



EFEE

13th WORLD CONFERENCE
ON EXPLOSIVES AND BLASTING

KRAKÓW 2025
POLAND

PROCEEDINGS

Editor:

J. Honkanen et al



To view individual papers,
simply click on the
relevant description in the
following Table of
Contents section

Copyright © 2025 European Federation of Explosives Engineers.

All rights reserved. No part of this publication or the information contained herein may be reproduced, stored in a retrieval system, or transmitted in any form or by any means, electronic, mechanical, by photocopying, recording or otherwise, without written prior permission from the publisher.

Although all care is taken to ensure the integrity and quality of this publication and the information herein, no responsibility is assumed by the publishers nor the author for any damage to property or persons as a result of operation or use of this publication and/or the information contained herein.

Published by: European Federation of Explosives Engineers.
www.efee.eu

ISBN 978-0-9550290-9-7

Produced in The UK by Agora Marketing Communications Ltd.

Table of Contents

Foreword by the Chairman of the Technical Committee	VII
Acknowledgement	IX

1. Blast vibration and seismology

Models of blast vibration <i>D.P. Blair</i>	3
Comparison of international vibration standards for quarry blasting close-in to a residential structure in Michigan, USA <i>C. Aimone-Martin, C. Dowding, D. Rudenko, C. Loukides, S. Newman, J. Gjødvad & M. Jern</i>	13
EFEE Environmental committee vibration standards comparisons - current update <i>C. Aimone-Martin, J. F. Gjødvad, M. Jern & T. Moser</i>	23
Effect of blasthole diameter on vibration from signature hole blast monitoring <i>R. Yang, J.P.R. Proulx, J. Desrochers & L. Etulle</i>	33
Optimising signature hole analysis: evaluating charge quantity and confinement effects on seed waveforms <i>T. Moser</i>	43
A hybrid modelling approach to vibration prediction <i>M. Jern & M. Fagerlund</i>	53
Perforating drill holes as additional method of reducing ground vibrations level <i>A. Brzozowski</i>	63
Compliance of blast vibration standards: the predominant role of weather and seismic activity in residential crack displacement <i>R. Gómez & J. Gjødvad</i>	75

2. Blasting work experiences

The critical importance of the bottom 30 centimeters of the blasthole in close-in construction blasting <i>J. Wallace</i>	87
Challenges and restrictions of drilling and blasting in the Arctic Circle <i>A. Gagnon & A. Pina</i>	97

Investigation of sheared boreholes in the Kiirunavaara Mine <i>H.C. Vikum Sri Mendis, N. Petropoulos & C.R. San Miguel</i>	107
Effect of detonation propagation and blast confinement on vibration and VOD in hard rock <i>T. Hänninen & M. Aarnio</i>	121
Snowy 2.0 pumped-hydro project - Marica upstream surge shaft <i>R. Domotor & M. Parkinson</i>	131
Paper or electronic records - ways to improve blast documentation and record-keeping <i>T.A. Davidsavor, R.E. Burnham, J.K. Ratliff & C. Aimone-Martin</i>	151
The influence of mine water on blast design - Paper 2 <i>T.N. Little</i>	163
A methodology to define drill and blast designs for drawbells at the Chuquicamata underground mine <i>A. Berg, P. Arriagada, R. Cortés, N. Jamett, J. Henríquez, H. Cortés & D. Morales</i>	175
Impact of using water gel explosives (Aquagels) compared to heavy ANFO in open-pit mining operations <i>E. Núñez Recabal, R. Gil Greene, L. Manríquez & E. Tobar</i>	183
Electronic initiation system applied in tunnel construction: focusing on the contour holes and decking mode <i>S. Kara, M. Laing, S.J. Lee, Y. Park & N. Hwang</i>	191
 <i>3. Demolition blasting</i>	
Numerical simulations for enhanced precision and safety in structural blasting <i>A. Gerat & C. Rapps</i>	203
Precision demolition: executing large-scale multi-structure power plant implosions <i>A. Konya, R. Gilbert & C. Moles</i>	209
Prediction of vibration levels from falling masses using DIN 4150-1 in practice <i>C.A. Andersen</i>	219
 <i>4. Drill and blast design methods and modelling</i>	
Influence of empty guideholes in single-blasthole experiments for tunnel blasting <i>C. Rodrigues San Miguel, C. Yi, N. Petropoulos & D. Johansson</i>	229
Development of Scaled Damage Criteria for Blasting <i>J. Alvarez</i>	237

Optimisation of blast design in complex geological environments via advanced digitalisation <i>S. Taoum, E. Coarita, O. Riahi & G. Lokou</i>	249
Impedance-based energy optimisation for improved fragmentation and NOx management <i>J. Navarro, L. Martínez, P. Vergara, P. Couceiro, F. Tapia, J. Blázquez, P. Ravanal, C. Astorga, J. Madrid & M. Galleguillos</i>	261
Techniques for quarrying dimension stones through the controlled use of explosives <i>M. Cardu, C. Todaro & J. Seccatore</i>	271
AI-integrated ore body movement modelling for enhanced blast outcomes <i>R. Sahu</i>	281
Advanced edge blasting techniques and digital simulation to reduce material spillage and optimise mining efficiency <i>G.S. Huerta Valer, B. Diallo & E. Thibault</i>	291

5. Explosive detection for security

Findings on the development of efficient methods for neutralising improvised explosive devices <i>R. Laszlo, E. Ghicioi, C. Jitea, B. Simon & A. Nicola</i>	301

6. Health and safety in drilling and blasting

Prediction of flyrock distances using XGBoost machine learning method <i>Y. Ağan & T. Hüdaverdi</i>	315
A pathway to detonation of ammonium nitrate due to heat energy <i>G. Cavanough, A. Torrance & G. Stenzel</i>	325
Blasting at 120°C (250°F) in opencast coal mining using thermal insulating blast hole sleeves <i>G.J. Stenzel</i>	335
Gaseous emissions of hydrogen peroxide emulsion explosives tested in an underground mine <i>A. Järvinen, H. Kuutti, M. Rumjantsev, V. Silvonen, T. Rönkkö & T. Halme</i>	343
Perforating gun test series <i>J. Tatom, J. Romano & J. Hoffman</i>	351

7. Innovation and technical development

An empirical study on the estimation of detonation pressure by separation distance of electronic detonators using the underwater explosion impact <i>H. Kim, S. Kim, J. Choi, Y. Lee, S. Oh & B. Choi</i>	373
Wireless blasting solutions in European mining operations <i>J.A. Kreivi & J.S.C. Marijnissen</i>	387

Development of gasless lead-free time-delay compositions for electric and non-electric detonators <i>M. Gerlich, W. Trzciński & M. Hara</i>	397
HMX based electronic initiation system for blasting in extremely reactive ground <i>K.M. Dufresne</i>	409
Eliminating the TNT risk for opencast boosters <i>G.J. Stenzel & R.P.G. Nel</i>	417
The Shock-Wave Laboratory at the TU Bergakademie Freiberg <i>T. Schlothauer, K. Keller, E. Kroke, G. Heide & M. Kilo</i>	423
Pioneering transformation of explosive practices in Indonesia: overcoming geological heterogeneity with dual salt emulsion technology <i>M. Arbi & H. Prasetya</i>	435
Health, safety and environmental considerations in use of HPE explosives <i>T. Halme</i>	447
Prilled hydrogen peroxide mining explosive <i>M. Araos</i>	453
 <i>8. New applications and training</i>	
Optimising sublevel cave for sustainable mining <i>M. Lovitt</i>	463
 <i>9. Sustainability and environmental topics in drilling and blasting</i>	
Benefits of Shell GTL fluids reducing toxic emissions in emulsions explosives <i>M. Rivera-Torrente, N. Petropoulos, C. Rodriguez-San Miguel, J. Russell, M. Perryman, R. Hunt T. Wilkinson, A. Pienimäki & V. Klippmark</i>	475
Improving airblast predictability with composite wave modelling <i>A.S. Romphf & C.K. McKenzie</i>	485
Bioremediation technology combined with explosive soil treatment (bioexplosion) <i>A. Szalay, I. Zádor, J. Földesi, I. Diószegi, B. Fehér & P. Habi</i>	497

Foreword by the Chairman of the Conference and Technical Committees

As Chairman of the EFEE Conference and Technical Committees, I would like to extend my sincere thanks to all members of the Organising and Technical Committees for their dedication and commitment to the 13th EFEE World Conference on Explosives and Blasting, held in Kraków, Poland in 2025.

Together with my colleagues on the Technical Committee, we have carefully selected the contributions included in these proceedings. We received a record number of abstracts and were truly impressed by the high quality of the papers submitted. I would like to thank all authors for their time and effort in preparing the technical papers featured here.

The proceedings cover a wide range of important topics, including:

- Blast Vibration and Seismology
- Blasting Work Experiences
- Demolition Blasting
- Drill and Blast Design Methods and Modelling
- Explosive Detection for Security
- Health & Safety in Drilling and Blasting
- Innovation and Technical Development
- New Applications and Training
- Sustainability and Environmental Topics in Drilling and Blasting

This breadth of content reflects the value of the EFEE World Conference in bringing together manufacturers, shotfirers, explosives engineers, managers, and researchers from across the blasting industry to share knowledge and explore new developments.

The biannual EFEE conference offers a unique opportunity for all European and International blasting professionals to learn of new and advanced techniques and product offering, interesting project experiences, and other important developments which affect our preconditions and potential to work with explosives. As demonstrated by the papers in these proceedings, the conference has successfully fulfilled its mission. I encourage all of you to continue documenting your experiences and sharing your findings by submitting abstracts to upcoming EFEE conferences.

EFEE, Stowarzyszenie Polskich Inżynierów Strzałowych (SPIS), and our event organiser Tyler Events are proud to have hosted this important event in 2025. I trust you will find the conference in Kraków both enjoyable and professionally rewarding.

Jari Honkanen
Chairman, EFEE Conference and Technical Committees

Acknowledgement

As Chairman for the EFEE Technical Committee, I would like to extend my sincere thanks to all members of the EFEE Conference Committee, the EFEE Technical Committee, and our event organiser, Tyler Events, for their dedication, commitment, and enthusiasm in delivering the 13th EFEE World Conference on Explosives and Blasting, held in Kraków, Poland, in 2025.

In particular, I would like to express my gratitude to the members of the EFEE Technical Committee for their invaluable support throughout the paper selection process and for their efforts in editing the various contributions included in these proceedings.

Jari Honkanen
Chairman of EFEE Conference and Technical Committees

Technical Committee

Jari Honkanen (Chairman), Finland
Espen Hugaas, Norway
Kristen Kolden, US
Igor Kopál, Slovakia
Mateusz Pytlik, Poland
Jörg Rennert, Germany
Nathan Rouse, Ireland

Assisting Members

Andrzej Biessikirski, Poland
Travis Davidsavor, US

Conference Committee

Jari Honkanen (Chairman), Finland
Elżbieta Błotny, Poland
Marilena Cardu, Italy
Alan Dolan, Ireland
Johan Finsteen Gjørdvad, Denmark
José Carlos Góis, Portugal
Roger Holmberg, Sweden
Donald Jonson, Sweden
Igor Kopál, Slovakia
Krzysztof Pietkiewicz, Poland
Simon Price, UK
James Tyler, UK

1. Blast vibration and seismology

Models of blast vibration

D.P. Blair

Blasting Geomechanics Pty Ltd

ABSTRACT: Blast vibration models are discussed in detail including their relative advantages and disadvantages. In order of complexity, these models are: the vibration impact model (VIM), based solely on charge weight scaling; the Heelan solution in its various forms (standard, scaled, and a new non-linear scaled version); Waveform Superposition Models (WSM) including a new, very fast, version (FWSM) and a Monte Carlo version (MCWSM), and finally, the Full Field Solutions (FFS) which require significant computing time. The Heelan and FFS account for explosive properties, blasthole design (diameter, length, primer location) and properties of the surrounding rock mass (wave velocities, density). VIM and WSM only use the charge mass (or deck) in each blasthole as well as its delay timing and distance to a monitor. The VIM, Heelan and WSM approaches are applied to some practical blast designs. The FFS is applied to some designs for a single, large diameter blasthole.

1 INTRODUCTION

Many approaches are used worldwide for the prediction of blast vibration, including a large group of more recent ones employing Artificial Intelligence (AI). However, the predictions of such models are site-specific and do not give any physical insight to the particular mechanisms related to the various blast design parameters. Thus, AI methods are not considered. From a practical viewpoint, the most important vibration measure, especially with regard to legislative requirements and rock mass stability, is the vector peak particle velocity, VPPV, typically in mm/s. The prediction of VPPV falls into 2 major categories: the predictions due to a single blasthole (using Heelan or FFS) and the predictions due to production blasts (using charge weight scaling or WSM).

1.1 Single blastholes

Blast wave radiation from single blastholes is typically modelled as a series of stacked squat cylindrical charge elements each of length L , whose radiation is delayed by a time interval L/V_D between elements, where V_D is the explosive velocity of detonation. In the simplest of models, the displacement radiation from each element under a prescribed pressure load can be given by the solution of Heelan (1953). The pressure load, $P(t)$, as a function of time, t , sometimes used is given by the scale-independent function:

$$P_t(t) = P_{VN} (e/n)^n (\beta t)^n \exp(-\beta t) \quad (1)$$

Where: P_{VN} is the (von Neumann) borehole pressure at the detonation front, e is the exponential number, β is a pressure decay

parameter and n is an integer; the peak, P_{VN} , occurs at a dimensionless time given by $\beta t = n$. The form of equation (1) was originally chosen based on past experimental data for use in dynamic modelling (Shastri & Malumdar 1982, Grady *et al.* 1980, Larson 1982). However, Trivino *et al.* (2009) have (justifiably) criticised functions of this form because the shape has a large rise time not typical of the shorter rise times seen in more recent experimental data (see, for example, Cavanough & Onderra 2011). In order to overcome this limitation, Blair (2015) devised an improved version of the pressure function given by:

$$P(t) = \left[1 - \left\{ 1 + \frac{1000}{\sqrt{n}} \left(\frac{\beta t}{n} \right)^m \right\}^{-1} \right] P_0(t) \quad (2)$$

For a fixed value of n (typically 6), increasing the value of m decreases the risetime of the pressure function, but only until the peak value is reached. However, the particle velocity induced in the surrounding material is determined by the second time-derivatives, $P''_0(t)$ and $P''(t)$, of the pressure functions. These functions can be determined analytically from equations (1) and (2). The lower plots in Figure 1 show normalised shapes of the pressure functions given by $P_0(t)$ for $n=6$, and $P(t)$ for $n=6, m=32$, for which the peak values of $P_0(t)$ and $P(t)$ occur when $\beta t=n=6$. The upper plots show the normalised shapes of their second derivatives.

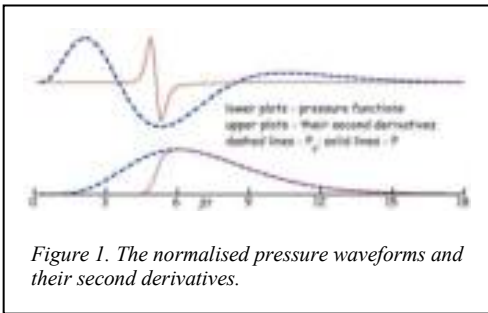


Figure 1. The normalised pressure waveforms and their second derivatives.

The Heelan and Scaled Heelan models given in Blair & Minchinton (1996, 2008) only employed functions of the form $P''_0(t)$ in order to predict the vibration particle velocity. In the present investigation the more realistic functions of the form $P''(t)$ are to be used. It is worthwhile noting that, under its basic assumptions, the general solution derived by Heelan (1953) for the particle displacement due to a pressurised cylindrical element is physically and mathematically well

founded (Meredith 1990). However, the assumptions were somewhat restrictive, thus Blair (2010) derived a full-field solution (FFS) based on the pressure load given in equation (1) and found that the Heelan solution is a valid approximation only at sufficiently large distances and only for certain vibration frequencies produced by the charge. More recently, Gómez *et al.* (2020, 2024) developed an alternative FFS with a detailed pressure load consistent with the shock physics on the detonation shock/rock interface. This approach describes the action of the pressure load in a much more realistic manner than equation (2) within the FFS described by Blair (2010). However, the shock physics approach coupled to any FFS is far too expensive and time-consuming to employ on a blast-by-blast basis within an operating mine, especially if multiple blastholes are considered. Thus, a detailed consideration is only given to a scaled Heelan version with a pressure load described by equation (2); this model is aimed at producing a fast and acceptable estimate of VPPV.

1.2 Production blasts

Blair (2011) gives a detailed review of charge weight scaling and general WSM approaches for the predictions of VPPV from production blasts and concludes that they both have their advantages and disadvantages. The advantages of charge weight scaling are that it is very simple to implement and is based solely on the measured VPPV data as well as the known weight of the maximum instantaneous charge (MIC) and some measure of average distance to the monitoring station. Its main disadvantage is that it gives no insight to other important parameters that govern VPPV such as delay sequence, blasthole spatial pattern, individual charge weights and individual distances. This situation makes it difficult for the blast engineer to alter these vibration-dependent parameters in order to control the VPPV. The main advantage of the WSM approach is that it does account for all these vibration-dependent parameters. However, it has two main disadvantages. Firstly, the seed waveforms (required in all WSM approaches) should always be measured in the rock mass of specific interest, and this is often impractical. Secondly, the rock mass undergoes significant dynamic alteration during a blast, and the interaction of seed waveforms with such varying ground conditions is not well understood. Nevertheless, the WSMs used in the present investigation incorporate an advanced screening/damage algorithm, which attempts to account for such variabilities.

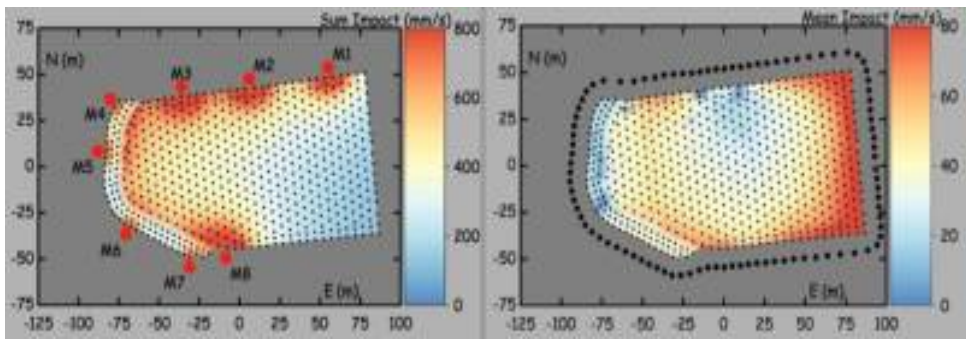


Figure 2. VIM results: a) For (25 kg, 75 kg); b) Based on actual charge weights.

All these models are now discussed and illustrated in order of complexity: Vibration Impact Model, Scaled Heelan, Waveform Superposition Models (both Fast Waveform Superposition and Monte Carlo Waveform Superposition) and, finally, the Full Field Solution.

2 VIBRATION IMPACT MODEL (VIM)

With regard to the vibration produced by an extended pattern of blastholes, the VIM is a special class of vibration prediction that gives information on the behaviour of each blasthole within the pattern. Here, the basic idea, as introduced by Blair (2015), is to select a monitoring location at distance, d , from a particular blasthole with charge weight W , then calculate the value of $A(d/\sqrt{W})^{-B}$ and map this VPPV value back to the location of the blasthole in question; A and B are the vibration site constants. Figure 2a shows the VIM predictions for a blast having 595 blastholes; all the closely-spaced blastholes on the lhs of the pattern were assumed to have a charge weight of 25 kg, and the remainder assumed to have 75 kg each. The results of this (25 kg, 75 kg) combination are shown for 8 monitor stations placed 5 m from the blast boundary. It is quite clear that the impact is associated with the proximity of each of the monitors, with low impact in the 25 kg region and higher impact in the 75 kg region. Furthermore, as expected, for each station the impact dies off with distance according to the charge weight scaling law. The VIM was then run using the actual charge weights, with 95 monitor stations lying on an expanded concave hull region 10 m from the blast boundary. The results are shown in Figure 2b. It is clear that some of the blastholes were

quite ineffective (blue zones) and that the upper-central region was under-charged (possibly due to poor ground conditions). In this regard, it should be noted that the VIM also has value for checking aspects of the blast design; the timing and charge weight designs are shown later in Figures 7a and 7b.

3 SPECIALLY MODIFIED ADVANCED SCALED HEELAN MODEL (SMASHM)

The general Heelan model only predicts relative values of peak vibration whereas the scaled Heelan model reported in Blair & Minchinton (2006) was devised to predict the particle velocity in mm/s. This involved normalising the second derivative of the pressure function of the form $P_0(t)$ given by equation (1) and then altering the standard Heelan solution using the rock damage site constants $K=700$, $A_1=0.5$ and $B_1=1.5$ suggested by Holmberg & Persson (1979). This is the basis of the scaled Heelan model. Considering blast vibration rather than rock damage, it seems more appropriate to use the standard vibration scaling laws, which only involve 2 constants, A , B . In this regard the Australian Standard suggests $A=1100$ and $B=1.6$. However, most vibration-sensitive mining sites will determine their own local values for A and B , and SMASHM allows for such constants as input. This new program also uses the more realistic applied load given by the solid lines in Figure 1. It should also be appreciated that, for all previous Heelan-type models, each of the n time-delayed stacked elements within the charge column is assumed to produce a vibration proportional to its charge weight, w , and thus the vector peak particle velocity (VPPV) is governed by the total charge weight $W = nw$, of a blasthole. However, this is inconsistent with world-wide

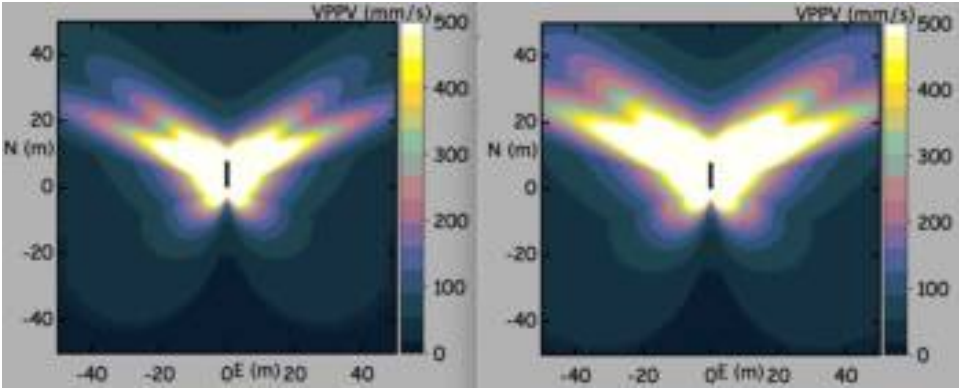


Figure 3. a) Non-linear superposition; b) Linear superposition.

measurements of VPPV for mine blasts, which clearly show a non-linear dependence given by $W^{B/2}$. In this regard, the time-delayed stacked elements within SMASHM have an ever-diminishing effect as they initiate away from the primer in such a way that the combined VPPV influence all elements is given by $(nw)^{B/2}$ rather than $n(w)^{B/2}$. Blair (2008) gave a detailed discussion of this approach and illustrated the basis for the non-linear algorithm at one specific primer location within a blasthole. SMASHM now uses this algorithm for a primer located at any point within the charge column; all the results in this regard assume $A=1100$, $B=1.6$ as per the Australian Standard.

3.1 Vertical blastholes

The first cases to consider are those for vertical

blastholes with diameter: 165 mm, length: 8 m, explosive density: 1176 kg/m³, VoD: 6000 m/s, Vp: 4000 m/s and Vs: 2309 m/s. Figure 3a shows the SMASHM predictions for a vertical blasthole with non-linear superposition, and Figure 3b shows the results for linear superposition. The VoD is greater than the P-wave velocity and thus produces a P-Mach wave as well as an S-Mach wave; both these dominant features are clearly seen in the VPPV radiation patterns. It is clear that the unrealistic assumption of linear superposition significantly overestimates the vibration. It can also be seen that the VPPV for regions below the primer are almost identical in both cases; this is due to the fact that the first charge element in each blasthole emits an identical radiation. It is only the subsequent charge elements in the non-linear case that emit a progressively weaker radiation as the column initiates away from the primer. However,

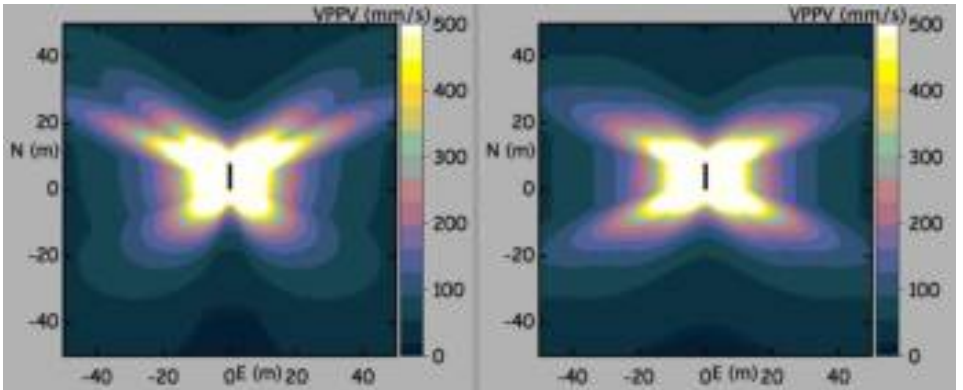


Figure 4. a) Primer 1.5 m from base; b) Centre-priming.

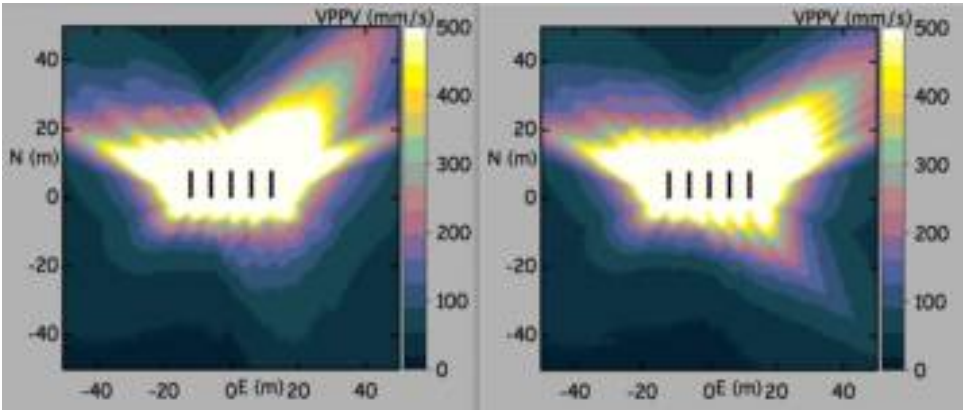


Figure 5. Five blastholes, fired from left to right; a) Delay interval 1.0 ms; b) Delay interval 2.0

the program also has an option in its header file for either linear or non-linear superposition.

All the following SMASHM predictions are shown only for non-linear superposition of charge elements. Figure 4a shows the results for a primer placed 1.5 m from the base of the charge and Figure 4b shows the results for centre-priming. Comparing Figures 3a and 4a shows that there is a sufficient charge length above the primer to form clear P- and S-Mach waves, however in Figure 4a there is an insufficient charge length below the primer to form these waves. Furthermore, for the case of centre-priming (Figure 4b) there is barely enough charge length either side of the primer to form clear Mach waves.

Figure 5 shows the results for 5 blastholes, base-primed with a firing direction from left to right; the delay time between holes in Figure 5a is

1.0 ms, and that for Figure 5b is 2.0 ms. These delay intervals are sufficiently close in time to produce significant wave interaction between blastholes. However, for the VoD of 6000 m/s, the total time duration of each charge column of 8 m is 1.33 ms, and the total time for all charges to initiate is 4 ms for Figure 5a, and 8 ms for Figure 5b. The complicated patterns seen in Figure 5 are due to such delays as well as the wave travel time delays to all points in the maps.

3.2 Angled blastholes

The non-linear version of SMASHM for angled blastholes is taken from a particular layout for a sublevel cave blast. Figure 6a shows the results for this blast having 89 mm diameter blastholes of variable length, explosive VoD 4277 m/s,

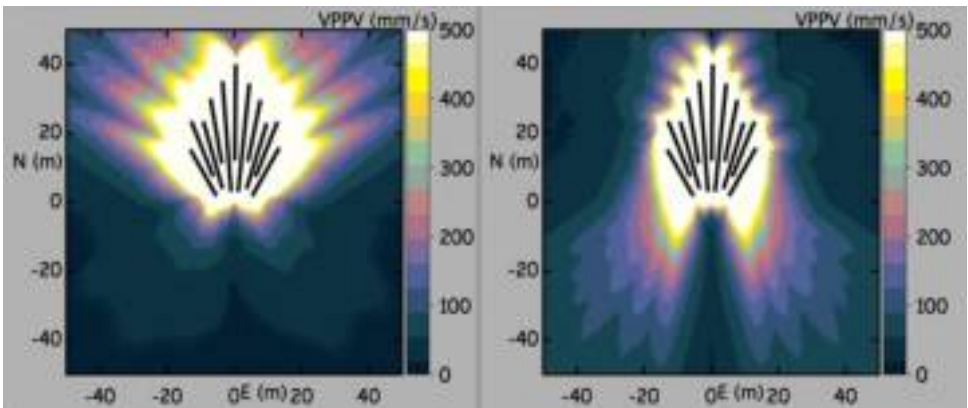


Figure 6. Sub-level cave blast; a) Base-priming; b) Top-priming.

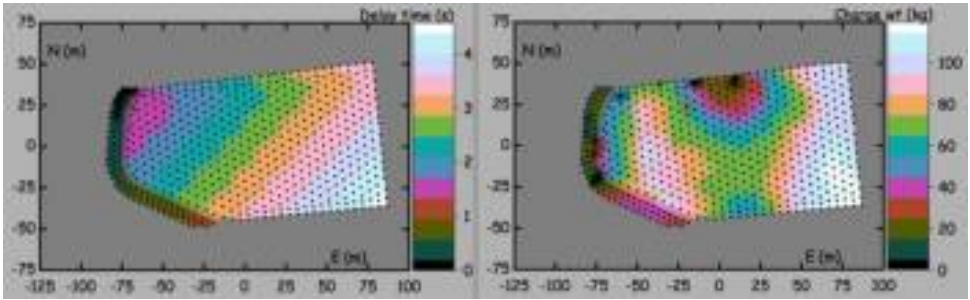


Figure 7. a) Planned delay times; b) The charge weight distribution.

explosive density: 830 kg/m^3 , V_p : 6000 m/s and V_s : 3460 m/s. In this case the VoD is less than the p-wave velocity, V_p , but greater than the S-wave velocity, V_s , thus the dominant radiation features are due to S-Mach waves only. The site constants were chosen as per the Australian Standard. The primers were placed at various distances between 0.7 m and 1.6 m from the base of each blasthole. There was a 25 ms delay interval fanning out each side of the central hole. Figure 6b shows the results when all primers are placed 0.5 m from the top of each blast column. It is quite obvious that such top-priming directs much of the energy downwards. Each of these models of 11 blastholes took approximately 70 s to execute in FORTRAN.

4 FAST WAVEFORM SUPERPOSITION MODEL (FWSM)

Consideration is now given to the case of blasts having a significant number of blastholes, laid out in any particular surface pattern. Previous models

of Monte Carlo Waveform Superposition (MCWSM) have been reported in Blair (2008); many references and discussion of allied work including other approaches is given in Blair (2011). The Monte Carlo aspect involves variabilities such as a natural scatter in delay times, random scatter in seed waveforms and fluctuations in charge weights. Thus, many simulations (usually 100 or more) of a particular blast are required at each point in a gridded region surrounding the blast. However, if one is prepared to forgo such variabilities (the increased use of electronic delays reduces the scatter in delay times) then a significant decrease in computer run times can be achieved. Further decreases in execution time is obtained by having fewer subroutines, with the final result that the FWSM can achieve a speedup of more than 100 compared to the MCWSM.

In any WSM the response of each blasthole is given by the measured 3-component waveform of a single blasthole firing; this is a seed waveform

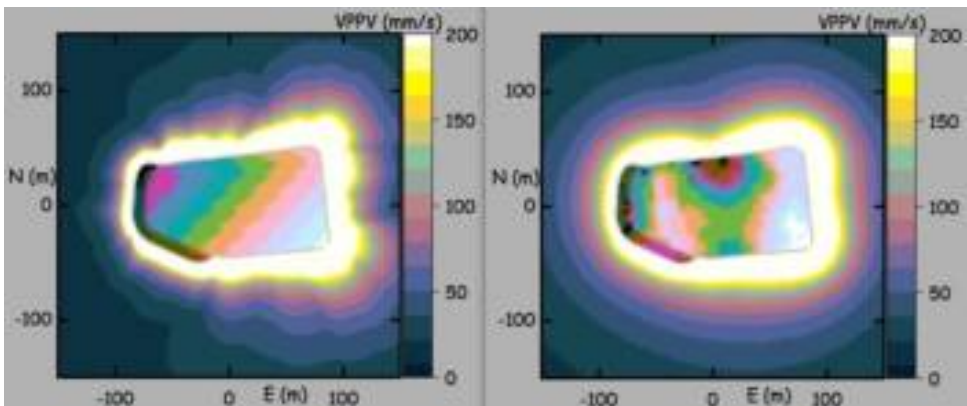


Figure 8a. The FWS predictions. Figure 8b. The MCWSM predictions.

and assumed to be representative of the vibration from each delayed blasthole in the extended pattern. The FWSM is now used to predict the outcome of the blast previously used in the VIM; this is a vibration shield blast designed to protect sensitive structures in regions to the left of the pattern. Figure 7a shows the planned delay times across this pattern of 585 blastholes (small circles), and Figure 7b shows the charge weight distribution.

Figure 8a shows the VPPV predictions using the FWS, and Figure 8b shows the results using the MCWSM. The insert over the blast pattern in Figure 8a shows the delay-timing map, and that in Figure 8b shows the charge weight distribution. Using a grid of 101 x 101 points, the FWSM took 1.87 minutes to execute, whereas the MCWSM took 253.25 minutes.

5 THE FULL FIELD SOLUTION (FFS)

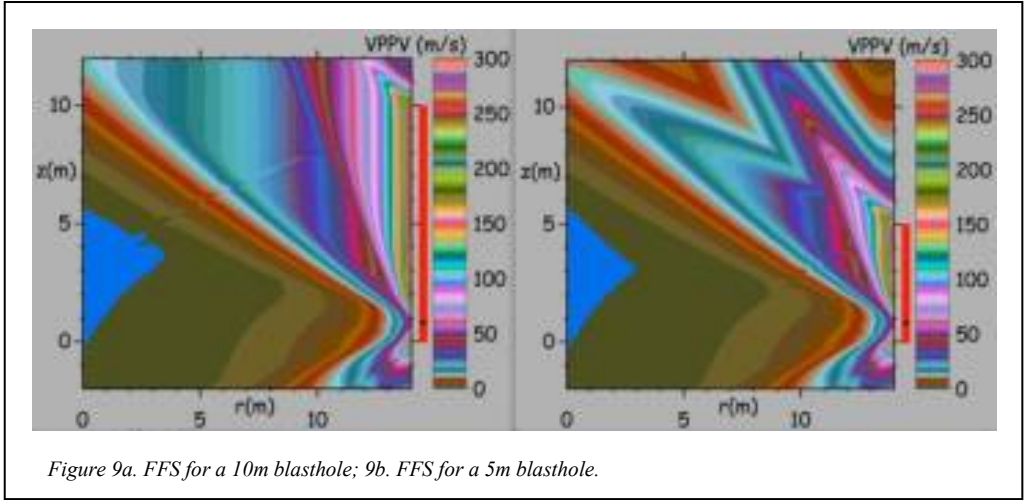
As noted in the Introduction, the Heelan models are an approximation insofar as they are only valid under certain frequency and distance conditions. The FFS overcomes such restrictions but comes with a significant overhead in computing time. In fact, it required over 25 hours to construct each of the separate maps shown in Figures 9, 10 later. The mathematical detail of the model is given in Blair (2010) with recent modifications to account for VoD run-up, a variable viscoelastic Q as a function of radial distance from the blasthole and an applied pressure load as described earlier in Figure 1. These models are applicable to a certain mine site having blastholes of 244 mm diameter and prescribed lengths. The rock mass properties are described by a density of 2820 kg/m³, a P-

wave velocity of 5065 m/s and a Poisson's ratio of 0.21; the explosive had a VoD of 5816 m/s at full run-up and is thus expected to produce both P- and S-Mach waves in the rock mass. The minimum viscoelastic Q -value was set at 2.0 close to the blasthole and maximum Q -value was set at 50.0 near the far boundary.

Figure 9a shows the VPPV map for a blasthole of length 10 m, and Figure 9b shows the results for a blasthole of length 5 m. The small black square in each column indicates the primer location. In each map there is a large range in VPPV values that cannot be effectively conveyed using any standard colour scheme. The colour scheme used here consists of 250 separate colours carefully constructed to be colour-blind friendly with a maximal colour difference between each (see Krzywinski, M <http://mkweb.bcgsc.ca/colorblind>).

These high-resolution maps clearly show the detail within each of the Mach waves as well as the far-field variations and, most importantly, the detail in the very near field of the blasthole. The behaviour in this regime cannot be explained using the Heelan models (Blair, 2007), and is due to coupled P- and S-waves travelling up along the blasthole wall. Due to the limited size of the maps, in Figure 9a, the blasthole is too long to show the full emergence of the P- and S-Mach waves; however, these separate waves are clearly seen in Figure 9b.

In Figure 9a a curved disturbance is clearly seen in the map and this is a real feature due to the interference between the unloading P-wave from the top of the explosive column and the direct SV-wave. Aspects of this interference are also seen in all 4 maps, including regions below the primer. Figure 10a shows the FFS results for a 5 m



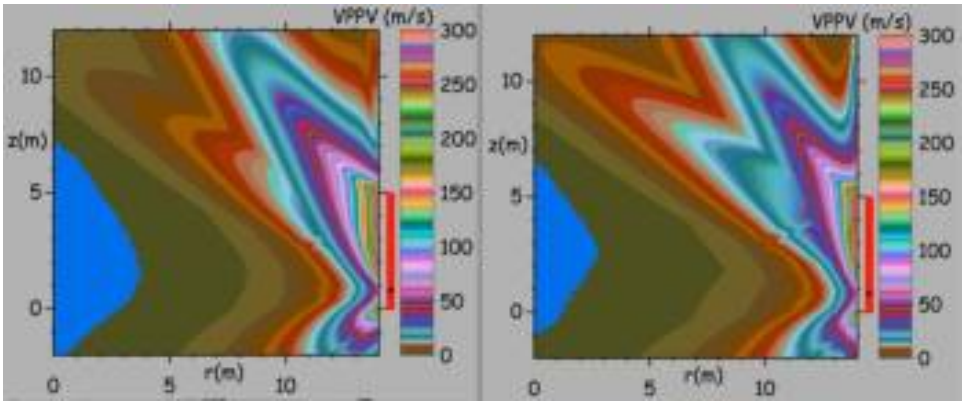


Figure 10a. Low viscoelastic Q ; Figure 10b. Original pressure function P_0 .

blasthole in which the viscoelastic Q was varied between 1.0 and 10.0 (Low Q) and Figure 10b shows the results using the original Q and pressure function, P_0 (see Figure 1).

6 CONCLUSIONS

The work has provided insights and results for various models applicable to blast vibration. In each case a detailed set of prediction results was provided along with a discussion on the relative advantages and limitations of each model. The VIM was shown to have practical application to blast charge weight and timing designs. All the Heelan models are shown to be useful in mapping the VPPV distribution around vertical and inclined blastholes, and the FWSM was shown to provide a fast and realistic estimation of the VPPV surrounding a large blast pattern. For the case of a shield blast the FWS and MCWSM showed very similar results and also the reduction of VPPV in sensitive regions west of the blast (Figure 8). The high fidelity VPPV patterns produced by the FFS showed the influences due to changing variables such as blasthole length, viscoelastic attenuation and the type of applied pressure loading.

REFERENCES

- Blair, D.P. 2008. Non-linear superposition models of blast vibration. *International Journal of Rock Mechanics and Mining Sciences*, 45, 235-247.
- Blair, D.P. 2010. Seismic radiation from an explosive column. *Geophysics*, 75, E55-E65.
- Blair, D.P. 2011. A probabilistic analysis of vibration based on measured data and charge weight scaling. In: *EFEE Sixth World Conference on Explosives and Blasting Techniques, Lisbon, Portugal*, 319-337.
- Blair, D.P. 2015. Wall Control Blasting. *Tenth Int. Symp. Rock Fragmentation by Blasting, Sydney, Australia*, 13-26.
- Blair, D.P. & Minchinton, A. 1996. On the damage zone surrounding a single blasthole, in *Proceedings Fifth International Symposium on Fragmentation by Blasting - Fragblast 5*, (ed: B Mohanty), pp 121-130 (A A Balkema: Rotterdam).
- Blair, D.P. & Minchinton, A. 2006. Near-field blast vibration models. *Eighth Int. Symp. Rock Fragmentation by Blasting, Santiago, Chile*.
- Blair, D.P. 2007. A comparison of Heelan and exact solutions for seismic radiation from a short cylindrical charge. *Geophysics*, 72, no. 2 E33-41.
- Cavanough, G. & Onderra, I. 2011. Development of pressure and temperature gauges to monitor in-situ performance of commercial explosives. *Mining Technology*, 120(2), 74-79.
- Gómez, S. Sanchidrián, J. A. & Segarra, P. 2020. Near-field vibration from blasting and rock damage prediction with a full-field solution. *International Journal of Rock Mechanics and Mining Sciences*, 134, 104357.

- Gómez, S. Sanchidrián, J. A. & Segarra, P. 2024. Full-field solution from an oblique shock to estimate ground motion from blasting. *International Journal of Rock Mechanics and Mining Sciences*, 176, 105688.
- Grady, D.E., Kipp, M.E. & Smith, C.S. 1980. Explosive fracture studies on oil shale, *Society of Petroleum Engineers Journal*, October: 349-356.
- Heelan, P.A. 1953. Radiation from a cylindrical source of finite length, *Geophysics*, 18:685-696.
- Holmberg, R. & Persson, P.A. 1979. Design of tunnel perimeter blasthole patterns to prevent rock damage. *Seventh International Symposium on Rock Fragmentation by Blasting*, Beijing, p595-604.
- Krzywinski, M. <http://mkweb.bcgsc.ca/colorblind>
- Larson, D.B. 1982. Explosive energy coupling in geologic materials. *Int J Rock Mech Min Sci & Geomech Abstr*, 19:157-166.
- Liu, Q. Tran, H. Counter, D. & Andrieus, P. 1998. A case study of blast damage evaluation in open stope mining at Kidd Creek mines, in *Proceedings of 24th Annual Conference on Explosives and Blasting Techniques*, Vol II, pp 323 - 336, (International Society of Explosives Engineers: Cleveland).
- Meredith, J.A. 1990. Numerical and analytical modelling of downhole seismic sources: the near and far field, PhD thesis, *Massachusetts Institute of Technology, Boston*.
- Shastri, S.M. & Malumdar, S. 1982. A dynamic finite element analysis of stress-wave propagation and rock fragmentation in blasting, in *Proceedings Fourth International Conference on Numerical Methods in Geomechanics*, (ed. Z Eisenstein), pp 437-447, (A A Balkema: Rotterdam).
- Trivino, L.F. Mohanty, B. & Munjiza, A. 2009. Seismic radiation patterns from cylindrical explosive charges by analytical and combined finite-discrete element methods. *Proc. Ninth Int. Symp. Rock Fragmentation by Blasting*, Granada, Spain. 415-426.

[RETURN TO CONTENTS PAGE](#)

Comparison of international vibration standards for quarry blasting close-in to a residential structure in Michigan, USA

C. Aimone-Martin

Aimone-Martin Associates, LLC, USA

C. Dowding

Professor Emeritus Northwestern University, USA

D. Rudenko

GeoSonics, Hazelton, PA, USA

C. Loukides

uWave Monitoring Systems, Golden CO, USA

S. Newman

Nomis Seismographs, Irondale, AL, USA

J. Gjødvad

Sigicom AB, Denmark

M. Jern

Nitro Consult AB, Sweden

ABSTRACT: Responses of a residential structure in Michigan, USA to adjacent quarry blasting are compared in this article to regulatory requirements from eight different countries. This report continues cooperative effort among international experts representing the European Federation of Explosives Engineers (EFEE) and International Society of Explosive Engineers (ISEE) to coherently document often conflicting regulations. This nearly decade long project is supported by more than 21 domain experts from Austria, France, Germany, Norway, Sweden, Switzerland, United Kingdom (UK) and United States (US). These measurements and regulatory comparisons compliment those made with another residential structure in Ohio USA and is the sixth of a series of papers resulting from this project that are archived in the EFEE and ISEE conference proceedings. The seventh paper in this series that compares in detail the eight regulations for controlling blasting vibrations is a companion paper in these proceedings.

This Michigan structure was instrumented to record the regulatory required blast induced ground and structural motions to compare with the resulting structural velocity, strain, and crack responses. Geophones were placed on the foundation and in the ground in accordance with each country's regulation to record peak velocities as a function of frequency or distance to blasting. Geophones were attached to the structure corners at strategic locations to calculate global wall tensile strains. In addition, micrometer sensitive displacement gauges were placed across existing cracks to compare vibration induced crack response to that induced by changes in temperature and humidity. This extensive instrumentation provided a quantitative mechanism to compare eight differing regulatory compliances with the actual structural responses of concern. The structural responses of concern are strain in the wall covering that if high enough could cause cosmetic cracking and dynamically induced changes in the width of existing cosmetic cracks for comparison with those produced by climatological effects.

This one-story residential wood-frame structure with a brick-and-mortar exterior and full basement was vibrated by quarry blasting as near as 71 m (232 ft). Structure responses were documented for some 34 blasts conducted to the north and east of the structure from March through October 2023 of which 16 blasts were selected for detailed analyses. The highest in-ground peak particle velocities (PPVs) recorded were 49 mm/s and 37 mm/s at the NE and SE corners, respectively. Measurements from these blasts plotted against each country's limits provided a comparison of stringency among the countries. The percentage of compliance limits among countries ranged from an average of 141% for the US, UK, and Austria limits to the highest of 327 and 634 % for Germany and Switzerland respectively.

Despite these high percentages of non, or out of, compliance for all countries, the global tensile strains induced by these blasts were at most only 37 μ -strains. This strain occurred with the blast that produced a PPV exceeded each country's limit by 112 to 478% except Austria at 91% of compliance. This high percentage of non-compliance associated with a low strain level demonstrates the restrictive nature of the regulations studied. Peak dynamic crack opening and closing by the blast with a PPV of 33 mm/s was only 24 m-m (0.00096 in) while the response to the summer-winter change in heating was some 420 m-m (0.016 in).

1 INTRODUCTION

This project resulted from recognition by seismograph monitoring experts around the world that norms, standards and regulations employed to control vibrations are highly divergent and restrictive for identical rock blasting works. European limits are set to prevent threshold or cosmetic cracking of structure materials. However, no scientifically based observation of cosmetic cracking by Europeans has been found over this now more than seven-year investigation. The notion that limits are adopted to allegedly prevent threshold cracking in structure materials when vibration limits are exceeded cannot be supported when in fact there have been few, if any, documented cracks originating from rock blasting-induced vibrations around the world (Noren-Cosgriff et al. 2002).

Throughout this study, the authors have concluded that many vibration standards are confusing, poorly documented by regulatory authorities and subject to arbitrary changes or site-specific modification without scientific justification. There is little consistency among limits. For instance, vibration limit amplitudes range from peak particle velocity measured in the ground to measurements on the foundation in terms of peak component, vector sum or vertical component. Vibration limits are plotted against zero-crossing frequency at the peak amplitude, converted to Fourier transform (FFT), or distance

from blasting. What limits should be followed often form the basis of unnecessary legal battles, where the divergency in the limits are in question.

These restrictive and divergent vibration limits result in unnecessarily high construction and mining operating costs, requiring a large number of smaller blasts using explosives. As a result, more time is needed to transport and handle explosives potentially increasing workplace hazards and worker complacency while exposing the community near blasting operations to an unnecessary large number of blasts and possible increased disturbance, dust and fumes, and increased use of natural resources. These factors make blasting operations less cost-efficient.

The European Union (EU) has had long-standing goals to harmonise regulations including those involving blast vibrations. The present unnecessary blast vibration restrictions and divergences described herein work in opposition of EU goals. A uniform standard across Europe should be an important goal for the EU.

2 STUDY BACKGROUND

Comparison of vibration standards began with an initial literature study and a survey among EFEE members. The result of the study presented by Gjødvaad & Jern (2019, 2020) clearly confirmed the unnecessary low limits and divergence among the standards, norms and regulations. The second part of the project Gjødvaad *et al.* (2022a, 2024)

involved installation of seven seismographs from two different manufactures, three crack gauges and two microphones on a commercial building at the Aitik mine in northern Sweden comparing the eight different national limits during blasting operations. The third phase began in 2023 and involved instrumentation of two residential structures near quarries in Ohio and Michigan in the United States. Results of the Ohio study were published earlier this year (Aimone-Martin *et al.* 2025) and this article presents the Michigan results. Both of these studies involve calculations of blast-induced global wall strains and measurement of an existing crack response to vibratory and climatological influences to determine if any of the standards can reflect the risks of blast-induced cosmetic cracking. A companion paper in these proceedings, 'EFEE Environmental Committee Vibration Standards Comparisons – Current Update', presents in depth description of details of the eight divergent regulations compared in this paper and need for documentation of origin of and data supporting them.

3 STUDY SITE

The study structure is owned by a quarry near Maybee, Michigan and subjected to 34 blasts between March and October 2023. Of these blasts 16, shown in Figure 1, were chosen as

representative of high PPVs that exceeded regulatory standards. Resulting ground and response motions were employed for detailed vibratory structure response analyses. Two of the 16 were chosen at a large distance in order to include shots at large scaled distances from blast to the structure.

