DRILLING FOR GEOLOGY II
EXTENDED ABSTRACTS

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Bulletin number 64
ISBN 0-9750047-6-X
ISSN 0812 60 89

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Australian Institute of Geoscientists
PO Box 576
Crows Newst NSW 1585
Australia
On behalf of the Queensland branch of the Australian Institute of Geoscientists (AIG), I welcome all delegates to the Drilling for Geology II Conference in Brisbane.

The inaugural Drilling for Geology conference was held in 2008, during what some industry pundits were calling “a twenty-year boom” to be driven by China, then India and other developing regions. Much water has passed under the bridge since then, as the minerals industry peaked, experienced the global financial crisis, rebounded, then went through a significant industry downturn. ‘Green shoots’ started emerging in 2016, but the overall minerals industry remains sluggish, except in selected commodities.

So, this is the setting for the second instalment of the hugely successful Drilling for Geology conference held back in 2008. Drilling for Geology II follows the same format as the original conference. The first two days comprise excellent technical presentations with a concurrent trade exhibition. All conference delegates will receive a hardcopy and softcopy of the Extended Abstracts volume containing thirty papers across six themes:

- Industry overview
- Drilling techniques
- Drilling logistics
- Sensing and geophysics
- Drillhole sampling and logging methods
- Making better use of drilling data

The third day is devoted to professional development, and we have nine scheduled workshops – a mix of half-day, full-day and a three-day workshop – all presenting relevant drilling-related content to geoscientists at all stages in their career.

Drilling for Geology II is not just a Queensland or Australian-focused event. We have five international papers and conference delegates (at the time of writing) either based or working in Indonesia, Philippines, PNG, New Zealand and USA.

In keeping with AIG ideals, this conference is designed to be inexpensive and promote healthy discussion and interaction within the geoscience community. Low registration fees are only possible because of the generous financial support from industry, the time devoted by speakers to prepare their contributions, and the work of the committee (who give their time on a voluntary basis).

Financial support from sponsors, exhibitors and supporters of Drilling for Geology II has been tremendous and has underpinned the low costs for the conference. What other organisation offers its members a two-day technical conference with conference volume, welcome drinks and conference dinner for $500, less than this for students and retired members! In addition, three of the professional development workshops are free for conference delegates, and all other workshops are competitively priced to attract delegates to attend.

My thanks go to all the conference contributors (presenters, sponsors, exhibitors and supporters), as well as the conference organising committee comprising Rod Carlson, Josh Leigh, Fergus O’Brien, Graham Pope and Michele Pilkington.

I hope you find the conference stimulating and educational, and enjoy the company of your conference colleagues.

Mark Berry
Conference Chair
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EXHIBITORS

GENERAL SPONSORS

Daly Bros Drilling Contractors
<table>
<thead>
<tr>
<th>Scheduled Professional Development Workshops – 28 July 2017</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Drilling for Non-Drillers</strong></td>
</tr>
<tr>
<td><em>Australian Drilling Industry Association</em></td>
</tr>
<tr>
<td><strong>From Data to Informed Decision Making</strong></td>
</tr>
<tr>
<td><em>Imdex Limited</em></td>
</tr>
<tr>
<td><strong>Structure Logging of Drill Core</strong></td>
</tr>
<tr>
<td><em>SJS Resource Management Pty Ltd</em></td>
</tr>
<tr>
<td><strong>Drilling Methods – Impacts on Sampling, Measurements and Tests for Deposit Characterisation</strong></td>
</tr>
<tr>
<td><em>JKTech</em></td>
</tr>
<tr>
<td><strong>Laboratory visit</strong></td>
</tr>
<tr>
<td><em>Australian Laboratory Services (ALS)</em></td>
</tr>
<tr>
<td><strong>Qld Govt Exploration Data Centre visit</strong></td>
</tr>
<tr>
<td><em>Department Natural Resources and Mines</em></td>
</tr>
<tr>
<td><strong>Drilling Services Company visit</strong></td>
</tr>
<tr>
<td><em>Mitchell Services</em></td>
</tr>
<tr>
<td><strong>Lessons Learnt from Auditing Drilling Programs</strong></td>
</tr>
<tr>
<td><em>AMC Consultants Pty Ltd</em></td>
</tr>
<tr>
<td><strong>3D Modelling of Drilling Data using Leapfrog Geo</strong></td>
</tr>
<tr>
<td><em>Dale Sims Consulting Pty Ltd</em></td>
</tr>
</tbody>
</table>
# TABLE OF CONTENTS

## INDUSTRY OVERVIEW

<table>
<thead>
<tr>
<th>Topic</th>
<th>Author(s)</th>
<th>Page</th>
</tr>
</thead>
<tbody>
<tr>
<td>Australian Drilling Trends: Boom to Bust and Beyond</td>
<td>Mark Berry</td>
<td>3</td>
</tr>
<tr>
<td>Drilling for Fast Discovery and Development</td>
<td>Joe Cucuzza and Adele Seymon</td>
<td>9</td>
</tr>
<tr>
<td>The Resilience of Australia’s Drilling Industry, but Where to Next?</td>
<td>Peter Hall</td>
<td>15</td>
</tr>
</tbody>
</table>

## DRILLING TECHNIQUES

<table>
<thead>
<tr>
<th>Topic</th>
<th>Author(s)</th>
<th>Page</th>
</tr>
</thead>
<tbody>
<tr>
<td>The Coiled Tubing Drilling Rig for Mineral Exploration: The RoXplorer®</td>
<td>Soren Soe</td>
<td>25</td>
</tr>
<tr>
<td>Millimetre-Wave Directed Energy Deep Boreholes</td>
<td>Paul Woskov</td>
<td>27</td>
</tr>
</tbody>
</table>

## DRILLING LOGISTICS

<table>
<thead>
<tr>
<th>Topic</th>
<th>Author(s)</th>
<th>Page</th>
</tr>
</thead>
<tbody>
<tr>
<td>Drilling for Studies in Remote Locations</td>
<td>Mark S. Rynhoud</td>
<td>39</td>
</tr>
<tr>
<td>Multiple Intersection Directional Drilling on a Lowest Cost per Intersection Model</td>
<td>Mat Scarlett</td>
<td>43</td>
</tr>
<tr>
<td>Successfully Managing Drill Programs</td>
<td>Simon Shakesby</td>
<td>47</td>
</tr>
</tbody>
</table>

## SENSING AND GEOPHYSICS

<table>
<thead>
<tr>
<th>Topic</th>
<th>Author(s)</th>
<th>Page</th>
</tr>
</thead>
<tbody>
<tr>
<td>Reconnaissance Drilling and Sampling with the RoXplorer®: A New Tool for ‘Prospecting Drilling’</td>
<td>David Giles</td>
<td>57</td>
</tr>
<tr>
<td>The Measurement of Permeability and other Ground Fluid Parameters</td>
<td>Ian Gray</td>
<td>59</td>
</tr>
<tr>
<td>The Use of Automated Core Logging Technology to Improve Estimation of Fracture Mineralogy and Weathering for Geotechnical Index Calculations</td>
<td>Cassady L. Harraden, Matthew J. Cracknell, James Lett and Ron Berry</td>
<td>73</td>
</tr>
<tr>
<td>Logging-While-Drilling with Diamond Rigs</td>
<td>A. Kepic, M. Carson, H. Nguyen, A. Greenwood, A. Podolska and C. Dupuis</td>
<td>81</td>
</tr>
<tr>
<td>Application of Hand-held Laser Induced Breakdown Spectroscopy to Drilling Samples: New Technology Providing New In-field Analytical Capabilities</td>
<td>Andrew Somers</td>
<td>89</td>
</tr>
</tbody>
</table>
# TABLE OF CONTENTS

## DRILLHOLE SAMPLING AND LOGGING

<table>
<thead>
<tr>
<th>Title</th>
<th>Author(s)</th>
<th>Page</th>
</tr>
</thead>
<tbody>
<tr>
<td>The Use of Large Diameter Rotary Drilling (BAUER) for Bulk Metallurgical Sampling for the Boyongan Porphyry Cu-Au Deposit, Philippines</td>
<td>J. J. N. Lamson and C. J. C. Manipon</td>
<td>99</td>
</tr>
<tr>
<td>Bulk Density Measurement: Learning Through Case Histories</td>
<td>Ian Lipton</td>
<td>103</td>
</tr>
<tr>
<td>Drilling for Industrial Minerals: Quality Procedures and the End User</td>
<td>Patrick Maher and Andrew Scogings</td>
<td>109</td>
</tr>
<tr>
<td>Rethinking Drilling and Sampling: Smart Connected Systems</td>
<td>Viv Preston</td>
<td>115</td>
</tr>
<tr>
<td>Accountable Geo-Logging</td>
<td>Ray Slater</td>
<td>119</td>
</tr>
<tr>
<td>RC Drill Sample Optimisation: The Quantity – Quality Balance</td>
<td>Rene Sterk and Duncan Franey</td>
<td>125</td>
</tr>
</tbody>
</table>

## MAKING BETTER USE OF DRILLING DATA

<table>
<thead>
<tr>
<th>Title</th>
<th>Author(s)</th>
<th>Page</th>
</tr>
</thead>
<tbody>
<tr>
<td>Drilling for Mineral Processing Plant Design and Performance</td>
<td>Rob Chesher and Ian Lipton</td>
<td>143</td>
</tr>
<tr>
<td>Making Better Use of the Core That We Drill Using Detailed Petrographic Analysis</td>
<td>Rowena Duckworth</td>
<td>147</td>
</tr>
<tr>
<td>The Drill Down on Drill Core</td>
<td>Susan Faulkner</td>
<td>151</td>
</tr>
<tr>
<td>Stress in the Ground</td>
<td>Ian Gray</td>
<td>157</td>
</tr>
<tr>
<td>Leveraging the Value of Drilling for Orebody Knowledge</td>
<td>J. Jackson</td>
<td>177</td>
</tr>
<tr>
<td>3D Modelling for Exploration Geology: How to Get the Most from Data</td>
<td>Ben Jupp, Matthew Greentree and Melanie Sutterby</td>
<td>187</td>
</tr>
<tr>
<td>The Australian National Virtual Core Library: Investigating Mineral Systems Across the Australian Continent</td>
<td>Carsten Laukamp, Suraj Gopalakrishnan, Monica leGras, David Green, Lena Hancock, Ian Lau, Peter Mason, Alan Mauger, David Tilley, Belinda Smith, Peter Warren, Rob Woodcock</td>
<td>193</td>
</tr>
<tr>
<td>‘All that Glitters is not Gold’: Exploring for More Than the Pay Dirt Whilst Drilling for a Sustainable Future</td>
<td>Justin Legg</td>
<td>197</td>
</tr>
<tr>
<td>Drilling in the Fourth Dimension: The Practice and Utility of Drill Sample Collection, its Current and Future Potential, and the Role of Government</td>
<td>Fergus O’Brien</td>
<td>207</td>
</tr>
<tr>
<td>Data Upcycling</td>
<td>Julian Vearncombe</td>
<td>215</td>
</tr>
</tbody>
</table>
INTRODUCTION

At the inaugural Drilling for Geology conference held in October 2008, drilling trends based on Australian Bureau of Statistics (ABS) quarterly statistics were presented that documented drilling-related trends for the Australian minerals industry from 1997 through to 2007.

Much water has passed under the bridge since then, as the minerals industry peaked, experienced the global financial crisis (GFC), rebounded, then went through a significant industry downturn. ‘Green shoots’ started emerging in 2016, but the overall minerals industry remains rather sluggish, except for selected commodities.

In this paper, the latest ABS statistics are presented to provide an update to the original 2008 presentation, and provide a snapshot of current Australian mineral industry exploration and drilling activity.

SOURCE OF DATA

The ABS routinely publishes a range of quarterly and half yearly drilling-related data collected from the private minerals and petroleum exploration industries operating in Australia. The type of data routinely collected by the ABS now includes:

- Metreage drilled and expenditure for both the minerals and petroleum sectors.
- Minerals industry expenditure by state and territory.
- Minerals industry expenditure by commodity groups.
- Petroleum industry expenditure by state and territory.
- Petroleum industry expenditure for onshore and offshore areas.

The ABS has a range of definitions and guidelines for companies to use when completing the quarterly census (refer to Table 1 for an explanation of some of these terms). Importantly, the expenditure figures reported by ABS in its quarterly statistics are total Exploration Expenditure rather than Drilling Expenditure (refer to Table 1).

Some guidelines have changed with time and this can make direct comparisons of data difficult and assessment of trends ambiguous. An important change was made in 2003, when the ABS changed the way it reported the categories of minerals industry drilling activity. Up until 2003, statistics were reported for drilling undertaken on Production Leases and drilling undertaken on All Other Areas. From 2003, onwards statistics are reported for Existing Deposits and New Deposits. Consequently, for this review, data from July 2003 to March 2017 have been compiled to ensure consistency in the data presented.

AUSTRALIA-WIDE DRILLING AND EXPENDITURE STATISTICS

Figure 1 presents statistics for the Australia-wide total Exploration Expenditure and total mineral drilling metres (Existing Deposits and New Deposits) completed from July 2003 to March 2017. The key observations are:

- Drilled metres and Exploration Expenditure rose progressively from July 2003 to a peak in June 2008, prior to the onset of the GFC.
- The GFC resulted in a short, sharp correction, but drilling and expenditure started rising again in the June 2009 quarter, where it peaked throughout the 2011/12 financial year.
- Drilling and expenditure fell sharply from July 2012 and continued to fall through to March 2015.
- From March 2015 to March 2017, expenditure has been relatively flat, but the statistics for drilled metres shows that it bottomed in the March 2015 quarter and is slowly but steadily increasing.
- Present drilling and expenditure rates are similar to those experienced in 2005 and 2006. Exploration Expenditure is in dollars-of-the-day, and therefore does not take inflation into account.

1 Director and Principal Geologist, Derisk Geomining Consultants Pty Ltd
Table 1. Selected glossary used by ABS for exploration census data collection (summarised from ABS Publication 8412.0, March Quarter 2017).

<table>
<thead>
<tr>
<th>TERM</th>
<th>ABS DEFINITION</th>
</tr>
</thead>
<tbody>
<tr>
<td>Minerals</td>
<td>Are a naturally occurring inorganic element or compound having an orderly internal structure and characteristic chemical composition, crystal form, and physical properties. These, for example, comprise of metallic minerals, such as copper, silver, lead-zinc, nickel, cobalt, gold, iron ore, mineral sands, uranium and non-metallic minerals such as coal, diamonds and other precious and semi-precious stones and construction materials (e.g. gravel and sand).</td>
</tr>
<tr>
<td>Exploration</td>
<td>Activity involves searching for concentrations of naturally occurring solid, liquid or gaseous materials and includes new field wildcat and stratigraphical and extension/appraisal wells and mineral appraisals intended to delineate or greatly extend the limits of known deposits by geological, geophysical, geochemical, drilling or other methods. This includes drilling of boreholes, construction of shafts and adits primarily for exploration purposes but excludes activity of a developmental or production nature. Exploration for water is excluded.</td>
</tr>
<tr>
<td>Existing Deposit</td>
<td>Exploration that is delineating or proving up an existing deposit, including extensions and infill, which has been classified as an Inferred Mineral Resource or higher.</td>
</tr>
<tr>
<td>New Deposit</td>
<td>Exploration on previously unknown mineralisations or known mineralisations yet to be classified as an Inferred Mineral Resource or higher. They include:</td>
</tr>
<tr>
<td></td>
<td>• Exploration resulting in finding mineralisation that was previously unknown.</td>
</tr>
<tr>
<td></td>
<td>• Exploration on previously known mineralisation that has not been subjected to modern exploration.</td>
</tr>
<tr>
<td></td>
<td>• Exploration within an existing mining tenement for the purpose of finding new sources of mineralisation that have not already been classified as at least an Inferred Mineral Resource.</td>
</tr>
<tr>
<td>Exploration Expenditure *</td>
<td>Covers all expenditure (capitalised and non-capitalised) during the exploratory or evaluation stages in Australia, Australian waters, and the Joint Petroleum Development Area. Costs include cost of exploration, determination of recoverable reserves, engineering and economic feasibility studies, procurement of finance, gaining access to reserves, construction of pilot plants and all technical and administrative overheads directly associated with these functions. Examples are costs of satellite imagery, airborne and seismic surveys, use of geophysical and other instruments, geochemical surveys and map preparation; licence fees, land access and legal costs; geologist inspections, chemical analysis and payments to employees and contractors. Cash bids for offshore petroleum exploration permits are also included.</td>
</tr>
<tr>
<td>Drilling Expenditure</td>
<td>Includes wages and salaries paid to employees; purchase, rental, hiring as well as operation and maintenance of drilling equipment together with activities associated with accessing the areas where drilling is to occur (e.g. road creation, vessel/transport hiring, site preparation and restoration). Also includes expenditure on drilling done by contractors.</td>
</tr>
<tr>
<td>Other Expenditure</td>
<td>Includes all other exploration costs, other than those associated with drilling expenditure. This expenditure includes purchase of capital and non-capital items, rental or hiring fees, service fees relating to surveying and analysis, administrative and legal fees associated with obtaining licences/permits, land access, map preparation, feasibility studies, environmental impacts studies and restoration costs.</td>
</tr>
</tbody>
</table>

* This is the quarterly expenditure that is reported by ABS, and presented in this paper.
DRILLING AND EXPENDITURE ON EXISTING DEPOSITS AND NEW DEPOSITS

Figure 2 presents statistics for the Australia-wide Exploration Expenditure (expressed as $/m drilled) and metres drilled targeting Existing Deposits, and Figure 3 presents the same statistics for New Deposits. The key observations are:

- In 2003/2004, drilled metres on Existing Deposits and New Deposits was similar, but industry started focusing more attention on Existing Deposits from 2005 and this trend has continued to the present day, becoming more pronounced after the GFC. For the last three years drilled metres at Existing Deposits comprises between 70-80% of the total drilled metres in Australia.
- Drilled metres on New Deposits showed a steady growth up to the GFC, but did not rebound after the GFC, unlike drilling at Existing Deposits. From December 2013 to June 2016, drilled metres on New Deposits languished at rates significantly less than that observed in the early-mid 2000’s.
- In the last four years, drilling on Existing Deposits has been relatively static, and in fact has fallen slightly in each of the last three quarters. In contrast drilling on New Deposits in the last year is showing signs of improvement.
- Exploration Expenditure, expressed as dollars/metre drilled, progressively rose during the mining boom for both Existing Deposits and New Deposits:
  - For Existing Deposits, expenditure averaged approximately $150/m drilled in the early 2000’s and peaked at approximately $400/m in 2011/12, before dropping back to levels of around $400/m now.
  - For New Deposits, expenditure averaged approximately $110/m drilled in the early 2000’s and peaked at approximately $400/m, but the peak didn’t occur until 2013/14 (two years after the peak in Existing Deposits), and well after the peak in actual drilled metres. Presumably, the industry was undertaking technical studies and other activities and less drilling. Expenditure rates have taken a long time to drop back to $400/m, only reaching this level in the last three quarters, in parallel with increasing drilled metres.
Figure 2. Quarterly drilling and expenditure statistics for existing deposits: July 2003 to March 2017 (Source ABS).

Figure 3. Quarterly drilling and expenditure statistics for new deposits: July 2003 to March 2017 (Source ABS).
EXPLORATION EXPENDITURE BY STATE AND TERRITORY

The ABS quarterly reports do not provide drilling statistics grouped by Australian state and territory. Figure 4 shows the proportion of total Exploration Expenditure spent in each Australian state and the Northern Territory. The key observations are:

- In broad terms, the proportion of total Exploration Expenditure spent in each state and the Northern Territory has remained relatively consistent for most of the 14 years covered by the plot. Western Australia consistently gets 50-60% of expenditure, followed by Queensland with 15-25%. New South Wales, South Australia and the Northern Territory each have 5-10%, with Victoria and Tasmania receiving generally less than 5%.

- Some specific observations include:
  - South Australia experienced a three-year expenditure boost from early 2006 – driven mostly by base/precious metals, but expenditure has proportionally declined since then.
  - Queensland experienced a four-year expenditure boost from mid-2010 – driven mostly by coal, but expenditure has proportionally declined since then.
  - In the last year, expenditure has proportionally increased in Western Australia, reaching 65% of total expenditure, driven by gold and specialty metals e.g. lithium.
  - Victoria has seen a steady reduction in the proportion of expenditure spent in the state.

Figure 4. Quarterly exploration expenditure for Australian state and territory: July 2003 to March 2017 (Source ABS).

EXPLORATION EXPENDITURE BY COMMODITY

The ABS quarterly reports do not provide drilling statistics grouped by commodity. Figure 5 shows the proportion of total Exploration Expenditure spent on each of ten commodity groupings. The key observations are:

- Copper. Exploration Expenditure proportionally spent on copper significantly increased from 2003 through to 2007, and has then remained generally stable, except for a short decline during the GFC.
- Lead, zinc and silver. Expenditure proportionally spent on lead, zinc and silver has remained stable, except for a period of higher activity from 2005 to 2008.
- Nickel and cobalt. Expenditure on nickel and cobalt was proportionally highest from 2003 through the GFC, but has progressively fallen since then.

Figure 5. Quarterly statistics: July 2003 to March 2017 (Source ABS).
• Gold. In 2003, gold exploration comprised just over 50% of total exploration, but this progressively fell in the lead up to the GFC, falling to approximately 30%. From the GFC through to the end of 2014, gold exploration comprised approximately 20% of total expenditure, but has strongly recovered since then, now comprising approximately 45% of expenditure.

• Iron ore. Proportional expenditure on iron ore grew steadily from 2003 (less than 10%) to comprise approximately 30% of expenditure from 2009 through to 2014. Since 2014, expenditure on iron ore has fallen back in line with its price, now at 15% of total expenditure.

• Uranium. Proportional expenditure on uranium steadily fell from 2003 to 2010, and was negligible from 2010 to 2011. From 2011, proportional expenditure on uranium has risen again, but has not reached the same levels since in the early 2000’s.

• Coal. Proportional expenditure on coal fell from 2003 to 2007, but then grew to comprise 15-20% of total expenditure from 2011 to 2015. Proportional expenditure on coal dropped sharply in 2016 to its current levels.

• Diamonds. Expenditure on diamonds progressively fell from 2003 to the GFC and has remained at very low levels since then.

Figure 5. Quarterly exploration expenditure for commodities: July 2003 to March 2017 (Source ABS).

CONCLUSION
The routine ABS census collected for the Australian minerals industry is a valuable source of drilling related data. The cyclical nature of drilling and Exploration Expenditure is clearly evident, illustrating the ramping up of the mining boom, the short-term effects of the global financial crisis, a rebound followed by a significant industry downturn. The data clearly shows that activity has started again since 2015/2016, but the overall minerals industry remains rather sluggish, except for selected commodities.

REFERENCES

INTRODUCTION

The days of easy discoveries are long gone. This is evident by the significant decline in the rate of world-class discoveries over the last several decades. There are many reasons for this of course and one is clearly that the shallow mineralisation has been found, particularly in the more “mature” areas. We now need to look deeper for those world-class discoveries, whether in areas of deeper cover or deeper in existing mines. With this in mind, are the old methods and techniques and indeed ways of going about the exploration process going to be enough to improve discovery rates in areas of deeper cover?

We believe that to achieve a step change in discovery rate, a new modus operandi will be required. This new modus operandi will require a new level of collaboration between explorers, access to new pre-competitive data, and development of new technologies including better data analytic tools. Indeed, we may need new decision making approaches, for example applying novel ideas around gamefication. According to Sérgio Brodsky, Head of Strategy at Initiative, gamifying strategy could contribute to a new business model that can readily embrace complexity, smoothen change, and has the potential to transform how people work in an enterprise (pers. comm.).

AMIRA International’s Roadmap for Exploration Under Cover (AMIRA International, 2015) points the way in terms of what research and development (R&D) needs to be done to improve our chances of success in areas of deeper cover. For the first time this ambitious initiative provided a strategic opportunity for Australia's mineral explorers, government geological survey organisations, and the research community, to define the longer-term integrated R&D, and accelerated data compilation and acquisition program that ultimately will improve the performance and success rate of mineral exploration in areas of Australia concealed under post-mineralisation cover. The size of the challenge highlighted by the Roadmap underscores the critical importance of the need for collective endeavour in implementing the Roadmap outcomes. Regrettably there is already evidence that some stakeholders particularly in the research community, are being driven by self-interest which may pay off marginally in the short term if they are able to secure some government funding, but will be destructive to the greater common cause in the long term. The cause that should be driving collective action is the vision outlined by the Roadmap, endorsed by the Roadmap sponsors, and one that cannot simply be achieved through anything but true collaboration. There are certainly challenges in implementing the Roadmap, not the least of which is how sustainable funding can be secured to ensure that a credible program can be realised. It may mean a more modest start with an appropriately designed research vehicle that will be tasked to undertake the necessary R&D. AMIRA International is committed to work with all stakeholders who have a more strategic view to create such a vehicle and find ways that the necessary programme can be funded.

DISCOVERY TO DEVELOPMENT LEAD TIME

Anecdotal evidence suggests that, depending on the commodity and the location relative to known mining camps and existing infrastructure, it could take up to ten years to discover a commercial resource. Unfortunately, there are no readily available statistics on discovery timelines but intuitively one would expect that the situation will get worse as explorers move more into areas of deeper cover. This is another reason why we must do things differently. The mantra must be to minimise risk of failure but to fail fast and fail cheaply.

Once we have been fortunate enough to make an initial discovery another crucial challenge industry faces is to speed up the development process. Again, anecdotal evidence suggests that, depending on the nature of the orebody and its location relative to infrastructure, it could take up to 10 plus years from discovery to production. Notwithstanding the problem of government regulations which can introduce significant delays, it is clear that reducing the development time would create considerable value for all stakeholders.

1 Managing Director, AMIRA International Limited
2 Program Director and Global Business Development Manager, AMIRA International Limited
We are not aware of data that show the changes in timelines from discovery to production in minerals over time, but we recently came across some data from the petroleum industry in the deepwater of the U.S. Gulf of Mexico which is instructive (Wessler, 2011). The Gulf of Mexico is arguably an extreme environment with water depths of 1,400 m or more. The petroleum industry experienced a marked acceleration of the timeline from discovery to production from the 1980s, 1990s and 2000s, when the average interval for deepwater fields was 11.4 years, 5.9 years, and 3.3 years, respectively (Figure 1).

Figure 1. Duration between field discovery and production by decade in the deepwater Gulf of Mexico (Wessler, 2011).

What is remarkable about the data is that the petroleum industry was able to reduce the average timeline by more than 70% in some thirty years. It appears that the proximity to existing production and closeness to infrastructure are the primary reasons for this incredible change. The importance of infrastructure is underscored by the fact that a number of 2000s-vintage discoveries at locations inaccessible to existing infrastructure have taken 7 to 10 year to reach production. However, improved and new technology also played an important role. For example, the advent of 3D seismic and improved drilling and production equipment designed for ultra-deep waters have facilitated exploitation of deeper and more remote resources. One should not discount the possibility that the petroleum industry in the Gulf of Mexico did not face the level of regulations that miners currently face when developing a mine.

It is possible that in the minerals industry we may have experienced similar acceleration in time from discovery to production during the same 30+ years’ time frame, particularly for those orebodies that are close to existing mines and infrastructure, but it is doubtful that they would reach the magnitude of the acceleration reported here. It is true that sometimes a decision to develop a mine may take some time to reach. Discoveries without a decision to develop represent substantial value – to maximise the value the deposit needs to be put into production quickly. However, it often takes a long time before a discovery is considered sufficiently commercial to be mined. Indeed, for all sorts of reasons, less than half of all discoveries made in the world since 1950 have been put into production (Schoe, 2014), as shown in Figure 2. For those mines that do end up in production the average delay between discovery and production is 12.4 year, although this does vary across commodities, e.g. it is about 18.4 years for copper. The former is remarkably close to the average experienced by the petroleum companies in the Gulf of Mexico in the 1970’s.
There is no doubt that in the future, location will continue to play a key role in determining development timelines for both new and existing discoveries. But in venturing to areas of deeper cover, unless we do things differently, develop new technologies and acquire new data, it is difficult to see how the discovery timeline is going to be accelerated.

THE ROLE OF DRILLING IN FAST DISCOVERY AND DEVELOPMENT

So how will drilling play a big role in “Fast Discovery and Development”?

Let’s look first at the discovery process. Drilling is going to be critical to prove a discovery, we don’t have the good fortune that the petroleum explorers have, where given the right circumstances they are able to see direct indication of hydrocarbons in appropriately processed seismic data. Apparently, the key factor in this capability has been the use of information-preserving processing of the seismic data, which allows both the conventional balanced amplitude and the true amplitude to be displayed—the former for structural interpretation and the latter for more positive identification of the presence of hydrocarbons. In the absence of indirect detection methods allowing us to improve our ability to identify mineralisation, we need to drill to obtain evidence of mineralisation. However, a number of things need to change before drilling is going to be of greater utility in improving discovery rates.

Firstly, drilling needs to be cheaper, faster and more steerable. This does not mean that explorers are going to drill fewer holes, indeed it’s highly likely that more holes will be drilled. The fact of the matter is that all things being equal the more holes we drill, obviously in the right place, the higher the chance of making a discovery.

Drilling technology is certainly not staying still. Recently, a prototype coiled tubing drill rig, the RoXplorer®, was developed by the Deep Exploration Technologies Cooperative Research Centre (DET CRC) and underwent its first extensive test at Port Augusta in South Australia. Coiled tubing drilling differs from conventional drilling in that the drill string is a continuous, malleable steel coil, in contrast to being comprised of individual steel rods that must be connected and disconnected. The RoXplorer® rig is a hybrid rig and first drilled, cased with steel pipe and cemented the top ~30 m of the hole. The main hole was then drilled through the cement and into open formation with a downhole hammer and percussion bit powered by a downhole motor. The successful trials represent the culmination of some $20 million in a research project by the DET CRC to develop a next generation drill rig for greenfields mineral exploration that can drill at a cost of A$50/m to a depth of 500 m. The DET CRC was the result of an AMIRA International led industry-sponsored bid based on AMIRA’s Drilling Technology Roadmap.
However, this type of technology is still in its infancy and unsuitable to drillholes down to two kilometres.

Secondly, we need to develop technologies that will provide in-situ data on chemistry, mineralogy, geotechnical properties, and ideally assays, in real or near-real time. Historically we have not made good use of drillholes. There are good reasons for this of course, cost and lack of technologies for example. However, the advent of new technologies should address this. The petroleum industry is miles ahead, albeit they don’t have the hole-size restriction that we face in minerals. Vectoring technologies that will utilise downhole chemistry and other data need to be developed that will point towards the epicentre of economic mineralisation. Over the last 14 years, industry, through AMIRA International, has invested over AUS10 million, in present dollar terms, with the ARC Centre of Excellence in Ore Deposits (CODES) and more recently with the ARC Research Hub for Transforming the Mining Value Chain at the University of Tasmania, and partners developing geochemical vectors using samples from both outcrop and drill core. Some remarkable results have been achieved although a readily field-implementable solution still eludes us - the work continues. Once this is cracked however, we need to transfer this capability to drill technology and in near-real time. Those who feel that this result can be achieved without considerable investment are kidding themselves.

Thirdly, we need to adopt the petroleum approach of only coring where it’s absolutely necessary. The RoXplorer® may lead the way but more needs to be done that will enable us to access petroleum-like capability in our drillholes particularly at greater depths.

Despite some advancements therefore, at the exploration end of the business we do not yet have the capability to drill faster, cheaper, and smarter to the sort of depths that we may need to drill in areas of deep cover. Neither do we have access to cost-effective technologies that will enable explorers to extract the maximum amount of data from drillholes and indeed to turn such data into valuable information.

Once a discovery has been made of course the aim is to determine whether there is a commercial resource. This depends on many factors, one important element of which is the prevailing and future commodity prices. The challenge is to reach a decision on whether or not to develop the orebody as quickly as possible. At this stage, typically closer spaced holes are drilled and resultant data is then used to build geological models, which help establish quantity and quality of the ore, and resource models, which help establish the size of the reserves and resources present.

The feasibility studies undertaken at this juncture in the process help determine which mining and processing options should be utilised, and what marketing opportunities there may be that ultimately will establish whether the ore body can be mined profitably at that point in time.

Feasibility studies can take a considerable amount of time as they need to address issues that include environmental impact, effects on local communities, the demand on resources such as water and power and of course mine closure implications. This does not include the sometime significant delays that result from seeking the necessary state and federal government approvals. This stage in the process is a critical step towards the development of the resource model leading to estimates of grade and tonnage. Here, as with greenfield exploration, there is a need to be able to make maximum use of the drillholes.

Ideally, we want to develop a resource model in real time and with the minimum number of holes necessary for the type of ore body under investigation to ensure that a bankable feasibility study is achieved quickly. This potentially could be accelerated through new technologies that enable better imaging of the ore body so that grade continuity, geomechanical and other important properties can be determined down and across holes. But the aim must be to develop the technology that makes possible real time ore body definition and enables the resource model o be updated in real time.

Although significant progress has been made in applying seismic and cross-hole tomography in hard rock environments, the results are still heavily dependent on the nature of the ore body and other factors. Clearly seismics, particularly down hole, can provide valuable information on the geomechanical properties of the ore body and host. Using the right geophysical method can potentially help to reduce uncertainty but as Chris Wijns (Wijns, 2017) recently pointed out at the recent Target 2017 Conference there are still challenges, for example “translating the geophysical response into useable ore body characteristics”. Addressing this challenge will help to accelerate the development process.

Another technology that will accelerate the development process is one that will do away with the need for continuous coring and minimise or indeed eliminate the requirement to dispatch samples to the laboratory.
for analysis. Uranium producers have been able to achieve market acceptance for in-situ analyses to determine grade. Ultimately, we need to aim to achieve similar outcome with other commodities through in-situ chemistry, mineralogy and of course grade. This will obviously not be simple but it is a great ‘moon shot’ aim. In the interim, technologies that provide data obtained at the rig are an important step forward. Lab-at-Rig® developed by DET CRC participants, CSIRO, Imdex and Olympus Scientific Solutions Americas provides a means of measuring drillhole geochemistry and mineralogy at the drill site within minutes of the drilling process. AMIRA International is currently preparing a project that will develop the ColdBlock™ digestion technology that will replace the conventional fire assay method for gold analysis by offering on-site and near real-time analytics to improve data turn-around. ColdBlock™ digestion is the first sample digestion technology that uses short-wave infrared radiation.

There is no doubt metallurgical test work may be informed by in-situ measurements through appropriate proxies but it is unlikely that grinding and leaching tests, for example, can be undertaken without taking a representative physical sample of the ore. But can these tests be undertaken on carefully positioned “plug” samples?

CONCLUSION

From the foregoing discussion, it is clear that drilling is going to be playing a critical role in finding our next world-class ore body under cover. But a great many challenges need to be addressed if the time between discovery and development is not to increase beyond the levels that have been the experienced in the past. Indeed, the aim must be to accelerate discovery and development. This is possible through development of new technologies and indeed through better collaboration between explorers and other exploration stakeholders. In respect to the latter, perhaps the industry should do away with the restrictive practice of keeping all the data proprietary – if a company owns the ground maybe it will be more advantageous to share data with those around in order to be able to develop a better understanding of the geology from regional scale down to prospect scale. This type of data sharing has occurred successfully in other industries, i.e. the pharmaceutical sector. There is simply no doubt that sharing of data and information can not only create efficiencies, but more importantly it can create value by improving the chance of making a big discovery.

The availability of new technologies that will improve our ability to target mineralisation, and permit more information to be extracted from the drillhole and access to new pre-competitive data along with a more responsive regulatory environment combined with the ability to build of more flexible modular mines will enable the minerals industry to emulate the sort of shortening time line that the petroleum explorers experienced in the Gulf of Mexico.

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THE RESILIENCE OF AUSTRALIA’S DRILLING INDUSTRY,
BUT WHERE TO NEXT?

Peter Hall¹

INTRODUCTION

Nothing stays still for long in the drilling industry and this paper will examine what the state of the current market is for drilling contractors, how did this evolve and where things look like heading in the coming few years. The industry enjoyed a longer than usual period of high activity, followed by the worst downfall it has ever experienced, to a point we it is believed there is now some light at the end of the tunnel.

WHERE HAVE ALL THE RIGS AND PEOPLE GONE?

The ADIA can list 150 drilling companies that have closed their doors or become insolvent since the end of the exploration boom in 2012 and the final number is likely to be larger than this. These were companies ranging in capacity from smaller owner-operator enterprises, through to prominent drilling companies with large equipment fleets. Some of their equipment has since been recycled, ending up with other contractors still in operation.

Often rigs that cost $1.5-2 million in the latter stages of the boom have sold more recently for a quarter to one third of this value. This has been good for the drilling contractors, but not so helpful to the rig manufacturers. Other rigs have been scrapped or will be scrapped as there is only so much time an older rig can remain parked and still be functional without regular maintenance.

Likewise, experienced drilling personnel are always in demand due to the revolving nature of drilling contracts. Due to personnel shortages in the boom, there were a lot of people operating rigs that had limited experience and it is now likely that a lot of them have exited the industry. When activity does pick up a few notches, regrettably there will again be a shortage of experienced people capable of doing the work.

Whilst much of what a driller learns comes from practical experience gained at the job site, there is a requirement for all crew members to have a set of industry approved qualifications. The number of registered training organisations (RTOs) who manage and provide this training, has also significantly diminished because of lack of demand over recent years. Therefore, as demand returns, there will be fewer resources out there to train new entrants to the workforce.

This will correct itself over time, however crew shortages will be a reality. Drilling contractors are already stating that they are finding it difficult to find suitable new employees, especially at short notice which is often the case due to being awarded new work with short lead times to start.

MORE RIGS WERE THE ORDER OF THE DAY!

From about 2003 to the end of 2012, there was unprecedented growth in exploration drilling, not just in Australia, but on a global scale. The period from 2005 to 2007 saw a shortage of drill rigs on quite a massive scale, which was great news for the manufacturers who pumped hundreds of new rigs into the Australian market place alone. Contractors also found that if they could secure a rig, there was also then the problem of trying to kit it out with drill rods and tooling and to find competent crews to run it. These necessary requirements were all in short supply, which resulted in the manufacturing and service companies having to invest in their businesses to expand their capacities.

During the earlier part of this period, a lot of the drilling focus remained on base metals and the gold sector, which had begun to make its own comeback. Don’t forget the gold price crash of 1998 which heralded the end of the previous mining and drilling boom period.

¹ CEO, Australian Drilling Industry Association (ADIA)
THE GFC WAS A BRIEF INTERRUPTION

When the global financial crisis (GFC) hit in 2008, confidence quickly deserted the mining sector and as usual drilling became one of the first areas to be cut, as it does when investment and exploration budgets dry up. Fortunately, before drastic cuts were made widely across the sector, activity once again accelerated, this time with a focus on the bulk commodities of coal, coal seam gas and iron ore. For a period, this caused an exodus of drilling contractors from west to eastern Australia to enter the coal markets of NSW and Queensland. This was beneficial at the time but most of them have now returned to their home bases.

Figure 1. Drill rig utilisation levels.

LIFE AFTER THE GFC

2009 to 2011 saw a further increase in exploration drilling due to increased demand in the coal and iron ore sectors, with the drilling industry again stretched to maximum. During this period the general consensus from governments, investment communities and market analysts alike was that the “super cycle” as it was then being called, was going to last for another 20 years or more.

This forecast had the knock-on effect of drilling contractors looking to further increase the size of their fleets, which also meant a corresponding increase in people and infrastructure needs.

Rig utilisation during this period was at 90% which basically means full utilisation, with some rigs always parked up for repairs and maintenance. The number of rigs working in an exploration capacity during this time was somewhere around 800.

THE DOWNTURN

Drilling people who have been in the exploration game over a long period have the hard-won experience to know that the potential for a downturn in activity is ever existent but, also difficult to pick. One wise head once commented that the time to start pulling back and consolidating is when drilling companies start to sell for silly, over-inflated prices!

Regardless, late in 2012 there was a reduction in activity with the question on most people’s minds being for how long will it last! Was it another short term GFC blip, or was it something more permanent?
THE CURRENT SITUATION

Rig utilisation which had reached its 90% peak in 2011 had bottomed to about 30-35% late in 2015. There has been a slight pick-up in drilling activity over the last 12 months, however any major rise in utilisation would have occurred more from rigs being written off and scrapped. Utilisation now is somewhere at the mid 35% - 40% level, with a much higher percentage being in underground drilling rather than anything to do with surface and greenfield exploration. Reverse circulation (RC), waterwell, environmental and geotechnical methods of drilling are all reasonably active in select geographic locations.

SUPPLY AND DEMAND

Whilst the market has seen a bit of a pick-up, any drilling contractor will tell you that this has not resulted in any increase in drilling rates. This is a simple fact of supply and demand, and whilst there remain drilling companies who need to keep their rigs turning and their people employed, competitive bidding will continue to benefit the client. When rig utilisation does move higher and rigs become less readily available, will be the time when rates can be expected to start to rise.

There are issues with underbidding to win work which can reflect negatively on the drilling contractor and also the client:

- The contractor may not be depreciating his equipment adequately or putting sufficient aside for future maintenance and replacement needs.
- The relationship with the client can deteriorate simply because the contractor is not making sufficient income on the job. This can result in cost cutting efforts that may have flow on effects of demoralising the drill crews, whilst not giving the client the full-service expectation.

THE NEED TO DRILL

The adage that a driller is only making money when the drill bit is turning to the right couldn’t be more true. During the boom period, there was a client focus on many sites that resulted in drillers regularly spending less time drilling and a higher percentage of time on non-drilling activities. The client often paid sufficiently for these non-drilling activities during this period, but when contractors were asked to take large rate cuts from 2013 onwards, the time available to actually drill became much more critical.

Typical examples of these non-drilling activities include length of onsite roster for crew, timing of safety and toolbox meetings, site inspections, location of spares and other resources, training needs, crib breaks, site access, and fuel and water delivery. They are however also great opportunities of where positive collaboration can result in more metres being drilled per shift to the benefit of both parties.

WHERE TO FROM HERE?

The Australian drilling industry has weathered many industry downturns throughout its history and has always come out the other end leaner and smarter and ready to rebuild when demand increases. The creative and innovative thinking within the sector has a “can do” attitude and is always looking for competitive advantage and a more efficient way of working.

This is happening currently, with several product manufacturing and research companies working in conjunction with the mining sector and direct with drilling contractors to target better ways of doing things, with a win – win outcome the goal.

Most drilling contractors are having to constantly invest money and resources in equipment upgrades to satisfy the compliance requirements of their clients. This is positive for the industry and the flow on benefits to safety and the general welfare of the drill crews. Naturally it does come at a cost and somehow must be factored into contract rates.

EQUIPMENT NOW AND IN THE FUTURE

The method of exploration core drilling hasn’t changed significantly for over 50 years, other than some incremental improvements to drill bits and the rest of the drill string. Rotation of the drill bit and penetration rates can’t change too much further with the current drill rig types on the market and their capability to...
spin the rods. Even if the rigs could spin faster, problems with drill rod vibration would likely result as different diameter pipe already has optimum spin speeds under usual operating conditions. This means that other methods need to be developed to increase the amount of time that the bit remains on the bottom of the hole.

RC drilling has seen more recent advancement when compared with core drilling, largely due to the ability to provide more air capacity to the hammer. This has resulted in improved RC hammer designs and the ability to drill to greater depths.

WHERE IS THE INNOVATION IN EQUIPMENT?

A high level of engineering research has been directed towards hands-free rod-handling systems that remove direct human interaction away from the rig’s moving parts. It has already been proven that this will greatly reduce the potential for hand and back injuries on a drill site. Productivity can also be improved on deeper holes as the rod handler does not suffer from fatigue like a drill crew would.

Another area of innovation has been to give the rig more automated drilling capability. The primary intent here is to allow the rig to keep drilling while the crew is occupied doing other tasks and to get that extra metre or two drilled each shift. Over a month this can have quite a positive impact on income generation, however as with any equipment investment, there is a payback time to consider.

The development of rod handling systems and rig automation can only be satisfactorily achieved by increasing the amount of electronics on a drill rig. The use of electronics also opens other possibilities such as monitoring the rig remotely from another location, improving the ability for a company to better manage the rig’s maintenance and transfer of rig operating parameters in real time.

Another area to watch closely over the next few years will be the availability of new technology in down hole instrumentation and data logging. These tools have the potential to provide the client with endless information in a more efficient package. Some of the work being undertaken by the Deep Exploration Technologies Cooperative Research Centre (DET CRC) is quite revolutionary in this regard as it encompasses a complete new drilling system from the rig down to the bit and how the information is retrieved from the hole.

There is certainly a likelihood that in the next 10 years’ time, clients will have more choice in how they want their holes drilled and this will likely be dictated by the type of information needed and what its intended use is. There will be drilling contractors who embrace this new technology as they want to stay at the forefront of drilling method capabilities. This will require further investment not just in new hardware but also in training, maintenance planning, more spare parts, engineering, compliance manuals and other documentation.

WHAT ARE THE COST IMPLICATIONS?

Some of these requirements are often hidden costs that can go unbudgeted for. Drilling contractors in return need more certainty and protection in their client contracts for the ability to recoup their investment. At the end of the boom in 2012, there were many contractors left “high and dry” as drilling contracts were reduced or terminated and it is unlikely that this would occur in too many other industries without compensation.

In an era of low rates for work undertaken, it will continue to be hard for contractors to fund these necessary equipment upgrades. The fact is however, that most of them need to have the latest equipment offerings to retain their client base, or risk not being eligible for the work. The most recent reporting season again revealed that most of the major drilling companies are still not turning a profit, perhaps not incurring losses on the scale of previous years but still the questions needs to be asked of how long can this continue?

The ADIA realises that there are market forces at work and that the normal supply and demand scenario needs to play itself out, but at what permanent damage to the drilling industry? It is in our shared interests that the Australian drilling industry is in a healthy state to be able to meet the future demands from the mining and exploration sectors.
DRILLING TECHNIQUES
HARNESSING THE POWER OF EXTREME VELOCITY FOR DRILLING: A PARADIGM SHIFT IN ACCESS TO THE WORLD’S GEOLOGIC RESOURCES

MARK C. RUSSELL 1 AND TIM ELDER1

INTRODUCTION

Repetitive hypervelocity rock impact and robotic automation technologies has the potential to be used to solve challenging problems in energy, transportation and materials production. HyperSciences Inc (HSI) is focusing its research and development activities on tunnelling, mining and hard rock energy drilling. Funded by Shell for the past few years, HSI has developed its hypervelocity drill-by-impact platform technology and is commercialising its steerable HyperCore™ tools for HyperDrill™ and Hyper Tunneling and Mining system (HTBM), using continuous hypervelocity repetitive projectile impacts downhole for energy drilling and a high-speed replacement for drill and blast mining and rotary tunnelling.

HYPERCORE™

HyperCore™ technology provides low cost, extremely fast robotic/remote underground repetitive hypervelocity projectiles shot from tubular ram accelerators to speeds of 2 km/s (4,500 mph) using simple industrial gases (diesel/methane/air) as chemical propellants to achieve repetitive impacts every few seconds.

Hypervelocity uses the power of $V^2$ through repetitive short-duration impact with extreme velocity projectiles (Mach 4-6 or 1,500-2,000 m/s) impacting with a dynamic pressure ($\frac{1}{2}\rho V^2$) that is 10 - 100 times the compressive strength of the hardest rock (Figure 1), efficiently breaking and pulverising rock faster and at a lower cost than the best-in-class rotary cutting or drill and blast technologies.

Figure 1. Hypervelocity rock impact pressure vs rock strength.

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1 HyperSciences Inc
HYPERDRILL™

The HyperDrill™ steerable tool will use continuous hypervelocity repetitive projectile impacts from the HyperCore™ technology downhole at the drilling face, offering 10 times the rate of penetration (ROP) over conventional rotary drilling. A paradigm shift in energy access through low cost, extreme depth drilling, HyperDrill™ has the potential for low-cost access to deep energy, providing access to conventional oil and gas plays and deep engineered geothermal system (EGS) power generation.

HyperDrill™ is compatible with existing rigs. The dynamic pressure at impact is 10 - 100 times the compressive strength of the rock, resulting in up to 10 times the ROP increase compared to rotary drilling in high strength and high borehole pressure environments. This significantly improves conventional hard rock resource economics and enables access to new and unconventional oil, gas, and geothermal resources (Figure 2).

Figure 2. Hypervelocity ROP vs hole depth.

Key technologies of HyperDrill™ have been demonstrated. Experiments have shown that projectile penetration in rocks of interest and under simulated borehole conditions is consistent with analytical predictions. Furthermore, conceptual designs of the HyperDrill™ downhole tool have been completed, along with drill wells on paper exercises that show the potential economic impact of the technology.