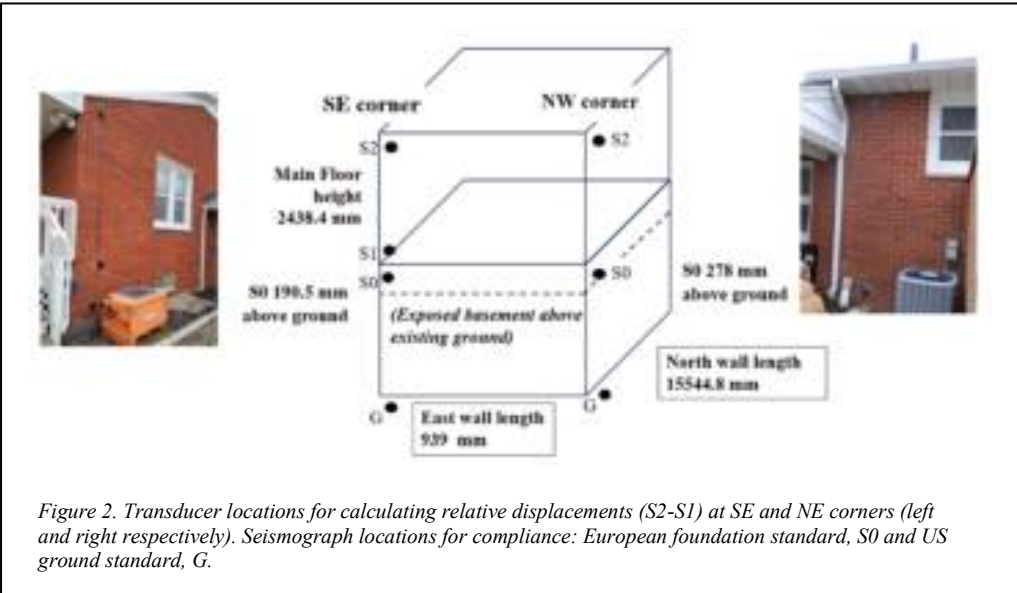
The limestone quarry rock is overlain by some 6 m (20ft) of glacial lacustrine clay and silt on which the structure was founded. Quarry blasting was conducted using 140 mm (5.5 in) diameter holes drilled on variety of patterns; for example, 4 x 4.5 m (13 x 15ft) to 3.7 x 4.3 (12 x 14). Bench heights ranged from 6.7 to 17.4 m (22 to 57 ft). Emulsion explosive based powder factors ranged from 0.50 to 0.62 kg/m³ (0.84 to 1.06 lb/yd³).

Explosive charge weights per delay for the 16 blasts varied 51 to 61 kg (113 to 134 lb) at blast distances from 60 to 372 m (190 to 1220 ft) to the structure resulting in scaled distances between 8.1 to 47.5 m/kg^{1/2} (18 to 105 ft/lb^{1/2}). 14 of the 16 were detonated at scaled distances between 9 and 11 m/kg^{1/2} (10 to 24 ft/lb^{1/2}) Closer shots were decked. Blast timing was designed to last less than 0.5 to 1 second.

4 STRUCTURE INSTRUMENTATION

Instruments were installed to acquire the wide variety of regulatory required motions as well as structural response to calculate strains and





measure crack response. Geophones were secured on structure walls and buried in the ground at corners identified in Figure 2 in compliance with regulations dictated by eight countries outlined in Table 1. Geophones were buried in the ground (G) at NE and SE corners and mounted on the exposed foundation (noted as S0) at each structure corner in compliance with most European standards.

Motions from upper and lower corner geophones (S2 and S1) were employed at the NE and SE corners to calculate blast-induced shear and global wall strains. These strains can then be compared with strain amplitudes that may induce threshold or cosmetic cracking in interior drywall. Seismographs recording motions at S2 and S1

were connected in series to simultaneously record all measurements upon triggering of the master unit in the ground, G. LVDT (Linear Variable Differential Transformer) displacement gauges were mounted over an existing drywall crack shown in Figure 3. Gauge responses were recorded each hour, herein called static response. The same gauge over the same crack was also automatically triggered by in-ground geophones at the NE corner of the garage to measure dynamic crack response when a blast occurred.

5 RESULTS

Peak particle velocities (PPV's) measured at the

Table 1. locations of vibration measurements in terms of velocity plotted against frequency or distance used for regulatory compliance in eight counties.

Country (regulation)	Location of compliance geophone	Compliance velocity measurement	Compliance - Frequency or distance
Austria (ONOM S 9020)	foundation <0.5 m from ground (S0)	time correlated vector-sum	none
France (Circular No. 96-52)	foundation (S0)	peak component	zero-crossing frequency of peak component
Germany (DIN 4150-3)	foundation <0.5 m from ground (S0)	peak component	zero-crossing frequency of peak component
Norway (NS8141)	foundation (S0)	peak component or vertical component	distance

NE and SE corners for the 16 blasts are plotted in Figure 4 with regulated limits depicted as lines for each of eight countries. The 14 blasts exceeding regulatory limits show a wide variation in exceedance. As summarised by the wide variety of regulatory constraints in Table 1, producing these graphs was challenging. Appropriate PPVs varied by location, direction, vector sum, and maximum component, and allowable magnitude varied by frequency (peak or FFT) or distance. Comparison of Austrian, US, UK, German and Swiss standards show that German and Swiss standards are unusually restrictive.



Figure 3. LVDT micro metre displacement gauges placed over an existing vertical drywall crack (above outlet) and null gauge on uncracked drywall (lower right) both below the living room window (molding shown above).

Figure 5 compares time histories produced by the closest blast on 16 October. Crack response is compared on the right with required compliance measurement of vibrations recorded on the foundation, S0, (six European countries) and in the ground, G, (UK and US). Displacements at the upper (S2) and lower (S1) corners are compared on the left with global tensile strains in the South wall, which contains the monitored crack. The tensile strain plot is based on shear strain, calculated as the absolute values of the time-correlated difference between integrated upper and lower recorded velocities divided by the wall height, h , from the floor to the upper wall ceiling of 2438 mm (8 ft) shown in Figure 2, where:

$$\text{Shear strain} = \text{Abs}(S2-S1) / h \quad (1)$$

The global tensile strain is derived from shear strain as calculated along the diagonal length of the wall of interest at each time increment, where:

$$\text{Tensile strain}_{(\text{peak})} = \text{Shear strain} [(\sin \theta) (\cos \theta)] \quad (2)$$

and θ is the angle between the diagonal and the south and east wall lengths of 15544 and 9397 mm (50.1 and 30.8 ft).

6 COMPARISON OF PERCENT COMPLIANCE WITH WALL STRAIN RESPONSE

Correlation of the possibility of cosmetic cracking with compliance can be assessed by comparing blast induced wall strains with the degree of regulatory compliance. For quantitative comparison the degree of regulatory compliance is defined as the ratio of appropriately measured PPV with the frequency or distance constrained allowable PPV defined in Figure 4. This comparison for the US, Germany and Switzerland is presented in Figure 6. Despite PPVs that exceed the regulatory limits at the closest corner for the small, scaled-distance blasts for all eight countries, the highest wall strain was 37 m-strains. This 37 m-strains is below the strain magnitude of 200 m-strains that is generally believed to induce cosmetic cracking in weak wall covering materials in residential structures.

Comparison of compliances for the closest blast shows that percentages of compliance can vary by factors of 6 for the same blast. The 16 October blast occurred at scaled distances of 8.9 and 10 m/kg^{1/2} from the NE and SE corners. It produced PPVs of 40 and 33 mm/s with F_{peaks} of 18 and 17 Hz in the ground and on the foundation PPV_{vertical} of 26 and 20 mm/s with F_{peaks} of 37 and 17 at the NE and SE corners respectively. The percentages of out of compliance at the NE corner nearest the blast ranged from 134 for Austria, to the 180s for UK and US to a high of 634 for Switzerland.

7 BLAST-INDUCED AND CLIMATE-INDUCED CRACK RESPONSE

Blast induced changes in crack opening and closing produced by PPVs several 100's of percentages above those permitted by regulations are negligible compared to those produced by seasonal (summer to winter) changes in temperature and humidity. Figure 7's year-long response of the living room crack in Figure 3 visually demonstrates this insignificance. The amplitude of the dynamic blast induced response of the crack to the blast on 16 Oct in the boxed center inset can be compared to amplitude of the crack's 365 day long response (between the red horizontal lines) produced by the yearly changes in indoor, HVAC moderated, temperature and humidity. The Oct 16 blast was the closest to the

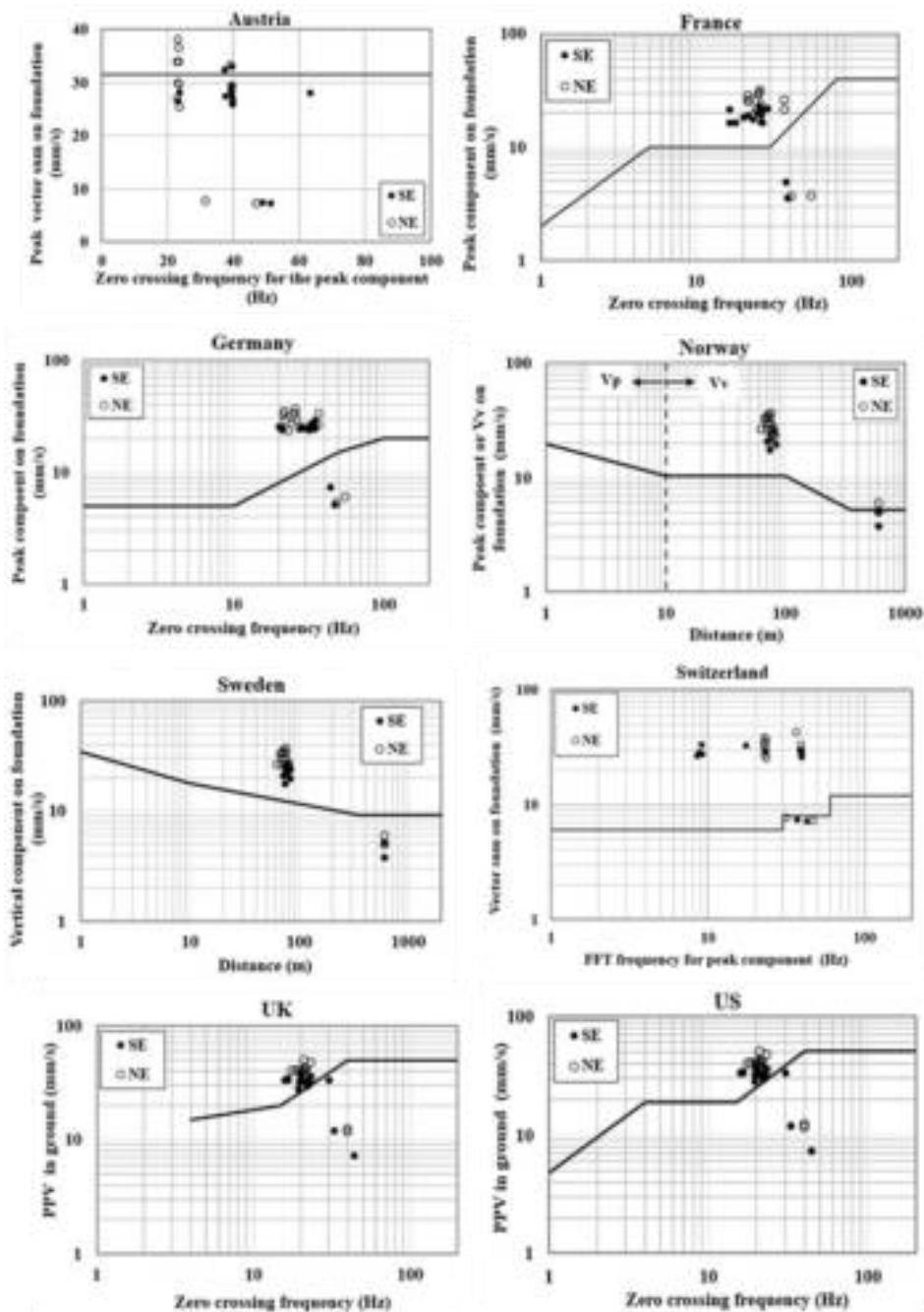


Figure 4. Vibration measurements for the 16 selected blasts plotted with regulated compliance limits for eight countries. Velocity compliance measures (e.g. max axis PPV vs vector sum PPV), levels (e.g. 47 mm/s vs 38 mm/s at NE), and moderation factors (e.g. frequency vs distance) vary widely. Blasts were chosen to be the most energetic except for two to show the effect of large distances and low PPVs.

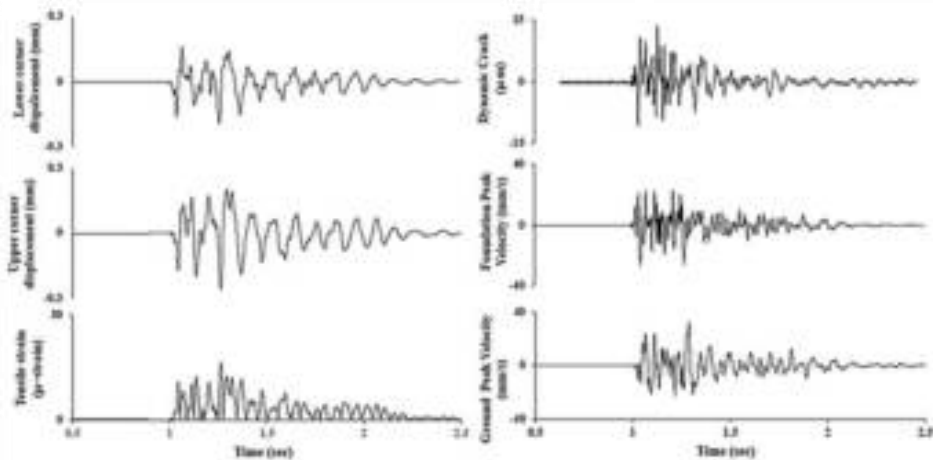


Figure 5. For one of the closest blasts (16 Oct) upper and lower corner driving displacement time histories are compared with the resulting south wall tensile stains on the left. Upper displacements (S2) are slightly greater than lower (S1) showing a slight amplification. The living room crack response is compared on the right with the peak compliance velocities in the ground (G) and on the foundation (S0) at the SE corner.

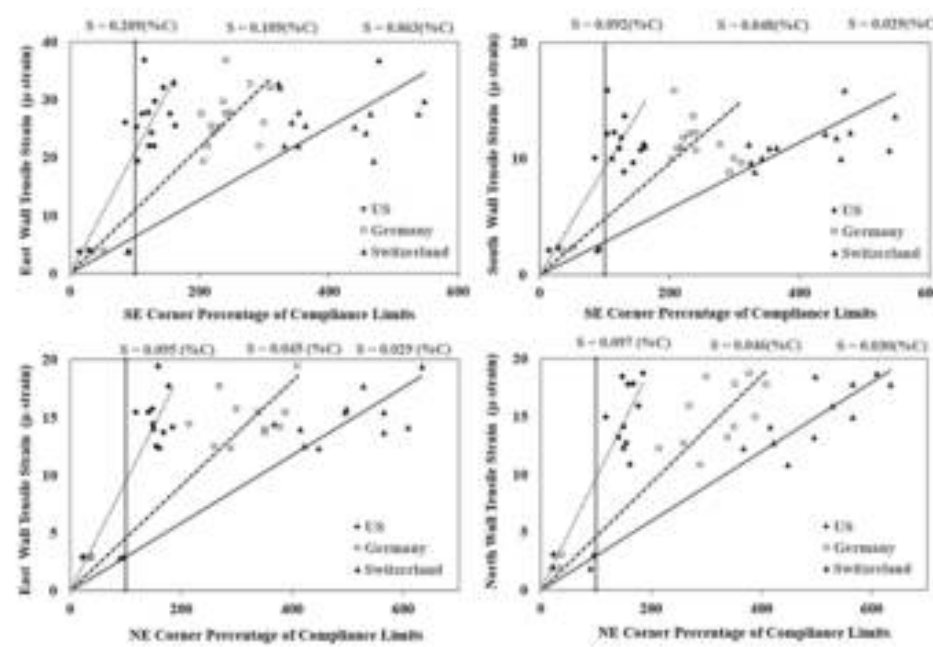


Figure 6. Comparison of South, North and shared East wall global tensile strains as a function of percentage of regulatory compliance at SE (upper) and NE (lower). Shots were chosen to exceed the limits, except for the two at a large distance. Tensile strains calculated from motions at the SE and NE corners for the same east wall differed, but were still small, despite exceedance of compliance limits.

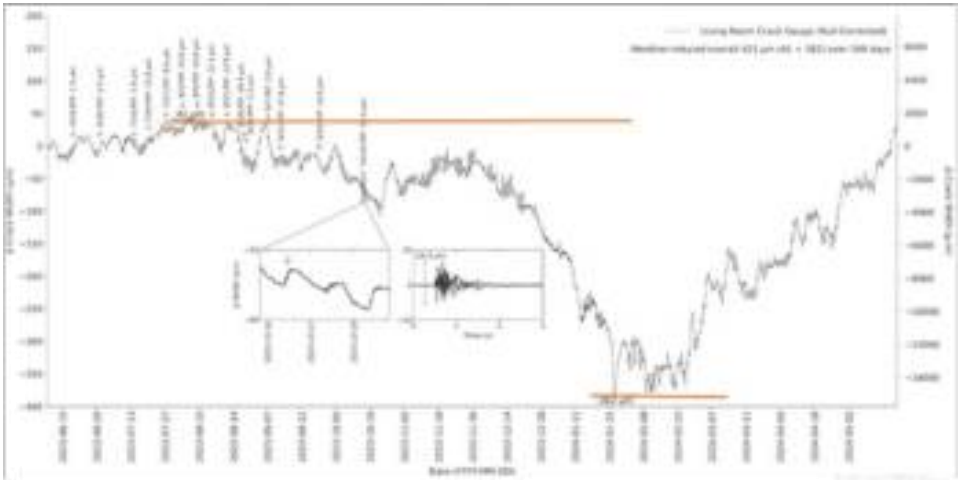


Figure 7. Comparison crack response changes induced by summer-winter heating seasonal of 420 μm , with response of 24 μm to the blast on Oct 16 produced a PPV of 33 mm/s (1.3 ips).

house and produced a PPV at the SE corner of 33 mm/s (1.3 ips) nearest the crack. This motion, which exceeds the US limit by 155 %, produced a 24 μm response from the blast. In comparison the change from summer to winter produced 420 μm and the passage of a weather front produced 75 μm . Thus the 155 % out-of-compliance blast by US standards produced only 6 % of the crack response that occurs every year and only 33% of that which occurs every week or so with the passage of a weather front.

8 SUMMARY AND CONCLUSIONS

A comprehensive study of blast-induced responses of a residential structure was conducted to compare the percentage of compliance with regulated vibration limits used in eight different countries throughout the world. The study focused on 16 selected blasts, 14 of which occurred at small, scaled-distances between 8.9 and 10 $\text{m/kg}^{1/2}$ at the NE corner. Major conclusions of this study include the following:

- Comparisons of velocities measured in accordance with regulated limits showed that at a small scaled distance of 8.9 $\text{m/kg}^{1/2}$ appropriately measured peak particle velocities were out of compliance by a wide range of 134% for Austria, to the 180s for UK and US to a high of 634% for Switzerland for the same blast.

- Despite high percentages of non, or out of, compliance for the suite of blasts, the calculated global tensile strains induced by any of these blasts were at most only 37 $\mu\text{-strains}$. These low calculated strains from these blasts with high out of compliance PPVs demonstrate the restrictive nature of the regulations studied.
- Unusually long surveillance of the living room wall crack showed that the crack response to the nearest blast, which greatly exceeded PPV compliance, was only 6% of the crack response produced by changing from summer cooling to winter heating of the home's interior.

As a group the regulation limits placed on vibrations compared in this study are complicated to compare and vary widely both in level and method of determination. Those with the lowest allowable peak particle velocities cannot be justified based on measured low dynamic structure response measurements of global wall strains and the dynamic blasting-induced response of a crack.

9 ACKNOWLEDGEMENTS

The authors wish to express appreciation to the team of specialists listed in Table 2 who donated their time and resources to guide the direction of this study, work diligently on-site to deploy and

Table 2. Participants in the study.

Regulatory		Seismograph Contributor	
Representer	Country	Representative	Seismograph
Mark Ganster	Austria	Johan Gjødvdad	Sigicom
Theirry Bernard	France	Cathy Aimone-Martin & Brent Meins	White Seismology
Johannes Kutschera	Germany	Douglas Rudenko & Siavash Mahvelati	Vibratech
Robert Farnfield	UK	Cory Loukides	Instantel
Karin Norén-Cosgriff	Norway	Keith Best & Scott Newman	Nomis
Mathis Jern	Sweden	Joel Kurtz	Inzwa
Thomas Moser	Switzerland	Observer	Country
Charles Dowding	USA	Nathan Rouse	Ireland
		Vitor Rosenhaim	Brazil
		Travis Davidsavor & Ken Eltschlager	USA

maintain equipment, remotely download time histories, analyse measurements, and work closely with the blasters to record and preserve blast records.

REFERENCES

Aimone-Martin, C., Dowding, C.H., Gjødvdad, J.F. & Jern, M. 2024. Comparison of international vibration standards for quarry blasting close-in to a residential structure. *51st ISEE conference*, Cherokee, NC. USA.

Gjødvdad, J. & Jern, M. 2019. Vibration monitoring standards connected to the use of explosives in Europe. *Proc. of the 10th World Conf. on Explosives and Blasting*, EFEE, Helsinki, Finland.

Gjødvdad, J. & Jern, M. 2020. Vibration monitoring standards connected to the use of explosives in different countries. *Proc. of the 46th Annual Conference on Explosives and Blasting Technique*, Denver, CO, USA.

Gjødvdad, J., Jern, M., Dowding, C. & Bernard, T. 2022a. Instrumentation of a structure for comparison of regulations worldwide - *Vibration Monitoring Standards Connected to the use of Explosives in Different Countries*,

Part II , 49th ISEE conference. Las Vegas, NV, USA.

Gjødvdad, J. & Jern, M. 2022b. Workshop on vibration standards. *11th World Conf. on Explosives and Blasting*, EFEE in Maastricht, Netherland.

Gjødvdad, J.F., Jern, M., Dowding, C.H., & Aimone-Martin, C. 2024. Comparison of compliance measures for blasting near an industrial structure. *Vibration Monitoring Standards Connected to the use of Explosives in Different Countries, Part III. Proc. of the 50th Annual Conference on Explosives & Blasting Technique*, Savannah, GA, USA.

Noren-Cosgriff, K.M., Ramstad, N., Neby, A. & Madshus, C. 2020. Building damage due to vibration from rock blasting. *Soil Dynamics and Earthquake Engineering* 138.

APPENDIX 1

Table 3. Details of the shots evaluated showing motions measured at the NE and SE corners on the ground (US) and on the foundation (Switzerland).

Blast no.	Date	Distance	MIC	Scaled distance	US				Switzerland			
					In-ground PPV SE corner				Foundation time-correlated vector sum Vs SE corner			
					PPV	Fpeak	PPV-limit	%limit	Vs	peak composite ref FFT	Vs-limit	%limit
		m	kg	m/kg ^{1/3}	mm/s	Hz	mm/s	%	mm/s	Hz	mm/s	%
12	20230523	83	60.3	15.72	32.00	20.80	26.4	121	28.31	39.63	8.0	354
14	20230614	82	54.0	11.1	33.00	20.00	25.4	130	26.5	40	8.0	331
15	20230619	605	53.5	62.73	7.24	44.60	50.8	14	7.11	43.13	8.0	88
16	20230626	80	58.5	10.45	36.78	20.00	25.4	145	25.98	39.75	8.0	325
18	20230719	81	55.3	31.34	34.11	16.70	21.2	181	25.78	39.88	8.0	322
19	20230714	605	112.5	16.92	11.80	33.00	41.9	28	7.29	37.25	8.0	91
20	20230720	82	70.3	21.04	32.15	22.00	27.9	115	28.68	23.38	6.0	478
21	20230727	77	74.8	8.94	33.11	30.50	36.7	85	27.38	38.63	8.0	342
22	20230803	76	63.5	9.58	33.15	16.00	20.3	163	29.14	39.50	8.0	364
23	20230809	78	69.4	9.34	36.81	23.00	29.2	123	28.25	39.50	8.0	353
24	20230815	75	68.0	9.08	27.80	19.60	24.9	112	27.89	9.00	6.0	495
25	20230821	80	68.5	9.64	34.04	20.40	25.9	131	32.88	9.13	6.0	548
26	20230828	84	61.7	10.68	28.18	22.20	28.2	103	26.39	8.83	6.0	440
29	20230912	76	58.1	9.90	30.56	23.00	29.2	105	28.15	8.88	6.0	490
31	20230920	77	54.9	10.44	31.70	19.80	25.1	126	27.41	9.26	6.0	457
35	20231018	72	51.3	10.00	32.90	16.70	21.2	155	32.31	17.63	6.0	539

Blast no.	Date	Distance	MIC	Scaled distance	US				Switzerland			
					In-ground PPV NE corner				Foundation time-correlated vector sum Vs NE corner			
					PPV	Fpeak	PPV-limit	%limit	Vs	peak composite ref FFT	Vs-limit	%limit
		m	kg	m/kg ^{1/3}	mm/s	Hz	mm/s	%	mm/s	Hz	mm/s	%
12	20230523	80	60.3	15.32	41.3	20.1	25.5	162	26.8880	23.10	6.0	448
14	20230614	77	54.0	10.49	41.91	21.3	27.1	135	25.3300	23.80	6.0	422
15	20230619	611	53.5	83.46	11.99	40.2	50.8	23	7.183	47	8.0	90
16	20230626	75	58.5	9.84	41.7	21.8	27.7	151	33.222	39.2	8.0	415
18	20230719	76	55.3	31.34	49.74	21.1	28.8	186	36.544	73.4	6.0	609
19	20230714	611	112.5	16.92	12.27	40.2	50.8	24	7.686	31.5	8.0	96
20	20230720	76	70.3	21.04	36.79	21.6	27.4	141	29.682	23.5	6.0	495
21	20230727	71	74.8	8.22	39.43	20.7	26.3	130	29.404	39.6	8.0	368
22	20230803	70	63.5	8.76	40.17	18.8	23.9	168	33.907	23.4	6.0	565
23	20230809	71	69.4	8.54	38.32	20.3	25.8	149	29.847	21.2	6.0	497
24	20230815	68	68.0	8.22	35.06	23.1	29.6	118	33.915	23	8.0	565
25	20230821	72	68.5	8.76	47.05	23.3	29.6	159	38.519	23.2	6.0	634
35	20231018	64	51.3	8.9	40.03	17.7	22.5	178	42.263	37	6.0	528

RETURN TO CONTENTS PAGE

EFEE Environmental committee vibration standards comparisons - current update

C. Aimone-Martin

Aimone-Martin Associates LLC, U.S.

J. F. Gjødvad

Sigicom AB, Denmark

M. Jern

Nitro Consult AB

T. Moser

SSE, Switzerland

ABSTRACT: The paper provides updates to a joint EFEE-ISEE study involving blast vibration standards and limits intended to prevent cosmetic cracking of weak wall covering materials. The project aims to understand vibration standards, bases, and applications from Austria, Britain, France, Germany, Norway, Sweden, Switzerland, and the U.S. that are often misunderstood due to poor detailing, a lack of historical development, or language differences. Some U.S. blasting projects must adhere to German or Swiss standards without understanding geophone placement, coupling methods, or vibration calculations. Several international standards use complex formulas considering ground conditions, structure materials, foundation types, and type and frequency of blasting, rendering limits difficult to apply. Compliance limits intended to prevent cosmetic cracking of wall materials are often restrictive, leading to time-consuming and costly rock extraction. This presentation encourages audience participation to verify standards, understand if practices deviate from standards, and explore where standards might be modified to ensure blasting is performed safely and economically.

1 BACKGROUND

Ongoing work of the EFEE Environmental committee since 2018 has focused on understanding similarities and differences among norms, guidelines, or standards imposed on blasting vibration velocity limits in Europe and abroad. Findings show that standards and the implementation of seismograph measurements are

highly divergent and restrictive for identical rock blasting works. Limits are allegedly set to prevent cosmetic cracking of wall covering materials while no scientific evidence has been produced to support this concept or explain the diversity in standards (Gjødvad & Jern 2019, 2020).

Field measurements were initiated in May 2021 with the installation of seven seismographs from two manufacturers, three crack gauges, and two

microphones on a commercial building at the Aitik mine in northern Sweden, comparing different country limits during blasting operations and summarised in Gjøvdad *et al.* (2022a, 2024). This study was the first attempt to instrument a structure in compliance with various standards and geophone mounting locations. The far distance from blasting to the structure did not result in significant vibration amplitudes but provided an opportunity to introduce the concept of global wall strains and the response of an existing wall crack to vibrations compared to the effects of weather changes.

A third phase of study began in 2023 with instrumentation of residential structures near quarries in Ohio and Michigan in the U.S. and continued the focus on cosmetic cracking potential based on global strain rather than on ground vibrations alone (Aimone-Martin *et al.* 2025a & 2025b). Although blast-induced ground vibrations exceeded the U.S. limits for several blasts, they far exceeded international standards but did not produce large enough wall strains that would produce cosmetic cracks.

The highest computed wall strain at the Ohio house for 68 blasts was 12 μ -strain in tension and 72.1 μ -strain in shear while most blasts exceeded international standards up to 6-fold without the formation of wall cracking. Blasting near the Michigan house produced wall strains of 37 μ -strain in tension and 151 μ -strain in shear with no observed wall cracking. The highest exceedance occurred for the Swiss standard at a vector sum of 35.1 mm/s that far exceeded the limit of 6 mm/s.

The notion that limits are adopted to allegedly prevent threshold or cosmetic cracking in wall materials in structure wall materials when vibration limits are exceeded cannot be supported when in fact there have been few, if any, documented cracks originating from rock blasting-induced vibrations well above current standards. Two studies in which structures were subjected to increasingly high levels of blasting vibrations to intentionally cause material wall cracking validate that existing standards are highly conservative and cannot lead to wall cracking.

In a study reported by Noren-Cosgriff *et al.* (2002) no visible cracking was observed in test structures, made from unreinforced concrete and plaster coated block with mortar joints, subjected to peak particle velocities (PPV) in the ground in excess of 260 mm/s, resulting in a high wall strain of 1750 μ -strain. Aimone-Martin *et al.* (2022) documented a new blast-induced vertical crack in stucco coating over a one-story cinder block wall

residential structure at 280 mm/s horizontal ground motion, well above the U.S. limit of 51 mm/s, resulting in 700 μ -strain wall shear strain. This strain amplitude was consistent with observed mortar wall cracking during lab studies reported by the U.S. Bureau of Mines (Stagg *et al.* 1984).

2 CURRENT STANDARDS FOR RESIDENTIAL STRUCTURES

A summary of compliance limits and monitoring details for counties included in this study is provided in Table 1. Plots of vibration standard limits as currently interpreted are given in Figure 1. Allowable limits are shown on the Y-axes with the X-axes representing either the distance between the blast and monitoring location, zero-crossing peak frequency, or FFT (Fast Fourier Transform) frequency for the highest component velocity. For this discussion, these standards apply to inhabited residential structures of normal or average sensitivity residing near mining or quarry blasting as was employed at the Ohio and Michigan house studies. In some cases, standards also apply to construction blasting. Standards do not apply to historic or commercial structures, nor do they address human response or annoyance.

A discussion of each standard is presented herein. The information is intended to be general in nature and not necessarily comprehensive, complete, accurate, or up to date. Many of the details have been translated from non-English languages possibly affecting the meaning and not completely recognising cultural undertones. As indicated, some details for compliance determination are unclear. It is the authors' goal that clarity can be accomplished in the near future.

2.1 United States (U.S.) and United Kingdom (U.K.) Standards

The U.S. and U.K. standards are similar and unique among the other nations in that seismograph geophones are coupled to the ground rather than mounted on structures. The U.S. limits were recommended as safe blasting criteria developed from structure response studies for blasting at surface coal mines and quarries (Siskind *et al.* 1980). The U.S. Bureau of Mines (USBM) chart is currently used for all types of blasting and routinely adopted as construction project and mining specifications and as standards within the National Fire Protection Association (NFPA) 495 Code in Chapter 11 for fire marshals that permit construction blasting. They have further been adopted as regulation by the Office of

Table 1. Interpretation of various standards for comparisons for eight counties.

Country	Reference for standards and guidance	Seismic component	Limit frequency based?	Seismic frequency range (Hz)	Location of geophones in actual practice	Measurement methods	Compliance limits for ground motions (observed, residential structures and quarry blasting)
Austria	ONORM S1020 (2011)	Peak motion	yes	0-200	In foundation (basement) or depth more 0.2 m above the ground	Boring	See Eurocode 8: $V_{Sd} = V_{Ed} \cdot V_{E1} \cdot V_{E2}$ V_{E1} = location correction of 0.1 m/s² $V_{E2} = 2.5$ for average sensitivity building class (conservatively) $V_{E2} = 1.0$ for building with ductile performance, 2.0 for performance 0.75 for larger buildings
France	Eurocode 8: EN 12010 relating to seismic requirements, Appendix B: Vibration Measurements Method	Peak component	yes	0-100	Maximum wall fully occupied with the structure	Boring, measuring on structure	(1) none for ground motion amplitudes at all frequencies
Germany	DIN 41329-1 (2004), 9.10.10.10 relating to seismic requirements, Appendix B: Vibration Measurements Method	Peak component	yes	0-200	In foundation (basement) or more than 0.2 m above the ground	Boring	See Eurocode 8: $V_{Sd} = V_{Ed} \cdot V_{E1} \cdot V_{E2}$ V_{E1} = location correction of 0.1 m/s² $V_{E2} = 2.5$ for average sensitivity building class (conservatively) $V_{E2} = 1.0$ for building with ductile performance, 2.0 for performance 0.75 for larger buildings
Hungary	MSZ 01-1:2012a Standard for seismic action for construction work, design and planning and effects	Peak component at 0 + 10 m vertical component at 0.01 Hz, upper 0 m above 0.01 Hz (vertical) 0.01 Hz	yes	1.0-400	No fixed-bearing foundation (fixed to ground)	Boring	See Eurocode 8: (1) none for ground motion amplitudes at all frequencies $V_{Sd} = V_{Ed} \cdot V_{E1} \cdot V_{E2}$ V_{E1} = location correction of 0.1 m/s² $V_{E2} = 2.5$ for average sensitivity building class (conservatively) $V_{E2} = 1.0$ for building with ductile performance, 2.0 for performance 0.75 for larger buildings
Sweden	SIS 460 4000 (2011)	Vertical component	yes	0-200	No fixed-bearing foundation (fixed to ground)	Boring	See Eurocode 8: (1) none for ground motion amplitudes at all frequencies $V_{Sd} = V_{Ed} \cdot V_{E1} \cdot V_{E2}$ V_{E1} = location correction of 0.1 m/s² $V_{E2} = 2.5$ for average sensitivity building class (conservatively) $V_{E2} = 1.0$ for building with ductile performance, 2.0 for performance 0.75 for larger buildings
Switzerland	SIS 0320-01.2a (2004)	Three component at similar depth	yes	0-100	Attached to foundation (wall on the outer of building placed on the floor when structural is 0.01 Hz) 0.01 Hz, otherwise attached to concrete structure when structural is 0.01 Hz) 0.01 Hz, otherwise attached to concrete structure (ground or floor)	Boring	See Eurocode 8: (1) none for ground motion amplitudes at all frequencies $V_{Sd} = V_{Ed} \cdot V_{E1} \cdot V_{E2}$ V_{E1} = location correction of 0.1 m/s² $V_{E2} = 2.5$ for average sensitivity building class (conservatively) $V_{E2} = 1.0$ for building with ductile performance, 2.0 for performance 0.75 for larger buildings
United Kingdom	BS 5345 Part 2: 1996, Part 2: quality of design based on ground borne vibration	Peak component	yes	0-200	At wall or wall surface measured at distance from the base or joint line	Boring, tapping, sandbagging, boring	See Eurocode 8: (1) none for ground motion amplitudes at all frequencies $V_{Sd} = V_{Ed} \cdot V_{E1} \cdot V_{E2}$ V_{E1} = location correction of 0.1 m/s² $V_{E2} = 2.5$ for average sensitivity building class (conservatively) $V_{E2} = 1.0$ for building with ductile performance, 2.0 for performance 0.75 for larger buildings
United States	USAF 4107-1-1 (1970) relating to Surface Mining (OSM) regulations Sections 0.01, 0.02, 0.03 and 0.04 (1970) relating to Surface Mining (OSM)	Peak component	yes	0-200	Attached to foundation (wall on the outer of building placed on the floor when structural is 0.01 Hz) 0.01 Hz, otherwise attached to concrete structure when structural is 0.01 Hz) 0.01 Hz, otherwise attached to concrete structure (ground or floor)	Boring, tapping, sandbagging, boring	See Eurocode 8: (1) none for ground motion amplitudes at all frequencies $V_{Sd} = V_{Ed} \cdot V_{E1} \cdot V_{E2}$ V_{E1} = location correction of 0.1 m/s² $V_{E2} = 2.5$ for average sensitivity building class (conservatively) $V_{E2} = 1.0$ for building with ductile performance, 2.0 for performance 0.75 for larger buildings

Surface Mining (OSM) for coal mine blasting. The U.S. limits are based on observation of cosmetic cracking in weak wall materials where structure response studies correlate horizontal ground motions with global wall shear and tensile strains.

This is very important as measured velocity was translated into displacements based on dominant frequencies, recognising that cosmetic

cracking is not related solely to peak particle velocity in the ground but rather to strains from horizontal displacements of the responding structure and its walls.

The concept of wall strains as a compliance criteria is recognised in the U.S. in New York City as adopted into Chapter 56 of the 2022 Fire Code based on over five years of structure response

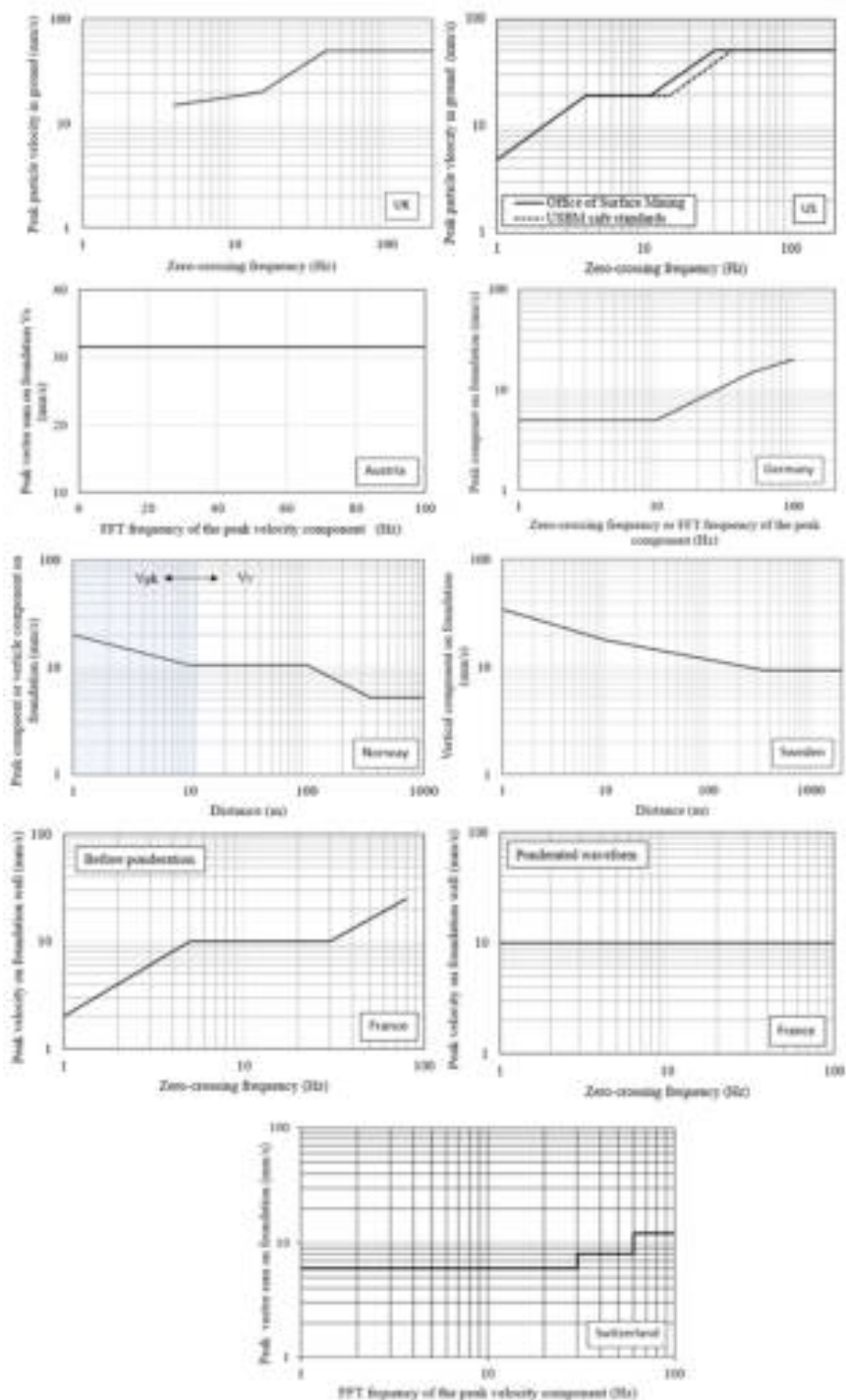


Figure 1. Graphical interpretations of various standards.

studies to support this criteria (Meyers & Aimone-Martin 2019).

The U.K. limits based on BS 7385 provide guidance on safe blasting limits that closely follow the USBM safe criteria above which cosmetic cracking *could* occur. Limits to prevent minor and major damage are also discussed without describing these categories of structure damage. Guidelines further include methods of seismograph deployment, recording, and analysis of vibrations. Included is a discussion of building strains related to ground strains based on P- and S-wave velocities and relative displacements leading to structure cracking support for frequency-based damage criteria.

2.2 Austria

The Austrian blasting standards ÖNORM S 9020 vibration protection for above-ground and underground systems are based on peak vector sum as a function of FFT frequency. Terms include structure sensitivity class, how often blasting occurs (referred to as frequency class), and duration of events. Geophones are bolted on the foundation and the allowable vibration is computed as:

$$V_{RW} = V_B * E_F * A_F \tag{1}$$

Where:
 V_B = baseline vibration of 9 mm/s
 E_F = building sensitivity factor 3.5 for average sensitivity (residential) building class

A_F = reduction factor for frequencies up to 10 Hz and beyond 40 Hz = 1

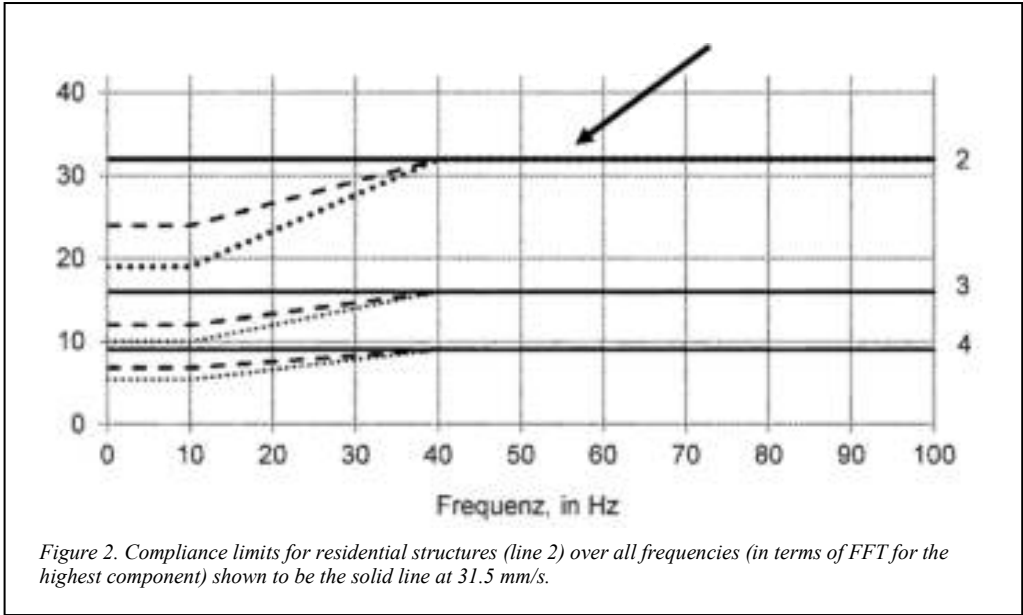
For quarry and construction blasting, the frequency class of ‘seldom’ likely applies and defined as up to 1,000 events per year. The reduction factor, A_F , for impulsive vibration occurring seldomly equals unity for frequencies up to 10 Hz and above 40 while guidance for the interval 10 to 40 Hz is similarly interpreted resulting in $A_F = 1$ over all frequencies as shown in Figure 2 where ‘frequency’ is the FFT frequency.

The key/relevant frequency must be determined for the strongest vibration component. It should be determined using an FFT analysis with a line width/resolution of no more than 0.5 Hz or 2 sec window.

2.3 Germany

DIN 4150-3 (1999) prescribes structure measurements in two horizontal directions on the top structure floor or can be recorded at the foundation. Since frequency was not specifically defined within the main body of the DIN standard, a zero-crossing frequency for the peak is often assumed.

However, the DIN 4150-3 document (English translation) provides an example recorded vibration velocity time history (Figures D-1 and D-2 in the standard) for a mechanical machine start up that is transformed into the frequency



domain to identify what is referred to as the 'significant' frequencies. The FFT peak is then used to find the allowable limit from the compliance curve for residential dwellings (Line 2) given in Figure 1 in the standard. The measured peak velocity is compared to the compliance limit at the FFT peak in Figure 1 to show that the measurement falls below the limit and is within safe limits. This implies that the German standard X-axis should be FFT frequency rather than zero-crossing frequency. *It is important that this is clarified within the main body of the guidelines.*

A second indicator of this finding is given on the SVANTEK website under the 'Building vibration section'.

(<https://svantek.com/applications/building-vibrations/>)

It states that the German DIN standard uses FFT to define dominate frequency (DF) and that each pair of PPV and associated DF are used as point coordinates that are compared with the vibration limit curve to evaluate compliance with the standard. The SVANTEK document states the following:

- 'Building structure vibration standards' - The most recognisable standards for structural vibration measurements are ISO 4866, British BS 7385-2, and German DIN 4150-3. All three standards use the Peak Particle Velocity (PPV) method and FFT to define the Dominant Frequency.
- *What are PPV and Dominant Frequency?* - The method for PPV Peak Particle Velocity and Dominant Frequency (DF) is described in German Standard DIN 4150-3. The PPV is the maximum value of the amplitude of the vibration velocity time-domain signal. To detect a dominant frequency of PPV, the method requires performing an FFT analysis. The result of such analysis is the PPV value and its corresponding Dominant Frequency for each axis (X, Y, Z). Each pair of PPV and its DF are used as point coordinates that are compared with the vibration limit curve.

This website claims that the British standard also uses FFT to define dominant frequency.

Both of these examples use mechanical vibrations and not blasting which generates a wide range of peaks in the FFT domain as opposed to a mechanically induced vibration with one predominant frequency. *It is recommended that the 1999 German standards clearly state which frequency is being used for compliance and provide a compliance example for a complex blasting time history.*

2.4 Norway and Sweden

The Norwegian and Swedish guidelines are distance-based and similar in the consideration of the factors contributing to guidance levels. These include overburden geology near the structure and structure details such as building type or use and materials utilised in construction.

The Norwegian standards are distance-based and employ six factors multiplied by a base vibration limit of 20 mm/s to include the influences of site geology, building foundation fill material, structure type and building materials, building type, and vibration source. Factors are found in various tables and noted below for the Ohio study house case.

Vertical compliance limit:

$$V = V_o * F_g * F_b * F_m * F_f * F_d * F_k \quad (2)$$

Where:

V_o - uncorrected vibration (current standards set at 20 mm/s).

F_g - ground condition factor.

F_b - building type factor (1 for residence).

F_m - material and building detail factor (1 for unreinforced).

F_f - foundation factor (0.8 for soil).

F_d - distance factor (varies with a complex function).

F_k - blasting source factor (0.5 for mines and quarries).

The Swedish standard is also based on distance where the base vibration is a function of geology with other multiplying factors applied similar to the Norwegian standard.

Vertical compliance limit:

$$V_v = V_o * F_b * F_m * F_d * F_t \quad (3)$$

Where:

V_o - vertical component of uncorrected vibration velocity depending on geology (18 mm/s).

F_b - building factor (1 for residence).

F_m - construction material factor (1.0 for plain concrete or brick).

F_d - distance factor depending on geology (varies with a complex function).

F_t - project time factor (1).

The standards do not consider the frequency of vibrations which is important in the analysis of strain response. Most likely frequency can be inferred with the type of geology near or under the structure based on P-waves but the connection is not clear. Another issue is the use of the vertical

component of structure response measured on the foundation rather than by the lateral components that properly relate to wall strain for compliance limits. A discussion of these and other issues with recommendations for changes to the Swedish standards is presented by Eriksson & Jansson (2018).

2.5 France

FFT analysis appears to be used to analyse and adjust vibration amplitudes in terms of peak velocity as it affects alleged damage potential to structures in the French Standard Circular No. 96-52 of 02/07/96. Adjustments of velocity are made using a ponderation (or amplification/deamplification) function based on FFT frequency shown in Figure 3. The function is applied at low frequencies (< 5Hz) to adjust the velocity measurements upward and at higher frequencies (>30Hz) to adjust measurements downward.

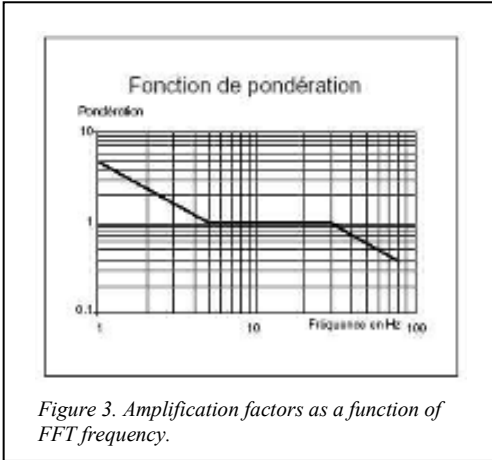


Figure 3. Amplification factors as a function of FFT frequency.

The algorithm used to compute pondered peak velocities can be confusing and difficult to understand by the layman. However, a simple explanation was provided over the interval < 5 Hz (R. Wheeler, personal communication, 11 March 2025). At each frequency, N, the vibration is multiplied by a weighting factor 5/N, such that at 1 Hz, the factor is 5/1 or 5, at 2 Hz it is 5/2, at 4 Hz it is 5/4, and 5 Hz, it is 5/5 or unity (no amplification).

A comparison example is provided in Figure 4 (T. Bernard, personal communication, 2 February 2025) to show the results of ponderation where the low frequency amplitudes (<5 Hz) shown within the unponderated plot (top) are increased in a decreasing manner, shown in the bottom plot for

the final compliance. The ponderated data must not exceed a compliance limit of 10 mm/s. Note, in this example there are no higher frequency components greater than 40 Hz to increase amplitudes.

No explanation is given for this data transformation within the French guidelines. *It is desirable that the reason for and implementation of this process is clarified to fully understand the implications to the standard.*

2.6 Switzerland

The Swiss standards are considered to be the most restrictive of the standards considered for this analysis. Limits are based on the maximum time-correlated vector sum of all three geophone components plotted against the FFT frequency for the component with the highest or peak velocity amplitude.

For residential structures of normal sensitivity and frequent blasting, such as in mining and quarrying, a peak vector sum limit of 6 mm/s is applied for all FFT frequencies up to 30 Hz. Beyond 60 Hz the allowable vector sum is 12 mm/s, which is over four times smaller than the U.S. and U.K. peak velocities limits verified to prevent wall cosmetic cracking. Justifications for such highly restrictive limits to prevent structure crack damage cannot be made.

It is often the case that the Swiss standards are applied to construction projects in the U.S. by engineers writing project specifications without providing guidance as to the deployment of geophones nor the plotting and analysis of data. Geophones are routinely buried in the ground and peak particle velocities are correlated with zero-crossing frequencies. Blasters have difficulty complying with such low standards and projects routinely suffer constant stop-work orders from vibration exceedances until structural engineers can survey all adjoining buildings for cracking. The author has experienced this situation in large cities where the costs to achieve such low vibration amplitudes can be exorbitant based on the need to design a large number of small blasts.

3 CONCLUDING REMARKS AND RECOMMENDATIONS

The diversity of international vibration standards applied to blasting operations near residential structures selected for this study demonstrate the differences among guidelines for deployment and measurement of vibrations ranging from peak

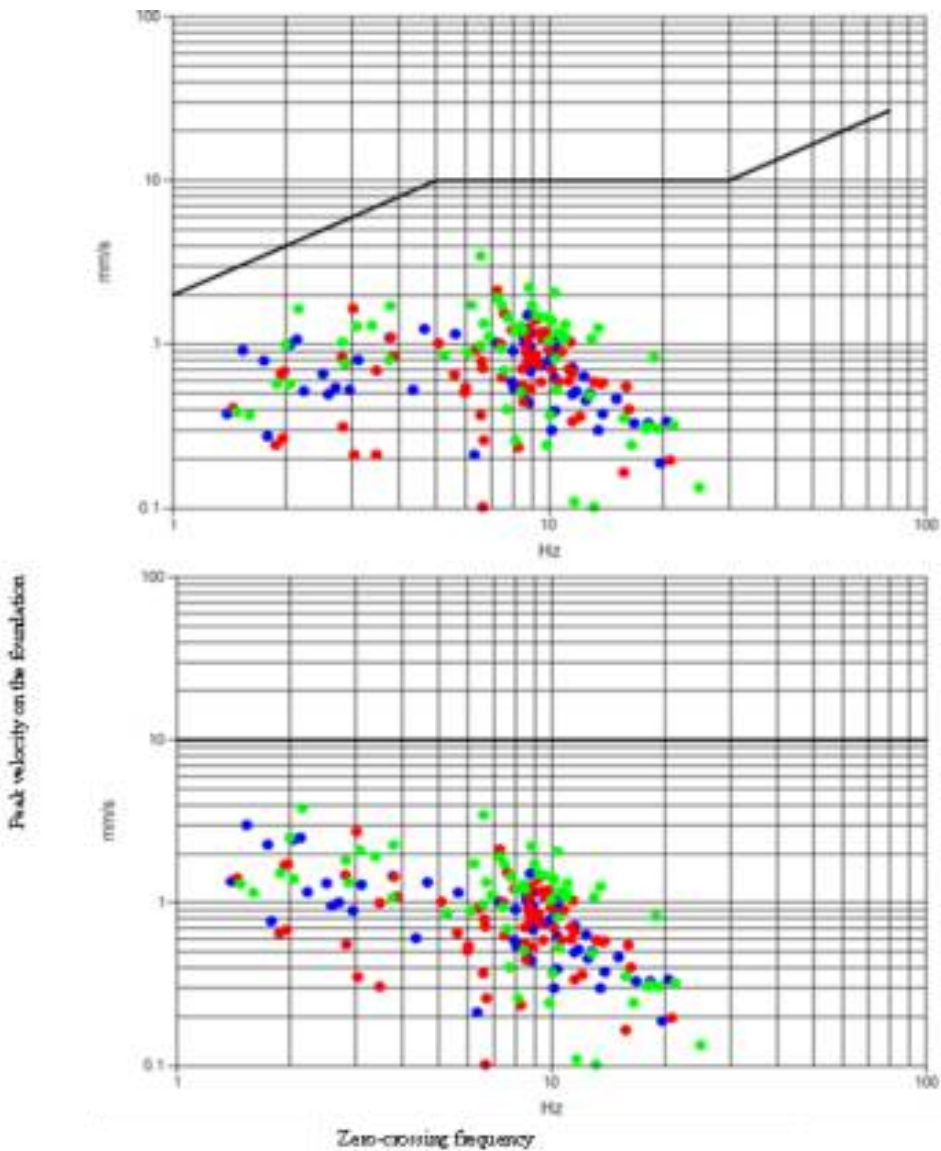


Figure 4. Recorded measurements are shown on non-regulatory plot (top) and ponderated amplitudes plotted on the compliance chart showing the maximum allowable PPV of 10 mm/s (bottom).

component, peak vector sum, and vertical component. The dependent metric correlating velocity ranges from zero crossing frequency, FFT frequency, or distance between blasting and structures.

The high allowable limits in the U.S. are compared in Figure 5 with the Swiss standard.

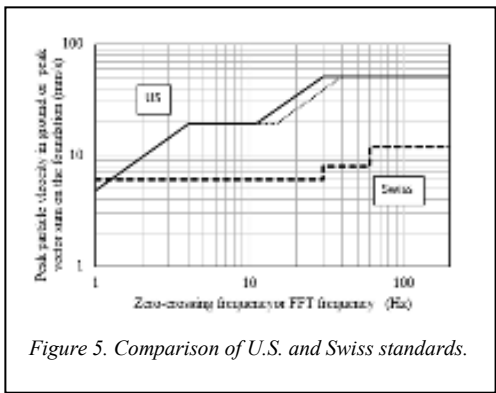
Although the metrics used for guidance are slightly different, the wide difference shows how diverse limits can be when allegedly set to prevent material wall cracking based on the misinformation on safe standards, fear of excessive complaints, and not understanding cost implications.

3.1 Suggestions for future work

It is the desire of the participants of the EFEE-ISEE on-going study to further interactions among different country members in an attempt to unify blasting vibration standards and align the standards to address cosmetic cracking potential in wall materials based on strain. Simplification of the parameters affecting structure response and concepts that make standards a challenge to implement is the first step in this process.

The following are action items to consider for near-future work:

- Educate regulators, persons enforcing standards and guidelines, engineers responsible for developing blasting specifications for mining, quarrying and rock removal works by blasting on the science of structure response, causes of material cosmetic cracking and vibration limits at which cracking might occur.
- Develop standards that address the fundamentals of the degrees of blast-induced cracking (e.g. threshold vs minor) in structures to incorporate velocity and frequency in combination as they relate to displacements acting differentially in structure walls in above-ground structures that might lead to excessive strains and, in turn, exceed yield or failure stresses.
- Develop standards for blasting near underground structures and consider sensitive structures such as hospitals, computers, bridges, etc.



- Establish instrumentation guidelines and protocols to ensure accurate and representative measurements used to evaluate structure cracking potential.

- Work toward a published ‘White Paper’ that summarises relevant peer-reviewed research on blast effects on structures to educate and guide decision makers, seismograph manufacturers and operators, consultants, project engineers, etc., on strain-based criteria to protect structures.

REFERENCES

Siskind, D.E., Stagg, M.S., Kopp, J.W. & Dowding, C.H. 1980. Structure response and damage produced by ground vibrations from surface mine blasting. *Bureau of Mines Report of Investigation* 8507, 74 pp.

Stagg, M.S., Siskind, D.E., Stevens, M.G. & Dowding, D.H. 1984. Effects of repeated blasting on a wood-frame house. *Bureau of Mines Report of Investigation* 8896, 82pp.

ITA Activity Group - Excavation 2016. Vibration control in urban drill and blast tunneling. *Report no. 8. ISBN: 978-2-9701013-7-6.*

Ericksson, M.D. & Jansson, A. 2018. Assessment of the Swedish Standard for blasting induced vibrations - A parametric study on wave propagation in rock and clay using the finite element method. *MS thesis. Department of Mechanics and Maritime Sciences, Division of Dynamics, Chalmers University of Technology, Gothenburg, Sweden.*

Meyers, J. & Aimone-Martin, C. 2019. Strain-based vibration criteria for blasting in New York City. *Proc. of the 45th Annual Conference on Explosives & Blasting Technique, ISEE Nashville; Published in The Journal of Explosives Engineering, Vol. 36, No.5, Sept-Oct., pp 6-13.*

Gjødvaad, J.F. & Jern, M. 2019. Vibration monitoring standards connected to the use of explosives in Europe. *Proc. of the 10th conf. on Explosives and Blasting, EFEE Helsinki, Finland, pp 51-60.*

Gjødvaad, J.F. & Jern, M. 2020. Vibration monitoring standards connected to the use of explosives in different countries. *Proc. of the 46th Annual Conference on Explosives and Blasting Technique, Denver. USA.*

Noren-Cosgriff, K.M., Ramstad, N., Neby, A. & Madshus, C. 2020. Building damage due to vibration from rock blasting. *Soil Dynamics and Earthquake Engineering* 138.

Gjødvad, J., Jern, M., Dowding, C. & Bernard, T. 2022a. Instrumentation of a structure for comparison of regulations worldwide - vibration monitoring standards connected to the use of explosives in different countries, Part II. *49th ISEE conference in Las Vegas*, NV.

Gjødvad, J. & Jern, M. 2022b. Workshop on vibration standards. *11th World Conf. on Explosives and Blasting, EFEE* in Maastricht, Netherland.

Gjødvad, J.F., Jern, M., Dowding, C.H. & Aimone-Martin, C. 2024. Comparison of compliance measures for blasting near an industrial structure vibration monitoring standards connected to the use of explosives in different countries, Part III. *Proc. of the 50th Annual Conference on Explosives & Blasting Technique*, Savannah, GA.

Aimone-Martin, C., Lenker, S., Senules, J., Meins, B. & Meins, O. 2022. Structure responses to a confined opening cut blast, Sumterville, FL. *48th Annual Conference on Explosives & Blasting Technique*, Las Vegas.

Aimone-Martin, C., Dowding, C., Gjødvad, J. & Jern, M. 2025a. Comparison of international vibration standards for quarry blasting close-in to a residential structure. *51st Annual Conference on Explosives & Blasting Technique*, Cherokee NC.

Aimone-Martin, C., Dowding, C.H., Rudenko, D., Loukides, C., Newman, S., Gjødvad, J. & Jern, M. (2025b) Comparison of international vibration standards for quarry blasting close-in to a residential structure in Michigan, USA. *Proc. of the 13th Conf. on Explosives and Blasting*, Krakow, Poland.

Chapter 56 of the 2022 Fire Code (Section 2.1 above).

https://www.nyc.gov/assets/fdny/pdfviewer/viewer.html?file=chapter-56-2022.pdf&section=firecode_2022

RETURN TO CONTENTS PAGE

Effect of blasthole diameter on vibration from signature hole blast monitoring

R. Yang & J.P.R. Proulx

Orica USA Inc.

J. Desrochers & L. Etulle

Orica Canada

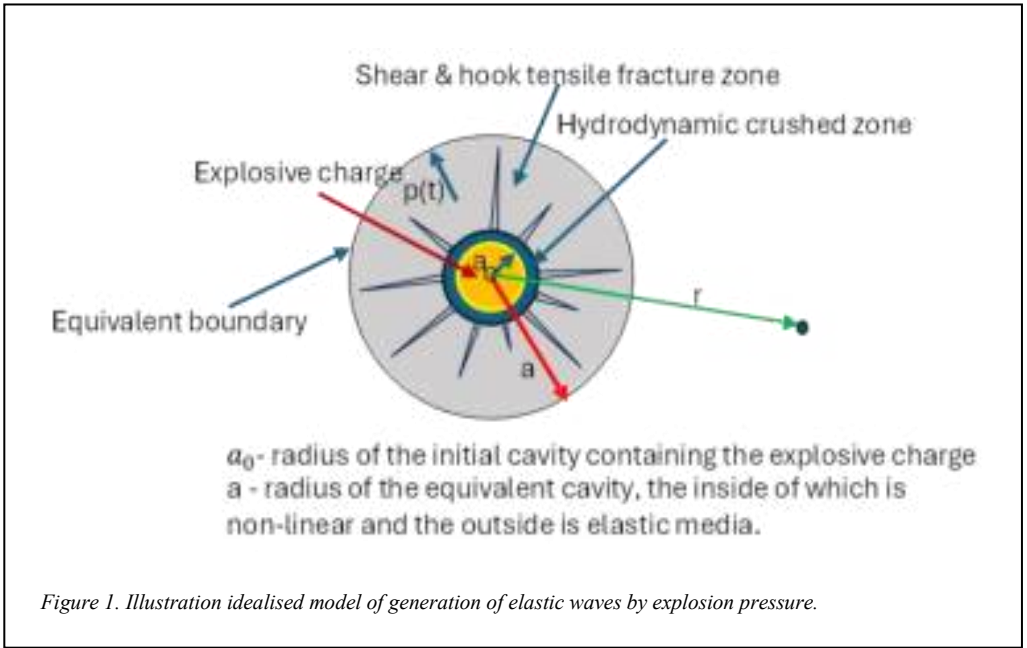
ABSTRACT: This study investigates the impact of blasthole diameter on blast vibration, a crucial factor in controlling vibrations near highwalls and structures. A field study monitored vibrations from signature hole blasts of three diameters: 203 mm, 254 mm, and 349 mm. These holes were drilled into confined ground and fired with long delays to isolate vibrations from an individual blasthole. Triaxial accelerometers recorded data at scaled distances range of 0.8 to 6.0 m/kg^{0.5}. Results showed that at scaled distances over 3.5 m/kg^{0.5}, peak particle velocity (PPV) and peak particle acceleration (PPA) were only negligibly affected by diameter. However, at scaled distances under 3.5 m/kg^{0.5}, larger diameters increased vibration amplitudes, aligning with theoretical predictions. These findings suggest that for far-field vibrations, amplitude mainly depends on charge weight, while near-field vibrations may be influenced by diameter. The study's insights are valuable for designing blasts to minimise vibrations near sensitive areas.

1 INTRODUCTION

The impact of blasthole diameter on blast vibration has been a longstanding topic of interest in the field of blast vibration control, particularly concerning the safety and stability of highwalls and nearby structures. Despite its significance, the literature lacks comprehensive theoretical frameworks and reliable empirical data to draw definitive conclusions. This gap in knowledge presents practical challenges for mining operations, where decisions regarding the investment in various drill sizes are critical. Mines often face questions about how to select blasthole diameter to optimise blast outcomes, such as minimising vibrations affecting highwalls, achieving desired rock fragmentation distributions,

and controlling airblast levels. Additionally, there is the question of whether the investment in different drill sizes is justified by the potential benefits.

To address these challenges, this paper investigates the effects of blasthole diameter on blast vibrations, focusing on the near-field region. The study combines theoretical analysis with field data collection and interpretation aiming to provide a comprehensive understanding of how blasthole diameter influences blast vibration characteristics. By examining signature hole blasts of varying diameters 203 mm, 254 mm, and 349 mm, drilled into fully confined ground, the research aims to isolate and analyse the specific impact of diameter on vibration amplitudes. The findings are expected to offer valuable insights for



blast design, particularly in selecting appropriate blasthole diameters to control vibrations near sensitive areas such as highwalls and residential structures.

2 THEORETICAL PREDICTIONS OF BLASTHOLE DIAMETER ON BLAST VIBRATION

A recently published paper (Yang 2025) proposed a theory on the mechanism of blast vibration generation that can consistently interpret many field observations. The new theory proposed that blast vibration is influenced not only by the detonation shock pressure but also the detonation gas expansion and interaction with surrounding rock. Sharpe theoretically deduced that generation of blast vibration from a spherical charge of the radius a_0 in an infinite large medium is related to a spherical boundary of radius a ($a > a_0$). The rock between radius a_0 and a will respond to the explosive load nonlinearly, undergoing crushing, fracturing, and plastic deformation. The blast vibration will propagate radially outward from the spherical boundary of radius a . The rock outside the radius a behaves elastically to the stress wave generated by the detonation of the spherical charge. Sharpe (1942) referred to the spherical boundary of radius (a) as the equivalent cavity. Figure 1 illustrates the idealised model of the generation of elastic waves by explosion pressure.

According to Sharpe's theory, Yang (2025) revealed that the peak particle velocity (PPV) of blast vibration from a spherical charge is:

$$PPV \propto \frac{P_0 a^2}{G} \quad (1)$$

- directly proportional to the peak pressure (P_0) at the equivalent cavity that depends on the blast loading and the rock yield strength.
- directly proportional to the area of the equivalent cavity ($4\pi a^2$), a stronger explosive or a charge with a large diameter generates a larger equivalent cavity.
- directly proportional to the ratio of pressure amplitude at the equivalent boundary (P_0) to rigidity of the rock (G).

When an explosive charge is fully confined within rock, the detonation pressure and expanding explosion gases generate a fully developed equivalent cavity with a maximum radius (a). This results in a high PPV outside the equivalent cavity radius. Additionally, the pressure amplitude (P_0) at the equivalent cavity reaches the rock's yield or damage strength.

A charge with the large diameter contains more explosive mass and energy. The large charge will provide a higher effective shock pressure and more gas expansion energy. It will generate a large equivalent cavity.

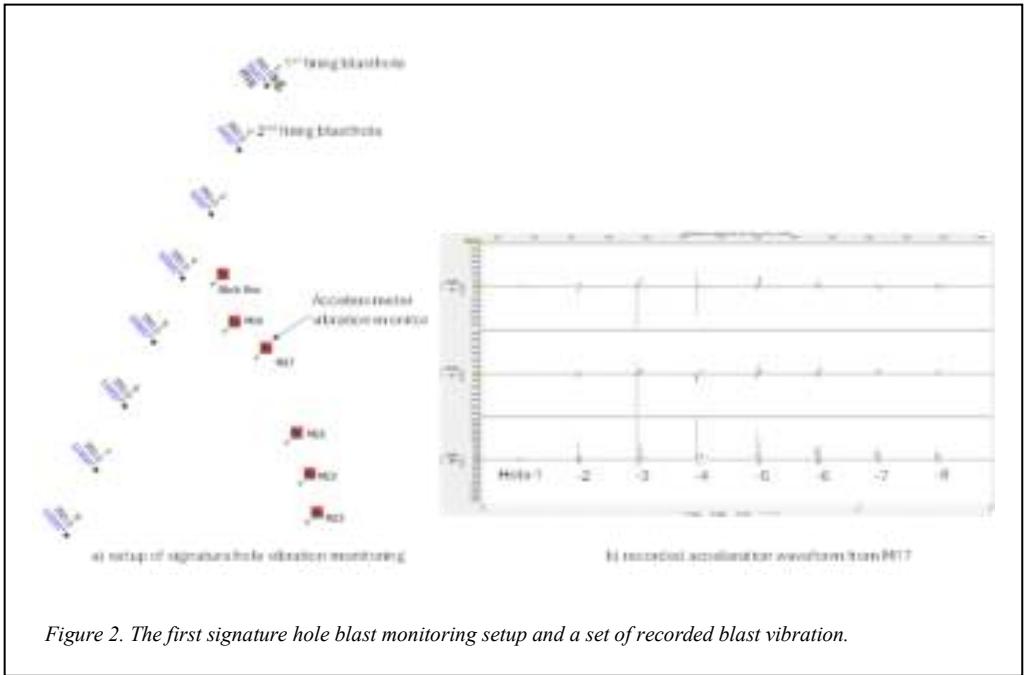


Figure 2. The first signature hole blast monitoring setup and a set of recorded blast vibration.

Conversely, if an explosive charge is detonated with a small burden, although the initial shock pressure may be independent of the burden, gas pressures may be released earlier, before the equivalent cavity is fully developed. This premature venting results in a smaller radius ‘a’ for the equivalent cavity. However, the pressure at the equivalent cavity could be the same as when fully confined because the plastic wave travels slower than the elastic precursor, leading to a lower PPV.

It appears that the rock crushing, fracturing, and plastic deformation between radius r_{cr} and r_{pl} significantly contribute to blast vibration, like how the interactions along the sides of a fault largely contribute to earthquake magnitude.

The dominant frequency of the blast vibration from a spherical charge is proportional to the ground p-wave velocity and inversely proportional to the size of the equivalent cavity ($\bar{a}$).

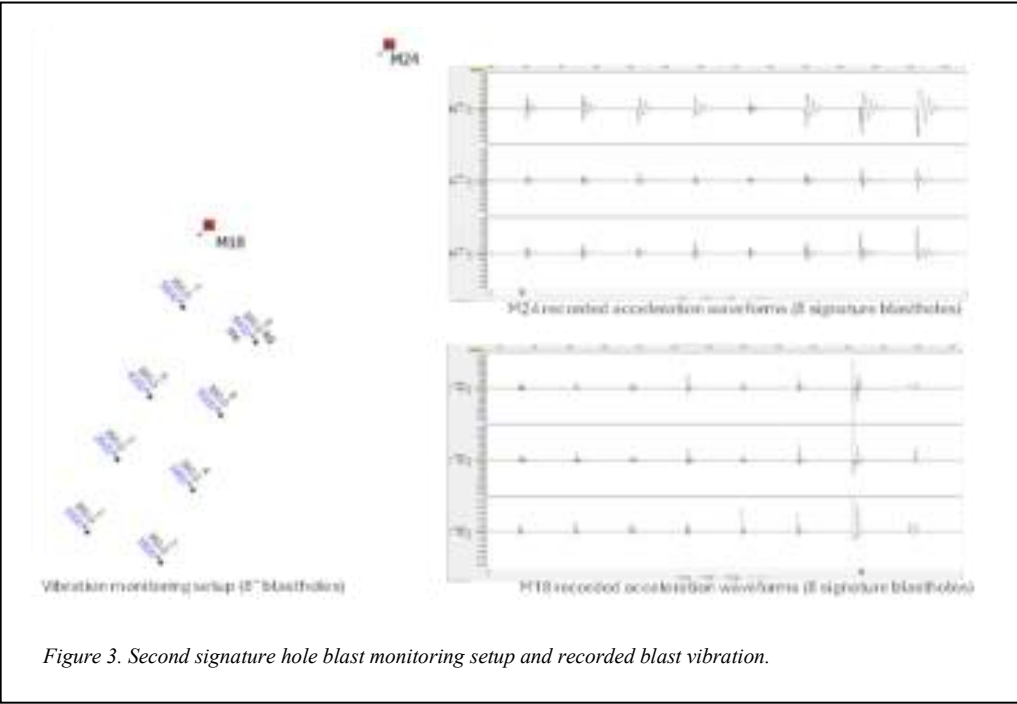
$$f = \frac{v_p}{2\pi\bar{a}}, \quad \bar{a} \propto \frac{r_{pl}}{a} \quad (2)$$

This implies that a competent hard rock blasted with a small charge diameter would produce a higher dominant frequency than with a large diameter charge. A blast in a low sonic velocity ground results in low frequency vibration. This trend is consistent with field observations (Yang *et al.* 2024a).

3 FIELD SIGNATURE HOLE BLAST VIBRATION MONITORING

Signature hole blast vibration was tested for different blasthole diameters at an open pit iron mine in Canada. Three sets of signature hole blasts were conducted at two areas of the mine. At the first area of the mine, two sets of signature hole blasts were monitored to compare the near-field blast vibration from the blastholes of two diameters 349 mm and 203 mm. The first test contained eight signature blastholes of 349 mm diameter. The blasthole depth was 15 m on average and loaded with 350 kg emulsion explosives and 12 m stemming. Figure 2 shows the setup of the blast vibration monitoring and an example of the recorded particle acceleration waveforms from the eight signature blastholes. The delay time between the signature holes was 800 ms to ensure the waveform from each signature blasthole is unaffected by other blastholes. The spacing between the signature holes was 11.1 m, significantly larger than the blasthole spacing and burden of the production blast at the site, which was 6.5 m. With a charge length of 3 m in each signature hole, there was negligible broken ground filtering from earlier firing signature holes affecting the blast vibration.

The second test was conducted at the same pit of the mine. The blasthole diameter was 203 mm and depth was 15 m on average. Eight signature



holes were loaded with 120 kg of emulsion explosives and 11 m stemming on average. Again, the delay time between the signature holes was 800 ms. Again, the signature hole spacing was 8 m compared to the typical production blast pattern of blasthole spacing and burden of 5.4 m for the 203 mm diameter blastholes. Figure 3 shows the setup

of the blast vibration monitoring and the recorded particle acceleration waveforms from the eight signature blastholes at the two monitor locations.

At the second pit of the mine, the third signature hole test was conducted with nine blastholes of different diameters, four of which were 349 mm in diameter and loaded with 350 kg

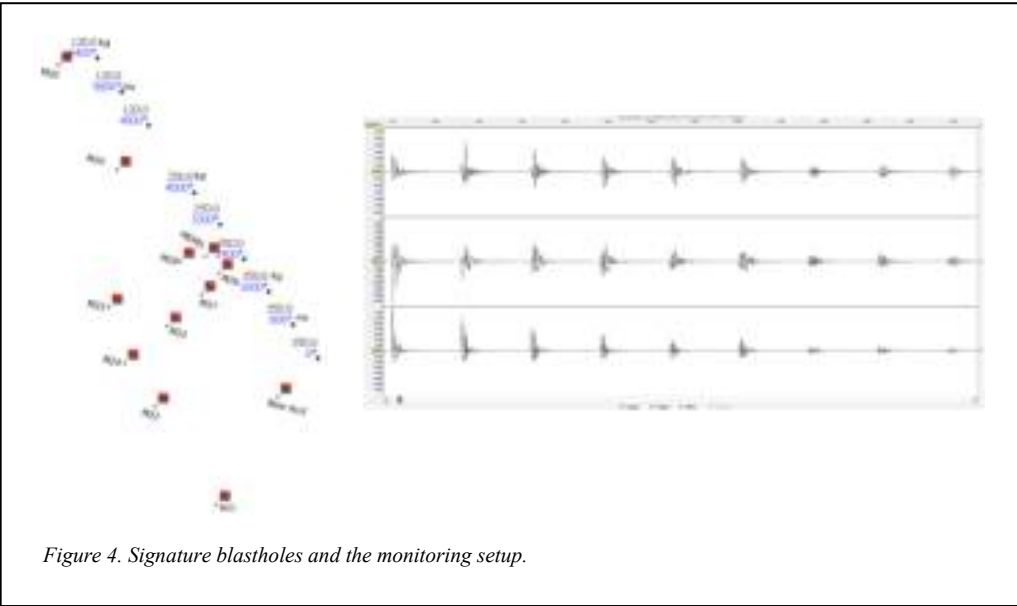


Table 1. Rock properties of the test sites.

Rock	Density (kg/m ³)	UCS (MPa)	Young's modulus (GPa)	Poisson ratio
Amphibolite	3068	100	57	0.26
Iron formation	3106	218	73	0.36
Average of two	3087	159	65	0.31

explosives, two of which were 254 mm and loaded with 250 kg explosives, and three of which were 203 mm in diameter and loaded with 120 kg explosives. Figure 4 shows the signature blastholes and the monitoring setup with 12 accelerometer-based monitors employed.

The charge lengths for different diameters of blastholes are comparable, ranging from 3 m to 4 m. Therefore, the effect of charge length variation on blast vibration is minimal. The explosive used was a 100% ammonium nitrate and fuel oil-based emulsion. The nominal velocity of detonation (VoD) for blasthole diameters of 203 mm and 349 mm should be the same or show negligible differences.

The rock at the mine contains the iron formation and the waste rock amphibolite. The former is the ore bearing lithological unit at the mine. The iron rich units can be described as medium, well banded, homogeneous grey to pale blue rock, composed of hematite and quartz with minor amounts of magnetite. The amphibolite is relatively softer. However, the first test site contains the mixture of the two rock types. Therefore, there is no clear distinction for the rocks at the first sites. The second test was conducted mainly in the iron formation. The basic rock properties for the test area were provided by the mine and are shown in Table 1. The average values of the rock properties from the two rock types may be more representative to the first test area. The iron formation is more representative to the second test area. It should be noted that the iron formation is marginally harder than the Amphibolite waste, in particularly, the density and the Young's modulus are not significantly different.

4 FIELD MEASUREMENT RESULT

The recorded triaxial particle acceleration waveforms are integrated to triaxial particle velocity waveforms. Peak particle acceleration (PPA) and peak particle velocity (PPV) are obtained from the waveforms, respectively. The site characteristic vibration attenuation laws have

been established - peak particle acceleration (PPA) and velocity (PPV) versus the charge weight scaled distances. Since the signature hole blast vibration is not affected by blasthole vibration superpositions, the signature hole vibration attenuation laws are the characteristic blast vibration response of the site to the chosen blasthole loading configuration, which can be used for production blast vibration analysis and modelling.

The frequency distribution of the vibration from signature hole blasts can be used to approximate the ground resonant frequency under the blasthole loading conditions (Yang *et al.* 2009), which can be used to manage the dominant frequency of the blast vibration through the delay timing design of a blast (Yang *et al.* 2009, 2024).

4.1 PPA and PPV comparison from different blasthole diameters

Figure 5 presents a comparison of the peak particle acceleration (PPA) from signature hole blasts with two diameters: 349 mm and 203 mm. There are 89 data points for the 349 mm diameter signature holes and 42 for the 203 mm diameter blastholes, both of which exceed the small data sampling criterion of 30 measurements. The recorded vibration data sets from signature holes with a diameter of 254 mm are fewer than 20 and are not included in the comparison.

The figure indicates that at a charge weight scaled distance of less than 3.5 m/kg^{0.5}, the PPA from the larger 349 mm diameter is higher than that from the 203 mm diameter. At scaled distances greater than 3.5 m/kg^{0.5}, the difference becomes negligible. Notably, at a scaled distance of 2 m/kg^{0.5}, the difference can be as large as 50% of the PPA for the 349 mm diameter. Additionally, Figure 6 compares the peak particle velocity (PPV) from signature hole blasts with the same diameters: 349 mm and 203 mm. The figure shows that at a charge weight scaled distance of less than 3.5 m/kg^{0.5}, the PPV from the larger 349 mm diameter is higher than that from the 203 mm diameter. At scaled distances greater than 3.5

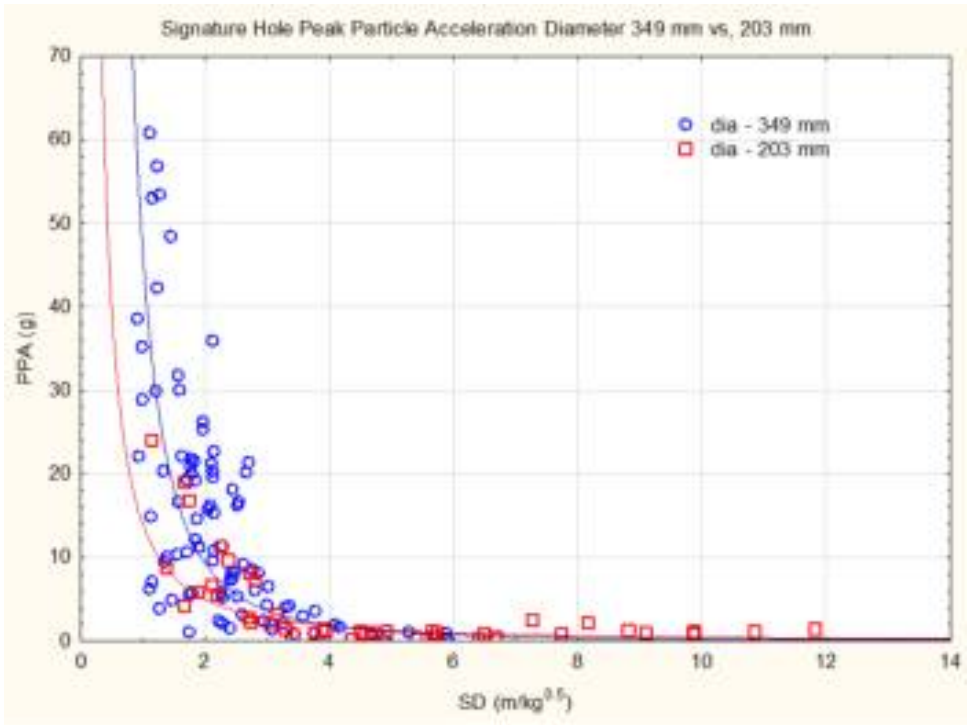


Figure 5. Comparison of the peak particle acceleration (PPA) from signature hole blasts with two diameters: 349 mm and 203 mm.

$\text{m/kg}^{0.5}$, the difference is negligible. The closer to the blasthole, the greater the PPV difference, with a 50% difference at a scaled distance of 2 $\text{m/kg}^{0.5}$ compared to the 203 mm diameter. The trend shown in Figures 5 and 6 is consistent with the theoretical predictions above in equation (1); the larger charge diameter creates a larger equivalent boundary that results in higher vibration amplitude – PPV.

Using small diameter blastholes can be an effective method for vibration control at small charge weight scaled distances. In addition, smaller diameter blastholes typically result in reduced explosive charge per hole, which can additionally help in minimising the intensity of vibrations generated during blasting. This approach is beneficial in sensitive areas where controlling ground vibrations is crucial to prevent damage to nearby structures or to comply with regulatory limits. By optimising the blast design with smaller diameters, it is possible to achieve the desired fragmentation while maintaining control over the environmental impact of the blasting operations.

4.2 Amplitude frequency spectrum comparison from different blasthole diameters

The Fast Fourier Transform (FFT) was chosen to be performed on the particle acceleration waveforms rather than the particle velocity waveforms because filters had been applied to the acceleration waveforms to obtain the latter, which can introduce errors into the results. The amplitude frequency spectrum was obtained by applying it over the average of the triaxial components of the particle acceleration waveforms of the blast vibration, represented as:

$$a_f(f) = \frac{a_x(f) + a_y(f) + a_z(f)}{3} \quad (3)$$

This approach considers the relative amplitudes of the triaxial components in the analysis. Secondly, a Fast Fourier Transform (FFT) is performed over a time window that includes vibrations from blastholes of the same diameter, as illustrated in Figure 7 for the analysis of blast vibrations recorded in the third test (refer to Figure 4). The time window for the FFT analysis of the

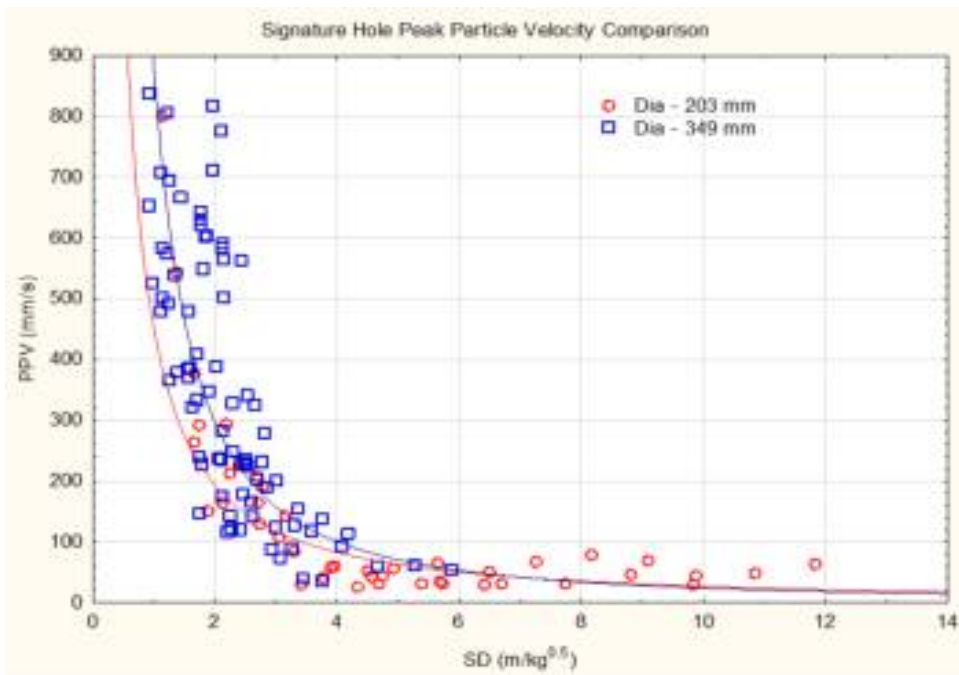


Figure 6. Comparison of the peak particle velocity (PPV) from signature hole blasts with two diameters: 349 mm and 203 mm.

first and second tests includes vibrations from the eight signature holes of the same diameter in each test. This method allows for obtaining the average

amplitude spectrum for vibrations from blastholes of the same diameter.

Figure 8 shows the amplitude spectrums from

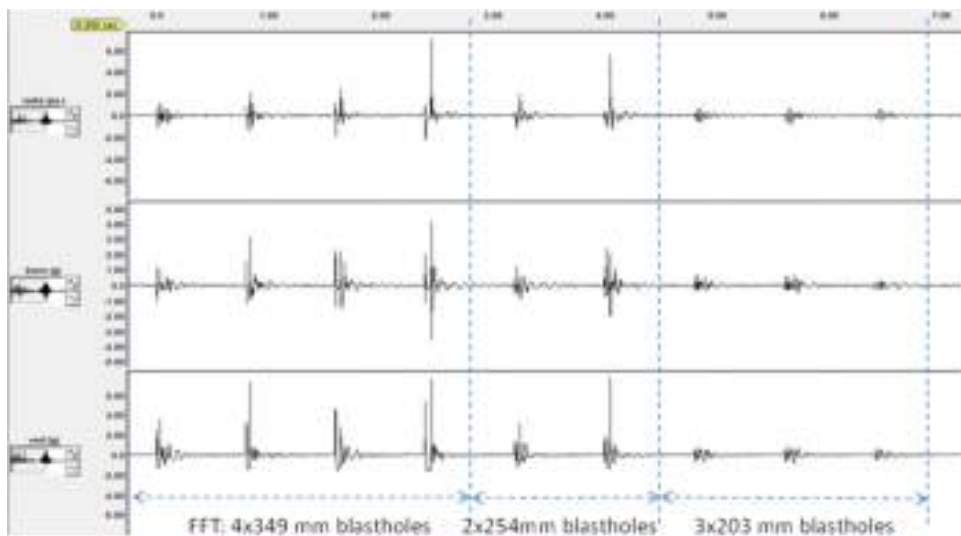
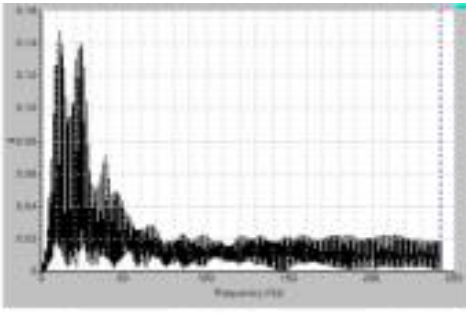
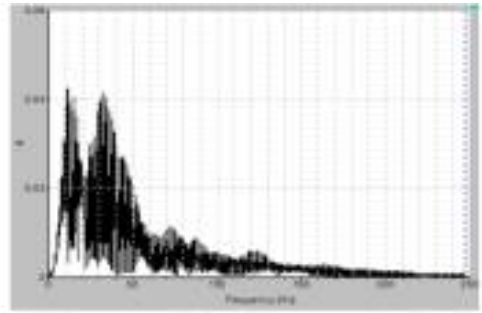


Figure 7. A time window containing the vibrations from the same diameter blastholes was used for FFT.



a) Eight signature hole of 349 mm at M22 (Fig. 2)



b) Eight signature hole of 203 mm at M24 (Fig. 3)

Figure 8. Comparison of amplitude frequency spectrum: diameter 349 mm vs. 203 mm from the tests 1 and 2.

the blastholes of the diameter: 349 mm and 203 mm in the first and second tests. As described above, the two tests were in the same pit of the mine. The distances from the monitors to the blastholes in the Figure a) and b) are comparable. Figure 8 clearly shows that the vibration from the signature blastholes of the diameter 203 mm contains more higher frequency components (> 30 Hz) than that from the blastholes of diameter 349 mm.

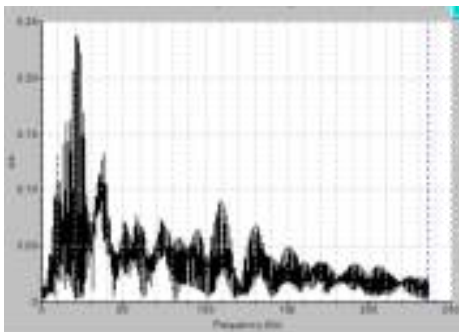
Figure 9 compares the amplitude spectrums from the third signature hole tests. Again, the distances from the monitor 25 to the four-signature hole of diameter 349 mm is similar to those of diameter 203 mm. Again, the figure clearly shows that the vibration from the signature blastholes of the diameter 203 mm contains more higher frequency components (> 30 Hz) than that from the blastholes of diameter 349 mm.

The trend shown in Figures 8 and 9 is consistent with the theoretical prediction in

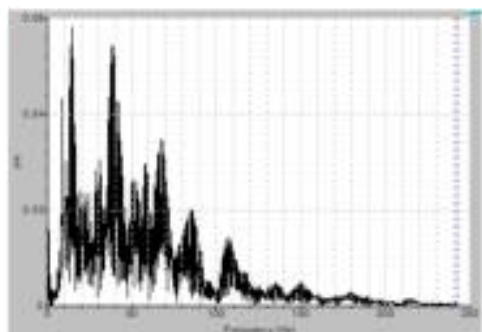
equation (2), indicating that a small diameter charge produces more high-frequency components than a large diameter charge for near-field blast vibrations, such as at a charge weight scaled distance of less than $6 \text{ m/kg}^{0.5}$. It is commonly accepted that higher frequency components cause less damage to structures than lower frequency components for the same amplitude of blast vibration. Therefore, using small diameter blastholes can be beneficial in sensitive areas where controlling ground vibrations is crucial to prevent damage to nearby structures or to comply with regulatory limits.

5 DISCUSSION AND CONCLUSIONS

The results indicate that at scaled distances greater than $3.5 \text{ m/kg}^{0.5}$, both Peak Particle Velocity (PPV) and Peak Particle Acceleration (PPA) are essentially unaffected by the blasthole diameter, which ranges from 203 mm to 349 mm.



a) Four signature holes of 349 mm at M25 (Fig. 4)



b) Three signature holes of 203 mm at M25 (Fig. 4)

Figure 9. Comparison of amplitude frequency spectrum: diameter 349 mm vs. 203 mm from the third test.

This suggests that for far-field blast vibration, the amplitude depends mainly on the charge weight for a given explosive product. At scaled distances less than $3.5 \text{ m/kg}^{0.5}$, the data suggests that larger diameter blastholes may generate higher vibration amplitudes (PPA and PPV) compared to smaller diameter blastholes. This observation aligns with the theoretical prediction by Yang (2025) that an explosive charge with a larger diameter creates higher amplitude blast vibration in the near field. Such effects diminish at scaled distances greater than $3.5 \text{ m/kg}^{0.5}$ due to the vibration amplitude-dependent attenuation of rock as the vibration propagates.

As mentioned, the charge lengths for different diameters of blastholes are comparable, ranging from 3 m to 4 m. Therefore, the effect of charge length variation on blast vibration is minimal. The tests were conducted in a relatively hard, competent rock mass. It would be worthwhile to conduct similar tests in soft rock to compare the blast vibration from different explosive charge diameters.

More detonation energy per unit length of the charge is generated from a large diameter charge than from a small diameter charge because the larger charge produces the borehole wall pressure for a longer duration, causing more damage to the surrounding rock. The explosive used was a 100% ammonium nitrate and fuel oil-based emulsion. The nominal velocity of detonation (VoD) for blasthole diameters of 203 mm and 349 mm should be the same or show negligible differences, minimising the effect on blast vibration differences.

The findings have important implications for blast design and vibration control, particularly in selecting blasthole diameters near highwalls or residential structures. Blast vibration from small diameter blastholes would also have a higher dominant frequency compared to the blast vibration from large blastholes (Yang 2025). This research contributes to the development of more effective and efficient blast vibration control strategies in mining operations.

REFERENCES

Sharpe, J.A. 1942. The production of elastic waves by explosion pressures. I. Theory and empirical field observations. *Geophysics*, 7(2), 144-154. Doi:10.1190/1.1445002.

Yang R., Whitaker T. & Kirkpatrick S. 2009. PPV management and frequency shifting in soft ground near highwalls to reduce blast damage.

Blasting and Fragmentation, April, 3(1), 21-42.

Yang, R. 2025. Influence of explosive detonation shock loading and gas expansion on blast vibration. *14th international symposium on rock fragmentation by blasting*, April 7 – 10, Antalya, Turkey.

Yang, R., Proulx, J.P.R. & Rodriguez, A. 2024a. Shifting blast vibration frequencies and PPV at highwalls from near-field blasts using the multiple seed waveform (MSW) blast vibration model. *58th US rock mechanics/geomechanics symposium*, June 23-26, 2024.

[RETURN TO CONTENTS PAGE](#)

Optimising signature hole analysis: evaluating charge quantity and confinement effects on seed waveforms

T. Moser

Société Suisse des Explosifs (SSE) Switzerland

ABSTRACT: The use of electronic detonators with signature hole analysis represents a modern method to optimise and minimise blast-induced vibrations. A signature hole is detonated in isolation to generate a seed waveform that characterises the ground response of a single blasthole. These seed waveforms are used in conjunction with superposition analysis to evaluate how different delay timings between holes can achieve destructive interference and thereby reduce peak particle velocity (PPV). A central challenge in this process is determining what constitutes a good seed waveform and how best to obtain one in practice. This study explores how the charge quantity and confinement level of a signature hole affect waveform characteristics. Seven signature holes were drilled and fired under controlled conditions. Vibrations were recorded using seismographs arranged in three linear arrays at varying distances. The findings aim to offer field-practical guidance for generating effective seed waveforms while minimising disruptions and safety risks in production blasting.

1 INTRODUCTION

Signature hole analysis, when used with precise electronic detonators, is a powerful tool for optimising blasting designs by minimising blast-induced ground vibrations. A signature hole, isolated from the production blast, is used to record a seed waveform that captures the vibration response from a single explosive event. This waveform serves as the basis for waveform superposition analysis, which simulates the interaction of vibrations from multiple holes fired at different delays. The goal is to find delay timings that result in destructive interference and thus lower PPVs.

However, defining the ideal configuration for a signature hole remains controversial. Some argue

that the charge weight and confinement (burden) of the signature hole should replicate those of production holes. Others propose using a fully confined charge, one with such a large burden that no rock movement can occur, in order to reduce noise from the surrounding geology and produce a cleaner waveform.

In practice, several operational constraints emerge. Firing a signature hole as part of a production blast requires a 1 - 2 second delay, which can lead to flyrock or air-overpressure if the burden is insufficient. Firing it separately often results in a pre-shot face, requiring changes to the subsequent blast design and timing, which can impact safety and efficiency. Thus, there is a need to balance waveform quality with practical and safe execution.

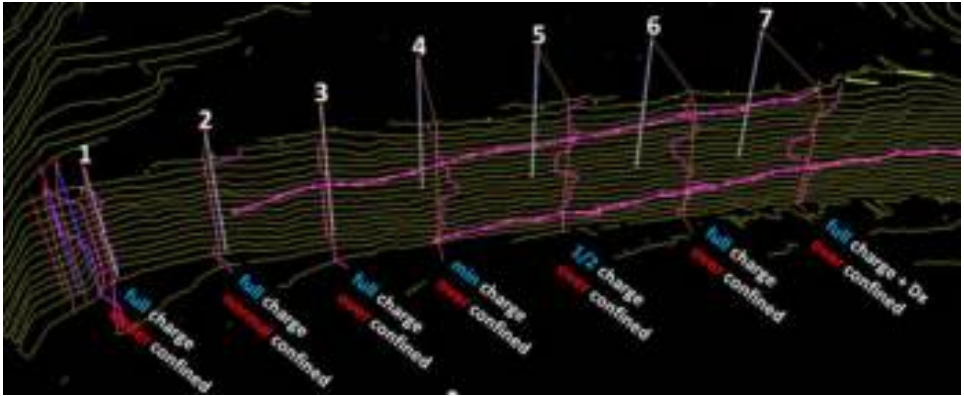


Figure 1. Depiction of the seven signature holes and their respective charges and confinements.

2 OBJECTIVES

To better understand what makes a signature hole effective for delay timing optimisation, this study aims to:

- Assess the impact of confinement level and charge quantity on seed waveform quality.
- Determine how these variables influence both the amplitude and frequency of the resulting vibrations.
- Evaluate whether reduced or differently confined charges can produce representative waveforms.
- Propose practical strategies for optimising signature hole execution without interfering with production.