Two embodiments of HyperDrill™ have been developed (Figure 3) that leverage the common HyperCore gun to generate the discrete projectile impacts:

- The pure HyperDrill™ has no bit integrated in the bottom hole assembly (BHA). The hole is entirely formed by projectile impacts. This version of the technology requires velocity in the 1.8–2 km/s range. In this velocity regime, each projectile impact creates a hole that is 2.7 to 3 times the diameter of the projectile.
- Augmented HyperDrill™ uses a gun integrated into a drill bit. The gun barrel exit is through the bit but is not centered on it. As the bit turns, the barrel rotates, tracing a circle around the hole bottom. The HyperCore gun used in Augmented HyperDrill™ is identical to the one developed for Pure HyperDrill™. The projectiles are used to create a crater and weaken the rock ahead of the rotary drill bit, increasing rate of penetration.
Figure 3. Pure HyperDrill™ and Augmented HyperDrill™ BHA configuration.
In November 2016, the Deep Exploration Technologies Cooperative Research Centre (DET CRC) launched the RoXplorer®, a coiled tubing (CT) drilling rig for greenfields mineral exploration, delivering a platform for low-cost, rapid, safe and environmentally-friendly drilling. The launch represented an important technical milestone in the development of CT drilling for mineral exploration, with the next steps being a series of field trials planned for the first half of 2017.

CT drilling technology has been used in the hydrocarbon industry for over 20 years. The majority of CT drilling applications are open-hole work-overs with a relatively small component of ‘active’ drilling, largely in unconventional oil and gas. The major advantage of CT drilling is the replacement of drill rods with a continuous ‘coiled’ drill string, thus eliminating the work, health and safety overhead associated with manual handling of drill rods and avoiding the time cost of making drill rod connections and disconnections. This has the potential to produce substantial improvements in bottom-hole time and hole-averaged rates-of-penetration. Further, in the absence of making rod connections the driller is able to continuously maintain in-hole pressures with the promise of improved hole stability. Despite these advantages, to date, CT drilling has not penetrated the mineral exploration drilling market.

The main challenges for CT drilling in mineral exploration are three-fold:

1. **Coil durability**: The action of winding and unwinding the coiled tube results in metal fatigue and eventually failure of the tubing. This places an upper limit on the number of holes that can be drilled with each drill string with a direct impact on cost efficiency. This is a greater challenge in mineral exploration applications – typically with more, shallower drillholes – than in hydrocarbon applications.

2. **Drilling hard rocks with low weight-on-bit**: CT drilling operates at lower weight-on-bit than conventional drilling and with the requirement that the entire drill string does not rotate. A key challenge is to design specialised bottom-hole-assemblies (BHAs) and associated operating procedures that are optimised to achieve high rates-of-penetration (ROP) when drilling both cover materials and hard, crystalline basement rocks with low weight-on-bit.

3. **Sample fidelity**: CT drilling does not provide the option of wireline recovery of a core barrel nor (at present) is there a reverse circulation option for CT drilling. Maximum ROP is gained utilising BHAs that produce cuttings with a range of particle sizes which are returned to surface within the drilling muds via the open hole. It is important to ensure that these cuttings are a representative sample of the rocks intersected by the drill bit, with minimal contamination or mixing en-route to surface.

DET CRC’s research agenda has been carefully planned to address these challenges in parallel and to incorporate innovations in each area into the ground-up design and build of the new CT drilling platform – the RoXplorer®. Our internally defined performance target for the RoXplorer® platform is ambitious by intention:

*To deliver a drill rig capable of drilling at a cost of $50/m to a depth of 500 m and weighing less than 10 tonnes with ancillary safety and environmental benefits.*

To this end, the RoXplorer® incorporates a number of revolutionary design elements:

1. The Reel is mounted vertically above the drillhole (in contrast the Reel is located horizontally behind the drillhole in conventional CT drill rigs), thus reducing the number of ‘bends’ experienced by the coil during each trip by a factor of three.

2. The CT material is a new-to-market steel tubing which delivers twice the fatigue life of conventional steel tubing.

3. The platform is able to operate with various BHAs, each optimised for specific drilling conditions. A significant effort has been made in this area and has involved the design of new drill bits and optimisation of operating procedures.
4. The operating platform offers computer-controlled electrics over hydraulics with several automated processes including live communication with the sensors in the BHA while drilling. These systems enable the high level of control required to efficiently run the down-hole drilling tools.

5. A closed fluid circulation system that incorporates fluid cleaning and delivers a fluid of very low solids load, as is required to run the BHA.

6. New drilling fluids that are suitable for the CT drilling system and have resulted in dramatic reductions in fluid loss and improved cuttings return when drilling broken or unconsolidated ground.

This work has called upon a diverse range of research skills, has required close collaboration between research organisations and DET CRC supplier participants Boart Longyear and Imdex Limited. The research has demanded a significant capital investment in engineering and advanced manufacturing. Such a collaboration would not have occurred without the facilitation of DET CRC.

Figure 1. DET CRC’s RoXplorer® drill rig on site at drilling trials near Port Augusta, February 2017.

Between December 2016 and July 2017, the RoXplorer® was subjected to a series of field trials. Firstly, in the controlled conditions of DET CRC’s Drilling Research and Training Facility at Brukunga in the Adelaide. Then at two locations with contrasting geological conditions, near Port Augusta in South Australia (Figure 1), and at Horsham in western Victoria. Whilst not available at the time of writing, the results of these trials will be presented in this talk.
ABSTRACT
The analytic basis of high-temperature, full bore directed energy opening in rock is presented including energy requirements, rates of penetration, high-temperature physics replacement of drilling mud, and diameter control. Millimetre-waves are ideally suited for long distance, high power guided transmission into typical borehole dimensions, but deep high pressure induced absorption will ultimately determine depth limits. The feasibility of this technology is established in the laboratory with granite and basalt specimens heated to temperatures in the 2,500-3,000 °C range using a low power 10 kW, 28 GHz gyrotron. A possible application to an enhanced geothermal system (EGS) is described using an engineered heat exchanger (EHE) with a potential useful lifetime of 100 years.

INTRODUCTION
Most, if not all studies of geothermal energy, such as that by Tester et al (2006), are done through the veil of existing drilling technology or small improvements thereof. Generally, they conclude that there is great potential for geothermal energy, but in fact one that is far short of the actual heat energy beneath our feet. Just 0.1% of the total heat content of planet Earth would be equivalent to the total world energy needs for about 20 million years (World Energy Council, 2013; US Energy Information Administration). This virtually limitless base power, green energy source cannot be practically accessed on large scales with current mechanical drilling technology or stimulated heat transfer reservoir approaches.

New game changing developments are needed. Such development is now possible with commercially available, powerful (1-2 MW) and efficient (>50%) millimetre-wave (MMW) (28-170 GHz) beam sources (Nusinovich et al, 2014) that have the potential to open self-cased boreholes in deep basement rock at faster penetration rates, greater depth limits, and at lower cost. Unlike infrared lasers, MMW gyrotrons are continuously more powerful, more efficient, and the wavelength of operation makes the beam energy immune to most losses suffered by shorter infrared waves in an industrial environment. The physics/technology advantage of MMW is so great that full borehole opening in crystalline rock can be reduced to just an energy-material interaction without any mechanical components in situ. All drilling functions performed with mechanical drilling systems: borehole stabilisation, sealing, and extraction can be performed with superior function by the high-temperatures and pressures generated locally to melt or vapourise the rock, circumventing the current drilling bottleneck to geothermal energy.

MELTING AND VAPORISING ROCK
Thermodynamics of Rock
The energy required to melt and vapourise rock can be expressed by the formula adapted from Maurer (1968):

\[ H = c_p(T) \cdot \Delta T + H_f + H_v \]  \hspace{1cm} [1]

where \( c_p(T) \) is the heat capacity at temperature \( T \), \( \Delta T \) is the increase in temperature, \( H_f \) is the latent heat of fusion if \( \Delta T \) includes the melting point, and \( H_v \) is the latent heat of vaporisation if \( \Delta T \) includes the vaporisation point. Granite and basalt basement rock, of interest to EGS, begin to melt at about 1,200 °C and are vapourised at somewhat over 3,000 °C at atmospheric pressure. The heat capacity of granite is plotted in Figure 1 with data from three references below 2,000 °C (Waples and Waples, 2004; Hellwege, 1982; Branscome, 2006) and extrapolated ad hoc to 3,000 °C. Waples and Waples (2004) show that the heat capacity of all rocks as a function of temperature is the same and only shifted in magnitude for different rock types. In their work the heat capacity of basalt is within 7% of granite.

\(^1\) Senior Research Engineer MIT, USA
Integrating Equation 1 over the plot in Figure 1 to 3,000 °C and adding the latent heat of fusion (0.9 kJ/cm³ for granite) results in about 12.5 kJ/cm³ that is necessary to heat granite up to the vaporisation temperature. The latent heat of vaporisation for rock is not well known, but from observations of stony meteor burn up in the atmosphere (Bornshten, 1983) it can be conservatively estimated to be about 14 kJ/cm³ for granite. It can take more energy to vaporise a rock than to just heat it to the vaporisation temperature. The electricity required, assuming 50% efficiency, would be about 7,000 kWh/m³ to melt and heat from 20 °C to 3,000 °C or about 15,000 kWh/m³ to fully vaporise near the surface. For a 10 km deep, 20 cm (8”) diameter hole the electricity would cost about 230 kSUS at $0.1 kWhr for heating to the melt temperature just below vaporisation. As a point of reference, a mechanically drilled hole to 10 km costs about 100 times more (Tester et al, 2006).

Rates of Penetration

The directed energy rate of penetration will depend on how quickly the energy can be delivered into the rock for a given method of extraction or displacement. It will be a function of beam power, borehole diameter, and required energy as expressed by:

$$ R_p = \frac{4}{\pi D^2} \left( \varepsilon_{mm} P_l - P_L \right) $$

where $D$ is the hole diameter, $H$ is the energy given by Equation 1, $\varepsilon_{mm}$ is the MMW emissivity (how well the beam is absorbed by the rock), $P_l$ is the beam power incident on the rock surface, and $P_L$ is the total power loss from the heated volume. Experiments have shown that the MMW emissivity of melted rock is about 0.7 and that the total losses, dominated by radiative heat transfer, are about 0.4 kW/cm² at ~3,000 °C (Woskov et al, 2014). Assuming all electromagnetic radiation is trapped in a deep borehole simplifies Equation 2 to $\varepsilon_{mm} = 1.0$ and $P_L = 0$.

A further assumption is necessary on whether the super-heated material is extracted or displaced to determine the appropriate value to use for $H$. As will be discussed below, very high local pressures that are generated, along with the compressible low melt viscosity, will make it possible to displace rather than to vaporise. Potential rates of penetration are plotted in Figure 2 under this assumption. The penetration rate scales linearly with power when $P_l > P_L$ and inversely with borehole diameter squared. Beam powers over 1 MW will be needed for rates > 5 m/hr in boreholes of 20 cm (8”) diameter or more. A 100 m/hr rate may be possible for small boreholes of 10 cm (4”) diameter or less with a 4 MW beam if the pressure is sufficient to suppress plasma breakdown.
REPLACING DRILLING MUD

High-Temperature Physics

All the functions of drilling mud: to stabilise the borehole, seal the wall from inflows (blowouts), and to extract or displace the rock, can be replaced by the physics of boring at high temperatures. The governing equation is the real gas law, which applies to gasses and super critical fluids. Its form in per mole of gas is given as Nordstrom and Munoz (1985):

\[ PV = ZRT \]  \hspace{1cm} [3]

where \( P \) is pressure, \( V \) is volume, \( Z \) is the compressibility factor, \( R \) is the gas constant, and \( T \) is absolute temperature. In a confined borehole volume if temperature is increased, the pressure will also increase independent of the starting local pressure. At high temperatures and pressures the compressibility factor also increases to a value greater than 1 (Nordstrom and Munoz, 1985), multiplying the pressure rise. For example, assuming a pressure gradient of 18 kPa/m (0.8 psi/ft.), a depth of 10 km (33,000 ft), and a factor of 10 increase in temperature, a pressure of up to 2 GPa (290,000 psi), about ten times ambient, could be produced. This would be sufficient to stress the local rock formation and displace the high-temperature melt. Rock melt viscosity decreases significantly with temperature, particularly for low silica basalts (Hobiger et al, 2011) and becomes compressible at high pressures (Sanloup, 2013) facilitating the displacement.

Collapse Strength

The displaced rock melt can form a strong dense glass casing. The volume of rock material inside a borehole before it is displaced is sufficient to make a diameter to wall thickness ratio of \( D/t = \sqrt{2} \), if all the melt could be moved to that distance without compression. Such a large casing wall thickness is outside the validity of pipe collapse strength formulas. Using 5% of Young' modulus to estimate collapse strength, which for glass is in the range of 50-90 GPa, would in principle allow an open borehole to depths all the way to the mantle at >30 km (100,000 ft.) to remain open after the pressure generated by the MMW beam is turned off. Another unique strengthening capability of full borehole directed energy opening is the possibility to shape and align the hole to remove ellipticity weakening in an asymmetric stress environment using an elliptical shaped beam.

CONTROLLING BOREHOLE DIAMETER

A metallic waveguide carries the MMW beam to some standoff distance from the target surface, which after launch into free space increases in size due to diffraction to make a borehole larger than the waveguide. The borehole in turn becomes a new waveguide analogous to a hollow fiber optic cable. The borehole diameter will be determined by the boundary of the MMW beam where the energy deposited into the wall equals the energy required to melt the rock (Equation 1 integrated through the heat of fusion). The
borehole diameter will increase until the wall deposited energy is less than the melting energy at which point it can no longer increase in diameter.

The power loading on the wall is determined by the borehole waveguide propagating losses, the backward reflection, and radiation from the hot melt surface. The attenuation constant for hollow dielectric waveguides is given by Marcatili and Schmeltzer (1964):

\[ \alpha = \left( \frac{X_{nm}}{2\pi} \right)^2 \frac{\lambda^2 (n^2+1)}{a^2 2\sqrt{n^2-1}} \]  

where \( X_{nm} \) is the guided HE\(_{nm} \) mode root, \( \lambda \) is the beam wavelength, \( a \) is the borehole radius, and \( n \) is the index of refraction of the wall. Note that the wall beam losses are inversely proportional to the cube power of the borehole diameter making the wall loading a strong function of beam size.

Assuming the forward and backward propagating beam power in the borehole is much larger than the radiative heat transfer contribution (Equation 7) to the wall, the borehole diameter can be estimated by:

\[ D \approx \frac{\alpha P_i (2-\varepsilon_{nm})}{\pi h R_p H_m} \]  

where \( h \) is the thickness of the wall layer into which heat is absorbed and \( H_m \) is the heat threshold for melting (about 4.5 kJ/cm\(^2\) for basalt and granite). The other terms are as previously defined. Values in Equations 2 and 5 need to be consistent so as to agree in diameter. For example, a 20 cm (8") borehole in basalt (\( n=2.6 \)) (Frasch et al, 1998) could be achieved with a 3.2 mm (95 GHz) wavelength beam of about 1.4 MW incident on a mode with a root (\( X_{nm} \)) of about 6.1 (~ 2% wall loss per m), a rate of penetration of about 9 m/hr (29.5 ft/hr), and a melt thickness of about 4 mm (consistent with laboratory observations). This example is very approximate to illustrate the connection between beam parameters and borehole diameter. In practice borehole diameter deviations from uniformity and surface roughness would cause higher propagation losses and variations in the rock composition/structure would cause variations in the index of refraction and absorption depth in the wall. Field experience will be needed to accurately establish the actual relations between borehole diameter and beam power, launched mode(s), and rate of propagation for specific rock types and saturation.

**TRANSMITTING MMW BEAM ENERGY**

**Near the Surface**

The most efficiently transmitted mode in a hollow waveguide is the lowest order hybrid mode, HE\(_{11} \), which is supported in hollow smooth dielectric and in metal waveguides having an internally corrugated surface. The required corrugations at MMW frequencies are small and can be fabricated with a tap (Nanni et al, 2012). When the fill medium of the waveguide is transparent such as low humidity surface air at frequencies of 95 GHz or 140 GHz or a pure nitrogen fill, the transmission losses in perfectly straight guide are primarily due to the wall surface resistivity. For optimised \( \frac{1}{4} \) wavelength deep corrugations in circular guide the loss factor is given by Nanni et al (2012):

\[ \alpha = \frac{R_s}{2Z_o} \left( \frac{2.405}{2\pi} \right)^2 \frac{\lambda^2}{a^2} \left( \frac{1-\frac{\lambda}{4p}}{(1-\frac{\lambda}{4p})^2} + 1 \right) \]  

where \( R_s \) is the surface resistivity, \( Z_o \) is the impedance of free space, \( p \) is the period of the corrugations, \( t \) is the corrugation wall thickness and the other parameters as defined previously.

The transmission \( (T = \exp(-aL)) \) of a 95 GHz beam as a function of distance \( (L) \) is plotted in Figure 3 for 127 mm (5") diameter copper and carbon steel waveguides. Near the surface there is the potential to transmit 20 km distances with 88% efficiency. This long distance guided efficiency in combination with multi megawatt power handling capability in typical borehole diameters is unique to the MMW range of the electromagnetic spectrum.
The insert in Figure 3 shows HE\textsubscript{11} transmission in a 200 mm (8") diameter smooth dielectric granite borehole (from Equation 4) after launch from the metallic waveguide. The beam could go about 100 m with 65\% efficiency guided by the borehole alone.

**Deep Transmission**

Gases transparent to MMWs at one atmosphere pressure become absorptive at high pressures due to molecular collision-induced absorption (Dagg et al, 1975). For example, at 95 GHz N\textsubscript{2} at 0.1 MPa (1 atm.) pressure has an 86.5\% transmission efficiency to 750 km, but decreases to about 4 m at 50 MPa (7,500 psi) and 200 °C. The most transparent nonpolar molecular gas, H\textsubscript{2}, at 0.1 MPa has the same transmission efficiency to 19,000 km and drops to about 70 m at 50 MPa. To reach deep subsurface distances it will be necessary to find a more transparent waveguide fill medium or operate the waveguide over most of its length at a pressure lower than that surrounding it.

The only apparent candidate for an alternative fill medium would be a pure noble gas. Argon, the 4th most abundant gas on earth, has been used as the zero-absorption calibration medium for the collision induced absorption studies of nonpolar molecular gases (Dagg et al, 1975). Unfortunately, there is no data for MMW transmission in pure supercritical noble fluids in the 0.1 to 10 GPa (1.45x10\textsuperscript{6} psi) pressure range that is of interest here.

A high-pressure window deep in the waveguide would not be practical. However, a dynamic pressure drop may be possible. The waveguide could be constricted for a short distance and a high velocity flow induced without significant beam loss to waveguide wall absorption. For example, a waveguide diameter reduction to 5.0 cm (2.0") in corrugated pipe for about 100 m length (0.99 transmission at 95 GHz) and a N\textsubscript{2} flow of about 0.17 m\textsuperscript{3}/s would produce a pressure drop of about 300 MPa (43,000 psi). Detailed analysis would be needed to determine the practicality of this approach.

The depth limit of full bore MMW directed energy penetration will ultimately be determined by the distance to which MMWs can be transmitted efficiently in a high-pressure environment.

**LABORATORY RESULTS**

The feasibility of melting and making holes in granite and basalt was demonstrated in the laboratory with a low power 10 kW, 28 GHz CPI HeatWave Model VIA-301 gyrotron. A custom transmission line system was built to convert the gyrotron beam from a circular, hollow azimuthally polarised beam (TE01) produced by the commercial system to an axially peaked linearly polarised beam (Woskov, 2014). Several waveguide launch ends were available. For the results shown here a smooth copper down taper to 20 mm (0.787") internal diameter was used to launch TE\textsubscript{11} (transverse electric) mode. Changing the waveguide launch antenna to achieve a different beam size, profile, and divergence is similar to changing a drill bit for a given job.
The transmission line included backward reflected power rejection to protect the gyrotron and a 137 GHz radiometer view superimposed onto the heating beam for real time temperature measurements. An air purge toward the target of up to 235 lpm (500 scfh) was also introduced into the transmission line to control plasma breakdown.

Rock specimens were exposed inside a steel test chamber that trapped all unabsorbed MMWs for safety and power balance measurements. The MMW beam waveguide was introduced vertically through the top of the test chamber with the specimen lying a short distance below the waveguide.

Both granite and basalt would melt within 2-3 minutes of beam exposure at about 2 kW incident power. Increasing incident power to 4.5 kW would typically raise the melt temperature into the 2,500-3,000 °C range in 5-10 minutes. Maximum temperature was limited by radiative heat loss as given by the Stefan Boltzmann equation:

\[
P_{\text{rad}} = \varepsilon_{IR} \sigma (T_{\text{hot}}^4 - T_{\text{cold}}^4) A
\]

where \(\varepsilon_{IR}\) is the infrared emissivity, \(\sigma = 5.67 \times 10^{-8} \, \text{W m}^{-2} \, \text{K}^{-4}\) is the Stefan Boltzmann constant, \(T_{\text{hot}}\) is the temperature of the melt, \(T_{\text{cold}}\) is the surrounding temperature, and \(A\) is the area of the melt. For the maximum temperatures and melt sizes achieved the radiated power equals the absorbed power for an infrared emissivity less than the MMW emissivity. MMW emissivity was measured to be \(\varepsilon_{\text{mm}} = 0.7 \pm 0.1\) for the rock melts at the highest temperatures.

Since it was not possible with this laboratory system to definitely reach the vaporisation point due to too low a power and too low a frequency (diffraction limited focusing), it was necessary to provide a leak path for the melt to make a borehole. Typically, a 12 mm (0.5") hole was drilled into the specimen for this purpose. Basalt melts flowed much better than granite melts.

Figure 4 shows a 50 mm diameter hole in a dense basalt specimen from Inner Mongolia (Coverall Stone, Inc., SeaTac, WA) with 4.5 kW TE\(_{11}\) incident power on the basalt located 35 mm from the waveguide. Due to beam diffraction of about 50° full angle after launch, the borehole diameter is larger than the outside of the waveguide, which was 42 mm (1.66") in this case. The MMW beam would have continued to diverge in free space after launch, but the borehole that was created confined and guided the beam.

Figure 4. 50 mm (2") diameter hole in 100 mm square, 30 mm thick basalt made with a 4.5 kW, 28 GHz beam (TE11) launched from a 20 mm diameter copper waveguide 35 mm away.
The very high temperature gradient (> 400 °C/cm) between the heated spot and the rest of the rock caused significant thermal fracturing as is evident in Figure 4. All the laboratory samples had to be constrained by steel bands around the periphery to keep them from breaking apart in these experiments. In a deep subsurface environment lithostatic forces would constrain the rock. Thermal stresses would add to pressure stresses to create fractures for melt displacement. Figure 5 shows a broken part of another basalt borehole revealing the melted wall to be about 4 mm thick without any pressure behind the melt, an indication of beam wall absorption depth used earlier here in beam controlled diameter calculation. This specimen was exposure to the MMW beam for 41 minutes, long after an equilibrium was established between the basalt and the MMW beam propagating through it. If the waveguide was advancing and not stationary the borehole would be smaller.

**Figure 5.** Cross section of a basalt borehole in a 38 mm (1.5”) thick specimen made with a 28 GHz 4.9 kW beam diverging at a full angle of about 50° launched from a 20 mm (0.787”) i. d. waveguide.

### APPLICATION TO EGS

The capability to open deep, self-cased sealed boreholes of moderate diameter in crystalline basement rock at lower cost creates the potential for an EHE approach to EGS. An ENE can be more predictably designed over a larger crustal volume and would have longer lifetimes at higher temperatures than a stimulated heat reservoir. The heat power that is mined by an EGS is governed by Fourier’s Law for thermal conduction:

$$P_{geo} = -k\nabla T A \quad \text{[8]}$$

where $k$ is the thermal conductivity of the rock, $\nabla T$ is the temperature gradient at the heat exchanger wall-rock boundary, and $A$ is the area through which the heat flows. The negative sign indicates that the heat is being extracted.

The thermal conductivity of hot dry rock (HDR) is low and decreases with depth, dominated by the temperature increase rather than the pressure increase. For granite, on average, it deceases from about 2.8 W/m/K near the surface to about 1.8 W/m/K at 20 km depth (Eppebaum et al, 2014). Strategies for ENE design need to maximise the temperature gradient and area since the conductivity is fixed by nature. The temperature gradient can be controlled by the volume rate flow of the heat exchanger fluid and the area by the size and number of boreholes.

Two-dimensional finite element heat transfer computations were carried out in a horizontal plane with parameters representative of granite at a depth of about 15 km (45,000 ft) for pairs of boreholes with various diameters, separations, and fluid fill temperatures. It was found that temperature gradients are larger for smaller holes. For a pair of 20 cm (8”) diameter boreholes separated 100 m (328 ft.) with a heat exchanger fluid flow maintained 25 °C cooler than the surrounding rock results in a temperature gradient starting from greater than 56 K/m in year 1 and decreasing to 37 K/m over a 100-year lifetime.

An area of 11.3 km² is required to achieve 100 MW heat power extraction with a temperature gradient of 44 K/m, corresponding to that after 15 years of operation. Such a surface area can be achieved with 106,
18.2 km (60,000 ft.) deep boreholes assuming the plane at 15 km depth is an average for the entire hole and discounting the top 1.5 km (5,000 ft.) length as too cold to contribute significantly.

The temperature at the bottom of these boreholes for an average crustal temperature gradient of 25 °C/km would be 470 °C. The heat capacity of water increases significantly near the supercritical point at 374 °C. Assuming an average heat capacity of 7 kJ/kg/K over the entire length of the borehole, a water flow of about 273 lpm (72 gpm) would be required to achieve an average temperature gradient of 44 K/m. In practice the temperature gradient would vary significantly as the heat capacity of the water increases with temperature. The resulting higher temperature gradients near the bottom of the boreholes would mean more of the heat power will be extracted from those depths.

One output well of larger diameter would be used for 30-35 injection wells to minimise heat losses on the way back up. A 30 cm (12”) return borehole per 35 input wells and a return fluid temperature of about 400 °C would introduce about 15% losses. This could be made up by additional injection boreholes.

An array of 106 boreholes on a 100 m grid separation would cover about 0.134 km² (33 acres) of surface area. The boreholes would be connected at the bottom by turning the directed energy beam horizontally at a 90° angle by a miter mirror bend at the waveguide launch aperture as is commonly done with MW gyrotron beams now. As discussed previously, the MMW beam could be guided by the basement rock borehole for distances of about 100 m.

The heat output would continually decrease as the rock around the boreholes cools. Figure 6 shows the temperature profiles in the granite with time. After 100 years, the rock temperature midway between the boreholes has dropped by 1.1%. The 100 MW heat output at year 15 would have been 126 MW after year 1 and 83 MW after year 100. Additional boreholes could be added as needed over time to maintain a desired power output and to extend the lifetime of the EGS site to beyond a century.

**DISCUSSION**

Intense energy will dominate matter and create a propagation path through it. Such continuous beam energy intensities are now commercially available in the MMW range of the electromagnetic spectrum at efficiencies to make full bore opening in basement rock practical. There is no need for mechanical components to interfere with the process. The beam launched from a waveguide brought to the proximity of the target rock diverges in size after launch to make a hole larger than the waveguide to allow it to advance. The diameter of the hole is determined, with knowledge of the properties of the rock formation, by controlling beam power, beam profile, and the rate of penetration. The high-temperature, high-pressure environment that is produced performs all the necessary borehole opening functions of stabilisation, sealing, extraction/displacement, and casing. The only consumable for this process is the electricity.
required to operate the high power MMW source. That cost could be as much as 100 times less than the current costs for mechanical drilling to depths of 10 km.

The resulting glass walled borehole makes an ideal high-temperature conduit for a heat transfer fluid that would be immune to chemical dissolution. Arrays of such holes to great depths of over 10 km and connected using the ability to turn a MMW beam optically at 90° to create an engineered heat exchanger has the potential to produce high-temperature, long lasting EGS power plants with potential lifetimes of a century or more. Additional studies will be needed to determine the optimum tradeoff between borehole number, diameter, and depth to achieve heat exchange areas of over 10 km² necessary for useful power plant sizes.

The cost could also be significantly lowered by engineering the heat exchanger in magma where it is accessible. Since magma is plastic it would take less energy to raise the temperature sufficiently to displace it out of the beam path and solidify the wall with a concurrent cooling gas flow. One could even imagine pursuing this approach to the mantle if it could be reached.

The depth limit of MMW directed energy boreholes is not known at this time. It will not be limited by subsurface temperatures or collapse strength during creation or completion. It will depend on how far a high power MMW beam can be efficiently transmitted into the deep high pressure environment. This will require either a MMW transparent supercritical fluid in the waveguide and/or a dynamic pressure drop to lower waveguide pressure over most of its propagation length. A recyclable noble gas, such as argon, seems to be the most likely candidate for the waveguide fill medium at this time. Waveguide diameter constrictions and high velocity gas/ supercritical fluid flow can produce high pressure drops to improve transmission. More studies will be needed to identify the best approach for propagating MMW beams into the pressure range of 0.1 – 10 GPa to determine the ultimate depth achievable.

The science is sound, the technology is largely available, and the outstanding engineering issues could be quickly resolved. The feasibility has been established in the laboratory. If successfully pursued into the field, MMW directed energy full borehole opening could have a greater impact on a shorter time scale than other approaches to expanding the viability of EGS power plants and could open new depths to geoscience research.

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DRILLING FOR STUDIES IN REMOTE LOCATIONS

MARK S. RYNHOUD

INTRODUCTION

A good ground model of the study area is critical to the successful design of any structure or mining project. The development of a ground model should always start with the geological mapping platform, which can inform both invasive (e.g. drilling) and non-invasive (e.g. geophysics) data collection. The development of a ground model using this approach is pretty standard, but can get complicated, and expensive, in remote environments.

Field work remoteness adds complexity to health and safety, logistics, communication, and resources, all of which can add significantly to the project schedule and budget. This can often lead to sharing of equipment for applications outside of normal use, for example using resource drill rigs to carry out shallow geotechnical drilling and testing. Exploration drilling crews experienced in deep resource drilling can find themselves carrying out geotechnical field tests and installing instrumentation in an attempt to manage costly task-specific drilling equipment and resources. Field staff also need to be converse with a broad range of field techniques to cover all aspects of the field campaign.

This paper expands on lessons learned from such scope-sharing field drilling programmes in remote locations, which do not always deliver on the schedule and cost saving intentions anticipated at the project outset.

FIELD WORK PLANNING

The success of a remote field work campaign will be dictated by the effort undertaken during the planning phase of the project. This planning effort should commence with a very clear understanding of the scope of work and what information is required to meet the project objectives. Remote field work often results in challenging immigration requirements. Staff resources need to be identified early and the appropriate visas and work permit applications submitted timeously with the relevant authorities. Once the preferred field staff have been identified, they need to make sure they carry out the necessary medical assessments to confirm a suitable level of fitness for the work involved, and to receive appropriate vaccination treatments for the region to be visited.

Once the resources have been identified the various drilling techniques and sampling procedures need to be confirmed to meet project objectives. The perfect solution may not always be possible because for example, preferred drill rigs may be too large for helicopter support in remote inaccessible areas. Information collection compromise may be required considering the practical constraints associated with the field campaign specifics.

A few common considerations to bear in mind during the planning phase of a field campaign include:

- Keep ground disturbance to a minimum (access, drill pads, helipads, geophysics).
- Local community participation (unskilled labour, security, protected sites).
- Extreme weather conditions (seasonal climate, down time, casualty evacuation).
- Sampling programme (testing at the rig, back at the core shed, offsite at the lab, quarantine procedures).
- Drilling team logistics (day shift, day-night shift, field crew movement).
- Occupational health and safety (remote assistance in case of emergency, communication constraints).

Once the broad planning decisions have been made and the potential constraints and hazards identified, the site investigation programme can be finalised and the specifics can be locked in. It is very important to establish clear lines of communication between the client, consultants and drill contractors with good document control procedures. Establishing a remote drill pad in the wrong place because of poor communication is a project shocker.

1 Senior Engineering Geologist, Klohn Crippen Berger
FIELD WORK CAMPAIGN

It’s generally a good idea to get the health and safety procedures sorted out early because if the field campaign cannot take place safely, it is never going to get off the ground. Remote field work often requires emergency medical crews’ involvement albeit on 24-hour standby where helicopter access is fairly easy, or as part of the field team overnighting in a field camp close to drill rig platforms. Casualty evacuation at night is often not achievable so patient stabilisation until daylight/chopper access is possible is often the preferred approach.

Helicopters are generally unable to service a rig site after dark, and cannot support a drilling programme under very low cloud cover. Contingency accommodation arrangements will be required for day-shift crews to remain at the drill sites. Emergency rations and good communication-support such as satellite phones are very important.

Drill sites need to take cognisance of large trees close to rigs and camp areas. Proximity to creeks is often a conundrum because the access to a good water source is important for the camp and the drill rig, but all field work activity needs to be aware of flood risks where high-intensity storms flash up very quickly. Bureau of Meteorology radars don’t work in the remote bush.

Drill core and soil sample collection needs to be very carefully managed to make sure good field data is gathered. As much data collection in the field as possible is a good idea because drill core can be disturbed during transportation. A well organised team should, however, be able to make sure core is carefully transported and packaged in a weather-proof core shed off-site to facilitate proper data collection. Be aware that high-tech electronic equipment can play-up in wet tropical conditions so robust, analogue equipment is often a preferred approach for field data collection. Avoid electronic equipment that requires batteries or regular recharging as much as possible.

LESSONS LEARNED

Over the years, the author has been involved in numerous remote, geotechnical field campaigns each with a different set of challenges that needed to be addressed to meet project objectives. Table 1 summarises some of the variables to consider for remote field campaigns.

Table 1. Variables to consider for remote field work campaigns.

<table>
<thead>
<tr>
<th>Variables</th>
<th>Comments</th>
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<tbody>
<tr>
<td>Site access</td>
<td>Light vehicle, quad bike, helicopter support, on-foot only</td>
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<td>Source of water</td>
<td>Drill rig requirements, hydrogeological testing, camp supply</td>
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<td>Communication</td>
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<tr>
<td>OHS</td>
<td>Evacuation procedures, overnight accommodation options</td>
</tr>
<tr>
<td>Local Community</td>
<td>Local source of labour, conflict resolution, no-go areas</td>
</tr>
<tr>
<td>Environment</td>
<td>Sensitive areas, sump management, water source sedimentation</td>
</tr>
<tr>
<td>Closure</td>
<td>Drill pad rehabilitation, local use (vegetables), instrument monitoring</td>
</tr>
</tbody>
</table>

Some field campaign variables are manageable and some are not. The manageable variables need to be carefully considered to provide the best outcome for your client. One of the key drivers is project schedule, which will often dictate whether the field campaign will encompass day-shift only, or day-shift and night-shift field work. Field work complexity is significantly influenced by the number of rigs drilling simultaneously and the day-shift/night-shift strategy. The quality of field data collection is almost always better when an engineer is stationed at each rig during the field campaign.

Experienced field personnel are essential for successful remote field work. There will often be no opportunity to call someone if you are unable to build up pressure in the packer bladders during hydrogeological testing, or to confirm the depths of vibrating wire piezometer tips prior to install. Over
and above field personnel technical skill, the ability to problem solve in the field and persevere during uncomfortable outdoor conditions is invaluable.

Client involvement to assist with community affairs issues is a variable that is often outside your control and any constraints imposed upon the field campaign need to be very carefully considered with input from all role players. Uncooperative or aggressive local communities can have catastrophic implications not only on drilling programmes, but ultimately on the longer-term opportunities to develop the project, post field-studies.

CONCLUSION

The planning of a field campaign is often common sense but it is critical to the success of any data collection field programme. If any of the variables are overlooked, or not considered thoroughly enough, it can have significant implications to the success of the field programme. It is hoped that this paper improves awareness and provides a useful reality-check during remote field work planning? These pretty basic planning principles can lead to huge embarrassment (and cost) if you get them wrong.
OVERVIEW

The purpose of the presentation is to unravel some of the key components required to utilise multiple intersection directional drilling on drilling programs. The objective being to deliver the given resource into the required JORC Code category at the lowest cost possible.

A brief history of directional drilling uses, techniques and tooling will be offered as a precursor to examples of where it has been utilised in typically hard rock hosted dipping ore bodies commonly found in Australia. A case study will detail technical and financial aspects of an actual multiple intersection directional project and then compare with alternate drilling options to achieve the same result.

MULTIPLE INTERSECTION DIRECTIONAL DRILLING

History

Directional drilling, which could be defined as “the use of drilling techniques to control the pathway of a hole with the purpose of hitting a specified target” has been used for many years across sectors such as oil & gas drilling, coal bed methane drilling, horizontal directional drilling and directional diamond core drilling.

Techniques

Across the different sectors using directional drilling, a range of techniques and tooling have been developed. The oil and gas industry has been at the forefront of these developments with other sectors adapting and developing to suit their niche requirements. The techniques most commonly used include:

- Downhole wedges – either permanent or retrievable options (or whipstock).
- Navi drills (or down hole motor).
- Steerable core barrels.
- Steerable rotary systems.
- Adjustment options of conventional bottom hole assemblies to alter direction of the drillhole.

Reasons to Consider Multiple Intersection Directional Drilling

Factors that can lead to benefit from using multiple intersection directional drilling can include:

- Logistical:
  - Environmental/cultural/access issues.
  - Underground workings.
  - Restricted underground access.
- Geological:
  - Ore variability.
  - Deep ore bodies.
  - Up-hole issues.
- Metallurgical.
- Financial i.e. multiple single holes vs one parent hole with single daughter.

1 DDH1 Drilling Pty Ltd
Survey

Integral to not only directional drilling but all drilling is adequate survey for drillhole tracking and placement. As the complexity of the drillhole path and multiple intersection components increase, more accurate survey and calculation is required. Methods used include:

- Eastman magnetic single shot.
- Electronic magnetic single shot.
- Electronic mems gyro (reference tool).
- Electronic north seeking gyro (memory wireline).
- Electronic north seeking gyro (live wireline).
- Measure while drilling (MWD).

Survey systems, and the use of, are a topic worthy of much more detail, but in the context of this presentation focus will maintain the importance of survey specifically when drilling multiple intersection directional drilling.

UNDERTAKING MULTIPLE INTERSECTION DIRECTIONAL DRILLING PROGRAMS

The decision to use directional drilling and/or bring it to a multiple intersection basis must start long before the drill bit hits the ground. Engagement between geologist and drilling contractor is to a level in another league to a “normal” drilling program.

Engagement

Initial contact would require a high level of information exchange between geologist and contractor. This would include at minimum – objective of the program, planned spacing of targets, geology to be expected, orientation of ore zone/s, available collar positions, target coordinates, historic drill reports for area, and historic survey results for area.

Evaluation and Proposal

Initial evaluation would focus on identifying suitable geology for directional drilling, underlying trends in survey data, production profile of ground and optimal intersection angles of ore body. From this a directional drilling model is constructed with assistance of directional drilling software or higher end geological software. The directional drilling model that is then presented to the client would be the optimised combination of new collars from surface and daughter holes to produce the most efficient drill program. This would be presented to the client in the form of proposed survey data, tables of meters to be drilled and technical overview.

Cost Estimation

Unlike more traditional drilling programs where estimation of time and cost are relatively easy to calculate by metres x $/m = cost, or metres/production = time, a directional drilling program is somewhat of an unknown on metres to drill, cost per metre and time to complete.

The drilling contractor will need to break down the program into metres to be drilled into RC/MR meters, coring of different sizes, nav drilling, wedging and accompany this with the expected rates of progress. This in turn needs to be aligned to the contract schedule of rates which will ultimately table the expected costs and program timing.

Execution

Effective execution of a multiple intersection directional program requires a very high level attention to detail from both drillers and geologist on the program. Geologist knowledge of the methods used and how/why they are employed is essential, which often means a lot of training/familiarisation at the beginning of such programs. The drilling is most likely going to be successful if parameters are set for target radius, preferred sampling intervals, etc. In unison as the program progresses, fine tuning of target position will be required due to actual intersection vs intended intersection as well as actual geology vs interpreted geology.
Case example: Multiple intersection vs single intersection

Comparison of actual multiple intersection directional drillhole compared with drilling single intersection holes from surface in a hard-rock dipping orebody typical to the goldfields of Western Australia. Focus of this is to highlight a lower program cost that has a lower cost per intersection but with an inflated baseline cost per metre.

Case example: Other factors

Examples are also presented of use of multiple intersection directional drilling where logistical, geological or metallurgical factors were a major factor.

Case example: Where multiple intersection directional is not appropriate

Example of a setting where the use of multiple intersection directional does not provide an advantage to the client and should not be proposed as best method.

CURRENT ADVANCEMENTS AND OPPORTUNITIES FOR MULTIPLE INTERSECTION DIRECTIONAL DRILLING

The mining sector through the downturn has predominantly retracted to extensions of existing mines still in production and generating free cashflow. This is against a backdrop in exploration drilling spend which traditionally focuses on finding new deposits. Hence there is proportionally more requirement for drilling to be concentrated around extensions of existing orebodies which are being mined deeper and deeper. The advantages of employing multiple intersection directional drilling grow exponentially with depth.

Recent advancements of tooling and the downturn in the oil & gas industry provide both a better-quality set of tools at a lower cost than ever experienced. This in turn provides a production benefit and/or a program cost benefit. A prime example of this is driller-operated north seeking gyro survey tools which provide a time and cost benefit over techniques previously used.
SUCCESSFULLY MANAGING DRILL PROGRAMS

SIMON SHAKESBY

INTRODUCTION

The results of drilling programs have significant effects on the companies involved. Poor-quality data and partially completed programs negatively impact the confidence stakeholders have in the operator and contractor. After choosing a drill contractor, rig type, drill method and staff, a drill program needs to run safely, deliver samples to specifications from the target zone and in a cost-effective manner within budget, time constraints and compliance parameters to be considered successful. Drilling often entails dealing with many unknowns and hostile terrain at the end of long, challenging supply lines. Managing the problems and deliverables successfully, requires a good client contractor relationship based on extensive planning, effective communication, ongoing monitoring and open collaboration. Successful completion of a program delivers confidence in the results to the stakeholders and allows informed, confident decisions to be made about a project’s future.

PLANNING

Planning begins with the client acknowledging it has a gap in its knowledge base that is best filled with data acquired via the drilling process. This can include the collection and analysis of samples, the in-hole measurement of geophysical, geotechnical or hydrogeological properties or the placing of instruments for ongoing data collection. Some programs may require the collection of more than one type of data from a hole.

It is important for the client to have a clear understanding of what they require from a program early on to ensure an accurate and concise contract is formulated and to avoid scope creep once a scope of work is finalised and a contract signed. The more advanced a program is, the more relevant this becomes. This is over and above requirements such as the number of metres and key performance indicators (KPIs) such as “x” metres per shift or “$x per metre. Regardless of cost or the rate of production, if a sample does not meet minimum specifications, was not collected or comes from the wrong place then the program has failed. Some examples of what a client may need for program to be considered a success are:

- Safe production - Normally applies to personal injury and environmental incidents to which there are many different KPIs that can be applied.
- Samples from the target zone – All drill holes have a target. How you assign the size of the target depends on the style of deposit you are testing for, the level of exploration you are undertaking (grassroots vs resource) or the kind of data you are collecting (hydro vs geophysical vs resource). It is important to be clear about this as the contract rates will reflect your accuracy requirements.
- Samples to specifications – This may relate to core size, sample weight, presentation style in a core tray, dry vs wet, style of core mark-up or a minimum recovery percentage.
- Successful downhole measurements/tool setup – A hole may be required to be drilled and maintained to allow the insertion of a tool to measure geophysical, geotechnical or hydrogeological properties and then have it removed. Another possibility is the permanent placement of instruments for ongoing monitoring of geotechnical or hydrological parameters.
- Accurate collar and hole trace locations.
- Compliance – During drilling and upon completion, drill holes have compliance parameters placed upon them by government or companies to ensure they are kept safe. This may include specific details about casing, capping, grouting, water flow management and sump overflow.

From a contractor’s perspective, their requirements for a program to be deemed successful are simpler than the clients. While they may want a technically successful program they only need at a minimum, safe production and a profit. A technical success in which the client is left happy but the contractor has lost money is not a success. Conversely a program that is not a technical success but allowed the contractor to make a profit may not have improved their professional reputation but does leave them able to continue doing business.

1 Geologist, Exploration and Advanced Project Management
Significant time and effort should be put into identifying the major risks to your program. This will require the input of numerous people with significant experience in their field and the area you will be operating. The aim is to be able to show a contractor a contract that identifies the major risks and put in place measures to eliminate them. Some examples of areas to cover are:

- **Personal safety** – This covers everything from accidents on a rig, health and hygiene to personal interactions with co-workers who may be from a different culture.
- **Environmental compliance & safety** – This covers items such as uncontrolled contaminated water releases, chemical spills and damage to flora and fauna.
- **Civil unrest** – Can be wide ranging and cover incidents ranging from country wide riots or a coup to localised unrest about employment opportunities or roadside holdups.
- **Camp standards** – Items such as electrical wiring, kitchen/food preparation, ablution and accommodation standards need to be covered here.
- **Compliance** – This relates to activities that are regulated by government and will bring your program to a stop if not adhered to. They will change from one jurisdiction to another. The following should be considered, land clearing/ground disturbance, water usage and disposal, rubbish and sewage disposal, camp size, employment requirements, work permit requirements, compensation, local community permission, casing, capping and grouting of holes.

This is a relatively simple but time consuming process of working through the program tasks in a stepwise fashion, identifying the key steps and associated risks, quantifying them and putting in place mitigation plans to eliminate or bring the risks to an acceptable and manageable level.

It is an iterative and ongoing process as things will change once for example, a rig or contractor is chosen as they may have their own requirements and standards.

Depending on where you are working, the community and how you and your contractor interact with them can have a significant impact on your activities. Generally, it is the environment and employment that are the community’s primary concerns. As the contractor is most likely to only have a short to medium term involvement in the project you need to ensure they start with, maintain and leave with community good will in place.

As the long-term manager of the project you are going to have to live with the repercussions of poor community engagement and so you should have in place guidelines and resources to help and guide the contractor through the employment and community engagement process.

Community engagement style and content changes if a project produces good results, and a company can see a longer-term future at the site. Initially the engagement is made and managed through the exploration team, often with minimal professional community affairs personnel engagement. The site team set the tone for future work and it is important to get it right. Early stage aims are relatively short term and are to gain access, test the target and either get out or then do more work. To achieve this, compromises may be made with the community around wages, numbers employed, employment standards etc. This can directly affect the operation of the drilling contractor. While compromise works in the short term, it is critical to keep a longer-term view in mind as success is always only one hole away. As a project progresses, a more structured approach to community support, business development and training is required so if a mine is developed, the majority of items have been negotiated and a stable structure established.

Some items to consider are:

- **Have a simple, clear project story and timeline.** This needs to lead from exploration to mining and explain that it is a long, stop, start process.
- **Try to set up clear rules and be consistent from the start.** Consider pay structure, communication lines, conflict resolution, charity requests etc.
- **What is the pay structure?** In some areas when dealing with groups who have little or no paid work experience, ensuring a uniform and systematic approach to pay between contractors and client will help avoid confusion for the local community.
- **How will the contractor liaise with the community?** This is most likely to be focused around employment but will cover such things as requests for donations, training programs etc. While the contractor will want to operate as an independent company and build its reputation in the district the
community liaison should be overseen by the client to ensure a consistent message that fits with the project aims delivered.

- How will the contractor employ people? This will vary from one jurisdiction to another. In some areas, long term contractor staff will travel with the equipment and do all work on site. In other areas, there may be government or community requirements to employ local people (possibly broken down into ratios based on tribal affiliation). All positions should have a job description coupled with age, educational, language and physical requirements. This is very helpful if the community is trying to dictate who will be employed. The contractor needs to retain the right to have the final say in who they employ.

- What training can the contractor provide? Many projects are in areas with limited opportunity of employment and training for the local community. Opportunities will vary on the size of the contract and the type of contractor but being able to show the community and employees that they will have improved their skill base at the end of the contract helps build enthusiasm for the project.

- How will the contractor resolve conflict with community members? This will normally be done as a collaborative process with community, contractor and client to ensure the long-term project goals are not compromised.

Drilling projects are often in remote and exotic locations at the end of long supply lines. It is important to understand what you need, when you need it and how you will get it there. If you don’t get it right your program will fail as and has been understood for eons. Alexander the Great noted “My logisticians are a humourless lot … they know if my campaign fails, they are the first ones I will slay”. As more rigs are used on a project the logistics of water supply, pad preparation, juggling rig capability vs hole depth etc. become more complex. Things to consider when planning the supply and logistics of a program are:

- Visit the project area and the supply route to get an idea of topography, road conditions, bridge widths and load limits, site storage options etc.
- Talk to local suppliers about what are their capabilities.
- Set up accounts early. In many jurisdictions building credit credibility can take time.
- Where is the water? What is the maximum head that will be pumped? How reliable is the water supply?
- What are the site gradients and rig capabilities on a slope?
- How big will pads have to be?
- Who is responsible for supply of what goods?
- How many helicopter or truck runs are required per week?
- What is the estimated production rate and hence the rate of consumption for consumables?
- How big an inventory is required to cater for the unexpected?

The adage that an army travels on its stomach can also be applied to the success of a drilling program. Hard physical work, temperature extremes and the common multicultural mixes of workers bring with them differing expectations which creates challenges when providing appropriate food and accommodation in remote locations. You may be required to choose the ethnicity of your workforce to suit the available food and accommodation standards.