3 EXPERIMENTAL DESIGN

3.1 Signature hole configurations

A total of seven signature holes were drilled and charged under varying conditions of confinement and charge weight as shown in Figure 1 and Table 1. Each hole was fired independently, the produced vibrations were recorded by three linear arrays of seismographs.

3.2 Field considerations

The signature holes were placed to achieve standardised test conditions, ensure safety, and maintain proximity to a monitored structure. Hole 1, the most under-confined, was positioned in a corner between the production bench and a finished highwall to minimise the risk of flyrock. Holes 2 and 3, representing normal burden/confinement conditions, were located in

homogeneous hard limestone. Hole 4, with a reduced charge weight, was placed to eliminate the influence of intersecting geological formations. Holes 5 - 7 were located as far from the monitored structure as possible to reduce the risk of elevated peak particle velocities (PPVs) due to over confinement.

Table 1. Signature hole configurations of charge weights, burden, and confinement.

Signature Hole	Explosives (Tovex Se/Tovex T) Charge wt.	Burden	Confinement
(#)	(kg)	(m)	
1	36.20	2.3	under
2	36.20	3.3	normal
3	36.20	5.0	over
4	4.20	11.5	over
5	18.40	11.5	over
6	36.20	11.5	over
7*	36.48	11.5	over

* Loaded with detonating cord: Detonex 24g/m

3.3 Seismograph setup

Seismographs (SYSCOM and Menhir shown in Figure 2) were deployed in three linear arrays as illustrated in Figures 3 and 4, each consisting of three seismographs oriented toward the west, north, and east of the blast site. All the



Figure 2. Seismographs used for this study: Menhir (left) and SYSCOM (right).

seismographs had their X-axis oriented towards the signature hole blasts. All geophones were firmly anchored to rock or concrete foundations using screws and plastic plugs. Menhir devices were mounted with brackets to ensure secure coupling.

4 GEOLOGICAL HOMOGENEITY ASSESSMENT

To verify that geological variability did not significantly affect the results, scaled distance regression analyses were performed using ground



Figure 3. Overhead map of seismograph linear arrays with Stations 1-9.

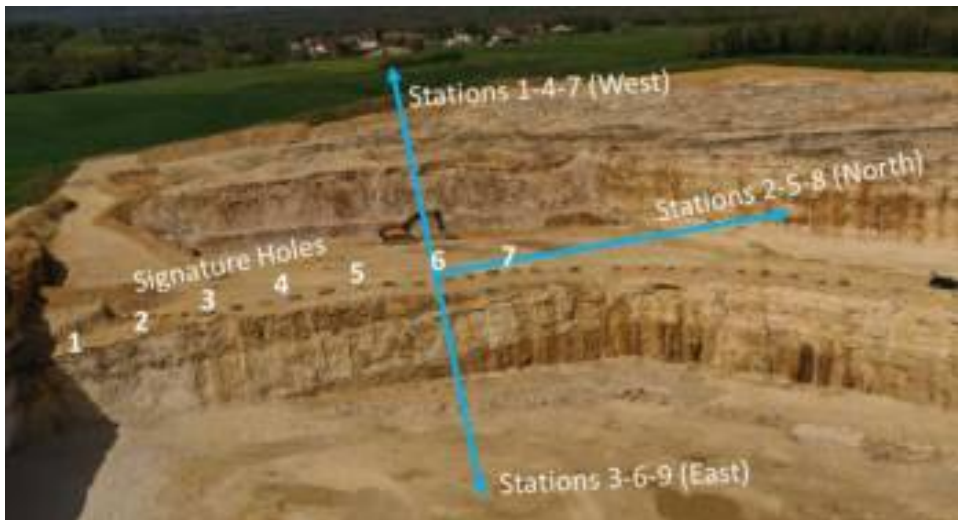


Figure 4. Arrangement of all signature holes and seismograph linear arrays with Stations 1-9.

motion measurements from the seven signature hole blasts. Figures 5.-.7 show the regression plots for each of the three linear arrays. Each linear array consisted of three seismograph monitoring stations. Figure 8 presents the regression plot for the combined dataset from all seismographs in the arrays.

Table 2. Seismograph settings.

Seismograph settings	
Sample rate	1000 Hz
Trigger	0.1 mm/s 2s pre /10s post / 30s max
Frequency range	Syscom: 1-315 Hz Menhir: 0 -unlimited

The regression correlations ranged from 79% to 98%, as illustrated in the figures: West - 79%, North - 98%, East - 89%, and all stations combined - 80%. The regression analysis demonstrates a good level of correlation across the full length of the linear arrays and among all seismograph monitoring stations, indicating a good degree of geological homogeneity. Although some scatter was observed in the western array, the overall consistency confirms that geological variability likely has minimal influence on the recorded waveforms. This homogeneity enhances

the reliability of waveform comparisons between the different signature holes, allowing for more accurate analysis and evaluation of confinement and charge effects.

5 WAVEFORM SUPERPOSITION ANALYSIS

5.1 Methodology

Waveform superposition involves simulating a production blast by duplicating and time-shifting a seed waveform across various delays. Calculations were performed using 30 theoretical blastholes and delay ranges between 1 - 50 ms in 1 ms increments.

All calculations were conducted under the assumption that the blastholes were co-located, meaning they occupied the same spatial point within the rock mass. This assumption was made deliberately to eliminate the influence of spatial separation on the resulting waveforms, such as variations in propagation path length, geological heterogeneities, or directional attenuation. By doing so, the analysis could focus solely on the intrinsic characteristics of the waveforms generated by different charge and confinement configurations without the confounding effects introduced by varying source locations. This approach enables a more controlled and direct comparison of the signature hole responses, improving the reliability of waveform-based interpretation and analysis.

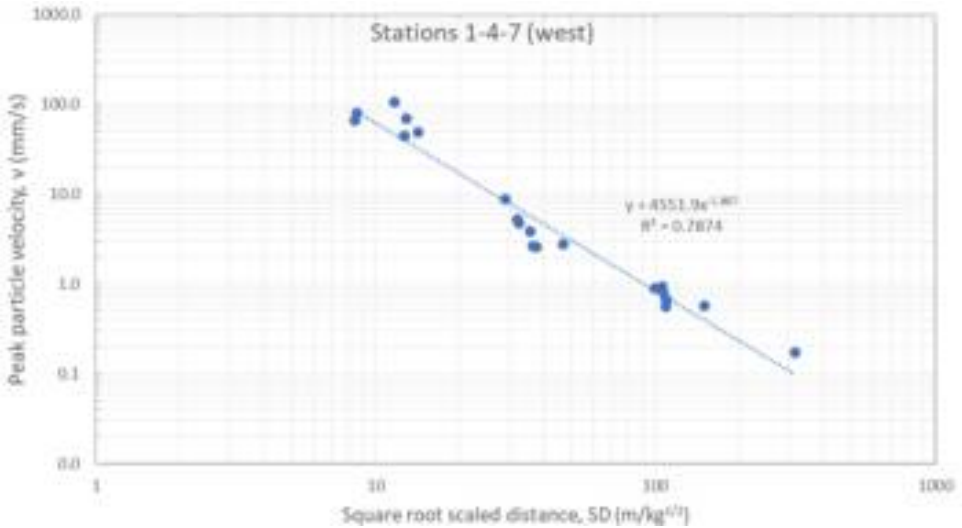


Figure 5. Peak particle velocity versus square root scaled distance for stations 1-4-7 (west).

Figures 9 through 17 show the delay intervals versus theoretical Peak Particle Velocities (PPVs) obtained from waveform superposition calculations for each signature hole at seismograph monitoring stations 1 through 9. The colored lines in each plot represent the calculated PPVs for different signature holes at each station. The index numbers in the legend indicate the monitoring station and signature hole number—for example,

"1_3" refers to Station 1, Signature Hole 3.

Troughs in the plotted curves represent delay times where destructive interference is maximized, resulting in lower theoretical PPVs. Conversely, peaks indicate delay times where constructive interference results in higher PPVs. The vertical dotted lines on the plots indicate delay times where the waveforms align, representing points of constructive interference that would theoretically

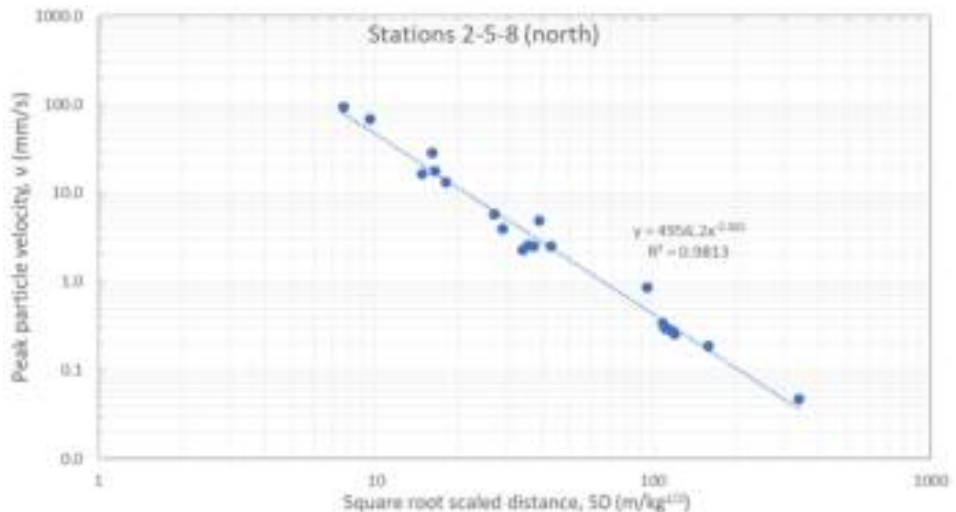


Figure 6. Peak particle velocity versus square root scaled distance for stations 2-5-8 (north).

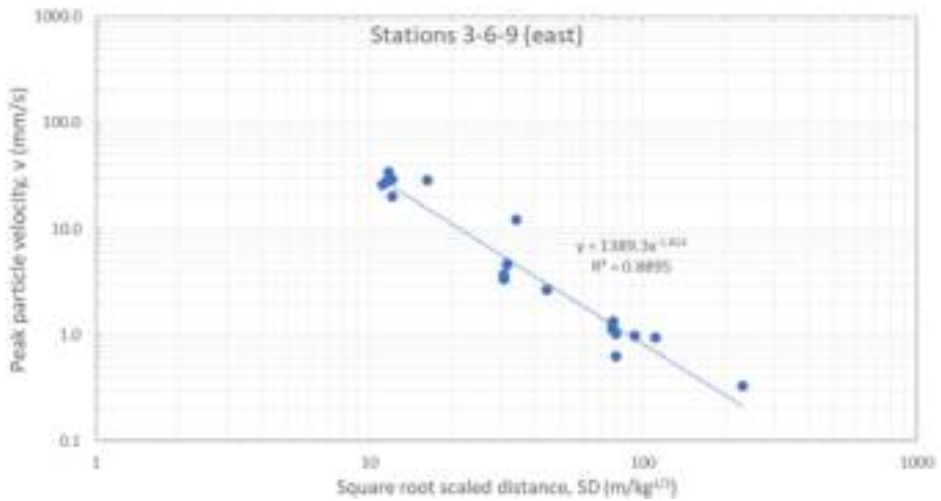


Figure 7. Peak particle velocity versus square root scaled distance for stations 3-6-9 (east).

result in higher PPVs. In contrast, the red ovals highlight time intervals where destructive interference is maximized, suggesting a potential for lower PPVs at those delays.

6 DISCUSSION

The analysis of theoretical PPVs across the nine seismograph stations reveals key insights into waveform correlatability and the broader

implications for timing optimisation. It is noted that a high correlatability was observed at Stations 3, 5, 6, 7, 8, and 9. These stations consistently show clear constructive and destructive interference patterns across multiple signature hole waveforms. The strong alignment of these patterns, despite variations in charge quantity and confinement, suggests that optimal delay times are governed more by propagation characteristics and site geometry than by blast parameters alone.

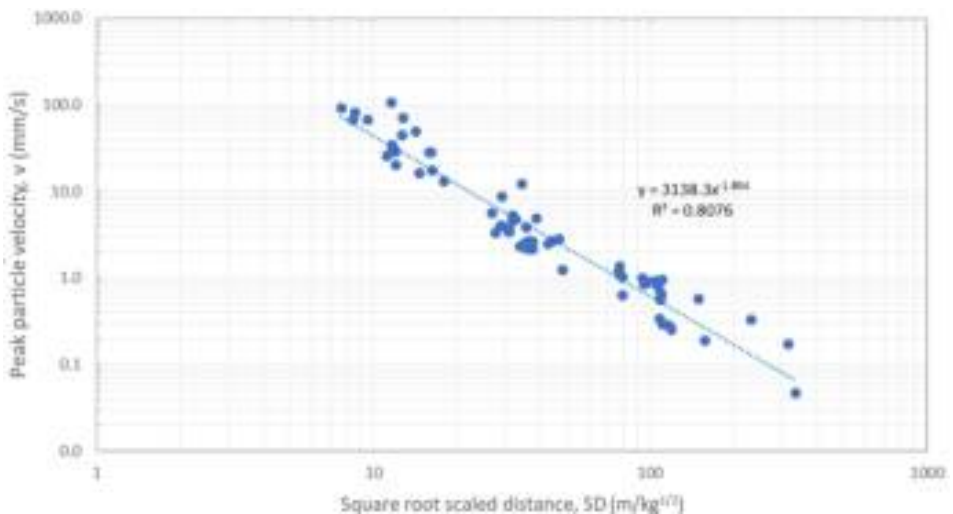


Figure 8. Peak particle velocity versus square root scaled distance for all stations.

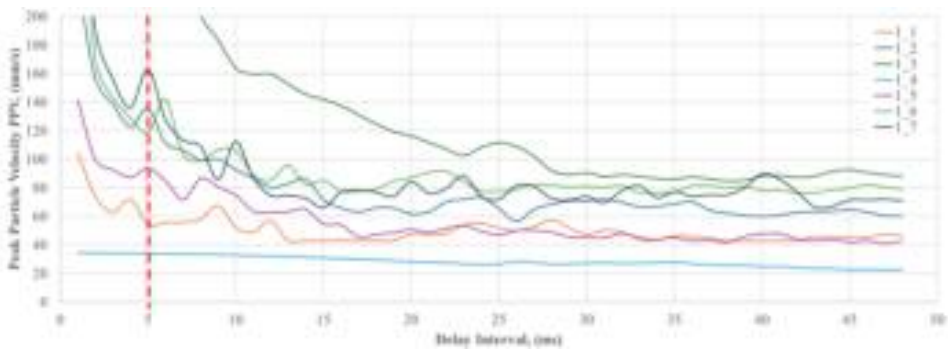


Figure 9. Waveform superposition analysis station 1.

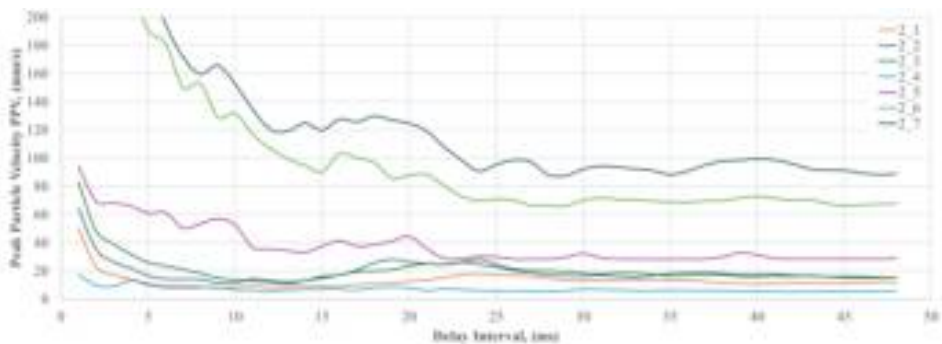


Figure 10. Waveform superposition analysis station 2.

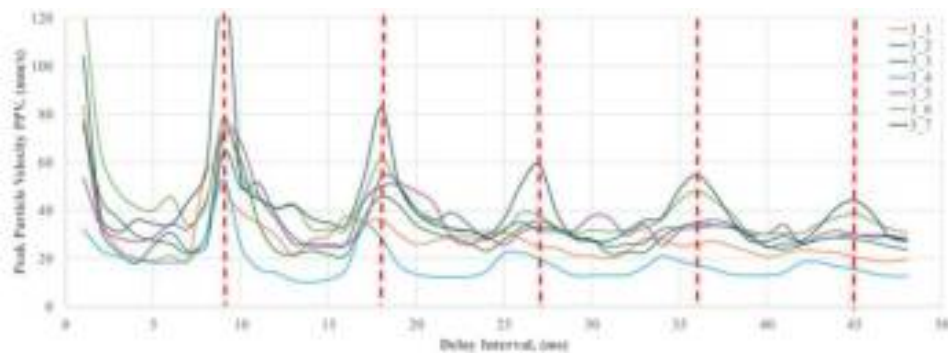


Figure 11. Waveform superposition analysis station 3.

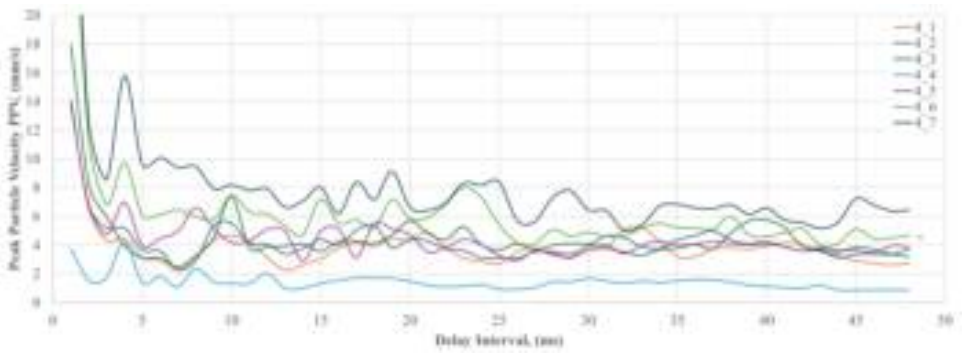


Figure 12. Waveform superposition analysis station 4.

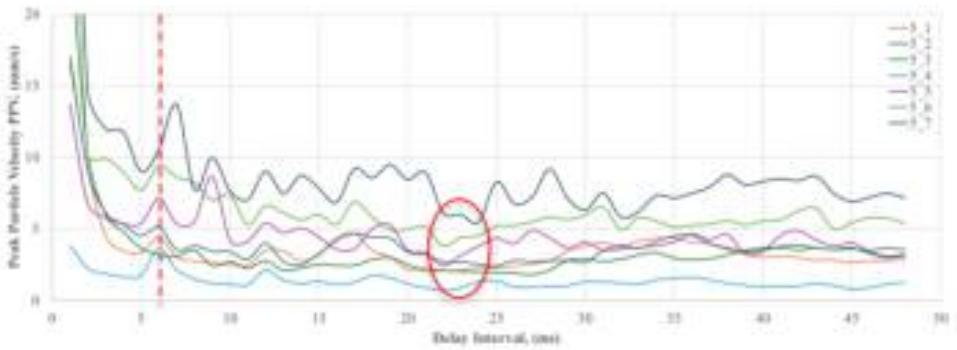


Figure 13. Waveform superposition analysis station 5.

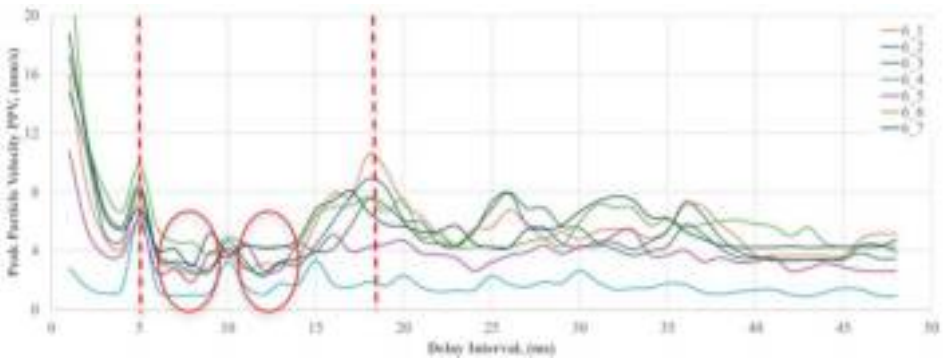


Figure 14. Waveform superposition analysis station 6.

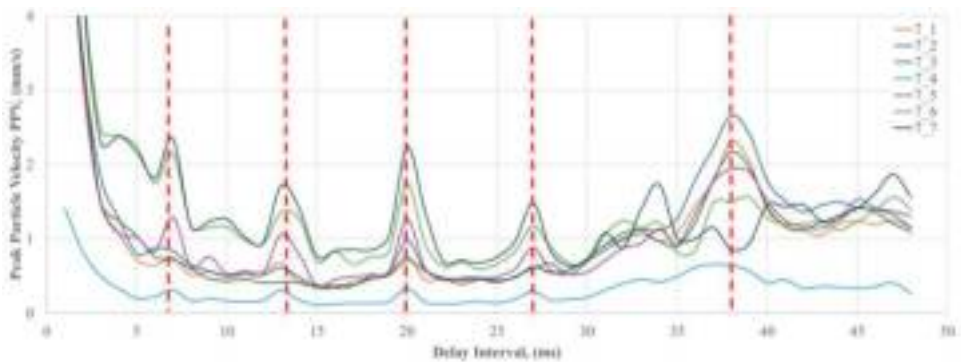


Figure 15. Waveform superposition analysis station 7.

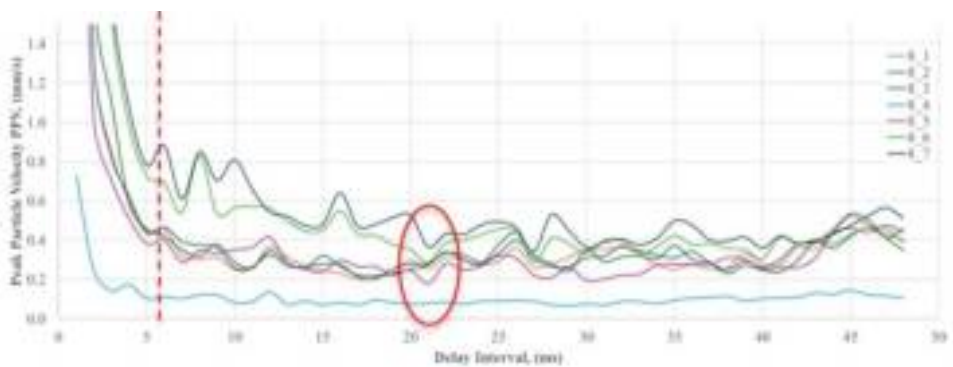


Figure 16. Waveform superposition analysis station 8.

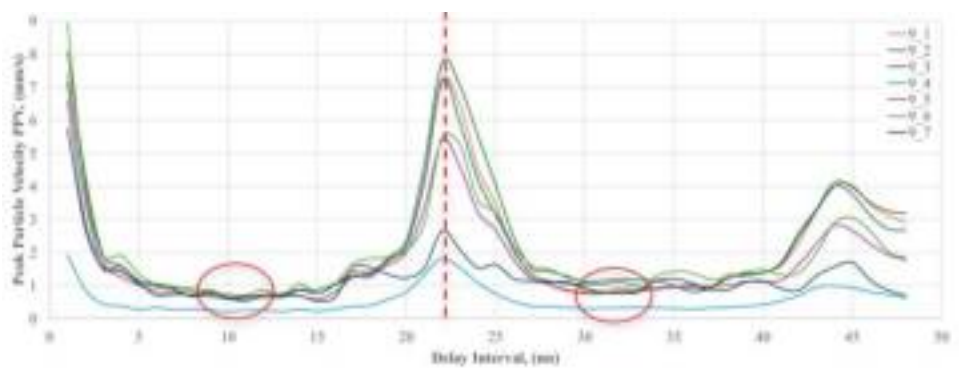


Figure 17. Waveform superposition analysis station 9.

Moderate to low correlatability is noted at Stations 1, 2, and 4. At these locations, greater variability in waveform behaviour is observed. Stations 1 and 4 are situated behind the blast zone, where waveforms appear more dispersed and less coherent due to the direct path of travel and possible scattering effects. The variability at Station 2 is linked to its mixed exposure to direct and indirect vibration propagation paths.

Timing optimisation is found to be independent of charge quantity and confinement. The consistent patterns of optimal delay times across many stations, particularly where signature holes differ in charge mass and confinement, indicate that the timing for maximising destructive interference is, relatively independent of explosive loading parameters. This highlights the robustness of the superposition method for identifying effective timing strategies, which is based primarily on wave interaction rather than blast design.

The study demonstrates that reduced charge weights under full confinement can yield high-quality seed waveforms. This allows a single hole, positioned between production rows, to serve as a signature hole without disrupting normal drill and blast activities. Additionally, waveform consistency across configurations supports flexibility in signature hole design.

Future work will include a statistical analysis to evaluate the distribution of theoretical Peak Particle Velocities (PPVs) derived from the waveform superposition calculations across the different signature hole waveforms. This analysis will quantify the degree of similarity between the waveforms by assessing their statistical variance. Measures such as standard deviation, confidence intervals, and possibly the coefficient of variation will be used to characterise the spread and consistency of the theoretical PPVs. This additional work will provide valuable insight into the reliability of the waveform alignment and help identify the magnitude of uncertainty or error in the modeling process, leading to more robust conclusions.

7 CONCLUSIONS

The following are conclusions based on this research:

- Seed waveform amplitude is sensitive to confinement and charge weight.
- Seed waveform frequencies are less sensitive to confinement and charge weight.

- Reduced, fully confined charges can generate usable seed waveforms that allow for efficient signature hole placement within active benches.
- No significant difference in waveform quality was observed when using detonating cord.

7.1 Recommendations for future work

The following recommendations are offered for future work in this area:

- Further analysis should include statistical comparisons of optimum delay times using standard error and confidence intervals to assess the variability in the optimised delay timings from the different signature hole seed-waveforms.
- Optimised delay times derived from signature hole superposition calculations should be iteratively refined, especially when accounting for other blasting design parameters.
- Signature hole analysis results derived from superposition calculations are site-specific and should be repeated in different geological settings to account for changing ground conditions.
- Document all signature holes and blast conditions for future drill and blast operations to eliminate safety and operational problems from pre-blasted rock.
- Note that ideal vibration patterns may conflict with other blast performance goals such as fragmentation, back break, toe break, highwall stability, muckpile shape, or flyrock control.
- The spatial distribution and timing sequence of blast holes to each other and monitoring sites can have a significant effect on waveform arrival times and interactions. This must be taken into consideration and re-evaluated if and when these conditions change.

REFERENCES

- International Society of Explosives Engineers 1998. *Blasters' Handbook, 17th edition*. Cleveland: International Society of Explosives Engineers.
- Stagg, M.S. & Engler, A.J. 1980. Measurements of blast-induced ground vibrations and seismograph calibration. *USBM Report of Investigations* 8506: 21–22.

A hybrid modelling approach to vibration prediction

M. Jern & M. Fagerlund

Nitro Consult AB, Sweden

ABSTRACT: This paper addresses a critical challenge in blast vibration prediction: the weak performance of Scaled Distance (SD) Models and the inability of machine learning models to extrapolate out-of-distribution. When projects move closer to measurement points, distances and/or Maximum Instantaneous Charge (MIC) may diverge from previously observed ranges, causing XGBoost (XGB) (and similar) models to perform poorly. Scaled Distance Models (SDM), on the other hand, has the ability to extrapolate. A hybrid model combining a SDM with XGB is proposed to overcome these limitations. The hybrid model uses a SDM for base predictions and XGB to model residuals. It's shown that given the out-of-distribution MICs and distances, the hybrid model (R^2 of 0.665) outperforms the Scaled Distance Model (R^2 of 0.371) and XGB (R^2 of 0.365).

1 INTRODUCTION

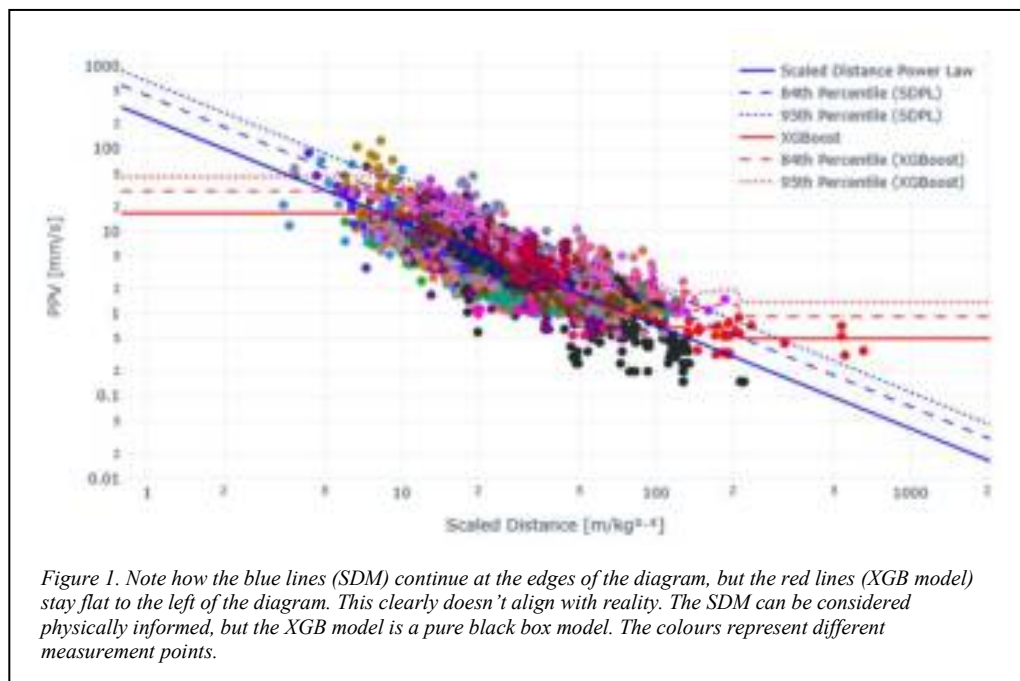
Blast vibration prediction is critical for construction and mining operations to ensure compliance with regulations and prevent damage to surrounding structures. Empirically based models like Scaled Distance Models (SDMs) are widely used for this. However, these models often deliver limited prediction accuracy as they cannot account for complex site-specific factors like geology variations, blast design differences, and local topography.

Machine learning models, such as XGB (Chen & Guestrin 2016) have demonstrated superior prediction accuracy when tested on in-distribution data. These models can capture complex, non-linear relationships and incorporate multiple features beyond just distance and charge weight, including elevation differences, soil types, and historical blast patterns. Testing shows XGB models routinely outperform SDMs on training data. XGB has previously been used for predicting vibrations from blasting, see for example Hosseini *et al.* (2023) and Feliciano *et al.* (2025).

However, machine learning models face a critical limitation: their inability to extrapolate beyond the boundaries of their training data - to out-of-distribution data. As shown in Figure 1, when a project advances and measurement points experience Scaled Distance values outside the training range, XGB predictions deteriorate dramatically. Testing reveals R^2 values plummeting from 0.8942 on training data to just 0.365 on test data with lower Scaled Distance values.

This limitation creates significant risk in real-world blasting operations, where underestimating vibrations in new project phases could lead to regulatory violations or structural damage. Conversely, relying solely on extrapolation capable but less accurate SDMs may result in unnecessarily restrictive blasting parameters, increasing project costs and timelines.

The ideal solution would combine the extrapolation capabilities of empirically based SDMs with the superior in-domain accuracy of machine learning approaches. This paper proposes such a hybrid model, leveraging the strengths of



both methods while mitigating their individual weaknesses. In Figure 1 you'll find two different models, and their blast vibration predictions. There's a clear difference in the limits of the model. Specifically, the traditional model assumes linearity outside of the known distribution of the model - meaning it assumes that vibrations will behave similarly outside of the seen data as it does inside the seen data.

The more complex (XGB) model is better at predicting values within the existing data, but when data exists outside the data the model was trained on, it assumes vibrations will behave as the existing data does at the limit. Note that the complex model in Figure 1 is kept simple to be plottable in 2D, in this example it only contains the MIC, Scaled Distance and Distance features.

This work combines these two types of models into a hybrid model that provides better predictions within the existing data but is also physically informed so that predictions outside of the existing data take the underlying physical reality into account. In Figure 2, you'll note how the hybrid model (the green line) follows the linear model when outside of the existing data but maintains its advantages within the existing data.

1.1 The test project

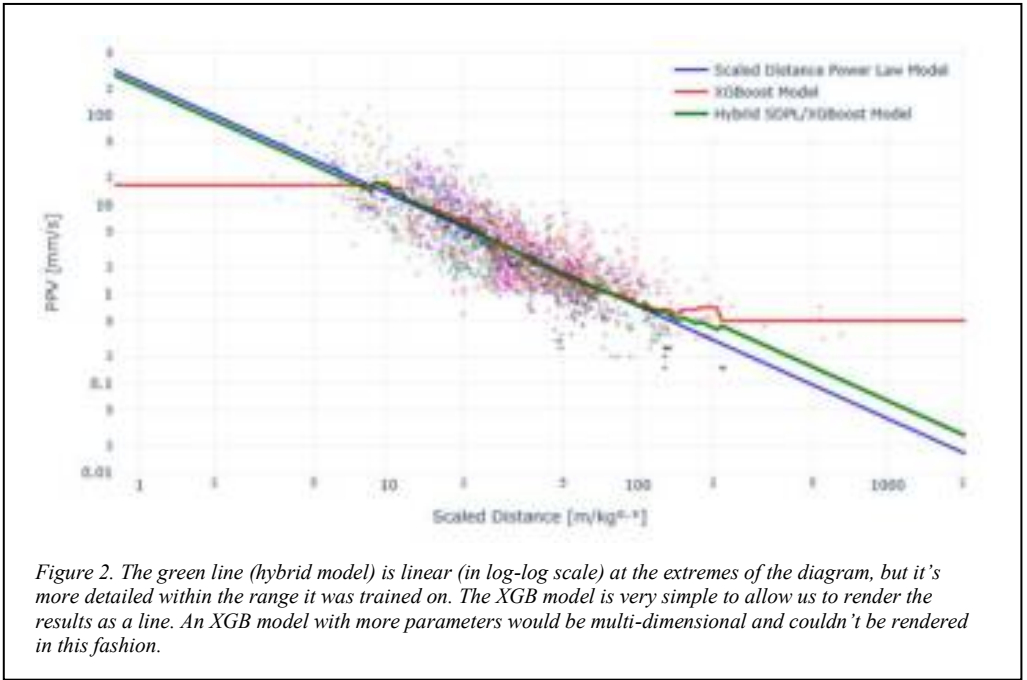
To evaluate the model, data from the City Link (CL) project was used. CL is a railway tunnel

under central Stockholm that was constructed between 2009 and 2017. A total of 9,846 blasting operations were carried out during the project, with 625 vibration measuring points of which 383 are included in this work. In total there were 991 measurement points of different types. Vibration data and blasting logs from the project are stored in Nitro Consult's web application NCVIB, which makes it possible to analyse this relatively large dataset. The vibration monitoring work carried out during the project has previously been described by Jonson (2012).

1.2 Input data

It is important to note that in an analysis of this type, we must work with the data that is available, not necessarily the data we would ideally want. The model input consists of accessible data, i.e. blasting logs from the project and geological information obtained from public sources (www.sgu.se). The quality of the raw data varies; blasting logs and position data may sometimes be inaccurate. For instance, the blasting log system allows for copying previous entries, which facilitates the work for the blaster but can occasionally result in incorrect data being entered. The geological data used in the model is based on geological maps, which are not highly detailed.

Information that likely has a significant impact, such as delay times, initiation direction, and



fracture zones, is not included. It is therefore probable that a model incorporating this type of information would yield improved results. The purpose of this study is to investigate what can be achieved using the data that is currently available.

2 METHOD

2.1 Data preparation and filtering

To ensure reliable model development, a systematic data cleaning approach was implemented. Starting with vibration measurements from six large tunnel construction projects within the CL project, the initial dataset contains 53,244 measurements.

An outlier detection approach using an initial SDM was used. Residuals between model predictions and actual measurements were computed, and the top 1% of outliers based on residual magnitude were excluded. This removed data points with exceptionally high prediction errors while retaining the vast majority of valid measurements. These are assumed to be faulty measurements rather than limitations of the model though both are possible. Faulty measurements can have many causes, for instance incorrect measurement position, incorrect blast position, incorrect MIC, incorrect timing, overlaps between different blasts or sensor issues.

The data used in this paper isn't optimal, there are quality issues, that is however something that can be expected and must be dealt with in all projects. We work with the data we have, not always the data we want. After removing 533 measurements, 52,711 measurements are left in the dataset. The initial SDM improves from R^2 of 0.577 before cleaning to 0.621 after cleaning.

2.2 Training-test split to simulate domain shift

To effectively evaluate the models out-of-distribution capabilities, a specialised training-test split was created that simulates what happens when blasting operations move closer to measurement points which is a common scenario that would produce Scale Distance (SD) values outside the original training range. Rather than using random sampling, which is typical, the dataset is sorted by SD and the bottom 5% of SD values were used exclusively for testing. The remaining 95% with higher SD values was used for training. The train set contains SD values in the range 1.207 to 3.314 and the test set contains SD values in the range 0.088 to 1.207.

This approach created a challenging evaluation scenario where the test set contained values in ranges never seen during training, precisely mimicking the real-world extrapolation problem. This significant domain shift is not uncommon when blasting operations progress closer to

measurement points or blasting charges increase, where higher vibration values might be expected but are often underestimated by machine learning models.

After being filtered as described above, the dataset contains 50,076 training measurements and 2,635 testing measurements.

2.3 Model development

Three distinct modelling approaches were implemented:

- SDM model: A collection of physics informed SDMs were created. The R^2 on training data was 0.607 and 0.371 on test data. See below.
- XGB model: An XGB model using multiple features beyond just distance and charge weight was created – see below. This gives us a pure black box prediction (not physically informed). This model is only used for validation. The R^2 on training data was 0.894 and the test was an R^2 of 0.365. The stand-alone XGB model performs worse than the stand alone SDMs. The fall-off aligns well with the claim that XGB doesn't extrapolate well in this situation.
- Hybrid model: A proposed hybrid model that combines the two previous models was created. It uses SDMs as a base prediction. A new XGB model is then trained to predict the residual (difference between SDM prediction and actual vibration). This dual-model approach provides physical extrapolation capabilities while maintaining the superior pattern recognition of machine learning within the training domain. These two models are combined by adding them in log space. The R^2 on training data was 0.931. On the test data, the R^2 was 0.665, better than either of the part models.

2.4 Prediction of blast-induced vibrations using SDMs

Prediction of peak particle velocity (PPV) resulting from blasting activities is commonly performed using only two parameters: distance from the blast source (R) and maximum instantaneous charge (Q). These three variables can be expressed in a general power-law function as follows:

$$PPV = A \cdot R^a \cdot Q^b \quad (1)$$

This equation is often simplified to the so-called scaled distance laws for practical and computational reasons, as well as due to their empirical reliability. Two commonly used forms are:

- The square root scaled distance (SRSD) model (Siskind *et al.* 1980):

$$PPV = A \left(\frac{R}{\sqrt{Q}} \right)^{-n} \quad (2)$$

- The cube root scaled distance (CRSD) model (Ambraseys & Hendron 1968):

$$PPV = A \left(\frac{R}{\sqrt[3]{Q}} \right)^{-n} \quad (3)$$

These simplifications are widely adopted because they reduce the number of variables, making the models computationally efficient while maintaining a good empirical fit to observed data.

In this paper we have used the term SDM (scaled distance model) and SD (scaled distance). These terms refer to the square root scaled distance model respectively the square root scaled distance.

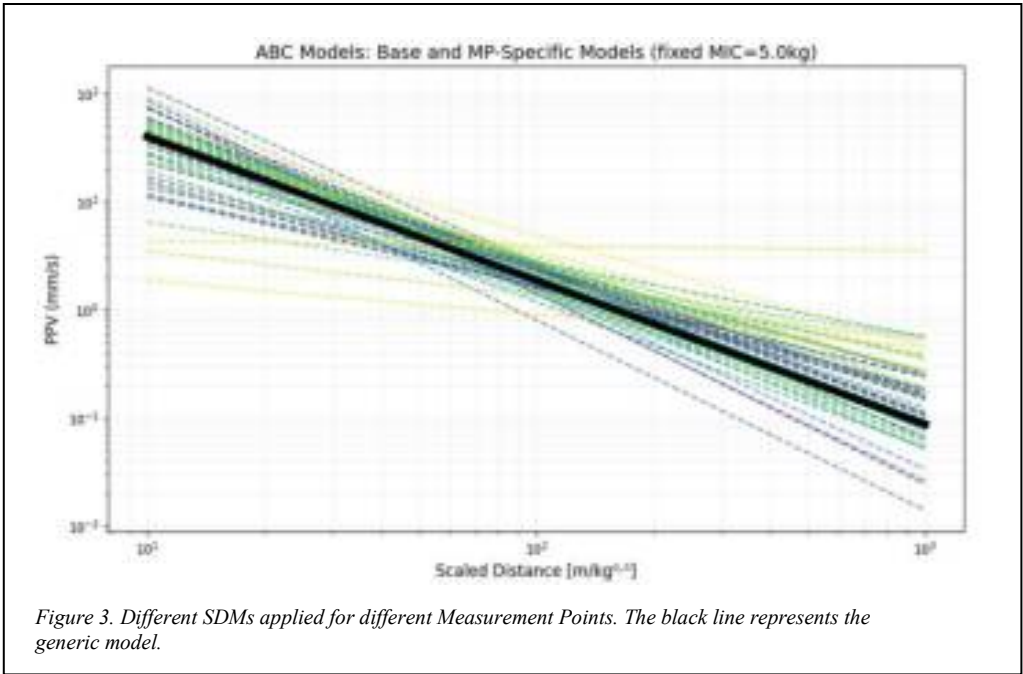
2.4.1 Measurement point specific SDMs

Measurement point-specific SDMs were created to capture location-specific characteristics that a generalised model might miss. This approach was implemented selectively, based on strict criteria to ensure improved performance without overfitting.

Specifically, Measurement Point (MP)-specific models were developed only when two conditions were met: (1) the MP had at least 40 data points in the training set, and (2) the MP-specific model demonstrated a minimum 0.5% improvement in R^2 over the general SDM model during cross-validation.

Further analysis identified 226 eligible measurement points with sufficient data for potential specialised modelling. After rigorous evaluation, 52 of these points qualified for MP-specific models, showing significant predictive improvements. A new SDM model was trained for the remaining measurement points - excluding the ones that were given their own models.

The MP-specific models improved the training performance R^2 from 0.568 to 0.607 on training data and from 0.205 to 0.371 on test data, compared to the general model. This approach allowed us to handle 26.4% of the training data (13,226 records) with specialised models, while the remaining 73.6% (36,850 records) utilised the general model.



The integration of multiple specialised models created a foundation that could better account for geological variations, historical blast response patterns, and measurement point installation characteristics, all factors that contribute to the complex nature of blast vibration propagation.

2.5 XGB model feature importance

Table 1 presents the features used in our XGB regression model for blast vibration prediction, ranked by their relative importance. XGB is an optimised distributed gradient boosting algorithm designed to be highly efficient, flexible, and portable.

The feature importance percentage represents the relative contribution of each variable to the model's predictions. These percentages loosely quantify how much each feature improves the performance metric when used in the model's decision trees. Higher percentages indicate features with greater predictive power.

Notably, Scaled Distance seems to account for approximately 40% of the model's predictive power. However, if that parameter is removed, the model can still recreate the performance using other parameters (MIC and Distance) and reach almost the same result.

Several of the parameters have no obvious physical explanation of why they contribute positively to the model, especially for the

parameters that have a relatively low influence. However, the idea for this test has been to use all accessible data and to use the parameters that are beneficial for the model.

2.6 Hybrid model design – combining models

The hybrid model developed in this study combines an empirically based Scaled Distance Model (SDM) with a machine learning XGB model through a residual learning approach.

A two-stage prediction pipeline is implemented:

- Base Prediction: The SDM model provides initial vibration predictions following established physical principles. This empirically based model gives our predictions a solid foundation that aligns with known vibrational propagation laws.
- Residual Prediction: The XGB model is trained not to predict vibration values directly, but rather to predict the residuals (errors) between the SDM predictions and actual measurements. This allows the machine learning component to learn complex patterns that the simplistic physical model cannot capture. It basically learns to fix the SDM errors.
- Combined Prediction: The final prediction is generated by adding the machine learning residual prediction to the base SDM

Table 1. XGB Features and their importances.

Rank	Feature	%	Description
1	Scaled Distance	39.93	Distance / MIC ^{1/2}
2	Ground Distance	20.71	Ground (horizontal) distance between blast and MP
3	_MP	10.44	Measurement point identifier
4	MIC	5.35	Maximum Instantaneous Charge (kg)
5	Distance	4.09	Direct distance between blast and measurement point (m)
6	MP Soil Depth	3.77	Soil depth at measurement point (m)
7	MP Elevation Orig	2.00	Original elevation of measurement point (m)
8	Charge Total	1.56	Total charge in the blast (kg)
9	Frequency	1.48	Average dominant frequency of last 5 blasts on the particular Measurement Point (Hz)
10	MIC * Hole Count	1.48	Product of MIC and number of blast holes. This is often similar to Total Charge, but sometimes Total Charge is missing.
11	Hole Count	1.45	Number of holes in the blast
12	Blast Elevation	1.43	Elevation of the blast point (m)
13	Delta E	1.42	East coordinate difference between blast and measurement point
14	Blast Elevation Orig	1.20	Original elevation of blast point (m)
15	Measurement Point Blast Count	0.98	Count of blasts recorded at this measurement point
16	Delta N	0.91	North coordinate difference between blast and measurement point
17	Fired Month	0.86	Month when the blast was fired (relevant due to different water levels and freezing)
18	Normal E	0.49	Normalised east direction component
19	Normal N	0.47	Normalised north direction component

prediction. All calculations are performed in logarithmic space to handle the wide range of vibration values effectively:

$$PPV_{\text{hybrid}} = 10^{\log_{10}(PPV_{\text{SDM}}) + XGBoostResidualCorrection} \quad (4)$$

2.7 Confidence levels

Almost all models presented in this paper use a confidence level of 50% (the mean predicted value). A confidence level of 50% means that there is an equal probability that the actual blast vibration will be higher or lower than the model's prediction. In practical applications, it's typical to work with higher confidence levels, such as 84% or 95% depending on circumstances, to provide more conservative estimates for safety and regulatory compliance.

Both the Scaled Distance model, the XGB model, and the hybrid model can be modified to predict using higher confidence levels, though the focus of this paper is on mean predictions to simplify the comparative analysis. R² values are

primarily meaningful for evaluating mean predictions, as they become difficult to interpret when comparing actual vibrations against predictions with an applied confidence level adjustment.

In figures 1 and 2, where the limitations of extrapolation for XGB models is shown, 84% and 95% confidence levels are included to demonstrate that they behave as expected.

2.8 Theoretical advantages

This hybrid approach was designed specifically to address the extrapolation problem:

- When predicting vibrations for scenarios within the training domain, the residual prediction from XGB refines the SDM prediction, capturing complex environmental and operational factors.
- When extrapolating to new scenarios (such as those with low Scaled Distance values), the SDM component provides physically plausible predictions following known attenuation laws, while the machine learning

contribution naturally diminishes in areas outside its training domain.

3 RESULTS

Our study demonstrates significant improvements in blast vibration prediction through a hybrid modelling approach. This method addresses a critical industry challenge: predicting vibrations accurately when blasting operations move closer to measurement points, where traditional models often fail.

3.1 Model performance comparison (R² VALUES)

Table 2. Results for the different models in training and in tests.

Model	Train R ²	Test R ²
SD Model	0.607	0.371
XGB Model	0.894	0.365
Hybrid Model	0.931	0.665

This implementation demonstrated a significant accuracy advantage when predicting vibrations at low Scaled Distance values (close proximity/high charge scenarios).

3.2 Robustness check – larger hold-out set (15 % test)

To test how sensitive the models are to the size of the unseen data, the train/test split was repeated with a 15% hold-out (≈ 7 900 blast records). All three algorithms lose some predictive power, as expected when the evaluation set becomes harder, but the hybrid model still outperforms the standalone approaches by a wide margin.

Table 3. Results for the different models on larger hold-out set. Test-set R² remains more than three times higher than the best single-model baseline (SDM).

Model	Train R ²	Test R ²
SD Model	0.5858	0.1784
XGB Model	0.8796	0.0984
Hybrid Model	0.9233	0.5360

3.3 Interpretation.

- SDM and XGB degrade sharply—especially XGB, whose test R² falls below 0.10—confirming that these single-model strategies generalise poorly when the hold-out covers a broader range of blasting scenarios.
- This robustness check underscores the Hybrid model’s consistent superiority across different validation schemes, bolstering confidence in its deployment for blast-vibration forecasting.

4 CONCLUSIONS

The hybrid approach offers several practical advantages for blast engineers:

- Safer predictions in critical scenarios: When operations move into new phases with lower Scaled Distance values (closer to structures), the hybrid model maintains reliable predictions where pure machine learning models fail.
- Measurement point specialisation: Our implementation includes measurement point-specific SDM models where sufficient data exists, further improving accuracy.
- Explainable results: Unlike black-box machine learning approaches, the hybrid model maintains a connection to physical principles, making results more transparent and defensible.
- Operational flexibility: The model can be continuously updated as new blast data becomes available while maintaining extrapolation capabilities.
- Overconfidence in advanced models is likely and risky: XGB models look deceptively good for known data (in-distribution). SD models look more modest. Once a project moves out-of-distribution, XGB performance decreases significantly. Overconfidence in models is risky.

The study highlights the significant potential of using AI-based tools for vibration prediction; however, what’s presented here is by no means a fully developed or finalised tool.

Further studies are needed to determine which parameters should or could be integrated into the model, as well as which ones should be excluded. Additional parallel tests are also required. Nevertheless, the ability to process large volumes of data and identify correlations relatively easily makes this type of tool highly useful for addressing problems of this nature.

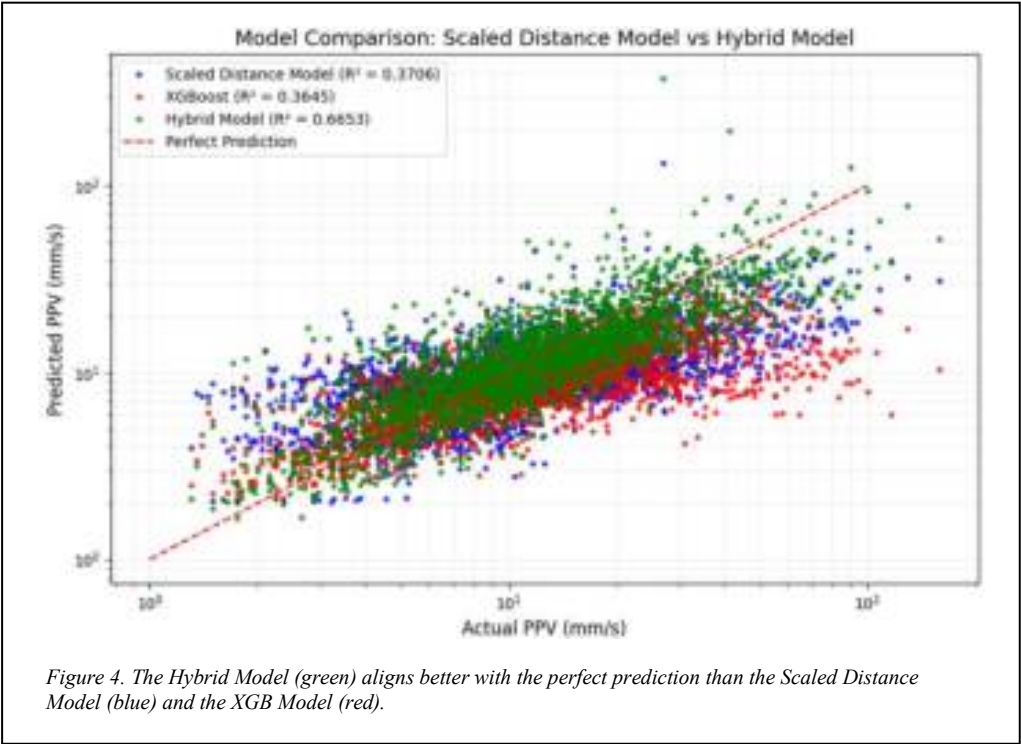


Figure 4. The Hybrid Model (green) aligns better with the perfect prediction than the Scaled Distance Model (blue) and the XGB Model (red).

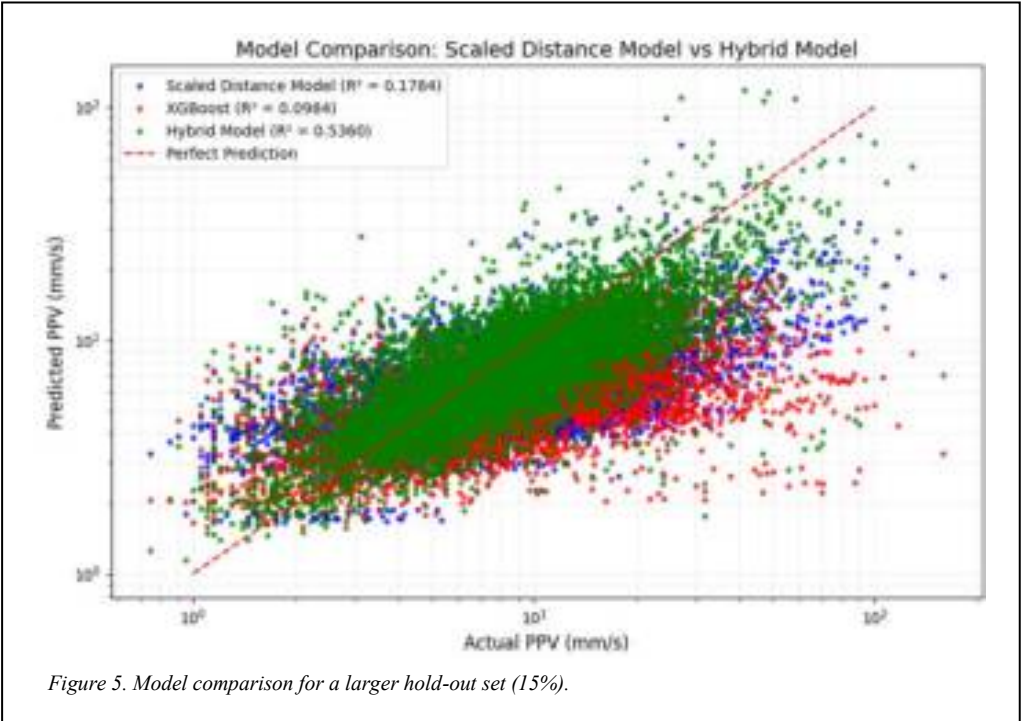


Figure 5. Model comparison for a larger hold-out set (15%).

5 ACKNOWLEDGMENT

The authors would like to thank the Swedish Transport Administration for letting us use the data from the City Link project.

REFERENCES

- Ambraseys, N.N. & Hendron, A.J. 1968. Dynamic behaviour of rock masses. *John Wiley & Sons*.
- Chen, T. & Guestrin, C. 2016. XGB: A scalable tree boosting system. In *Proceedings of the 22nd ACM SIGKDD International Conference on Knowledge Discovery and Data Mining* (pp. 785–794). New York, NY, USA: ACM. <https://doi.org/10.1145/2939672.2939785>
- Hosseini, S., Pourmirzace, R., Armaghani D.J. & Sabr-Sabri, M. M. 2023. Prediction of ground vibration due to mine blasting in a surface lead-zinc mine using machine learning ensemble techniques. *Sci Rep.* 2023 Apr 21;13(1):6591.
- Jonson, D. 2012. Controlling shock waves and vibration during large and intensive blasting operations under Stockholm City, *Fragblast 10* (New Delhi, India, 26-29 November 2012).
- Siskind, D.E., Stagg, M.S., Kopp, J.W., & Dowding, C.H. 1980. Structure response and damage produced by ground vibration from surface mine blasting. *Report of investigation 8507 US Bureau of Mines*, 74 pp.
- Soares Feliciano, C. E., da Cunha Meireles, A. & Garcia, P. 2025. Conformal symbolic regression applied to rock blasting vibration prediction. *Proceedings of the 51st Annual Conference on Explosives and Blasting Technique Cherokee North Carolina USA*.

RETURN TO CONTENTS PAGE

Perforating drill holes as additional method of reducing ground vibrations level

A. Brzozowski

Holcim Polska, Dubie, malopolska, Poland

ABSTRACT: Dubie dolomite quarry is located between two villages. The main focus of blasting engineers is to reduce negative seismic influence on nearby housing. After many trials with changing direction of excavation, implementation of electronic detonators, differentiating place of initiation in holes, trials with many different maximum instantaneous charges we found our optimal solution. After finding resolutions for each of the above aspects we tried to find additional methods of reducing ground vibration level. Dubie quarry has been measuring ground vibration level permanently (24/7) since 2015, we also cooperate closely with local authorities to be as preventive as possible. Having that in mind we'd like to achieve additional reduction we came up with idea to interrupt progression of vibration through rock deposit. The first trial was in 2023 with a group of holes drilled on the edge of the quarry from the village side. The results after their completion will be shown in this paper.

1 INTRODUCTION

Dubie dolomite quarry is located in the southern part of Poland, less than 30 km from Krakow city. It started excavation as State owned company in 1968. The quarry was located in the middle of the forest with no dwellings nearby. It was very small at the beginning with only 20,000 tonnes of excavation per year. During it's slow (but stable) growth Dubie reached, in 1976, excavation level at around 350,000 tonnes which was the annual standard for next twenty years. With the quarries growth the local area started to be more attractive as a place with good infrastructure (power supply, telephone, asphalt road). Taking it into consideration it's understandable that local villages were coming closer and closer to the borders of the quarry area. In 1970s and 1980s, housing nearby was no problem for the owner of the Dubie quarry because there was no legislation

for local communities to interfere with excavation. There was no measurement of ground vibrations or shockwaves - in general the quarry was only focused on fly rock reduction. In 21st century the situation become more tense due to houses being located less than half a kilometre from the quarry boundary. Since 2008 annual excavation level is above 1 million tonnes (1.4 mln in 2024).

Dubie quarry uses no subcontractors in the drilling and blasting process. Holcim Poland in KOSD region (four quarries: Dubie, Klikuszowa, Porąbka, Tenczyn) use an in-house drilling and blasting team. Dubie blasting team is mainly focused on development and optimisation of the process. As shown on the Figure 1 Dubie is located in-between two villages and near a big limestone quarry (2.5 mln tonnes). All manipulated images in this paper are generated in Stravos platform.



Figure 1. Dubie quarry location (Dubie is on the right).

2 GROUND VIBRATION MEASURING AND REDUCTION PROCESS

The main focus of blasting engineers is to reduce negative seismic influence on nearby housing.

Unfortunately, a first-class dolomite deposit is located on south-west part of quarry which is in the direction of Siedlec village. Excavation in 2024, as shown in Figure 2, is focused mainly in this crucial part.

In 2013 blasting engineers were using 3D laser profiler for every blast.

This changed in 2023 when Dubie drilling and blasting team switched to photogrammetry-based drone systems.

After many trials with changing direction of excavation, implementation of electronic detonators, differentiating place of initiation in holes, trials with many different maximum instantaneous charges we found the optimal solution. After finding resolution for each of the above aspects we tried to find additional methods of reducing ground vibration level.

Dubie quarry has been measuring ground

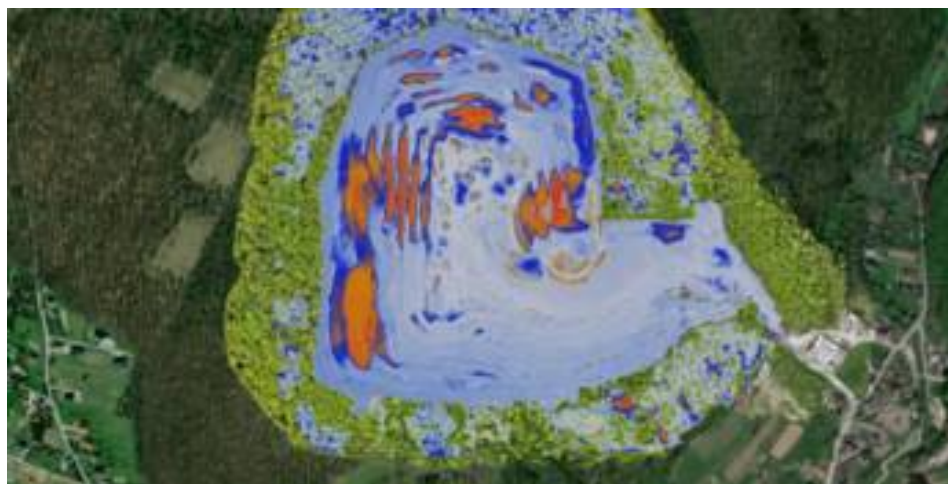


Figure 2. Places of excavation (orange and dark blue) in 2024.



Figure 3. Location of ground vibration monitors through years.

vibration level permanently (24/7) since 2015; measuring it in thirteen different locations.

Ground vibration monitoring is provided by AGH University of Krakow. We've decided to cooperate with the University to be as transparent as we can. Dubie quarry's team also cooperate closely with local authorities to be as preventive as possible.

Nevertheless, quarry management is aware that from the perspective of local communities the best result on ground vibration monitoring is zero.

With that in mind, we'd like to achieve additional reduction in vibration and came up with idea of interrupting progression of vibration through rock deposit.

3 PERFORATING DRILL HOLES – IDEA

In the second half of 2023, Dubie quarry expanded in the south-western area. It is the first time in the history of quarry we have been so close to local housing, and we wanted to reduce ground vibrations levels even more. We were recording results of around 2 mm/s, which for us was not a desired level. From our experience if the level exceeds 1.5 mm/s we receive a lot of phone call complaints from the local community. As mentioned in Section 2 we've reached our limit of reduction using blast optimisation methods, so we came up with idea that maybe when we interrupt the uniformity of the dolomite deposit we could reduce the propagation of ground vibration.

To achieve that we choose to use drill holes as a suppressor. We thought that when we drill proper length holes with narrow spacing the ground vibration propagation process will be interrupted by them. The idea was to create a barrier that will be 50 metres long which, in our opinion, should be sufficient.

4 PERFORATING DRILL HOLES – TRIALS AND RESULTS

4.1 First series of perforating trials

In December 2023 we started to drill the first series of holes. The idea was to have the holes vertical of a length that exceeded the face height. Holes were located on the southern part of deposit on bench VII.

That location was chosen to reduce the vibration level on the southern part of quarry. Everything is shown in Figures 4 to 6.

After making these holes we started the blasting process in the southern deposit area. All the blasts taken into consideration were from bench VII to VI. We took six months of observations and compared before and after on the blasts that were the most similar. In comparison, (see Figure 7) the results were very satisfying. We did not predict that level of reduction (over 30%) will be that high. Both quarry management and drilling and blasting team were very pleased with the results so we prepared to conduct a second trial



Figure 4. Location of first series of holes (orange rectangle).

with perforating drill holes. This time we were focusing on south-west part of the deposit. The holes should cover a much bigger area so the

length of this series was almost 100 metres and, as before, it was one row. Again, the length exceeds the face height. All holes were vertical and spacing



Figure 5. Front view of first series of holes.

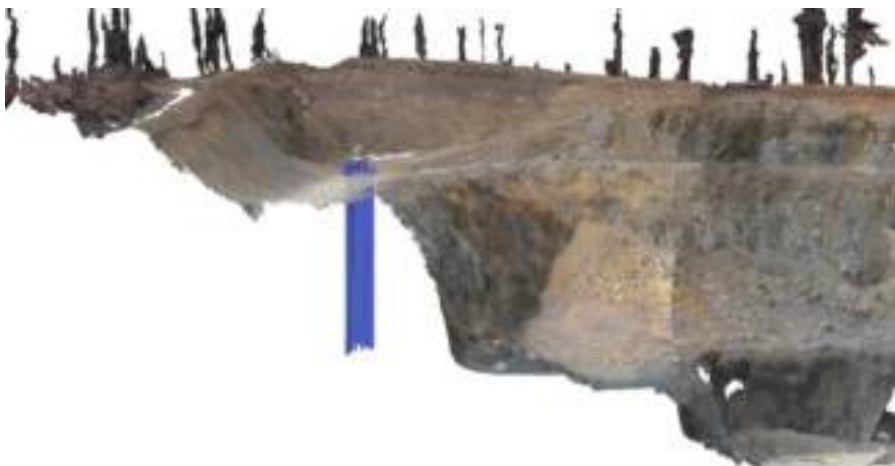


Figure 6. Side view of first series of holes.



Year	1990	1991	1992	1993	1994
1990	1,000	1,000	1,000	1,000	1,000
1991	1,000	1,000	1,000	1,000	1,000
1992	1,000	1,000	1,000	1,000	1,000
1993	1,000	1,000	1,000	1,000	1,000
1994	1,000	1,000	1,000	1,000	1,000



Figure 7. PPV results of first series of blasts. Red indicates 'before' and green indicates 'after' perforations.



Figure 8. Location for first pair of before (red) and after (green) blasts.



Figure 9. Location of second pair of before (red) and after (green) blasts.

was narrow. Location of the second series is shown in Figure 8.

4.2 First trial, full PPV results

See Figure 7.

4.3 First trial comparison of four pairs of blasts.

See Table 1 to 4 and Figures 8 to 12.

Table 1. Comparison details for first pair of blasts.

Comparison No. 1	Before (Red)	After (Green)
Blast number	D123/2023	D05/2024
Number of holes	22 pcs	18 pcs
Number of rows	3	3
Average hole length	12.3 m	11.5 m
Number of decks	2	1
Type of explosives used:		
ANFO	-	-
Cartridged emulsion	850.0 kg	775.0 kg
TNT / dynamite	22.0 kg	9.0 kg
Point of initiation	top deck	bottom
MIC	19.8 kg	43.6 kg
Total charge	872.0 kg	764.0 kg
Av. burden	3.0 m	3.0 m
Av. spacing	3.2 m	3.5 m
Distance to perforating holes	0 m	48 m
Distance to vibration monitor	502 m	498 m
PPV	1.82 mm/s	1.00 mm/s

Table 2. Details about second comparison of blasts.

Comparison No. 2	Before (Red)	After (Green)
Blast number	D197/2023	D80/2024
Number of holes	21 pcs	10 pcs
Number of rows	3	1
Average hole length	12.0 m	19.6 m
Number of decks	1	3
Type of explosives used:		
ANFO	-	500.0 kg
Cartridged emulsion	825.0 kg	150.0 kg
TNT / dynamite	10.5 kg	13.98 kg
Point of initiation	bottom	top deck
MIC	39.8 kg	22.1 kg
Total charge	835 kg	664.0 kg
Av. burden	3.1 m	3.1 m
Av. spacing	3.2 m	3.3 m
Distance to perforating holes	43 m	19 m
Distance to vibration monitor	505 m	535 m
PPV	1.05 mm/s	0.75 mm/s



Figure 10. Location of third pair of before (red) and after (green) blasts.

Table 3. Details about third comparison of blasts.

Comparison No. 3	Before (Red)	After (Green)
Blast number	D129/2023	D38/2024
Number of holes	28 pcs	14 pcs
Number of rows	3	1
Average hole length	11.5 m	18.4 m
Number of decks	1	2
Type of explosives used:		top deck
ANFO	-	-
Cartridged emulsion	975.0 kg	900.0 kg
TNT / dynamite	14.0 kg	14.0 kg
Point of initiation		
MIC	35.3 kg	32.6 kg
Total charge	989.0 kg	914.0 kg
Av. burden	3.1 m	3.0 m
Av. spacing	3.4 m	3.7 m
Distance to perforating holes	62 m	23 m
Distance to vibration monitor	506 m	539 m
PPV	1.42 mm/s	0.76 mm/s

Table 4. Details about fourth comparison of blasts.

Comparison No. 4	Before (Red)	After Green)
Blast number	D02/2024	D23/2024
Number of holes	9 pcs	12 pcs
Number of rows	1	2
Average hole length	19.4 m	18.9 m
Number of decks	2	2
Type of explosives used:		
ANFO	-	-
Cartridged emulsion	600.0 kg	775.0 kg
TNT / dynamite	9.0 kg	12.0 kg
Point of initiation	top deck	top deck
MIC	33.8 kg	32.8 kg
Total charge	609.0 kg	787.0 kg
Av. burden	3.1 m	3.1 m
Av. spacing	3.7 m	3.7 m
Distance to perforating holes	39 m	17 m
Distance to vibration monitor	554 m	552 m
PPV	1.17 mm/s	0.88 mm/s



Figure 11. Location of fourth pair of before (red) and after (green) blasts.



Figure 12. Location of second series of holes (purple rectangle).

4.4 Second series of perforating hole trials

After the second series was complete we started to gather the results. This time we were focused on a three-month period and again we'd like to find the most similar blast and directly compare them. The

results after the second series of perforating drill holes are shown in Figure 13.

As seen in the 1st series, again we came up with a big reduction of ground vibration levels. This time the average level of reduction was around 40%.

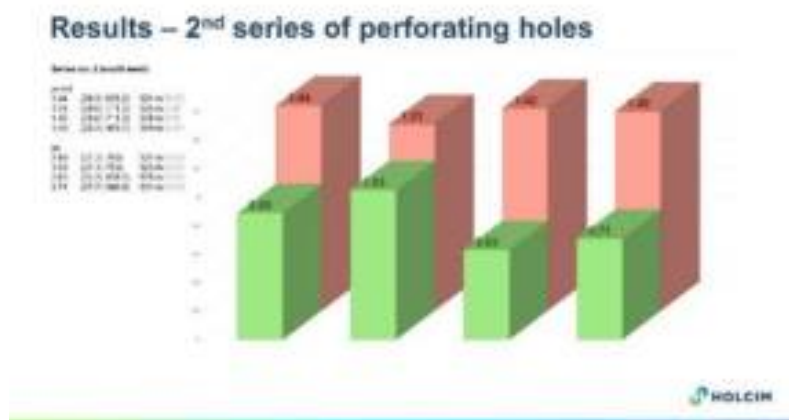


Figure 13. PPV results of second series of blasts. Red indicates 'before' and green indicates 'after' perforations.



Figure 14. Location of fifth blast comparison according to series of perforating holes.

Table 5. Details about fifth comparison of blasts.

Comparison No. 5	Before (Red)	After (Green)
Blast number	D142/2024	D149/2024
Number of holes	11 pcs	14 pcs
Number of rows	2	2
Average hole length	15.9 m	16.5 m
Number of decks	2	2
Type of explosives used:		
ANFO	-	-
Cartridged emulsion	625.0 kg	750.0 kg
TNT / dynamite	10.23 kg	13.02 kg
Point of initiation	top deck	top deck
MIC	28.9 kg	27.3 kg
Total charge	635.2 kg	763.0 kg
Av. burden	3.0 m	3.1 m
Av. spacing	3.7 m	3.7 m
Distance to perforating holes	19 m	15 m
Distance to vibration monitor	521 m	521 m
PPV	1.44 mm/s	0.89 mm/s

4.5 Second trial with perforating holes - comparison of four pairs of blasts.

See Table 5 to 8 and Figure 14 to 17.

Table 6. Details about sixth comparison of blasts.

Comparison No. 6	Before	After
Blast number	D114/2024	D158/2024
Number of holes	12 pcs	14 pcs
Number of rows	3	2
Average hole length	16.7 m	15.9 m
Number of decks	2	2
Type of explosives used:		
ANFO	-	-
Cartridged emulsion	700.0 kg	750.0 kg
TNT / dynamite	11.184 kg	13.02 kg
Point of initiation	top deck	top deck
MIC	29.6 kg	27.3 kg
Total charge	711.2 kg	763.0 kg
Av. burden	3.0 m	3.0 m
Av. spacing	3.6 m	3.7 m
Distance to perforating holes	23 m	15 m
Distance to vibration moni- tor	525 m	523 m
PPV	1.31 mm/s	1.05 mm/s



Figure 15. Location of sixth blast comparison according to series of perforating holes.



Figure 16. Location of seventh blast comparison according to series of perforating holes.



Figure 17. Location of eighth blast comparison according to series of perforating holes.

Table 7. Details about seventh comparison of blasts.

Comparison No. 7	Before	After
Blast number	D88/2024	D152/2024
Number of holes	12 pcs	12 pcs
Number of rows	2	1
Average hole length	16.7 m	19.1 m
Number of decks	2	2
Type of explosives used:		
ANFO	400.0 kg	-
Cartridged emulsion	300.0 kg	825.0 kg
TNT / dynamite	11.184 kg	11.160 kg
Point of initiation	top deck	top deck
MIC	29.6 kg	34.8 kg
Total charge	711.2 kg	836.2 kg
Av. burden	3.0 m	3.1 m
Av. spacing	3.5 m	3.5 m
Distance to perforating holes	44 m	31 m
Distance to vibration monitor	558 m	528 m
PPV	1.42 mm/s	0.63 mm/s

Table 8. Details about eighth comparison of blasts.

Comparison No. 8	Before	After
Blast number	D125/2024	D146/2024
Number of holes	18 pcs	17 pcs
Number of rows	3	2
Average hole length	15.2 m	15.3 m
Number of decks	2	2
Type of explosives used:		
ANFO	-	-
Cartridged emulsion	950.0 kg	925.0 kg
TNT / dynamite	16.740 kg	15.810 kg
Point of initiation	top deck	top deck
MIC	26.9 kg	27.7 kg
Total charge	966.7 kg	940.8 kg
Av. burden	2.9 m	3.1 m
Av. spacing	3.7 m	3.5 m
Distance to perforating holes	44 m	36 m
Distance to vibration monitor	559 m	551 m
PPV	1.40 mm/s	0.71 mm/s



Figure 18. Location of all five series of holes.

5 PERFORATING DRILL HOLES – OUTCOME

After conducting two trial series of perforating holes we took the decision not to do any more trials (we stopped doing blasting before drilling perforating holes) to reduce ground vibration level.

We did an additional three perforating holes series and we have added this to our standard operational procedure (SOP). All the perforating drill holes series are shown in Figure 18.

We considered perforating drill holes as an additional method of reducing ground vibration levels in Dubie quarry. In the future we're planning to extend this method outside of our quarry to other sites in Holcim group.

6 SUMMARY AND CONCLUSION

From our analysis implementation of perforating drill holes lowers ground vibration level by around 30% on Peak Particle Velocity results in Dubie quarry conditions. This method is in constant use. It helps reduce ground vibration levels in all our blasts in the areas which are in closest proximity to village housing. We'd like to share this method with our colleagues from the drilling & blasting industry as an example that an unorthodox method can gave results.

REFERENCES

There were no external sources used in this paper. Everything is the author's own work.

[RETURN TO CONTENTS PAGE](#)

Compliance of blast vibration standards: the predominant role of weather and seismic activity in residential crack displacement

R. Gómez

SOL Ingeniería SpA, Santiago, Chile

J. Gjørdvad

Strategic Partnership manager Sigicom AB, Sweden

ABSTRACT: This study presents the results of a community-based monitoring initiative aimed at evaluating the influence of mining blast vibrations, seismic activity, and environmental conditions on residential structural cracks. Real-time instrumentation was installed in two homes near an active mine site, measuring ground vibrations, temperature, humidity, and crack displacements. Vibrations from blasting events remained within Chilean regulatory limits and did not produce structural damage. In contrast, a magnitude 5.5 earthquake triggered measurable but non-critical responses. A comparative frequency analysis between the earthquake and a high-energy blast event revealed that low-frequency vibrations such as those from seismic activity have a higher potential to induce structural displacement than high-frequency vibrations generated by blasting. Temperature and humidity showed stronger and more consistent correlations with crack behaviour than vibration events. This approach illustrates how community engagement and technical instrumentation can support transparency and risk management in mining-affected areas.

1 INTRODUCTION

Mining activities often generate ground vibrations that propagate over significant distances, potentially impacting nearby infrastructure. In residential areas, such vibrations are frequently perceived as disturbing or even damaging, especially in buildings with pre-existing structural weaknesses. As a result, monitoring and regulating vibration levels is essential to ensure not only structural integrity but also social acceptance of mining operations in populated zones.

In this context, a real-time structural monitoring project was conducted in the El Cobre Alto community of Andacollo, Chile. The initiative was launched in direct response to

community concerns about the possible effects of blasting operations from the nearby Carmen de Andacollo mine. Two residential houses were instrumented with extensometers, triaxial geophones, and sensors for ambient temperature and relative humidity to evaluate the behaviour of existing cracks and their potential evolution under the influence of blasting events and environmental conditions.

The technical framework for the study is based on the Chilean standard NCh3577:2018, which provides a methodology for evaluating ground vibrations produced by blasting and their impact on infrastructure. This national standard classifies structures into three groups heavy (Group I), residential (Group II), and light or historical



Figure 1. Aerial view showing the instrumented houses.

(Group III) and defines threshold values for peak particle velocity (PPV) as a function of structural type and dominant frequency. NCh3577:2018 is conceptually aligned with international references such as DIN 4150-3 (Germany) and UNE 22381-93 (Spain), which are widely used in the assessment of blast-induced vibrations.

While DIN 4150-3 emphasises long-term structural response and human perception, and UNE 22381-93 extends its scope to heritage structures and measurement protocols, the Chilean standard adapts these principles to local geotechnical and construction contexts. This study illustrates how such standards can be effectively applied in a community-driven monitoring project, and examines the relative influence of blasting vibrations versus environmental factors particularly temperature and humidity on the behaviour of structural cracks.

2 COMMUNITY-ORIGINATED OBJECTIVE

This initiative was not conceived as an academic research project. Instead, it emerged directly from the concerns of the local community. Residents had long reported structural damage, and in response, Carmen de Andacollo mine, deployed a transparent monitoring system to demonstrate

compliance with regulatory limits and assess real impact. Results were shared with the community throughout the process.

3 METHODOLOGY

To evaluate the effects of blasting and environmental conditions on structural cracks, a real-time monitoring system was deployed in two residential homes in the El Cobre Alto community, near the Carmen de Andacollo mine.

The houses were selected after a technical field inspection conducted on July 23-24, 2024, during which several properties were assessed. The two chosen houses El Cobre 1080 and El Cobre 14 presented visible, structurally relevant cracks and offered suitable conditions for sensor installation. El Cobre 1080 showed damage in interior walls, while El Cobre 14 had a critical crack near a slope modified by neighbouring construction.

Photographic documentation was captured during the fieldwork to illustrate the monitoring setup inside both houses. The following images show the installation of extensometers across structural cracks, the placement of triaxial geophones for vibration monitoring, and the sensors used to record temperature and relative humidity. These visuals provide practical context



Figure 2. Instrumentation setup inside the monitored homes, including extensometers, triaxial geophones, and temperature and humidity sensors installed at El Cobre 1080 and El Cobre 14.

on the structural conditions and the arrangement of the equipment deployed during the study.

Monitoring was conducted between August 7 and October 10, 2024, totalling over 1,500 hours per house. The system recorded:

- Crack displacement using extensometers installed across selected cracks.
- Ground vibration in three components using triaxial geophones.
- Ambient temperature and relative humidity via integrated sensors.

All data were transmitted in real time via a 4G cellular network to an online platform, enabling continuous remote access and analysis. This setup ensured high-frequency, synchronised datasets for evaluating the influence of both vibrations and environmental factors on crack behaviour.

3.1 Correlation analysis methodology

To explore the relationships between crack displacement and external variables, a statistical

correlation analysis was performed using the collected time-series data. This included:

- Ground vibration records (vertical, longitudinal, and transverse components).
- Ambient temperature.
- Relative humidity.
- Crack displacement values from the extensometers.

Blasting events and seismic activity were marked within the dataset to allow focused examination of their potential influence on crack movement.

The Pearson correlation coefficient was used to measure the strength and direction of linear relationships between each variable and the recorded displacement. The formula for Pearson's r is:

$$r = \frac{n(\sum xy) - (\sum x)(\sum y)}{\sqrt{[n\sum x^2 - (\sum x)^2][n\sum y^2 - (\sum y)^2]}} \quad (1)$$

Where:

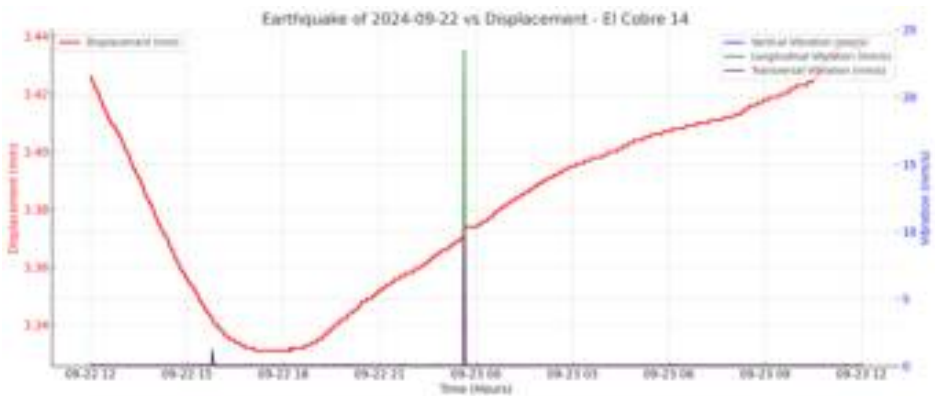


Figure 3. Crack displacement and seismic vibration El Cobre 14.

- r is the Pearson correlation coefficient.
- n is the number of data pairs.
- x and y are the two variables being compared (e.g., vibration vs. displacement).

The result of r ranges from -1 to 1:

- 1 indicates a perfect positive correlation,
- -1 indicates a perfect negative correlation,
- 0 indicates no linear correlation.

This methodology allowed identification of which factors vibration, temperature, or humidity had the most consistent statistical relationship with structural behaviour. The results of this analysis are presented and discussed in Chapter 4.

4 RESULTS AND ANALYSIS

The monitoring campaign produced a continuous,

Table 1. Pearson correlation coefficient figures for both houses.

Factor	Correlation		Comparative Interpretation
	El Cobre 1080	El Cobre 14	
Total Vibration	0.005	0.004	Total vibration shows virtually no correlation at either location, indicating it does not significantly affect crack displacement measured by the extensometer in either house.
Temperature	0.223	-0.679	Temperature has opposite effects in the two homes. In El Cobre 1080, the correlation is moderately positive (0.223), while in El Cobre 14 it is moderately to strongly negative (-0.679). This suggests that the influence of temperature depends on the structural and environmental characteristics specific to each location.
Relative Humidity	-0.321	0.569	Relative humidity also shows contrasting influence between the two houses. In El Cobre 1080, the correlation is moderately negative (-0.321), whereas in El Cobre 14 it is moderately positive (0.569). This difference may be related to exposure to moisture, building materials, or the structural condition of each house.

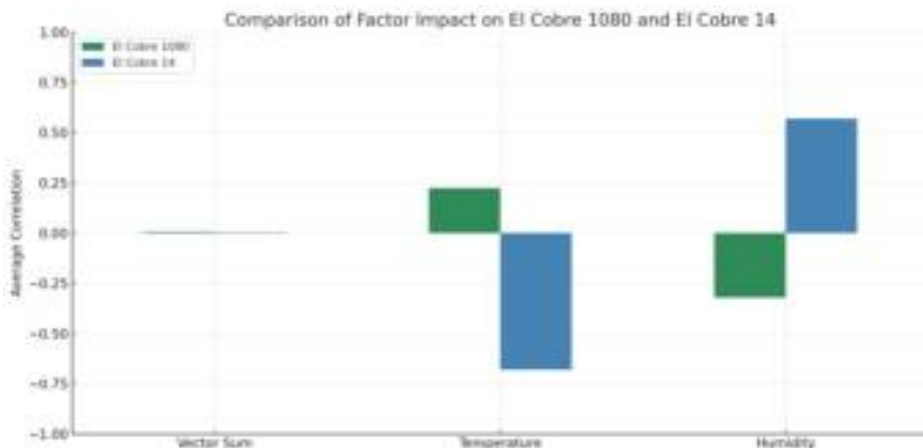


Figure 4. Pearson correlation coefficients El Cobre 1080 and El Cobre 14.

high-frequency dataset that allowed for a detailed evaluation of the factors influencing crack behaviour in two residential homes exposed to mining and environmental dynamics. The analysis covered both natural and anthropogenic vibration events, as well as ambient conditions such as temperature and humidity, over a total of more than 3,000 monitoring hours combined.

4.1 General observations

Displacement remained minor throughout the monitoring period, with values not exceeding 0.3 mm in either home. All recorded blasting events were found to comply with the vibration thresholds specified by the Chilean standard

NCh3577:2018 for Group II residential structures. This suggests that the operational blasting practices were not a direct structural threat to the monitored houses. However, a magnitude 5.5 earthquake that occurred on September 22, 2024, approximately 20 km north of the site, generated vibration levels that exceeded the standard's thresholds at El Cobre 14. Despite this, no abrupt or critical structural movements were observed.

The corresponding displacement response was minimal but measurable, and will be illustrated in a dedicated plot.

4.2 Correlation analysis

A Pearson correlation analysis was conducted to

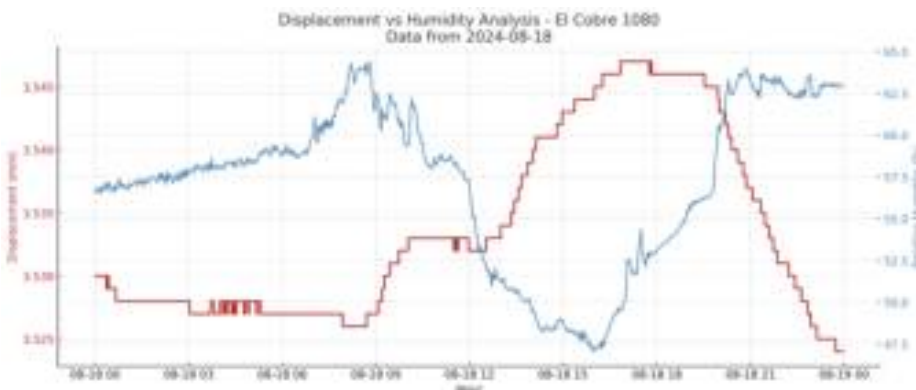
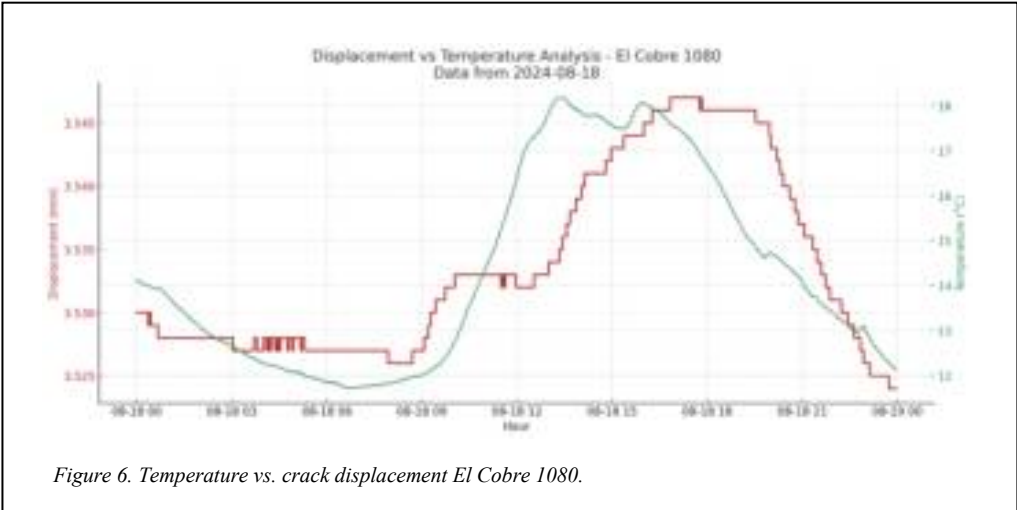


Figure 5. Relative humidity vs. crack displacement El Cobre 1080.



assess the relationship between crack displacement and three key variables: total vibration magnitude (from blasting and seismic events), ambient temperature, and relative humidity.

The results showed weak or negligible correlation between vibration and crack movement in both houses. In contrast, temperature and humidity exhibited more consistent correlations with structural displacement.

Table 1 presents the Pearson correlation coefficients calculated for each variable in both houses, El Cobre 1080 and El Cobre 14. Additionally, it includes a comparative interpretation that highlights how the influence of each factor varies depending on the structural and contextual characteristics of each location. This table provides an overview of which variables had the strongest linear relationship with crack

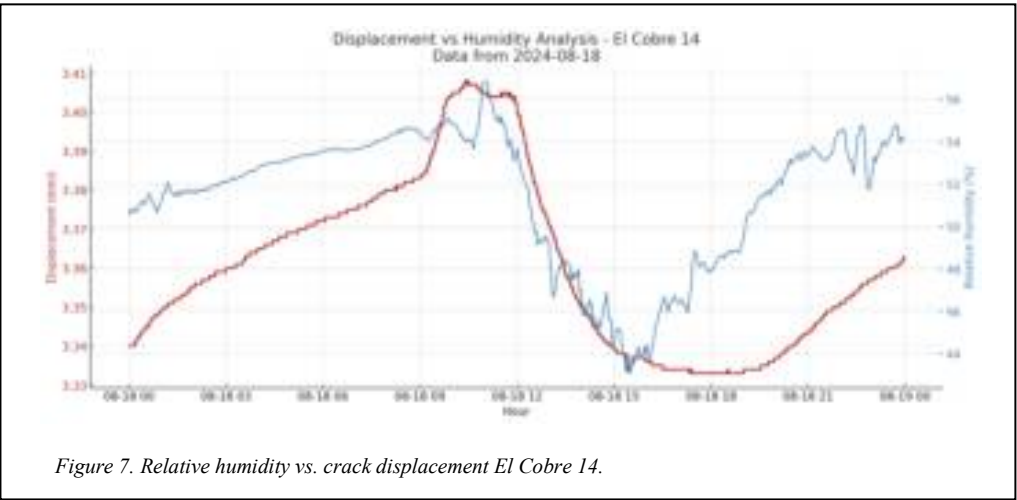
behaviour, reinforcing the conclusion that temperature and humidity were more influential than ground vibration.

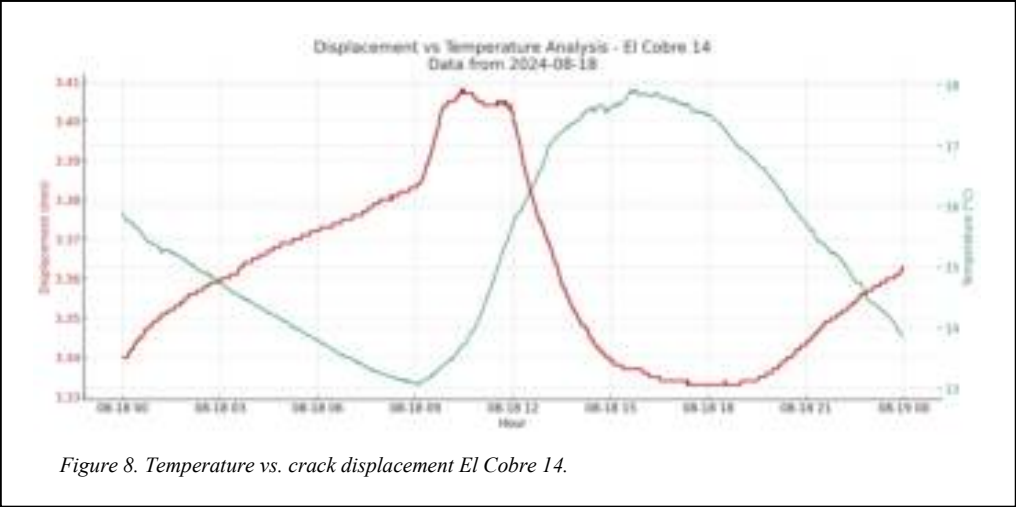
4.3 Environmental influence on displacement

To better understand the behaviour of cracks in relation to environmental conditions, time-series comparisons were plotted for each house:

El Cobre 1080: A negative correlation was observed between relative humidity and displacement, indicating a tendency for crack contraction in more humid conditions. A moderately positive correlation was also identified between temperature and displacement, suggesting slight expansion during warmer periods.

El Cobre 14: Similar patterns were observed, but with stronger correlation values. In particular,



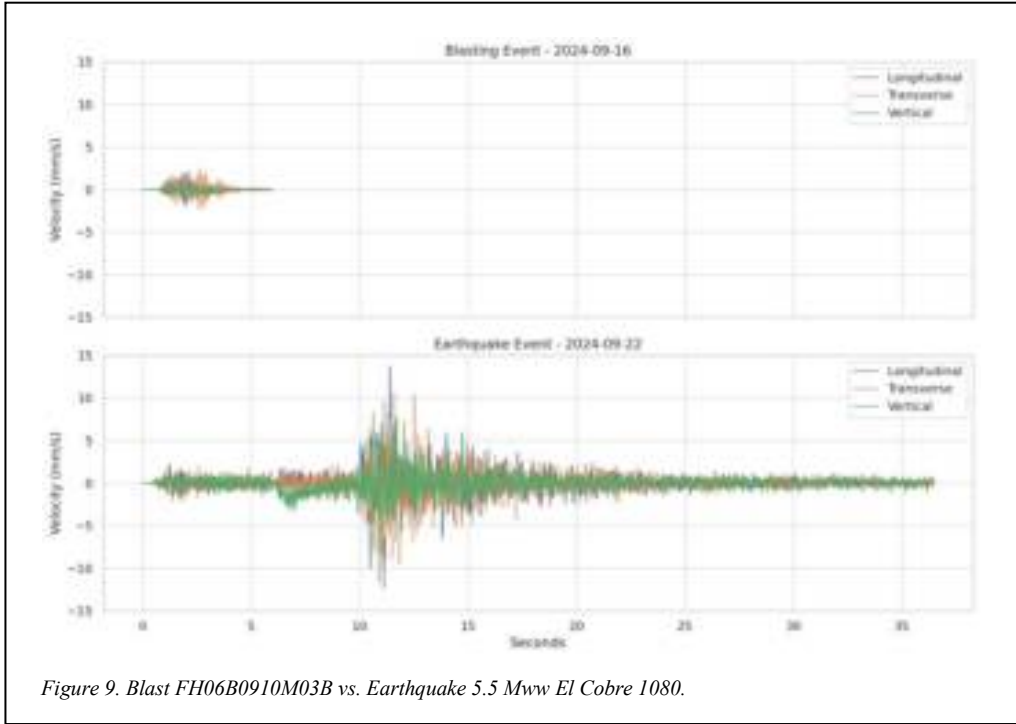


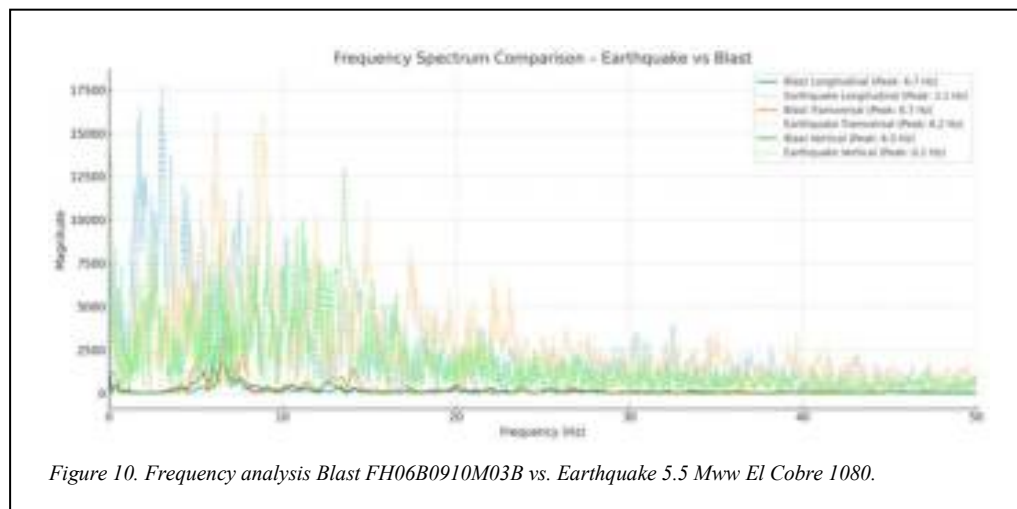
temperature variations were more strongly associated with crack movement, showing a moderately strong negative relationship, suggesting crack contraction during warmer periods. Relative humidity, conversely, showed a moderate positive correlation, indicating crack expansion in more humid conditions. These visualisations provide evidence that temperature and humidity, rather than vibration levels, were the dominant factors influencing

minor structural movement during the study period.

4.4 Frequency Influence on Structural Response: Earthquake vs Blasting

During the monitoring period, multiple vibration events were recorded, both from blasting activities and natural earthquakes. For this analysis, the two most significant events in terms of recorded





intensity were selected:

- A magnitude 5.5 Mww earthquake that occurred on September 22, 2024, at 23:38, located approximately 20 km north of La Serena.
- A production blast identified by the code FH06B0910M03B, carried out on September 16, 2024, at 11:53.

Both events represented the highest vibration levels recorded by the network of monitoring stations during the observation period. For this specific analysis, additional instrumentation was used, installed in the garden of El Cobre 1080, approximately 10 m from the sensors used throughout the rest of the study. This station is part of the permanent monitoring system operated by the mining company and uses the same type of sensors employed in this research, ensuring data comparability.

To characterise both events, two types of graphs were generated in Figure 9:

- Vibration over time (velocity in mm/s), to observe the behaviour of each component (longitudinal, transverse, and vertical).
- Frequency analysis (spectrum), to determine the predominant type of vibration, distinguishing between high and low frequencies in Figure 10.

The results reveal clear differences:

- The earthquake exhibited dominant low frequencies, below 10 Hz, with a significant peak around 0.05 Hz in the vertical component. This type of vibration is associated with long-period waves capable of causing greater displacements in structures

such as houses, even when the amplitude is not extreme.

- The blasting event exhibited dominant frequencies between 6.5 and 6.7 Hz across all components, which fall within the range where structures like houses are more susceptible to resonance. However, the overall amplitude was low, and the duration short, indicating that the potential for structural damage remains limited despite the overlap in frequency range with typical building resonance.

This analysis complements the evaluation conducted with sensors installed in homes, confirming that not only the intensity of the motion matters, but also its frequency content. In particular, low-frequency vibrations like those generated by the earthquake tend to have a greater structural impact than high-frequency vibrations typical of controlled blasting.

4.5 Summary of findings

Overall, the data demonstrate that:

- Blasting vibrations, although perceptible and frequent, remained within the regulatory limits established by Chilean standard NCh3577:2018. The recorded frequencies were predominantly between 6 and 7 Hz, with relatively low amplitudes and short duration, which limits their capacity to cause structural damage to nearby houses.
- Seismic activity, particularly the magnitude 5.5 earthquake on September 22, 2024, exhibited dominant low-frequency content below 10 Hz, with a significant peak around 0.05 Hz in the vertical component. These

long-period waves are more likely to produce larger displacements in structures, potentially affecting crack behaviour even in the absence of high amplitudes.

- Temperature and relative humidity were the most influential environmental variables, showing consistent correlations with crack movement. Seasonal changes in these parameters were associated with expansion and contraction cycles in the materials, although the degree and direction of influence varied between House 1 and House 2.

5 COMPLIANCE WITH NCh3577:2018

The study applied the NCh3577:2018 standard, which sets particle velocity thresholds by structural category and dominant frequency. Both homes fall under Group II (residential). Vibrations from blasting remained within these limits. Only the seismic event briefly surpassed the thresholds in one home, validating the normative safety criteria under typical operational conditions.

6 CONCLUSIONS

This community-driven monitoring initiative demonstrated the value of real-time structural instrumentation to assess how residential cracks respond to mining-related vibrations and environmental conditions such as temperature and humidity. While the project was not designed as a scientific study, it effectively addressed the concerns of local residents regarding potential damage from blasting activity, and it also generated a robust dataset suitable for further technical evaluation.