The scope of work should be completed before the tender process starts as it provides the technical basis for a successful tender process. It is important that you have a reasonable idea of what a program may cost and good idea of your budget before putting the scope of work together as you do not want to put out a tender that will never be completed. It needs to outline and estimate the following type of items:

- The client’s needs as discussed above.
- Timeframe – The completion date of a program is driven by many differing demands such as end of fiscal year reporting dates, feasibility study completion deadlines or a simple desire from management to either upgrade a project or reject it in the quickest possible time. A realistic start date should be included as well.
- The type of equipment required – While some of this will be decided in negotiation with the contractor there may be some that client experience says is mandatory. This may be something like providing all tracked vehicles as opposed to wheeled or a requirement for a specialised grout mixer and pump.
- Size of program – this may be quantified as the number of metres, number of holes, number of measurements taken, number of intersections of an aquifer etc.
• Location of work.
• Key logistical considerations such as the quality of the road network, nearest port, need for helicopter support, facilities available in the nearest town, travel requirements etc.
• Camp facilities – a description of the types of rooms, recreational facilities, number of personnel per room, the type of meals.

The scope of work is the basis on which you plan a program and build your contract, its rates and KPIs around.

A contract is the reference manual for the business relationship between client and contractor. In its most basic form it needs to articulate what you are doing, where are you doing it, when you are starting and finishing it, how the work is broken down and charged for and the minimum safety expectations. Topics to cover include:

• The scope of work.
• A schedule of rates that needs to marry with the scope of work. Style will vary with some companies wanting it broken down into minute detail and others wanting a single flat rate for say metres drilled. Marrying it to the scope of work is important, as you can for example get the situation where work time on a contract that is simply drilling a hole and extracting a sample is different from a contract where a hole is drilled to both collect a sample and do downhole testing. In the former case the work time is controlled by the contractor to advance the hole while the latter is controlled by the client wanting to collect good-quality information.
• Clear definitions of the terms in schedule.
• What can and cannot be charged.

While poorly formulated items that relate to day-to-day activities can have incremental and detrimental, but usually manageable effects on either party, it is the poorly formulated or lack of items that address the premature program ending events that have major impacts on a company’s finances. Identify your major risks and take the time to ensure that they are covered in the contract.

A contract’s use will be almost always restricted to the field unless a major legal or financial issue arises. The language and structure needs to consider that it will be geologists and drillers using it not lawyers. The general hope is that lawyers are involved in writing it and then are not involved again. If they are your program has probably failed.

COMMUNICATION

Drill program management is coloured by the client and contractor relationship. Combative and acrimonious relationships are not conducive to optimising performance as ideas are unlikely to be shared easily and problems become harder to solve. A relatively structured and open approach to communication helps keep things organised and provides forums to openly identify, discuss and solve problems. Some suggested tools and ideas to use are:

• Contract alignment meeting – Often a contract will have been put together by a company’s procurement department and a drill company’s senior executives none of whom will be using it. Before work commences hold a meeting with the field crews responsible for undertaking and managing the program. Have them discuss the contract and work through the on-ground implications of the clauses and definitions, what is and is not chargeable, who the key personnel are, acknowledge the needs of each party, what the unknowns are and what may be missing and how you will deal with it.
• Agree to and sign the drill plods daily. This ensures any issues are fresh in your mind and the details easy to remember.
• Every hole should have a work plan formulated and agreed to by the client and the contractor. Programs comprised of many short holes may have a generic one. Your work plan should be completed, discussed and agreed to prior to drilling. It needs for example, to define the hole setup parameters, the target and its size, survey requirements and maximum deviation limits, key geological and structural features, hole size, casing depth targets, possible aquifers, and the end of hole requirements.
• Basic stock tracking system for key consumables. While the supply and logistics team will track the levels of consumables in detail it is good for the client and contractor management team to track the quantity of five to six key consumables daily against a minimum safe level.
- Have in place a clear handover system for fly-in fly-out (FIFO) situations and ensure the decision-making process is not compromised by a supervisor being off site.
- Many projects have a varied mix of cultures, languages and education levels that require a thorough and patient approach to all types of communication.
- Conflict between clients and contractors during a drill program is going to happen. How you resolve it and put in place actions to prevent it happening again will colour your future relationship and the efficiency of your operation. Strategies can include:
  - Discuss sensitive issues in a structured and documented environment. Morning meetings or a weekly meeting specifically for resolution of issues are good forums.
  - Document actions and assign timeframes for resolution.
  - Use a layered approach. In the first instance try to solve issues by engagement between the frontline supervisors. If this is not resolved quickly then it should be shelved and sent up the management chain. The aim here is to ensure that the client and contractor crews in the field who you want to be working together will continue to do so happily.
  - Structure your meetings so that issues that are raised are acknowledged, recorded, discussed and the resolution or actions and timeframes for completion are documented.
  - Acknowledge the issue and needs of the other party.
  - Work within the framework of the contract but I recommend retaining an open mind and be flexible about how you apply the contract due to the ability of drilling activities to throw up unforeseen problems.
- Sharing data and skills when problem solving will provide ownership to those undertaking the solutions. Collaboration brings groups together that have different resources, databases and skills. Dictating solutions to problems to a contractor will seldom get the optimal solution. Drilling programs are generally undertaken to collect geological, geotechnical and hydrogeological information. They all affect the drilling process and data from them can be fed back to the drillers to improve performance the second time. Build a model of your project and share it with the drillers. The whereabouts of faults, clay zones, zones of high water flow, hard ground etc. if known only improve the planning process.

**DOCUMENTATION AND OPTIMISATION**

A program that is well documented from the planning to the demobilisation phase will run more effectively and be easier to manage and improve than one that is not. Whether it be a well written contract that gives you the financial and legal certainty to proceed or a plod system that allows accurate and timely invoicing, systematic and accurate documentation of your work can pay big dividends over time.

A contract is the documentary base for the work you do and is a good reference tool when questions arise, for example about what is chargeable or what methods are expected to be used. Meeting minutes whether they be daily, weekly or otherwise are good references to what decisions have been made, what actions have been decided, who is responsible and when they are expected to be completed. Document the issues, plans and outcomes when solving complex problems. This allows an analysis of what did and did not work and provides a reference point should the problem arise again.

The management phrase “What gets measured gets managed” (derived from Lord Kelvin’s "If you cannot measure it, you cannot improve it.") is relevant to drilling because as technology becomes more complex and clients and contractors seek to continually optimise drilling activities useful detail can be lost if not captured and stored correctly. The capture and storage of drilling data in digital form is standard in the oil and gas industry and becoming more so elsewhere. On large programs and on sites that have long lives the information can be used to optimise the drilling process and reduce costs. The data collected varies from simply the rig ID, date, shift and the number of metres drilled to capturing all data on a plod from names of staff, to bit types and consumable volumes used. It is also possible on some rigs to capture digital data in real time on metrics such as bit weight, rotation speed, penetration rate and bit torque over time. When stored in a relational database coupled with the contract details, hole location and depth the information can be used to:

- Provide an invoice checking system that is particularly useful on large programs.
- Plot and track many different metrics over time to pick subtle trends that would otherwise go unnoticed.
• Compare drillers penetration rates between shifts and seek to understand why they differ.
• Compare bit on bottom time between drillers and shifts.
• Plot penetration rates in 3D to show where there is good and bad ground.
• Compare production from different rig types.
• Compare drill crew’s consumable usage between shifts.
• Plot consumable usage in 3D to show where usage is higher.
• Compare the effect of using a different consumable type or introducing a solids recovery unit has on production
• Store drilling knowledge. In an industry that experiences large boom and bust cycles, storing digital drilling data is one way of maintaining site and company knowledge about drilling details on a site.

Stored data is useless and expensive clutter unless used constructively. The main aims of analysing and utilising data are to optimise an ongoing or future program and to make planning and budgeting more accurate. Using the comparisons and images described above you can predict penetration rates and consumable usage for geological formations and structural zones in certain parts of a project, give advice on what rotation speed or bit weight may be optimal in a particular formation, predict how long a program may take to complete in a specific area, estimate costs of future programs and plot the progress of a current program.

Formulating and then implementing optimisation plans can be difficult as it involves change to the status quo. Some useful ideas on how to get it to work successfully are:

• Agree on a simple metric that will measure the improvement, for example metres per shift.
• Engage with site personnel to make a list of suggested improvements.
• Leave nothing off limits, from human resources to bit type.
• Assign each item a measure of improvement.
• Assign each measure an ease of implementation score. Keep this simple like easy, moderate or hard.
• Some improvements may give only a small improvement in performance. Consider using them cumulatively and/or over time to make implementing them worthwhile.
• Rank the items by best to worst improvement and then assess on ease of implementation.
• Choose an item and decide how to implement it, what time frame you will test it for and how to capture the data that will prove or disprove the improvement.

CONCLUSION

Planning, communication, data management and program optimisation are fundamental to good management process. When done well they develop and foster strong and stable, professional working relationships between the client and contractor so that when the inevitable problems arise there is a good base for discussion, debate, collaboration and agreement on how to solve the issue. That will inevitably lead to a successfully managed drilling program.

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RECONNAISSANCE DRILLING AND SAMPLING WITH THE ROXPLORER®: A NEW TOOL FOR ‘PROSPECTING DRILLING’

DAVID GILES

In the first half of 2017 the Deep Exploration Technologies Cooperative Research Centre (DET CRC) completed a series of field trials with its newly commissioned RoXplorer® coiled tubing (CT) drilling rig. The RoXplorer® has been designed for greenfields mineral exploration, with performance targets to drill at a cost of $50/m to a depth of 500 m and deliver a representative sample of the geology being drilled. If RoXplorer® is able to meet these targets it will provide a platform for low-cost, rapid, safe and environmentally-friendly drilling, and in turn, will enable a paradigm shift in mineral exploration under cover. A greater density of drillholes, combined with the ability to design and modify a drilling program on-the-fly, based on multiple streams of real-time data from the rig, permits a ‘prospecting drilling’ approach which systematically builds knowledge of the mineral system and vectors toward mineralisation.

CT drilling does not provide the option of wireline recovery of a core barrel nor (at present) is there a reverse circulation option for CT drilling. Maximum rate of penetration (ROP) is gained utilising bottom hole assemblies (BHAs) that produce cuttings with a range of particle sizes which are returned to surface within the drilling muds via the open hole. Key to the DET CRC’s CT drilling strategy is that the drill cuttings can be accurately assigned to a depth interval, are representative of the subsurface geology and amenable to near real-time Lab-at-Rig® analyses. The main challenges for sampling and real-time analysis for CT drilling in mineral exploration are thus:

1. **Fluid and sample loss:** In CT drilling with the RoXplorer®, the drill cuttings are returned to surface by the drilling fluid. Loss of circulation results in loss of sample. Mitigating against fluid loss requires an agile fluid management approach with the ability to call on a range of techniques depending on the situation – from minor tweaks of the fluid characteristics to significant interventions (e.g. casing the drill hole).

2. **Depth fidelity:** In order to accurately locate drill cuttings to their derived depth we require efficient and predictable upward transport of the cuttings within the drilling fluid. This requires monitoring and optimisation of fluid velocity (and flow regime), fluid characteristics (density and viscosity) and cuttings characteristics (particle size and density distribution). Inefficient up-hole transport can result in sample mixing which serves to blur geological boundaries and reduces the ability to resolve narrow intervals.

3. **Contamination:** The open hole nature of CT drilling with the RoXplorer® introduces the potential for contamination of the sample from the walls of the drillhole – either by collapse of the hole wall or erosion of the wall rock by the drilling fluid. This challenge is strongly dependent on the nature of the material being drilled and is most profound in unconsolidated cover materials. In addition there is potential for contamination of the sample from the drill bit and fluid additives.

4. **Representative subsample:** Once the drill cuttings have been delivered to surface, our challenge is to collect a representative sample from the drilling fluid and prepare it for analysis. This requires fit-for-purpose sampling equipment and methodology which is appropriate for the high volumes of fluid and particle size distribution generated by CT drilling.

5. **Robust, rapid and reliable real-time analyses:** At-site, real-time analysis that provides relevant geological data of sufficient quality to drive decision making processes is vital to the prospecting drilling strategy.

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1 Future Industries Institute, University of South Australia and Deep Exploration Technologies Cooperative Research Centre
DET CRC’s research agenda has been carefully planned to address these challenges in parallel, moving toward a working prototype of an integrated sampling and analytical platform for the RoXplorer®. The key elements of this platform include:

1. The ability to drill in a conventional top drive mode (in addition to CT drilling) and thus to run casing into the drillhole if required.

2. The design of new drilling fluids that are suitable for the CT drilling system and have resulted in dramatic reductions in fluid loss, improved cuttings return and minimal wall rock contamination when drilling broken or unconsolidated ground.

3. Detailed understanding of the cuttings produced by CT drilling has allowed optimisation of the fluid velocity and characteristics to ensure efficient up-hole transport.

4. Routine collection of drill chips for visual inspection provides a means of identifying contamination and, in combination with real-time analyses of the drill cuttings, provides the potential to quantify the extent of contamination.

5. The design of new sampling devices and in-field sample preparation procedures to enable consistent, objective and unbiased batch sub-sampling at intervals representing <1 m of subsurface geology.

6. Protocols to present the sample to the Lab-at-Rig® analytical platform, previously developed by DET CRC for application to diamond drilling and licensed to Imdex Limited.

Our recent trials at Brukunga (in the Adelaide Hills), near Port Augusta and at Horsham have provided the opportunity to test and tweak many aspects of the drilling and sampling workflow under a range of drilling conditions. These conditions simulate the operational environments in which the RoXplorer® will be required to perform if it is to reach its potential as a reconnaissance drilling tool. In this talk I will provide a summary of the outcomes of the drilling trials and use our (growing) knowledge of the operating performance of the RoXplorer® and its attendant sampling technologies to foreshadow how it might be rolled out to facilitate a regional ‘prospecting drilling’ exploration strategy.
THE MEASUREMENT OF PERMEABILITY AND OTHER GROUND FLUID PARAMETERS

IAN GRAY

ABSTRACT

Different branches of engineering tend to adhere to historic practices that are used by their discipline alone, and tend to ignore developments from other groups.

In the case of permeability measurement, civil and mining engineers tend to stick to packer testing and the determination of an index called a Lugeon value. This bears little relationship to a real value of permeability. In soils, there is a tendency to use falling head tests. These also have problems associated with their analysis, which generally renders them invalid.

The hydrogeologist wants large scale well tests that are frequently of long duration and suitable for productive aquifers. The analysis of these is usually based on drawdown.

The petroleum reservoir engineer wants a quick test down a deep hole. However, because of the dollars involved in these tests, a lot of thought has been put into their analysis. This enables the separation of the real permeability of the ground from damage to the well bore area brought about by drilling. The key to this success is in ending the test with a zero-flow period where the test zone is closed in.

Sigra has operated in all of these disciplines, and has developed equipment and techniques that are as simple as a packer test to conduct, but yield all the advantages of tests conducted by the petroleum industry. These are a true assessment of formation permeability, a measurement of well bore damage, the ground fluid pressure and an indication of the mean effective radius of the investigation.

What single borehole testing cannot do is provide information on the storage characteristics or the anisotropic nature of permeability. To achieve this, other pressure sensing points (piezometers) must be installed. This brings the testing process full circle, to one where a full pumping test might be used with at least three correctly placed piezometers to enable the determination of anisotropy in permeability, and the storage behaviour. The problem with this approach is that it is not possible to differentiate between anisotropy and inhomogeneity.

There is an alternative that enables the measurement of both inhomogeneity, anisotropy and storage parameters. This is the pulsed drill stem test (DST) approach. It involves sequentially testing individual boreholes and placing piezometers in these as each borehole test is finished. The next borehole to be tested sends a pressure transient to those boreholes drilled before and fitted with piezometers. The most convenient way to test each well is by conducting a DST in each of the test zones. Hence the name “pulsed DST”.

This method enables multiple measurements of mean permeability and directional permeability, so that both inhomogeneity and anisotropy may be assessed. This provides comprehensive information on the groundwater/liquid regime that is generally ignored by mining and civil engineers.

A lot of poor quality measurements with intrinsic flaws do not make up for a few good ones!

THE REASONS FOR MEASURING PERMEABILITY AND STORAGE PARAMETERS

The reasons for measuring groundwater parameters include:

- Slope groundwater behaviour – for stability.
- Dewatering needs:
  - quantity of fluid.
  - magnitude and extent of drawdown.
- Settlement due to dewatering.
- Containment and removal of contaminants.
- Water supply.

1 Sigra Pty Ltd
The prime reason for measuring parameters in the world of petroleum fluids is to delineate a reserve and deduce a production scheme.

**THE NEED TO UNDERSTAND THE GEOLOGY**

No matter what technique is being applied to measure ground fluid behaviour, an understanding of the geology of the area is essential so as to be able to interpret the results. Having a measurement in unidentified strata means little, and certainly precludes the extrapolation of this measurement elsewhere. Preferably all data available should be used to identify the geology. This might include core logs, geophysical logs, geochemistry and seismic survey information. It may also include well test data itself, as this can be used to identify the rock type and such features as reflections from barrier boundaries or the recharge zones.

**TESTING PRACTICE**

Each engineering discipline has developed its own way of measuring permeability and storage behaviour, and we do feel that these practices have often proved inadequate, specifically in reference to civil and mining projects. These could be improved with some understanding of the process and the use of suitable technology. Let us examine these practices.

**The Hydrogeologist**

The hydrogeologist who requires groundwater will choose to drill a well or wells into a suitable aquifer, and will pump from this to determine the production rate versus pressure (head) depletion in the well. If economics permit, they have observation wells and will then observe pressure (head) changes in these so that the storage behaviour of the aquifer may be deduced, but without them this is not possible.

Long term pumping tests of wells and the response in adjacent wells is a very good way to determine aquifer characteristics. Hydrogeologists will interpret their measurements in terms of hydraulic conductivity, which is a combination of absolute permeability of the ground and the viscosity of water.

They will describe the storage behaviour in terms of yield of fluid per unit area per unit fluid head change. The storage term used for a confined aquifer is storativity, and for the unconfined case, specific yield.

**The Petroleum Reservoir Engineer**

The petroleum engineer will drill a well which is usually deep and expensive. They wish to test the formation (ground) as quickly as possible because drill rig time is real money. They seldom use pressure measurements in adjacent wells to determine storage parameters, due to the prohibitive costs of such wells for pressure sensing.

Because of the variation in fluids with location and time, the petroleum engineer focuses on absolute permeability and in determining the fluid parameters, such as viscosity and density, separately. The determination of storage behaviour is as much as possible determined by laboratory studies of core. The key words regarding storage are porosity and compressibility (change in porosity) of the rock, and the compressibility of the fluids. They are a function of fluid pressure.

The hydrogeologist and the petroleum reservoir engineer both want to know permeability and storage. In addition, the hydrogeologist will probably want to know something about the recharge mechanism of groundwater. Any field testing undertaken by either of these disciplines will focus on examining transient behaviour.

The hydrogeologist will usually want to pump at a constant rate until the transient behaviour of the well can be defined, while the petroleum engineer will frequently use a DST to produce fluid for a short period, then shut in the production zone and wait until the transient recovery behaviour is well established.

Both disciplines will want to obtain information on their reservoir or aquifer away from the test well. This is accomplished by the use of pressure measurement in surrounding wells, or by suitably designed test methods and analysis.
The Civil and Mining Engineer

The civil engineer, and to some degree the mining engineer, want to know whether water will be a problem for whatever structure they are designing and building. Their concern then is likely to be water make into an excavation, tunnel or mine, or water loss from a dam or through an embankment.

Very frequently they wish to know what the pressure distribution is within the ground, as it directly affects the effective stress, and therefore the potential for failure.

Sometimes the civil or mining engineer will employ a full pumping test with associated pressure observation by piezometers. These cases are however unusual. Time and cost pressures have tended to lead to a series of short term tests that have been historically used. In soils, these are typically falling head or slug tests, in which a hole is filled with water and the rate of change of head and hence volume change within the hole, is monitored for a period.

In rock, the test method is typically the packer test, in which a section of hole is sealed, and water is pumped in at a fixed pressure of one atmosphere as measured at surface, and the rate of inflow is monitored. The final supposedly steady state (10 minute) flow rate is measured in Lugeons (litres/metre/minute), a value that was developed originally to simply determine whether the ground would take cement grout. Neither of these tests can be analysed to produce real values of permeability, and by definition single hole tests cannot provide any information on the storage behaviour of the ground.

Problems with a Falling Head Test

The falling head test produces a varying rate inflow. This causes a problem in separating pressure loss around the well bore, usually associated with drilling, from the response of the soil outside the zone of influence of the well. This problem is made much worse because the process of injection almost invariably carries soil particles into the zone around the well bore, thus changing the near well bore behaviour. This means that it is not generally possible to separate near-well bore pressure (head) loss from the transient response in the ground.

The results of such tests are therefore misleading—if the test is left for long enough to come to stabilisation it can yield information on the groundwater fluid level, and very little else.

Problems with a Packer Test

The civil engineering industry has used packer tests for many years. The basic technique was put forward by Maurice Lugeon (1933). In this, a section of borehole is straddled by packers and water is injected with 1 atmosphere pressure at surface. The hydraulic conductivity is expressed in terms of the Lugeon value, which is empirically defined as the conditions required to achieve a flow rate of 1 litre per minute per metre of test interval under a reference water pressure equal to 1 MPa. If the flow rate is higher the Lugeon value is proportionately higher. The inflow period is normally 10 minutes. This sort of test takes no account of the initial fluid potential (level) in the hole. It also assumes that the flow rapidly becomes steady state and it takes no account of near-well bore effects. These three assumptions mean that the test results are not interpretable in terms of real values of permeability. The method has some use in empirically defining the degree of fracturing in the rock mass being tested, but should not be interpreted beyond this.

The methodology of the packer test is that it should reach a steady state of fluid injection. If in fact it does so, it is an indication that the pressure drop between the rock and the fluid pressure within the hole are dominated by near well bore losses, typically by the size of the joint openings to and adjacent to the borehole. The real information from such tests on the rock mass being tested is lost, because no attempt is made to determine the transient response of the ground. Neither does the test provide information on the fluid pressure (head) within the ground, nor take this into account in how it affects the inflow rate.

Literature abounds on how to interpret such tests, and spurious correlations are published between the value of Lugeons and units of hydraulic conductivity, and by consequence, permeability.

We therefore have two tests that are widely used by the civil engineering industry which cannot provide the information that is required. Indeed, the results obtained are misleading, and their adoption could lead to serious errors in design. What can be done to remedy this?

The short answer is to change test methods.
IMPROVEMENTS TO CURRENT PRACTICE

The most effective way to improve matters is to adopt the analogue of the oilfield DST for civil and mining applications. Sigra has used these extensively for coal seam gas and coal mining clients. We developed equipment and analysis to suit these applications.

The test needs a period of flow followed by zero flow from the test zone, during which pressure stabilisation is achieved. This is followed by an inflow period and then a period of no flow from the test zone, during which the pressure build-up is monitored. This build-up time must be long enough to get a meaningful answer. By focusing measuring on a period without flow, it is possible to remove the effects of pressure drop through a damaged near-well bore area.

Low permeability ground tends to take a lot of time for the pressure recovery to deliver results with a useful measure of permeability. If there is no need to measure permeability down to low levels, then the test may be terminated early without providing a precise value.

While flow from the test zone is the best choice, as it avoids contamination of the well bore with foreign fluids and clay particles, it is sometimes more practical to inject for a period at a constant rate, or by falling head, in the drill string. Changing the flow direction does not change the basis of analysis though changing flow due to particle blockage on the well bore may complicate or invalidate the test.

This test approach may be used in rock or soil.

DST in Rock

The test equipment for conducting a DST test in rock may take several forms. One used by Sigra is its system which enables it to test though an HQ core system. The operation of this is shown in Figure 1, with sequencing described below.

1. The wireline DST tool is lowered through the HQ drill string.
2. The DST tool is shown landed and locked into the core barrel:
   a. A head seal is placed at the top of the drill pipe.
   b. Compressed air is used to push down the water level in the drill string.
3. The packers are inflated:
   a. Compressed air is bled off.
   b. The test zone is allowed to come to equilibrium.
4. The valve is opened so inflow can take place.
5. The valve is closed so that a head build up can take place.
6. The packers are deflated and the tool can be pulled out of the hole.

Figure 2 shows an example of the inflows and pressures during a DST test. This is a test in a particularly low permeability coal seam. As a consequence, there is very little inflow in either of the two inflow periods, and the recovery times after shut in are very long. However, this permitted the detection of very low permeability. What is also apparent is the very rapid rise in pressure in the test zone after the closure of the valve, indicating a very high well bore damage (skin).

Figure 3 shows the derivative of pressure with respect to Agarwal time in this test. The flat area indicates stabilised conditions, when the build-up may be used for permeability analysis. This period is the straight portion of the Horner build up plot shown in Figure 4.
The Measurement of Permeability and Other Ground Fluid Parameters

Figure 1. Operation of Sigra’s through-the-string DST tool.

Figure 2. Pressure-time trace for a very long time interval DST. The red trace shows the drill pipe pressure; the blue trace shows the test zone pressure while the pink trace shows the packer pressure. Depending on permeability and storage characteristics of the ground most tests take a few hours.
Equation 1 describes the Horner build up plot which has a slope $m$.

$$p \text{ versus } \ln \left( \frac{T + \Delta t}{\Delta t} \right)$$  \[1\]

Where: $T$ is the flowing time, $\Delta t$ is the time after shut in.

Equation 2 is used to derive permeability from the Horner Plot.

$$k = 9.21 \cdot 10^{-10} \frac{q \mu}{mD}$$  \[2\]
The Measurement of Permeability and Other Ground Fluid Parameters

I. Gray

Where: $k$ is the permeability in m$^2$.
$q$ is the flow in m$^3$/day.
$m$ is the slope of the Horner plot in kPa per natural log cycle.
$D$ is the thickness of the unit being tested.
$\mu$ is the viscosity (N·s/m$^2$).

In addition to permeability, the reservoir pressure can be determined from the Horner plot. It is obtained from the extrapolation of the pressure to the zero value of $\ln \left( \frac{T + \Delta t}{\Delta t} \right)$ at the left-hand side of the plot in Figure 3. It is also possible to derive a value of the extent of the test. Sigra uses the term mean effective radius of investigation. It is important, because too small a zone of investigation may mean that the value of permeability obtained has little relevance. This is particularly the case where the permeability is very low and the well bore damage high. The well bore damage describes the degree to which the near well bore permeability deviates from the general reservoir permeability. The term ‘skin’ is used to define well bore damage. It is given in Equation 3. It may be related to an effective well bore radius through Equation 4.

$$ P_{\text{Loss}} = \frac{\mu q S_k}{2 \pi k D} \quad [3] $$

Where: $P_{\text{Loss}}$ is the wellbore pressure loss.
$S_k$ is the skin term.
$q$ is the flow rate.
$u$ is the fluid viscosity.
$k$ is the permeability in consistent units.
$D$ is the test zone thickness.

$$ r_{\text{we}} = r_w e^{-S_k} \quad [4] $$

Where: $r_{\text{we}}$ is the effective well bore radius.
$r_w$ is the nominal well bore radius.

The value of $S_k$ can be calculated by examining the difference in well pressure at the end of the flow period from the calculated reservoir pressure. Using the well equation linearised by a log approximation, this takes the form of Equation 5.

$$ S_k = \frac{1}{2} \left( \frac{P_{\text{diff}}}{m} + 1.722 - \ln \left( \frac{Q}{r_w^2 \phi c m} \right) \right) \quad [5] $$

Where $P_{\text{diff}}$ is the pressure difference between the flowing well and the reservoir pressure.
$m$ is the slope of the Horner plot in units of pressure per natural log cycle.
$Q$ is the cumulative flow over the flow period (assuming uniform flow rate).
$\phi$ is the product of porosity and total compressibility.
$r_w$ is the wellbore radius.
$D$ is the test zone thickness.

In solving Equation 5 an assumption must be made as to the value of the compressibility porosity product, as this cannot be determined from a single test alone.

In addition to determining the value of permeability alone, it is sometimes possible to detect the presence of sealing faults or zones of recharge, by deviations from the normal build-up behaviour.

**DST in Soil**

It is possible to also use a form of DST test in soils. This is designed to replace the slug test. It involves setting a standpipe and filter in a test formation. The preference here is to have a developed test zone over an entire soil type interval rather than at a point. This forces radial flow. The procedure involves the use of a packer fitted with a pressure transducer and a data logger.

The procedure is then to:

1. Set a packer in the standpipe below the water level.
2. Wait for pressure stabilisation.
3. Fill the standpipe above the packer.
4. Deflate the packer and monitor the water outflow.
5. If necessary add water to the standpipe at a known rate to maintain flow.
6. Inflate the packer and monitor pressure stabilisation.

In the event of a high groundwater level, water must be pumped. This can be achieved by air lifting through the packer, bailing or using a pump. At the end of production the packer must be sealed within the standpipe to enable recovery.

The use of the packer acts as a valve in the standpipe, removing problems with well bore storage as the water level changes in it. This is a very important feature of the process. The analysis is similar to those described for rock.

**DST Test Results: Summary**

A DST can be used to provide information on:

- Permeability.
- Groundwater head.
- A radius of investigation for the test – how much ground is tested.
- Well bore loss behaviour, which may be expressed in terms of an effective well bore diameter – to determine to what extent the test has been dominated by well bore issues.
- An indication of the features of the tested rock, such as boundaries and fractures.

What it cannot do is provide information on the storage characteristics or the anisotropic nature of permeability.

**Pumping Tests**

Pumping tests involve pumping for a period from a well. Most analysis is undertaken on the basis of constant flow, and deviations from this practice cause complications in the determination of reservoir versus well bore loss behaviour. The exception is a test in which the rate is deliberately increased in steps. The purpose of such a test is to determine the non-linear well bore loss characteristic of the well.

While some pumping tests are conducted in a single well without monitoring piezometers, information may be gained on storage terms (porosity, compressibility product or storativity and specific yield in hydrogeological terminology) by placing piezometers around the pumping well. This constitutes an interference test.

If information is sought on the anisotropy of the ground then three piezometers are the minimum that may be used. These piezometers should not be placed diametrically opposite each other.

The problem with this approach is that it is not possible to differentiate between anisotropy and inhomogeneity. The characteristic of the test is dominated by the single pumping well, and it is always possible to determine a precise solution for anisotropic permeability from the three piezometers. In fact the apparent anisotropy may simply be an artefact of inadequate sampling points in an inhomogeneous formation.

**Pulsed DST Testing**

By increasing the number of locations from which fluid is withdrawn, it is possible to get more information on the mean permeability. If each of these wells is then fitted with a piezometer, it is then possible to obtain a directional permeability from the subsequent test well to the piezometer. This approach has the advantage that the test programme may be developed as each test is undertaken. This overcomes the normal problem with an interference test, of trying to determine what spacing to place piezometers from the pumping well before the testing has been undertaken. The use of a DST test for fluid withdrawal is convenient, because it may be conducted in slim core holes without the complications of setting up a pumping well.

A hypothetical well layout is shown in Figure 5. This shows five test wells drilled and tested sequentially, and then each fitted with a piezometer. These produce five measurements of mean permeability and ten measurements of directional permeability. The five wells do not necessarily need to be drilled and tested. The process could be terminated sooner or extended, depending on how much information is required. The spacing of wells is dependent on permeability and storage parameters and is determined as the testing progresses. Figure 6 shows a characteristic DST test in one of the wells while Figure 7 shows the pulse.
The measurement of permeability and other ground fluid parameters received in an adjacent piezometer. This also shows the best fitted curve between well theory and the experimental data. The fit is very good and provides directional permeability and storage information.

*Figure 5. Hypothetical well layout for pulsed DST testing.*

*Figure 6. A DST test used as part of pulse testing.*
The Measurement of Permeability and Other Ground Fluid Parameters

I. Gray

Figure 7. A pressure trace from a piezometer caused by a two inflow DST test.

Pulsed DST testing provides information on:

- Permeability.
- Storage, compressibility-porosity product, storativity or specific yield.
- Anisotropy.
- Inhomogeneity.
- Well characteristics, namely well bore loss, effective well bore diameter.
- Boundaries to the system, such as recharge or barriers.
- Fractures.

PIEZOMETERS

There is a need to measure fluid pressure in a formation either for testing purposes or simply for long term monitoring. This is more complex than it may first appear, and there are many traps in achieving a good outcome. The keys to good pressure sensing are certainty of what is being measured, and reliability. Certainty can only be achieved if the system can be tested. Reliability comes from good equipment and good installation practice.

The options to install pressure sensors include removable systems and those that are permanently installed. The removable systems are usually packer based and used for installation inside casing that has been perforated to connect it with the formation. While it may be considered that the removable systems are superior because they can be serviced, the cost of doing so is considerable as it requires removing the packer and transducer system from a live well that needs to be kept under control. The cost of this may easily exceed that of the initial installation.

The permanent transducer installations usually involve cementing transducers into the well. As such they rely on communication between the pressure sensing diaphragm of the pressure transducer and the formation to be monitored. To establish this communication the cement used must have some permeability and yet not enough permeability to enable pressure connection between the various formations intersected by the well. These are conflicting requirements.

The process of cementing transducers into a well is further complicated by the filtration of the cement grout mixture into the formation under hydrostatic pressure within the well. This leaves a very dense impermeable mixture next to the formation to be monitored. The consequence of this is that the transducer response may be slowed by some months (Neels and Gray, 2014).
The solution to these problems was the development of cement grouted installations with the displacement of cement from around the filter of the transducer and the borehole wall. It enables the use of low permeability cement grouts that avoid problems with intra-well connection between transducers and permits the transducers to operate in an installation that is independent from the cement grout-formation interaction. It also provides a means by which intra-well connection can be tested and where communication with the formation to be monitored may also be tested.

This type of installation involves the placement of permanent pressure transducers in a borehole. In this application, a cement grout pipe is fitted with pressure transducers, cables and cement displacement lines and then lowered into the borehole. By preference, this is undertaken inside a casing or wireline drill pipe (in this case Boart Longyear HQ pipe of 89 mm OD and 78 mm ID).

This drill pipe is then withdrawn over the grout pipe assembly. The assembly is picked up from what will be a helically buckled form in the hole and hung from the hole collar so as to locate the sensors at the predetermited positions. This is the situation shown schematically in Figure 8. Here a single transducer is shown hanging in the well with a cable attached. The transducer is connected to a fitting to which a filter is attached. The filter separates the transducer diaphragm from borehole. A pressure pressure relief valve is also attached to the fitting. This pressure relief valve is designed to support the water column above it contained in an injection tube. This injection tube is either nylon or stainless steel depending on the depth and application.

Figure 8. The installation sequence of a pressure transducer by cement displacement. A shows the operation of pumping water through the injection tubing to clear the filter. B shows the hole being cement grouted. C shows the filter being cleaned with a small quantity of water. D shows the cement grout being displaced by water injection when it has reached a plastic state.

The actual installation process involves installing transducers on tubing containing pockets to protect them. This tubing is of steel, or where mining may take place, it has been replaced by fibreglass. In shallow holes it may be plastic. The transducers are connected by cable to the surface. Each transducer diaphragm is connected to a common chamber with a filter and a pressure relief valve, that is in turn connected to the surface by the injection tubing.
Case Study of Piezometer Installation

The case study involved a coal seam bearing sub basin in Queensland. A density log of this is shown in Figure 9 for borehole 1 of the group of holes drilled. It shows an upper coal seam (1) at 292 to 297 m and then a group of seams from 356 to 378 m depth. The latter group comprises a main seam (2) from 356 m to 361 m which is separated from seam 3 by a tuff band and then a sequence of interbedded seams and shales. In the latter, seams (4) and (5) are identified. Pressure transducers were installed in seams 1 to 5 by the method described above. Their locations are marked in Figure 9.

Figure 9. Location of piezometers in coal seams in Borehole 1.
Five transducers were installed in the hole as closely as two metres apart. Injection testing through the capillary tubing following the setting of the cement grout, showed no interconnection within the borehole. Decay of pressure took place, indicating connection to the coal seams being monitored. Figure 10 shows the results of pressurisation and pressure decay during this test period. The large pressure spikes and their subsequent decay are caused by pumping into each filter zone. The small responses in adjacent sensors are due to the expansion of the nylon injection tubing in the adjacent filter zone.

Figure 10. Water injection into each seam level and the associated pressure spike and decay.

The transducers used were vibrating wire devices that are generally used in civil and mining applications. They are extremely stable over very long periods and produce a frequency output that can be read to 1/1000th of a Hertz in a signal lying between 2,000 and 3,000 Hz bandwidth. The theoretical sensitivity is therefore 1 in 10^6. Practically the sensitivity achieved was within a bandwidth of 0.04 kPa (0.006 psi) in a 10,000 kPa transducer, corresponding to 1 in 250,000. The accuracy of these devices is really dependent on the quality of calibration. With careful calibration this can reasonably be within 0.05% of full scale. As they are a fraction of the cost of quartz sensors and provide a digital (frequency) signal, they are a good, cost effective transducer for reservoir monitoring.

This case was for an installation between 290 to 380 m depth. Others have been completed without complications to 600 m and no major problems are foreseen in using the system to greater depths. In deeper installations the nylon can be replaced by stainless steel capillary tubing. The system is subject to patent applications.

CONCLUSION

This paper examines typical practice to determine groundwater parameters in the civil engineering and mining environment. It compares this with hydrogeological and petroleum engineering practice. The packer tests, falling head or slug tests used in the civil and mining industries are found to deliver incorrect information about permeability due to the test processes and their analysis. The principal problem with
these is that they do not deal with well bore damage nor do they properly analyse the transient pressure response from which permeability information may be derived.

The use of a modified petroleum industry DST is shown to overcome most of the problems associated with these tests, without undue extra complexity. The results are tests which provide a real permeability as opposed to some uncertain measure of near well bore damage brought about by drilling.

The modified DST test provides information on:
- Permeability.
- Well bore loss.
- Mean effective radius of investigation.
- Possible boundaries.

However, as a single hole test it does not provide information on:
- Storage terms.
- Anisotropy.

While the traditional pumping test with surrounding piezometers (interference test) provides information on:
- Storage terms.
- Anisotropy.

However, it does not enable the precise separation between anisotropy and inhomogeneity. Furthermore the risk of an incorrect choice of pump well size and pump type and piezometer locations means that there is a potential for the test to be designed incorrectly.

The pulsed DST test approach suggested in this paper enables information to be gained progressively and economically, while maximizing the information gained and reducing the potential for disasters.

A new method is presented to install piezometers. This overcomes the storage issues associated with standpipes that will delay response to piezometric change. It also overcomes problems associated with pressure transducers that are grouted into a hole and have poor connectivity to the ground or too much intra-hole connectivity. This new installation method is testable by pumping water into the test zone and observing the pressure decay without intra-hole pressure variations.

It should be appreciated by the reader that this paper merely scratches the surface of the subject of well testing and analysis. The object of writing it is not to replace a full study of the subject but rather to overcome current bad practice, and replace it with systems that are not dissimilar in implementation but yield reliable results.

ACKNOWLEDGEMENTS

The author wishes to acknowledge the contribution of his colleagues from Sigra. Specifically Bruce Neels who helped develop the transducer installation system, Russell Noonan, Edward Cawley and Darryl Smith who installed them and carried out the DST testing. Darryl Smith also assisted in the analysis of the results while John Nooyen helped put together the instrumentation.

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THE USE OF AUTOMATED CORE LOGGING TECHNOLOGY TO IMPROVE ESTIMATION OF FRACTURE MINERALOGY AND WEATHERING FOR GEOTECHNICAL INDEX CALCULATIONS

CASSADY L. HARRADEN1, MATTHEW J. CRACKNELL1, JAMES LETT2 AND RON BERRY1

ABSTRACT

In underground mining, it is vitally important to characterise the rock mass conditions as they relate to ground support requirements. Rock mass characterisation is often achieved through the calculation of geotechnical indices. These indices reflect the properties affecting rock mass stability. Fracture parameters such as spacing, density, roughness and orientation are used to design appropriate ground support. The mineralogical properties within and immediately adjacent to fractures also affect the rock mass strength. Therefore, the relative hardness, thickness, and weathering of minerals along fractures are key input parameters in determining overall rock mass characteristics.

Commonly, geotechnical index parameters are collected manually by geotechnical engineers and geologists on drill core. Recent advances in automated core logging technology provide an opportunity to rapidly and consistently collect surface topography (3D laser height data) and mineralogical information (hyperspectral data) from drill core. From a combination of the 3D laser image data and a series of experience-based logical image processing steps, fracture surfaces can be automatically identified and extracted as a group of neighbouring pixels. The mineralogical data is co-registered with the 3D laser data, so the mineralogy of a pixel group representing a fracture can be queried. Mineral hardness and weathering effects can be estimated by analysing the mineralogy within and surrounding a fracture. Fracture fill thickness can be calculated using the number of pixels of each mineral across a fracture. The automated quantification of fracture fill characteristics ensures that these parameters are collected consistently, greatly improving the calculation of geotechnical indices.

INTRODUCTION

Understanding the conditions of a rock mass provides critical information in an underground mining scenario, particularly as it relates to ground support requirements (Hoek et al., 2000). While many of the geotechnical parameters relate to the morphological properties of the rock (e.g. fracture spacing, roughness, etc.), the mineralogical properties are vitally important in understanding geotechnical characteristics. Current practice is to collect mineralogical geotechnical data from drill core by visual inspection and manual logging by a geologist or geotechnical engineer. This method is effective, but can be both time consuming and inconsistent.

Automated hyperspectral drill core logging technologies are becoming increasingly important in ore deposit characterisation, particularly in the area of geologic modelling. The advantages of these automated scanners include fast throughput, low running costs and high resolution (Keeling et al., 2004). Automated geotechnical assessment is not a new concept. The mining industry commonly uses acoustic televiewer down hole logging systems to measure fracture orientations in situ down hole (Shigematsu et al., 2014; Trofimczyk and Du Pisani, 2009). While the televiewer system provides accurate fracture orientations, it does not provide mineralogical information. The Corescan automated core logging system utilises the Hyperspectral Core Imager Mark-III (HCI-3) logging technology for the rapid, non-destructive analysis of drill core. Three high resolution data sets are collected: (1) red-green-blue (RGB) visible imagery (50 µm pixel size), (2) 3D laser height profiles (200 µm pixel size), 15 µm vertical resolution), and (3) visible near-infrared and short wave infrared (VNIR-SWIR) spectra (3.84 nm spectral resolution, 0.5 mm pixel size). These three datasets are co-registered, so the core images and mineralogical data can be draped over the surface model of the drill core (Figure 1).
The aim of this paper is to discuss the use of automated core logging systems to extract fracture aperture, infill mineralogy, and weathering effect information to calculate fracture condition and alteration parameters as they relate to specific geotechnical index calculations.

**GEOTECHNICAL INDEX CALCULATIONS**

In order to assess the ground support requirements for the installation of underground development and extraction infrastructure, consistent rock classification is required (Brady and Brown, 2013; Hoek et al., 2000). Various authors have proposed geotechnical rock mass characterisation indices, many of which are designed to be calculated using properties measured from drill core. Two commonly used indices are the rock mass rating (RMR) index proposed by Bieniawski (1989) and the Norwegian Geotechnical Institute’s tunnelling index (Q-index) developed by Barton et al. (1974). Both indices give guidelines for the excavation type and ground support requirements based on the calculated index value. It should be noted that both the RMR and Q-index calculations refer specifically to ‘joints’. Joints are only one type of discontinuity observed in drill core; however, the general term ‘fracture’ will be used here to reference all types of naturally occurring breaks in the drill core, regardless of geologic origin.

Originally developed for civil engineering applications, the RMR classification is now commonly used to assess underground rock mass conditions (Bieniawski, 1989). The RMR is calculated by the sum of six rock property parameters, one of which is fracture condition which accounts for fracture aperture, hardness of infill, and weathering rating (Table 1) (Bieniawski, 1989). The Q-index was developed by Barton et al. (1974) after comparing numerous case studies of rock behaviour in underground mines. The index is calculated by assigning a numerical value on a scale from 0.001 to 1000 to six geotechnical parameters. The fracture alteration number (Ja) is one of the six parameters required to calculate the Q-index and accounts for fracture infill, aperture and mineralogy (Table 2) (Barton et al., 1974).
The Use of Automated Core Logging Technology

Table 1. Criteria for determining the fracture condition in the RMR system. Modified from Bieniawski (1989).

<table>
<thead>
<tr>
<th>RMR Fracture Condition Guidelines</th>
<th></th>
<th></th>
<th>Weathering</th>
</tr>
</thead>
<tbody>
<tr>
<td>Separation (aperture)</td>
<td>Infilling (gouge)</td>
<td>Weathering</td>
<td></td>
</tr>
<tr>
<td>Description</td>
<td>Rating</td>
<td>Description</td>
<td>Rating</td>
</tr>
<tr>
<td>None</td>
<td>6</td>
<td>None</td>
<td>6</td>
</tr>
<tr>
<td>&lt; 0.1 mm</td>
<td>5</td>
<td>Hard filling &lt; 5 mm</td>
<td>4</td>
</tr>
<tr>
<td>0.1 - 1.0 mm</td>
<td>4</td>
<td>Hard filling &gt; 5 mm</td>
<td>2</td>
</tr>
<tr>
<td>1 - 5 mm</td>
<td>1</td>
<td>Soft filling &lt; 5 mm</td>
<td>2</td>
</tr>
<tr>
<td>&gt; 5 mm</td>
<td>0</td>
<td>Soft filling &gt; 5 mm</td>
<td>0</td>
</tr>
</tbody>
</table>

Table 2. Criteria for determining the fracture alteration values in the Q-index system. Modified from Barton et al. (1974).

<table>
<thead>
<tr>
<th>Q-index Ja Guidelines</th>
<th>Rock wall contact</th>
<th>Ja value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Description</td>
<td>Ja value</td>
<td></td>
</tr>
<tr>
<td>Tightly healed, hard, non-softening, filling</td>
<td>0.75</td>
<td></td>
</tr>
<tr>
<td>Unaltered fracture walls, surface staining only</td>
<td>1.0</td>
<td></td>
</tr>
<tr>
<td>Slightly altered fracture walls, non-softening mineral coatings, clay-free disintegrated rock, etc.</td>
<td>2.0</td>
<td></td>
</tr>
<tr>
<td>Small clay-fraction (non-softening)</td>
<td>3.0</td>
<td></td>
</tr>
<tr>
<td>Softening or low-friction clay mineral coatings</td>
<td>4.0</td>
<td></td>
</tr>
</tbody>
</table>

GEOTECHNICAL PROPERTIES OF MINERALS

The geotechnical behaviour of a rock mass is, in part, determined by the mineralogical properties present within and surrounding fractures. The hardness, swelling potential, and friction potential of the minerals present and the relative abundance of these minerals influence the geotechnical properties of a rock mass. The Corescan system is capable of detecting a number of minerals that are geotechnically significant with respect to their hardness, swelling potential, and friction potential properties (Table 3). Minerals of interest can be divided into four main groups: (1) hard minerals (H), (2) soft, high-friction, non-swelling minerals (SHFNS), (3) soft, low-friction, non-swelling minerals (SLFNS), and (4) very soft, low-friction, swelling clays (VSLFS).

Minerals in group H have consistent geotechnical properties. These minerals are considered to be primary minerals and do not represent weathering as it relates to geotechnical index calculations. Minerals belonging to the remaining three groups (SHFNS, SLFNS, and VSLFS) display very different geotechnical properties than those of group H. These minerals have variable geotechnical properties, but all three are likely the result of weathering affects and must be accounted for in geotechnical index calculations.
Table 3. Geotechnical properties of minerals detected by the Corescan VNIR-SWIR system.