The main conclusions are:

- Blasting-induced vibrations remained consistently below the limits established by the Chilean standard NCh3577:2018 for Group II residential structures, even during high-energy detonations. Frequency analysis revealed that these events primarily produced vibrations in the 6 - 7 Hz range, with low amplitudes and short durations, limiting their structural impact. No damage attributable to blasting was observed in the monitored homes.
- The magnitude 5.5 earthquake of September 22, 2024 resulted in the highest vibration levels recorded during the study. Frequency analysis indicated dominant low-frequency content, with a peak around 0.05 Hz in the vertical component. These long-period waves have a greater potential to cause structural

displacement, and although the recorded movement was modest (below 0.01 mm), the event confirms the importance of low-frequency content in seismic risk assessment.

- Crack displacement did not correlate strongly with ground vibrations from either blasting or seismic events. This suggests that short-duration vibrations, even if perceptible, are less significant drivers of structural change than longer-term environmental factors.
- Temperature and humidity emerged as the most influential variables affecting crack behaviour:
 - Higher humidity was generally associated with crack widening.
 - Increased temperatures tended to cause contraction of the cracks. These patterns were more evident at El Cobre 14, likely due to differences in exposure and construction.
- The combined use of extensometers, geophones, and temperature and humidity sensors, supported by real-time data transmission, enabled continuous monitoring and early detection of potential risks. This approach provides a replicable model for other mining-affected communities.

This project underscores the importance of integrating community engagement with rigorous technical practices. By aligning local monitoring efforts with both national regulations and detailed vibration analysis, mining operations can enhance transparency, reduce conflict, and improve long-term risk management.

7 ACKNOWLEDGEMENTS

This project was made possible thanks to the residents of El Cobre Alto, and the commitment of Carmen de Andacollo mine to transparent environmental practices.

This paper benefited from the use of AI-based tools (ChatGPT, OpenAI) to support the cross-analysis of extensive environmental and structural monitoring datasets. All interpretations, conclusions, and content were verified and approved by the human author.

REFERENCES

Deutsches Institut für Normung. (1999). DIN 4150-3:1999-02 Structural vibration Part 3: Effects of vibration on structures. Berlin, Germany.

Dowding, C. H. 1996. Micrometer crack response to vibration and weather. *Northwestern University*.

National Institute for Standardisation. (2018). NCh3577:2018 Methodology for assessing vibrations from blasting on infrastructure. Santiago, Chile.

National Seismological Center, University of Chile. (2024). *Earthquake Catalog of Chile*. Retrieved from <https://www.sismologia.cl>

Oriard, L.L. 2002. The effects of vibrations and environmental forces. *International Society of Explosives Engineers*.

Spanish Association for Standardisation and Certification (AENOR). (1993). UNE 22-381-93 Assessment of the effects of vibrations caused by blasting on buildings. Madrid, Spain.

[RETURN TO CONTENTS PAGE](#)

2. Blasting work experiences

The critical importance of the bottom 30 centimeters of the blasthole in close-in construction blasting

J. Wallace

Wallace Technical Blasting, Inc., Woodland, Washington, USA

ABSTRACT: 45 years of construction blasting has shown to this author the critical importance of the bottom 30 centimetres of the blasthole to success in construction blasting in hard rock. Whether foundations or trenching, the charge located within the bottom 30 centimetres of the blasthole must be able to break the rock and move the burden at that location. The bottom of the hole is especially critical when decking holes in close-in blasting, where the distance to structures means that the maximum instantaneous charge is below the charge weight that would be necessary for the volume of rock in the entire hole. Reducing the charge weight to comply with vibration concerns cannot be allowed to reduce the charge weight within the critical bottom 30 centimetres to below what is necessary to accomplish the work. If that charge weight is insufficient the rock will not dig to grade (be excavated without additional problems and actions). Working within the parameters may require reducing the burden, increasing the hole diameter, excavating a free face for the material to move into, or any or all the above. A recent blasting job for wine cellar construction when it is necessary to excavate to a lower elevation than the existing structure foundation, but also to tie to that foundation well, is discussed to showcase this system in the field and give a rough guideline to charge type and quantity that has proven successful.

1 CLOSE-IN CONSTRUCTION BLASTING

This paper focuses on close-in blasting for (a) poured-to-solid foundations where subdrill is either minimal or not allowed and (b) trenches where the cut depth exceeds the distance to nearby structures. My observations and suggestions are informed by over 45 years in the field as a blaster and blast consultant on over a hundred construction blasting projects. Much of my experience has been in the hard basalt of the U.S. Pacific Northwest. As every blaster knows,

blasting techniques must be chosen with the specific geological conditions in mind.

The working definition of close-in construction blasting used here is blasting which:

- includes foundation blasting within 5 metres of structures and/or trench blasting where the required trench depth is greater than the distance to structures, utilities, or other sensitive encumbrances,
- must leave behind intact geology for construction purposes, usually including a floor



Figure 1. Close-in construction blast site at Friday Harbor, San Juan Island, Washington, USA.
Photo credit: Geri Woods.

- suitable for subsequent pour-to-solid foundation construction (as opposed to foundation poured in forms or on a compacted-fill base,
- must create a rock-free void of specified dimensions, typically including an essentially flat floor,
- must minimise overbreak beyond the specified dimensions,
- must reduce the rubble to manageable size, but where achieving the design dimensions of the void is a higher priority than producing rubble of a specific size.

An example of a close-in urban construction project is shown in Figure 1. Here, blasting made way for new foundations to be tied into an existing building (shown on the right). Blasting was conducted within metres of in-use buildings, and roads with heavy traffic especially during ferry loading times. The site is on an island, necessitating special permitting and coordination of blasting, explosives delivery, and ferry schedules.

2 SAFETY IS ALWAYS FIRST

Blast design and practice in close-in construction projects, of course, share many basics with other

blasting projects. [See ISEE Blaster's Handbook, chapter 33, surface construction blasting.] However, the fact that these projects require blasting close to existing structures or other sensitive areas heightens safety concerns. Special attention must be paid to the potential need for traffic and crowd control measures. Blast planning will need to include a survey of any nearby structures as well as the activities taking place within them. Extra public relations efforts are likely to pay positive dividends in terms of stakeholder satisfaction. Figure 2 shows the site of construction blasting within 1.5 metres of a school building which had children inside during blasting. Extremely cautious blasting procedures, including the use of blasting mats, coordination with school officials, and a complex safety plan were required.

3 CRITICAL ELEMENTS OF CLOSE-IN BLASTING

Close-in construction blasting typically presents technical considerations which can make such projects particularly complex and challenging.

Perhaps the most challenging aspect of close-in construction blasting is that it requires a reversal in the blaster's typical point of view. In most applications such as mining, road construction,



*Figure 2. Blasting up 1.5 m of a school. With students inside. St Helens, Oregon, USA.
Photo credit: Doug Harding.*

general sitework and the like, the primary focus of blasting is to reduce a rock mass to rubble of an appropriate size for removal and, often, for subsequent use (for example as fill material) or processing (for example for crushing or mineral extraction). Beyond perhaps leaving behind a passageway or hole of relatively rough dimensions, the success of the blast is likely to be assessed mostly in terms of the state of the rubble produced. A blaster with a philosophical bent may experience an ‘Alice in Wonderland’ moment when first considering the design and execution of close-in construction blasting where suddenly the primary ‘product’ shifts from the rubble itself to the highly-specified void that blasting leaves behind. I invite the reader to step into that ‘void’ with me now.

3.1 The critical toe of the blasthole

A primary challenge in close-in construction blasting is ‘to pull the toe,’ that is to break the rock to grade at the bottom of the blasthole. In blasting for a pour-to-solid construction design, breakage below design grade needs to be minimised.

Based on many years of field experience, I have adopted a strong rule-of-thumb that defines the area of the toe to be the bottom 30 centimetres

of the blasthole. Vibration prediction calculations to establish maximum instantaneous charge, followed by choice of hole diameter, burden, drilling, priming, loading and blasting within this zone has proven to be critical to the success of the shot. The following subsections discuss the technical elements which, together, result in effective and precise blasting at the toe.

3.2 MIC calculations, the first step

In these situations, structures nearby will have limits that impact the maximum charge weight per delay, or maximum instantaneous charge (MIC). The MIC is frequently controlled by regulation, commonly to vibration limits, scaled distance formulas, or vibration regression formulas. At other times no formula is established but frequency dependent maximum peak particle velocities are the guiding principle.

When blasting close-in, the MIC is commonly less than the total charge weight necessary per hole for proper rock breakage. That MIC then must properly break the rock at the toe/grade elevation, with additional separated deck charges further up the hole. At times as many as 5 or more individual charges are necessary.



Figure 3. An example of a bench with blastholes being loaded for a poured-to-solid foundation. Blasting to tie in to the sewer tank just behind the excavator bucket (containing stemming material). Sweet Home, Oregon. Photo credit: Bill Parke.

A useful formula for arriving at the MIC, developed based on his significant experience and data from blasting in the hard basalts in the Pacific Northwest is the Lewis L. Oriard formula:

$$PPPV=K (SD)^{-1.6} \quad (1)$$

PPPV, predicted peak particle velocity, K, constant variable dependent upon ground conditions and relief but varies commonly from 100 to 400, SD, scaled distance, -1.6, is the decay slope. This is the Imperial units formula.

The formula in metric units is then:

$$PPPV = K (SD)^{-n} \quad (2)$$

K varies, with the decay slope approximately -1.6 to -1.8 for close-in blasting work.

Both these are square-root formulas. At times when blasting exceptionally close these predictions come in higher than the actual readings so one can use a cube-root formula to determine charge weight. Be advised, one must record the vibration levels and make certain that using a cube-root formula does not violate a regulation or ordinance.

The MIC must be sufficient to break the toe at grade, and importantly, and based upon my experience, it must be loaded into the bottom 30 centimetres of the drill hole. This necessity will determine the burden dimension and the hole diameter. For example, if the MIC weight that can comply with vibration limits is 2 kg, then using a blasting agent with a density of 1.2 g/cc, with a 450-gram cast booster as primer, the hole diameter must be at least 75 mm to accommodate that charge weight in the 30 cm length. That charge weight will break a burden of 1.3 metres, or slightly more depending upon a free face and relief. The free face offers two advantages: a larger burden and reduced vibration.

3.3 Drilling

Following determination of the MIC, the hole diameter to contain that charge in the bottom 30 centimetres, and the burden dimension that MIC will satisfactorily break, the drill pattern can be laid out. Drilling accurate, properly located, proper depth/length and proper diameter holes is critical to success in this type work. Without accurate drilling the rest of the equation simply falls apart. Figure 5 shows an example of a complex pour-to-solid design where proper drilling was critical.

3.4 Proper priming

Helpful background information on priming can be found in the Atlas Powder Company handbook, Explosives and Rock Blasting, pages 205-231. While this handbook was published in 1987, the laws of physics have not changed since.

It is critical to get the MIC to steady state in as short a distance and time-period as possible for proper breakage at the bottom of the hole. This requirement affects the choice of primer. Cast boosters are the optimal choice, since run-up to steady state in other primer products can mean that the bottom 30 centimetres do not receive the full and desired energy necessary to break the toe. For example, if a 50 mm by 200 mm cartridge of density 1.43 g/cc dynamite is used at the primer, with 3-times the charge diameter run-up to steady state, with a 75 mm long delay detonator installed properly at the base, the cartridge does not reach steady state. 75 mm plus 3 times 50 mm equals 225 mm, a distance to steady state that exceeds the length of the cartridge, and encompasses nearly the entire 30 cm length of critical toe of the hole.

If a 50 mm by 400 mm cartridge is used then approximately 175 mm of the upper end of that cartridge fires at steady state. This produces 92



Figure 4. Complex industrial site work, trenching and foundations with multiple structures both overhead and in the ground. St. Helens, Oregon. Photo credit: Bill Parke.

kbar of energy at a detonation velocity of 5,000 metres per second for 0.52 kg/520 gram starting 225 mm above the hole bottom, again a huge reduction in energy where it is most needed.

Conversely, when using a 450-gramme cast booster embedded into 1.2 g/cc blasting agent for full wall coupling, that bottom 30 cm of hole fires immediately at 230 kbar of pressure, with a detonation velocity approximately 7,000 metres/second. This is significantly more energy than provided in the two previous scenarios, and is key to breaking the rock at grade.

3.5 *Blasting*

This significant increase in energy at the toe allows for reduced subdrill, or in the case where no subdrilling is allowed, for a flatter angle of breakage between holes at the floor thereby reducing the amount of mechanical breakage necessary to bring the poured-to-solid floor down to grade.

This also allows the burden to be increased when blasting to a free face. Empirical evidence from decades of blasting in the hard basalt formations in the Pacific Northwest shows that a range of 0.40 kg to 0.45 kg per 30 cm of hole per 30 cm of burden will break even the toughest toe

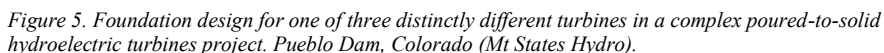
at grade, relief and timing dependent. This shooting to a free face must be accompanied by careful attention to flyrock control, hence the following precaution.

PRECAUTION! This high energy source at the toe does increase the risk for flyrock in highly elastic basalt formations. Similar to the reaction when stretching a strong rubber band to the breaking point, the response in highly elastic rock is much more intense when the bonding is strong than when the bonding is weak.

Relief is key to reduced vibration for a given MIC's. Vibration levels can be reduced to as little as 25% of that predicted and recorded on a fully confined charge by shooting to full face relief.

4 CASE STUDY

The following case study showcases the application of the critical close-in blasting elements. In this project, sitework for a wine cellar at a private residence require excavation in hard basalt. When the homeowner constructed the house about a decade prior mechanical means of rock breakage had been attempted. Following failure of that method, drilling and blasting was used. This earlier successful experience led the owner to be amenable to using the drill and blast



The rock was hard to very-hard basalt, with three-dimensional fracturing, and a nearly vertical dominant joint set. The project area was bordered on the south by a natural area, with hiking trails and public access.

The pool house, with the foundation we were tying in to, was at 0 distance, but based upon personal experience blasting near similar foundations, and with the agreement of the owner, we felt that 250 mm at high frequencies would not cause damage.



Figure 6. Wine cellar, tying in to foundation of structure to right, house 12 metres beyond. Blasting mats used to limit flyrock. Looking north. Photo credit: Jerry Wallace.

Experience in the rock type had shown that 1.5 kg of explosives per metre of burden, but loaded into the bottom 30 cm of hole, would break and move the rock especially when firing to a free face. The selected burden would then require 1.8 kg of explosives, allowing for a reduced MIC nearest the pool foundation that we were tying in to. 1.8 kg of emulsion explosives at 1.2 density would fit into 30 cm of a 75 mm drillhole when slit, with the 350 gram cast booster gently shoved into the slit cartridge for full wall coupling.

Using the basic rule of spacing not exceeding 0.5 of the hole depth, a square pattern of burden and spacing of 1.2 metres was chosen for the entire pad. Along the north and east final walls additional half-spacing holes were drilled for the closest 2 rows.

Blasting began at the point furthest downslope, with care taken to not toss material onto the natural area bordering the project. A maximum of 3 rows of blast holes, stretching half the width of the foundation, were fired per shot. Their limit a function of the area that could be covered by available blasting mats. Following the first blast the face was excavated to grade with the resulting material stacked just at the edge of the foundation, creating a berm further protecting the natural area.

A total of 20 blasts, 7 in each half, plus 6 small blasts in the roadway/driveway decline into the foundation, were conducted. Vibration monitoring with 5 instruments showed we stayed within regulatory limits, with the maximum at the pool house estimated to be about 175mm based on the reading at the nearest seismograph. No rock left the worksite so there was no impact on the natural area.

During the blasting the homeowner/project owner plus his adult son plus the neighbour were each allowed to fire a blast from the usual safe location, behind the excavator situated off the northwest corner of the cellar. Each really enjoyed the experience.

Drone footage captured each blast, plus served to view the natural area for hikers as additional security.

4.2 Case study results

When proper toe priming is used, combined with accurate drilling to the required diameter and depth, the results are predictable. Blasting can be accomplished to flush against existing engineered structures. Properly designed blasts can use the discontinuity between the rock being blasted and



Figure 7. Overview of project from blast report #13. Looking north. Photo credit: SubTerra, Inc.

the existing concrete structure as a reflective point, with the resulting tension wave back into the rock mass sufficient to detach the rock from the concrete.

5 CHALLENGES TO IMPLEMENTING CLOSE-IN METHODOLOGY

In my experience the main challenges to properly implementing recommended methods for successful close-in construction blasting have been over-prescriptive specifications and a focus on false economies.

5.1 Improper specifications

Overly prescriptive specifications require the use of formulas regarding hole diameter and charge diameters that are not applicable to the situation at hand. Requiring by regulation or code the maximum hole diameter and charge diameter regardless of the demands of the geology can lead to higher vibration levels, increased flyrock risk and poorly broken rock that will need additional mechanical breakage.

The suggestion that one cannot prove a negative gets a real challenge when following prescriptive specifications. Investigations have shown failures directly related to following the

incorrect-for-the-site means-and-methods. One recent project required the blaster to use less than 30% of what my calculations and empirical evidence from decades of blasting in that geology tell me would work. The results proved the negative. The shot would not dig, even with massive excavating equipment.

5.2 False economies

False economies frequently come from money managers further up the chain, two-degrees separated from the activities on the bench. These false economies are pushed downward to workers by those without enough knowledge about blasting to understand the negative effect these false economies will have on the finished product.

Replacing over-blasted rock that cost, say, \$25 per cubic meter to drill and blast with lean concrete is rarely cost effective. On a recent project that concrete would have to have been flown in by helicopter, hugely increasing cost unnecessarily.

Micro-managing labour, with labour too often seen as a begrudged expense rather than as an asset that generates profit, and micro-managing which explosives are used purely on cost per unit rather than on work produced are all too common.



Figure 8. Successful completion. Note oxidized existing joint structure. No damage to existing foundation, no vibration issues, no flyrock.

It takes time, and labour cost, to inspect each drillhole prior to loading the shot to ensure that all are clear and to the proper depth. Clearing of detritus from the bottom of the holes can be necessary. At times on projects that are publicly contentious that detritus extends beyond the usual mud, water and drill-cuttings mix to include survey stakes, beer bottles and even dirty baby diapers. The hole must be clean to allow loading the primer at the point of maximum confinement, which except in rare circumstances, is the bottom 30cm of the hole.

It takes additional time to properly place the charges at the toe. Loading crews have frequently been directed to simply pull the cast boosters up to make sure they are within the emulsion as an option. This is an expensive alternative to proper loading when the toe is not pulled nor broken to design grade.

Micro-managing explosives cost without considering how critical the toe charge is leads to undershooting the toe, causing 'bottom-bounce' where the proper breakage begins further up the hole. This result often is addressed by adding subdrill, but in pour-to-solid floors additional subdrill is not allowed, hence more mechanical secondary work is necessary.

A narrow focus on drill and blast costs too often results in higher cost for the project as a whole. Given that the foundation drilling and blasting is commonly only a very small percentage of the total cost of a power generation facility, or sewer treatment plant upgrade, or a new school, attempting to incorrectly micro-manage such a small part can have huge risk and lead to unnecessary additional costs.

There have been numerous studies, and research papers published on the holistic mine-to-mill approach to support improved blasting. It is this same approach that must be taken to close-in construction blasting.

6 CONCLUSION

Using a systematic approach to the unique challenges at the specific site can provide proper results. In this case a wine cellar foundation was safely and efficiently drilled and blasted, with the duration of the foundation excavation segment of the project much shorter than had been predicted if mechanical methods had been used, and at a lower cost.

Advice from the mentors in my past:

- taking the time to do the job properly the first times pays off in the long run.
- do the math, then do the work once. ⁽¹⁾
- breaking the toe at the proper grade is key to success. ⁽²⁾
- hire, and pay fairly, quality employees. ⁽³⁾

REFERENCES

- Atlas Powder Company 1987 *Explosives and rock blasting*, Dallas, Texas.
- Dowding, C.H. 2000 *Construction vibrations*, Northwestern University, Evanston, Illinois.
- Dowding, C.H. 2008 *Micrometer crack response to vibration and weather*, ISEE, Cleveland, Ohio.
- Floyd, J. 1993 *Efficient blasting techniques*, Blast Dynamics, Inc., Steamboat Springs, Colorado.
- ISEE Blasters' Handbook, 2001 18th Edition, Cleveland, Ohio.
- Lucca, F.J. 2003 *Tight Construction Blasting: Ground vibration basics, monitoring, and prediction*. Terra Dinamica, LLC.,

Oriard, L.L. 2005 Explosives Engineering, *Construction vibrations and geotechnology*, ISEE, Cleveland, Ohio.

Oriard, L.L. 1999 *The effects of vibrations and environmental forces*, ISEE, Cleveland, Ohio.

Persson, P.A. Holmberg, R. & Lee, J. 1994 Rock Blasting and Explosives Engineering, CRC Press, Inc. Boca Raton, Florida.

Siskind, D.E., Ph.D. 2000 *Vibrations from blasting*, ISEE, Cleveland, Ohio.

Teller, A.E. 1985 BLASTING TECHNIQUES:1. *Estimating the blasting project*. Explosives Services Corporation, Issaquah, Washington.

Teller, A.E. 1993 BLASTING TECHNIQUES:3. *Controlled blasting*. B.E.R.C., Issaquah, Washington. ⁽²⁾

Wallace, V.A. 1961 Personal conversations. ⁽¹⁾ and ⁽³⁾

[RETURN TO CONTENTS PAGE](#)

Challenges and restrictions of drilling and blasting in the Arctic Circle

A. Gagnon & A. Pina

Agnico Eagle Mines, Baker Lake, Nunavut, Canada

ABSTRACT: This paper examines the complex dynamics of drilling and blasting near the Arctic Circle, explicitly focusing on the Meadowbank complex at Agnico Eagle Mines in Nunavut, Canada. The study analyzes various challenges, including weather conditions, logistical issues, and regulatory restrictions. The goal is to comprehensively understand the difficulties encountered in this harsh environment using a rigorous methodology that incorporates documentation of operational experiences and key lessons learned. The study identifies specific challenges such as drilling through permafrost, managing frozen muck piles that necessitate frost shots, and dealing with weather conditions that impact the effectiveness and integrity of blast patterns. Furthermore, regulatory frameworks and indigenous rights significantly influence operations by imposing restrictions such as vibration control to protect fish in nearby lakes and measures to safeguard wildlife. This research underscores the importance of balancing economic interests with environmental protection and respecting indigenous rights. It offers recommendations aimed at promoting sustainable practices in the region.

1 CONTEXT

This paper focuses on close-in blasting for (a) poured-to-solid foundations where subdrill is either minimal or not allowed and (b) trenches where the cut depth exceeds the distance to nearby structures.

1.1 Location

The Meadowbank Complex is located in Canada's far northern region, specifically at the 65th parallel north. It is situated within the Arctic territory of Nunavut, which is recognized as the ancestral land of the Inuit community.

1.2 Procurement & Logistics

The mine site is not connected by road to the southern part of Canada, which makes logistics and procurement particularly challenging. Most materials and equipment are delivered by boat once a year during the summer months. Since the waterways are frozen for much of the year and the boat needs to make a lengthy journey around the country, delivery delays can reach six months or longer. As a result, it is impractical to use explosive products such as ANFO and packaged emulsion; by the time these products are manufactured and arrive on-site, they often exceed the expiration date recommended by the supplier.



Figure 1. Meadowbank location.

1.3 Mines and History

At the Meadowbank Complex, two open-pit mines and one underground mine are currently in operation. These mining activities take place in permafrost terrain, which presents unique challenges. Notably, the main pit has been developed below the former Whale Tail Lake, leading to unusual water infiltration conditions for mining in permafrost areas.

2 EXPLOSIVE INTEGRITY & MANAGEMENT

2.1 Pre-split material

During the winter, average temperatures at the Meadowbank Complex range from -32°C to -21°C , raising concerns about the integrity of explosive materials. The most critical product is the pre-split explosive, which consists of a detonator-sensitive emulsion combined with a detonating cord. This explosive tends to lose effectiveness in colder temperatures, and ice accumulation around the product has resulted in misfires, particularly in areas with high water infiltration.

Heated explosive magazines are available on-site, but they have limited capacity. Given the logistical challenges of transporting explosives to this remote location, storing all pre-split materials in the heated magazine is not feasible. Therefore, a procedure is implemented to rotate the product between the standard and heated magazines based on the demand for pre-split materials. The pre-split explosives must remain in the heated magazine for at least 12 hours before being authorised for use.

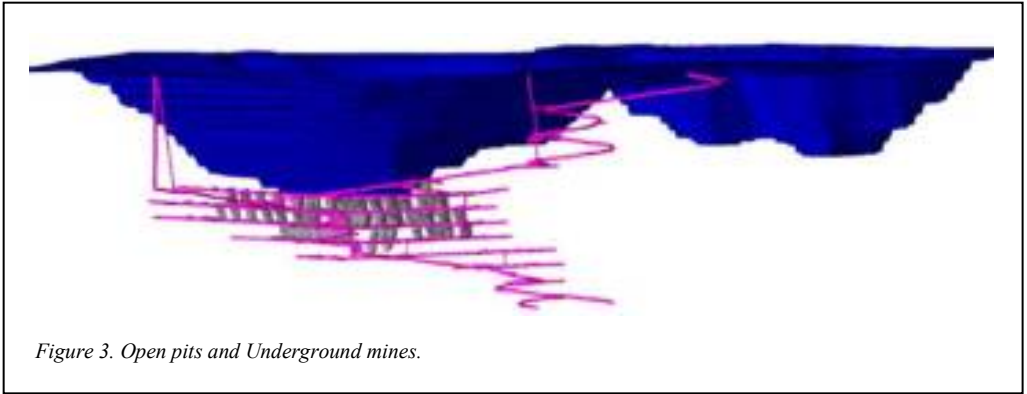
Further details will be discussed in the next section.



Figure 2. Meadowbank procurement route.
Photo credit: Jerry Wallace.

2.2 Electronic detonators

There have been some issues with electronic detonators. During the initial phase of the transition from nonel to electronic detonators in 2020, several misfire events were experienced due to high leakage in the detonators. Investigating these incidents revealed that the high leakage was caused by water condensation in the clip and wiring of the electronic detonators. When a hole was drilled into the permafrost during the winter, the temperature inside the hole was found to be warmer than the temperature in the magazine. These temperature changes led to water



condensation in the electronic components. However, the supplier developed an Arctic-grade electronic detonator, and since its implementation, no other misfires were encountered.

3 PERMAFROST AND WATER INFILTRATION

3.1 Drilling

As previously mentioned, when a hole is drilled into the permafrost, the temperature within the hole drops below freezing, leading to the formation of moisture. This moisture then freezes again, resulting in ice buildup inside the hole. This phenomenon is not unique to the Meadowbank site; it can also be observed in other mining operations in the Arctic. Establishing the main open pit below the former Whale Tail Lake adds to the situation's complexity. In some areas of the pits, groundwater infiltration can be observed

during drilling. Occasionally, water can rapidly rise in the drill hole upon intercepting a water source. Since the groundwater is constantly in motion, it remains in liquid form until it rises within the drill hole and begins to freeze within a few minutes.

3.2 Mass Loading

According to the mine health and safety regulations in Nunavut, 'No person shall drill or allow drilling to be conducted within 5 metres of a hole containing explosives.'

In the open pits, the burden and spacing are approximately 4 metres by 4 metres, which means that a distance of two burdens or spacings must be maintained between drilling and loading activities to comply with the regulations. However, maintaining the 5 metres distance becomes challenging in areas with high water infiltration. Freezing events occur rapidly, and by the time the



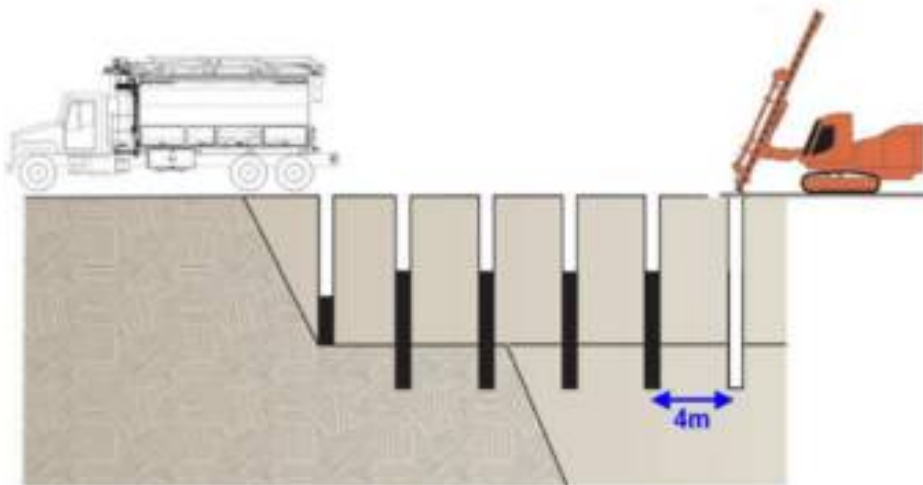


Figure 5. Hot loading derogation.

next hole is drilled, the first hole has already frozen and needs to be cleaned out.

Given these conditions, the Nunavut authorities (WSCC) granted a derogation to reduce the required distance from 5 to 4 metres. A 'Hot Loading' procedure has been implemented to ensure compliance with regulations.

3.3 Pre-split Loading

As mentioned in the second section, a rigorous procedure has been established to ensure the integrity of the pre-split material is maintained. This procedure outlines two typical scenarios encountered in open pits: drilling and loading under dry conditions and drilling and loading under water conditions.

In dry conditions, loading occurs immediately after drilling the pre-split line. The primary concern in this scenario is the temperature of the explosive material. The pre-split explosive should preferably be taken from the heated magazine; however, if the temperature recorded in the drilled holes is warm enough, it is permissible to load with frozen material, provided it remains in the holes for at least 12 hours before blasting.

In water conditions, drilling the entire pre-split line before loading is not feasible due to the previously mentioned freezing conditions. Therefore, loading must be performed as soon as possible and adhere to the 'hot loading' procedure. A limit of 30 holes per blast has been established to minimise the number of holes requiring cleanup and to reduce the risk of ice forming around the

pre-split explosive in the holes. The pre-split explosive must come from the heated magazine, and both loading and blasting should be carefully timed to minimise the exposure of the pre-split explosive to water and cold conditions.

4 FROZEN BLAST

4.1 Overburden

Due to the sites' location and existing regulations, the space allocated for the ore stockpile is limited to a specific footprint. Additionally, challenges such as fuel shortages, weather conditions, and potential closures due to wildlife migration are always possible. To ensure that operations are not compromised and that a sufficient supply of blasted ore is available to sustain milling operations, strategic planning is implemented to maintain a high inventory level.

To achieve this, an inventory of approximately 2 million tons of broken material is consistently kept within the open pits. The blast patterns are prepared well in advance and remain undisturbed in the pits. This planning approach aims to provide operational flexibility in the event of unplanned occurrences. However, this strategy also introduces challenges to the drilling and blasting operations when the bottom portion of the blasted muck in the pits begins to freeze, creating a layer of frozen overburden above the drilling pattern. Additional sub-drilling is incorporated into the blast patterns to ensure that the excavators can reach the expected floor grade.

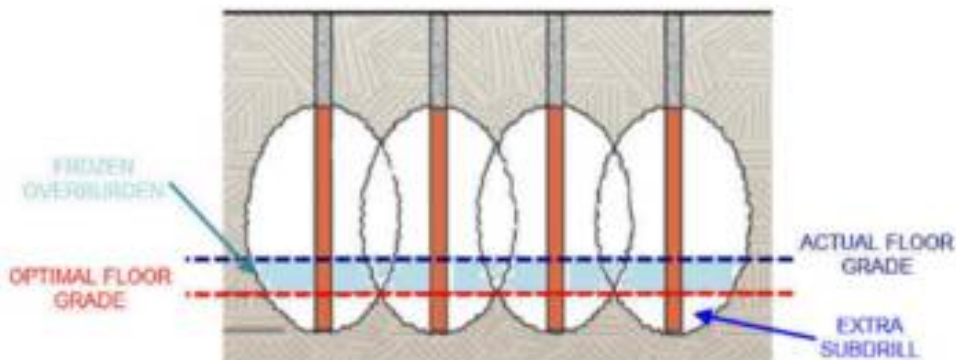


Figure 6. Extra subdrill frozen overburden.

Depending on the mining sequence and weather conditions, frozen overburden can often pose a challenge. Large chunks of frozen overburden can be found in certain areas of the pits, significantly diminishing the quality of the top layer of the muck pile. This reduction in quality can, in turn, decrease the productivity of the mucking equipment. Effective water and sump management

have become crucial in areas where blasted muck remains for an extended period.

4.2 Stockpile and waste dump

The phenomenon of permafrost freezing also affects stockpiled muck when it remains in one location for too long without being moved. When



Figure 7. Frozen overburden.



Figure 8. Borehole liner.

this occurs, there is no equipment on site strong enough to loosen these frozen piles. Consequently, drilling and blasting become the solution to this problem. However, drilling and blasting into frozen muck piles can be challenging and have risks and uncertainties. Key factors to consider include fly rock and vibrations.

Difficulties can arise when drilling and loading frozen piles, such as hole caving and the presence

of cavities or gaps in the holes. Additionally, the proximity of infrastructure to the blast site complicates matters further. Since using packaged emulsion is not an option at this site, it is recommended to load boreholes with liners when blasting near infrastructure.

Additionally, blasting mats can be used to minimise the projectiles created by a blast; however, they can also pose hazards themselves. During a frozen stockpile blast, flying mats have been observed being projected higher and farther than the fly rock itself.

5 WILDLIFE PROTECTION

5.1 Fish and blast monitoring program

The mine's location presents challenges related to wildlife and the regulations in place to protect it. The open pits are situated approximately 600 meters from Nemo and Kangislulik lakes, raising concerns about potential harm to the fish and their habitat caused by the vibrations from blasting activities.

To minimise the impact of blasting on fish and their habitats, a comprehensive blasting program



Figure 9. Flying blasting mats.

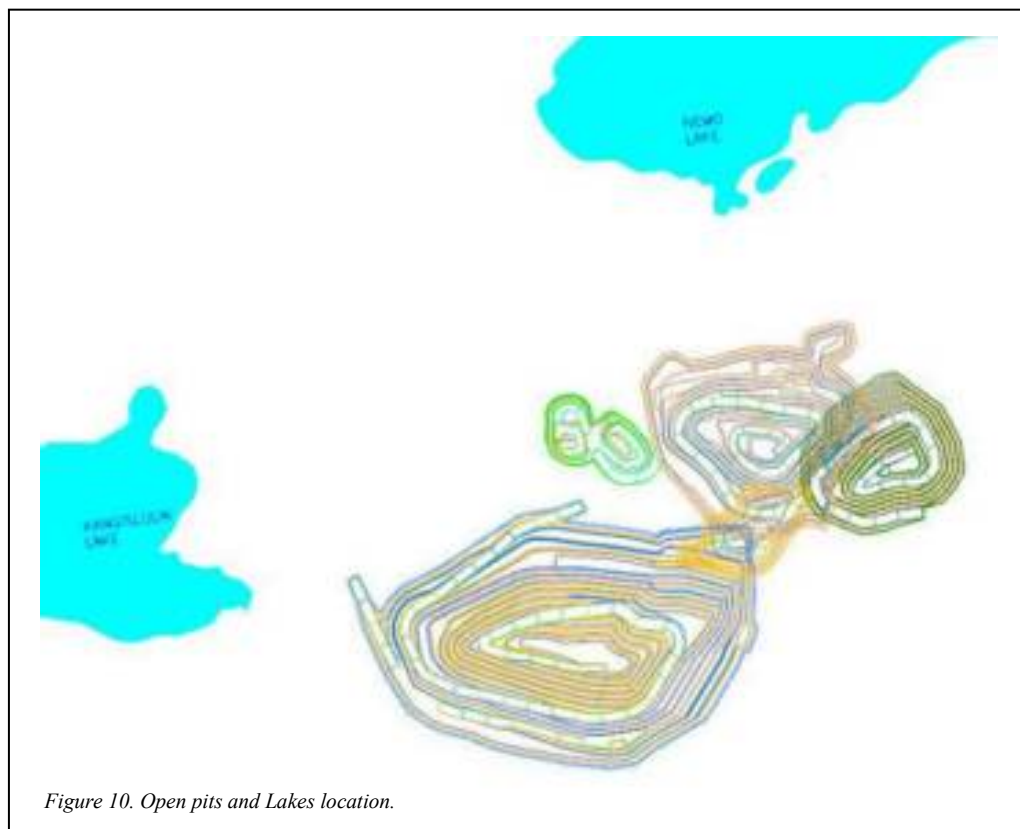


Figure 10. Open pits and Lakes location.

has been developed in consultation with the Department of Fisheries and Oceans Canada (DFO) and the Government of Nunavut. The blast monitoring program outlines several key requirements that Agnico Eagle must consistently adhere to:

- Compliance with guidelines: Follow the guidelines for the use of explosives in or near Canadian Fisheries Waters (Wright & Hopky 1998), as modified by the DFO for northern regions.
- Adherence to guidance documents: Abide by the guidance in the report ‘Monitoring explosive-based winter seismic exploration in waterbodies, NWT 2000-2002’ (Cott & Hanna 2005).
- Monitoring and mitigation plan: Develop a monitoring and mitigation plan in consultation with the DFO and secure DFO approval for the blasting program annually prior to the start of blasting activities.
- Minimisation of ammonium nitrate use: Reduce the use of ammonium nitrate to lessen its effects on receiving water quality.

Regarding vibration limits, the DFO has established specific requirements:

- No explosives should be detonated in or near fish habitats if they are likely to cause an instantaneous pressure change (overpressure) greater than 50 kPa in the swim bladders of fish.
- No explosives that produce or are likely to produce a peak particle velocity (PPV) greater than 13 mm/sec in a spawning bed during the egg incubation period should be detonated.

As part of the blasting monitoring program, it is mandatory to record the vibrations from all blasts in the open pits and underground mines to ensure compliance with these outlined requirements.

5.1.1 Blast monitoring station

To record blast vibrations, a Minimate Blaster or Micromate seismograph equipped with a geophone has been installed near Nemo Lake. A permanent monitoring station anchored on the bedrock was set up at the closest point possible to the lake, and its precise location was surveyed. The geophone,

which records the blast vibrations, is securely attached with a 3/8-inch bolt and nut.



Figure 11. Geophone setup.

For Kangislulik Lake, an IMS xES station with a triaxial monitoring sensor was permanently installed in 2024. This station eliminates the need to install a portable seismograph for every blast since it sends the data remotely to the office.

5.1.2 Data management and interpretation

After each blast, the results are stored in a database, and a report is saved in the library for



Figure 12. IMS Geophone setup.

future reference. The blast monitoring results are analysed, and a blast mitigation plan is implemented immediately if the vibrations or overpressure exceed the permitted limits. The Department of Fisheries and Oceans (DFO) is notified within 72 hours if these levels are exceeded. Additionally, the data are included in the Meadowbank Complex Annual Report, which is submitted annually to the Government of Nunavut (GN), the Nunavut Impact Review Board (NIRB), Environment and Climate Change Canada, and the Nunavut Water Board.

5.1.3 Vibration prediction

The effects of blasting are typically assessed in terms of Peak Particle Velocity (PPV). The U.S. Bureau of Mines has established a relationship between peak particle velocity and scaled distance, which is described by a specific formula (Equation 1):

This formula allows for the estimation of PPV to determine whether it will exceed the established limits before the blasting occurs. A regression model for vibration amplitude is continually developed and updated using a database of recorded events (PPV) from seismographs, which



Figure 13. IMS xES setup.

includes data on the distance from the blast and the charge weight per delay. This model is then used to update the K and b factors.

$$PPV = k \cdot (R/W^{0.5})^b$$

Where:

- PPV = Peak Particle Velocity, mm/s
- R = Distance from blast to point of concern, m
- W = Charge weight per delay, kg
- k = confinement factor – specific to site
- b = site factor

The fish bladder's 50 kPa overpressure limit has been analysed using formulas from ‘Guidelines for the use of explosives in or near Canadian Fisheries waters’ (Wright & Hopky 1998) along with site-specific variables. The calculations demonstrate that a peak particle velocity (PPV) of 42.72 mm/s would be required to exceed the 50 kPa overpressure limit. Consequently, only the 13 mm/s PPV limit is implemented for vibration control.

5.1.4 Mitigation plan

In the event of high predicted vibrations, a mitigation plan is implemented. The first adjustment involves reducing the number of holes blasted on the same delay. This measure aims to minimise the number of explosive blasts occurring simultaneously. If the model continues to predict high vibrations, techniques such as casing and decking are used to decrease the amount of explosive in each hole. If vibrations still exceed acceptable levels, drilling with a smaller diameter

is employed to reduce the explosive quantity per hole. The final option would be to use an emulsion with a lower density. However, this solution is not ideal for the site because the emulsion is produced in batches at the emulsion plant, and any remaining emulsion from the batch cannot be reused for other blasts in the pits.

5.1.5 Caribou migration & wildlife presence

The site is home to various wildlife, including caribou, gray Arctic wolves, Arctic foxes, wolverines, and a variety of birds. Unfortunately, the mines’ infrastructure has been constructed along the route of the caribou migration path. Caribou migrate during the spring and fall, travelling from the boreal forests of Manitoba and Saskatchewan to their calving grounds in the tundra of Nunavut. As part of the blast monitoring program, there are restrictions on blasting when wildlife is nearby. For example, blasting is postponed if caribou are present within a 5 km radius of the blast site.

6 SUMMARY AND CONCLUSION

Drilling and blasting operations within the Arctic Circle, particularly at the Meadowbank Complex in Nunavut, Canada, face unique challenges due to extreme weather, permafrost conditions, logistical constraints, and stringent environmental and regulatory frameworks. This study documents critical aspects of these operations and provides a comprehensive overview of the adaptations necessary to maintain safety, efficiency, and compliance in this remote environment.

Among the most significant technical challenges are preserving the integrity of

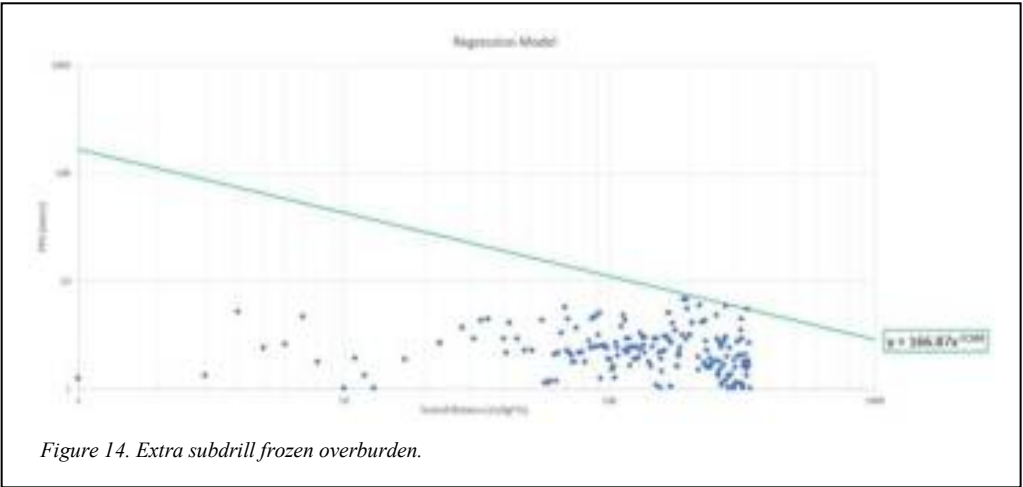


Figure 14. Extra subdrill frozen overburden.

explosives in sub-zero temperatures, managing water infiltration and frozen muck, and executing blast patterns under dry and wet permafrost conditions. Operational adjustments, such as hot loading procedures, rotated heated storage, and real-time decision-making based on environmental conditions, are essential for mitigating risks and ensuring the effectiveness of blasting.

Additionally, environmental stewardship is a fundamental aspect of blasting operations in the Arctic. This includes rigorous monitoring to protect fish habitats from vibration-related impacts and coordinated efforts to avoid disrupting caribou migration. Developing tailored blast prediction models and mitigation plans has proven vital for ensuring regulatory compliance and minimising ecological disturbance.

Overall, the experience at Meadowbank highlights the importance of flexibility, planning, and innovation in Arctic mining. Integrating engineering expertise with a deep respect for environmental and cultural values provides a model for sustainable resource development in northern climates. The lessons learned from this experience enhance operational resilience and offer valuable insights for similar projects across the circumpolar region.

[RETURN TO CONTENTS PAGE](#)

Investigation of sheared boreholes in the Kiirunavaara Mine

H.C. Vikum Sri Mendis & N. Petropoulos

LKAB, Sweden;

N. Petropoulos & C.R. San Miguel

Webrec/ Luleå University of Technology, Sweden

ABSTRACT: Borehole instability remains a major challenge in deep underground mining, affecting both safety and production efficiency. This study focuses on the Kiirunavaara Mine, where production borehole shearing is prevalent due to the large-scale sublevel caving mining method. Two crosscuts (Ø158 and Ø161 at block 15, level 1051) were studied through four borehole filming sessions to investigate shearing behaviour relative to blast front advancement. Results showed high instability rates: 75 % of boreholes in Ø158 and 68 % in Ø161 exhibited shearing. Most sheared boreholes occurred 20 – 45 m from the collar, with 32 % (Ø158) and 20 % (Ø161) showing shearing exceeding 65 %, posing significant detonation failure risks. Borehole shearing increases as blast fronts approach sheared zones, with blockages frequently observed near these areas. The study concludes that shearing is a gradual process influenced by prolonged drilling-to-blasting delays, stress redistribution, and geological structure activation. Redrilling in heavily sheared areas is important to reduce production disruptions.

Keywords: *Sheared boreholes, Borehole filming, Borehole stability, Sublevel caving method, Underground mining.*

1 INTRODUCTION

Sublevel caving is an effective mining method for large-scale ore bodies, characterised by systematic drilling and blasting at different sublevels to create controlled caving for efficient ore extraction. This method requires production boreholes to be drilled well in advance, often months or even years before charging and blasting. While necessary for operational efficiency, this extended timeline poses critical challenges as boreholes face stability issues over time, particularly at the Kiirunavaara Mine in northern Sweden where borehole shearing has emerged as a significant operational concern affecting ore extraction efficiency and safety.

The problem occurs as independent rock

sections along the borehole which can move separately, creating differential displacement that reduces the borehole's cross-sectional area. Figure 1 illustrates such a sheared borehole located 12 m from the blast front, showing two independent rock sections that can move separately, indicating significant structural damage.

This movement can compromise blast performance and potentially cause detonation failure, leading to undesirable outcomes such as hanging roof formation, poor fragmentation, and production disruptions. In the Kiirunavaara Mine, boreholes are 115 mm in diameter and charged with emulsion explosive along with PETN cord. The current approach involves cleaning and redrilling sheared zones with 70 mm diameter, but

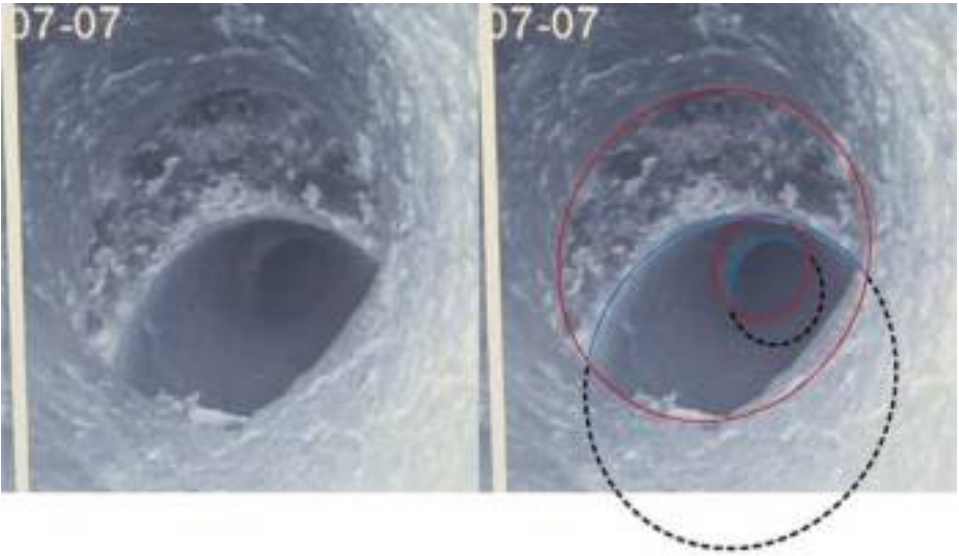


Figure 1. Sheared borehole located 12m (4 rings) away from the blast front.

questions remain about the effectiveness of this procedure.

Previous investigations by Ghosh (2012) at Malmberget Mine and Lundin (2020) at Kiirunavaara Mine have identified sheared boreholes as significant contributors to borehole instability issues. However, limited understanding exists regarding how these sheared boreholes behave as production advances and the blast front approaches the affected zones.

This study aims to address this knowledge gap by investigating the behaviour of sheared boreholes when the blast front approaches the sheared zone, utilising field data collection

campaigns and quantitative analysis to understand the progression patterns.