<table>
<thead>
<tr>
<th>Mineral Name</th>
<th>Relative Hardness</th>
<th>Low-friction Potential</th>
<th>Swelling Potential</th>
<th>Geotechnical Mineral Group</th>
</tr>
</thead>
<tbody>
<tr>
<td>amphibole</td>
<td>hard</td>
<td></td>
<td></td>
<td>H</td>
</tr>
<tr>
<td>apophyllite</td>
<td>hard</td>
<td></td>
<td></td>
<td>H</td>
</tr>
<tr>
<td>epidote</td>
<td>hard</td>
<td></td>
<td></td>
<td>H</td>
</tr>
<tr>
<td>prehnite</td>
<td>hard</td>
<td></td>
<td></td>
<td>H</td>
</tr>
<tr>
<td>silica</td>
<td>hard</td>
<td></td>
<td></td>
<td>H</td>
</tr>
<tr>
<td>tourmaline</td>
<td>very hard</td>
<td></td>
<td></td>
<td>H</td>
</tr>
<tr>
<td>carbonate</td>
<td>soft</td>
<td>X</td>
<td></td>
<td>SHFNS</td>
</tr>
<tr>
<td>iron carbonate</td>
<td>soft</td>
<td>X</td>
<td></td>
<td>SHFNS</td>
</tr>
<tr>
<td>iron oxide</td>
<td>soft</td>
<td>X</td>
<td></td>
<td>SHFNS</td>
</tr>
<tr>
<td>chlinochlore</td>
<td>soft</td>
<td>X</td>
<td></td>
<td>SLFNS</td>
</tr>
<tr>
<td>chlorite</td>
<td>soft</td>
<td>X</td>
<td></td>
<td>SLFNS</td>
</tr>
<tr>
<td>kaolinite</td>
<td>soft</td>
<td>X</td>
<td></td>
<td>SLFNS</td>
</tr>
<tr>
<td>phlogopite</td>
<td>soft</td>
<td>X</td>
<td></td>
<td>SLFNS</td>
</tr>
<tr>
<td>sericite</td>
<td>soft</td>
<td>X</td>
<td></td>
<td>SLFNS</td>
</tr>
<tr>
<td>dickite</td>
<td>very soft</td>
<td>X</td>
<td></td>
<td>SLFNS</td>
</tr>
<tr>
<td>gypsum</td>
<td>very soft</td>
<td>X</td>
<td>X</td>
<td>VSLFS</td>
</tr>
<tr>
<td>laumontite</td>
<td>very soft</td>
<td>X</td>
<td>X</td>
<td>VSLFS</td>
</tr>
<tr>
<td>montmorillonite</td>
<td>very soft</td>
<td>X</td>
<td>X</td>
<td>VSLFS</td>
</tr>
<tr>
<td>nontronite</td>
<td>very soft</td>
<td>X</td>
<td>X</td>
<td>VSLFS</td>
</tr>
<tr>
<td>vermiculite</td>
<td>very soft</td>
<td>X</td>
<td>X</td>
<td>VSLFS</td>
</tr>
</tbody>
</table>

H = hard, high friction potential, non-swelling minerals
SHFNS = soft, high friction potential, non-swelling minerals
SLFNS = soft, low friction potential, non-swelling minerals
VSLFS = very soft, low friction potential, swelling minerals

AUTOMATED FRACTURE CONDITION METHODOLOGY

The extraction of geotechnical index parameters requires fracture locations extracted from the laser height data and hyperspectral mineralogical data to determine the minerals present in and around the fracture. The RMR and Q-index input parameters list specific minerals in the classification schemes; however, the infill aperture and extent of weathering away from the fracture are also required. Using the VNIR-SWIR hyperspectral mineralogy and abundance in combination with a series of experience-based, logical, ordered processing steps, the fracture condition and Ja values required in the RMR and Q-index calculations can be automatically determined by the following steps:

- extract fracture infill aperture and mineralogy.
- determine weathering effects away from the fracture.
- calculate fracture condition and Ja.
EXTRACT FRACTURE INFILL APERTURE AND MINERALOGY

To extract the mineralogy and mineral aperture of each fracture, fractures must first be identified. Since drill core is a relatively consistent cylindrical shape, deviations from this shape represent potential fractures. Using a slope analysis of the 3D surface of the drill core, slopes greater than those associated with the curvature of the drill core surface can be filtered, identifying the location of fractures (Harraden et al., 2016). Once neighbouring pixels associated with each fracture are identified, the apparent aperture of the fracture is calculated by multiplying the number of fracture pixels across by the pixel size of 200 µm. In most cases, a fracture is intersected by the drill core at an oblique angle, so this approach calculates apparent aperture. To measure the true aperture, the orientation of the fracture is required. Using the 3D orientation methods outlined in Harraden et al. (2016), simple trigonometry can be used to calculate the true aperture:

\[ t = L \cos \rho \]

where,
- \( t \) = true aperture.
- \( L \) = apparent aperture.
- \( \rho \) = angle between the pole to fracture plane and the drill hole.

(Charlesworth and Kilby, 1981)

Extracting the mineral abundance of the fracture is achieved through a simple query of the mineralogical data coinciding with the selected fracture pixels. A relative mineral abundance for each fracture is calculated by querying the mineralogy of each pixel within the fracture, then calculating the proportion of the minerals present in the fracture.

DETERMINE WEATHERING EFFECTS AWAY FROM THE FRACTURE

Weathering can affect both fracture surfaces and the wall rock surrounding the fracture. The weathering characteristics of a fracture can be calculated by tracking changes in the abundance of weathering minerals away from the fracture. By buffering the fracture pixels at distances of 5 mm and 10 mm, mineralogy away from the fracture can be used to determine if weathering effects extend beyond the boundaries of the fracture surface. This change is monitored by comparing the relative abundance of weathering minerals within the fracture to the relative abundance of weathering minerals within the 5 mm and 10 mm buffers (Figure 2). If the mineralogy within and immediately adjacent to the fracture is dominated by group H minerals, no weathering effects are observed. Where the fracture is dominated by group H minerals with weathering minerals (SHFNS, SLFNS, and VSLFS groups) detected in the buffers, minor weathering effects are present. Where the mineralogy within the fracture and buffers is dominated by minerals in the SHFNS, SLFNS, and VSLFS groups, significant weathering effects are present within the fracture and extend into the wall rock outside of the fracture boundaries.
Figure 2. By tracking the changes in mineralogy away from the fracture boundary, weathering effects can be assessed. A. The boundary of the fracture (red line) as determined from the surface topography analysis. B. The 5 mm buffer (blue line) relative to the fracture boundary. C. The 10 mm buffer (pink line) relative to the fracture boundary. In this example, the mineralogy within the fracture is similar to the mineralogy of the 5 mm buffer, but changes within the 10 mm buffer. This shows that the weathering effects extend a short distance away from the fracture surface.

CALCULATE FRACTURE CONDITION AND JA

From the mineralogical and weathering information extracted from the Corescan data, a series of logical rule sets are used to assign geotechnical index parameter values to each fracture. In the RMR system, three specific ranking guidelines are used to calculate the total rating for the fracture condition parameter: separation (aperture), infilling (gouge), and weathering. The aperture value is assigned by comparing the measured fracture aperture (calculated in previous steps) to the RMR aperture criteria (Table 1). For the RMR system’s fracture infill, a series of logical rule sets that account for the infill aperture and mineral hardness within the fracture are used to rank the infill value (Table 4). Minerals in group H represent hard filling, while minerals in groups SHFNS, SLFNS, and VSLFS represent soft filling. The RMR infill value is assigned based on the criteria outlined in Table 4. Determining the RMR weathering value requires a comparison of the amount of each mineral group present in the fracture, within the 5 mm buffer, and within the 10 mm buffer. The criteria used in the RMR infill processing step is summarised in Table 5. The Q-index requires a single value to calculate the Ja parameter. To assign the Ja value, abundances of the geotechnical mineral groups within the fracture used. Table 6 outlines the specific criteria used in the Ja value rule sets.
### Table 4. Decision criteria for the infill RMR parameter using Corescan mineralogical data.

<table>
<thead>
<tr>
<th>RMR Infill Value</th>
<th>Measured Aperture</th>
<th>Geotechnical Mineral Properties</th>
</tr>
</thead>
<tbody>
<tr>
<td>6</td>
<td>0 mm</td>
<td>H &gt; 50%</td>
</tr>
<tr>
<td>4</td>
<td>&lt; 5 mm</td>
<td>H &gt; 50%</td>
</tr>
<tr>
<td>2</td>
<td>&gt; 5 mm</td>
<td>H &gt; 50%</td>
</tr>
<tr>
<td>2</td>
<td>&lt; 5 mm</td>
<td>SHFNS + SLFNS + VSLFS &gt; 50%</td>
</tr>
<tr>
<td>0</td>
<td>&gt; 5 mm</td>
<td>SHFNS + SLFNS + VSLFS &gt; 50%</td>
</tr>
</tbody>
</table>

H = hard, high friction potential, non-swelling minerals  
SHFNS = soft, high friction potential, non-swelling minerals  
SLFNS = soft, low friction potential, non-swelling minerals  
VSLFS = very soft, low friction potential, swelling minerals

### Table 5. Decision criteria for the weathering RMR parameter using Corescan mineralogical data.

<table>
<thead>
<tr>
<th>RMR Weathering Value</th>
<th>Fracture Mineralogy</th>
<th>5 mm Buffer Mineralogy</th>
<th>10 mm Buffer Mineralogy</th>
</tr>
</thead>
<tbody>
<tr>
<td>6</td>
<td>H &gt; 50%</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>5</td>
<td>SHFNS + SLFNS + VSLFS 40 – 60%</td>
<td>H &gt; 50%</td>
<td>-</td>
</tr>
<tr>
<td>3</td>
<td>SHFNS + SLFNS + VSLFS 40 – 60%</td>
<td>SHFNS + SLFNS + VSLFS 40 – 60%</td>
<td>H &gt; 50%</td>
</tr>
<tr>
<td>1</td>
<td>SHFNS + SLFNS + VSLFS 40 – 60%</td>
<td>SHFNS + SLFNS + VSLFS 40 – 60%</td>
<td>SHFNS + SLFNS + VSLFS 40 – 60%</td>
</tr>
<tr>
<td>0</td>
<td>SHFNS + SLFNS + VSLFS &gt; 60%</td>
<td>SHFNS + SLFNS + VSLFS &gt; 60%</td>
<td>SHFNS + SLFNS + VSLFS &gt; 60%</td>
</tr>
</tbody>
</table>

H = hard, high friction potential, non-swelling minerals  
SHFNS = soft, high friction potential, non-swelling minerals  
SLFNS = soft, low friction potential, non-swelling minerals  
VSLFS = very soft, low friction potential, swelling minerals

### Table 6. Decision criteria for the weathering Q-index Ja parameter using Corescan mineralogical data.

<table>
<thead>
<tr>
<th>Q-index Ja Value</th>
<th>Fracture Mineralogy</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.75</td>
<td>H &gt; 55%</td>
</tr>
<tr>
<td>1.0</td>
<td>H 50 – 55%</td>
</tr>
<tr>
<td>2.0</td>
<td>SLFNS + VSLFS &lt;10%</td>
</tr>
<tr>
<td>3.0</td>
<td>SLFNS + VSLFS 10 – 25%</td>
</tr>
<tr>
<td>4.0</td>
<td>VSLFS &gt; 25%</td>
</tr>
</tbody>
</table>

H = hard, high friction potential, non-swelling minerals  
SHFNS = soft, high friction potential, non-swelling minerals  
SLFNS = soft, low friction potential, non-swelling minerals  
VSLFS = very soft, low friction potential, swelling minerals
The Use of Automated Core Logging Technology

C. L. Harraden, et al

CONCLUSION
Since the mineralogical properties of a rock mass dictate rock behaviour, understanding these properties is vital for the production of robust geotechnical models. Recent advances in automated core logging technology provide an opportunity to rapidly and consistently collect coincident surface topography (laser height data) and mineralogical information (hyperspectral data) from exploration and production drill core. By combining core surface topography and mineralogical data obtained from the Corescan automated logging system, fracture infill aperture, infill mineralogy, and degree of weathering of the wall rock surrounding the fracture can be estimated. Key mineralogical and weathering properties affect the geotechnical response of a rock mass, so applying logical, ordered image processing steps allows mineralogical geotechnical index parameters to be rapidly and consistently calculated. Integration of 3D laser data with hyperspectral derived mineralogical data generates large volumes of consistent data for geotechnical assessments, increasing the accuracy and efficiency of geotechnical rock mass characterisation.

ACKNOWLEDGEMENTS
This research was conducted by the ARC Research Hub for Transforming the Mining Value Chain (project number IH130200004). The authors would like to thank Maya Secheny, Chris Chester, Stephen Guy, and Ronell Carey. We would also like to thank Anthony Harris, Neil Goodey, and David Cooke for their review of this work and continued support.

REFERENCES
LOGGING-WHILE-DRILLING WITH DIAMOND RIGS

A. KEPIĆ¹, M. CARSON¹, H. NGUYEN¹, A. GREENWOOD¹, A. PODOLSKA¹ AND C. DUPUIS¹

ABSTRACT

The Deep Exploration Technologies Cooperative Research Centre (DET CRC) has developed a range of geophysical sensing techniques for two logging-while-drilling (LWD) methods that work with diamond drilling. These methods, the AutoSonde and AutoShuttle provide a logging platform that works well with diamond drilling rigs requiring no modifications to the rig and may be used by the drillers as part of their normal operations. Initially measuring natural gamma, magnetic susceptibility and electrical conductivity, the tools are able to relate the geological-physical properties with depth in an unsupervised manner; the “auto” aspect of the methods. Thus, many of the barriers to using geophysical wireline tools to gather information about the rocks are removed. With the exception of coal and iron ore mine development there is little geophysical wireline logging performed in mineral exploration and development. The DET CRC AutoSonde and AutoShuttle provide a low effort and cost method for mineral exploration geologists to have the sort of information that is regularly used in the petroleum industry.

INTRODUCTION

Mineral exploration heavily depends on borehole drilling to map and discover deposits (Bieniawski, 1989). In-depth knowledge of the chemical and physical rock properties is paramount and assays of core and chip samples is standard practice, yet surprisingly, there is a lack of in-situ down-hole rock property measurements collected. The reluctance to collect down-hole rock property measurements is driven by cost and risk in the mining sector. Mineralisation is, in general, associated with complex geometries, fluid movement, fractures, faults and intrusive structures; the type of rock environments that make borehole environments unstable. Thus, for borehole wireline surveys to be conducted safely in mineral exploration settings, it is common practice to have a drill rig standing over the borehole, adding considerably to the wireline survey costs. Wireline logging was adopted by the petroleum industry because it provided an opportunity to provide important data without a “core sample”; thus, allowing new drilling technologies (such as the tri-cone) to reach greater depths and enhance other information recovered from reservoirs (Hyne, 2012). While the petroleum well often represents a production asset, a borehole in the mineral industry always represents an exploration cost. As such, an unwillingness to increase the investment per borehole has largely been responsible for the sluggish adoption of wireline logging in the mining industry.

LWD eliminates most of the risk and extra costs of wireline operations; additionally, adding significant value to the drilling process by gathering data quickly that is easily transmitted worldwide. LWD emerged over time as the natural solution to the challenges imposed by ever-deeper reservoirs and more complex settings (Collett et al., 2012; Goldberg et al., 2003; Hyne, 2012). The real-time information generated by LWD systems reduces exploration risk and improves efficiency. Currently, LWD is common practice in the petroleum industry (Collett et al., 2012; Goldberg et al., 2003; McMinnies et al., 2007) with the technologies developed to take advantage of relatively large diameter drilling technologies and no requirement to remove material up the inside of the rod-string. Whereas in diamond drilling for base-metals and gold, the small inner diameter (<100 mm) of drill strings (with only a 40-50 mm diameter of space in NQ size), and the requirement to allow core to be removed through the drill string makes it technically difficult to instrument the bottom-hole assembly. In addition, much lower costs per metre in minerals drilling has restricted the investment into LWD technologies for minerals (Reynolds, 2011; Witherly, 2012). Existing LWD petroleum technologies would require significant adaptations for minerals use and the mining market for LWD tools is fragmented with slim margins. Additionally, the energy markets tend to assume geophysical experts will assist in collecting and analysing the data. Thus, it is very unlikely that the petroleum and coal sector technologies will “trickle down” soon to help mineral exploration without costly expert assistance to adapt rigs with extra sensing abilities and then operate these tools.

¹ Deep Exploration Technologies CRC, Curtin University
AUTOSONDE

The autonomous sonde – AutoSonde - developed by the researchers of the DET CRC has been designed to integrate seamlessly with normal drilling processes for conventional diamond drilling. The autonomous sonde is deployed inside the drill string once the borehole has reached its target depth. The sensor portion of the sonde engages past the drill-bit and therefore has an un-obstructed view of the rock-mass. This method of deployment significantly simplifies the logistic of logging in unstable ground since the formation remains supported by the drill string. From this vantage point the autonomous sonde can log the entire borehole while the rods are pulled out of the borehole.

To accomplish the same thing with traditional wireline, the tool would have to make a return trip to the bottom of the borehole every time a rod is pulled. This is because the cable that is threaded through the drill rods makes it impossible to disassemble the string without damaging the cable. In practice this means that a large portion of the borehole may not be surveyed with traditional wireline since the rig standby quickly becomes too onerous. A memory tool, making continuous measurements in time and storing the results, could be then used to remove the requirement of a wireline; however, that would require an up-hole sensing system to monitor rod retrieval to relate measurements logged in time to depth. Such a system has been developed by DMT for example in coal exploration (http://www.dmt-group.com/en/products/geo-measuring-systems/dmt-boreholeshuttle.html), which superficially resembles the DET CRC AutoSonde. However, it requires some work to interface to the drill rig and there appears to be no means to relate time to depth when using a hoist plug to retrieve the rods (an encoder needs to be mounted onto the mast and interfaced to a recording instrument). So to use such a system requires a commitment by the drilling company to use this technology in future or regularly to recoup the investment in outfiting the rig. In the case of the AutoSonde the tool can be delivered to site in an instrument case similar to any gyro or core orientation tool and used similarly.

Generally, the AutoSonde is attached to the bottom of core-barrel assembly (an example of a multi-parameter AutoSonde in Figure 1), as this assembly is normally in place during the lifting of the rods. The drill string is pulled back by one rod length, say 3 m, to allow the AutoSonde to protrude beyond the bit during logging. Normally the whole core-barrel assembly is pumped down prior to removing the rods as per usual practice in Diamond drilling. Before being lowered into the drill string, the autonomous sonde is initialised using a hand-held remote. This initialisation provides a system status and initialises the recording of data from the sensors like any other memory tool. Once initialised, the AutoSonde is lowered to the bottom of the drill string normally used for deployment of the core-barrel. Once the assembly is in place at the bottom of the drill-string the driller then proceeds with drill-string recovery as per usual. The AutoSonde is recovered from the final rod once it reaches the surface and the data are downloaded via the same handheld that was used to initialise the sonde tool. The stored data are related to depth by using information supplied by driller: bit depth at the start of removing the drill string and rod length. Extra sensors within the AutoSonde are used to sense the movement and progression of removing the drill rods so that the measured physical properties of the rock are mapped to depth, producing a normal geophysical log from the hole. To operate the tool the driller starts the tool after entering in the start depth, puts the handset to one side and removes the rod string from the hole normally, then uses the handset to recover the data from the tool when it appears of the surface.

The AutoSonde with gamma is being commercialised by Boart Longyear and has undergone many trials to prove up the quality of the data and viability of its auto-depthing ability. There have been over 10,000 m of logs generated from more than 30 separate logging operations. Data quality is equivalent or better than most slim-line wireline tools when the rod tripping rate is less than 30 m/min. Good depth registration is at the heart of the autonomous sonde concept. With reasonable care by the driller the repeatability of the depth encoding scheme is very good, better than 0.3 m (figure 2) and the results obtained are on par with traditional wireline data.
Figure 1. Multi-sensor prototype AutoSonde with natural gamma, magnetic susceptibility and galvanic resistivity mechanically coupled together via e-threads and controlled via a single handset. The handset is used to start the surveys allow synchronisation of the three tools and data harvesting after the survey is performed. Other sensors additional to the petrophysical sensors are used in post-processing on a local PC/tablet to create a geophysical log from knowing the starting bit depth and length of rod retracted on each withdrawal. While the prototype tool is long, at 3 m, it proved to be quite manageable.

Figure 2. Example log showing the repeatability and accuracy of the AutoSonde. In this example, the natural gamma (red traces) petrophysical changes detected are reproduced to better than 0.3 m. Each natural gamma log was collected as the rods were withdrawn to replace bits. Note that the petrophysical sensors detect changes in the rock differently to that mapped by visual logging of core (lithology column).
Other measurement capabilities are being introduced to the system so that many useful petrophysical measurements may be made on the single run i.e. galvanic resistivity, magnetic susceptibility, total count gamma, chargeability or induced polarisation (IP). All of these measurements are useful in calibrating geophysical exploration surveys and also provide useful input to changes in rock-type or alteration. An example of such a log (from our Brukunga-Adelaide Hills test site) is shown in Figure 3, where the various changes are easily identifiable.

Figure 3. Multi-petrophysical AutoSonde. In this example, the tool shown in Figure 1 was used to log four geophysical properties in a 180 m deep hole at the DET CRC Brukunga drill testing facility. This area has approximately 150-200 m of metasediments with bands of pyrite and pyrrhotite before entering a zone of significant sulphide mineralisation. In this case, the drilling has just started to enter the main sulphide-rich zone at approximately 165 m. The IP chargeability and magnetic susceptibility respond strongly to thin bands of pyrite and pyrrhotite respectively. The dips in the total count gamma (green) and electrical resistivity (blue) signals are due to dolerite intrusives. There are often thin bands of massive sulphide adjacent to the dolerites.
AUTOSHUTTLE

The AutoShuttle has been designed to be deployed with the bottom-hole assembly (BHA) above the core barrel and below the back-end assembly. The AutoShuttle is initiated at surface with each deployment of the empty core tube and the data is farmed at the end of the core run. So it is a true LWD system in that it collects data during the penetration into the rock. Currently, the sensors incorporated into the AutoShuttle utilise natural gamma ray emissions due to the need to see properties of the rock formation through the drill rods. This LWD system for diamond core drilling has been field tested at the DET CRC research facility at Brukunga, and the Mineral Systems Drilling Program, (http://www.minerals.statedevelopment.sa.gov.au/geoscience/pace_copper/about_pace/mineral_systems_drilling/new_technology - sponsored by the Government of South Australia and Industry), and is shown in Figure 4.

Figure 4. AutoShuttle with handset to start and gather data. The prototype AutoShuttle is compatible with both NQ and HQ sized inner core and back-end assembly with couplers. A lidar sensor (placed on the top drive unit with magnets – red circle) wirelessly transmits the top-drive position with respect to the ground back to the handset to track drilling progress for later co-ordination with the downhole tool to assign depth to the data.

The spectral gamma version of the AutoShuttle continuously logs the energy of natural gamma rays to form a spectrum of energies every minute. Some elements such as potassium (K), thorium (Th), and uranium (U) have spectral peaks that are distinctive and may be used to identify geological changes. The coloured area in Figure 5 displays the gamma ray spectrum versus depth with the amplitude represented by colour. This is not typically used in interpretation by itself, but as a quality control tool for tool operation. The outputs normally used for interpretation are estimates of K, U and Th, plus the total count trace to represent overall activity.
Figure 5. An example of AutoShuttle data from the Brukunga test area (30 m interval of drilling in DH08). The coloured panel displays the raw spectrum with depth and gamma ray energy on the vertical and horizontal axes respectively. The amplitude of the gamma ray intensity per unit energy is displayed as a colour, with red representing relatively high intensity and blue-purple low intensity. The very high dynamic range of the raw spectra makes for difficult interpretation, so the data is normally windowed over energy peaks that allow estimates of naturally occurring radio-elements (K, U, Th) to be made (the three traces to the right of the coloured panel). Note that the high intensity regions (hot colours in left portion of the colour panel) indicate rock with lighter elements (low-Z). The black trace is an indication of average atomic number HMI, and tends to mostly reflect Fe concentration.
An additional capability introduced by DET CRC researchers is an analysis of the pattern of scattered gamma ray energies, the heavy mineral index (HMI). The values of HMI may be used to determine the presence of heavier elements, including the approximate fraction of iron in iron-rich formations (Kitzig and Kepic, 2016). Figure 5 displays an example of spectral gamma data plus relative K, U Th concentrations and HMI in data provided by the prototype spectral system. The resulting log shows subtle changes in the rock minerology with vertical resolution better than 0.3 m. In contrast, the core related to data presented in Figure 5 is visually unremarkable, and looks to the (untrained) eye to be a very uniform sequence of rocks. This level of spectral gamma quality and the resulting fine detail cannot be achieved with conventional wireline tools, only a true measurement-while-drilling approach works. The AutoShuttle method exploits the relatively slow advance of drilling, 0.3 m/min, versus 3 m/min for a slow wireline logging rate. As the gamma rays are very penetrative the information comes from a radius of about 0.2 m, providing a robust indication of the rock properties. Spectral gamma measurements from the AutoShuttle demonstrate not only cost savings over wireline deployment, but an ability to make quality measurements not feasible with wireline tools.

Other sensors such as water pressure and an accelerometer are used to monitor down-hole activity (water pressure changes with flushing-drilling/rotating rods) to help register the shuttle’s depth with an up-hole masthead sensor during the drilling progress. At the end of the drill run (usually up to 3 m) the shuttle is retrieved along with the rock core and the data is downloaded from the shuttle to the hand unit via infra-red communication. This downhole data is combined with the start-depth of drilling provided by the driller plus the mast-head sensor (Figure 4) to create a log section from the coring activity. Multiple coring runs add to the log section. This additional petro-physical data is collected without significant disruption to the drilling process and the AutoShuttle is easily operated by the drilling crew.

CONCLUSION

We demonstrate that the AutoSonde and AutoShuttle LWD tools developed by DET CRC provide a new capability to produce data that is rarely collected by most gold and base metal prospects or mines. The petrophysical data collected is similar to that collected from wireline logging. These tools don’t require the mobilisation of specialised technicians with associated rig stand-by or separate dedicated winches normally associated with petrophysical logs. Nor are there changes to the drill-rig, or disruption to the normal drilling operations. Thus, their use will not add significant extra cost to the drilling process. The extra data collected about the rock formations adds significantly to the visual and geochemical analysis of core to assist in exploration and development.

ACKNOWLEDGEMENTS

The work has been supported by the DET CRC whose activities are funded by the Australian Government's Cooperative Research Centre Programme. This is DET CRC Document 2017/1014.

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APPLICATION OF HAND-HELD LASER INDUCED BREAKDOWN SPECTROSCOPY TO DRILLING SAMPLES: NEW TECHNOLOGY PROVIDING NEW IN-FIELD ANALYTICAL CAPABILITIES

ANDREW SOMERS

ABSTRACT

Laser induced breakdown spectroscopy (LIBS) is a form of atomic emission spectroscopy that allows the identification and measurement of the elemental composition of materials. Using a laser, plasma is created on the surface of materials. This plasma interacts with the sample by ablating a small amount and the resultant optical emissions related to the cooling of the plasma are collected and plotted spectrally. Each element emits at specific wavelengths and the intensity of the emission is directly related to concentration. The analysis of a wide range of elements is possible using this technique, most notably including very light elements such as Li, Be, B, C, O, F and Na that have previously not been possible with conventional hand-held analytical devices. In addition to this, improved sensitivities for important rock forming elements such as Mg presents exciting opportunities for improved lithological and mineralogical determination to be made in the field. LIBS analysers have been commercially available in a laboratory environment for several decades and with recent advances in technology, miniaturised, hand-held LIBS analysers have now become commercially available.

The application of this new technology to drilling samples such as rock chips from percussion drilling techniques and drill-core from diamond drilling present new in-field analytical capabilities for geoscientists. This includes the opportunity to analyse elements that were previously not possible using hand-held analytical means and also to map, at a micro-scale, the distribution of elements within samples in a field setting. In addition to a comprehensive elements suite LIBS allows a high degree of spatial resolution allowing discrete analysis of individual minerals and veins. This technique can complement existing field portable analysis with field portable X-ray fluorescence (fpXRF), near infra-red, short wave infra-red (NIR/SWIR) and laboratory analysis through pre-screening and selection of high priority samples for both conventional bulk geochemistry and micro-analytical techniques such as laser ablation inductively coupled plasma mass spectrometry (LA ICP-MS), electron microprobe and micro XRF. LIBS also presents opportunities for integration with existing automated core scanning systems that incorporate XRF, NIR/SWIR and other sensors.

INTRODUCTION

The ability to rapidly and effectively characterise geological samples in field allows better decisions to be made to manage mineral exploration and mine based geological drilling. The application of analytical techniques such as field portable and, more recently, on-line XRF and NIR/SWIR has improved objectivity in the interpretation of geological samples for the purpose of mapping and modelling sub-surface geology. LIBS analysis can provide valuable additional data that than complement and augment these conventional techniques especially with regards to light element detection for elements such as Li, Be, B, O, F, C and Na. LIBS spectral data collected over a wide bandwidth e.g. 200-900 nm, contains a comprehensive geochemical fingerprint that can be used effectively to determine specific mineral phases and/or rock types (Alvey et al., 2010; Harmon et al., 2005).

This study looks at the application of hand-held LIBS to both diamond drilling and percussion drilling samples. Although technically speaking the LIBS technique can be applied to solids, liquids and gases the ideal sample type for the SciAps Z300 hand-held LIBS used in this study is a solid. With regards to the scope of this study ideal samples could be ranked from best to worst being sawn diamond core, drill chips and fines respectively. The fine unconsolidated sample generated with percussion drilling techniques presents challenges with LIBS analysis due to the very hot plasma expanding air between particles in the sample resulting in poor sample density in the plasma and typically poor results. Reconstitution of the sample by hydraulic press or at the very least mounting sample on adhesive tape stabilises the sample to allow proper testing.

1 SciAps Inc
LIBS also presents an advantage over conventional fpXRF analysis due to the ability for users to safely hold samples including very small samples such as drill chips manually in front of the analyser without concern about X-ray radiation (Figure 1).

Figure 1. Hand-held LIBS operator safely holding a small rock chip in front of the analyser.

Understanding not only what a sample is made up of chemically but also how the specific elemental components are distributed within mineral phases provides valuable information in exploration, mining and mineral processing applications. This LIBS system used in this study allows in field analysis of the relative spatial distribution of elements and could complement conventional microanalytical techniques employed in laboratories, albeit at a coarser spatial resolution than most of these techniques. This could benefit in allowing better sample selection in field and even allow in field determination of mineral chemistry and elemental deportment.

THE LIBS ANALYTICAL TECHNIQUE

LIBS is an optical spectroscopic technique that uses a focused solid-state laser pulse to ablate a small amount of sample to a plasma plume. The excited molecules of sample matter will return to its ground state by emitting characteristic lines for each element. A dispersive spectrometer and CCD detector is used to collect the light from the plasma in order to resolve the signatures of the excited atomic species by way of recording intensities at specific wavelengths. Each element in the periodic table emits light in the 200-900 nm spectral range under the conditions created with LIBS analysis. As such, a complete chemical fingerprint can be captured simultaneously with a single laser pulse in parts of a second. (De Lucia and Gottfried, 2011; Harmon et al., 2013; Connors, Somers and Day, 2016; Harmon et al., 2005). Typical detection limits are in the 10’s to 100’s of ppm range for most elements when averaged across the field of sample collection points possible with the rastering capability of the Z300. The binary presence or absence of specific elements such as Au that are typically inhomogeneously distributed within samples can be observed at far lower concentrations especially when using high density rastering.

For this study, the Z300 hand-held LIBS analyser was utilised. This hand-held LIBS analyser is portable, has a broad spectral range from 190 to 950 nm, argon purging, XYZ stage for beam rastering and in-built camera for targeting and documentation of analysis. The Z300 uses a proprietary PULSARTM 1064 nm
Nd:YAG pulsed laser with 50 μm focused beam size. This diode pumped solid state laser delivers 6 mJ to the sample with a 1 nS pulse duration and is used with a 10 Hz firing rate. The instrument is capable of argon purging utilising small, replaceable 21 gm cartridges that fit in the handle or by connection to a larger gas cylinder for stationary use. It is also possible to operate the analyser with a helium purge or without purge gas i.e. air. If argon purge is selected by the user, flow is started approximately 300 ms before the laser pulse train to displace any accumulated air and ceases after the last shot to minimise argon usage. Argon pressure is monitored before every measurement and the flow is controlled by a microsolenoid valve. The gas is delivered directly to the area of the sample surface where the LIBS plasma occurs (Loree and Radziemski, 1981).

The resultant signal is collected and passed through internal spectrometers that use time gated CCD detectors with resolution of 0.1 nm full width at half maximum (FWHM) below 400 nm and 0.3 nm above 400 nm. Data are typically collected with 1 μs delay times (the time between laser shot and beginning of emitted light integration). This device can analyse a raster pattern of up to a 2 mm x 2 mm area. While the stage is capable of 12.5 micron steps, the raster used to collect data is normally 125 to 500 microns between locations and tests between 1 and 256 locations dependant on the mode of use selected. The raster pattern and spacing can also be customised by the user. To improve data quality multiple analytical shots can be collected at each location and non-analytical cleaning shots can be performed prior to the collection of data. The number of cleaning shots (no data collected) and data shots is also user selectable (Loree and Radziemski, 1981). Figure 2 illustrates the schematic of the analytical components of a typical LIBS system.

The Z300 is capable of generating a range of different data products including spectra at individual points or averages across the raster pattern and quantitative analysis of these spectra. Using the static heat maps describing relative distribution of elements can also be generated on the analyser or exported for more advanced processing using third party software such as Reflex ioGAS (Figure 3 and Figure 4). The 2 operation modes or Apps utilised in this study used were the qualitative GeoChem Pro App for the diamond drilling samples and the quantitative GeoChem App for percussion drilling samples.
DATA AND DISCUSSION

In order to identify and illustrate fit for purpose applications of this technology for field based analytical work on drilling samples several studies were undertaken.

Case Study: Diamond Drill Core Samples

Flat slabs of rock analogous to the typical surface of cut drill core were studied using the high-density sampling mode possible using the GeoChem Pro application on the Z300. This allowed a high density 16 x 16 raster pattern to be tested at 125 um spacings. These samples were specific samples cut from core for petrographic description i.e. the remaining slabs from thin section preparation or samples that had been cut from drill core for other microanalytical techniques. Comparison of LIBS data with conventional microanalysis is an important step in validating the data generated by this new technique to better understand sensitives, strengths and limitations (Figure 5).

The primary objective of the work carried out on these samples was to mimic conventional microanalytical approaches to show differences in mineral chemistry and elemental distribution related to mineralisation and associated alteration. In most cases similar trending is observed for many elements. Some limitations and controls were observed and for best results with relative element distribution maps being that samples
should be flat although not necessarily polished. Samples should exhibit partitioning of elements into different minerals to show good contrast.

Due to the laser spot size of 50 um, samples with particle size larger than this is preferable to avoid “mixels” or single spectra that have sampled multiple grains with variable matrix types resulting in a mixed signal within the pixel. While a particle size of 250-300 um seems ideal using the step size of 125 um reducing the step size may yield better performance in finer grained samples. Careful targeting and recording of sample site is critical for good comparison with other data.

Previous work had also yielded data that was uploaded to The Spectral Geologist [TSG] software typically used for NIR/SWIR data to spatially locate LIBS spectra along drill core demonstrating the ease with which the data from this technique can be stored and viewed alongside more conventional data using industry standard data management and visualisation tools (Figure 6).

Figure 5. Examples of heat maps generated on drilling samples as compared to XRF microprobe analysis. Australian Synchrotron’s XRF microprobe (XFM) data courtesy of Shaun Barker (University of Waikato) and Jeremy Vaughan (Barrick Gold Exploration).

Figure 6. LIBS data presented in TSG showing spatially registered spectra with elemental lines of interest. TSG data courtesy of Jon Huntington (CSIRO).
Case Study: Percussion Drilling

Lithium Australia has been working with SciAps Inc. to develop the hand-held LIBS product for several years now through the provision of lithium bearing mineral samples across a range of different sample media types and mineralisation styles. This work has culminated with the application of the LIBS analyser to guide exploration drilling at a remote location in Sonara, Mexico. Keeping in mind the challenges presented when testing with LIBS on particulate samples the data presented herewith was generated in the field using well prepared and presented samples. The preparation of the samples was performed using the Reflex SP30M field samples press which allowed hydraulically pressed drilling samples to be delivered to the LIBS analyser at a similar rate to the actual drill rig (Figure 7). The analysis was carried out using the quantitative GeoChem App with customised empirical calibration for Li. This allowed a 12 location raster using a 4 x 3 pattern at 500 um spacing. Each test was conducted in 3 different locations within the sample and averaged using the on board software on the analyser to yield concentration values for Li (Figure 8).

Figure 7. SciAps Z300 analysers with Reflex SP30M sample press in field location. Courtesy of Lithium Australia. Inset: Close up of sample diameter = 25 mm.
Figure 8. Data comparison of conventional laboratory results to in-field Z300 results (Courtesy of Lithium Australia).

AF-17-001 Real-time Laboratory and LIBS Lithium Assay comparisons.

AF-17-002 Real-time Laboratory and LIBS Lithium Assay comparisons.

AF-17-003 Real-time Laboratory and LIBS Lithium Assay comparisons.
The data generated shows clearly how high-quality quantitative in field analysis of geological samples is possible using this technique, if proper sampling and sample presentation methodologies are employed. The company carrying this work out expressed how the success of this initial phase of field work has given them the confidence needed to use the field-portable LIBS analyser for operation control during the remainder of the drilling. Lithium Australia believes this real-time control will have a substantial effect on exploration cost optimisation during the first round of drilling (Griffin, 2017).

CONCLUSION

The advent of hand-held LIBS analysers certainly presents exciting opportunities for field based analysis of drilling samples generated using the standard conventional drilling methods. Depending on the objectives of the investigation LIBS can provide both quantitative bulk analytical results for a comprehensive element suite on well prepared, representative samples or detailed elemental distribution maps over discrete sample areas on sawn drill core or slabs thereof. The initial data are very encouraging of applications for in field microanalytical, targeted mineral chemistry and bulk chemistry studies to complement and augment both conventional field portable and laboratory based techniques.

With regards to the ability to map spatial distribution of elements further work to allow quantification of individual or grouped pixels of a similar matrix type within the analysis area is underway and will create new opportunities to in field analysis of drilling samples and geological samples in general.

This study demonstrates the ability to test elements in field that were not possible before the availability of field portable LIBS systems and has allowed the effective exploration using direct analysis of analytes such as Li. Being able to better understand specific discrete parts of samples without a loss in sensitivity experienced with the use of fpXRF allows effective understanding of mineral chemistry and potential geochemical vectoring application that are now emerging.

The ability of LIBS to provide a comprehensive geochemical fingerprint for rocks and minerals also presents opportunities for spectral mineral identification approaches.

REFERENCES


THE USE OF LARGE DIAMETER ROTARY DRILLING (BAUER) FOR BULK
METALLURGICAL SAMPLING FOR THE BOYONGAN PORPHYRY CU-AU
DEPOSIT, PHILIPPINES

J. J. N. LAMSON1 AND C. J. C. MANIPOP2

ABSTRACT

For continuous advancement in understanding the geometallurgy of the Boyongan-Bayugo Porphyry Copper-Gold (Cu-Au) Deposit, various drilling and sampling programs had been conducted from 2001 to 2015. This includes a deep auger drilling program in 2014 dedicated to collect bulk samples for metallurgical test work as part of the definitive feasibility study (DFS) for the deposit. For this purpose, a total of 721.1 m of advance from six holes with depths ranging from 36.2 to 150.5 m were drilled by BAUER using a large diameter rotary drill rig (BS 110). BS 110 utilised Kelly extension bars with various types of augers, buckets, core barrels and modified cross cutter with 1,200 and 1,500 mm diameter. As the drilling process involved trimming and chiselling of samples prior to retrieval, recovered samples were mostly slurry and deemed unsuitable for pilot tests for crushing and grinding indexes. Drilling advances of 3 to 6 m were documented and tagged for dispatch to sample preparation.

Samples were then subjected to crushing, homogenisation and splitting, generally following those procedures employed for the past exploration programs with modifications based on additional considerations for bulk sampling. Representative sample splits were then nitrogen purged and sealed in drums and pails. Samples were dispatched for routine gold and major elements analysis and copper speciation and multi-elements using inductively coupled plasma optical emission spectrometry (ICP/OES) and X-ray fluorescence (XRF). Quality assurance and quality control (QA/QC) analysis of the insertions (25%: blanks, certified reference materials (CRMs), internal control standards (ICS), pulp and coarse duplicates) for gold and copper showed no contamination, good accuracy of results, precision of the laboratory and reproducibility of the sampling and preparation.

This paper outlines sampling procedures and QA/QC protocols employed to establish and affirm credibility of results for the bulk samples sent for further laboratory tests using quantitative evaluation of minerals by scanning electron microscope (QEMSCAN), X-ray diffraction (XRD) and final metallurgical testing as part of the DFS for the deposit.

INTRODUCTION

Boyongan-Bayugo Porphyry Cu-Au Deposit, in the Surigao Mineral District, is located on the north-eastern tip of Mindanao, Philippines (see Figure 1). It is approximately 750 km southeast of Manila. Since its discovery in August 2000, it has been subjected to various drilling and sampling programs by Anglo American Exploration Philippines Inc., (AAEPI), Philex Mining Corporation (PMC), and Silangan Mindanao Mining Co., Inc. (SMMC). These include intermittent programs for continuous advancement in the understanding of the geometallurgy of the deposit as early as 2001, to 2015.

In 2001-2003 and 2006-2007, AAEPI submitted Boyongan drill core samples for metallurgical testing and analysis to Anglo American Research Laboratories in Johannesburg, South Africa. To supplement this and include the Bayugo Deposit, PMC-SMMC in 2009 and 2012-2013, submitted composited drill core samples to G&T Metallurgical Services and ALS Metallurgy in Kamloops, Canada and to METCON Research and KD Engineering in Tucson, Arizona. Whereas these two earlier major programs utilised samples from regular diameter diamond drilling, more recent programs conducted in 2014 to 2015 by PMC-SMMC involved additional bulk sampling procedures, including tunnel sampling and large diameter rotary drilling. This paper outlines the procedures and particulars of the large diameter rotary drilling program commissioned to collect sufficient bulk samples of the top portion of the Boyongan deposit for the purpose of metallurgical testing.

1 Geology Department, Silangan Mindanao Mining Co., Inc.
2 Technical Study Group, Silangan Mindanao Mining Co., Inc.
METHODOLOGY

Six holes with a total of 721.1 m drilling advance were drilled by BAUER with depths ranging from 36.2 to 150.5 m. For this purpose, BS110 (shown in Figure 2A & B) utilising Kelly extension bars with various types of augers, buckets, core barrels and modified cross cutters (shown in Figure 2C) were used. Generally, drilling and sampling procedures involved digging of “rat holes” as temporary laydown for accessories including extension bars and bit assembly. Then auger drilling followed by sequential trimming using core barrels and sample retrieving using buckets were done. In addition, modified cross cutters were used after grouting and re-drilling at certain intervals. With this, recovered samples were mostly slurry and deemed unsuitable for determination of crushing and grinding indexes using pilot tests.

For sampling and documentation, procedures generally followed those employed for the past exploration programs with modifications based on additional considerations for bulk sampling. Drilling advances ranging from 3 to 6 m were documented by geologists and tagged for dispatch to sample preparation. Samples were then subjected to manual crushing and homogenisation in a mixing pad by using bobcats for windrow construction. Then, samples were split using two-stage manual quartering with diagonal discards as coarse rejects. Sample discards were returned to container bins for storage while sample splits were further homogenised and reclaimed manually producing representative sample splits. These samples were then nitrogen purged and sealed in drums and pails as required.

DISPATCH AND ANALYSES

The purged pails and drums were then dispatched to ALS Metallurgy in Balcatta, Australia for metallurgical testing. Consequently, representative samples in pails were also dispatched for sample preparation following PMC-SMMCI procedures and QA/QC considerations in the company’s in-house facility. Then, samples with QA/QC insertions were sent for routine gold and major elements analysis, and copper speciation and multi-elements by ICP/OES and XRF to Intertek in Metro Manila, Philippines.
Use of Large Diameter Rotary Drilling (BAUER) for Bulk Metallurgical Sampling

J. J. Lamson and C. J. Manipon

AIG Drilling for Geology II: 26-28 July 2017
Brisbane, Australia

QA/QC ANALYSIS

Gold and copper results of the analyses for the QA/QC insertions were then subjected to routine analysis for evaluation of the sampling, sample preparation and analyses procedures. All in all, blanks, CRMs, ICSs, pulp and coarse duplicates inserted accounted for 25% of the total number of samples. Results showed no significant contamination as shown by the blanks insertions. Also, results for coarse duplicates showed good precision and reproducibility of the sampling and sampling preparation. CRMs and ICS insertions also showed good accuracy of the results that are well within the margin of 10% absolute relative difference (ARD) from the corresponding certified values. In addition, results for pulp duplicates showed good precision and reproducibility of the assay laboratory.
CONCLUSION

The use of large diameter rotary drilling is a suitable alternative method to collect sufficient ore samples for the purpose of bulk metallurgical testing. Furthermore, QA/QC analyses of the assay results showed that modified sampling and sample preparation procedures and protocols employed to accommodate bulk samples are of good quality and well within the bounds of the QA/QC monitoring employed for the deposit. However, overgrinding of clayey ore material with this method is likely. Further investigations in this matter is recommended so that refinements in the drilling techniques could be forwarded.

REFERENCES


ABSTRACT

Bulk density is one of the three parameters that are fundamental to any resource estimate; the other two being volume and grade. Over the last 15 years there has been a significant improvement in the attention paid to measuring and estimating bulk density. Measurements are more frequent and geologists are generally more aware of the pitfalls of poor measurement practices. As always, there are areas for improvement, some common errors and frequently asked questions that warrant some discussion.

The critical issues in building dry bulk density into a mineral resource estimate are:

- Sample selection – what we collect and measure, and the risk of bias.
- The physical characteristics of the samples – and how these will constrain the range of density measurement methods that are feasible.
- Measurement methods – there are several alternatives and many variants from which to choose.
- Quality control and quality assurance (QA/QC) – reducing errors and risk; building confidence.
- An adequate number of samples – how many is enough?
- Modelling the data.

Lipton and Horton (2014) discussed sample selection and presented several methods for measuring density, ranging from laboratory tests on small scale samples to estimates based on bulk sampling. The advantages and limitations of each method, and the importance of quality control were discussed.

This paper addresses a range of common questions with illustrations from exploration and mining projects. For brevity, the term “bulk density” is used to refer to dry bulk density or DBD.

WHAT’S WRONG WITH SELECTING SMALL SAMPLES?

Sampling for density measurements should satisfy the golden rule of all sampling: every particle in the interval (the lot) should have an equal probability of selection. This ensures that the sample is representative of the interval. The selection of small sub-samples of core for density measurement, unless truly random or from homogeneous core, is likely to be biased towards the more competent sticks of core. This almost always results in bias towards more dense material.

This is a common problem in weathered or intensely altered rocks in a variety of deposit types. The potential impact of sub-sampling is illustrated by data from the Marlborough nickel laterite deposit. A typical tray of core may vary from massive limonitic ore, crumbly saprolite and vughy silica boxworks.

Figure 1 shows the bulk density of full 1 m core samples versus 10 – 15 cm subsamples from within the same intervals (Lipton, 2000). The variance of the sub-samples is higher than the continuous core samples, as is expected. The mean of the sub-samples is biased high due to preferential sampling of the most competent materials. The very low density material measured by the continuous core was not tested by the sub-samples.

A further problem with sub-sampling is that the density results cannot be directly compared with the assay data for the full samples or geometallurgical data measured over continuous intervals. Measuring bulk density over full assay sample intervals is therefore preferred.
Figure 1. Comparison of the bulk density of continuous core samples vs small sub-samples, measured on nickel laterite core (Lipton, 2000).

DO I NEED TO SEAL MY (SUB-)SAMPLES?

Many rock-types, especially hydrothermally altered rocks or clastic sediments, are porous. In extreme cases, they may contain vughs and large cavities. Porosity must be considered when measuring bulk density using water displacement methods. One option is to seal the sample to prevent water filling up the voids. Wax can be used but waxing is time-consuming, expensive, and contaminates the sample. Various other sealants including hairspray and varnish have been used to seal samples but their effectiveness has not always been demonstrated.

Plastic food wrap has been used in some projects to seal samples prior to using a water immersion method. There is an obvious risk that the plastic may trap air, particularly if the core has broken ends. Furthermore, information on the density of the wrap is required.

Figure 2 shows a case history where two methods were used to measure the bulk density of drill core samples; the caliper method and an Archimedean method (Lipton and Horton, 2014). The figure shows the ratio of the two density values obtained for each sample, in chronological order. The blue points are samples for which the Archimedean method allowed the sample to become saturated with water. The correlation between the data for the blue points was good, with only a small positive bias in the caliper data relative to the Archimedean values. The red points represent samples that were wrapped in food wrap to seal them and the Archimedean method was changed to an unsaturated procedure. As can be seen, entrapped air caused the weight in water to be under-estimated and severely biased the density data obtained with the Archimedean method.
Vacuum-sealing, using good quality equipment, may be a suitable alternative. For many rocks, however, there may be no need to seal the samples because, as illustrated above, the Archimedean method for saturated samples can provide correct bulk density data (AS2891.9.2-2005).

**DO DIFFERENT METHODS PRODUCE DIFFERENT RESULTS?**

The laws of physics are immutable, so if bulk density measurements by any valid method are correctly applied and if assumptions (such as, the core is not porous) are valid, different methods applied to the same sample will produce the same results.

Scogings (2015) compared up to five different methods on the same samples of chromitite and pyroxenite core from the Batlhako mine in South Africa. Table 1 presents the data. Although the data set is very small, the comparison clearly shows that the methods produce essentially the same results. The small variations are experimental errors. The largest difference compared to the caliper estimate is shown by the use of clingwrap which, as has been demonstrated, is prone to underestimation of bulk density due to entrapped air.

When different methods produce different bulk density results, this is simply an indication that one, or maybe both, methods is not being correctly or appropriately applied.