1.1 Borehole instability mechanisms

Borehole instabilities in sublevel caving operations result from complex interactions between stress redistribution, geological structures, and mining-induced changes. When boreholes are drilled, the removal of rock material disrupts the natural stress field, leading to stress concentration around borehole walls as shown in Figure 2. In deep mining conditions, these stress concentrations can exceed rock strength, causing deformation or

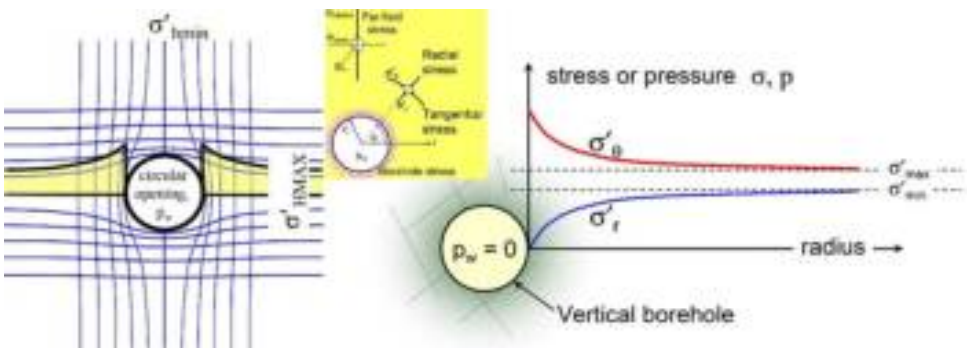
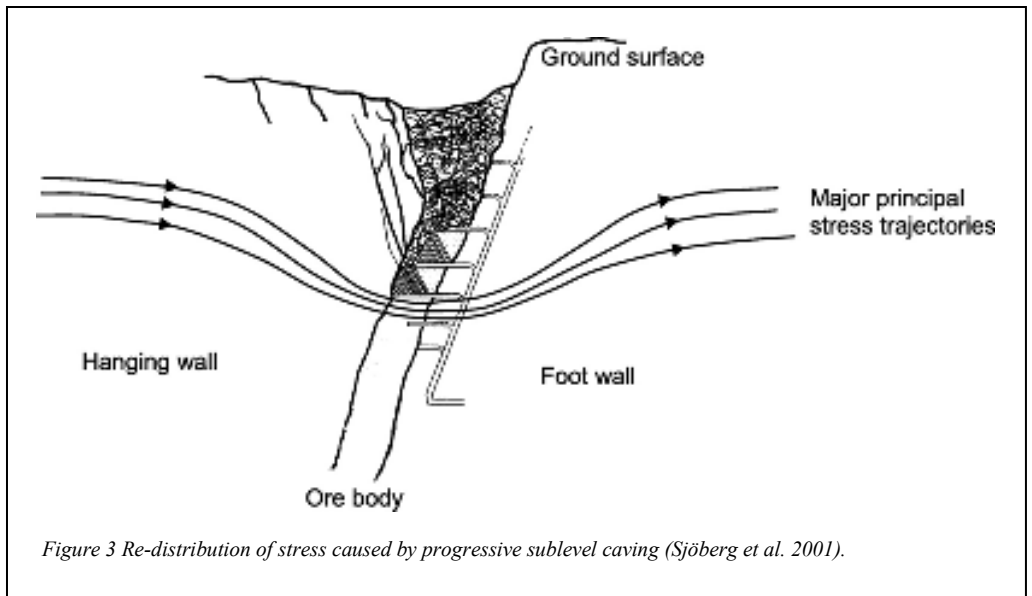


Figure 2. a) Stress trajectories after the redistribution (left), b) Variation of tangential and radial stresses with the distance from the borehole, Dusseault 2005 (right).



fracturing (Dusseault, 2005).

Progressive sublevel caving operations create additional stress redistribution patterns, with major principal stress concentrating below the caving front as demonstrated in Figure 3 by Sjöberg *et al.* (2001). This redistribution can activate pre-existing geological weaknesses, leading to various borehole instabilities including shearing, fracturing, deformation, and blockage.

Sheared boreholes specifically result from movement along discontinuities or fractures that intersect the borehole (Peng & Zhang 2007). These geological features may remain stable

initially but become reactivated due to production activities, stress changes, and seismic events associated with caving operations. Studies at LKAB mines have demonstrated that borehole instabilities are strongly linked to production activities, with movement of geological structures and fractures in the hanging wall induced by progressive caving as mining advances (Ghosh *et al.* 2015). The consequences of such instabilities include detonation failure when borehole cross-sectional area is reduced below critical diameter for reliable explosive performance (Zhang, 2016).

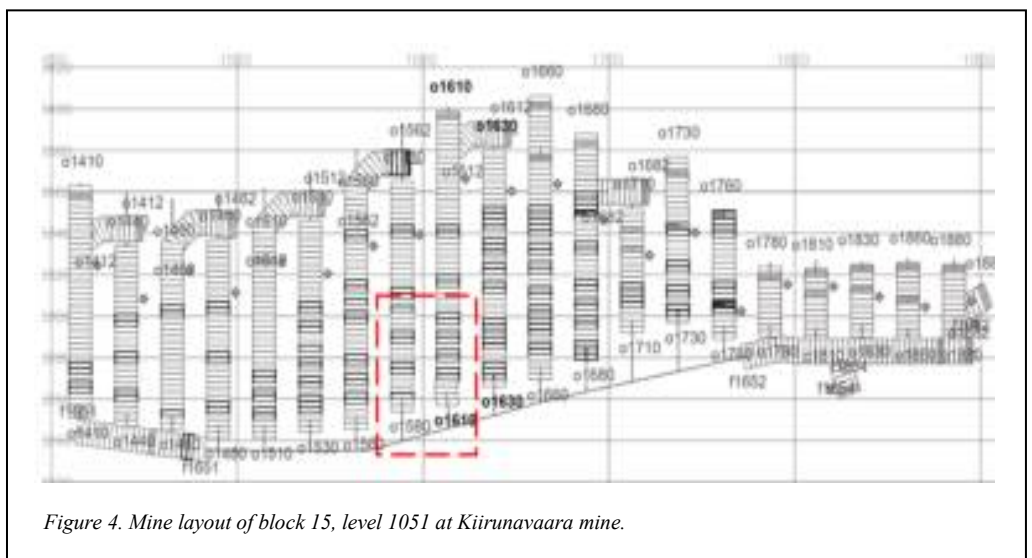




Figure 5(a & b). Instrumentation used for borehole filming: (a) RIDGID SeeSnake camera (left); (b) RIDGID standard reel with the 99 m long push cable (right).

2 METHODOLOGY

2.1 Study area selection

Two crosscuts Ø158 and Ø161 in block 15 on level 1051 (Figure 4) were identified as potentially containing sheared boreholes. The identification process involved consultations with the mine planning team, charging team, and other relevant personnel. The criteria for selecting areas focused on locations with higher occurrence of borehole cleaning and redrilling issues.

Crosscut Ø158 contained 9 blastholes per ring, with 5 holes (H3, H4, H5, H6, H7) filmed in each ring across rings 27 - 34, covering a 21 m span. Crosscut Ø161 contained 8 blastholes per ring,

with 4 holes (H3, H4, H5, H6) filmed in each ring across rings 45 - 52 spanning 18 m in the crosscut. This selection enabled systematic monitoring while minimising disruption to production activities.

2.2 Instrumentation

Borehole filming was conducted using a RIDGID SeeSnake standard camera with a 35 mm diameter camera head protected by a hardened stainless-steel frame. The camera featured built-in sonde for location finding, self-levelling capability, and high-intensity LED lights for enhanced visibility. The high-dynamic range sensor provided excellent image quality for borehole inspection. The camera



Figure 5(c & d). Instrumentation used for borehole filming: (c) RIDGID CS65xr digital monitor (left); (d) LKAB's charging truck (right).

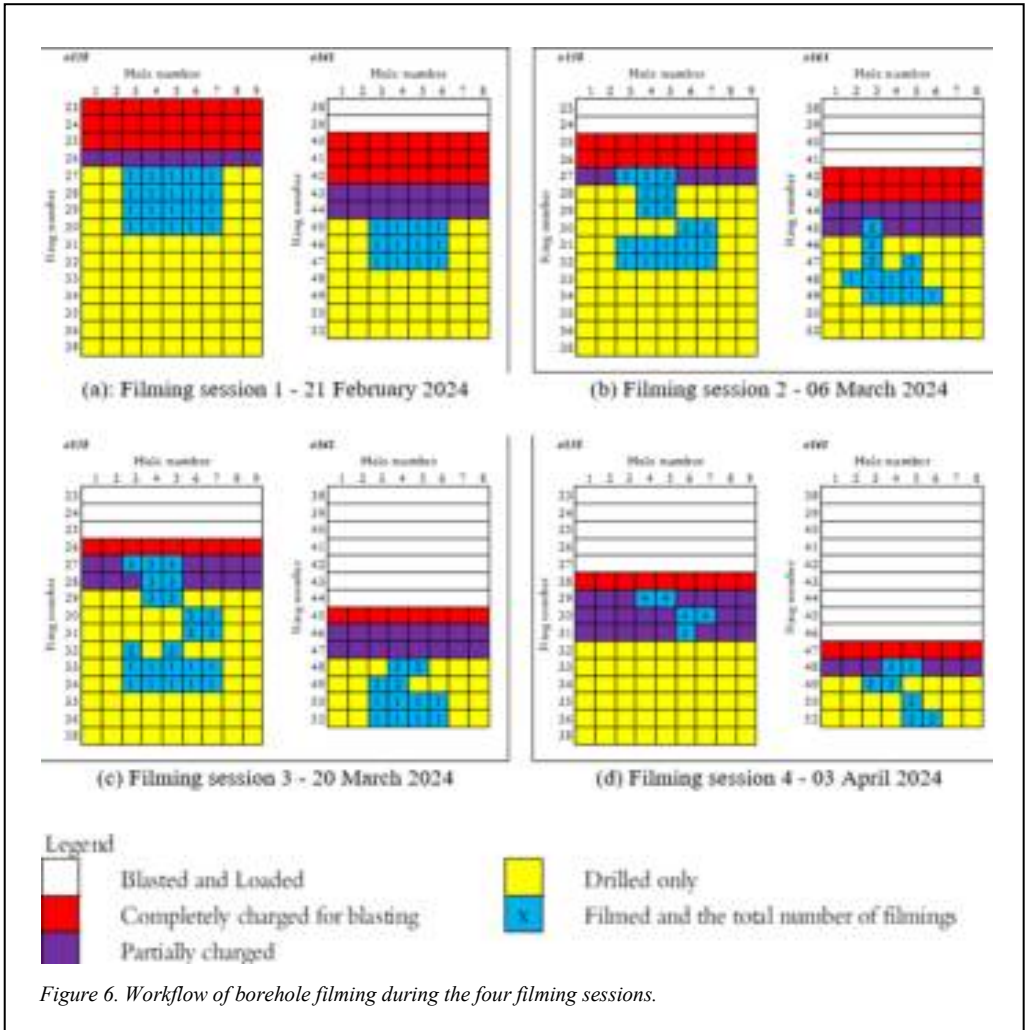


Figure 6. Workflow of borehole filming during the four filming sessions.

was paired with a 99 m long push cable with 11 mm diameter, mounted on a reel with an integral counter for accurate distance measurements. Real-time viewing and data collection were performed using a RIDGID CS65xr digital monitor with USB connectivity for data storage and transfer (RIDGID 2024).

LKAB's charging truck was used to push the camera inside boreholes, with the camera head mounted to the top of the charging hose. Adobe Photoshop CC 2019 was used for data analysis to delineate sheared borehole boundaries and measure sheared areas.

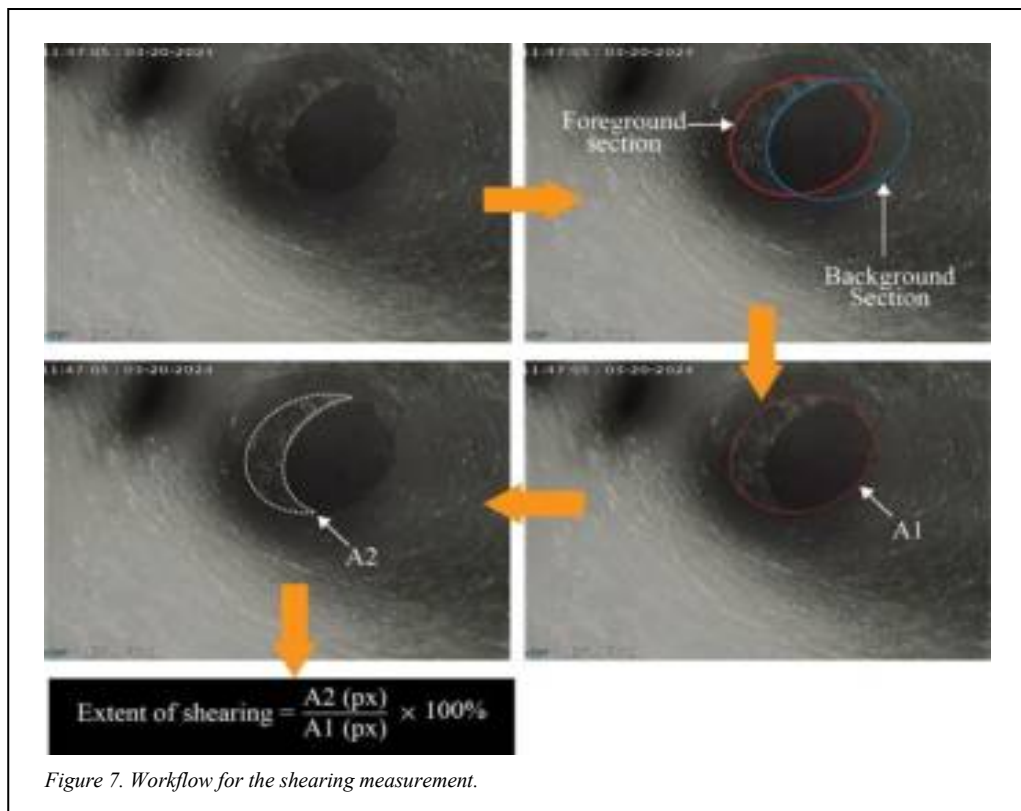
2.3 Data collection methodology

Four filming sessions were conducted from February 21, 2024, to April 3, 2024, totalling

68 holes filmed across both crosscuts in 110 individual recordings. The initial filming session targeted rings nearest to the blast front that were not loaded with explosives to confirm the presence of sheared boreholes in the selected crosscuts.

For each filming session, the camera setup was calibrated by setting the borehole depth to zero at the collar on the digital monitor. Filming was performed using the digital monitor to observe and record anomalies throughout the borehole length.

The data collection workflow involved filming after each production blast once the blasted material was mucked out as shown in Figure 6. Each session started from where the previous session ended, allowing systematic monitoring of sheared borehole behaviour as the blast front approached the sheared zones. Due to production constraints, only two blastholes from each blast



ring were selected for repeated filming in later sessions, with these holes remaining unloaded with explosives as long as possible.

Shearing locations were documented by recording the distance from the collar to each sheared point. These distances were measured using the integral counter of the camera reel and cross-checked with measurements from the charging truck's distance counter to ensure accuracy.

2.4 Borehole shearing measurement

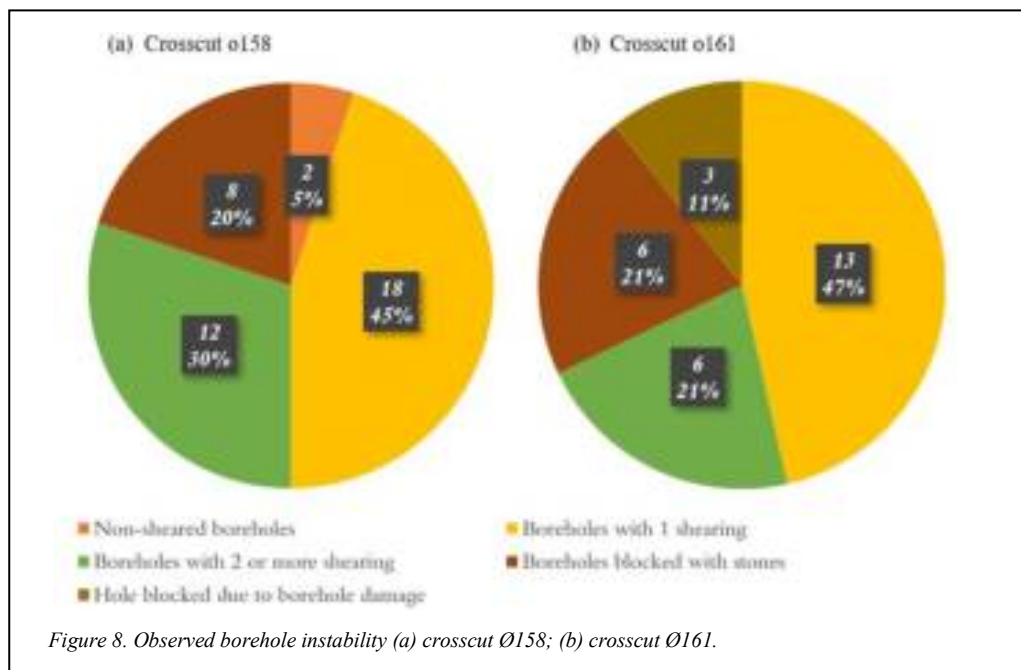
The collected video data was analysed to determine shearing percentages through visual analysis using Adobe Photoshop. Images of sheared locations were captured from borehole videos and imported into Adobe Photoshop. The boundaries of the two halves of the sheared borehole were then delineated, with the foreground section marked with a red line and the background section marked with a blue line as shown in Figure 7. The area of the foreground section (A1) was measured in pixels, and the visible sheared area (A2) was determined by subtracting the

background section from the foreground section. Finally, the extent of shearing was calculated using the formula:

$$\text{Extent of shearing} = \frac{A2 \text{ (px)}}{A1 \text{ (px)}} \times 100\% \quad (1)$$

This percentage-based approach mitigates perceptual inaccuracies caused by the varying angle of the borehole direction, shearing plane orientations, and camera positions within the borehole. The identification of sheared boreholes and boundary delineation relied on visual inspection rather than automated software analysis, providing a practical solution for field conditions.

The methodology allowed for systematic tracking of shearing progression in relation to blast front advancement, providing data on how borehole shearing changes as production blasting approaches sheared zones. This approach enabled the collection of quantitative data on shearing percentages while maintaining operational flexibility within the active mining environment.



3 RESULTS & ANALYSIS

3.1 Borehole instability overview

The investigation revealed extensive borehole instability across both study areas. In crosscut Ø158, filming covered 40 boreholes across 8 rings, where only 2 boreholes (5 %) showed no shearing or blockage, while 75 % exhibited at least one sheared location. Specifically, 45 % had single sheared locations, 30 % had multiple sheared locations, and 20 % were blocked with stones.

Crosscut Ø161 showed even higher instability rates, with all 28 boreholes filmed across 7 rings showing either shearing or blockage. Among these, 68 % had sheared locations (47 % single, 21 % multiple), 21 % were stone-blocked, and 11 % showed damage with cavities and broken rock pieces. Figure 8 shows the boreholes instability distribution observed during this study.

The severity of borehole instabilities prevented complete filming in most cases, with only 22 % of boreholes in Ø158 and 14 % in Ø161 filmed to full depth due to impassable zones caused by shearing or blockages.

3.2 Behaviour of borehole shearing relative to blast front

Boreholes filmed three or more times enabled analysis of shearing behaviour relative to blast

front advancement Figure 9). In crosscut Ø158, 10 sheared locations across 9 boreholes within 4 rings were analysed and the results showed consistent patterns of increased shearing as the blast front approached. Overall shearing increases ranged from 1 % to 22 % over blast front advances of 9 - 15 metres. Despite the general increasing trend, three locations experienced minor decreases (< 2 %) between filming sessions.

Linear regression analysis confirmed strong relationships between blast front distance and shearing percentage, with most locations exhibiting R^2 values exceeding 0.9. All regression slopes were negative, confirming increased shearing as blast fronts approached.

Similarly, crosscut Ø161 analysis examined 5 sheared locations across 4 boreholes within 2 rings (Figure 10). Overall shearing increases ranged from 2 % to 24 % over blast front advances of 15 metres. Two instances showed slight decreases (< 2 %) between filming sessions.

Linear regression analysis showed more variable correlation strengths depending on location with R^2 values ranging from 0.46 to 0.99. All regression slopes were consistently negative, confirming increased shearing as blast fronts approached.

3.3 Borehole shearing distribution

The spatial distribution of the sheared locations

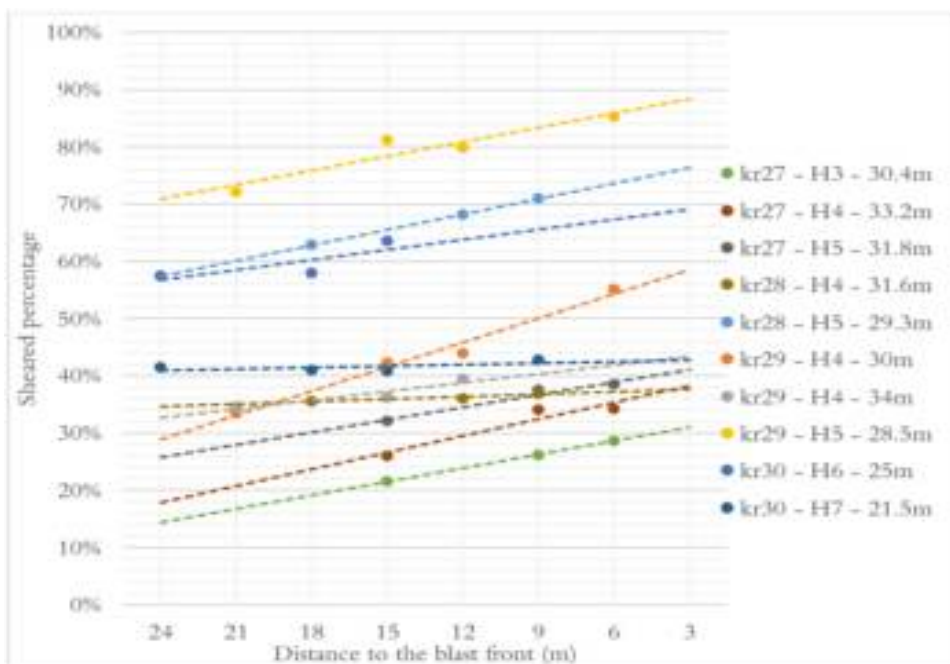


Figure 9. Shearing progression versus blast front distance in crosscut Ø158.

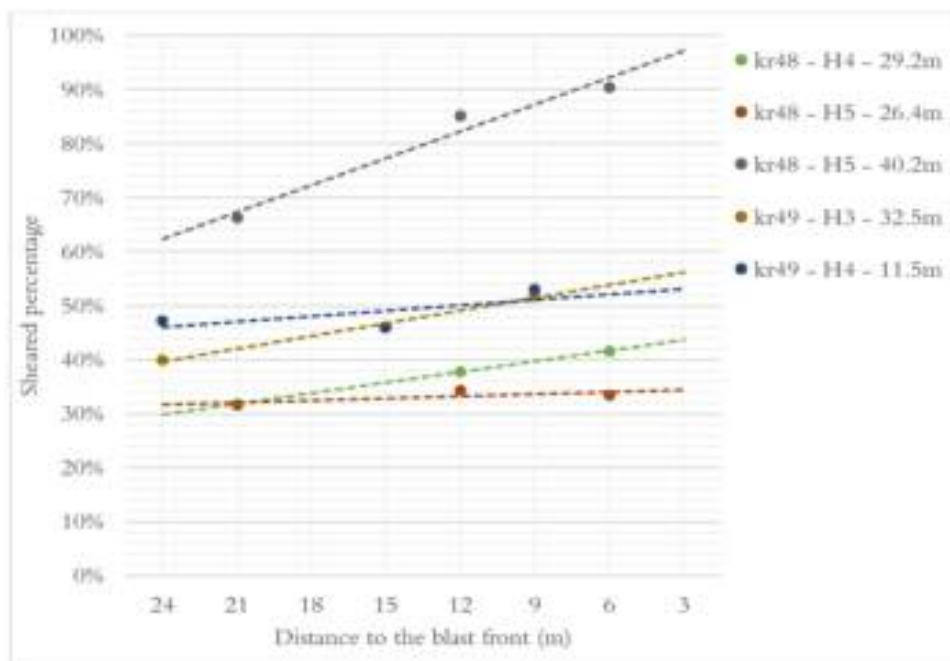


Figure 10. Shearing progression versus blast front distance in crosscut Ø161.

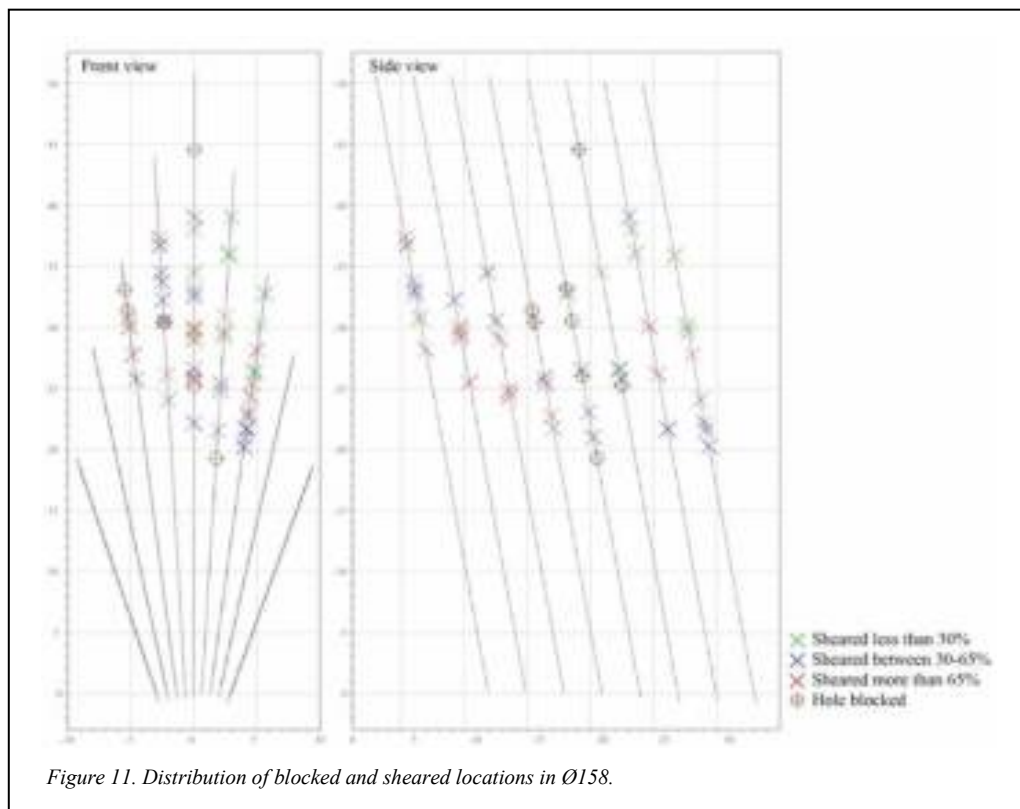


Figure 11. Distribution of blocked and sheared locations in Ø158.

was analysed with them being categorised into three groups based on severity: less than 30 %, 30-65 %, and greater than 65 % shearing. In crosscut Ø158, shearing occurred predominantly between 20-40 m from borehole collars, with heavily sheared zones (> 65 %) distributed between 22-37 m. No clear spatial pattern emerged regarding shearing severity distribution. Among all sheared locations, 25 % showed less than 30 % shearing, 43 % showed 30-65 % shearing, and 32 % showed severe shearing exceeding 65 % (Figure 11).

Crosscut Ø161 showed shearing ranging from 11-45 m from borehole collars, with 88 % concentrated between 24-45 m. Heavily sheared zones (> 65 %) were distributed between 25-42 m. Similar to crosscut Ø158, no clear spatial pattern was evident for shearing extent distribution. Among the sheared locations observed, 32 % showed less than 30 % shearing, 48 % showed 30-65 % shearing, and 20 % showed severe shearing exceeding 65 % (Figure 12).

3.4 Blocked Boreholes

Borehole blockages showed strong spatial

correlation with sheared zones. In crosscut Ø158, blockages occurred only in rings kr30, kr31 and kr32 between 19-45 m from collars. Crosscut Ø161 exhibited more widespread blockage distribution across all rings, ranging from 12-45 m depths.

4 DISCUSSION

The borehole filming conducted in crosscuts Ø158 and Ø161 provided a substantial dataset to investigate borehole shearing behaviour within the Kiirunavaara mine. The rings investigated were located after the connection to the caving of the upper level, where high rates of borehole shearing and blockages were found. These findings align with previous studies by Ghosh (2012) and Lundin (2020), which observed that borehole stability problems tend to increase after connection to upper-level caving, likely due to stress transformations caused by contact with the level above.

The observed high occurrence of borehole instability, with only two boreholes in crosscut Ø158 showing no shearing or blockage, and none in crosscut Ø161 may be related to proximity to a

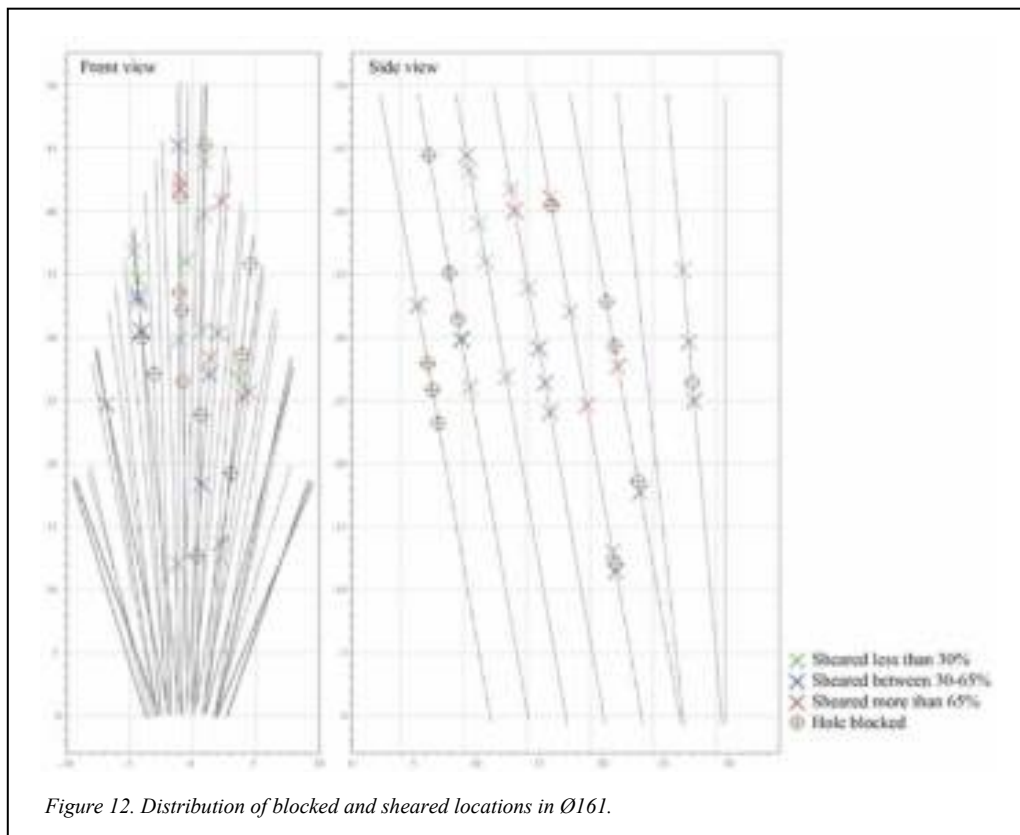


Figure 12. Distribution of blocked and sheared locations in Ø161.

pillar in the mine's northern part, which can lead to high-stress conditions and increased seismic activity. Ghosh (2012) observed similar issues near pillars, where borehole instabilities fluctuate when drifts are divided by pillars.

The relationship between borehole shearing and blast front distance revealed an overall trend of increased shearing as the blast front moved closer to boreholes. This observation was consistent in both crosscuts, aligning with previous studies by Ghosh (2012) and Lundin (2020). However, the majority of sheared locations showed smaller overall increases as the blast front approached. Although overall shearing increased, occasional minor decreases ($< 2\%$) occurred between filming sessions, indicating complex local behaviour patterns that suggest shearing is a gradual process influenced by mass movement within the rock rather than sudden failure.

Linear regression analysis confirmed that decreased distance to blast front corresponds to increased borehole sheared percentage, supported by consistently negative slope coefficients. However, relationship strength varied

significantly, with some locations showing R^2 values exceeding 0.9 while others showed weaker fits, suggesting factors beyond blast front distance significantly influence borehole shearing. No new sheared locations were identified as the blast front moved closer, similar to Lundin's (2020) findings. The absence of new sheared locations, combined with slower increase rates, suggests that shearing is a gradual process, and identified sheared boreholes may have sheared earlier given that drilling was completed over four years before filming campaigns.

The sheared locations were concentrated within specific zones: 20-40 m in Ø158 and 24-45 meters in Ø161 from borehole collars, similar to Lundin's (2020) observations of most shearing between 30-40 m depth. These locations were above the floor level of upper level 1022, where production was completed more than four months before the first filming (Figure 13). This distribution could be attributed to stress redistribution and activation of geological structures resulting from level 1022 caving above. As production was completed at the upper level, rock mass confinement decreased,

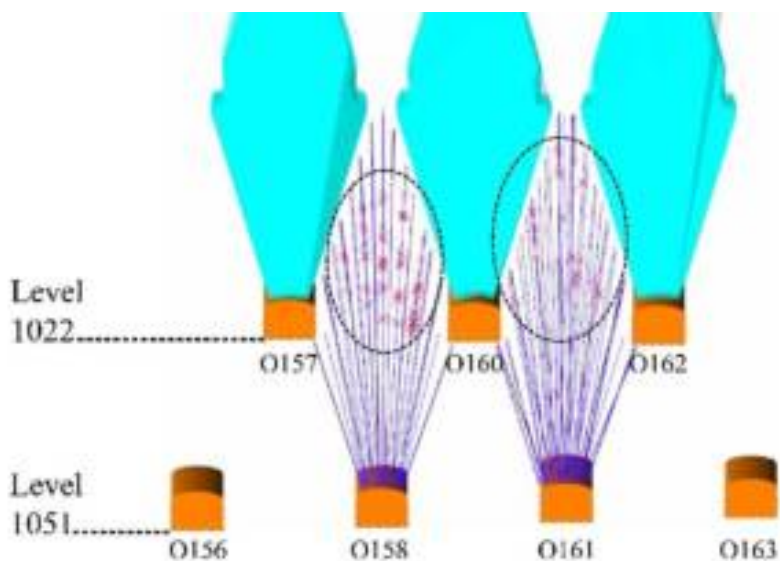


Figure 13. Distribution of sheared boreholes between levels 1022 and 1051.

allowing structures to become reactivated. The shearing patterns correspond to one of the most common geological structures identified in the

mine. These structures are spaced approximately 2-2.5 m apart, dip at 50-70 degrees toward S-SSW, with dip direction of about 200 degrees

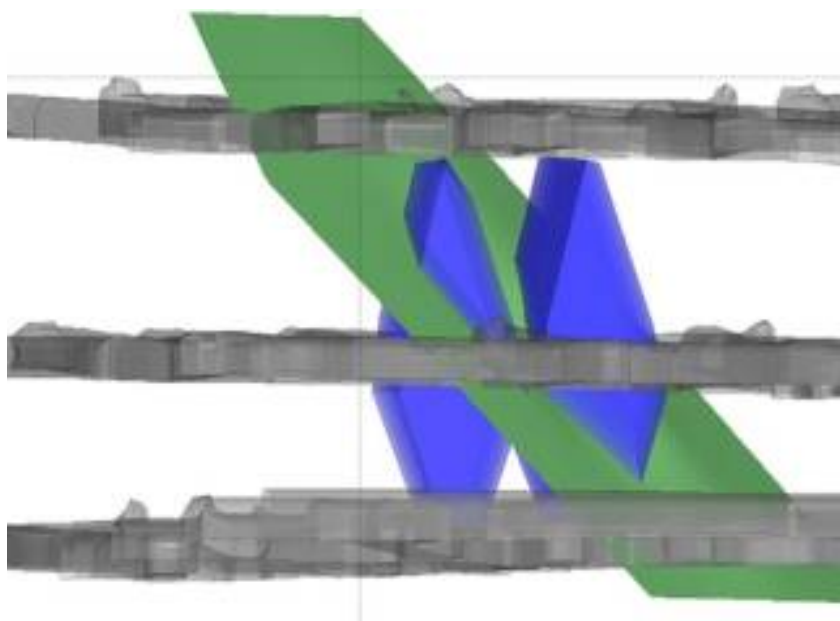


Figure 14. Existing geological structures in block 15, Kiirunavaara mine (Courtesy of Ann Bäckström, LKAB, 2024).

(Figure 14).

Heavily sheared locations with over 65 % shearing are a critical concern as they significantly reduce borehole cross-sectional area and blast performance. Results show 32 % of sheared locations in Ø158 and 20 % in Ø161 exceeded 65 % shearing. These heavily sheared locations must be redrilled to prevent detonation failure and associated operational challenges including poor fragmentation, hanging roof formation, reduced ore recovery, and environmental concerns from undetonated explosives.

Borehole blockages by stones were frequently observed, preventing complete filming. These results align with Lundin (2020), who found blocked boreholes were the primary reason for redrilling requests. Blocked boreholes were located within similar zones as sheared locations: 19-45 m in Ø158 and 12-45 m in Ø161. The correlation between blockage and shearing locations suggests a causal relationship, with evidence showing progression from shearing to stone blocking over time. For example, as shown in Figure 15, Hole 4 of ring 49 in crosscut Ø161 progressed from 79.7 % shearing during the second session to complete stone blockage by the fourth session.

Blocked locations often coincided with sheared zone depths, reinforcing the connection between shearing and subsequent blockages. This suggests that hole blockages observed during this study may result from borehole shearing that occurred earlier, supporting findings by Kangas (2007 as cited by Lundin 2020) where sheared boreholes often had nearby boreholes blocked at similar depths. All observations suggest that the hole blockages observed during this study may result from borehole shearing that occurred earlier.

5 CONCLUSIONS & RECOMMENDATIONS

The high occurrence of borehole instability may have been caused by the completion of production at the level above, existing geological structures in the area, the proximity of these crosscuts to a pillar, and the long delay between drilling and blasting. This investigation provides new insights into the progressive behaviour of sheared boreholes as production advances in sublevel caving operations, establishing that borehole shearing progresses gradually with blast front advancement at modest rates typically below 10 %. The study revealed a direct relationship between shearing and hole blockage by stones. One-third of sheared locations exceed critical 65 % shearing, which is critical for the blast performance. The methodology demonstrates the effectiveness of systematic borehole filming for quantitative shearing assessment in active mining environments, providing a practical tool for ongoing stability monitoring.

The findings encourage minimising drilling-to-blasting delays, given the demonstrated relationship between time and instability development, though operational constraints in active mining environments present challenges. Future investigations should focus on recently drilled boreholes to capture early-stage shearing development patterns currently missing from this long-term post-drilling study. Integration of seismic monitoring data with borehole filming results may reveal triggering mechanisms for shearing initiation. Numerical modelling studies focusing on stress redistribution effects from upper-level caving could enhance predictive capabilities for identifying susceptible zones before drilling occurs. Enhanced filming

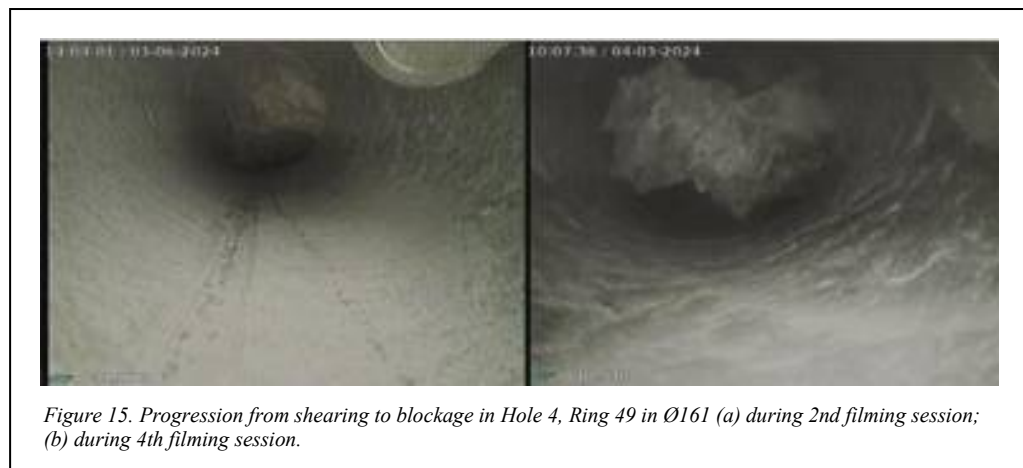


Figure 15. Progression from shearing to blockage in Hole 4, Ring 49 in Ø161 (a) during 2nd filming session; (b) during 4th filming session.

equipment, including camera centralisers and improved measurement systems, would increase data accuracy.

6 ACKNOWLEDGMENTS

The authors thank LKAB for funding this research and providing access to the Kiirunavaara Mine. This research was conducted as part of a master's thesis at Luleå University of Technology. Special appreciation goes to Johanna Johansson and Frida Kyrö for fieldwork assistance.

REFERENCES

- Dusseauult, M. 2005. Borehole stability: Stresses and how analysis is done. <https://isrm.net/download/media.file.8add23a5af653626.31333031333231373232335f645f626f7265686f6c655f73746162696c69742e706466.pdf>.
- Ghosh, R. 2012. Investigation of borehole instability in Malmberget underground mine. [Thesis, Luleå Tekniska Universitet]. <http://www.diva-portal.org/smash/get/diva2:1020595/FULLTEXT02.pdf>.
- Ghosh, R., Zhang, Z. & Nyberg, U. 2015. Borehole instability in Malmberget underground mine. *Rock Mechanics and Rock Engineering*, 48(4), 1731–1736. <https://doi.org/10.1007/s00603-014-0638-1>.
- Kangas, J. 2007. Orsaker till hållrensnings Malmberget. [Internal report]. LKAB Kiruna: Not published.
- Lundin, M. 2020. Chargeability and borehole stability in production borehole—Kiirunavaara mine [Thesis, Luleå Tekniska Universitet]. <https://www.diva-portal.org/smash/get/diva2:1426613/FULLTEXT01.pdf>.
- Peng, S. & Zhang, J. 2007. Engineering geology for underground rocks. *Springer Science & Business Media*.
- RIDGID. 2024 February 14. SeeSnake standard camera with TruSense | RIDGID Tools. <https://www.ridgid.com/us/en/seesnake-standard-camera-with-trusense>.
- Sjöberg, J., Lundman, P. & Nordlund, E. 2001. Analys och prognos av utfall i bergschakt, KUJ 1045. Slutrapport. *LKAB internal report*.
- Zhang, Z.X. 2016. Rock fracture and blasting: Theory and applications. *Butterworth-Heinemann*.

RETURN TO CONTENTS PAGE

Effect of detonation propagation and blast confinement on vibration and VOD in hard rock

T. Hänninen

Forcit Consulting Oy & Aalto University, Helsinki, Finland

M. Aarnio

Forcit Consulting Oy, Helsinki, Finland

ABSTRACT: Blast induced vibrations are important considering environmental impacts and public acceptance of mining operations. MIC, distance and bedrock properties are commonly used for estimating vibration levels but they depend on other factors too. This study combines two field tests regarding detonation propagation direction and blast confinement due to stemming. In the first test setup hole charges were initiated from different ends. Results showed that PPV is the highest when detonation propagates towards a monitoring point but no clear differences were noticed when the monitoring point was adjacent to charges. VOD behaved normally regardless of the initiation method. In the second test setup half of the holes were filled with gravel after charging while the other half were left without stemming so the blasting gases could erupt and expand freely out of the hole. Both hole types produced fairly similar PPV and blast confinement due to stemming did not increase the values.

1 INTRODUCTION

During a bedrock blast multiple factors affect peak particle velocity (PPV) and other factors of blast induced vibrations. Traditionally the maximum instantaneous charge (MIC), the distance between blast and monitoring point and properties of bedrock are considered to play key roles.

Nevertheless, PPV variation between blasts cannot be fully explained with the traditional factors so the vibration must be relative to other things too. For example, direction of the detonation propagation and blast confinement may influence the vibrations regarding to research studies and practical observations during mining and rock excavation operations.

Nowadays blasting is often executed in more and more complex environments such as city

centres and near to hospitals and laboratory facilities. Deeper understanding of blast induced vibrations and all tools to control them will help to execute demanding projects in a safe and efficient manner.

1.1 Direction of the detonation propagation

Detonation of a cylinder charge has a beginning and an end. Traditionally a hole charge is ignited with a single detonator located near to the bottom of the drill hole. The hole charge can also be ignited from the top or from the middle of the charge. In poor rock conditions or long holes, detonation of the whole hole charge is often ensured with additional detonators. Because of delay dispersion and tolerance of pyrotechnically delayed detonators, igniting two or more

detonators in the same explosive charge was not precise in the past. Electronic detonators make it possible to initiate hole charges with more control and with two or more simultaneous detonation fronts. Due to controlled dual initiation the fronts collide and so-called shock wave collision occurs. This method has stated to enhance fragmentation (Zhang 2016) and to possibly increase vibration levels, according to Leng *et al.* (2021).

Effect of the detonation propagation on vibration has two possible approaches. During one open pit blasting project PPV-levels were higher when an upright cylinder hole charge and a vibration monitoring point were horizontally adjacent, i.e. side by side compared to another monitoring point which was straight below the same hole charge (Vuolio & Halonen 2010). On the other hand, in the simulations of Liu *et al.* (2016) vibration strain was the highest when detonation propagated straight towards the monitoring point and the lowest when the propagation direction was opposite. Strain levels were somewhere between when monitoring point was side by side with the charge. Gao *et al.* (2019) and Triviño *et al.* (2012) made similar findings.

1.2 Blast confinement

Confinement's influence on vibration is an uncertain and controversial topic. Simply put, bedrock blast confinement means all factors which resist the explosion reaction and breakage of bedrock. For example, too small powder factor due to big burden or drilling errors leads to higher blast confinement. High confinement can result in poor fragmentation, undesired movement of the blasted rock or to uneven bottom. Dead pressing effect of a deep hole charge due to high ground water pressure prior to a blast can also be considered as a confinement issue so misfires can result from confinement too.

Although blast confinement can lead to various blasting problems, does it affect blast induced vibrations? Generally, there are two basic approaches to the problem and a clear answer to the issue cannot be found between research studies or practical experience. According to the first assumption the high level of confinement would lead to higher vibration levels because less explosive's chemical energy is turning to rock breakage and movement and bigger amount to seismic energy. The second assumption is based on the idea that detonation shock wave and vibration waves are already long gone before the rock does or does not start to break and move, i.e.

the poor fragmentation and movement of the bedrock wouldn't affect the vibration levels.

Previous common-sense explanation of different approaches to the possible connection between blast confinement and vibration levels is highly simplified and the mechanism behind the phenomenon seems to be much more complex. Olsson, Niklasson & Beyglou (2017) studied confinement with two different methods. In the first part vibration results from successful tunnel blasts were compared to misfired and re-blasted tunnel rounds. In the second part the effect of burden was studied while measuring vibrations from single charges with different burden lengths. Neither of the study parts showed any clear correlation between confinement level and blast induced vibrations.

1.3 Structure, goals and preliminary challenges of the study

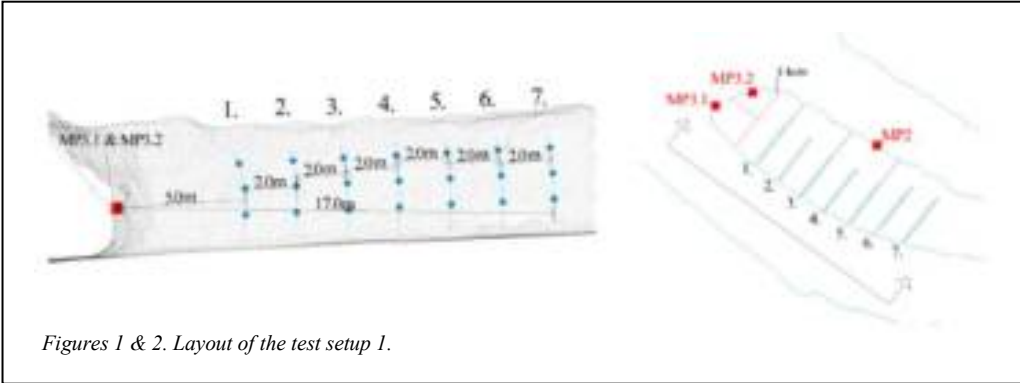
This study compiles two different test setups which were organised in the same area of a hard rock underground mine in Finland. Emulsion explosive cartridges and electronic detonators were used to build up hole charges which were blasted one at a time and monitored as single events. Vibrations were monitored during both test setups and velocity of detonation (VOD) was measured during the first test setup. Research questions were following:

- Does the detonation propagation direction affect PPV-levels?
- How does the VOD behave and is there a risk of a dead pressing effect if the detonation propagates towards the solid hole bottom?
- Does the formation and expansion of blasting gases affect PPV-levels?

The main problem while studying vibrations and bedrock blasting in general is the heterogeneous structure of bedrock. Due to fractures and joints vibration propagation and attenuation can vary a lot to different directions from the blast. Mounting of monitoring equipment is crucial too. Results from two monitoring points can vary even if the geophones are mounted the same way to the same structure and very close to each other. The differences while comparing results from different monitoring points may lead to problems especially in research projects.

2 FIELD TEST SETUPS

Meeting the goals and tackling the problems mentioned in the previous chapter built up the



baseline for field test setup design. Two main guidelines were adopted:

- Hole charges and monitoring points locate so that vibration waves from comparable blasts propagate through the same bedrock.
- Results only from a single monitoring point (three-axial geophone) are compared together at the same time.

Regarding previous guidelines also distances between comparable blasts and monitoring points were similar as well as explosive product and MIC within each test setup.

Hole charges located in solid bedrock so burden and spacing can be considered infinite. Although no bedrock movement was expected, fracturing of rock mass near the hole charge should happen which was noted during planning. Distances between hole charges were designed to avoid fracturing from adjacent charges. During the blasting sequence the furthest hole charges from the monitoring point were ignited first so generated bedrock fractures wouldn't affect the vibrations from subsequent hole charges.