Anglo American Technical Solutions Research in collaboration with Micrometrics developed a gas pycnometer (marketed as the AccuPyc II 1340) that has a large enough chamber to measure the bulk density of core up to PQ diameter and 280 mm length. Makhuvha et al (2014) report comparisons between an Archimedean immersion method and the AccuPyc using three alloy cylinders of known bulk density (certified reference materials). Both methods produced results with very good precision (Figure 3). Interestingly, the Archimedean results were biased high by about 0.35%. This is approximately the difference between measurements using water at 4°C (the temperature at which the density of pure water is 1.00 t/m³) and water at 25°C.
Table 1. Chromitite and pyroxenite dry bulk density estimated by various methods (from Scogings, 2015).

<table>
<thead>
<tr>
<th>Method</th>
<th>Diameter cm</th>
<th>Length cm</th>
<th>Volume cm³</th>
<th>Mass in air g</th>
<th>Mass in water g</th>
<th>Sealant mass g</th>
<th>Sealant density g/cm³</th>
<th>Sealant volume cm³</th>
<th>Density g/cm³</th>
<th>Difference vs. Colliper</th>
</tr>
</thead>
<tbody>
<tr>
<td>Collipper</td>
<td>6.3</td>
<td>11.35</td>
<td>353.95</td>
<td>880.25</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>2.49</td>
</tr>
<tr>
<td>Vacuum pack</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>887.85</td>
<td>530.75</td>
<td>7.6</td>
<td>0.9</td>
<td>8.26</td>
<td>2.52 [1.5%]</td>
</tr>
<tr>
<td>Paraffin Wax</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>887.35</td>
<td>536.9</td>
<td>7.1</td>
<td>0.9</td>
<td>7.89</td>
<td>2.57 [3.3%]</td>
</tr>
</tbody>
</table>

**SG2 oxidised pyroxenite**

<table>
<thead>
<tr>
<th>Method</th>
<th>Diameter cm</th>
<th>Length cm</th>
<th>Volume cm³</th>
<th>Mass in air g</th>
<th>Mass in water g</th>
<th>Sealant mass g</th>
<th>Sealant density g/cm³</th>
<th>Sealant volume cm³</th>
<th>Density g/cm³</th>
<th>Difference vs. Colliper</th>
</tr>
</thead>
<tbody>
<tr>
<td>Collipper</td>
<td>4.76</td>
<td>19.6</td>
<td>348.93</td>
<td>1145.85</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>3.28</td>
</tr>
<tr>
<td>Archimedes</td>
<td></td>
<td></td>
<td></td>
<td>1145.85</td>
<td>801.05</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>3.32 [1.2%]</td>
</tr>
<tr>
<td>Cling Wrap</td>
<td>1147.95</td>
<td>786.1</td>
<td>21.2</td>
<td>0.9</td>
<td>2.28</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>3.19 [-3.0%]</td>
</tr>
<tr>
<td>Vacuum pack</td>
<td>1153.5</td>
<td>796.4</td>
<td>2.1</td>
<td>0.9</td>
<td>8.32</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>3.29 [0.0%]</td>
</tr>
<tr>
<td>Paraffin Wax</td>
<td>1154.85</td>
<td>800.2</td>
<td>9</td>
<td>0.9</td>
<td>10</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>3.32 [1.2%]</td>
</tr>
<tr>
<td>11R0007</td>
<td>22.1±22.8</td>
<td>21.2</td>
<td>1.55</td>
<td>0.9</td>
<td>1.72</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>4.21</td>
</tr>
<tr>
<td>Paraffin wax (1/2 core)</td>
<td>175.8</td>
<td>132.65</td>
<td>1.55</td>
<td>0.9</td>
<td>1.72</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>4.21</td>
</tr>
<tr>
<td>Spray Lacquer (1/4 core)</td>
<td>94.45</td>
<td>71.85</td>
<td>0.4</td>
<td>0.9</td>
<td>0.44</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>4.24</td>
</tr>
</tbody>
</table>

Figure 3. Comparison of density measurements on standard alloy cylinders by core pycnometer and Archimedes methods (from Makhuvha et al, 2014).

Makhuvha et al (op cit) continued the comparison by using 248 core samples of quartz monzonite, breccia and oxidised breccia from the Los Bronces copper mine. Duplicate analyses showed that the repeatability of both methods was excellent. However, comparison of the two methods applied to the same samples revealed a significant bias (Figure 4).
The scatterplot shows two important features of the data; the AccuPyc results are biased high relative to the Archimedean method and the AccuPyc results appear to be truncated at about 2.65 t/m$^3$. The most likely explanation, in the view of the author, is that the pressurised gas in the pycnometer penetrated the pore spaces resulting in the underestimation of the volume of the samples and hence overestimation of the bulk density. The measurements appear tending towards grain density values (equating effectively to reduction of porosity) rather than bulk density.

WHAT’S THE BEST METHOD FOR MEASURING BULK DENSITY?

From review of the geological features that affect bulk density and the various methods available for measuring bulk density (Lipton and Horton, op cit), it should be apparent that other than a strong preference for measuring continuous core rather than sub-samples, the best method will be specific to the geology of the deposit and the practicalities of the project.

The critical point is to have adequate QA/QC procedures in place and, preferably, to demonstrate the reliability of the selected method by duplicate measurements using a second method. QA/QC programs should include:

- Detailed documentation of procedures and standards, including recording of immersion time and water temperature.
- Testing of methods prior to implementation.
- Regular calibration and validation.
- Daily checks on scales and balances using standard reference samples.
- Repeat measurements at the end of the day or round.
- Duplicate measurements made by independent laboratories (i.e. external check samples).
- Independent audit of processes.
A good QA/QC program provides confidence in the results, an audit trail, and confidence that the data can be used to estimate mineral resources.

CONCLUSION

Much progress has been made in improving the measurement of bulk density and its estimation in mineral resource estimates in the last 15 years. The author’s experience from auditing sites and reviewing company reports is that more data appears to be collected and documentation is generally more complete.

The 2012 JORC Code is more prescriptive than its predecessor, and Table 1 of the JORC Code now specifies:

“[Bulk density] whether assumed or determined… If determined, the method used, whether wet or dry, the frequency of measurements, the nature, size and representativeness of the samples.

The bulk density … must have been measured by methods that adequately account for void spaces (vugs, porosity, etc) moisture and differences between rock and alteration zones within the deposit”

These important additions to Table 1 require the collection of good quality bulk density data. If these principles are applied, projects will avoid some of the errors and disappointments of the past.

In almost all cases, collection of good quality bulk density data will start with the geologist making sound scientific choices of sampling method and measurement method.

REFERENCES

AS2891.9.2-2005 Australian Standard, Methods of sampling and testing asphalt - Determination of bulk density of compacted asphalt - Presaturation method


INTRODUCTION

Industrial minerals are normally classified and specified according to their end uses. Common examples are gypsum in wallboard and talc in body powder, where properties such as colour and chemical purity are important. It is essential to understand the mineral quality and mineral distribution of a deposit in order to meet the specifications as defined by a manufacturer or end user.

For any type of industrial mineral project, the main economic driver will be the market available for the mineral’s non-metallurgical properties.

Classification of industrial mineral deposits requires an understanding of exploration drilling procedures, given that drilling methods could affect intrinsic properties of minerals.

Good control on the quality of data generated during the exploration drilling and project evaluation phase is vital. It is the data produced during a drilling campaign that forms the basis for all subsequent decision making on the specific end uses of an industrial mineral.

QUALITY ASSURANCE AND QUALITY CONTROL (QA/QC) PROCEDURES IN INDUSTRIAL MINERAL EXPLORATION DRILLING PROGRAMS.

Publicly traded companies are now required by the Australian Securities Exchange (ASX), and other bourses, to release data that is accompanied by an outline of sampling and QA/QC procedures used during the collection and analysis of exploration samples. Many financial institutions now require an impartial audit.

Work carried out on graphite and vermiculite drilling projects detail the fact that a good QA/QC program is one that is active and regularly reviewed throughout the data collection process (Scogings and Coombes, 2014). This ensures quality information flow for end use study.

Twinned Holes

One method used to verify the quality of sampling and assaying on exploration projects is the use of twinned holes. This is used to compare the results from a reverse circulation (RC) rock chip drillhole by twining the hole with a diamond drill (DD) hole producing core.

Twinned holes are specifically referred to in the JORC Code (JORC, 2012) Table 1 for the verification of sampling and assaying, and are traditionally drilled for verification of historic data or confirmation of drillhole data during geological ‘due diligence’ studies.

“Twinned holes are typically drilled less than 5 metres apart and are best compared according to geological units, individual samples or equal-length composites. Repeatability of analytical results and bias must be statistically quantified. Compositing of short and or variable length samples into composites of a larger size converts samples into common length data, necessary for geostatistical evaluation. In addition, grouping samples to larger composites (by geological boundary) helps to minimise the noise exhibited by individual samples,” (Abzolov, 2009).

Case Study: Graphite Twinned Holes

QC work carried out on an African graphite project used the process of twinning drillholes to check drilling results (Scogings and Coombes, 2014). Three pairs of twinned holes were drilled to compare the percentage of graphitic carbon (% Cg) in RC vs DD samples. A visual comparison of mineralised intersections in the twinned RC and DD holes suggests an overall similar representation of the mineralisation (see Figure 1 for an example of one pair), especially when the 1 m samples are composted to 3 m lengths to reduce noise. The similarity in assay results is evidenced in a quantile-quantile (QQ) plot comparing the grade percentile

1 CSA Global
from the two data sets (Figure 2). On the basis of the similarity between the RC and DD twinned drilling results, it was concluded that RC and DD data could be combined for resource estimation purposes (% Cg only, not flake size or quality) since there is no significant bias between the twinned drillholes.

Figure 1. Two comparisons of twinned RC (left borehole) and DD (right borehole) showing % Cg assay results. Left hand side pair shows 1 m sample intervals (logged barren intervals in DD hole were not sampled). Right hand side pair shows samples composited to 3 m lengths (Scogings and Coombes, 2014).

Figure 2. QQ plot comparing % graphitic carbon in twinned RC and DD holes (Scogings and Coombes, 2014).
Assuring Quality Data from a Laboratory

Other methods used to verify the quality of sampling and assaying on exploration projects are the use of standards, blanks, duplicates and external laboratory checks.

Standards or certified reference materials (CRMs) are samples of known or accepted value that are submitted to assess the accuracy of a laboratory. A systematic difference from the expected CRM result indicates a bias within or between assay batches.

Standard samples may be purchased commercially or may be prepared internally and it is recommended to submit standards that span the practical range of likely assay values (e.g., % Cg) or market performance tests (e.g., bentonite fluid loss).

Abzolov (2008) recommends insertion of 3-5% of the standard(s) in each sample batch to identify bias, whereas Verly (2012) suggests 6% (Table 1).

Table 1. QA/QC sample insertion rates suggested by Verly (2012).

<table>
<thead>
<tr>
<th>Sample Type</th>
<th>Sample sub-type</th>
<th>Insertion rate</th>
</tr>
</thead>
<tbody>
<tr>
<td>Duplicates</td>
<td>Field samples</td>
<td>2%</td>
</tr>
<tr>
<td></td>
<td>Coarse duplicates</td>
<td>2%</td>
</tr>
<tr>
<td></td>
<td>Pulp duplicates</td>
<td>2%</td>
</tr>
<tr>
<td>CRMs</td>
<td>CRMs</td>
<td>6%</td>
</tr>
<tr>
<td>Blanks</td>
<td>Coarse blanks</td>
<td>2%</td>
</tr>
<tr>
<td></td>
<td>Pulp blanks</td>
<td>2%</td>
</tr>
<tr>
<td>Checks</td>
<td>Check (umpire) samples</td>
<td>4%</td>
</tr>
</tbody>
</table>

Source: Scogings and Coombes, 2014.

Blanks are barren samples with an expected low grade (value) relative to the mineralisation being evaluated and are submitted to check for contamination during sample preparation or assaying of major and trace elements. Although blanks are commonly used during exploration for precious metals, base metals and some industrial minerals such as graphite (Cg), phosphates (P₂O₅) and potash (K₂O, MgO, NaCl), they are less likely to be used for other industrial minerals where market performance is generally more important than elemental analysis.

Blanks are effectively another type of CRM, albeit with very low values of the element(s) in question. Verly (2012) suggests an insertion rate of 4% blanks (Table 1).

Duplicates are samples collected, prepared and assayed in an identical manner to an original sample, to provide a measure of the total error of sampling. There are several types of duplicates possible: field duplicates are collected at the drill rig or trench, while laboratory duplicates may be produced by taking a second split after crushing, before the pulverising stage, or a third split of pulp after pulverising (e.g., 90% passing 75 micron).

Importance of Laboratory Checks

External laboratory checks generally rely on pairs of pulverised exploration samples (also known as umpire samples) to define inter-laboratory precision and bias. It is suggested that at least 5% of samples be tested by an umpire laboratory.

Recent work on a vermiculite project highlights the importance of using an umpire laboratory (Scogings and Coombes, 2014). During the QA/QC process it was noted that an external laboratory used a surfactant that gave artificially low values – this demonstrates how crucial it is to use consistent laboratory methods.

A comparison between 46 samples from an external laboratory and the original samples showed an overall difference in mean grade. A scatterplot between the check samples and corresponding original laboratory shows a significant positive bias towards the original laboratory (Figure 3).
This bias between the sample data sets was also evident on comparative histograms and a QQ plot. Further investigation into the bias highlighted differences in analytical procedures, with the original lab reporting vermiculite inclusive of some entrained impurities and un-exfoliated material, whereas the umpire laboratory process produced a more refined product and hence lower recoveries as is evidenced in the statistical plots. Such umpire data may be used to adjust the original data by means of regression, which is the process of fitting a line to data and applying the best-fit equation, for example $y = 0.82x + 1.43$ (Figure 3).

Drilling With the End User in Mind

Procedures used in a high purity quartz (HPQ) project in Northern Queensland show the importance of selecting the right drilling method to classify the deposit considering the highly specialist end user requirements.

The project owner had requested a cheaper reconnaissance drilling method such as RC for the initial testing and sampling of the deposit area. A review of the market specifications (see Table 2) for HPQ indicated that very high levels of sample quality would be required by the drilling method.

<table>
<thead>
<tr>
<th>Type or Application</th>
<th>SiO2 minimum %</th>
<th>Other Elements maximum %</th>
<th>Other Elements maximum ppm</th>
</tr>
</thead>
<tbody>
<tr>
<td>Clear glass grade sand</td>
<td>99.5</td>
<td>0.5</td>
<td>5,000</td>
</tr>
<tr>
<td>Semi-conductor filler, LCD and optical glass</td>
<td>99.8</td>
<td>0.2</td>
<td>2,000</td>
</tr>
<tr>
<td>‘Low grade’ HPQ</td>
<td>99.95</td>
<td>0.05</td>
<td>500</td>
</tr>
<tr>
<td>‘Medium Grade’ HPQ</td>
<td>99.99</td>
<td>0.01</td>
<td>100</td>
</tr>
<tr>
<td>‘High Grade’ HPQ</td>
<td>99.997</td>
<td>0.003</td>
<td>30</td>
</tr>
</tbody>
</table>

*Modified from Richard Flook and the December 2013 Issue of the Industrial Minerals Magazine (p.25)*

The only drilling method capable of producing the quality of sample required was DD. The project owner realigned their budget and exploration approach based on the highly specific end user requirements.

Recent work on a vermiculite project further highlights the importance of choosing the right drilling method. In this case, the initial exploration on the project had been carried out using the air core (AC)
Drilling method. During the QA/QC phase to check on the quality of the information produced for end user commercial assessment, two of the AC holes were twinned with DD holes.

When plotting QQ graphs for the fractions (in mm ranges) of sieved vermiculite material, it was noted that there was a positive bias for AC fraction 0.425 to 0.71 mm (fines). It was also noted that the bias changes as the size fraction increases.

It was concluded that the AC drilling method shreds vermiculite flakes and reduces their size relative to flakes in core samples and that samples with a large proportion of vermiculite flakes appear to be most affected by the AC method. Hence more extensive DD drilling was recommended to produce the required flake size analysis to satisfy the end users requirements.

Case Study: Drilling for Bulk Density

Industrial mineral resource estimations rely on three main inputs: i) grade, ii) volume and iii) bulk density, of which the latter is often relatively neglected during exploration. A study of exploration drilling work on a sodium bentonite mine in Queensland, demonstrates the importance of selecting the correct drilling and sampling technique for the estimation of bulk density.

The bentonite beds were deposited within a high energy fluvial/lacustrine environment of Upper Jurassic to Lower Cretaceous age. Several bentonite beds have been identified on the property; these range up to ~4 m in thickness and consist predominantly of dioctahedral smectite (montmorillonite) with accessory minerals including feldspar, kaolinite, quartz and zeolite (Scogings, 2014). The beds are capped by volcaniclastic rocks identified petrographically as either tuff or ignimbrite in addition to cross-bedded volcanogenic sandstone.

Measuring the in-situ bulk density (ISBD) of sodium bentonite presents a whole set of challenges related to the fact that such material absorbs water and swells; therefore, direct immersion in water cannot be used with much confidence.

All exploration drilling was carried out by an open hole method known as rotary air blast (RAB) using a bladed bit, which results in small drill chips unsuitable for water immersion or the calliper method. An alternative drilling method was considered in order to measure ISBD and after discussion with the contractor, the RAB rig was modified to drill core (without water) at several strategic locations. On reclaiming the cores, all samples were sealed in plastic bags to retain in-situ moisture before estimating density. The core samples were then trimmed with a hacksaw to yield regular cylindrical shapes from which volumes could be estimated using the calliper method (Figure 4), and moisture content derived from the ‘shavings’. Density values of between 1.72 and 1.84 t/m3 were obtained and it was elected to use 1.8 t/m3 for estimation of in-situ ‘wet’ bentonite resources (Scogings, 2015).

Figure 4. Bentonite core trimmed for calliper method. The core ‘shavings’ were used for moisture analysis.
REFERENCES


ABSTRACT
The software and hardware evolution into smart, connected systems is reshaping many industries. Real-time data streaming and integration is allowing improved insights into the business, and providing smarter decision making. Is the drilling and sampling industry ready for the transformation, and can they make the transition?

Technology stacks are a fundamental part of smart, connected systems, which integrate live data collection, hardware, sensors, data storage, microprocessors, software, and connectivity in many ways. These systems are delivering a new era of operational insight, real-time decision making, improved efficiencies, reduced spending, and corporate-wide reporting. Smart connected systems in the agriculture world are now connecting live data from farming equipment into irrigation systems, as well as soil and nutrient sources with information about the weather and commodity prices to optimise overall farm performance.

In the drilling and sampling business, digital data capture to improve business performance, is not new. However, the level of acceptance varies, and is often carried out as a siloed activity, with little integrated into the entire business workflow. To improve this situation, there needs to be a wider acceptance of streaming live data from drilling and sampling activities into integrated smart connected systems. Data from drill rigs, geological information and laboratories is streamed and integrated in real time into a database. Implementation costs have been substantially reduced to a point where these systems are affordable to all companies. There have been vast improvements in processing power, device miniaturisation, and wireless connectivity, all offer the drilling and sampling process opportunities for new functionality, greater reliability, higher product utilisation, reduced risk, and better insight and reporting capabilities that cut across traditional business processes.

In this article, we look at the nature of smart, connected systems and how they can redefine or improve operationally functionality within the drilling and sampling process. Live streaming and integration of data from contracts drilling, sampling, logging, geophysics, sample analysis and reporting are all hosted in a system of systems providing improved business efficiencies and performance.

WHAT ARE SMART CONNECTED SYSTEMS
Smart, connected systems have three core components:

1. Physical components include the mechanical and electrical parts. For example, with a drill rig it includes the engine block, compressor, downhole tools. With the laboratory, this includes the analytical instruments, and on the geological side it includes the physical sample.

2. Smart components include the sensors, microprocessors, databases, controls, software, operating system and user interface. With the drill rig this includes sensors such as plc/scada systems, laboratory software to capture the results and quality control (QC), data schema to capture and host all geological information.

3. Connectivity components include ports, antennae, and protocols, enabling wired or wireless connections between the products and within the system. Connectivity allows products to communicate with each other and with databases. For example, data from the sensor on a drill rig is transferred wirelessly to a data schema. This data can be streamed to a cloud environment, where the data can be monitored and controlled.

Smart systems improve the capabilities of the physical components, while connectivity improves the capabilities of the smart components. The result is a continuous cycle of value adding.
WHAT CAN SMART CONNECTED PRODUCTS DO

Smart connected systems provide functionality that can be grouped into four areas: monitoring, control, optimisation, and autonomy. Each function is capable in its own right, and also influences the other functions:

1. Monitoring: Smart, connected systems enable live monitoring of integrated data. Drilling parameters can alert users to changes in machine performance. This information integrated with geological information can improve predictive drilling analysis. Monitoring of historic data allows companies to track the operational performance and better understand how best to drill the ground. The collection of QC data from the laboratory can track sample performance, and how best to establish analytical best practice.

2. Control: Smart, connected systems can remotely control products through commands built into the devices. Products will respond to specified changes in their condition, for example, “if pressure gets too high, shut off the valve”. This also allows the customisation of product performance.

3. Optimisation: The flow of live data within a smart, connected systems can help optimise various activities. For example, if drill rigs could be connected to anticipated geology, the drill rig can be better optimised. Algorithms and analytics applied in real time to historical data can improve drilling performance in specific geological environments.

4. Autonomy: Combining the functionality of monitoring, control and optimisation will allow smart, connected products to achieve a certain level of autonomy. A number of drilling companies now run autonomous rigs helping reduce the need for operators, but also more importantly, improving the safety in dangerous environments.

CENTRAL DATABASE

Live streaming of data is growing at an exponential rate in terms of volume, variety, veracity and velocity, as well as complexity and strategic importance. To add value to this data, it is no longer desirable for drilling or sampling data to be managed in siloed databases. To get the most out of data generated, smart connected systems require a data management system that consolidates and integrates data collection and storage, allowing for consolidated reporting and analytics. Good data is fundamental in a smart connected system, providing improved understanding of the metrics of the whole business. Only with good data can drilling and sampling activities have the insight into what good performance looks like. Companies that successfully use good data outperform their peers by up to 20% (EY, 2014).

A common problem with mining and exploration companies is the existence of siloed databases, resulting in a data integration gap (EY 2014). Companies need to move away from these disparate data management solutions, and move to an end to end solution. Each part of the business is optimised, not on its own, but as part of the business system. Addressing the integration gap is key to the business benefits of a smart connected system. This transition to end to end data solutions will involve a change in organisational culture that empowers the workforce and provides improved insight to the drilling and sampling business.

DATA ANALYTICS

Disparate datasets may be valuable, however companies can often unearth useful insights by integrating these drilling and sampling datasets. For example, information from individual sensors, such as a drill rig’s engine temperature, torque, head position and fuel consumption, as well as geological data, can help monitor drill rig performance against engineering specifications. Capturing such insights is the domain of data analytics, which blend mathematics, computer science, and business analysis techniques.

A challenge for smart, connected system is to capture live data in a variety of formats and present the data for analytical analysis. Data from sources such as sensor readings, geology, downhole tools, laboratories need to be stored in a native format and integrated, and then from there the data can be studied with various analytical tools that fall into four categories:

- Descriptive: Capture data from hardware and software.
- Diagnostic: Examine causes of reduced system or product performance.
- Predictive: Detect patterns that signal on impending event.
- Prescriptive: Identify measures to improve the outcomes or correct a problem.
THE NEW TECHNOLOGY STACK

Smart, connected systems require a “technology stack” (Porter and Heppelman, 2014). This stack is made up of multiple layers, including product hardware, embedded software, operating systems, network communications to support connectivity, cloud environments containing a database, as well as an analytics platform, security tools, access for external information sources, and integration with enterprise business systems. The technology landscape of today has made smart, connected systems technically and economically feasible. These include hardware and software performance improvement, miniaturisation, energy efficiency of sensors, highly compact, low-cost computer processing power and data storage.

Some of the challenges for technology stacks is that:

- Devices are always on.
- Operating in real time.
- Seamless connectivity.
- Centralised databases, not siloed.
- System to collect, analyse and store continuous data.

RETHINKING THE DRILLING AND SAMPLING BUSINESS

The powerful capabilities of smart connected systems are making organisations rethink the drilling and sampling business. Functionality of one product is optimised, in association with other connected product, adding value to the data and improving business insights. Integrating drill rigs, geological data collection, and laboratory systems, enables better overall performance of equipment and improved business insights. Departmental boundaries expand allowing enterprise wide visibility of data and business performance.

In the future could we see smart connected systems in mining that connect live grade control data into block modelling systems, with information about the weather and commodity prices to optimise overall mine performance (Figure 1).

Figure 1. Schematic presentation of a Smart Connected System for Grade Control (after Porter and Heppelman, 2014).

CONCLUSION

Smart, connected systems are creating opportunities to add value to the data generated during drilling and sampling activities, and in so doing improve the business process. However, the acceptance of such systems is still in its infancy. Siloed data management is still wide spread through the industry. There needs to be a cultural shift in management to recognising the business benefits of smart, connected systems. The transition to such systems maybe unsettling and destabilizing, however it needs to brought centre stage. Smart, connected systems will improve the efficiencies around the organisation, share software that doesn’t get used much, and to get more out of the products that we already have. And smart, connected products will also allow people to work more productively.

REFERENCES

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ACCOUNTABLE GEO-LOGGING
RAY SLATER¹

INTRODUCTION
The conference theme ‘drilling for geology’ evokes memories of being on-site at a drilling rig logging chips and core, since this task is forefront to obtaining geological data from drilling.

We know drilling remains a key element of exploration, both as an activity and as a cost. From start to finish, a drilling campaign can involve a lot of time and energy and effort. Someone must ensure everything is carefully thought out and executed; also that data generated is comprehensive, reliable, and properly recorded and saved.

That someone is usually a geologist, and as exploration is the biggest employer of geologists, it stands to reason that most of us will be required to produce geological logs of drillholes at some point.

My own experience is mostly coal industry related, however the principles and messages from this presentation apply equally in any field of exploration.

GEOLOGICAL LOGGING
Geo-logging is not difficult, however some fundamental aspects of drilling and logging must be appreciated and understood in order to achieve what I call “accountable” logging. In this context “accountable” means everything must add-up properly!

When logging core, this is as important as the geological content.

Before delving further into this, and since most drilling also involves chip holes or holes which are only partially-cored, it’s important first to briefly touch on chip drilling and the importance of knowing your hole depth before starting to core.

LOGGING CHIPS
Clearly, there are limitations on the level of detail and accuracy possible when logging a chip hole. Chip holes are described by reference to drilled depth, with samples collected and laid out on a metre-by-metre basis.

Under ideal conditions, an experienced, observant geologist can record a drillhole log from chip samples alone which is considerably more detailed and accurate than 1 m sample piles. “Ideal conditions” however are the exception and not the rule!

Collecting and laying-out chip samples usually falls to the “off-sider”. Six metre drill rods are the common standard, and some off-siders lay out samples in rows of six (or multiples of) to correspond to each drill rod, others place the samples in rows of ten to simplify mentally tallying the progressive drillhole depth.

As a rig geologist, you should specify how and where you want the samples placed, but whatever is decided should be kept consistent.

If you must be absent from the rig and later return, don’t automatically assume the total number of samples laid-out on the ground represents the current hole depth, nor that the sample material is representative of the interval intended.

Samples can be missed due to caving, lack of circulation, lack of awareness, and lack of enthusiasm. Samples can also be simply ground-up and washed (or blown) away in the drilling process.

If you’ve just drilled fifty metres through a target formation, at depth, using mud circulation and all you have to show for it are 50 puddles of uniform grey slop that washes straight-through your sieve, how do you know what you’ve actually drilled through and what to record?

¹ Principal Geologist, Ray Slater & Associates Pty Ltd
In itself, missing chip samples may not be a big deal. Fortunately, a geologist’s chip log can often be refined later (in detail and accuracy) by reference to down-hole geophysical logs. Like many processes, drilling is imperfect and 100% recovery of both chips and core is uncommon. This is not a criticism, but it does introduce some variables which become especially relevant when logging core.

**TRUE HOLE DEPTH**

If you’ve begun drilling a hole where you plan to start coring at 158 m depth, how do you know when you’ve reached 158 m depth?

Do you rely on the driller’s advice, or do you independently calculate the drillhole depth yourself?

- How long is the ‘kelly rod’ and did you measure it in order to calculate “stick up”?
- Is stick-up consistent between holes, or even on a single hole?
  - For example, have the blocks under the jacks settled and/or did the driller jack-up the rig further to re-level it since commencing the hole?
- Was there a stabiliser rod in the drill string and did you measure it?
- How long are the drill rods?
  - Did you simply assume standard six metre rods?
  - Did you measure (some/any of) them?
  - Are they all the same length?
- When the ground conditions changed and the driller changed drill bits, did he also change subs, and did you measure it and/or the replacement drill bit?

Rig geologists need to be accountable for knowing these things, and not simply assume.

If the driller tells you he’s cored the first three metres starting from 158 m depth, how is that three metre interval measured?

When the core barrel was run to the bottom of the hole, did you discuss with the driller, and place a mark somewhere on the mast, chains or Kelly rod as a reference point from which to measure progress?

Do you take it ‘as gospel’ when the driller says he drilled three metres? Is this three metre core barrel the same as the one you were using on the last project when you had a different drilling contractor, rig and equipment?

What if there’s a 3.05 m long stick of unbroken core in the inner tube? Do you even know for sure what length of core can physically fit in this “three metre” core barrel?

Core barrels are like cars – there are various makes and models, but they’re all different. One popular HQ three metre wireline core barrel in current widespread use has an inner tube length of 3.135 m. So, dependent on choice of drill bit, it is entirely possible to fit up to 3.20 m of core in that particular core barrel. Compared to 3.0 m, that’s almost a 7% discrepancy, which, over multiple core runs, could tally to a very significant cumulative error!

When a core run is pulled from the hole, the first thing to record is the recovered core length. This is usually measured to the nearest centimetre, but what is the point if you don’t know how much was drilled in the first place!

If you’re required to log the core to centimetre detail, then clearly you must also measure and record how much was drilled to the nearest centimetre.

An off-sider hanging from the mast, reaching up with the brush from the rod grease tin to slap a mark on the kelly is not an acceptable method for marking a reference point against which to measure the amount drilled!

Having measured and recorded the recovered core length, the next task – often – is to photograph the core (prior to disturbing, or removal of any samples). To be useful, core photos need some reference and usually include a minimum of hole number, top- and base-of-run depths, and a measuring scale.
As you log each core run, you might also be collecting and removing samples from the core, using labelled core blocks or core foam to mark the position of removed samples, and/or transferring the core into core trays.

So, ………as your cored hole progresses:

- Are all the run depths shown in your core photos correct?
- Are all your log depths correct?
- Is the labelling of the core trays correct?
- Are the core run interval and depth block markers in the core trays correct?
- Are your recorded sample interval depths correct?

The answer to these questions may often be “NO” because – and despite the best efforts of drillers – perfect core recovery is a myth in most instances.

Lots of factors can contribute to losing core, but less than perfect recovery might render a cored interval useless and is why drilling contracts specify minimum acceptable core recovery.

Lost core might later be recovered, and if you think about this, it is the reason that while ever you are in the process of coring – and until the drillhole or the entire cored interval is completed – you cannot know for certain what the true depth is at any given point along the core.

If this is NOT evident, consider any series of two or more consecutive core runs where a single cored interval recovers more core than was drilled. For example, 3.05 m recovered, against 3.0 m drilled.

The upper 0.05 m of core from that run MUST be moved up into the previous run, and NOT thrown into the mud pit, though I have seen this done!

If in turn that 2nd-to-last run had 100% recovery, then the upper 0.05 m from the 2nd last run also must be moved up into the previous run, and so on and so forth.

Since it’s entirely possible the next or any subsequent core run might itself recover >100% of what was drilled, this whole adjustment process cannot be completed until the drillhole itself is completed (or at least until the cored interval is completed).

LOGGING CORE ‘BY DEPTH’ VERSUS ‘BY THICKNESS’

There is no right or wrong way to log core, but if you use a coding sheet which has only a ‘depth’ field and hence logging everything by reference only to depth, you may need to make lots of corrections once the hole is completed.

This applies not just to the lithology record, but anywhere “depths” were recorded, including core photos, and any other forms or templates such as:

- Drilling record sheets.
- Core run sheets.
- Sample record sheets.

…and similarly for the likes of:

- Core trays.
- Start- and end-depth marker blocks.
- Core run depth marker blocks.
- Sample spacer blocks.
- Physical samples (if depth labelled).

Fundamentally, this is why it makes a whole lot more sense to log core “by thickness” rather than “by depth”.

Most field geology is about observation – when logging core, you can directly observe and measure the thickness of the different units, but you cannot “see” the depths. So my advice is to simply log what you can see – measure and record thickness, and depth will take care of itself later.
I remain a firm advocate for field logging using pencil and paper – this could be coding sheets but irrespective of the format, the original field log data should always be kept and saved.

Once drilling and logging is completed, the “log accounting” (mathematical correction) is an essential part of the process and should be done immediately by the geologist who logged the hole. Not later, and certainly NOT by someone else!

Ask anyone who regularly works with drillhole databases how frequently they encounter simple maths-based data validation errors, and it becomes very apparent that lots of geologists don’t understand that their original recorded depths may require correction, why, or how to adjust and correct their log.

Table 1 shows a hypothetical field log, showing the end of an interval of open-hole drilling, and an initial core run logged and recorded “by thickness”:

Table 1. Hypothetical initial, uncorrected field log.

<table>
<thead>
<tr>
<th>Base Depth (m)</th>
<th>Thickness (m)</th>
<th>Rock Type</th>
<th>Lith Adj 1</th>
<th>Colour</th>
<th>Texture</th>
<th>Sample #</th>
</tr>
</thead>
<tbody>
<tr>
<td>-155.00</td>
<td>2.0</td>
<td>sandstone</td>
<td></td>
<td>light grey</td>
<td></td>
<td></td>
</tr>
<tr>
<td>-156.50</td>
<td>1.5</td>
<td>siltstone</td>
<td></td>
<td>grey</td>
<td></td>
<td></td>
</tr>
<tr>
<td>-158.00</td>
<td>1.5</td>
<td>mudstone</td>
<td>coaly</td>
<td>black</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

*** End of open-hole interval ***

Core Run 1 158m - 161m; 3.0m D; 2.96m R; 0.04m KL (core loss)

<table>
<thead>
<tr>
<th>Depth (m)</th>
<th>Rock Type</th>
<th>Lith Adj 1</th>
<th>Colour</th>
<th>Texture</th>
<th>Sample #</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.08</td>
<td>siltstone</td>
<td></td>
<td>grey</td>
<td></td>
<td></td>
</tr>
<tr>
<td>0.34</td>
<td>sandstone</td>
<td>silty</td>
<td>grey</td>
<td>light grey</td>
<td></td>
</tr>
<tr>
<td>0.56</td>
<td>siltstone</td>
<td></td>
<td>grey</td>
<td></td>
<td></td>
</tr>
<tr>
<td>0.27</td>
<td>mudstone</td>
<td>carbonaceous</td>
<td>dk brown</td>
<td></td>
<td></td>
</tr>
<tr>
<td>0.16</td>
<td>coal</td>
<td>undifferentiated</td>
<td>black</td>
<td>broken</td>
<td>1</td>
</tr>
<tr>
<td>0.45</td>
<td>coal</td>
<td>dull</td>
<td></td>
<td></td>
<td>1</td>
</tr>
<tr>
<td>0.01</td>
<td>coal</td>
<td>bright</td>
<td>friable</td>
<td>2</td>
<td></td>
</tr>
<tr>
<td>0.57</td>
<td>coal</td>
<td>dull, few bright bands</td>
<td></td>
<td>2</td>
<td></td>
</tr>
<tr>
<td>0.06</td>
<td>coal</td>
<td>dull, numerous bright bands</td>
<td></td>
<td>2</td>
<td></td>
</tr>
<tr>
<td>0.16</td>
<td>carb mudstone</td>
<td>coaly laminae</td>
<td>black</td>
<td>soft &amp; puggy</td>
<td></td>
</tr>
<tr>
<td>0.30</td>
<td>mudstone</td>
<td>slightly carbonaceous</td>
<td>dk brown</td>
<td></td>
<td></td>
</tr>
<tr>
<td>0.04</td>
<td>core loss</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

In this first core run, 3.0 m was drilled but only 2.96 m recovered. There is a loss of 0.04 m and this core loss should be assigned; now – not later. You don’t know what might happen in the next core run – or the 3rd, or 4th or 5th, and you should not make assumptions about later recovery of lost core. If you don’t assign the core loss, you immediately have an accounting error in your log.

Assume the next core run drills 2.0 m, with 1.95 m recovered and a loss of 0.05 m (Table 2).
Table 2. Original, uncorrected log of Core Run #2.

<table>
<thead>
<tr>
<th>Base Depth (m)</th>
<th>Thickness (m)</th>
<th>Rock Type</th>
<th>Lith Adj 1</th>
<th>Colour</th>
<th>Texture</th>
<th>Sample #</th>
</tr>
</thead>
<tbody>
<tr>
<td>Core Run 2</td>
<td>161m - 163m;</td>
<td>2.0m D;</td>
<td>1.95m R;</td>
<td>0.05m KL (core loss)</td>
<td></td>
<td></td>
</tr>
<tr>
<td>0.05</td>
<td>siltstone</td>
<td>grey</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>0.29</td>
<td>siltstone</td>
<td>grey</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>0.12</td>
<td>sandstone</td>
<td>grey</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>0.87</td>
<td>sandstone</td>
<td>grey</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>0.62</td>
<td>sandstone</td>
<td>grey</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>0.05</td>
<td>core loss</td>
<td>grey</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Any number of additional consecutive core runs could follow. Each discrete core run could involve core losses, gains, or both, and each must be dealt with on its merits, as it is drilled, recovered and logged.

Core recovery exceeding the amount drilled must be moved up into the preceding core run. This again emphasises the importance of knowing, for each core run, how much was actually drilled.

For the purpose of this example, let’s assume a third and final core run is drilled (Table 3).

Table 3. Original, uncorrected log of Core Run #3.

<table>
<thead>
<tr>
<th>Base Depth (m)</th>
<th>Thickness (m)</th>
<th>Rock Type</th>
<th>Lith Adj 1</th>
<th>Colour</th>
<th>Texture</th>
<th>Sample #</th>
</tr>
</thead>
<tbody>
<tr>
<td>Core Run 3</td>
<td>163m - 166m;</td>
<td>3.0m D;</td>
<td>3.10m R;</td>
<td>0.10m core up</td>
<td></td>
<td></td>
</tr>
<tr>
<td>0.10</td>
<td>sandstone</td>
<td>grey</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>0.32</td>
<td>sandstone</td>
<td>grey</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>0.56</td>
<td>sandstone</td>
<td>grey</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>1.24</td>
<td>sandstone</td>
<td>grey</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>0.56</td>
<td>siltstone</td>
<td>grey</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>0.32</td>
<td>mudstone</td>
<td>grey</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

In this final core run, 3.0 m was drilled, but 3.10 m of core was recovered. The uppermost 0.10 m of core from Run 3 must therefore be moved up into Run 2. In turn, that means Run 2 now has >100% recovery, core loss is eliminated, and the topmost 0.05 m must also be moved up into Run 1. When it was recovered and logged in Run 1, there was an assumed loss of 0.04 m. That loss is now accounted for, but there is still another 0.01 m “leftover”.

If you’re satisfied and confident with your measurements, in order to properly correct and reconcile your log, the start depth of the first core run must be amended to 157.99 m.

THE IMPORTANCE OF ‘LOG ACCOUNTING’ (LOG CORRECTION)

I’ve deliberately chosen the numbers in this example to yield this trivial discrepancy, and it could be argued that any similar errors can be accounted for simply by measurement error or not allowing sufficiently for expansion of core due to natural and drilling-induced defects.

Both possibilities are very real and very true, but nevertheless this is a perfectly reasonable example of what can and does happen, and demonstrates the need to correct and adjust logs, and also illustrates why the ‘log accounting’ process can only be done upon completion.
In this example, it also means that the very last open-hole chip interval logged must be corrected from 1.5 m to 1.49 m thickness, and reinforces why (in this example) every recorded core interval, if logged ‘by depth’ will be wrong and must also be corrected.

This log correction process is vital, and although seemingly trivial, a discrepancy of just 0.01 m can still cause data validation errors if transposed from a coding sheet or migrated into a database from a logging software application.

Note that ‘log correction’ in this context is separate, and wholly unrelated, to the process of correcting to geophysical logs.

Finally, and in closing, if we acknowledge and accept that drilling is an imperfect process, we should also acknowledge that geology itself is interpretive and often imprecise, but this is no excuse for a lack of accuracy and accountability when it comes to geologist’s drillhole logs.
ABSTRACT

In the last few years, design improvements in reverse circulation (RC) percussion drilling sampling systems have reignited the debate on the usefulness and appropriateness of samples collected by RC drilling. Where RC drilling has been sometimes criticised as being a suboptimal drilling technique for the collection of quality data for use in mineral resource estimation work, new RC sampling systems now promise that bad sample splitting and poor sample recovery are problems of the past.

However, having a good RC sample splitting system doesn’t necessarily mean a good sample is collected. In particular, the skills of the RC driller to provide a consistent sample to the sample splitter are just as critical as the sampling system.

In this paper, the authors describe cost-efficient ways to ensure the quality of drilling and sampling through well-designed standard operating procedures (SOPs), which incorporate practical and user friendly systems. Such procedures permit RC drillers to better understand the implications of drilling actions on sample quality, and facilitate better communication between the drillers and geologists.

Combining sample-drilling quality metrics with production metrics such as rig availability and rig efficiency, allows the drill rig managing geologist to manage and improve the sample quality on a daily basis. For example, real-time control plots can be used to identify when something in the sample quality is likely to be out of order (instead of the resource geologist finding such issues a week or month later). Problems identified this way can be discussed between driller and rig geologist, as they occur, and immediately corrected.

In this paper, examples of such a quality monitoring system are provided along with examples of how the system has improved metre delineation, sample recovery, sample splitting, water and dust management in exploration drilling programs.

INTRODUCTION

Each year in the Australian mineral industry between 2,000 km and 3,000 km is drilled for mineral exploration, of which RC drilling is the major component (Australian Bureau of Statistics, 2017). Most exploration professionals consider that RC drilling can provide a good-quality sample that is cheaper and faster to collect than diamond core (DC) drilling samples, albeit the RC drilling does not have the same high sample quality usually attributed to DC drilling.

In the last few years, design improvements in RC drilling sample-splitting systems have reignited the debate on the usefulness and appropriateness of samples collected by RC drilling. While RC drilling has been criticised by some as being a suboptimal drilling technique for the collection of quality data for use in mineral resource estimation work, new RC sampling systems promise that bad sample splitting and poor sample recovery are problems of the past.

However, having a good RC sample splitting system doesn’t necessarily mean a quality sample is collected. The ability of the RC driller to provide a consistent sample to the sample splitter are just as critical as the sampling system.

Significant opportunities for sampling quality improvement are often missed because:

- The quality of the sample, which is the most important aspect of the entire drilling exercise, is rarely included as a deliverable in any drilling agreement.
- The two main parties (the geologists and the drillers) often communicate poorly, due to either lack of experience and/or incomplete understanding of each other’s requirements.
• There are no real-time systems in place to adequately monitor quality or performance of the drilling and sampling.

The goal of this paper is therefore to review how the quality of samples delivered by an RC rig can be improved through the incorporation of the latest updates in sample-splitting technology with robust sampling procedures and processes.

THE DRILLING CONTRACT

Quality sampling starts with a good understanding between the two parties on the exploration program requirements. Quite often, for smaller exploration companies, the drilling contract between driller and explorer is often an afterthought of a programmed design, and is often a standard contract quickly cobbled together from versions from prior works. Most drilling contracts between larger companies and drillers span many pages, with the contract’s primary focus on production management, safety, and legal clauses.

However, a drilling contract clause that specifically discusses sample quality is seldom included. Such a clause should always be included into the drilling contract, to bring sample quality into the contract discussions, and to create a framework to discuss the quality of the results. In this way, there is a clear understanding that the primary service to be provided is the quality of samples, not sample production rates.

A sample quality clause can be included in an appended schedule and should include expectations on:

• Sample recovery
• Sample delimitation
• Sample extraction
• Dust control
• Sample splitting accuracy
• Water control
• Collaring practices
• Hole tolerance guidelines
• Sample splitting precision
• Drillhole deviation

Defining sample quality constraints in a drilling contract can be challenging. However, with the use of clear and unambiguous wording, frequent communication between drillers and geologists, good monitoring systems, and an open and positive approach to sample quality, the net benefit in terms of sample quality will always be positive. The cost-benefit of the small financial investment required to implement quality sampling systems is readily demonstrated in all exploration endeavours.

COMMUNICATION

A contract agreement regarding sample quality only works well when the rig geologist and driller can work collaboratively. Quite often, the geologist in charge of a RC rig may be young and inexperienced, and both geologist and drillers may lack specific training in project management communications.

A rig-management geologist may not fully understand the subtleties and intricacies of RC drilling, unless the geologist has significant time at an operating RC drill rig drilling in a variety of ground conditions. As such, the inexperienced geologist will not understand how subtle changes in ground conditions can affect sample quality. Unfortunately, most geologists who do build up practical knowledge regarding RC drilling often end up being promoted out of the field to behind a desk before they have transferring this knowledge onto the next generation of young rig geologists.

Importantly, geologists and drillers need to discuss the issues that may affect sample quality before the drilling program starts. Specifically, there needs to be a pre-agreement on what criteria defines a quality sample, which comes from mostly good communication.

Initiatives that facilitate a better understanding and communication regarding exploration drilling include:

• Training of geologists before they are put in to the position of managing a drilling program. Such training should include theory on quality assurance and control, the workings and drilling principles of an RC rig, explanations of various important components of a rig, and drilling/sample recovery situations commonly encountered in different ground condition. There are several industry service providers that can provide this type of training if in-house expertise is unavailable.
• All drilling programs should include:
  – An office-based commencement workshop for the drillers and geologists, where the program objectives are explained, the expected ground conditions are discussed, and access and logistical issues are made clear. Later in a field-based workshop, the driller should explain how the rig works and key safety issues related to the drilling process. The more detail, the better, such as demonstrating to the geologists a dry-run of a rod run-through of some possible sampling scenarios. The driller should also collapse the rig mast so the geologists can inspect the sampling system from bit to sample bag through the whole RC sampling collection system.
  – Weekly quality meetings during the program, where drillers and geologists meet to discuss sample quality aspects along with data and graphs of quality monitoring results.
• The implementation of a quality management system that helps facilitate communication as described further below.

RESOLVING RC SAMPLE QUALITY ISSUES
Creating a monitoring system for the quality of the samples is only possible if the sources of potential errors are understood. The following sections discuss some of the primary sources of errors.

Quality of the Primary Sample
As discussed above, specification of the expected quality of the primary sample is very seldom addressed in pre-program discussions or drilling contracts. Often, geologists, when preparing post RC drilling reports, will prepare many pages of graphs regarding the performance of laboratory quality standards, but then fail to include a single word on the monitoring of the quality of the primary sample. At best, report discussion will include the quality of sampling from the rig splitter, with the quality of the primary lot implied to be good. Importantly, the primary lot is collected at the bottom of the drill string, where the hammer is breaking up rock over a designated interval.

The importance of the primary lot point is acknowledged in “Table 1” of the JORC Code (JORC, 2012). The first entry in Section 1 of Table 1 requires a Competent Person to discuss the quality of the primary sample (although the layout of Table 1 makes this an unnecessarily confusing task). However, too many practitioners only include information on the RC sample splitting (for example the ‘field duplicate’), and frequently include information regarding laboratory preparation and analytical processes, which should be described in other subsections of Table 1. These reporting trends demonstrate that the industry is not focussing on the quality of this primary sample and to some extent there is confusion as to how to discuss the primary lot under JORC Code guidelines. The key omission here is that the largest sample errors occur at the primary sampling stage.

Mineral resource estimation (MRE) is based on interpolating or extrapolating assay results from drilling data to estimate the respective grades of larger (mine planning) blocks. The assay point data are derived from the RC subsampling, which often reflect one-metre down-hole drilling intervals. Explorers not immediately familiar with estimation process may not always appreciate the fact that ultimately, for example that a 30 gram fire assay result may be used to inform the grade estimate of >750,000,000 g mining block. Given these order-of-magnitude mass differences between assay results and block, it is intuitively important for this assay to be representative as possible of the primary, meaning the sample accuracy and precision are suitable for block grade estimates.

Sampling situations, issues and problems can be discussed using the standard nomenclature developed by Pierre Gy in the 1960s (Gy, 1979). For splitting processes the sampling errors pertinent to RC subsampling are the Extraction Error (EE), Delimitation Error (DE), and Preparation Error (PE) as discussed in the subsections below.

Sample Delimitation Errors
Ideally, the assay resulting from the subsampling and analysis of a one-metre down-hole drilling interval is considered somehow representative of the actual metre drilled. Usually this means that the assay result is within what is deemed to be an acceptable error variance from the true (and unknown) primary lot grade and that there has been no significant bias in the sample collection of the targeted primary sample lot.
Compared to DC drilling, RC drilling has lower boundary resolution because RC sampling on fixed intervals means that the sample interval often transects geological and grade estimation domain boundaries. Specifically, with DC drilling – sample intervals can be selected at key geological contacts, but with RC drilling, the sample delimitation is controlled by the regular sampling interval. As such, RC drilling may not be suitable method for narrow tabular mineralisation because the sample DE of the planned sample is not representative for this type of deposit.

Even when RC sampling is considered appropriate for the mineralisation under consideration, each sample is expected to be (exactly) a one-metre interval. However, if attention is not paid to the advancing drill string a DE will occur and the resulting subsampling will not represent a one-metre primary lot. The outcome of such delimitation errors can be an increase in the data variability due to the difference in sample support between samples. For confidence in MRE work the assay results should reflect the true variability of the one meter intervals, not the inflated variability that occurs when short samples are introduced in the MRE database.

DE problems are common on RC drilling programs when:

- The metre marks are often not properly delimitated on the rig pull down chains and the systems in place to mark the end of a metre are based on general visual judgments by drillers or off-siders. Marks on the mast or chain are often quickly obscured by dust and/or grease and difficult to see.
- Inattention by driller in observing the meter marks when samples are dumped to the RC cyclone.
- Drilling practices such as slightly over drilling the last metre of a drill rod to protect the hammer resulting in the sample support of the last-rod-sample being say 1.1 m long, while the first-rod-sample of the subsequent drill rod is 0.9 m long.
- The sample bit is worn and does not achieve a hole that meets the expected hole diameter.
- Blowing out the hole too vigorously between rods resulting of hole wall cavitation and (over) delimitation error.

These are common issues and the resulting delimitation errors are often underestimated and ignored.

Sample Extraction Errors

As the rock is broken up at the hammer down the hole, the stream of air and sample cutting enters the RC inner tube due to built-up air pressure in front of the bit shroud, which is designed (largely) to prevent the air from the outer tube of the RC rods from escaping between the rods and drill hole walls. The sample stream then proceeds through the sampling system (cyclone and splitter system) to end up in the assay subsample bag(s) and reject bags. Ideally, all the rock fragments from a single metre of drill advance should come up through RC inner tube, and end up in the bags. When some part of or size fraction of the sample material does not end up in the sample bag, then an EE occurs. Like the DE, the EE may create an unacceptable bias and/or inflate the variance of the MRE data set.

EE can be introduced in RC drilling through:

- **Loss to outside return.** A worn shroud, too-narrow rods, or excessive tolerance between shroud diameter and bit diameter may lead to poor sealing of the air stream and instead of the all the sample stream returning through the inner tube, some fraction is returned to surface outside the rods. A 3-mm diameter difference between bit and shroud generally is the recommended maximum tolerance for good RC sample recovery. However, some drillers intentionally set greater tolerances to prevent bogging and increase drill advance rates by ‘wasting’ some of the primary sample. Drillers rarely communicate this information with the rig geologists and such sample quality trade-offs should be agreed before the drillers make this decision.
- **Lack of air pressure, worn O-rings, lack of rig compressor capacity.** Any of these conditions may result in a situation where there is insufficient air pressure to lift all the primary sample up the rod string during the metre advance. Typically, insufficient air pressure results in the primary sample being excessively re-ground and pulverised at the hammer, with sample losses material by passing the shroud to the outside return or excessive fines lost from the RC cyclone vortex finder.
- **Not allowing time to let samples clear the system.** At the end of each metre, time is required for all sample cutting to travel from the bit to the inner tube and the sampling system. Removing the sample
bags right on the time metre mark means some sample from that interval will still be travelling to surface and will inevitably end up in the next sample.

One solution is to pause briefly at the end of each sample interval to allow the sample to properly clear the system before the next metre is drilled. The pause interval needs be increased with the depth of hole and a simple calculation of air flow rates, pipe diameters and depth can give an indication of the time required. However, this requirement will reduce the average drill hole advance and is one of the contentious issues that should be dealt with at the drilling contract stage.

Some explorers' operating procedures require the driller to pause for up to 10 to 15 seconds at the end of each metre sampling interval and to “pull the rods of off bottom” and let the hammer “fire out”. This process means that the RC hammer will lose contact with the bottom of the hole and will quickly lock and the air flow should clear through the sample system of cutting. However, most drillers will not like to do this procedure because:

− The process can damage the RC hammer or the bit rim ring as the bit design is to hammer against some support not for the piston to fire against open air.
− When the bit is locked and not hammering, hole excessive air is lost through outside return and when the bit is replaced to the bottom of the hole, the pressure must be re-built up at a cost of increased diesel consumption.
− When the hammer face is returned to the hole face unplanned and excessive hole deviation may occur, especially in the case of using a large shroud tolerance.

A preferred solution is to let the hammer fire out at the end of each sampling interval but keep the bit rested on the bottom to keep the air pressure high and minimise the risk of bit damage. It requires significant experience to get this right.

Wet Drilling: Sample Preparation Errors

RC sampling is designed to occur under dry ground conditions. Compressed air powers the hammer and carries the drill cuttings to surface through the inner tubes in the RC drill string. Ideally, injecting very high air pressure down the hole creates an air pocket ahead of the drill shroud, so that each drilling interval is kept dry, even if the sampling interval is below the groundwater table. However, when a new rod needs to be added and the bit is below the water table, the temporary reduction in air pressure may result in flooding of the hole with groundwater. In this situation, the drilling re-commences under dry drilling conditions by first using a blow-down valve, where water is cleared from the hole through an outside return by a blast of high-pressure air before the next sampling interval is commenced.

Even with good-practice drilling, it is inevitable that water enters the hole when water-saturated broken-ground conditions are encountered, and/or the available air pressure is insufficient to drive water away from the advancing bit. While it may be possible to increase air pressure to keep the hole dry there is a risk that a too-high pressure could blow out the collar seal and the hole would have to be abandoned.

In the case of water inflow due to local conditions of broken ground, it may be possible to drill beyond the broken zone (producing wet samples) and then continue dry drilling as normal. In such situations, the geologists and drillers need make sure that the sampling system is appropriately cleaned of mud build-ups at the first available rod-change, and they should note that the condition of the sample collected is likely to be of poor or suspect quality.

In the case where significant continuous water inflows are encountered, the driller may need to secure collars by cementing them. This adds significantly to time and cost, and does not always outweigh the benefit of the occasional wet samples returned from the bottom of some drill holes. This is where discussion regarding expected ground conditions (particularly expected water inflows) before the start of a drill program and each drill hole are valuable.

Even though some authors consider that good quality samples is possible under wet drilling conditions (Carswell and Sutton, 2014), PEs always occur when RC drilling happens under wet conditions due to loss of fines in the slurry water. The best solution here is to switch to DC drilling when RC drilling is clearly not meeting quality sample requirements.
Quality of the Secondary Sample (Rig Split Sample)

After the sample cuttings have passed through the inner tubes and the sampling hose, the sample stream exits the RC cyclone to the sample splitting system. The primary sample mass then needs to be reduced to provide the laboratory with a manageable sample mass from the primary sample lot which is typically 35 kg to 40 kg for one-metre drilling interval. A manageable mass is typically in the range 2 kg to 4 kg.

There is a variety of RC subsampling systems on the market, and all require a cyclone to separate the high-pressure air from the particulate sample cuttings prior to subsampling (Figure 1). Importantly, a cyclone is effectively a particle size separation device and the micron sized fines from the sample stream are lost from the top of the cyclone unless specific measures are implemented to catch these fines. The coarse particles exit at the base of the cyclone.

Figure 1. A classic rig-mounted static cone sampling tower.
The sample cuttings are usually collected into a cyclone dump box, before this primary lot is gravity-fed into the splitter. Splitters typically used in RC subsampling are commonly static cones, rotary cones, or tiered riffle splitters. These sub-sample splitting systems may be mounted on the drill rig, on a separate trailer, or simply mounted on the ground as separate cyclone and splitting devices.

In the first sample splitting stage, several sampling errors can and usually do occur. The main argument against the use of RC drilling for sample collection for MRE work is that even under optimum conditions, a sample bias will always be present when sample fines are lost from the sample collection system. Conventional sampling systems such as the popular static cone splitters always produce a biased sample due to the loss-of-fines effect.

**Sample Delimitation Errors**

Sample delimitation errors are common on static cone and tiered riffle splitters and per the theory of sampling the designs of both these splitters, particularly the static cone, are incorrect. Notwithstanding the incorrect designs, these splitters also introduce additional biases due to improper feeding of the sample stream to the splitter and split and maintenance of the equipment.

For a static cone splitter, the sample stream should be uniformly distributed when delivered to top of cone as one part of the sample stream is directed to the laboratory subsample. A uniform distribution from a cyclone dump-box is problematic regardless of the dump-box door design (butterfly, sliding, single-flap) as the sample collected in the dump box will always be delivered preferentially to the cone, effecting a DE bias, particularly if the material in the dump box is segregated. Additional bias will occur when the splitting system isn’t perfectly level. Commonly, regardless of how the sampling system is mounted, static come splitter have high between subsample mass variability and exaggeration of the variability of the MRE data and possibly a systematic grade bias affect.

For tiered riffle splitters, most sample feed designs always incorrect with designs releasing material too quickly onto one side of the riffle splitter, choking the device and introducing biases due to sifting segregation. Almost all riffle splitters mounted on an RC sampling system are tiered, comprising three or four tiers of riffles. Tiered riffle splitters assume mixing of material between tiers, which is an unrealistic assumption and are therefore incorrectly delimited.

Sample delimitation errors occur with incorrect operating procedures of the equipment. Often, after years of usage, the sample shoots on the splitter, which regulate the size of the sample, may be blocked or damaged. Using a crowbar, or other device to force the cone splitter sample shoot sliders to move, may lead to bent chute blades and a biased sample. The two opposite cone splitter chutes are commonly operated like scissors and bending one side will lead to the two chutes giving a different split weight. Older sampling systems that are not well-maintained may often have either the knife valve or the lower cyclone dump box doors stuck and drillers may drill with these doors open. This practice will result in sample segregation of coarse and fine particles as the vortex airflow created in the cyclone will likely generate a preferential path for the material, with the result being a more substantial bias of one side of the splitter over the other.

Geologists need to communicate clearly with the drillers before contract commencement as to what sample systems are to be used and how the systems are to be maintained. The geologist must inspect the systems before the rigs and sampling systems are mobilised to site and should seek independent expert advice on mechanical issues that they may not have the skills to assess. There should also be a clear understanding on how sampling systems are to be used.

Sample delimitation is the primary focus for improvement of a new generation of RC sampling systems, such as the Metzke Splitter™ (Metzke, 2017) and the Progradex PGXTM sampling systems (Progradex, 2017). Such systems come with a price and before they are widely adapted and further finetuned by the industry, it pays for the geologist to match the purpose of the program with the targeted quality of the sample: it may not be a problem running a standard cone splitter for a greenfields base-metal drilling program, but a coarse gold resource definition drill-out will require optimum sample quality control.

**Sample Extraction Errors**

Similar to the primary sample, extraction errors occur often at the splitting stage. The following errors occur commonly and should be avoided where possible:
- Loss of sample as dust. As discussed above, dust loss is the main sampling flaw in RC drilling. The RC drill bit pulverises the sample at the bottom of the hole and a significant amount of this material travels up the sampling system as very fine material, smaller than 100 µm. As it enters the cyclone, the larger particles will travel down in the cyclone but a significant mass of dust will escape out the top of cyclone unless measures are taken to collect this material (Figure 1). Styles of mineralisation where the valuable material is preferentially contained (or depleted) in fines, will significantly bias the samples.

There are four ways to deal with dust. Each has its benefits and drawbacks:

- Bulk Dust collection can be done with dust sampling tools that are linked in with the normal sampling mechanism. The problem is that the dust collected cannot be attributed to a specific metre interval, and as such assaying the dust component only gives post sampling insight into the level of bias occurring due to dust loss.

- Metre-specific dust collection which requires a specialised sampling systems such as the Progradex sampler, which collects the cyclone dust for each sampling interval with the rest of the sample. While this is theoretically a good solution, the sampler is relatively expensive, leading to it not being widely adopted by exploration drilling contractors.

- Dust suppression, which involves spraying the dusted air with a water mist as it enters the sampling system. The concept here is to introduce sufficient moisture to make sure that the fine particles agglomerate and create particles heavy enough to gravitate through the cyclone into the splitter, but to not cause particles to stick to the cyclone lining and other parts of the sampling system. As such, dust suppression rarely meets the concept idea, and the geologists needs to be aware of the limitations and risks. Geologists need to talk to drillers to find out how much moisture they are applying and inspect the sampling system for build-up and clogged at the end of each rod drilled.

- Slurry creation, which involves adding sufficient water to the return sample stream to create a slurry of material, which can be sub sampled using a correct slurry splitting device such as a Vezin-style sampler like the Metzke rotary splitter. However, this then creates the problem of dealing with wet samples which must be dried before further sample preparation and possible loss of fines in the slurry water draining from calico bags if used.

- Sample left behind. As mentioned above, excessive dust control water, wet drilling, or drilling sticky materials may lead to material adhering to the walls of the sampling system. This means that some sample is left behind and doesn’t end up in the sample bag and can lead to worse biases when sample build up blocks up the sample system passages. A good indicator of the degree of sample builds is the presence and amount of layer-cake material that falls to the ground below and around the splitter, after the device has been inspected and scraped clean at rod change.

Sample Preparation Errors

The geologist should be aware of the common preparation errors that can occur with the sample splitting. Notably:

- Contamination between samples. Sampling systems get clogged up due to wet sampling, which causes contamination between samples. And samples cross-contaminate of proper metre delimitation isn’t adhered to (e.g. not pausing in between metres, see above)

- Preparation errors due to design flaws. Some splitting systems are not designed properly, for instance: the use of wide-rimmed sample chute edges instead of knife-edged ones (creating sample build-up and cross-contamination between samples), or single-flap valves to open the cyclone dump box (not allowing the sample to fall over the splitter from a central point and uniformly).

- Preparation errors due to system meddling. Ad hoc solutions to fix problems with sampling systems often create sample preparation errors. For example, changing the cyclone airflow by restricting it or otherwise modifying it, or not using the cyclone dump box the way it is intend to, can all cause errors.

- Preparation errors due to mishandling of the sample. Off-siders should take care to remove the calico sample bags from the sample shoots, make sure that material from large bags doesn’t spill. Each off-sider should follow the same process when handling the sample bags.
**Preparation errors due to incorrect sample bags/numbers.** For example, using the incorrect sample bag so that the sample is assigned to the wrong depth. This requires diligent checking and cross-checking during drilling to ensure that the samples are in the correct order and that the correct bags are being used by the drillers. Bar-coding of sample bags with bar-code readers used to ensure that the correct sample bags/numbers are being used is one method, albeit expensive and labour intensive, of minimising these types of errors.

Recent developments in sample splitting for RC rigs include the development of systems that can collect the entire sample, including the fine dust, in a theoretically sound manner (the Progradex sampling system (Progradex, 2017)), as well as systems that are less prone to delimitation errors and can deal with wet samples (Metzke Splitter (Metzke, 2017)).

**A PRACTICAL QUALITY ASSURANCE AND QUALITY CONTROL SYSTEM**

**Quality Assurance**

The most cost effective way to reduce sampling errors is for geologists and drillers to understand how sampling errors occur and then preventing the errors by implementing well-designed quality assurance (QA) processes. As discussed previously, drillers and geologists should discuss the quality issues mentioned above and include expectations in the drilling contracts. Good communication and discussion of the various sources of errors should happen before the drilling, so that all parties understand the expectations, as well as during the drilling. An ongoing discussion is required to keep refining the quality framework.

Drill contractors will aim to prepare the lowest cost bid to win a drilling tender and geologist need to consider that such bids may not consider high sample quality unless the drillers are informed of the specific quality requirements before the bid is submitted. Well-designed standard operating procedures (SOPs) can be used as a guideline for expectations on sample quality and can make sure everyone in the team, from rig geologists to field technicians, understands how the various tools and practices work.

**Quality Control**

Good QA is about error prevention, but a good quality control (QC) system is also required to correct errors as they occur. QC tools include the checks and balances that are used to measure the performance of the sampling system with a feedback loop for changes/improvements to be implemented as the sampling process is happening. For instance, at the laboratory, where the QC concept is well-engrained, such tools include the insertion of standards, blanks and duplicates, so that process control can be monitored, corrected as necessary and kept as a record that the laboratory assaying process are always in control.

The same principle can be applied to RC drilling, where subsample and/or reject bag masses can be used to give important information on sampling consistency. Most delimitation and extraction errors influence the subsample weights and therefore they can be used as a good proxy for drilling quality.

Note that the purpose of QC is real-time error correction, not post-drilling reviews weeks or months after the process has finished. Therefore, the bag weights data needs to be controlled as the rig is drilling, as well as reviewed at least daily, rather than handing result over for review at the end of the program.

**QC on Primary Sample**

Assuming relatively uniform rock types and density, dry drilling, uniform hole diameter, each drilled metre should result in the same expected sample mass. If that mass is higher or lower than the expected mass, then a sampling error has likely occurred to cause mass variation. For instance, if the mass of a metre of sample is 35 kg instead of an expected 40 kg, then it is highly likely the interval was not properly delimited, or there has been a substantial loss of sample during extraction. However, for the case of a 39 kg sample return, it is likely that mass variance is simply an acceptable (normal) random error that can be expected in any well-controlled physical process.

An effective QC process is to plot the total mass of each successive metre sample on a control plot, in a similar manner to the typical laboratory certified reference material control. Control limit lines can then be added to the plot using a moving range (MR) approach (Sterk, 2015). Samples having masses that plot outside the upper and lower control limits are outliers and indicate that something in the sampling process...
likely to be out of order and needs to be investigated. Equally, two out of the last three points above/below two MR-standard deviations, four out of the last five points above/below one MR-standard deviations, eight consecutive points on one side of the mean or target value, all indicate that something is likely to be wrong with the sampling system.

The approach described above implicitly assumes that the error in weights are normally distributed due to random process. However, a variety of rock types are encountered in drilling a hole with both fresh and weathered conditions, changing bit sizes down the hole, and samples can also be damp or wet, depending on ground conditions. As such, data in the sample mass control plots are automatically corrected for known or assumed densities, bit diameter changes, and wet samples are excluded from the analysis. The primary samples are weighed in total, including the bulk plastic bag and original and duplicate calico sampling bags. The data is entered into the control spreadsheet by a sampling technician as the samples are collected at the rig, so that there is a real-time monitoring process. The technician will enter any relevant comments, so that it is clear to end users of the data that a certain sample mass was deemed defective for a particular reason such as water in the hole, a blow-out or cave-in, collar debris re-drill, etc.; and, that the data should be excluded for MRE work for this reason.

Figure 2 is an example of a sample mass control plot where a "normal" pattern of variability can be observed in the wavering blue line in the top graph of the primary sample bag masses. The samples of weathered rock sampling intervals are not included in this analysis but can be treated separately if required. Drilling in this example hole was mostly dry, so not many samples wet were excluded from the analysis. A lot of this mass variance may well be "reasonably expected" for this type of drilling, and the results are consistent with largely random controlled variability. However, the samples automatically highlighted with yellow circles in Figure 2 indicate sample masses that are statistically unlikely due to just random variability, and such samples needs investigation and explanation. In this example, the corrective actions a rig geologist should implement are: first a discussion with the driller to learn if they are one-off causes due to drilling approach; and, if no cause can be found, drilling should continue under more scrutiny regarding sample quality. If a second outlier deviation occurs, this signals a need to stop drilling and inspect the drilling and sampling system.

Figure 2. Quality control sheet for RC drilling sample mass data. The first graph (blue shade) is a plot of primary sample masses on the vertical axis plotted against time sequence. The second graph (green shade) is a plot of the relative difference weights between the routine and duplicate samples. The tabulation below the charts is a record of the data collected and includes recovery estimates adjusted for each rock type and other systematic variations expected in the drillhole. Red and yellow dots identify special cause variation (too many points on one side of the mean and points outside warning lines resp).
Sample mass control plots also permit the monitoring of trends. A trend of higher sample masses for samples collected from the last sample in a drill rod run is common, which may indicate delimitation errors as discussed previously in this paper. Any mass trends can be further investigated by group-averaging the masses of 1st, 2nd, 3rd, 4th, 5th, and 6th metre of every rod and plotting these as a bar graph as depicted in Figure 3. In the example in this figure, there is a significant metre-delimitation problem that needs to be resolved, as metres are getting progressively heavier towards the end of each rod.

Figure 3. Average bag weights of 1st, 2nd, 3rd, 4th, 5th and 6th metre of each rod for one day of drilling (single driller on single rig).

With regards to expected mass accuracy of each primary sample, the total estimated recovery per metre can be calculated for each metre, and where this metric is higher or lower than expected, this mass bias be discussed with the driller so as to identify issues such as too much material being lost to outside return from excessive shroud tolerance or dust losses from the top of the cyclone. An estimated total sample recovery of 90% or better should be routinely achievable.

QC on Rig Split Sample

The mass monitoring system to check for delimitation and extraction errors can also be used to monitor the primary lot subsamples to provide a QC system for the sample splitting process. Ideally, a splitter should have zero mass difference between routinely collected sample (the original) and a duplicate sample collected from the same primary lot. If the splitter is providing uneven mass splits, then the difference in the routine-duplicate subsample mass will be highly variable.

The routine-duplicate sample mass differences can be measured and plotted in exactly the same way as the primary lot sample masses. Any samples having masses the control lines can signal splitting issues such as sample hang-ups, blockages and/or misalignment of the splitter. Real-time monitoring of the mass difference data offers the opportunity to make immediate corrections, and improve the quality of the sample splitting throughout the drilling program or drill hole under consideration.

In the example shown in Figure 2, the split-mass data reveal some suspect trends, with two periods having sample split masses that are clearly biased. These cases signalled the need to open up and clean the sampling system. Again, the sampling technician included comments in the data input spreadsheet so that end users of the data (such as the resource geologist) can decide whether to exclude or increase the risk weighting of these biased samples in MRE work.
Note that the split mass QC process requires the collection of a duplicate samples for each metre drilled. However, not all the duplicates need to be submitted to the laboratory and only a designated proportion of duplicates need be submitted, and the duplicates can be selected from likely mineralised material rather than collecting duplicates from obvious waste zones. The additional duplicates collected do have the cost of an additional subsample calico bag but this approach creates a resource of additional duplicates that have already been split using the same splitting tool and under the same conditions as the original samples. Logistically, this approach has advantages for future resampling, especially when the sample rejects are discarded. As such, the small extra cost of the calico bag is therefore easily justified.

Every QC system comes with its limitations and QC monitoring should not be used as a tool designed to immediately criticize a drillers skill and performance. In the author’s experience, most drillers are interested in understanding what impacts their actions have on the sample quality, and are keen to improve their drilling skills with the use of a positive feedback system. It is good practice to provide drillers with the control plots from each day of drilling, so they can review and discuss the results with the geologist before the start of their next shift as well at weekly toolbox meetings.

**MEASURING PRODUCTION VS QUALITY**

Given the trade-off between quality and production, it is useful to capture this balance in graphical format, so that better decisions can be made by drillers and geologists on a daily basis.

From the drilling contractors perspective, RC drilling performance is measured as the total metres drilled per day. This metric is a measure of the driller's general efficiency including the driller’s ability to get ready for the day, conduct their pre-checks, have fuel and water solutions at hand and not causing down-time, handle the ground conditions, and drill metres. More useful efficiency metrics include:

- **Daily Drilling Efficiency ratio**, measured as:

  $$\frac{100\% - \left(\frac{\text{Work Time} - \text{Down Time} - \text{Standby Time} - \text{Drilling Time}}{\text{Work Time} - \text{Down Time} - \text{Standby Time}}\right) \times 100\%}{(\text{Work Time} - \text{Down Time} - \text{Standby Time})}$$

- **Daily Utilisation Total**, measured as:

  $$\frac{100\% - \left(\frac{\text{Work Time} - \text{Down Time} - \text{Standby Time}}{\text{Work Time} - \text{Down Time}}\right) \times 100\%}{(\text{Work Time} - \text{Down Time})}$$

- **Daily Availability**, measured as

  $$\frac{100\% - \left(\frac{\text{Work Time} - \text{Down Time}}{\text{Work Time}}\right) \times 100\%}{(\text{Work Time})}$$

However, when comparing quality of drilling with production, it is best to look at production during active drilling hours only being the total metres drilled when the RC hammer is operating (often called "penetration rate"). In particular, a focus on drill penetration rate gives a measure of how too-rapid advance rates may be adversely affecting sampling quality for a given set of ground conditions.

Quality monitoring can be presenting using time or depth graphs of the precision and accuracy on both primary and rig split samples. There are five quality metrics that can be used to compare against the metres/active drilling production ratio:

- Precision of all bag masses, as an indicator of the control on delimitation and extraction errors on the primary sample, measured by the standard deviation divided by the mean for all dry, fresh and density-corrected data for the time period.

- Precision of average 1st, 2nd, 3rd, 4th, 5th and 6th metre of every rod as an indicator of the control on delimitation errors on the primary sample (e.g. "delimitation performance"). Measured by the standard deviation divided by the mean for the overall average 1st, 2nd, 3rd, 4th, 5th and 6th metre of every rod for the time period.

- Overall recovery for the day as a measure of accuracy of the primary sample, measured by the ratio of all dry, fresh and density-corrected weights over the theoretical weight for the time period.
- Precision of sample split masses, as an indicator of the control on delimitation and extraction errors on the rig split sample, measured by the standard deviation divided by the mean for all mass differences for the time period.
- Overall bias of the difference between original and duplicate sample masses as a measure of accuracy of the rig split sample, measured by the average of all split weight differences.

Examples of control plots are shown in Figure 4, Figure 5 and Figure 6. Figure 4 is a plot of the delimitation precision of the primary sample over time for one driller. The clustered light-to-dark-blue lines are the average masses for the 1st, 2nd, 3rd, 4th, 5th and 6th metre of every rod for each day. For the 10-day period between 14 to 24 May, the 6th sample of each rod was always the heaviest and the second metre the lightest. For the period between 6 April and 4 May rod-interval-sample-mass pattern was random, and lines for each sample sequence are clustered more closely together indicating the preferred target variability between sequential samples on each rod. The orange line in Figure 4 represents the principal delimitation precision, with low precision numbers representing better quality (e.g. low variance is desirable). Again, the period between 6 April and 4 May stands out as period of good sample quality because of the low precision. Discussions with the drillers identified that some of the quality issues related to the rig conditions (rigs were changed over on the 10th of May) as well as some ground condition issues.

One of the issues identified by this graph (and using the plot in Figure 3) was that the metre marks used to identify drilling advance intervals were incorrectly placed on one of the drill rigs.

Figure 4. Delimitation precision of the primary sample over time. See text for explanation.

Figure 5 is a plot of the delimitation and extraction precision, and the accuracy of the rig split sample over time. The blue points on the background of the plot are the routine-duplicate mass difference values plotted on the vertical axis date on the horizontal axis. The red dotted line is a 20-point moving average for the pair mass difference data.

The plot reveals at least three periods where average mass differed significantly as indicated by step changes in the average mass line (black) in Figure 5. In the first period the mass difference bias is \( \approx +400 \) g, with the duplicate always heavier than the routine split, which was targeted to be 5 kg. As such the +400 g bias represented a relative bias of \( \approx +8\% \). The second period, started on 29 March when the sample splitting device was changed from a static cone splitter (used for the first period) to a Metzke Splitter. During commissioning of this unit, the mass difference plot revealed that the new sampling unit produced a negative bias between routine and replicate samples of \( \approx -400 \) g up until 10 April. Following some small engineering modifications and with increasing experience with the new sampling system, the split masses returned on average to zero difference, and sample splitting continued without bias.
Figure 5. Accuracy and Delimitation & Extraction Precision of the rig split over time. See text for explanation.

Figure 6 is a plot of the primary delimitation precision against the metres/active drilling hour for a drill rig. Note how the drilling production improves towards the end of the program from 18-22 m/active hour to 22-24 m/active hour, with quality improving in parallel from 12-14% precision to 8-10% precision. Both metrics spike upwards after 21 April when a drilling crew changeover occurs and a new driller takes over. However, the chart then reveals that with time and feedback the new driller learns how to deal with the ground conditions and returned both metrics to levels of good production and good sampling precision.

Figure 6. Delimitation & Extraction Precision% vs Metre/Active Drilling Hours. See text for explanation.
CONCLUSION

RC drilling is usually subject to a range of sampling errors that occur in the primary or sample splitting stage. Understanding the source of these errors and putting systems in place is the first point of call for a robust system to improve the overall quality of the RC sample.

A QC system that is based on sample primary sample masses and secondary sample mass difference for replicated splits from the primary lot, can identify when delimitation and extraction errors may be occurring on a real-time basis. When plots of these metrics are prepared as easy-to-interpret communication tools for geologists and drillers, constructive discussions around the results improves both sample quality and drill production rates.

When these types of processes are implemented in RC drilling programs, the resulting data can be considered as high-quality for downstream MRE work.

ACKNOWLEDGEMENTS

The authors would like to thank John Graindorge and Mark Murphy for their comments and edits.

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MAKING BETTER USE OF DRILLING DATA
INTRODUCTION

As a minerals project moves from discovery to technical and economic evaluation, assessment of the potential processing routes for the ore becomes increasingly important. The ore and adjacent waste that may be mined as dilution, must be tested to determine their response to crushing, grinding, mineral extraction and, finally, disposal as tailings. The more work that can be done at the prefeasibility stage of a project to define and quantify possible options, the more reliable will be the decision regarding which path to choose for detailed examination in the definitive feasibility study.

BULK SAMPLES

Bulk sampling from test pits, shafts or large diameter bucket drillholes, remains an essential step for testing some ores at a scale close enough to production scale for beneficiation upgrade, comminution behaviour, mineral recovery, slurry rheology, and tailings settling characteristics to be forecast with confidence. Bulk sampling is particularly important for projects with non-typical ore where the optimum final processing route is not immediately obvious. This situation creates the necessity to evaluate multiple technologies and circuit arrangements. Bulk sampling can provide the large quantity of sample that will be required for a complex testing program. However, bulk sampling is increasingly difficult to complete due to the depth of excavation that is typically now necessary, environmental permitting for such excavation prior to approval of the project, and the consequent high cost of completing such a program.

Bulk samples may also be required for customer trials but typically this will be after project feasibility has been demonstrated.

DRILLHOLE SAMPLES

For the most part, the material available for metallurgical testing will be selected from drillhole samples. A range of drillhole types may be suitable depending on the circumstances.

NQ, HQ and PQ core are perhaps the most common sampling media, providing sufficient sample for many types of test. If stored in dry, under-cover conditions, drill cores of many ore types are quite stable and will remain suitable for metallurgical testwork for months or years. There are exceptions, such as the presence of marcasite and other unstable sulphide minerals which may deteriorate immediately on exposure to air.

If the core is prone to rapid deterioration due to oxidation, it may be necessary to take specific measures to ensure that it remains in the as-drilled condition and representative of the in-situ material. Individual core portions are often sealed in plastic bags with air excluded by flooding with argon (or nitrogen), then stored and transported under refrigeration.

Reverse circulation (RC) drillhole chips may also be considered for metallurgical sampling, however the disadvantages of RC samples include:

- The condition of the sample at the time that testing is required is difficult to evaluate. For example, if the sample has begun to oxidise, it may be unsuitable for flotation testing. Due to the fine particle size, RC samples would be expected to deteriorate faster than drill core.
- The sample is not suitable for most comminution testwork, since it has already been crushed.
- The sample is unlikely to be logged in as much detail as core, so knowledge about the characteristics of the material may be limited. This potentially would make it more difficult to interpret the results of the metallurgical testing.

1 Principal Consultant, AMC Consultants Pty Ltd
2 Principal Geologist, AMC Consultants Pty Ltd
While these uncertainties make drill cuttings less attractive at the feasibility stage, once a mine is in production the cuttings from RC grade control, or blast holes may provide valuable geometallurgical data. For example, at Batu Hijau, ore control includes flotation tests on blast hole samples to enhance short term forecasts of copper and gold recovery. The copper grades at Batu Hijau also show a relationship with mill throughput (Wirfiyata and McCaffery, 2008). The blast hole sample grades and flotation recoveries provide the opportunity to fine tune blending or settings in the crushing circuit and flotation plant.

SAMPLE SELECTION

Compared to exploration sample analysis, many metallurgical tests are expensive. Table 1 provides order of magnitude costs for a range of common metallurgical tests.

<table>
<thead>
<tr>
<th>Test</th>
<th>Purpose</th>
<th>Sample mass (kg)</th>
<th>Approximate cost ($)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Fire assay</td>
<td>Determine gold content</td>
<td>2 kg</td>
<td>$15</td>
</tr>
<tr>
<td>Screen fire assay</td>
<td>Determine gold content, accounting explicitly for coarse gold</td>
<td>2 kg</td>
<td>$100</td>
</tr>
<tr>
<td>Gravity recoverable gold test (three stage)</td>
<td>Determine gold content and proportion of gold potentially recoverable with a gravity circuit</td>
<td>30 kg</td>
<td>$8,000</td>
</tr>
<tr>
<td>ICP-AES</td>
<td>13 element suite, whole rock analysis</td>
<td>2 kg</td>
<td>$40</td>
</tr>
<tr>
<td>Flotation (base metals)</td>
<td>Determine the recovery of base metals sulphides to flotation concentrate</td>
<td>Up to 2 kg of minus 3.35 mm material depending on feed density requirements</td>
<td>Varies based on test type (bulk or kinetic, number of samples). Rougher test can be from $400 to $600</td>
</tr>
<tr>
<td>SMC test</td>
<td>Determine resistance to crushing and hence parameters for SAG and ball mill design and throughput</td>
<td>20 kg – 40 kg of minus 40 mm to ensure sufficient particles (about 5 kg) can be selected for the recommended fractions</td>
<td>$1,500</td>
</tr>
<tr>
<td>Drop weight test</td>
<td>Determine resistance to crushing and hence parameters for SAG and ball mill design and throughput</td>
<td>100 kg of minus 63 mm to ensure sufficient no. of particles can be selected for the recommended fractions. Will end up using approximately 30 kg for the actual testing</td>
<td>$3,500</td>
</tr>
</tbody>
</table>

The expense of metallurgical testing and the mass of core required means that selection of the best intervals to sample is particularly important. Furthermore, the nature of comminution tests such as Drop Weight, and the large sample masses required mean that in many cases the entire core from a mineralised intercept may be consumed by the testing. Sample selection needs to be considered from both the minerals processing and the geological perspectives.

Sample selection is the joint responsibility of the geologist and the metallurgist. The engagement between the two should be collaborative, enquiring, and iterative. The metallurgist brings expertise in minerals processing and metallurgical testwork procedures. The geologist brings in-depth knowledge of the orebody; its grades, rock types, mineral assemblages, textures and spatial disposition.
The metallurgist knows how the samples should be tested, how much material is required and what characteristics are likely to be important for the unit processes in likely processing flowsheets. The geologist knows how the chemical, mineral, and physical characteristics vary across the deposit. Jointly they can determine what parts of the deposit need to be tested. The mining engineer also provides important input to sample selection, bringing understanding of the potential stages in which the mine may be developed and the sequence in which ore types may be delivered to the process plant.

Until quite recently, the focus of metallurgical sampling was often the notion of testing “representative” samples. In many cases, “representative” was translated as a composited sample created to represent a blend of the major ore types or a ‘time blend’ to represent nominated production periods. In particular, the first few years of production are often targeted because they have the greatest impact on project value.

In these cases, sample selection from drill cores will likely focus on intervals of the major ore types with apparently average characteristics, from locations dictated by the mine schedule. In the case of flotation testing, composited samples of this type will often be assessed with lock cycle tests to simulate typical rougher/scavenger/cleaner circuit arrangements. Often pilot scale trials using the preferred circuit arrangement are conducted to simulate the anticipated processing method as closely as possible.

The major limitation with blended composites of this type is that they do little to increase the understanding of ore response and recovery on a practical short-term scale of hours or days. Short-scale fluctuations in ore fed to a plant can have a large impact on mineral recovery and plant performance. If a process plant has been designed with tolerances that are too narrow to cope with variations in ore feed characteristics, excess load may build up, reagent consumption may increase, slurry rheology may change, energy consumption may increase and tailings grades may increase.

More recently therefore, metallurgists and geologists have placed increased emphasis on understanding the natural variability of the ore characteristics and its impact on processing response. Samples for variability testwork should be selected to represent the range of ore types that may be fed to the mill. Selection therefore will be based on recognition of geological variations that may influence comminution or extraction response. Such features may include:

- Oxidation.
- Transition zone sulphides or mixed oxide-sulphide assemblages.
- Variations in hypogene sulphide assemblages.
- Grain size.
- Texture.
- Hardness; resistance to crushing and grinding, or abrasiveness.
- Fracture frequency and rock quality designation (RQD).
- Alteration (hydrothermal, mesothermal or supergene).
- Gangue minerals, such as clays, talc, carbon (graphite), carbonates.
- Chemistry; chemical variations that are not easily recognised by logging of mineral assemblage.

Variability sampling is designed to identify any problematic ore types and to determine the range of responses that the process plant will need to be able to handle. Designing for the extreme ore types is generally impractical, uneconomic, and usually made unnecessary by good ore control and blending practices. In most cases, process engineers will design the plant to run optimally on an average run-of-mine feed, but with sufficient flexibility to handle the 75th percentile of the ore. Circuits may also be designed for ease of reconfiguration on the strength of geometallurgical data and good mine planning. Significant capital expenditure can be saved by making provision in the initial plant layout for additional crushing, grinding or processing equipment that will be needed later in the project when new ore types enter the feed blend.

**DRILL CORE LOGGING**

Clearly, for sample selection to be based on variations in geological features, their occurrence must be recorded by the geological logging process. To be reliable, the geological logs need to be as objective and quantitative as possible. This is greatly assisted by capturing each type of observation in a separate field in the log and applying quantitative estimates, either by percentage or by quantitative categories (e.g., 0 =
absent, 1 = trace to 2%, 2 = 2 – 5%, etc). It is important to ensure that the absence of a mineral feature is as reliably recorded as its presence.

Objectivity is assisted by maintaining a reference set of core from each rock type and its variants. In addition to displaying macroscopic features, the reference set should be supported by thin and polished section microscopy, and multi-element geochemical analyses. Geologists should make sure they are familiar with the reference set, and are equipped with, and use; hand lens, tungsten carbide scriber, and dilute hydrochloric acid.

Hand-held X-ray fluorescence (XRF) analysers and near infrared spectral analysers provide additional semi-quantitative or quantitative assessments of chemistry and mineral content and are becoming more widely used. They can take much of the guess-work out of identification of fine grained minerals and alteration.

The nature of comminution tests such as the Drop Weight Test, and the large sample masses required mean that in many cases the entire core from a mineralised intercept may be consumed by the testing. It is vital, then, that the logging of the samples is as thorough as possible, accurately recorded and supported by good quality core photographs.

MANAGEMENT OF METALLURGICAL TEST DATA

As noted by Munro and Tilyard (2009), geologists are typically better custodians of data than metallurgists. This is presumably because geologists spend most of their careers collecting spatially located data that has a potentially long useful life, whereas metallurgists spend more time with plant data that is non-spatial and commonly of only day-to-day importance.

The information that most commonly gets lost is the identity of the metallurgical sample; the drillhole name and the depth-from and depth-to. This is common for metallurgical samples that are composited across multiple drillholes and intervals. The geologist likely already has a well-structured geological information management system (GIMS) (or if not, why not?). This database should be used to store, as a minimum, the metallurgical sample number, and the drillhole intervals from which it was derived. The records should also include the laboratory to which the sample was despatched, and an identifier such as the report number for the final results. Best practice is to store all the metallurgical test results in the GIMS.

Independent quality control is also sometimes overlooked in the metallurgical program. The geologist will typically have an array of certified reference materials for quality control during the assaying of the regular drill samples. These should be provided with the metallurgical samples on submission and the results included in the GIMS.

The results of metallurgical testing may also influence the evolution of the geological logging procedures for a project; placing new emphasis on particular geological characteristics, or demanding the collection of more accurate or more detailed data. These challenges should be embraced by the geologist.

CONCLUSION

Drillhole samples play a critical part in the selection of an optimum processing route for an orebody. Due to the size of sample required for metallurgical testing and the cost of those tests, sample selection needs to be carefully considered and based on good logging and assaying data. Sample selection is the responsibility of both the geologist and the metallurgist. Their engagement should be collaborative, enquiring and iterative, responding to the data and working to build a comprehensive three-dimensional understanding of the orebody, its response to processing, and its economic value.

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MAKING BETTER USE OF THE CORE THAT WE DRILL USING DETAILED PETROGRAPHIC ANALYSIS

ROWENA DUCKWORTH ¹

INTRODUCTION

Detailed descriptions of rock mineralogy, textures and paragenetic sequence are easily obtained using old fashioned optical microscopy. Sadly, it is a technique that is not used as widely as it used to be, despite its ability to provide information reasonably quickly and cheaply. Electron microprobe analysis (EPMA) is a more recent and proven analytical technique that utilises X-ray diffraction methods to provide definitive mineral chemistries and element identification.

Both techniques can be used on regular polished thin sections and these two techniques combined can therefore divulge abundant information about core, chip and outcrop samples that is useful for target generation and, later down the track, metallurgy and mineral processing.

THE CASE FOR USING PETROGRAPHIC/MINERAGRAPHIC ANALYSIS

Optical microscopy and EPMA techniques can effectively complement or contradict routine core logging descriptions of diamond drill core. Submitting one or two samples for petrology from each lithology encountered is highly recommended, as logging descriptions regularly prove to be unreliable, and should always be ground-truthed before mineralisation models and target generation plans are developed. An example of this is a suite of rocks logged as amphibolites that systematically failed to contain any amphibole, but did contain abundant chlorite, when observed using transmitted light microscopy. This type of hand specimen mis-identification is common, especially in pervasively hydrothermally altered rocks, and can lead to incorrect mineralisation models and hence, poor target generation.

Another use of petrographic and mineragraphic analyses is to identify where the target element is in the drill core samples that have been assayed. For example, is the gold visible? If so, is it related to an early pyrite or a paragenetically late pyrite? Identification of where the gold sits paragenetically, and the texture, i.e. porous pyrite vs clean pyrite for instance, and mineral association of each phase it is associated with, can directly benefit the logging and the assay sampling of the core. Figure 1 is a reflected light photomicrograph at x20 magnification showing bright gold associated in this case with arsenopyrite, which is replaced by later chalcopyrite. Ascertaining mineral associations and paragenetic sequences early via reflected light mineragraphy and EPMA can therefore save many thousands of dollars when it comes to mine planning and processing.

Figure 1. Reflected light photomicrograph at x20 magnification showing bright gold with white arsenopyrite and yellow chalcopyrite.

¹ Mintex Petrological Solutions/ Gnomic Exploration Services, Townsville, Qld 4810
As well as determining the major, minor and accessory minerals in a rock sample, optical microscopy can identify:

- Primary mineralogy and textures.
- Metamorphic minerals, textures, metamorphic facies, and the degree and sequence of ductile or brittle deformation events.
- Alteration mineralogies and facies.
- Vein and selvage mineralogy.
- Sequence of events (paragenesis) including the timing of ore mineralising events: did the ore minerals come in early or late?

Petrographic description of rocks can also help with the refinement of stratigraphy and structure, the identification of geological and hydrothermal processes and assist with ore grade vectoring from near miss drillholes.

EPMA can additionally provide crucial information with respect to the location of elements of interest: e.g. gold. Gold can occur as native gold, or bonded with other elements to form a mineral, e.g. sulphosalts, as inclusions within a mineral or bound up in the lattice of another mineral phase. Often assays will indicate the presence of significant gold, but if it is not free gold it may not be distinguishable in hand specimen and even under the optical microscope. EPMA imaging and energy dispersive chemical analysis can usually resolve this problem.

The back-scattered electron image in Figure 2 is from a deposit where the company was not sure from hand specimen, mineragraphy or the assay data, whether the gold was occurring as free gold or as telluride complexes. Microprobe analysis showed that there was native gold as well as a range of telluride complexes including, gold, gold-silver, silver-gold, iron and mercury tellurides. This highlights the need to understand the correlation between assay data and the mineralogy.

Figure 2. EPMA back scattered electron image showing dark grey pyrite grains that have been variably brecciated and replaced by several telluride phases (with differing brightness). At this scale, using reflected light microscopy there may be just a hint of another phase present in the pyrite and with the naked eye it may not be apparent at all.

For qualitative and semi-quantitative mineral analysis, Energy-dispersive X-ray spectrometry (EDS) is commonly utilised and has both advantages and disadvantages:

- Quick 20-50 second analyses that can tell you the major elements in the mineral, e.g. Fe, Cu, S, Ag, Zn.
- Can do semi-quantitative analysis if confident of the elements in the sample.
- Accuracy of EDS spectrum can be affected by various factors, e.g. many elements have overlapping peaks (i.e. S, Pb).
Wavelength dispersive spectrometry (WDS) can be used for quantitative analyses as it has a better resolution but is more time consuming, as it requires a calibration to be established for each element to be analysed (majors plus minor elements of choice, F, Cl, Cu, Co, Te, Ag, Ba etc.) and each analysis can take 3-4 minutes depending on the elements being measured. However, the results are definitive and can be extremely useful as exploration vectors. Quantifying and then mapping the amount of a specific trace element in a specific mineral is a powerful mineralisation vectoring method. WDS also has both advantages and disadvantages:

- WDS differs from EDS in that it uses the X-rays diffraction on special crystals as its raw data.
- WDS has a much finer spectral resolution than EDS.
- However, only one element can be analysed at a time, while EDS gathers a spectrum of all elements, within limits, of a sample.
- Analysis time varies therefore on the number of elements set up for analysis, but average time is ca. 4-8 mins.

Due to modern occupational health and safety regulations, staining techniques are not widely used these days as they utilise hydrofluoric acid. Hence microprobe analyses can also be helpful in determining carbonate chemistries and feldspar chemistries, which often cannot be determined optically. Add cathode luminescence (CL) to the package and the results can be very impressive, as CL allows discrimination of different generations of the same mineral as a result of differences in trace amounts of activator elements, i.e. it can pick up chemical zoning not visible using other energy sources. This is particularly applicable in calcite, dolomite, apatite and quartz and can therefore help map the incredibly short, episodic, hydrothermal history often shown in these minerals, especially in epithermal environments (Figure 3).

Figure 3. Intricate chemical zoning in dolomite in the silica-dolomite of the Mt Isa copper ore.

Microprobe analyses can also be useful in the processing stage of mining to identify any contaminants, adverse or beneficial, which may affect the final mineral concentrate. Polished blocks containing concentrate samples from differing shifts can be analysed to see if there is variability in contaminant abundances in different feeds.
Making Better Use of the Core That We Drill Using Detailed Petrographic Analysis

R. Duckworth

AIG Drilling for Geology II: 26-28 July 2017
Brisbane, Australia

AIG Bulletin No. 64
INTRODUCTION

The Geological Survey of Queensland (GSQ) began on 1st April 1868, with the appointment of Christopher D’Oyly Hale Aplin as Government Geologist for the southern region. Richard Daintree was appointed as Government Geologist for the northern portion of the colony on 12th May 1868. (Johns, 1976).

In April 2018, the GSQ will celebrate its 150th anniversary.

EARLY HISTORY OF QUEENSLAND DRILLING ACTIVITIES

The earliest recorded water well was dug using pick and shovel in 1870. The casing was timber and the depth was 15.20 m. It was located near Oakey, west of Toowoomba.

The GSQ has lead the way forward for the state with innovation and exploration. A good example is the early development of drilling and of downhole geophysics. GSQ has always trailed new technologies to demonstrate its application to the wider industry and community such as the introduction of three diamond drilling rigs circa 1882. One rig was placed in Charters Towers drilling for gold in 1883 and in 1884 was transferred to drill for coal in the Bowen Basin. The others were stationed at Burrum and the Darling Downs.

Coal was discovered at Blair Athol in 1864 when a seam was penetrated at 18.28 m depth (60 feet) whilst sinking a well for water. A mine was established and opened in 1890. Subsequent drilling proved the existence of a second seam a metre below the top seam, but it was not until about 1908 that deeper drilling showed that this second seam was of a much greater thickness (Dunne, 1950). Coal was discovered at Collinsville in 1886 and mining began in 1912. The Mammoth Colliery north of Blackwater began in 1905.

In 1878, T.E. Rawlingson claimed that during flood, the Barcoo River and Cooper’s Creek appeared to convey large amounts of water into the interior where evaporation alone was considered insufficient to explain its disappearance. Rawlingson thought this water would be found underground and advocated deep drilling to test the theory.

In an 1884 paper, John Falconer cited repeating patterns of stratified water-bearing layers recorded along Queensland’s western railway as evidence of the Great Artesian Basin.

The costs associated with deep drilling dissuaded attempts to properly test the theory. It was the drought in 1885 that provided a catalyst for the Queensland Government to establish a test bore. The then Government Geologist, Dr Robert Logan Jack, had been advocating to the government in the preceding years that artesian water could be found on the western plains. A site at Blackall was selected. Drilling commenced on the 3rd of December 1885, however, due to a series of technological issues and a blow out of wages, completion was delayed until 1888 when they reached 506 m (1,660 feet) with a good flow of water (Towner, 1962).

The discovery of artesian supplies in Queensland prompted much excitement. Backed by large amounts of British capital, pastoral investment companies tried their luck and succeeded. Canadian driller J.S. Loughead was contracted by the Squatting Investment Company for assistance and in 1886 began drilling on the Thurulgoona property near Cunnamulla finding a good supply of artesian water at 250 m (820 feet) (Henderson & Johnson, 2016).

He was also contracted by J.B. Henderson, the Hydraulic Engineer for the Queensland Government, to drill for water in Barcaldine. Drilling commenced on the 16th of November 1887 and one month later, at a depth of 200 m (656 feet), he struck water which flowed at a rate of approximately 720,000 litres (158,378 gallons) per day (Blake, 2015).
The Dagworth bore northwest of Winton was drilled in 1896 to a depth of 1,016.51 m (3,335 feet) and the flowing water temperature was 91.1°C (196°F). The bore produced 5,682,615 litres (1,250,000 gallons) per day. Figure 2 illustrates the lithology log from this bore.

By the end of the 19th century, over 500 bores had been drilled or were in the process of being drilled across Queensland, the majority on privately managed leasehold land, with a 95% success rate.

Although gas was found at Hospital Hill at Roma in 1896, it was not until 1960 when the Pickanjinnie Gas Field was discovered and established (Gray, 1969). Queensland saw the first commercial production of oil at Moonie in 1961.
THE IMPORTANCE OF PRESERVING DRILL CORE

Today it is recognised that industry invests millions of dollars in exploration drilling annually, making a significant contribution to the state’s economy. The drill core stored in the Department of Natural Resources and Mines (DNRM) facilities at the Zillmere Exploration Data Centre (EDC) and the Mount Isa John Campbell Miles Drill Core Storage Facility provide a very important data set and a record of the state’s geological history.

Figure 2. Lithology of Dagworth bore, 1896 (Powell, 2011).
Just like tree rings, the composition and deposition layers of cores contain detailed records of the climatological and ecological changes on the Earth dating back millions of years. Different coring communities recover cores for study from lake and ocean floors, ice sheets, glaciers and even the moon. The deepest hole to date on earth is the Kola Superdeep Borehole, on the Kola Peninsula in the northwest portion of Russia, which commenced drilling in 1970 and took 24 years to reach 12,262 km (7.6 miles) depth.

Drill core provides examples of stratigraphy, significant structural features and unusual geological features, and is a unique insight into the geological world beneath our feet and the water. Scientists are even trying to work out how to drill a sample of core on asteroids and bring it back to earth. Rock core provides evidence of a range of mineral commodities, styles of mineralisation and tectonic settings. Ice core presents a history of past climates and atmospheric conditions and is a direct archive of past atmospheric gasses including volcanic activity. One metre of ice core can hold 500 years of information. The deepest ice drilling to date reached 3.7 km (2.3 miles) in Antarctica.

The estimation of a mineral or energy resource is critical to all mining operations irrespective of size. The primary method of determining size, grade and mineability of a potential deposit relies on drill core. The data obtained from core is vital to industry for the planning and development of operations. It is also very important to academia for teaching and research, as well as to the staff of the GSQ for conducting regional studies.

A drill core collection that is catalogued and available to view and sample by industry, academia and the public is a valuable and irreplaceable asset. As new knowledge and understanding are accumulating at an astonishing rate there becomes a greater emphasis on the importance of what has been often considered as insignificant data. Its relevance may be realised 20–50 years later when mine workings and samples may not be available, but a library facility where core and samples are conserved allows for logging, sampling with appropriate approval, and ongoing analysis. Stored core is simply one of many items in the geologist’s toolbox that is frequently examined and reviewed as new ideas, technologies and models become available.

Figure 3 shows the nature of requests and a general trend on a monthly basis for access to core at EDC. Customers are made up of students, industry geoscientists, scientists from various organisations, universities and other State and Territory jurisdictions, Ministers and department staff, the staff of the GSQ, and on occasion members of the public. EDC and the Mount Isa facility also host international visitors, sometimes for several months at a time. EDC also offers workshops and courses from time to time.

Figure 3. Graph of the number of requests for access to core at EDC.
CONCLUSION

The geology can be read from the core like turning the pages of a book. An important difference is that the rock can also be sampled for detailed study and analysis to provide clues about the study area and ultimately add data about the State’s geological history. Core collections provide a piece of the wider exploration puzzle and give support for the future prosperity of Queensland by providing indicators of potential mineral and energy resource discoveries.

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STRESS IN THE GROUND
IAN GRAY

ABSTRACT
This paper examines stress in the ground, the processes used to measure these, and the bases for working out their distribution. The paper is primarily focused on stress in rock, with more attention given to sedimentary rocks.

WHAT IS STRESS?
Stress is by definition the force acting on a unit area. While this has the same dimensions as pressure there is a difference – stresses may vary with direction. Unlike fluid pressure, stress may act in a direction which is not perpendicular to a surface. In this case it may be divided into components that are normal and parallel to the surface in question. The latter parts are the shear components. This directional aspect of stress is embodied in the tensor notation $\sigma_{ij}$ where $i$ is the vector normal to the plane in question and $j$ is the direction in which the stress acts. Where $j = i$ the stress is normal to the plane $i$, and where $j \neq i$ the stress is a shear stress acting on the plane defined by the vector $i$.

The product of stress and the area of a surface is force. The net force on the faces of any body and field forces (normally gravity) must be zero or that body would accelerate. As the ground is not generally accelerating within the frame of reference of our planet it is generally in equilibrium, albeit with some complex stress distributions stopping it from doing so.

A full stress tensor has nine components which can be reduced to six as the shear stresses $\sigma_{ij} = \sigma_{ji}$. The tensor can be re-described into principal stresses which are major, intermediate and minor values of stress, oriented in such a way that the shear components are zero.

WHAT IS EFFECTIVE STRESS?
In materials such as rock and soil there are frequently fluids at pressure that fill pores and fractures. These fluid pressures are very important as they affect the effective stress within the ground. Equation 1 describes effective stress.

Figure 1. Showing surfaces on a cube of unit dimension described by vectors $e_i$ perpendicular to these, surface tractions (forces) $T^{(e_i)}$ and stress tensors $\sigma_{ij}$, (from Wikipedia).
The simplest definition of effective stress is given in Equation 1, (Gray 2017).

\[ \sigma'_{ij} = \sigma_{ij} - \delta_{ij} \alpha_i P \]  

where:

\( \sigma'_{ij} \) is the effective stress on a plane perpendicular to the vector \( i \) in the direction \( j \).
\( \sigma_{ij} \) is the total stress on a plane perpendicular to the vector \( i \) in the direction \( j \).
\( \delta_{ij} \) is the Kronecker delta. If \( i \neq j \) then \( \delta_{ij} = 0 \), while if \( i = j \) then \( \delta_{ij} = 1 \).
\( \alpha_i \) is a coefficient affecting the plane perpendicular to the vector \( i \). It lies between 0 and 1.
\( P \) is the fluid pressure in pores and fractures within the rock.

The Kronecker delta term is used because a static fluid cannot transmit shear. The directional subscript indicating direction in \( \alpha_i \) is not usual practice where, for measurement reasons, only a scalar value is obtained.

Equation 1 indicates that positive fluid pressure acts to reduce the effective stress. In a soil, it is generally simplified to Equation 2.

\[ \sigma' = \sigma - P \]  

Here no direction is ascribed to the stress (generally because it is not known) and the fluid pressure \( P \) acts in all directions in the soil over the full extent of any surface chosen within it. This may be simply understood for a granular soil with point contact. Experimental evidence shows that Equation 2 applies to clays as well as granular soils and other particle types in between.

For the case of rock, we need to revert to Equation 1. Here the value of \( \alpha_i \) is generally not unity. There are two entirely different approaches to consider what \( \alpha_i \) actually is. In one case we may consider it to be a fraction of a surface area on which fluid acts. This might be used to describe the open part of a planar joint in an impervious rock mass which has been partially filled with a sealing crystalline mineral such as calcite. The other way to look at \( \alpha_i \) is in terms of poroelastic behaviour. This describes the behaviour of a soil or rock in terms of how fluid pressure affects its deformation. In this form \( \alpha_i \) is Biot’s Coefficient (Biot and Wills, 1957).

To help describe the concept of poroelastic behaviour in rock we might think of the rock as being a giant sponge with some open and some closed pores. If we consider the sponge as having some elastic behaviour, when it is stressed it changes dimension and will recover its shape when it is de-stressed. If we then stress the sponge and then inject fluid pressure at the same pressure as the external stress and measure the amount of recovery of dimension obtained, then the fraction of dimension recovery is described by Biot’s Coefficient. Because of the elastic behaviour of the sponge, Biot’s coefficient is describing the effect that pressure has on deformation.

Mathematically, the relationship between the change in strain of a rock with orthotropic properties exhibiting poroelastic behaviour with a change in stress is described by Equation 3.

\[ \Delta \varepsilon_{ii} = \frac{1}{E_i} \Delta \sigma_{ii} - \frac{v_{ij}}{E_j} \Delta \sigma_{jj} - \frac{v_{ik}}{E_k} \Delta \sigma_{kk} - \Delta P \left( \frac{1}{E_i} \alpha_i - \frac{v_{ij}}{E_j} \alpha_j - \frac{v_{ik}}{E_k} \alpha_k \right) \]  

where:

\( \Delta \varepsilon_{ii} \) is the change in strain in the \( i \)th direction under the influence of the three Principal effective stresses in the \( i, j \) and \( k \) directions.
\( E_i \) is the Young’s modulus in the \( i \)th direction.
\( v_{ij} \) is Poisson’s modulus in the \( i \)th direction.
\( \Delta \sigma_{ii} \) is Poisson’s ratio describing deformation in the \( j \)th direction due to stress in the \( i \)th direction.
\( \Delta \sigma_{jj} \) is the change in total principal stress in the \( i \)th direction.
\( \Delta \sigma_{kk} \) is the change in fluid pressure.
\( \alpha_i \) is Biot’s coefficient in the \( i \)th direction.
FLUID PRESSURE

As fluid plays such an important part in effective stress it is important to consider the fluid and what pressure it might be at. Ultimately it is essential to measure fluid pressure.

At the ground surface the fluid pressure is generally that of the air or of the hydrostatic pressure of some body of water above the ground surface. In the vadose zone there is both air and water and the difference in pressure between these is determined by the capillary pressure which is dependent on the soil or rock type and the degree of saturation. At low saturations, capillary pressures may be much greater than atmospheric pressure. Therefore, the water pressures in unsaturated ground are theoretically well below absolute zero pressure, thus indicating that the water is acting in tension without vapourising. This is surprising but important, especially in clays which may develop very high capillary pressures.

At depths below the vadose zone, groundwater pressures tend to rise hydrostatically. Artesian pressures or perched water tables may exist but these are the exception. With increasing depth fluid pressures may exist that are well above hydrostatic. The upper limit on these pressures is the minimum principal stress in the rock. Above this pressure the rock would open up and the fluid leak off. Also at greater depths, the fluids in the ground may include petroleum liquids and gasses.

DOES STRESS MATTER?

The stresses in the ground affect its potential deformation. This deformation may be around an underground opening, an excavation or in an earthquake. Failure may be defined as excessive deformation. Excessive deformation in a high-speed railway may be 5 mm misalignment in 10 m length of track. Excessive deformation may also be a major open pit wall collapse or a fault scarp caused by an earthquake. In the latter two cases the rock stress will have exceeded its strength, leading to a decrease in strength and energy release. Stress is frequently important to the design of wells for the purpose of petroleum production.

To be able to design in the materials of the ground it is important to know what the stresses and material properties are. The material properties prior to loss of strength are the Young’s moduli, Poisson’s ratios and Biot’s coefficients. The values of these tend to vary with stress. In sedimentary rock the Young’s modulus tends to increase with the average stress, sometimes several fold, while Poisson’s ratio tends to increase with shear stress and decrease with mean stress. Biot’s coefficient tends to decrease with increasing average stress. When some stress state, usually defined by the Mohr-Coulomb or Hoek and Brown failure criteria, is reached, the rock’s shear strength will diminish, sometimes dramatically, leading to rapid deformation.

A useful rule of thumb for determining the importance of stress in contributing to the failure on intact rock in mine roadways is to see if the in-situ stress is more than a quarter of the uniaxial compressive strength. If it does so, then failure is likely and support methods will need to be considered more carefully.

WHAT LEADS TO STRESS IN THE GROUND

Most sedimentary deposition is in a marine or lacustrine environments. The particles settle through water and build up in thickness. The types of material in the sequence change with time so that different layers are built up. The deposited material is a soil with a very high void ratio (pore space). As the soil is compressed by its own mass, water is squeezed out, the pore space is reduced and it becomes more dense. This is the process of consolidation. In fine grained material this may be a very slow process. In coarser material the water is squeezed out much more quickly.

Earthquakes may cause coarser material to consolidate as they shake the particles down into denser packing. In very fine material the effect is less as the water cannot escape. Liquefaction sometimes occurs in finer soils subject to earthquakes. This is a process where the packing of the soil is disturbed and the soil would pack down into a finer form. It cannot however do this immediately, because the stress that was carried between the grains cannot escape immediately and the water carries the stress in the form of raised pressures. Until the grains repack this soil is very weak and unstable.

In packing down in various forms the soil may generate a range of lateral stress. If the soil lacks a lateral boundary or there is extension of its base the limit of lateral stress is defined by its active state. If the lateral
boundary compresses, the stress is defined by the passive state. For a soil without cohesion, but with an internal angle of friction $\varnothing$, the range of state of horizontal stress may be approximated by Equation 4.

$$\sigma'_v(1-\sin\varnothing) < \sigma'_h < \sigma'_v\left(\frac{1+\sin\varnothing}{1-\sin\varnothing}\right)$$  \[4\]

Where:
- $\sigma'_v$ is the vertical effective stress due to self weight and fluid pressure effects.
- $\sigma'_h$ is the horizontal effective stress.
- $\varnothing$ is the internal angle of friction of the soil.

If $\varnothing = 30^\circ$ then the limits of Equation 4 are given in Equation 5.

$$\frac{1}{3} \sigma'_v < \sigma'_h < 3 \sigma'_v$$  \[5\]

This is a very large range which may depart significantly from the plastic state where $\sigma'_v = \sigma'_h$.

The value of vertical effective stress may vary due to deposition, erosion or changes in ground fluid pressures. The concept of overconsolidation is one where a soil has been buried to a significant depth and has developed stresses associated with that burial, and then erosion has removed material from surface, lowering the vertical stress. This raises the ratio of horizontal to vertical stress until failure occurs at something approximating to the passive stress state. This occurs near the erosional surface first.

Lithification of sedimentary deposits will take place with time. In this, crystals are built up which change and interconnect the grains. How much of the soil stress is carried over into the rock that is created is undetermined. Lithification leads to the development of rock that has more of the properties of an elastic solid, albeit not necessarily one with linear elastic behaviour. Indeed the moduli of sedimentary rock may be very nonlinear as is shown in Figure 2.

Figure 2. Tangent moduli derived from testing two similar sandstone cores, one triaxially and one uniaxially. Tri-E1 is the Young’s modulus transverse to the bedding direction, Tri-E2 is the Young’s modulus in the direction of bedding. Uni-E1 is the modulus transverse to the bedding direction derived from a uniaxial test.
Diagenesis may take place altering the minerals and bonding within the rock mass, and changing stresses further.

Strain in the ground has a major effect on changing the state of stress. This strain may come from major tectonic plate movement, folding or faulting. Tectonic plate movement tends to set a regional principal stress direction trend. This is overlain by folding at different scales, imposing tensional and compressional components to the stress within the rock mass.

Faults are invariably stress relief features, and the force that is relieved (stress \times area) has to be carried somewhere within the rock mass. This shift of load leads to some other zone being stressed. Take for example the simple example of a reverse fault that is localised within one layer of strata and has limited lateral extent. Following the fault displacement, the force taken across the fault has diminished and is moved to the surrounding layers and to the end of the fault, thus raising the level of stress in these.

If continuing lateral strain is being applied to a rock mass, it is possible for it to reach a stress at which failure on a fault occurs. After this the stress regime is changed, and it is possible for the stress principal directions to be rotated, sometimes at 90°. With continuing strain the stress builds up in the fault again, and failure ensues in a stick-slip cyclic process. The Japanese earthquake of 2011, which affected the Fukushima area, was a classic example of this behaviour. Similar events can however be seen in the coal mining areas to the west of Sydney and Wollongong (Gray, Wood and Shelukina, 2013).

Igneous rocks are either intrusive or extrusive. Extrusive material can only carry stress at the time of its placement caused by gravitational loading. Dykes and sills are good paleo-stress markers as they extrude in a direction perpendicular to that of the minimum stress. Large intrusions will carry the stresses required for their placement. These will be reflected in the stress in the surrounding rocks and in the visco-plastic behaviour of the rock mass in the molten state, which then becomes more elastic as the material cools. As all hot rocks cool they will shrink. This cooling strain generally de-stresses the rock mass. Where the boundary cooling is quicker than that of the main body of rock, the rock outer boundary solidifies first and shrinkage of the mass continues. This induces a compressive stress in the boundary and is considered to be the cause of exfoliation fracturing of exposed batholiths.

Other forms of shrinkage may also exist that affect stress. One of these is the shrinkage of organic matter in the process of coalification. This shrinkage is thought to lead to the presence of cleating, a clear indicator that zero or at least very low lateral stress existed in the coal at some time.

THE CONCEPT OF TECTONIC STRAIN

The types of rock in the ground are variable, with very different stiffnesses. Under the influence of strain the stresses developed in each different stiffness of rock vary. To understand the stress distribution better the concept of Tectonic Strain (Gray, 2000) is useful. Tectonic Strain is the theoretical strain required to cause the rock mass to be at its current state of stress.

In the context of fairly horizontal stratigraphic units, the tectonic strains may be thought of as the horizontal strains that are required to change the horizontal stresses in the ground from those which would exist due to gravity alone in a zero lateral strain environment. They need not be due to gross tectonic movement. Rather they may be due to local faulting or folding. Indeed, a component of the tectonic stresses may come from soil like behaviour of sediments with normal or over-consolidation of these prior to lithification. Some dimensional change can be expected in such lithification and any further diagenesis of the rock, and appear as a component of the tectonic strains. The effects of temperature on inducing strains may also be bundled into the tectonic strains.

Despite these limitations fairly even tectonic strains are found to exist in several measurements in a rock mass, and it is possible to use these to interpolate stresses between measurements.

The mathematics of determining Tectonic Strain are as below.

The average total vertical stress is, over a wide area, a summation of the product of density of all the superincumbent stratigraphic units with each stratigraphic unit’s thickness multiplied by gravity, as shown in Equation 6.

\[
\sigma_v = g \sum \rho_i \Delta x_i
\]
Where: $\sigma_v$ is the total vertical stress.
\[ g \] is the gravitational acceleration.
\[ \rho_i \] is the density of the $i$th stratigraphic unit.
\[ \Delta x_i \] is the thickness of the $i$th stratigraphic unit in the vertical direction.
\[ z \] is the depth from surface.

The effective vertical stress is given in Equation 7.
\[ \sigma'_v = \sigma_v - \alpha_v p \] [7]
Where: $\sigma'_v$ is the vertical effective stress.
$\alpha_v$ is Biot’s coefficient influencing the vertical stress.
$p$ is the fluid pressure.

The total horizontal stress due to self weight in a laterally confined situation with zero lateral strain is given in Equation 8 and the effective horizontal stress due to self weight in a similar case is given in Equation 9.
\[ \sigma_{hsw} = \sigma'_v \left( \frac{v}{1-v} \right) + \alpha_h p \] [8]
\[ \sigma'_{hsw} = \sigma'_v \left( \frac{v}{1-v} \right) \] [9]
Where: $\sigma_{hsw}$ is the total horizontal stress due to self weight.
$\sigma'_{hsw}$ is the effective horizontal stress due to self weight.
$v$ is Poisson’s Ratio for strain in the horizontal plane brought about by stress in the vertical direction.
$\alpha_h$ is Biot’s coefficient influencing the horizontal stress.
$p$ is the fluid pressure.

If we now use a simplified elastic model which does not account for creep behaviour, then we can subtract the effective horizontal stress due to self weight from the horizontal principal effective stresses to arrive at what we will term here to be tectonics stresses. These are shown in Equations 10 and 11.
\[ \sigma'_{t1} = \sigma'_1 - \sigma'_{hsw} \] [10]
\[ \sigma'_{t2} = \sigma'_2 - \sigma'_{hsw} \] [11]
Where: $\sigma'_{t1}$ is the major tectonic horizontal stress.
$\sigma'_{t2}$ is the minor tectonic horizontal stress.

Assuming a ground surface that is free to move vertically the tectonic strain may be calculated using Equation 12 and 13.
\[ \varepsilon_{t1} = \frac{\sigma'_{t1} - \sigma'_{t2}}{E} \] [12]
\[ \varepsilon_{t2} = \frac{\sigma'_{t2} - \sigma'_{t1}}{E} \] [13]
Where: $\varepsilon_{t1}$ is the major tectonic strain.
$\varepsilon_{t2}$ is the minor tectonic strain.

To examine the average tectonic strain for a group of stress measurements, the procedure is to rotate the principal strains into direct N-S & E-W strain and shear strain components and to find the mean of these. The principal tectonic strains and their directions may be calculated from these three mean strains. If tectonic strains are relatively uniform between adjacent stress measurements they may be used to calculate stresses in rock of varying Young’s Moduli and Poisson’s Ratios. The process is the reverse of that used to derive the tectonic strain.

The effective stresses due to tectonic strain may be calculated using Equations 14 and 15.
\[ \sigma'_{t1} = \frac{E}{1-v^2} (\varepsilon_{t1} + \nu \varepsilon_{t2}) \] [14]
\[ \sigma'_{t2} = \frac{E}{1-v^2} (\varepsilon_{t2} + \nu \varepsilon_{t1}) \] [15]
The total effective horizontal stress may be calculated by adding the horizontal stress component due to gravity acting in a zero lateral strain environment as given in Equation 9 to the values arrived at in Equations 14 and 15.

This may seem like an unnecessarily complex process but it gives a consistent basis by which to assess the state of stress in strata where stress measurements have not been undertaken.

Figure 3 shows an example from a site in a sedimentary basin in eastern Australia. In it the major effective stresses from 68 stress measurements are shown in red. They are quite variable. The calculated tectonic strains are, however, quite even in each borehole and, with the exception of the first hole, they are quite even across the site.

Figure 3. Example showing the major effective stress and the major tectonic strain.

Not all sites are going to provide as even a set of Tectonic Strains as are presented in Figure 3. Where there is faulting the calculated values of Tectonic Strain tend to vary, as stresses are redistributed around faults. Similarly, quite different tectonic strains may be found above and below erosional surfaces. Tectonic Strain theory does however give a good basis for getting away from plotting horizontal stresses versus depth and quoting meaningless ratios between these and the vertical stress.
HOW DO YOU MEASURE STRESS?

It is impossible to measure stress in the ground without disturbing it. The disturbance required is considerable and will affect the value being measured. Thus, stress measurement is anything but exact. The next problem with measuring stress is that it tends to be so variable that a number of measurements are needed. These are point measurements and need a logical interpretation to permit sensible interpolation.

Soil Stress

The measurement of stress in-situ in soils is difficult. Anything put in the ground affects the stress and causes inelastic failure of some sort around the hole or plate penetration. Correlations exist with a variety of pressuremeter (devices that expand in a hole or slot) tests but essentially all are dubious. This is unlike the case for soil fill, where stress measurement sensors can be buried in the ground and can be expected to measure quite accurately.

Stress in Rock

The measurement of stress in rock can be accomplished with a variety of ways, with differing degrees of certainty. The principal methods used for rock stress measurement are:

- Overcoring.
- Hydrofracture.
- Borehole breakout.

Overcoring

Overcoring is the process where the stress change is determined by measuring a dimensional change within rock, when it is stress relieved, by coring over the top of it (Figure 4). Several variants of overcoring exist. To determine a stress field solution by overcoring requires that the rock properties remain elastic, and preferably linearly elastic.

Figure 4. The concept of surface overcoring.

The simplest overcoring method, which is often forgotten these days, is surface overcoring. This method provides a measurement of stress at the rock surface, often where it is most important – in a tunnel or at the bottom of an excavation.

In the past this used to involve the measurement of the change in diameter of points around a borehole after a central stress relieving hole was drilled (Tsur-Lavie and Van Ham, 1974). The diameter change was measured using a mechanical dial gauge with pivoted legs. These days it is best undertaken by smoothing the rock surface with a diamond grinding wheel on a disc grinder and gluing a strain gauge rosette to the rock along with a temperature sensor inserted into the rock. After the temperature has stabilised, the strain...
gauge and temperature sensor wires are folded over and attached to the rock, and a concrete sampling drill is used to core over these. At the end of coring the drill is removed, usually leaving the core in situ, and the strain gauges and temperature sensor are reconnected. A record of temperature and strain is then obtained, so that the strain may be related to temperature change and hence corrected back to the precoring temperature. The strain difference before and after overcoring is thus measured, and the rock can be removed to have its modulus measured. The strain, Young’s modulus and Poisson’s ratio are used to calculate stress. The author has used this technique at the bottom of deep foundations in Sydney and in a TBM tunnel in the Snowy Mountains. In the latter case four surface overcores were undertaken and a total stress tensor derived from their values.

Overcoring is, however, most commonly undertaken at the end of a borehole. Most of the devices used for this are glue-in strain gauge devices, the adhesion of which is either impossible or compromised in wet holes. For this reason, they are most commonly used in dry upward holes that readily drain away drilling fluid. The glue-in devices include:

- Doorstoppers.
- Leeman Triaxial Cell.
- CSIRO HI Cell.
- ANZSI cell.
- Borre Probe.
- Cone cell.

While the two main mechanical devices are:

- USBM deformation gauge.
- Sigra IST tool.

Doorstoppers

The doorstopper type cells (Leeman, 1971) used a strain gauge rosette glued to the end of a flat ended borehole and then overcored (Figure 5). These only provide measurement of the stress perpendicular to the hole. They were followed by the work of Saito in Japan (Gray, 1980), who drilled a flat-ended hole with a radius at the edge and whose doorstopper was fitted with additional strain gauges. Saito presented a solution for the full stress tensor for this device. The recent development in this area has been that of cone overcore devices that are glued into the end of a conical hole and permit the full stress tensor to be determined (Obara and Ishiguro, 2004). All of the doorstopper variants have the advantage that they can be overcored to achieve stress relief within a small drill advance distance. This means that they have a greater probability of being used successfully in jointed rock.

*Figure 5. Overcore stress measurement process using a doorstopper.*
Glue in Pilot Hole Overcore Devices

The other type of glued-in cells were fitted into a pilot hole and drilled ahead of the main borehole. The Council of Scientific and Industrial Research (CSIR) cell (Leeman, 1968) was the best known early version of these. It had a great advantage in that the strain gauge rosettes used contained four gauges at $45^\circ$ to each other. This could be used as a simple check that the gauge was actually adhered to the borehole wall, as the sum of strain changes of each of the two orthogonal gauges should add up to the same value. The disadvantage of this device is that it was not designed to provide progressive strain measurements during overcoring. Rather, a measurement could be made before and after overcoring alone.

The CSIRO Hollow Inclusion (HI) cell followed (Figure 6). This did not place the strain gauge on the borehole wall but, rather, epoxy resin is exuded between the cell, which has integral strain gauges, and the pilot hole. When this has set, the cell can be overcored. The system is used with a cable that carries strain readings from the 12 strain gauges to the borehole collar, so that the overcore may be progressively monitored. The need for a cable is a serious disadvantage. So too is the problem of delamination of the epoxy from the pilot hole where its expansion becomes too great due to de-stressing.

Figure 6. A cross section of an overcore containing a CSIRO HI cell.

The CSIRO device was followed by the Auckland New Zealand Soft Inclusion (ANZSI) cell (Mills and Pender, 1986) which used the inflation of a low-pressure rubber tube (packer) to push strain gauges against the pilot hole wall. Since then a number of other variants of these devices have been built by various groups.

Mechanical Pilot Hole Deformation Overcore Devices

In addition to the glue-in devices, there are the overcore devices that monitor the pilot hole diameter mechanically. The first of these is the USBM borehole deformation gauge (Obert, Merrill and Moran, 1962; Merrill, 1967). It is a biaxial device that measures the change in borehole diameter in three places. It is only suitable for use in dry holes or holes with very limited water pressure. This piece of instrumentation is still made and is very much in use, particularly in North America. It was followed by an attempt to build a device at The University of Queensland that used three triangular pin-beam type strain gauges (Leahy, 1984) pressed into the wall of the pilot hole. This suffered from problems of the pins slipping on hard rock surfaces, but was nearly successful.
The Sigra in situ stress tool or IST (Gray, 2000) is the modern development of the USBM tool. It has been in use since 1996 and has accomplished measurements at depths from 1.5 to 1,000 m. It is used integrally with HQ or PQ wire line coring. It has six pin pairs to measure diameter. This is three more than the number required for a solution and therefore gives a measure of redundancy.

It is lowered into a pilot hole on wireline where it mechanically locks into place. Throughout the test it electronically records the diameters across the pin sets, the temperature and the output of the three magnetometers and three accelerometers. These latter measurements permit the tool’s orientation to be determined. The limitation of this tool and that of the USBM device is that it is biaxial. It is necessary to therefore assume the axial (vertical) stress. The deformations that are measured and the stresses that are calculated are perpendicular to the axis of the tool. In its most common use, the tool is used in vertical boreholes where the vertical stress is presumed to be lithostatic. This is not an unreasonable assumption in most cases, especially where the rock is flat-lying sediments or is at a shallow depth.

In summary Sigra’s IST system is:

- A quick biaxial overcore system:
  - 100 m hole overcore in 1 hour.
  - 500 m hole overcore in 2.5 hours.
  - 1000 m hole overcore in 4 hours.

- Used primarily with HQ wireline coring system though it may also be used with PQ.
- Mostly used in vertical holes

The stress measurement procedure is shown in steps in Figures 7 and 8.

*Figure 7. Initial steps in the Sigra IST stress measurement procedure.*
Stress in the Ground

Figure 8. Further steps in the Sigra IST stress measurement procedure.

<table>
<thead>
<tr>
<th>STEP 5</th>
<th>STEP 6</th>
<th>STEP 7</th>
<th>STEP 8</th>
</tr>
</thead>
<tbody>
<tr>
<td>PULLING BACK THE RODS FOR A COMPASS READING</td>
<td>OVERCORING</td>
<td>PULLING THE CORE AND TOOL</td>
<td>EXTRACTING THE DATA</td>
</tr>
</tbody>
</table>

Figures 9 and 10 show photos from IST operations while Figure 11 shows the pin displacements during the overcore and Figure 12 shows the best fitted result to experimental data.

Figure 9. An IST tool being lowered into an HQ drill pipe prior to overcoring.
Figure 10. An IST tool set in a core of medium grained sandstone after overcoring.

Figure 11. Pin displacements during an overcore.
Overcore Accuracy

Converting an overcore deformation or strain measurement to a stress requires the mathematics relating the two, and detailed knowledge of the stress-strain behaviour of the rock. Overcore analysis is essentially based upon the isotropic linear elastic behaviour of the material surrounding a pilot hole in rock. The problem with this is that a lot of rock is not linearly elastic or isotropic. This is illustrated by the relation of total stress to Young’s moduli shown for a sandstone in Figure 2. The best that can be obtained in laboratory core testing are the orthotropic properties of the rock, though the two Poisson’s ratios related to stress and strain perpendicular to the core axes cannot be precisely determined. In the event that the rock is porous, Biot’s coefficients are also important and need to be measured. Prior to the paper by Gray (2017) Biot’s coefficient had not been incorporated into overcore theory, though its role is of some importance. This is shown in Equation 16.

\[ \Delta D_i = \frac{D}{E} \left[ 2\sigma_m + 4\sigma_d (1 - v^2) \cos 2\theta_i - \nu \sigma_z - ((1 - 2v) + (1 + \nu)\alpha_r) p \right] \]  \[16\]

where:
- \( D \) is the pilot hole diameter.
- \( \Delta D_i \) is the change in pilot hole diameter at angle \( \theta_i \) from the principal stress direction.
- \( E \) is Young’s modulus perpendicular to the hole.
- \( p \) is the fluid pressure at the overcore location.
- \( \sigma_m \) is the mean total stress acting perpendicularly to the axis of the hole.
- \( \sigma_d \) is the deviatoric stress (major stress – minor stress)/2 acting perpendicularly to the hole.
- \( \sigma_z \) is the total stress acting on a plane perpendicular to the axis of the hole.
- \( v \) is Poisson’s ratio.
- \( \alpha_r \) is Biot’s coefficient in the direction radial and perpendicular to the hole.

Equation 16 may be re-arranged to provide a solution for stresses based on the pilot hole deformation in overcoring.
Equation 16 may be broken into two parts. In addition to the effect of Young’s modulus, the mean diameter change is affected by the mean horizontal stress perpendicular to the hole axis ($\sigma_m$), the stress along the hole axis ($\sigma_z$), normally taken as overburden stress, Poisson’s ratio ($\nu$), Biot’s coefficient ($\alpha_r$), and fluid pressure ($p$). Thus, the potential for errors in the mean stress grows with depth associated with the axial stress and fluid pressure. The deviatoric component of stress ($\sigma_D$) may however be more accurately determined from the out of roundness changes to the hole diameter measured during overcoring and influenced only by the Young’s modulus and one minus the square of Poisson’s ratio, $(1 - \nu^2)$. The latter remains reasonably close to unity even if the rock is totally plastic ($\nu = 0.5$).

The accuracy of overcore devices is limited by the ability to measure borehole diameter change or strain, and the amount of deformation that occurs during the stress relief brought about by the overcore process. The Sigra IST tool can measure to a sensitivity of 0.1 micron in the laboratory, or practically to one micron in the field during a good drilling process across a 26 mm diameter pilot hole. In a 10 GPa stiffness rock this corresponds to a stress measurement sensitivity of 0.2 MPa. If the rock is however, 50 GPa, this sensitivity reduces to 1.0 MPa.

Overcore tools that use glued-on strain gauges may have a nominal accuracy of 0.2 microstrain in the laboratory; however, in the field two microstrain is more realistic. This corresponds to a stress sensitivity of 0.04 MPa in 10 GPa rock. Once again temperature and drilling fluid pressures limit this uncertainty to a significantly greater value – maybe 0.4 MPa.

Hydrofracture

Hydrofracture is the prime means used by the petroleum industry to measure the minor principal stress at great depth. It is also used by others at shallower depths. It involves sealing section of borehole and raising the fluid pressure until failure occurs. The pumping is then discontinued. Fracturing fluid then leaks off and the fractures close. A schematic of the concept of hydrofracturing is shown in Figure 13. Figure 14 shows an idealised single fracturing sequence with subsequent fracture closure.

Figure 13. Schematic of hydrofracture development around a borehole in a field with major principal stress $\sigma_H$. 
The fracture closure pressure is a measurement of the minimum principal stress. In recent years, intense activity has been focused by the petroleum industry on the use of hydrofracture as a means of production from low permeability reservoirs. The determination of the closure pressure for minimum stress has been of the utmost importance and such methods as G Function analysis have been developed (Martin et al, 2012).

While obtaining the minimum closure pressure and thus the minor stress may be possible from hydrofracture, obtaining the major stress is much more difficult. Firstly, the assumption must be made that the borehole is drilled in one of the axes of principal stress. Further, that pre-existing fractures do not exist. Then, making the assumption that the rock is linearly elastic and has a certain tensile strength, it is possible to theoretically calculate the fracture opening pressure in terms of the closure pressure and the major stress. This approach is unrealistic because the tensile stress of the rock is unknown. The approach is then generally taken that the fracture can be re-opened following initial closure by a second pressurisation cycle, and that the fractured rock will behave exactly as though it were a rock without any tensile strength.

This approach is unrealistic as the first fracture will generally not close perfectly and pressurised fluid from the borehole will progress rapidly along it on re-pressurisation. This will lower the subsequent re-opening pressure. In addition, the poroelastic effects of fluid in the rock need to be considered as well as non-linearities of the elastic behaviour. The process to obtain a value of the major principal stress perpendicular to the borehole becomes very complex and interpretation is uncertain.

Hydrofracture as it is usually practised in geomechanics has another problem, as packers are used to seal a section of borehole. The packer scaling pressure against the borehole wall must exceed the fluid pressure to avoid leakage. The consequence of this is that it is very easy for fracture initiation to be caused by the packers themselves. This might just be avoided if the packer scaling pressure is dynamically maintained just above the fracture fluid pressure. In most applications this is not the case.

Hydrofracture can however be usefully used to open joints and find the normal stress across these. Knowing this normal stress may be all that is required. If multiple joints of different orientations exist in a rock mass it is possible to hydrofracture each of these and get a closure stress, and then use these multiple values to calculate a stress tensor.
Borehole Breakout

Borehole breakout also provides a method of assessing biaxial stress distribution around a hole. The method of measurement here is the dimension of the failure of the borehole wall. Breakout is shown schematically in Figure 15. This breakout is usually measured by an acoustic scanner as shown by the trace in Figure 16. If the wall stresses are insufficient to induce compressive or tensile failure of the borehole wall, then no indication of the stress field may be made.

Figure 15. Stylised borehole breakout.

Figure 16. Borehole breakout shown in an acoustic scan. Depth on y axis. Wrap around view of hole wall from 0 to 360 degrees shown on x axis.
Generally, the measurement of breakout or tensile fracture only permits the direction of major stress perpendicular to a borehole to be estimated. The petroleum industry uses borehole breakout width in combination with hydrofracture closure pressure and uniaxial compressive strength estimates to arrive at two-dimensional stress measurements. However, core taken from the hole and tested along its axis will generally not provide an accurate value of the transverse uniaxial compressive strength in anisotropic rock. Neither do relationships between the sonic log and uniaxial compressive strength provide an adequate value of UCS for the determination of stress from breakout.

Other Rock Stress Measurement Methods

Attempts at measuring stress by the Kaiser effect whereby the core emits intergranular noise, when it is stressed beyond its original stress state, have been proven conclusively to not work by Hseih, Dight and Dyskin (2015). There has been a recent resurgence in the use of post elastic strain recovery of core as a method to estimate original rock stress (Wang et al, 2012). The method did not, however, generally work in the past and must at best be regarded as being of dubious accuracy today. This especially applies in lightly stressed near surface tunnelling applications. Generally, the more direct the measurement, the more accurate it should be, and recourse to alternatives should be avoided.

CONCLUSION

The determination of the stress regime in rock is a complex process. Not only is it necessary to use the right stress measurement technique for the rock type, but then there is a need to interpolate and possibly extrapolate the stresses away from the points of measurement. This requires a good knowledge of geology. Any model developed should take into account the lithology and structural geology. A real programme to measure stress may include multiple stress measurements by a number of techniques, then interpreted in terms of Tectonic Strain. This might be followed by an examination of borehole breakout to determine stress direction where other measurements have not been made. Faulting, as revealed by seismic reflection, may then be the key to understanding the regional stresses. What is certain is that the concept of some unique far field stress loved by numerical modellers is not a reality. Neither does stress necessarily increase monotonically with depth.

While hydrofracture may appear the most direct means to measure stress it is really only suitable for the determination of the minor stress if that is in a plane that runs through the borehole. The determination of the major stress is fraught with complexity. Practically, hydrofracture has significant problems associated with packer pressure initiating the fracture and with pressurised fractures that do not quite close. Overcoring has more uncertainty in the determination of the value of mean stress, but more certainty in the determination of the difference between major and minor stresses. The values of stress derived from overcoring are dependent on the material properties – Young’s moduli and Poisson’s ratios. These need to be measured properly, and nonlinearities and anisotropy taken into account properly in the analysis for stress. This is difficult and generally not attempted.

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LEVERAGING THE VALUE OF DRILLING FOR OREBODY KNOWLEDGE

J. JACKSON

ABSTRACT

In economic geology, drilling is undertaken to locate, extend and evaluate mineralisation. Once mineralisation has been located, the primary goal of these activities is to evaluate whether the mineralisation can be extracted economically, which involves more than defining just tonnes and grade. Orebody knowledge or geometallurgy involves the characterisation of subsurface material to enable the prediction of how the material will respond to processes within the mining value chain. These processes include blast fragmentation, loading, material handling, crushing, grinding, flotation and leaching. Due to increasing environmental standards and societal expectations, orebody knowledge has extended into the surrounding host rock relating to environmental performance of waste material from mining and mineral processing as well as long-term closure impacts.

There are three key approaches to acquiring orebody knowledge - proxy based models, mineralogical based models and test-based approaches. The selection of the approach is based on trade-offs between spatial coverage, cost, relative accuracy, sample type and mass, logistics, time constraints and project stage. Thus, there isn’t a simple one size fits all approach to obtaining orebody knowledge and the options are required to be evaluated in context. This paper will provide an overview of recent developments in obtaining data from drilling that is used to create orebody knowledge.

INTRODUCTION

In economic geology, drilling is undertaken to locate, extend and evaluate mineralisation. Traditionally, the focus has been on two aspects: 1) delineation of tonnes and grade and 2) understanding the genetic aspects to locate additional mineralisation. A third aspect, which has been gaining more prominence over the past decade, is that of orebody knowledge which considers the characterisation of the mineralisation from an extraction (mining, processing, disposal) perspective.

This aspect has long been recognised and included in regulatory codes such as the JORC Code as ‘reasonable prospects for eventual economic extraction’ (JORC, 2012) and ‘Modifying Factors’ (JORC, 2012) where the key assumptions for reasonableness must be explicitly disclosed and discussed. However, the rigour in developing and quantifying these factors has traditionally been limited by the cost and sample requirements for extraction-related test work. This resulted in significant assumptions of ‘processibility’ through the testing of limited samples that were ‘representative’ or an average of mineralisation for a geological domain, a geological ore type definition or a production period (e.g.: first 5 years).

This coupled with issues such as poor data collection and analysis, lack of integration between discipline and project life-cycle silos have been contributing factors to the very poor performance in meeting production, revenue and cost expectations (PWC, 2012; IPA, 2015; Mackenzie and Cusworth, 2016).

This paper outlines common approaches in acquiring orebody knowledge, considers recent developments and the impact of drilling practices on developing robust orebody knowledge.

OREBODY KNOWLEDGE

Orebody knowledge involves the characterisation and mapping of subsurface material, most commonly via products from drilling, to enable the prediction of how the material will respond to processes within the mining value chain. Synonyms to orebody knowledge include deposit knowledge and geometallurgy within which are a diverse range of definitions (Jackson, McFarlane and Olson-Hoal, 2011).

Determining the appropriate characterisation methodology can be relatively complex due to several major variables such as the mineralisation type and style, sampling platforms and media, the extraction processes under consideration, degree of precision, spatial resolution, users risk appetite and application/use of the orebody knowledge as issues with legacy data. There isn’t a one size fits all method because:

1 JKTech Pty Ltd
Leveraging the Value of Drilling for Orebody Knowledge

J. Jackson

- The sampling platforms and media for characterisation may be derived from the drill rig itself, the wall rock in the hole or samples in the form of core, chips or powders.
- The extraction processes under consideration can include processes such as blasting, handling (load, haul, dozing, stockpiling), caving, crushing, grinding, flotation, dense media separation, leaching, gravity, dewatering, tailings disposal, waste rock storage. Essentially any process within the value chain that is or may be inflicted upon a rock or derivative of a rock.
- The degree of precision relates to the precision of the characterisation methodology in a similar way to that for assays. Generally, the higher-precision methods come with higher costs.
- The spatial resolution is a topical and common question within orebody knowledge as there are many dependencies with other considerations listed here. There are two major types of spatial resolution – the sample/composite length or volume and the spatial density of these samples or volumes.
- The user of any orebody knowledge can impose significant constraints on the selection of the related variables due to their risk appetite and how they intend to apply/use the knowledge. This can be based on one or more of personal, site and corporate standards or perceptions. Common uses include domaining, equipment selection and optimisation, mining and process improvement, mine to mill optimisation, preliminary economic assessments and feasibility studies.
- Risk appetite, whether it’s at a corporate, project or personnel level.
- Budget.

The industry has operated within technical silos. In project development, a common practice is that a program of holes be drilled specifically for each technical silo. By far, most holes are drilled for geology which is generally focused on the two aspects noted in the introduction, tonnes and grade and genetic geology. Other drilling programs are undertaken for metallurgical testing, geotechnical assessment particularly stability, and for environmental purposes. This often leads to a poor understanding of interdependencies, sweeping assumptions and biases particularly in relation to studies, but also to short and medium term planning leading to poor outcomes (Logan and Jackson, 2016; Mackenzie and Cusworth, 2016).

However, irrespective of the mineralisation type, the size of project or project stage, common threads for all orebody knowledge programs include:
- For all involved to have a holistic perspective of the entire actual or potential processing value chain.
- Balancing the trade-offs in data acquisition from drilling to provide relevant data and information in a timely manner.
- To identify the geological drivers of the variability in the characterisation of process performance. These can be very different for each process, and the standard approach to oretypes may not be appropriate (Jackson and Young, 2016).

It is the second thread that will be attempted to be addressed herein where three basic approaches can be considered for the determination of a process response:
- Via test work, where a sample is subjected to tests that are directly related to the process response.
- Via mineralogy, where a sample is analysed for mineralogical aspects which can be related to the process response by calibration with test work on selected samples.
- Via proxies, where a sample is measured/analysed for variables that can be easily and cost effectively mapped through the orebody and correlated with test work on selected samples.

Due to the required sample mass and cost, it is rare for test work to be undertaken on all samples across all drillholes as is commonly done for assays. Conventionally, test work has been undertaken on selected samples which consist of long interval composites or multi-hole composites.

DATA ACQUISITION, DRILLING AND SAMPLING

As mentioned above, three basic platforms and media for the characterisation of subsurface material are:
- The drill rig and its response to the rock properties. This typically falls under the terminology of measurement-while-drilling (MWD).
• The wall rock of the drill hole and the surrounding volume. The most common method is indirect, by characterising contrasts in physical properties through wireline geophysical logging or more recently logging while drilling (LWD).
• Bringing to surface samples of the material in drill core, chip or powder form.

Each of these has their own requirements in providing the rock/sample in a suitable condition to obtain adequate measurements and tests. This is in addition to the limitations and issues around the measurements and tests themselves. Thus, all personnel involved in the production and use of the data need awareness of the measurements and tests, and their limitations.

An important parameter is that of ‘hardness’, which can be measured using many methods depending on one’s perspective. For a geotechnical or blasting perspective, hardness often refers to rock strength which is commonly measured by universal compressive strength (UCS), however there are other hardness measures. In terms of mineral processing, there are at least three ‘hardness measures’ which are based on the scale of the size reduction which is a function of the equipment being used. Crushing hardness relates to the crusher (typical size reduction 2 m to 200 mm) and is generally related to UCS. Autogenous grinding (AG)/semi-autogenous grinding (SAG) mill hardness is a measure of the material’s resistance to breakage through impact, either by media or other rocks in an AG or SAG mill (typical size reduction 200 mm to 6 mm). Grinding hardness related to ball mills is a measure of the energy required to reduce a rock down to a known size (typical size reduction 6 mm to 75 um) to enable separation.

MEASUREMENT-WHILE-DRILLING

Since the early to mid-1900’s the relationship between MWD data and rock parameters has been explored. It was firstly utilised to assist with optimisation of penetration rate and in more recent decades to estimate the rock properties, particularly rock strength such as UCS, from MWD data.

Simple strength indicators like penetration rate, specific energy or more complex formulations like the modulated specific energy have been largely used in coal mining for coal strata detection, due to the sharp change in rock strength between the overburden and coal. A recent study by Hatherly et al. (2015) compared MWD (specific energy and rate of penetration) to geophysical logging in a point to point comparison basis for several holes in an open cut coal mine. It determined that MWD can be used to estimate not only rock type but the sonic velocity in the media (see Figure 1).

Figure 1. Estimated and measured down the hole sonic velocity (Hatherly et al., 2015).
Basarir and Karpuz (2016) used an adaptive neuro-fuzzy inference system (ANFIS) to estimate a rock mass strength indicator in a wide range of lithologies from Turkish mines. This system was trained to develop a decision rule based on bit load (BL), penetration rate (PR) and bit rotation (BR); the final link between UCS and MWD is a set of eight linear equations for UCS as a function of BL, PR and BR, each one defined for a certain MWD data domain. The results are shown in Figure 2.

Two important aspects are often overlooked when utilising MWD. The first is the variability arising from the driller(s), which are difficult to identify and then take into account (Segui, 2015). The automation of drill rigs has and will continue to improve this, as when an error is present it will consistently repeat and thus more easily be identifiable and rectifiable. The second is the effect of the engineering properties of the rig, particularly damping or buffering components such as hydraulics, with each rig having its own characteristics.

MEASUREMENTS OF THE WALL ROCK

Measurement of wall rock properties is a long-established practice in the oil and gas and coal industries. Wall rock measurement can be categorised into two types i.e. physical properties such as density, sonic velocity, gamma, resistivity; and observational features such as grain size, fabric and structures. Both types have the capacity to be used in building orebody knowledge.

The acquisition of such data has traditionally been via wireline logging in open/uncased drillholes. Alternate systems such as through case/rod logging (Hinton, Clarricoates and Samworth, 2016) and LWD where sensors are either within the rods or behind the core barrel (Greenwood et al, 2015) have been evolving from the oil industry to the minerals industry.

Sonic-related logging, along with density, are probably the most utilised geophysical logging methods for rock mass characterization, as sonic velocity can be related to mechanical parameters such as hardness and uniaxial compressive strength (UCS). An example of this is shown in Figure 3 where Young’s Modulus derived from sonic logging can be related to the mill hardness parameter Axb across a wide range of mill hardness (Jackson and McFarlane, 2015).

In the early 2000’s Century Mine implemented routine logging of blast holes with natural gamma and magnetic susceptibility to obtain a higher resolution map of stratigraphic units as inputs to mine scheduling and dilution management (Basford et al, 2001).

The recent developments in borehole logging tools and integration into the rod string, both in minerals and oil and gas have enabled logging to be more logistically simpler. These advances include tools for spectral induced polarisation (SIP) that can be used to monitor geochemical and biogeochemical processes associated with mine remediation strategies.
Figure 3. Predicted versus measured Axb using Young’s Modulus derived from sonic logging.

Other logging tools such as gamma-gamma and prompt-gamma neutron activation (PGNAA) type tools have been used to determine elemental composition and coal quality, particularly ash content in coal deposits (Hatherly, 2013). PGNAA and related tools such as pulse fast and thermal neutron activation (PFTNA) are becoming more commonly available for mineral deposit characterisation to obtain fast near real time geochemistry (Smith et al, 2015).

Optical and acoustic borehole televiewers have been used for rock mass characterisation for many years. They provide rapid and accurate high-resolution oriented images of the borehole walls and can be used as a replacement for manual core orientation techniques and are used mainly in geotechnical applications particularly for mapping of discontinuity spacing, width, orientation and texture (de Fredrick et al, 2014). Such data is also applicable for the mapping of blastability index which is used to identify the number and type of fractures within blast or grade control holes which can result in variable blast energy distribution. An example of an optical televiewer image and fracture interpretation is shown in Figure 4.

Figure 4. Example of televiewer logging from a reverse circulation (RC) grade control hole to input into blast design. The lower (right hand) part of the hole has significantly more fracturing and open fractures.

However, many of these techniques can suffer from the impact of drilling-related issues such as:

- Variable borehole diameter due to bit wear.
- Variable wall roughness which often results in less accurate and variable measurements and interpretation.
- Excessive drilling muds and grease on the hole walls.
- Dirty water in drill holes.
- Opening of closed fractures by drilling.
- Collapsing holes, particularly collars.

Hence the hole condition at the end of the drilling must be at the forefront of driller’s and geologist’s minds when these methods are to be utilised.
DRILL CORE, CHIPS AND POWDERS

Drill core, chips and powders are the most common media for measurements and test work to be undertaken on in the minerals industry. Rather than review very common measurements such as assaying and the complex issues around sampling in detail, a select few of the newer measurement and test technologies that can be used in the development of orebody knowledge will be considered.

Drill Core: Mineralogy

The ability to create digital mineral maps of drill core has improved significantly with improved camera technology, training and supervision systems, hyperspectral imaging and computation power. Although digital imaging of drill core logging has been available for some time, the new generation of systems enable resolutions of 20-50 um in the visible bands and 500 um in the hyperspectral wavelengths. Improvements in detectors such as those in the thermal infrared wavelengths allow for expansion of identifiable minerals. Imaging at such resolutions allows for improved mineral identification as the probability of a pixel containing one to three minerals is high, thus improving the ability to unmix the spectral reflectance from each mineral and identification.

Detailed mapping of minerals and associations has direct implications for orebody knowledge in terms of classifying mineral-textural types and then relating these to processing attributes. Recent methods of semi-automated processing of classified mineral images have enabled the prediction of attributes such as mill hardness in metalliferous deposits and ash-yield in coal, as shown in Figure 5 (Nguyen et al, 2016).

Figure 5. Examples of prediction of processing variability from mineral based images Top left: example of a classified image, Top right: prediction of mill hardness for a drillhole (Nguyen et al, 2016), Bottom: prediction of yield as 15% ash for a hole from a coal deposit using hyperspectral core imaging.
The issues in using drill core imagery and particularly with hyperspectral imagery include:

- That the scanned surface needs to be as smooth as possible, be it whole core or half core.
- The presence of saw cuts in half core create artefacts within the reflectance spectra.
- Poor handling of core when recovered from the tube, during transport or core cutting create broken surfaces that impact reflectance spectra.
- The surface of the core needs to be dry.
- Care should be taken not to write on or mark the core that is to be scanned.

**Chip and Powder: Mineralogy**

Detailed mineralogy on chips and powders, be they from drill core, RC or blast holes, has become feasible both in field camps and on a routine basis on operational sites from both a bulk modal mineralogy and a mineral liberation perspective.

The Olympus portable and desktop Terra X-ray diffraction (XRD) series of instruments is such an example. These instruments are able to obtain XRD spectra using a 15 g sample of ground powdered material at approximately 75 um in a vibration chamber that presents the sample in ‘multifarious orientations of the crystalline structure’ (Olympus, 2015). Comparisons to the interpretation of mineral phases using spectra from larger laboratory XRD units has shown excellent correlations for mineral phases greater than 5% and reasonable detection for mineral phases less than 5% (Burkett, Graham and Ward, 2015).

This type of data allows for identification of mineral phases that are problematic in mineral separation but difficult to identify in routine elemental assays. As an example, the detrimental impact of phyllosilicates on copper recovery in a flotation process is dependent on the phase and quantity of the phyllosilicate present as shown in Figure 6 (Farrokhpay, Ndlovu and Bradshaw, 2014; Ndlovu, Farrokhpay and Bradshaw, 2013). The data can also be used to correlate bulk mineralogy with mill hardness.

Several smaller, more rugged scanning electron microscope (SEM) based instruments, such as the FEI’s MLA Express, Zeiss’s Miniscan, have been developed over the past five years to bring high-resolution mineralogical analysis to project and operational sites, allowing for faster turnaround. Such systems when coupled with the appropriate analysis software provide automated detailed mineralogical characterisation, such as liberation characteristics, mineral locking etc. These characteristics can be related to many processes such as grinding, flotation, ore leaching, waste rock and tailings leaching.

The sampling provenance of the powder used for mineralogy is critical due to the potential for segregation by size. Many conclusions have been erroneous due to those interpreting the results not understanding potential segregation processes during drilling and sampling.

**Core and Chip Hardness**

In the last decade, a number of mineral processing-related hardness tests have been developed which allow hardness testing on relatively small mass samples that can be incorporated into the standard geological sampling and assay workflow.
An example for AG/SAG mill hardness are those tests developed at UQ’s JKMRC which includes the JKRBT Lite and the JK AbExpress. These tests enable an indication/estimation of the impact breakage resistance parameter Axb using samples ranging from <1 k g to 5 kg as compared to 30 kg for the SMC test and 100 kg for the JKDrop Weight Test and particle sizes as small as 4.75 mm. These tests allow for mapping of AG/SAG mill hardness at the same scale as assays for drill core, RC or blast hole drilling. An example is shown in Figure 7 from a porphyry copper deposit where the variability of AG/SAG mill hardness is similar to that of the copper grade. In terms of metal production per period, for porphyry copper deposits in particular, knowledge of the throughput is as equally important as knowledge of the copper grade. The corollary of this is that mill hardness, a key driver of throughput, should be known to a similar level of confidence and thus be drilled and sampled at a similar spatial density to that of copper. This applies at long, medium and short term planning and scheduling timeframes.

Figure 7. Downhole variability at a 3 m interval from core from a diamond drill hole in a porphyry copper Top: AG/SAG mill hardness indicator A*bexp (note higher values = softer) and associated variogram and model, Bottom: Cu grade and associated variogram and model

In the case of chips from RC or blast holes, the type of rig and bits, and the driller can play a significant role in the particle size distribution generated from the hole. Depending on the characteristics of the material, this can impact interpretation when directly comparing one area/hole to another. The trends will still be apparent but the absolute values may be differing.

CONCLUSION

Orebody knowledge is predominately derived from measurements and tests from the products of drilling, be it the wall of the drillhole, drill core or chips/powders. Determining the appropriate characterisation methodology through a project’s lifecycle can be complex due to variables that include drilling, spatial density, logistics, required confidence and budgets. There is much development in this field particularly in the use of samples with smaller mass than previously and bore hole logging. Those involved in the creation of orebody knowledge need to be aware of options, issues and limitations of the combination of drilling types, sampling methods, measurements and tests and decisions made around orebody knowledge.

ACKNOWLEDGEMENTS

I’d like to thank Constanza Parades Bujes, Anh Nguyen and Dr Travis Murphy from SMI-JKMRC and SMI-BRC at the University of Queensland’s Sustainable Minerals Institute for sharing their insights for this paper.
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3D MODELLING FOR EXPLORATION GEOLOGY: HOW TO GET THE MOST FROM DATA

BEN JUPP1, MATTHEW GREENTREE2 AND MELANIE SUTTERBY2

ABSTRACT

Drilling is one of the largest costs associated with mineral exploration and the delineation of mineral deposits. It is important that the explorer obtains the most value from each drillhole from planning through to the interpretation of results. Successful exploration campaigns are reliant on good quality data and the facilitation of timely decisions based on drilling results. Modern computing power and software allow for routine analysis of drilling data in 3D. New packages capable of rapid implicit modelling of assay and geological data now provide an efficient integration of a diverse range of datasets. This paper highlights the advantages of applying 3D modelling to a project’s workflow to help build confidence as the planning process develops and reduce project risks. This paper draws on an example from recent exploration work conducted by Ausgold Limited (Ausgold) on the newly identified Datatine gold prospect (Ausgold ASX release, 2016) at the Katanning Gold Project, in south Western Australia.

GEOLOGICAL MODELLING FOR EXPLORATION

A geological framework provides an invaluable tool for explorationists. The emergence of 3D modelling software has become an essential tool that provides for ongoing interpretation of exploration drillhole data through the development of a project. A good geological model integrates all the available data and provides the basis for targeting and drillhole planning. Multi-million dollar investment decisions are based on the drillhole dataset of each project. Therefore, a thorough understanding of the 3D geology and subsequently testing of geometric assumptions of mineralisation can be critical to the continuation of the project. In more advanced exploration and mining projects, a geological model not only provides a basis for resource estimation, but also underpins geotechnical and mining planning. The application of best-practice modelling is aimed at risk mitigation and the reduction of uncertainty tied to geological interpretations.

Improved speed of desktop computers and 3D geological modelling technologies now allow for the integration and reconciliation of a diverse range of datasets. The 3D technology has largely replaced traditional map and cross-section interpretations. Integrated data can include geographical, drillhole logging (geology, assay, alteration, and structure), geochemistry and geophysics. 3D modelling systems provide more than just integrated visualisation of data, they enable geologists to construct and test geological hypotheses. 3D modelling allows geologists to distinguish relationships in data that may have otherwise not been apparent from 2D interpretations. This visualisation can form the basis of new ideas through exploration of the interrelationships between previously disparate datasets and as a result, geologists are producing robust mineral systems models.

Advances in implicit 3D modelling technologies such as Leapfrog and GOCAD-SKUA have improved the speed of 3D model creation and subsequent analysis. As a result, models can be evaluated in shorter timeframes, resulting in the reduction of associated costs such as staff time and improving confidence in the underlying geology and faster project advancement (or abandonment).

CASE STUDY

This case study is from the Katanning Gold Project (KGP) in Western Australia and illustrates the application of implicit modelling and an integrated approach for the interpretation of geochemical, geophysical, structural and drilling datasets.

1 SRK Consulting
2 Ausgold Limited
Geology

A number of greenstone belts have been identified in the south-west terrane of the Archaean Yilgarn Craton (Cassidy et al, 2006; Figure 1). The greenstone belts are metamorphosed to granulite facies and are structurally complex, having undergone multiple phases of deformation. The Southwest terrane is a relatively under-explored area of the Yilgarn, despite the significant gold endowment of the Boddington Deposit and numerous other gold deposits such as Katanning, Griffins Find, Edna May, etc. Despite increasing interest from explorers, there are still large knowledge gaps in the understanding of the deposits in this region.

![Schematic location map of the Katanning Greenstone Belt.](image)

Figure 1. Schematic location map of the Katanning Greenstone Belt.

Gold mineralisation in the Katanning region is interpreted as orogenic gold mineralisation that has undergone post-mineralisation deformation and high-grade granulite facies metamorphism. Gold mineralisation has been identified within a number of areas by Ausgold (Figure 2) at the company’s KGP. Exploration at the KGP has focussed along major crustal features and greenstone interpreted from aeromagnetic data. Exploration within the region was limited by a paucity of outcrop and limited drillhole data. Geophysical acquisition (e.g. aeromagnetics) and 3D drillhole data interpretation has proven a major tool in understanding of the overall structural architecture as well as providing a vectoring tool for further exploration drill targeting.

Geological modelling by Ausgold and SRK (Greentree, 2013; Greentree & Cairns, 2014) at the KGP has focussed on development of the genetic mineralisation model. The current interpretation is based on high resolution airborne magnetic imagery, assay data, structural logging of drill core and pit mapping across five deposits within the KGP. Integration of these datasets was conducted using the Leapfrog® 3D modelling software, which allowed for the evaluation of multiple structural scenarios for mineralisation across the deposits.

Based from this work, SRK defined a new structural model for the controls on the distribution of high-grade shoots, which characterise the deposits. Previously mineralisation was interpreted as stacked gold lodes, however, this model did not account for high grade gold mineralisation (>10g/t Au). The new 3D model revealed tight isoclinal folding that caused the local transposition of higher grades within fold hinge zones. The isoclinal fold geometry is highlighted within the Jinkas Deposit (Figure 3). The fold geometry is also recognised within aeromagnetic datasets and can be mapped in outcrop and drill core.
Figure 2. Location map of the KGP deposits and prospects within the Katanning Greenstone Belt.
Figure 3. Modelled gold assay data from the Jinkas Deposit illustrating isoclinal folding of gold with higher grade shoots occurring within the hinge zones (looking north).

Utilising Leapfrog’s® implicit modelling functionality, structural observations from drill core and interpretations of aeromagnetic data were also integrated into the modelling process. This enabled field observations to be tested against theoretical data. The recognition of the key structural characteristics that control the location of mineralisation within the region helped develop a new exploration model from which to refocus exploration efforts within the KGP.

Datatine Prospect 3D Model

The Datatine prospect is located 7 km north along strike from the Jinka’s and Dingo deposits. The stratigraphy is rotated and strikes E-W rather than the NNW strike of the main resource area. Datatine is marked by gold-in-soil anomalies that correspond with an interpreted NE striking synform. To improve the interpretation of mineralisation and the underlying geology, surface aeromagnetic datasets were combined with 3D magnetic inversion modelling along with available drilling data within Leapfrog®. This resulted in a refined fold geometry and interpretation with the interpreted synform reflecting the underlying magnetic granite-greenstone contact. 3D modelling of the gold grade distribution from drillhole data demonstrated that the geometry of gold mineralisation followed the synformal trend and provided a basis for further drill targeting (Figure 4). The 3D data integration, modelling and analysis enabled the exploration target size to be assessed early in the project development cycle and also facilitated better drill planning. With rapid updates to the 3D model as new results are received, the dynamic understanding of geological controls that lead to the grade distribution within provided for commission of a successful and effective exploration process at Datatine.
CONCLUSION

Developing geological knowledge from sparse data is key to successful mineral exploration. The collection, storage and interpretation of datasets is vital to the sustained success of the exploration process. Detailed analysis of drillhole data and integration with structural mapping, geophysical datasets and 3D modelling allows for robust interpretations of the subsurface to be developed. The KGP case study illustrates how 3D modelling provides a rapid low cost method to integrate larger datasets in the development of geological models.

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THE AUSTRALIAN NATIONAL VIRTUAL CORE LIBRARY: INVESTIGATING MINERAL SYSTEMS ACROSS THE AUSTRALIAN CONTINENT

CARSTEN LAUKAMP1, SURAJ GOPALAKRISHNAN2, MONICA LEGRAS3, DAVID GREEN3, LENA HANCOCK4, IAN LAU1, PETER MASON1, ALAN MAUGER5, DAVID TILLEY6, BELINDA SMITH7, PETER WARREN1, ROB WOODCOCK1

ABSTRACT

The characterisation of mineral assemblages is amongst the first steps for describing and analysing mineral systems. The identification of lithologies and alteration footprints usually starts with time consuming core logging next to drill rigs or in drill core sheds across Australia. Notwithstanding the geological expertise, the results of visual logging can be very subjective and are difficult to use for decision making, let alone in a quantitative way for resource characterisation. Detailed thin section analyses or X-ray diffractometry of selected samples are commonly used to characterise mineralogy. However, in many cases, mineral assemblages are calculated from geochemical data. This can deliver ambiguous results, especially when considering areas prospective for hydrothermal ore deposits that were overprinted numerous times or experienced considerable weathering. In contrast, modal mineral abundances can be obtained by calibrating drill core hyperspectral data with, for example, quantitative X-ray diffractometry or quantitative evaluation of minerals by scanning electron microscopy (QEMSCAN). Mineralogy is key and there’s now a substantial mineralogical database that covers many of the mineral systems present in Australia, termed the National Virtual Core Library (NVCL).

The NVCL is part of AuScope’s national earth science infrastructure program (http://www.auscope.org.au/nvcl/) and comprises close to 1,000 km of hyperspectral drill core data with continuing acquisition. Sourced from various geological environments and mineral deposit types across the Australian continent (see Figure 1); Huntington, 2016), the 800 spectral channel, line-profiling data are collected at approximately 1 cm resolution using HyLoggerTM systems (Schodlok et al., 2016) located at six NVCL nodes operated by various state and territory Geological Surveys. It is arguably one of the world’s largest collections of publicly available mineralogical data with associated high-resolution drill core imagery.

The prime objective of the NVCL has been to provide internet access to the vast resource of geological information from the upper 1 – 2 km of our earth’s crust that is stored in drill core libraries and core sheds across Australia. The hyperspectral drill core data are available to the research community and resources sector via AuScope’s data infrastructure and discovery portal (http://www.auscope.org.au/auscope-grid/), the Australian geoscience portal (http://portal.geoscience.gov.au/gmap.html) and directly from the state and territory Geological Surveys.

This paper demonstrates the huge potential of hyperspectral drill core data contained within the NVCL database for addressing the challenges of the Australian resources sector. Recent examples for applications include the mapping of hydrothermal alteration footprints associated with orogenic gold deposits (Arne et al., 2016), sediment-hosted gold (Wells et al., 2016), nickel sulphides (Burley et al., 2017), volcanic-hosted massive sulphides (Duuring et al., 2016), iron oxide copper-gold deposits (Mauger et al., 2016) as well as resource characterisation of nickel-cobalt laterites (Cracknell et al., 2016).

1 CSIRO Mineral Resources
2 Geological Survey of Queensland
3 Mineral Resources Tasmania
4 Geological Survey of Western Australia
5 Geological Survey of South Australia
6 Geological Survey of New South Wales
7 Northern Territory Geological Survey
Analysis of hyperspectral data from single drill cores can provide important background information prior to commencement of drilling programs (Gordon et al., 2016). Regional-scale, NVCL-based studies provide detailed insights into the complexity of mineral assemblages that are usually missed in conventional sporadic sampling. These larger drill core data sets enable, for example, mapping the extent of diagenetic overprints, such as in the McArthur Basin (Smith et al., 2017), but also of regional metamorphism and alteration, such as in the Central Lachlan Orogen (Downes et al., 2016). Furthermore, hyperspectral drill core data have provided useful insights into the understanding of hydrocarbon systems (Ayling et al., 2016; Hill and Mauger, 2016). Finally, Fox et al. (2017) showcase the geo-environmental applications of hyperspectral drill core data. The great variety of case studies and the vast potential for integrating the hyperspectral with other drill core data (e.g. petrophysical data) highlight why the Australian made earth science NVCL infrastructure should be part of every geologist’s toolbox.

**ACKNOWLEDGEMENTS**

The NVCL is supported by AuScope (www.auscope.org), the Australian Government National Collaborative Research Infrastructure Strategy (NCRIS), the CSIRO and Australian state and territory geological surveys. We would like to express our sincere thanks to those who successfully initiated the NVCL over 10 years ago and helped to build a strong infrastructure that new generations can build upon. First and foremost, we thank Jon Huntington (former NVCL Director) and Lew Whitbourn (former Hyperspectral Engineering Team Leader) who made the NVCL with their teams possible. Furthermore, we’d like to acknowledge all those hard-working people that collect and process the NVCL data on a day to day basis, helping to ‘uncover the mineralogy of the top two kilometres of the Australian continent’.
REFERENCES


‘ALL THAT GLITTERS IS NOT GOLD’:
EXPLORING FOR MORE THAN THE PAY DIRT WHILST DRILLING FOR A SUSTAINABLE FUTURE

JUSTIN LEGG

ABSTRACT

Groundwater accounts for approximately 0.6% of the earth’s fresh water supply and has increasingly been used as a source of water since the agricultural (or green) revolution of the 1960’s (Calder, 2005; National Ground Water Association, 2016). At least 50% of the global population’s drinking water is currently being sourced from groundwater. Groundwater extractions provide 43% of all water for irrigation used for agriculture, and currently accounts for 46% of Perth’s water supply (Franek et al. 2015; WaterCorp, 2016).

Currently the various Australian state governments manage groundwater and aquifer data in accordance with the 2007 Water Act, whilst the Bureau of Meteorology (BoM) manages the National Groundwater Information System (BoM, 2016; De Sousa, 2014).

To help better manage this increasingly important, yet impalpable resource, explorers of all disciplines have the opportunity to contribute invaluable geological and drilling data that can be used to assist the various catchment managers, government agencies and the BoM to provide more accurate information and thus develop more effective water governance strategies.

INTRODUCTION

Though the earth may be known as the ‘Blue Planet’ due to its water content, less than 0.01% of water is currently available in rivers and lakes. With residence times in these systems being comparatively short (i.e. less than a decade), they are vulnerable to changes in climate and associated rainfall patterns (de Chaisemartin et al. 2017; National Ground Water Association, 2016).

As 70% of this available water is utilised for agricultural purposes globally, there will be increasing demand for water as the population is predicted to increase by 30% to 9.6 billion by 2050 (Calder 2005; Food and Agriculture Organization of the United Nations, 2002). Currently, 50% of the global population’s drinking water is sourced from groundwater, with groundwater extractions providing over 40% of water used in irrigation for agriculture (Food and Agriculture Organization of the United Nations, 2002; Franek et al. 2015). In an Australian context, groundwater currently accounts for 46% of Perth’s water supply (WaterCorp, 2016). An obvious solution is to augment surficial water supplies with groundwater – which accounts for approximately 0.6% of the earth’s fresh water supply (National Ground Water Association, 2016).

In a modern context, this augmentation was undertaken during the 1960’s. The confluence of improved drilling techniques, introduction of hybrid strains of wheat, rice, and corn (maize) and the wholesale adoption of agri-chemicals (such as fertilisers and pesticides) saw the advent of the “Green Revolution” that sought to end world hunger (Food and Agriculture Organization of the United Nations, 1996). Facilitated by broad neo-liberal economic mechanisms that encouraged a pro-development productivist paradigm, socio-economic factors usually trumped environmental considerations (Calder, 2005; Wilson, 2004).

Subsequently, due its impalpable nature, and its reluctance to recognise established geopolitical boundaries, groundwater fell victim to a ‘tragedy of the commons’ (Hardin, 1968) whereby consumers abstracted groundwater solely in their self socio-economic interest, rather than considering adjacent users and/or longer-term implications of this behaviour (Foster & Garduño, 2013).
Given this invaluable resources’ influence on agriculture and drinking water, it is alarming to consider that groundwater governance (and by extension trading structures) were not introduced in Australia until the 1996 Council of Australian Government (COAG) meeting (McKay, 2006; Ross, 2016) – especially given Australia’s reliance on groundwater resources to augment surficial water supplies as shown in Figure 1.

*Figure 1. Groundwater use at a national scale as a percentage of total water use (Harrington & Cook, 2014).*

In 2000 the United Nations, concerned about increasing global population growth (expected increase by 30% by 2050), changes in rainfall distributions, and unregulated groundwater extractions - called for a “Blue Revolution”. This ‘revolution’ seeks to:

“...address wider issues of sustainable water management such as drinking water, food production, ecology and environment for subsequent generations.” (Calder, 2005; Food and Agriculture Organization of the United Nations, 2002).

As the spectre of climate change (and its associated impacts) is at the forefront of the political agenda, this paper will identify opportunities for the resources industry to contribute towards the development of effective groundwater management strategies, not only for the community at large, but for the miners themselves -providing a better baseline for their own water management and be seen to take action and responsibility for a finite natural resource on behalf of the community. It is an endorsement of their social licence and with time is likely to become a requirement to operate in the mining and exploration sector.

**AUSTRALIA: THE DESERT COUNTRY?**

Australia is endowed with several groundwater basins (Figure 2) including:

- The Great Artesian Basin, which covers some 20% of the continent.
- The Murray–Darling Basin, which is considered the food bowl of Australian Agriculture.
- The Canning Basin of northern Western Australia.
- The Daly Basin of the Northern Territory.
- The Otway Basin aquifers of south-east South Australia and south-west Victoria.
These groundwater basins are divided into broad groundwater management areas as shown in Figure 3.
MANAGING THE UNMANAGEABLE: WHAT IS CURRENTLY BEING DONE?

The 2004 COAG agreement established the National Water Commission (NWC) to implement the National Water Initiative (NWI) – “a national blueprint for water reform in Australia” that entailed provisions for secure water access entitlements, as well as integrated water management planning to incorporate social, economic and environmental considerations (De Sousa, 2014; Ross, 2016).

Under this initiative, the 2007 Water Act (Water Act 2007) was enacted, which seeks to enable the Commonwealth and state governments to manage and promote water resources “in a way that optimises economic, social and environmental outcomes” as well as “return to environmentally sustainable levels of extraction for water resources that are over-allocated or overused” (Water Act 2007).

Under the 2007 Water Act, state and local governments are required to develop legislation and guidelines that meet this objective – for example the Queensland Water Act 2000 requires local government areas to develop Water Resource Plans (WRP), Resource Operation Plans (ROP), Water Use Plans (WUP), Land and Water Management Plans (LWMP), manage permits and licensing, as well as monitor water allocations (Great Barrier Reef Marine Park Authority, 2013; Lower Burdekin Water, 2016).

Predictably, the NWC was abolished in 2015 with the Productivity Commission, the Bureau of Meteorology (BoM) and the Australian Bureau of Agricultural and Resources Economics and Sciences (ABARES) absorbing its functions.

Currently the various state governments manage groundwater and aquifer data in accordance with the 2007 Water Act, whilst the BoM manages the National Groundwater Information System (De Sousa, 2014).

The BoM now provides a wide variety of water information products including historical information and trends, current status and predicted forecasts to assist planners on both a local and national perspective (BoM, 2016 and 2017).

CURRENT STATUS OF GROUNDWATER MONITORING

There are some 160,000 registered bores in Queensland as shown in Figure 4, with bores coloured by type (BoM, 2017). Of these, some 23,000 are Queensland government-owned groundwater monitoring bores (Figure 5), of which half have fallen into disrepair and/or are over 30 years old (BoM, 2017; Loussikian, 2013).

*Figure 4. Location of registered water bores in Queensland by type (BoM, 2017).*
Whilst this may appear alarming, the BoM does obtain groundwater-monitoring data such as salinity and groundwater levels from private bores, drains etc. as shown in Figure 6.

Figure 5. Location of Queensland Government groundwater monitoring bores (BoM, 2017).

Figure 6. Location of Queensland BoM groundwater monitoring sites (BoM, 2017).
GEOLOGY TO THE RESCUE AGAIN

Geology has an uncanny knack of affecting environmental politics.

The eruption of the Toba Caldera Complex in Sumatra some 74,000 years ago is considered one of the biggest volcanic eruptions of the Quaternary (Chesner, 2012). This event provided the impetus for the theory of a ‘Volcanic Winter’ (Dörries, 2008; Ninkovich and Donn, 1976) whereby a magma-eruptions eject sufficient material (in this case some 2,800m³ in a single eruptive event) into the atmosphere to alter the climate (Chesner, 2012). This concept was later developed by Carl Sagan and his team in the 1980’s to describe a ‘Nuclear Winter’ that was predicted to result from bombardment by atomic weapons (Dörries, 2008; Sagan and Turco, 1993). It was this concern about climate change that formed one of the foundations of the mutually assured destruction (MAD) doctrine that formed the basis of nuclear ordnance and geopolitical policies during the latter part of the cold war (Knorr, 1985).

Hence it does not seem unreasonable that geologists can add meaningful discussion to the problem pertaining to groundwater.

THE TIDE IS TURNING

In December 2016, the Australian Bureau of Statistics (ABS) stated that exploration spending had increased during the December quarter 2016 by 3.4% ($12.0 million) to $369.5 million, with drilled metres increasing by 2.9% (seasonally adjusted) as shown in Figure 7 and Figure 8 (ABS, 2017). This trend along with changes in the global geo-political climate may indicate a potential increase in drilling and exploration activities (Ingram, 2017).

Figure 7. Seasonally adjusted exploration spending in Australia, December 2016 (ABS, 2016).

Figure 8. Seasonally adjusted exploration drilling in Australia, December 2016 (ABS, 2016).
As geologists return to logging core, sieving chips and wandering up streams from the “post mining boom” contraction that has characterised the last five years (Connolly and Orsmond, 2011; Downes, Hanslow and Tulip, 2014), they have an opportunity to assist with groundwater management by simply considering it as part of their ongoing activities.

Notably, recent changes to the Queensland Water Act 2000 as of December 2016 now require that:

“Mining companies that take or interfere with groundwater under this right will be subject to the obligations that currently apply to the P&G sector under Chapter 3 of the Act. This means landholders whose water bores are affected by the take of associated water will be protected by statutory certainty that companies will have to enter ‘make good’ agreements.” (Department of Natural Resources and Mines, 2017).

Hence the necessity for explorers and resource companies to start to consider water observations as part of their drilling strategy.

The BoM in concert with Geoscience Australia has developed the “National Aquifer Framework” – a nationally recommended set of terminology for naming and grouping hydro-geologic units in Australia (BoM, 2013 and 2016). By incorporating these codes into logging codes etc., not only could discrepancies between terms to describe geological units be reduced (by utilising a standardised nomenclature and coding system), but also the information could be utilised by third parties (such as the BoM) without the need for costly data formatting and/or translation.

**WHAT’S IN IT FOR ME?**

The obvious corollary of this suggestion is “What’s in it for me?” With limited exploration funding opportunities (Ingram, 2017), managers may be reluctant to consider the costs involved to alter established logging codes and database structures, for no immediate return.

Whilst this is a tangible argument, and the response proposed here somewhat esoteric, the longer-term outcomes (as opposed to the immediate outputs) may be far more beneficial than first thought.

**See It Once, Measure It Twice**

A study of 105 feasibility studies in 2003 identified a number of common commissioning and operational problems arising from inadequacies in the feasibility studies of projects (McCarthy, 2003). Some 35% of errors were attributed to geology, geo-tech, hydrogeology and metallurgy (Berry, 2009; McCarthy, 2003).

By considering hydrological observations in drill logs (via downhole geophysics, use of the ‘National Aquifer Framework’ coding and nomenclature etc.) geologists may consider observations that were previously overlooked – what has often been referred to as “the Data Gap” (Laing, 2008), thus reducing this potential for errors.

**Connecting to the Local NRM**

Landcare Australia was an initiative conceived in November 1986 by Joan Kirner, (then Victorian Minister for Conservation, Forests and Lands) and Heather Mitchell, (then President of the Victorian Farmers Federation). Since these humble beginnings, Landcare Australia has developed into a fundamental component of Australian Government’s commitment to natural resource management (NRM) – with the Australian Government investing $1 billion over four years from 2014-15.

Currently there are 56 regional NRM organisations across Australia that act as delivery agents under the regional stream of the national landcare program.

By collaborating with the local Landcare group, the data collected from field-work (drill logs, stream sediment results, soil maps etc.) could be utilised by the greater community. In doing so, an explorer has assisted obtaining and maintaining their social license to operate (SLO), which can be defined as:

“When a project has ongoing approval of the local community and other stakeholders.”

(Quigley and Baines, 2014).

Additionally, by considering hydrogeological aspects at the grass roots exploration stage, longer-term base line data (such as groundwater levels) for a particular prospect could be collected, thus reducing the
potential for conflict regarding water management as has been seen in the past between mining companies and the agricultural community (McCarthy and Gunders, 2016).

CONCLUSION

The proposed strategy presented here advocates explorers considering more than a pay-dirt intersection. Rather, this proposed strategy advocates the use of ‘National Aquifer Framework’ nomenclature in drilling logs and field codes so as to reduce ambiguity, as well as make the data transferable to third parties such as the BoM.

By collaborating with the local Landcare group in a particular prospect, the data (and or selected drill holes themselves) could be used for groundwater monitoring purposes, thus proving ‘free’ information to NRM decision and policy makers.

Whilst there is an assumed time: cost disbursement that is unlikely to recovered, the development of a social licence to operate may prove move valuable.

Ultimately, climate change is here to stay (Marder, 2013). By recognising that groundwater management that is essential for water supply augmentation (as shown in Figure 1), observational data can be collected that is not only beneficial for the exploration objective (i.e. opening a mine) but also for the greater good.

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Expanding for More Than the Pay Dirt Whilst Drilling for a Sustainable Future

J. Legg

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INTRODUCTION

Drillholes are very expensive and represent not only the direct drilling cost, but also the cost of each step in the exploration process to that point. The immediate (or realised costs) often belie the actual (the realised and unrealised costs) of the exploration and consultation process, which can add up to several thousand dollars per metre.

Table 1 provides an approximation of the immediate or realised cost of a 350 m diamond core drillhole (DDH) of NQ size.

Table 1. Example of the realised costs of diamond drilling.

<table>
<thead>
<tr>
<th>Item</th>
<th>Type</th>
<th>Unit</th>
<th>Unit Cost ($)</th>
<th>Units</th>
<th>Total $</th>
</tr>
</thead>
<tbody>
<tr>
<td>Drill rate</td>
<td>Realised</td>
<td>Meter</td>
<td>120</td>
<td>350</td>
<td>42,000</td>
</tr>
<tr>
<td>Geology</td>
<td>Realised</td>
<td>Day rate</td>
<td>700</td>
<td>5</td>
<td>3,500</td>
</tr>
<tr>
<td>Assays</td>
<td>Realised</td>
<td>Sample</td>
<td>40</td>
<td>25</td>
<td>1,000</td>
</tr>
</tbody>
</table>

As shown in Table 1, these realised costs are not insignificant and represent a significant investment on behalf of the shareholders. These realised costs however do not account for the huge investment of time and knowledge that has gone into targeting the drillholes. These unseen costs include:

- Data compilation and verification.
- Integration and formulation of the exploration model.
- Site clearance considerations.
- Cultural heritage negotiations.
- Statutory obligations.

These unrealised costs of the exploration process are presented in Table 2, which indicates that the costs (and therefore financial risks) are considerably higher than the cost of the drillhole itself.

Table 2. Example of the pre-drilling exploration costs.

<table>
<thead>
<tr>
<th>To get to this point, requires exploration.</th>
<th>Approximate cost</th>
</tr>
</thead>
<tbody>
<tr>
<td>Review of previous exploration and public datasets.</td>
<td>$28,000</td>
</tr>
<tr>
<td>Mapping</td>
<td>$21,000</td>
</tr>
<tr>
<td>Geochemistry. Soils, Streams</td>
<td>$20,000</td>
</tr>
<tr>
<td>Geophysics. Magnetics, IP.</td>
<td>$132,000</td>
</tr>
<tr>
<td>Approximate total cost:</td>
<td>$201,000</td>
</tr>
</tbody>
</table>

The purpose of this talk is two-fold:

- To examine what we as explorationists can do to maximise the benefit of each dollar spent in the exploration process, and in particular the drilling dollars.

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1 Technical Assessor, DNRM
To explore (pun intended) the role the government can play in reducing the upfront early exploration risks, make more data and knowledge available to companies, and make exploration a more attractive investment option – as well as hopefully leading to the successful discovery of new deposits.

**DATA MINING**

The use of analytic hierarchy processes such as Data-Information-Knowledge-Wisdom (DIKW) hierarchies is commonly used to develop exploration strategies i.e. knowledge management. These DIKW hierarchies (Figure 1) are integrated into assessment scenarios that are used for target ranking and decision-making (Schomaker, 1997; Mendoza and Prabhu, 2001).

*Figure 1. Analytic hierarchy processes (after Schomaker, 1997).*

By including additional data, capacity for information is increased as presented in Figure 2. In essence with more (good quality) information you can make better informed models and decisions and lower exploration risk.

*Figure 2. Increased capacity by the inclusion of additional data (after Schomaker, 1997).*
The acquisition of this additional (or historical data) is significantly cheaper than the costs of re-drilling a hole as outlined previously. The following examples illustrate where the judicious use of historical data in concert with contemporary data has led to significant resource developments.

**Case Study: Pasminco North No. 1 Mine, Broken Hill**

The North No. 1 pit area in Broken Hill had been mined intermittently since 1880’s with underground and later open pit operations concluding in 1992 (Liddy, 1987; Birch et al., 1999). In late 1999, Pasminco undertook an internal feasibility study to assess the potential remaining resources in three lens. This investigation entailed the compilation of historical drill data, re-logging of historical drill core as well as the drilling of 8 reverse circulation (RC) holes.

For a paltry drilling cost of $22,300 (1999 figures), a Mineral Resource of 195,000 t @ 6.2% Pb and 7.4% Zn was estimated (Legg, 1999). In this instance, access to historical core, geological logs and assay data was key to the project’s success.

**Case Study: ActiveEx**

ActiveEx is a junior explorer based in Brisbane focussed on gold and base metals in Queensland, and potash in WA. Proper preservation of drill core from previous exploration campaigns allowed ActiveEx to create a mineral resource estimate in accordance with the JORC Code at the Coalstoun porphyry copper deposit in Queensland. This was done through re-logging, re-assaying, re-interpretation and adding new density and petrographic information – all without the cost of drilling a new hole (Pers comm, J Hugenhotz, ActiveEx website).

*Table 3. Mineral Resources for Coalstoun copper deposit (from ActiveEx website).*

<table>
<thead>
<tr>
<th>Category</th>
<th>Domain</th>
<th>Tonnes (Mt)</th>
<th>Cu (%)</th>
<th>Cu (t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Inferred</td>
<td>Supergene</td>
<td>6.99</td>
<td>0.47</td>
<td>32,692</td>
</tr>
<tr>
<td>Inferred</td>
<td>Primary</td>
<td>19.87</td>
<td>0.35</td>
<td>69,984</td>
</tr>
<tr>
<td>Inferred</td>
<td>Total</td>
<td>26.86</td>
<td>0.38</td>
<td>102,677</td>
</tr>
</tbody>
</table>

In short, drillholes and the data and samples collected from them represent a significant investment of time, money, people and learning / experience.

**Case Study: CMR Coal**

There are cases where good quality pre-existing data has been used to make a new discovery. Re-logging of a 1966 GSQ drillhole (Figure 3) and reinterpretation and correlation with more recent drillholes, led to the identification at Dawson West of the first potentially mineable coal resource in the Moolayember Formation (Pers comm N Williams, Exploration Data Centre staff & CMR Coal website). This work resulted in an Inferred Resource of 120 Mt in the M Seam, with an average seam thickness of 2 m and maximum seam thickness of 2.8 m.

**PRESERVING HISTORICAL DATA**

To maximise the return on this investment, everything possible should be done to keep drillhole data (core, chips, assays, downhole logs) in a condition that reflects this investment and preserves the data to maximise the return on this investment. Not only for the explorer, but also for future explorers, which often includes the original explorer with a new perspective, or capability through updated models or better technology.

To have high quality well–preserved physical samples/records from an area (in a convenient location), is in effect providing very low cost access to an area, in addition to the existing data, new samples can be taken and core can be re-logged in light of new exploration concepts. All of which provides an up to date equivalent to the previous logs/assays and a baseline for interpreting data gathered at the same time and subsequently. This data can be used to test the hypotheses that drove the original exploration effort and collect modern data to support or replace it, and getting the maximum value from the previous data and the new data.
Figure 3. Queensland GSQ Drillhole log for DRD 7.
THE COST OF NEGLECT

Taking the costs presented earlier, the full (realised and unrealised) costs of that 350 m drillhole is $707/m. Figure 4 shows $5,657 effectively left lying in the bush, not to mention the unrealised future potential, the lost data, and the cost of redrilling this hole (should there be a discovery in the area), which given the area was selected for a drillhole is more likely.

Discarded core can also be considered unprofessional. We increasingly require high standards of accuracy, reporting, responsibility and integrity in exploration. Why then at the end of a project do we throw away the ultimate expression of our work as explorationists, without regard for its longer-term utility?

It is possible now to buy core trays that sit perfectly stable on top of each other sealing the tray beneath, all UV stabilised and very tough. Place these in a very basic shed and for approximately $3.50/m you have well preserved core, a cost less than 0.5% of the full per metre drilling cost to this point.

Figure 4. The cost of neglect.

SOME CAUTIONARY COMMENTS ABOUT DATA MINING

The 2012 JORC code outlines the MINIMUM requirements for public reporting of exploration results and resource estimates. The incorporation of historical data falls under this auspice. Hence consideration should be given to:

- an understanding of historical assay techniques.
- the historical assay technique used – and are these compatible with the current exploration philosophy.
- the inclusion of quality assurance and quality control data and assay certificates for data verification.
WHAT CAN BE DONE?

In an ideal world, explorers would adhere to an industry-accepted standard of practice for the recording and preservation of exploration data. This data would be retained by a central authority without bias, for the benefit of the people of Queensland and future explorers. There are a number of benefits to having this repository of high quality data:

- Easy low cost early assessment of the exploration potential in the state.
- Over time a high-quality database to use in testing new ideas/approaches and methodologies.
- Project generation.
- Long term independent data archive/backup for explorers.
- Easier data room set up and mineral project marketing.
- Quality assured data set.

Part of what is required is clarifying what state governments are currently able to do, the benefits of having this data and the impediments. Also, explorers/miners, the professional organisations and the industry they represent, must think about their current and future exploration efforts, and how to get the maximum value from their exploration work now and into the future. It is easier for potential investors to make a decision in favour of your project if they have access to a good quality, unbiased data set for the area.

This requires thinking about:

- What is best practice for drill data recording and maintenance.
- Who will be the champion and the guardian of this code of practice.
- Is the Government the best steward for the samples and data.
- This will require input from industry, service providers and the government.

WHAT CAN THE GOVERNMENT DO?

The government can:

- Compile historical data into exploration datasets e.g. Mt Isa Inlier (2001, 2010), North Queensland gold and base metals (Georgetown – part 1, Charters Towers – part 2), South East Queensland, Yarrol Conns Central Queensland.
- Assist with advice for storage strategies.
- Act as a custodian for historical data.
- Potentially facilitate core storage hubs in regional centres.
- Assist with ongoing industry education of the need for good data management practices.
- Develop an industry standard/guideline.

HOW?

Governance can be defined as:

“...the exercise of economic, political and administrative authority to manage a country’s affairs at all levels. It comprises the mechanisms, processes and institutions through which citizens and groups articulate their interests, exercise their legal rights, meet their obligations and mediate their differences.” (Foster, S. et al. 2010; ICPS 2016; UNDP 1997).

Hence the role of any government is to enact and enable these principles. If we the exploration and mining community articulate the need for better data and sample management to reduce costs and risks to exploration to the government – woe betide a government who doesn’t listen as we have seen in recent years.

REFERENCES


DATA UPCYCLING

JULIAN VEARNCOMBE

INTRODUCTION

Mineral exploration has witnessed recent and critical paradigm shifts, starting at the time when geological mapping began to drive exploration (1930s - 1950s), followed by the development of effective drilling techniques that allowed deep and under-cover deposits to be discovered (1960s to early 1980s). In the late 1990s and 2000s, digitisation of information, computer databases and 3D visualisation enabled the rapid and detailed understanding of how mineralisation behaves in three dimensions. And, referred to as “Big Data”, computational and technological advances are now allowing the collection of an ever-greater variety of data, in large quantities, at high speed, and reduced cost (Riganti and Vearncombe, in press). The consequent large datasets may be analysed computationally and are a focus of mining companies, consultants, software- and hardware-suppliers.

Here, I emphasise the role of “data upcycling” as a significant contributor to modern exploration. With the large volumes of new data routinely collected, it is easy to overlook previously obtained datasets, especially those acquired on paper prior to digitisation and often forgotten after passing through a myriad of agencies or companies. Older geological data have a wealth of applications once “upcycled” to fit present day workflows. Upcycling can take three basic forms:

1. Re-collecting data, for example the re-logging or re-assaying of remaining diamond core.
2. Adding to or improving existing data. This may include the analysis of old core with new methods, such as multi- or hyperspectral loggers, or more accurately locating historical collars using a differential global positioning system (DGPS).
3. Using data differently. ‘Old’ data may be re-processed using improved algorithms and/or new analytical techniques to reveal previously unrecognised patterns. This is common place in the geophysical industry, especially with seismic data.

CASE STUDY

I illustrate data upcycling with a recent published case study, at Mt Mulgine, Western Australia (Vearncombe et al., 2016). Significant Resources have been announced to JORC standards (Hazelwood Resources announcement to ASX, 5 November 2014; JORC 2012) located within 2 km of each other by using legacy data and without requiring the client to mobilise a drill rig. Mt Mulgine is an Archaean porphyry tungsten–molybdenum system, comprising endoskarn and exoskarn (at Mulgine Hill) and stockwork vein (at Mulgine Trench) mineralisation (Migisha and Both, 1991; Conner et al., 2012; Vearncombe et al, 2016). Tungsten exploration in the area dates back to the 1960s, with the majority of data available today collected in the 1970s and early 1980s. Exploration in the 1980s and 1990s focused on gold, with the result that of the 318 holes drilled over a number of successive campaigns not all holes were assayed for tungsten. Most of the original, handwritten Mt Mulgine diamond core logs are available through WAMEX, and contain the original assays. Handwritten logs are a valuable resource as notes and drawings of individual geologists cannot be easily transferred to a structured database format.

Data available for validation consisted of legacy diamond drilling (drilled 1972–1981) and RC drilling (drilled 2008–2014). A large proportion (about 84%) of the legacy diamond core remains stacked, racked and protected from the weather and in excellent condition given the time elapsed since drilling. Protected physical core is undoubtedly superior to legacy digital data of uncertain provenance. As part of the validation process, core was systematically re-marked with a conversion from feet and inches to metres, and the lithology, weathering, veining and mineralisation re-logged. Representative selections (about 6.5%) of the core were re-sampled to assess the reliability of original assays, and for the first time, specific gravity data were collected (Figure 1). Past work had not included specific gravity measurement and hence no modern resource evaluation was possible without new data. This was easily achieved using the legacy core but would have not been possible had the core not been preserved.

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Once the rigorous data validation exercises had been completed, mostly in the field, spreadsheets were amalgamated into a new database, and the data imported into standard exploration software for geological modeling. Verified data were used to create mineralisation wireframes and 3D geological models at both Mulgine Hill and Mulgine Trench. Drillhole data in combination with georeferenced surface maps, shaft diagrams and cross-sections with structural data enabled a confident interpretation of the geology. The modeling techniques employed at Mt Mulgine include both explicit and implicit methods. The implicit method used at Mulgine Trench implies defining a hypothesis that is then tested against the geology and used as a base for the tungsten mineralisation wireframes.

The Mineral Resources at Mt Mulgine have been estimated by upcycling primarily historical data. Having confidence in the veracity of the legacy data and geology was instrumental in defining the resources. At Mt Mulgine, most, but not all, of the data satisfy the test for a comparison between legacy collection science method with the modern scientific method, as authors were able to re-collect or directly verify the information (e.g. with down-hole geology and drill collars, and re-assaying of a representative selection of samples). Where legacy data did not exist (e.g. specific gravity), new data were collected from the preserved core. Re-logging diamond core drilled in the 1960s to 1980s equates to using cores from a recent program and produces similar results (although modern drilling would produce oriented core offering detailed contact, vein and fabric data). On the other hand, re-assaying has to take into account possible nugget effects. Despite some inherent weaknesses leading to a classification of the resource at Mulgine Trench as Inferred, this case study highlights how resource definition in this brownfields project is possible without having to incur the cost of additional drilling, but simply by upcycling and checking the veracity of existing legacy data.
CONCLUSION

Emerging technologies are reducing the human input in data collection, analysis and assessment, and are likely to change exploration in the decades to come — but to ensure ‘fit for purpose’, data must be fully integrated with geological knowledge, and be both verified and verifiable for future use. There are both opportunities and challenges for the exploration and mining industry.

ACKNOWLEDGEMENTS

We thank Angela Riganti, Dave Isles and Sian Bright for discussions that have significantly impacted our views and the content of this paper.

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