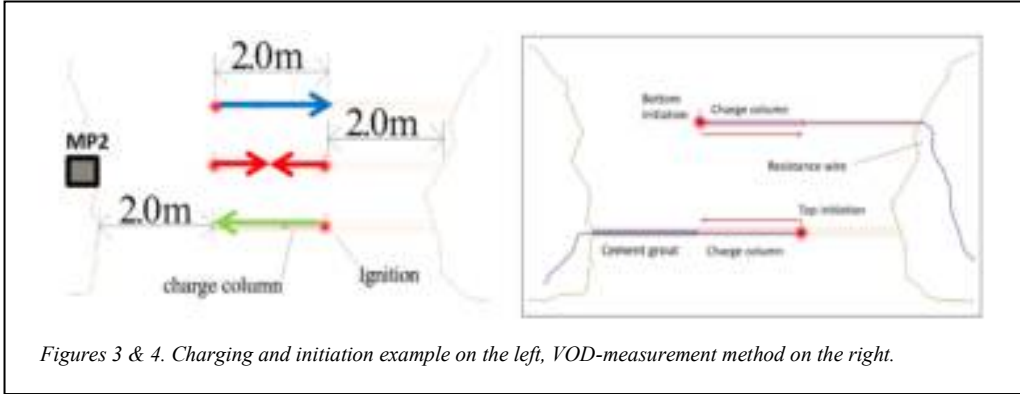
Both test setups began with calibration blasts which helped to verify explosive amounts, suitable delays and monitoring equipment settings.

2.1 Test setup 1 – Horizontal hole charges in a pillar

In the first test setup three different initiation methods were studied. One third of the charges were ignited from the bottom (bottom initiation), the second third were ignited from the top (top initiation) and remaining holes were ignited simultaneously from both ends of the charge (dual initiation).

To meet the design guidelines above, a pillar between two underground mine drifts was chosen for the test location. Seven horizontal three-hole sets were drilled and distance between main monitoring point MP3.1 and all the holes inside one set stayed constant (see Figure 1). Three additional monitoring points were installed too. MP2 located behind the pillar and MP3.2 next to MP3.1 as shown in Figure 2. MP1 was mounted to a lower level of the mine. MP1 and MP3.2 acted as reference monitoring points to verify results.

Each hole in one set was charged with two-metre-long emulsion pipe charge of about 440 grams of explosive but the initiation method varied as in Figure 3. In addition to vibration monitoring, VOD of bottom and top initiated charges were



Figures 3 & 4. Charging and initiation example on the left, VOD-measurement method on the right.



Figures 5 & 6. Layout of the test setup 2.

measured from three different hole sets with the resistance wire method. To enable VOD-measurement for top initiated charges one hole of each set 1, 3 and 5 was drilled through the pillar. Regarding these holes the resistance wire went through the pillar and the hole ends were blocked with about two-meter-long cement grouting prior the blast (see Figure 4).

Hole sets blasted from the furthest one (calibration hole set 7) towards the monitoring point MP3.1 so the hole set 1 blasted last. The T-hole showing in figure 2 is outside the scope of this study.

2.2 Test setup 2 – Vertical hole charges with and without stemming

Blast confinement is a complex phenomenon and it is related to many different factors (see section 1.2). This study concentrated on blasting gases forming from vertical hole charges in solid bedrock with infinite burden. Half of the holes were stemmed with several metres of gravel after charging while the other half blasted without stemming so the blasting gases could expand and erupt freely out from the holes and confinement was considered smaller.

In this test setup six two-hole sets were drilled. Like in test setup 1 holes in each set had a similar distance to the main monitoring point (Geophone no.1 in figure 5). A reference monitoring point was located on lower mining level due to verifying reasons.

Every hole contained one bottom initiated 530 mm long emulsion explosive cartridge including 1.25 kg of explosive. Each hole set included one stemmed hole and one hole without stemming. Drilling and charging designs are presented in figures 5-7. Due to heavier charge, distances between holes and to monitoring point needed to

be bigger than in test setup 1. Hole sets located 4 meters from each other and distance between hole set L0 and main monitoring point was 60.9 m. Blasting started from calibration hole set L0 and finished with hole set L5.

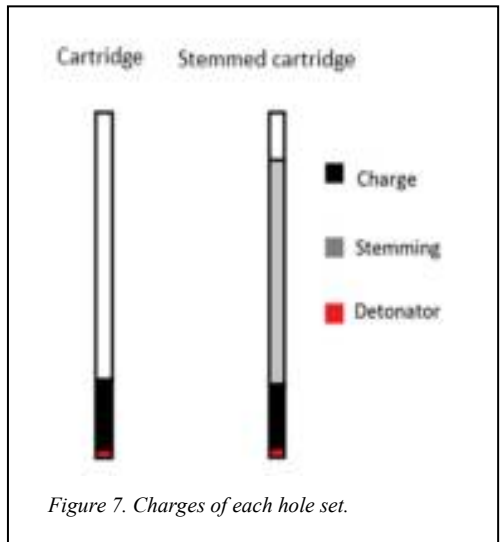


Figure 7. Charges of each hole set.

3 RESULTS

Since PPV is widely used factor regarding vibrations, the PPV-values were the main concern in this study. In the following results PPV-values are presented three-axially in vertical (V), longitudinal (L) and transverse (T) components. VOD was measured from six holes in test setup 1.

Both test setups included calibration hole sets which were not designed as a part of the actual study and therefore they are ignored in the results. The same applies with the reference monitoring points which were used to verify blast events and other data from the main monitoring points.

Table 1. Test setup 1 –
Results of hole sets 1-6 from MP3.1.

Hole ID	Initiation type	PPV (mm/s)		
		V	L	T
1_1	B	14.6	18.1	17.7
1_2	S	7.4	16.5	7.5
1_3	T	9.0	19.5	16.3
2_1	T	7.6	13.5	8.4
2_2	B	12.2	12.6	12.6
2_3	S	6.7	21.4	12.6
3_1	S	2.8	5.9	2.4
3_2	B	4.2	5.6	5.5
3_3	T	5.6	7.6	3.2
4_1	B	4.5	9.6	4.2
4_2	T	5.7	11.0	6.0
4_3	S	2.0	4.2	1.8
5_1	B	5.6	9.5	5.9
5_2	S	4.3	10.4	3.7
5_3	T	4.8	6.8	4.6
6_1	S	9.8	10.5	8.9
6_2	T	10.3	11.1	10.2
6_3	B	4.0	4.7	3.9

Table 2. Test setup 1 –
Results of hole sets 1-6 from MP2.

Hole ID	Initiation type	PPV (mm/s)		
		V	L	T
1_1	B	2.9	13.1	5.9
1_2	S	2.8	8.3	4.0
1_3	T	2.6	11.0	4.8
2_1	T	5.5	13.6	6.9
2_2	B	4.0	13.8	3.9
2_3	S	6.8	13.0	4.1
3_1	S	2.2	2.8	2.1
3_2	B	2.7	5.2	2.7
3_3	T	4.4	12.5	7.1
4_1	B	5.4	6.8	8.2
4_2	T	8.4	6.7	9.7
4_3	S	2.8	3.9	3.8
5_1	B	3.9	5.1	6.4
5_2	S	5.4	7.6	8.2
5_3	T	7.2	7.4	8.3
6_1	S	7.9	12.1	15.0
6_2	T	4.7	11.9	11.8
6_3	B	3.0	4.8	3.5

Table 3. Test setup 1 –
Results from VOD-measurements.

Initiation type	Hole ID	VOD (m/s)	average VOD (m/s)
B	5_1	4600	4700
	1_1	4600	
T	3_3	4800	4750
	1_3	4700	

Table 4. Test setup 2 –
Results of hole sets 1-5 from Geophone no. 1.

Hole ID	Stemming	PPV (mm/s)		
		V	L	T
L1.1	No	2.8	8.0	2.4
L1.2	Yes	2.8	7.9	2.7
L2.1	No	3.2	10.4	3.2
L2.2	Yes	3.8	10.7	3.0
L3.1	No	3.8	10.1	3.3
L3.2	Yes	3.3	11.6	3.5
L4.1	No	5.1	17.2	6.0
L4.2	Yes	4.6	13.8	5.1
L5.1	No	5.3	16.6	7.4
L5.2	Yes	5.9	11.5	4.9

3.1 Effect of the initiation method and direction

Test setup 1 provided results regarding initiation method and direction. Main monitoring points were MP3.1 and MP2 and their PPV-results from hole sets 1-6 are presented in Tables 1 & 2. In the column 'Initiation type' B means bottom initiated, T means top initiated and S means dual initiated (shock wave collision effect). Every hole set includes one result from each initiation method.

VOD was measured from three bottom initiated (B) holes and from three top initiated (T) holes. Both initiation methods included one failed measurement so data was acquired from four holes as shown in Table 3.

3.2 Effect of the blast confinement due to stemming

Blast confinement was studied in test setup 2 by adjusting the stemming level, i.e. the blasting gases eruption and expansion out of the hole. Results from hole sets 1-5 are presented in Table 4. No stemming means that the hole included only the charge. 'Yes' in the stemming column means that the stemming length was several metres and the hole was practically full of gravel.

4 ANALYSIS

Due to reasons stated in previous chapters, in both test setups holes of one hole set were designed as similar as possible considering explosive amounts as well as distances and bedrock properties between holes and monitoring point. With caution

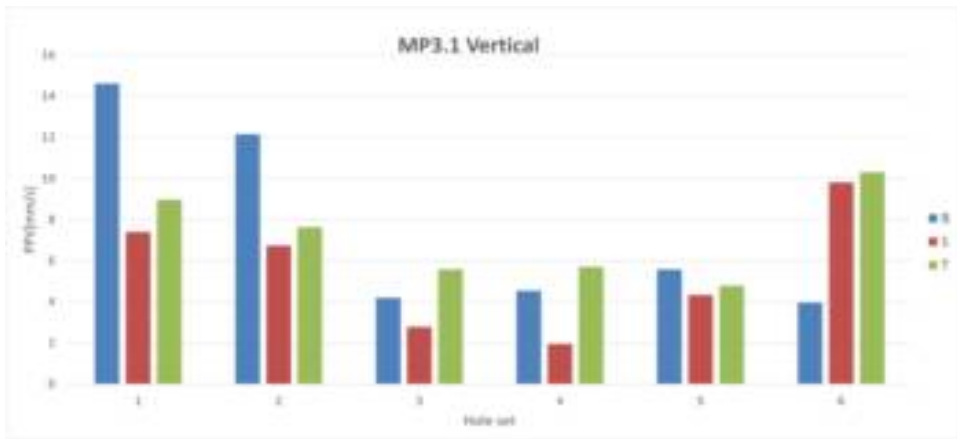


Figure 8. PPV-vertical comparison from MP3.1.

hole sets can be compared also together but this study concentrated mainly on comparing results from one hole set at a time. Different monitoring components (V, L and T) were compared separately too.

4.1 Effect of the initiation method and direction

Results from the test setup 1 were analysed in three different phases. Firstly, the PPV-values from MP3.1 were compared. Figures 8, 9 & 10 show that PPV from different initiation methods varied a lot. Depending on observed hole set and component, all initiation methods provided maximum and minimum values. According to the

results initiation method does not have obvious or uniform impact on the PPV when the monitoring point is adjacent to the hole charges. The finding is logical for bottom and top initiated charges but slightly surprising for dual initiated charge with the shock wave collision effect.

The second analysis phase concentrated on MP2 results from hole sets 3, 4 and 5. Because the holes of these sets were almost parallel with MP2 (see figures 1 and 2), detonation of each top initiated charge propagated towards the monitoring point while the detonation direction of each bottom initiated charge was opposite. PPV-values of the vertical component in Figure 11 show that top initiated charges produced the highest

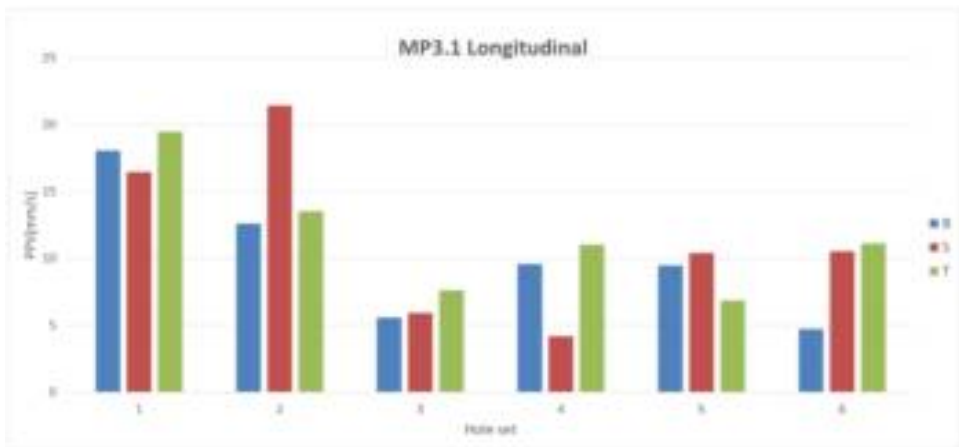


Figure 9. PPV-longitudinal comparison from MP3.1.

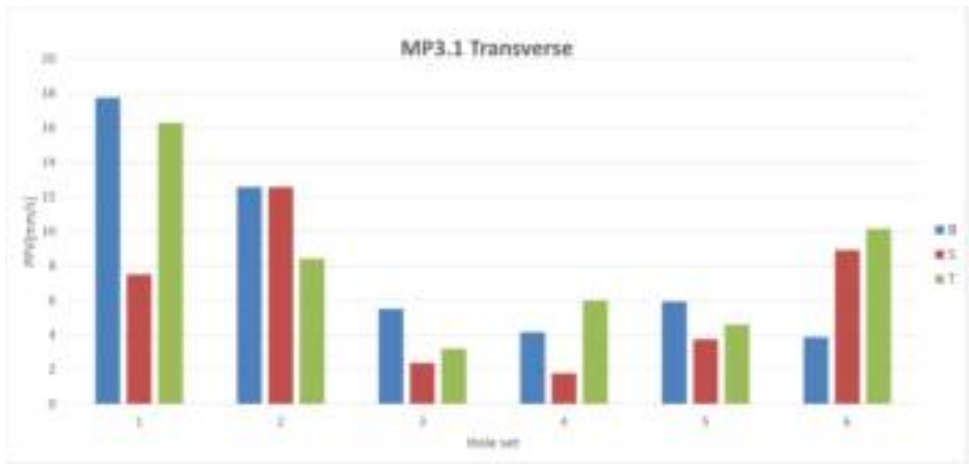


Figure 10. PPV-transverse comparison from MP3.1.

PPV-vertical in all three-hole sets and other components back up this finding quite unanimously. In this study PPV was highest when the detonation propagated towards the monitoring point, i.e. detonation seems to accumulate vibration to its propagating direction. Results between bottom and dual initiation were mixed which may result from superposition of shock wave collision effect.

During the third phase VOD behaviour was analysed. According to measurements, VOD was, on average, the same with top and bottom initiated charges and the measured velocities met the manufacturer specification (Forcit 2021). VOD-graphs were also similar regardless of the initiation method. Top initiated charges detonated successfully from the start to the end, i.e. no signs of dead pressing effect was noticed when the

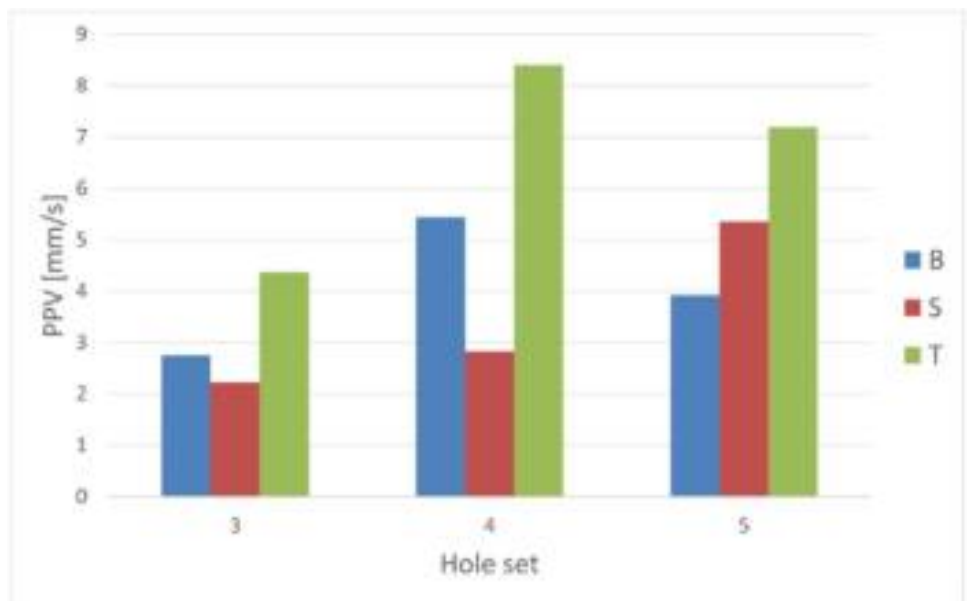


Figure 11. PPV-vertical comparison from MP2 – Hole sets 3-5.

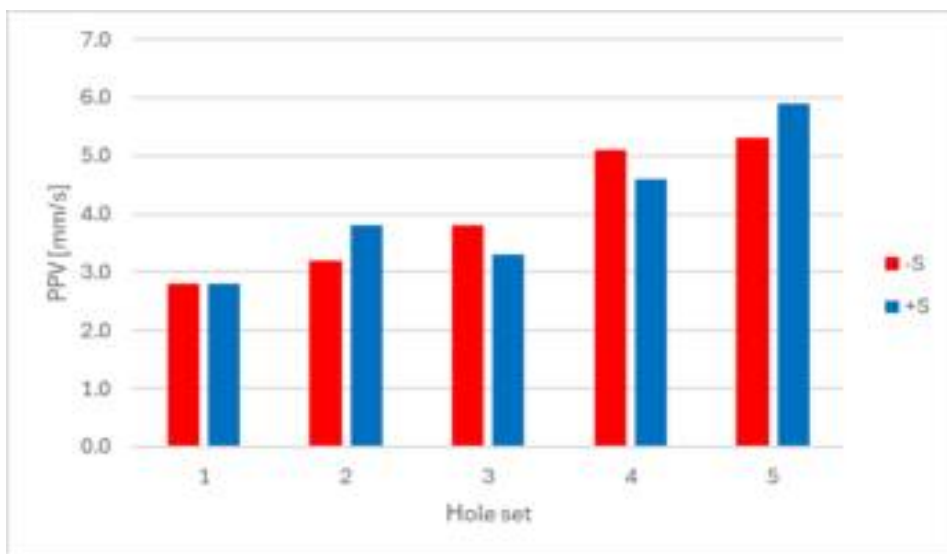


Figure 12. PPV-vertical comparisons from test setup 2.

detonation propagated towards the hole bottom.

4.2 Effect of the blast confinement due to stemming

Results from test setup 2 were combined in three charts according to different monitoring components (see Figures 12, 13 & 14). In the

charts -S means holes without stemming and +S holes with stemming. Regardless of the component no obvious or uniform correlation can be found between holes with or without stemming. Especially in hole sets 1-3 the results are close to each other while hole sets 4 and 5 have more dispersion. In some sets the hole without stemming produced higher PPV although the

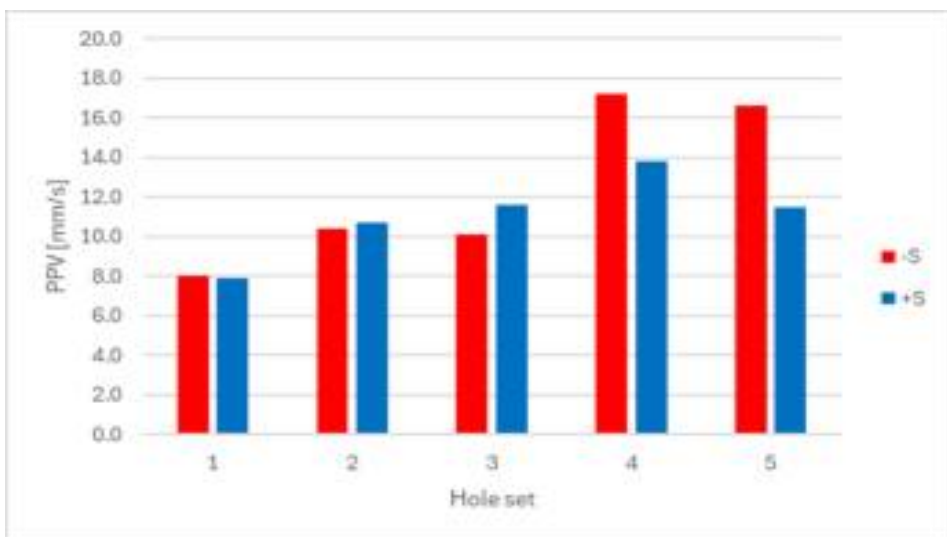


Figure 13. PPV-longitudinal comparisons from test setup 2.

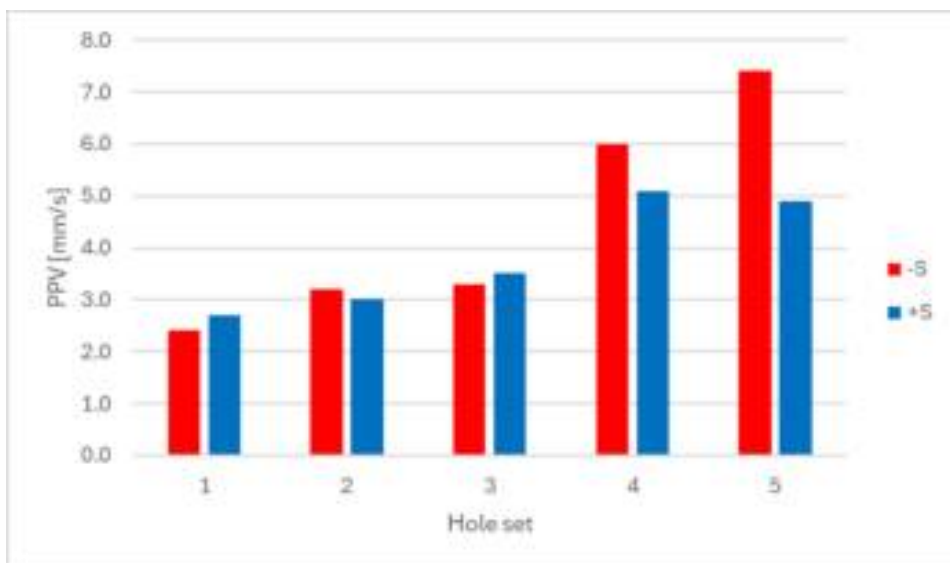


Figure 14. PPV-transverse comparisons from test setup 2.

blasting gases could expand and erupt freely out of the designated hole. According to the results stemming does not increase the PPV.

5 CONCLUSIONS

Based on the results of this study PPV values are higher when the detonation of a cylinder explosive charge propagates towards the vibration monitoring point compared to the situation when the detonation propagates to the opposite direction. However, if the monitoring point is adjacent (side to side) with the charge, no uniform difference in PPV seems to exist between top, bottom or dual initiated charges. Blast confinement generated by stemming does not increase the PPV of a single hole charge according to this study.

The VOD-measurements showed no difference in the behavior of bottom and top initiated charges. Detonation seems to propagate similarly towards the bottom of the hole compared to the collar point of the hole.

As mentioned before, it is crucial to control differences in bedrock properties, in measuring distances as well as in monitor mountings when studying blast induced vibrations. Furthermore, wave reflections in the tunnel structures, possible water in holes, high frequencies due to partially short distances, inaccuracies in explosive products and air blast phenomenon may generate errors in such test setups. Although previous variables were

taken into account in this study and their impact was minimised, they cannot be totally ignored while evaluating the reliability of the results.

The study provided good basic knowledge about how different factors affect the blast induced vibrations and velocity of detonation. Although findings in such laboratory scale tests cannot be adopted directly to bigger hole charges and multiple hole blasts with short delays, fundamentals stay the same while moving to industrial scale.

REFERENCES

- Forcit 2021. Tuoteluettelo, louhintaräjähteet.
- Gao, Q., Lu, W., Leng, Z., Yang, Z., Zhang, Y. & Hu, H. 2019. Effect of initiation location within blasthole on blast vibration field and its mechanism. *In: Shock and Vibration Volume 2019*: 1–18. Hindawi.
- Leng, Z., Sun, J., Lu, W., Xie, X., Jia, Y., Zhou, G. & Chen, M. 2021. Mechanism of the in-hole detonation wave interactions in dual initiation with electronic detonators in bench blasting operation. *In: Computers and Geotechnics Volume 129*: 103873. Elsevier.
- Liu, K., Li, X., Li, X., Yao, Z., Shu, Z. & Yuan, M. 2016. Characteristics and mechanisms of strain waves generated in rock by cylindrical

- explosive charges. In: *Journal of Central South University* 23: 2951–2957. Berlin Heidelberg: Central South University Press and Springer-Verlag.
- Olsson, M., Niklasson, B. & Beyglou, A. 2017. The influence of charge confinement on vibration level in blasting. In *R. Holmberg et al (ed.), Stockholm Conference Proceedings 2017, European Federation of Explosives Engineers: 83–93.*
- Triviño, L.F., Mohanty, B. & Milkereit, B. 2012. Seismic waveforms from explosive sources located in boreholes and initiated in different directions. In: *Journal of Applied Geophysics* Volume 87: 81–93. Elsevier.
- Vuolio, R. & Halonen, T. 2010. Räjätystyöt. Helsinki: Suomen Rakennusmedia Oy.
- Zhang, Z. 2016. Rock fracture and blasting: Theory and Applications. *Oxford: Butterworth-Heinemann Elsevier*

[RETURN TO CONTENTS PAGE](#)

Snowy 2.0 pumped-hydro project - Marica upstream surge shaft

R. Domotor

TS Engineer, Orica, Australia

M. Parkinson

Marica, Future Generation Joint Venture

ABSTRACT: Snowy 2.0 is a major pumped-hydro expansion of Australia's Snowy Mountains Hydroelectric Scheme, engineered with the aim of delivering clean and reliable energy storage and generation for the next 100 years. The Scheme is owned and administrated by Snowy Hydro Limited, the owner of the Snowy 2.0 project. The upstream surge shaft 'Marica' located high in the Snowy Mountains, is a 28 m diameter, 250 m deep shaft connecting to the Snowy 2.0 headrace tunnel, which brings water from the upper Tantangara Reservoir to the 2,200 megawatts power station, 800 m underground. Upon completion, the Marica surge shaft at the Snowy 2.0 pumped hydro scheme will rank among the largest diameter operational shafts worldwide.

The initial excavation down to 90 m from the surface level was completed by 36 t excavators using traditional mechanical excavation techniques, with the spoil hoisted to the surface in kibbles. At this point, drilling and blasting commenced to safely and efficiently increase the advance rate of the shaft. The major challenge with blasting was maintaining the integrity of the rock bolt and shotcrete temporary support structures. An initial vibration limit of 100 mm/s PPV limit was imposed, with the closest monitor placed 5 m above the bench level on the shaft wall. To comply with the 100 mm/s limit, only very small charge weights could be used to break the rock. This required complex blast designs of nearly 300 holes to blast the initial 2 m deep shot.

After several blasts were fired, the vibration results and the effect on ground support were reviewed. The blast design was extended to a 5 m deep cut, using two decks of explosives fired on separate delays. After further successful blasts without adverse impacts on the ground support, the project team was comfortable increasing the vibration limit to 250 mm/s. This allowed the blast to be extended to 7.5 m deep across the 28 m shaft, using decked and fully charged holes. Novel timing sequences ensured single-charge arrivals at the monitoring locations.

The advance rate was of significant interest to the project, and the design team considered innovative blasting technologies such as wireless blasting systems to increase the advance rate. This paper discusses the evolution of the blast design and the techniques investigated and used to improve fragmentation and project advance rate while managing blast vibration.

1 THE SNOWY 2.0 PUMPED-HYDRO PROJECT – MARICA SHAFT

The Snowy 2.0 project is a significant expansion of the Snowy Mountains Hydroelectric Scheme in

Australia, representing the country's largest committed renewable energy initiative. With a design life of 100 years, Snowy 2.0 will support the nation's secure and stable transition to a low-carbon emissions future while minimising

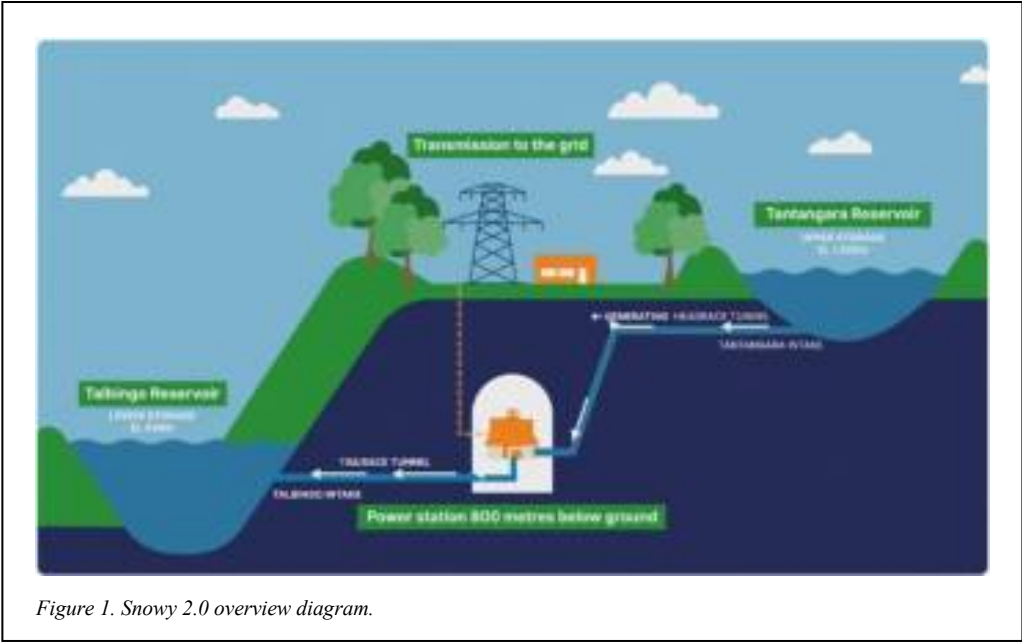


Figure 1. Snowy 2.0 overview diagram.

costs for consumers. Water will be pumped to the upper dam during periods of surplus renewable energy production and low energy demand, then released back to the lower dam to generate electricity when demand is high. This project will provide flexible, on-demand power by reusing water in a loop and maximising the efficiency of renewables by using excess solar and wind energy to pump water to the upper dam.

The project entails the construction of 27 kilometres of underground tunnels linking the Tantangara and Talbingo dams and an underground power station. It aims to deliver 2,200 megawatts of capacity and approximately 350,000 megawatt-hours of large-scale storage to the national electricity market, playing a pivotal role in Australia’s transition to a low-carbon emissions future and diversifying the national energy portfolio.

The upstream surge shaft Marica situated high in the Snowy Mountains, is a 28 metre-diameter, 250 metres-deep shaft that connects to the Snowy 2.0 headrace tunnel, which transports water from the upper Tantangara Reservoir to the 2,200 megawatts power station located 800 metres underground. The surge shaft serves multiple purposes: it will store water and provide vacuum relief when the power station starts up, and it will absorb the momentum of the water and provide pressure relief to the headrace tunnel. Once completed, the Marica surge shaft at the Snowy

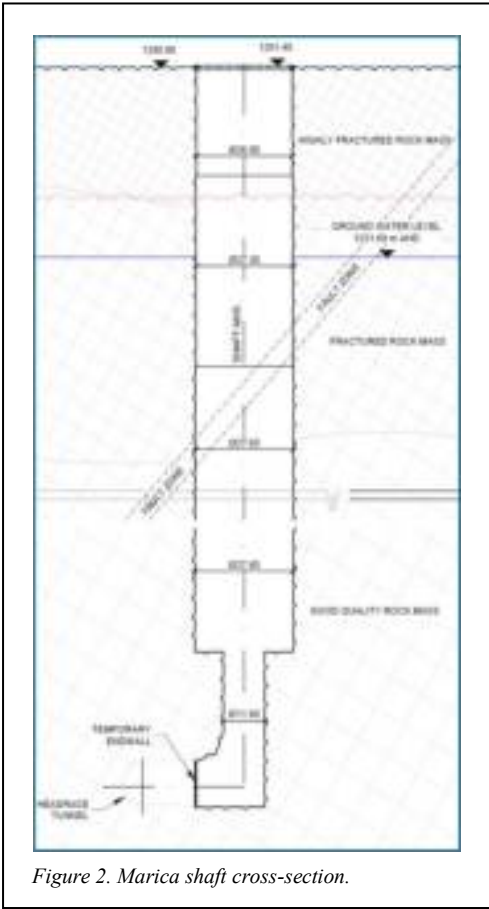
2.0 pumped hydro scheme will rank globally among the largest diameter operational shafts.

2 FUTURE GENERATION JOINT VENTURE

Future Generation Joint Venture (FGJV) is the principal contractor for the Snowy 2.0 pumped-hydro expansion project in New South Wales, Australia. Formed specifically for this nationally significant infrastructure development, FGJV brings together the combined expertise of three major construction and engineering firms: Webuild (Italy), Clough (Australia), and Lane Construction (United States). The joint venture leverages international best practices in tunnelling, hydroelectric infrastructure, and large-scale civil works to deliver one of Australia’s most complex renewable energy projects. FGJV operates with a strong focus on innovation, safety, and environmental sustainability, while also supporting regional economic development through local employment and community engagement.

FGJV engaged Orica to deliver blast design and execution services across the project’s four operational sites. The first blast at the Lobs Hole Tunnels took place in September 2022, and initial blasting activities at the Marica shaft began in November 2023. With a committed milestone for full power generation by December 2028, the upstream surge shaft (USS) must be excavated, lined, and commissioned on time to support the

wet commissioning activities scheduled for May 2028. The shaft is a critical component of the hydropower system, providing vacuum relief during power station start-up and absorbing pressure surges within the headrace tunnel. Therefore, completing the Marica shaft is vital for successful wet commissioning activities.



3 SHAFT BLASTING

The Marica Shaft was initially excavated in 2 m levels, with a maximum unsupported depth dependent on the designated Excavation Support Class (ESC), typically 2.5 m. This ESC depth limit was necessary to ensure that ground pressures were appropriately managed and the walls were supported to minimise deformation. Once excavated, ground support was installed according to the designated ESC. The ground support included cement-grouted rock bolts and layers of fibre-reinforced shotcrete.

The shaft’s excavation commenced without using drill and blast techniques. The upper levels of the shaft were characterised by highly fractured ground, which allowed the mechanical excavators and hammers to reach each 2m level with relative ease. As the shaft moved past this fractured zone, excavation became more difficult and progress slowed. When the shaft reached level 26 at a depth of 50 metres from the surface, the first blast was attempted.

Considering the civil infrastructure, a blast vibration limit of 100 mm/s Peak Particle Velocity (PPV) was initially enforced. This measurement was taken on the shaft wall, 5 m above the bench level. To manage the constraint of the vibration limit, the blast hole charge weights had to be carefully designed, ensuring that holes around the perimeter contained less than 1 kg of explosives by design.

The Level 26 blast was designed to blast one quarter of the shaft within a zone of more competent rock. It consisted of 87 holes of 76 mm diameter, arranged in a 1.6 m x 1.6 m square pattern, drilled to a depth of 3 m. The shot was designed to use 55 mm diameter Senatel™ Powerfrag packaged explosives in the body, with 26 mm diameter Senatel™ Powersplit packaged presplit explosives in the perimeter holes. The collars of the perimeter holes were offset from the wall profile by 1.0 m and angled with a lookout of 18 degrees from vertical. The intention of this blast aimed to test the blasting parameters, vibration predictions and facilitate excavation in the area.

Once drilling was complete, the loading crew attempted to load the blast. During the QA/QC checks, they discovered that many holes had collapsed due to ground conditions. The ground was competent enough to slow the excavation, but insufficient to support the blastholes. The decision was made to attempt loading the remaining viable holes. As loading commenced, the crew realised that the collars would collapse and block the holes during the loading process. Consequently, the Shotfirer cancelled the loading of the shot. The crew removed the explosives from the holes that could be unloaded, leaving approximately eight holes to be fired to safely dispose of the explosives. The holes were fired in November 2023, resulting in a safe yet unremarkable outcome.

Mechanical excavation continued for several months. The variable ground conditions presented challenges in achieving the desired production advance rates. However, the operation had to



Figure 3. Drilling the blast.

persist as the ground was not yet competent enough to maintain the integrity of the blastholes. Several more blasts were designed in anticipation of execution; however, it wasn't until June 2024 that the ground conditions improved sufficiently for drill and blast to be viable.

The next attempt to blast occurred at level 46, approximately 92 m down the shaft. The blast was designed to cover the entire shaft, except for a small area to the north-east where dewatering pumps needed to continue operating due to water inflow rates. The design comprised 275 76 mm diameter blastholes and 8 102 mm diameter reamers arranged in a revolver cut configuration. All holes were designed to be 2.2 m deep to reach the 2 m level. A traditional box and diamond hole configuration was employed for the burn cut, transitioning to rings of holes spaced 1.7 m apart on each ring. The burden on each ring varied from 1.0 m around the cut, progressively increasing to a maximum of 1.8 m as subsequent rings expanded geometrically.

Like the previous design, the shot was loaded with 55 mm diameter Senatel™ Powerfrag packaged explosives in the body, along with 26 mm diameter Senatel™ Powersplit packaged presplit explosives in the perimeter holes. The collars of the perimeter holes were offset from the wall profile by 1.0 m and angled with a lookout of 20 degrees. With the 100 mm/s PPV limit on the walls, the charge weights were reduced as the rows of holes moved closer to the perimeter of the blast.

There was no seed hole or prior blast vibration data to reference, so the Australian Standard vibration prediction constants of K (1140) and b (1.6) were used.

The blast was successfully fired on 19 June 2024. The post-blast inspection indicated good fragmentation; however, the maximum PPV recorded was 123.40 mm/s, which was above the 100 mm/s limit.

4 BLAST DESIGN REVIEW AND EXECUTION

Following the successful initial blast, the site was eager to enhance the advance rate for the subsequent blast. FGJV sought permission to increase the blast depth to 5.0 m for the next blast across Levels 49 and 50. This adjustment would enable them to effectively blast two levels by implementing a strategy that involves excavating one level first and installing ground support before excavating the second. The blasted ground would support the walls of the lower level while ground support was installed on the upper level.

This introduced some complexity because fully loading a deeper hole would increase the Maximum Instantaneous Charge (MIC), which, in turn, would increase the vibration. Revised site constants for vibration prediction were calculated based on the results of the first blast, with an MIC of 3.2 kg predicting less than 100 mm/s PPV in the perimeter holes using a 26 mm Powersplit™



Figure 4. Two drills operating in the shaft.

decoupled charge. This would allow the perimeter holes to be loaded with one deck of explosives. However, the production holes loaded with 55 mm Senatel™ would require two decks of explosives separated by an inert stemming deck, as the MIC of a single column of explosives would far exceed the vibration limit.

This would influence fragmentation, as an inert stemming deck of 2.5 m separating the charges was necessary to contain the energy of the top deck and prevent transmitted pressure from potentially damaging the explosives in the bottom deck. Orica discussed this with FGJV and explained that using deck loading to control vibration might compromise fragmentation.

FGJV requested permission to increase the vibration limit to accommodate full column charges. If approved, this change would boost the advance rate by reducing the drilling intensity for

the same powder factor, thereby speeding up drilling time. Independent third-party advice on increasing the limit was sought, however, for the moment, they remained at 100 mm/s. The 5 m deep shot was successfully fired on the 6th of August 2024. The recorded vibration was 73.14 mm/s, comfortably below the 100 mm/s limit.

FGJV conveyed to Orica that the advance rate was critical to the project schedule and requested they consider any possible design improvements achievable within the site constraints. The request included designing a ‘Triple Bench’ shot at 7.5 m deep and another double bench shot 5 m deep, enabling FGJV to explore options and assess the impact on production cycle times. Orica also suggested transitioning from packaged explosives to bulk emulsion, which would enhance cycle time through improved loading efficiencies.



Figure 5. Loading shot at Level 46 in Marica shaft.

Orica explored whether new technology could be employed to improve production cycle times and accelerate advance. With blast vibration being a function of MIC and distance, larger charges positioned farther away (deeper) from the monitors would produce the same vibration as smaller charges located closer. The advantage of using larger charges is that the patterns can be expanded for the same powder factor. This means an increase in cycle time by reducing drilling and loading time.

Going too deep introduces the risk of over-confining the bottom of the blast, which can

cause it to freeze and lead to significant excavation delays. However, if it were possible to ‘stack’ fully loaded blasts one on top of the other and fire the bottom blast after the first had been excavated and avoid over-confinement, this would significantly reduce cycle times through a process of single-pass drilling and loading multiple blasts that could then be fired off as required by the project. This novel concept has not been previously implemented, and it is only possible using a wireless initiation system.

While Orica explored this concept, the subsequent blast at level 51 was designed using

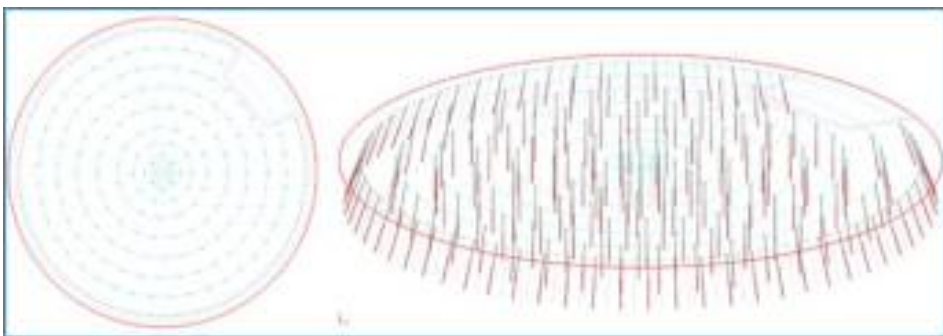


Figure 6. Level 46 shot plan and aspect.

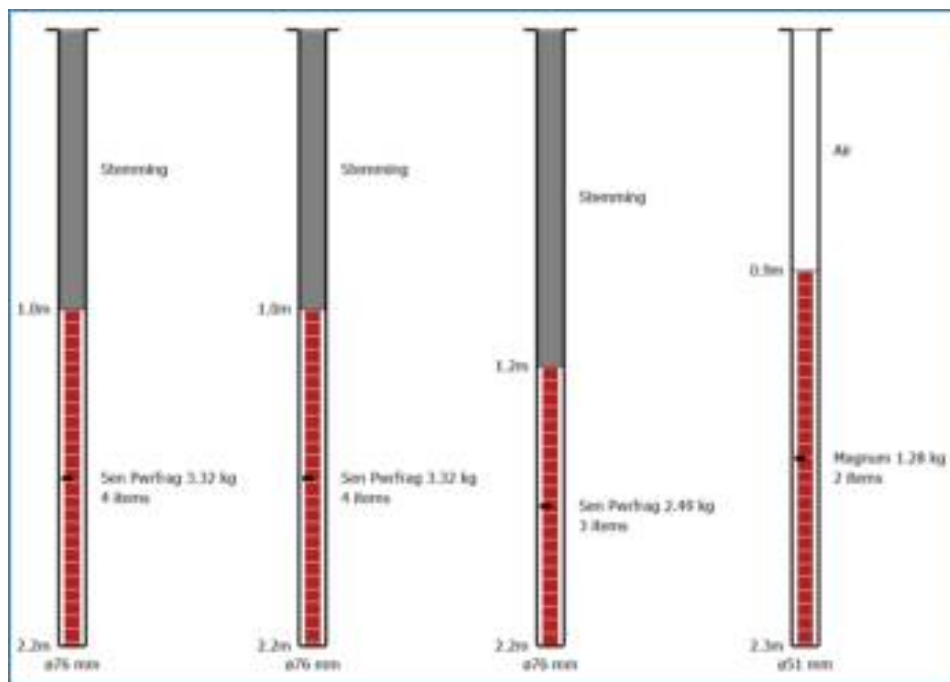


Figure 7. Level 46 design loading of cut, production, easer and perimeter holes.

bulk emulsion in the production holes. This blast was set to be 5 m deep and was the first loaded with emulsion. Since the cut holes were the farthest from the monitors, they were fully loaded as a single column of explosives. However, the remaining production holes were loaded with two decks of explosives separated by an inert stemming deck and fired on separate delays. The perimeter holes were loaded with 26 mm Powersplit™. The second 5 m deep shot was successfully fired on the 28th of August 2024. The recorded vibration was 79.98 mm/s, comfortably below the 100 mm/s limit.

Blasting continued while discussions progressed regarding the potential to increase the vibration limits. FGJV also sought approval to deepen the shots from 5 m to 7.5 m. Orica also spoke with FGJV and SHL about the opportunity to use the WebGen™ (WG) wireless initiation system to trial the new concept of single-pass drilling and loading multiple blasts. With interest in the technology from both parties, Orica mobilised WG equipment to the site to conduct suitability testing. This included signal transmission surveys and in-hole blast pressure testing.

The level 53 shot was a duplicate of the shot on level 51 and was fired on 15 September 2024. Initially, the fragmentation appeared good, however, the outcome was soon found to be less favourable than previous shots. The bottom of the shot around the perimeter didn't fracture the ground sufficiently and had to be mechanically hammered to the correct profile. Although mechanical trimming was usually required due to the perimeter hole standoff, level 53 needed far more hammering than initially anticipated.

As the shaft progressed deeper, the rock mass became harder and more competent. It reached a point where the low MIC and decoupled 26 mm PowerSplit™ didn't have enough energy to adequately fracture the more competent rock. The rock breaking required in this area significantly delayed the production cycle, and a revised blast design was needed for the next level.

5 WIRELESS TECHNOLOGY

Before the innovation of truly wireless electronic blasting, all commercial initiation systems required a physical connection between the firing system (source of the initiation signal) and the

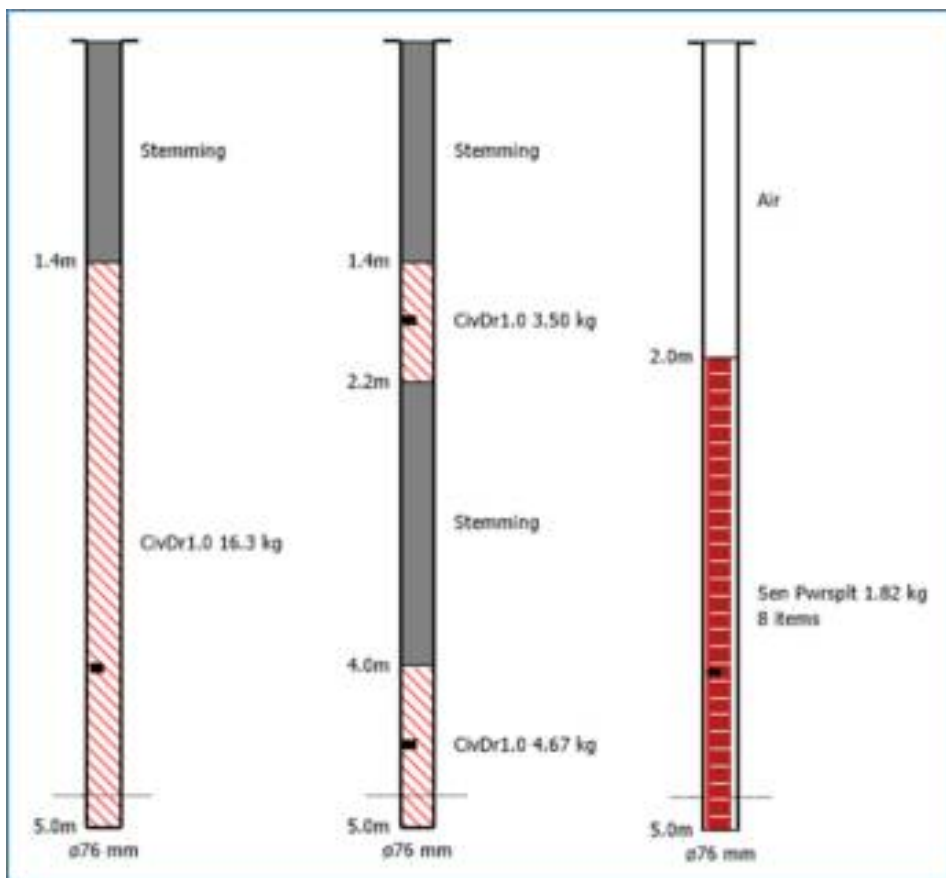


Figure 8. Level 51 design loading of cut, production, and perimeter holes.

in-hole detonator. Eliminating the need for a physical connection between the firing system and the in-hole detonator facilitated several new mining methods, including the safe passage of mining equipment over loaded blastholes.

Orica internally conducted early work on through-the-rock wireless initiation concepts, which evolved into WG, the world's first 'through-the-rock' commercial wireless initiation system. It was initially released as WebGen™ 100 in 2017 and further developed and refined into the current version, WebGen™ 200.

The WG system uses an encrypted electromagnetic induction (MI) signal to transmit firing commands through rock, water, and air to pre-programmed in-hole primers. The MI signal is used to wirelessly arm and initiate specific blasts with sub-millisecond timing accuracy while other blasts remain safely dormant. WG has eliminated

the need for physical connections to each primer, thereby removing the limitations they impose. This facilitated the exploration of single-pass drilling and the loading of multiple blasts in the Marica shaft.

A key focus of FGJV was on production rates and overall project efficiency, primarily achieved through optimising blast designs and implementing operational improvements. Alongside mucking spoil, the drilling and blasting process was the longest and most critical activity undertaken. Therefore, it was essential to continually seek new ways to innovate and enhance efficiency gains.

Each drill and blast campaign required a significant amount of time to lift the plant, equipment, materials, and personnel in and out of the shaft. WG wireless technology would allow FGJV to drill up to five benches in one drilling

campaign, effectively reducing the number of drilling campaigns by over 40%. Wireless blasting technology presented a unique opportunity to significantly enhance blasting efficiency and, therefore, reduce the overall project schedule and costs.

6 WIRELESS SYSTEM CONCEPT DESIGNS

A concept design was developed to explore the use of WG in achieving the deepest blast possible within the drilling and vibration constraints. Due to ground conditions, the maximum functional depth that the available drill rigs could accurately achieve was 16 m. The plan was to fire the maximum charge possible in each blast, which necessitated increasing charge weights as the holes moved further from the shaft walls. The design was divided into two sections, an upper and a lower section. The upper section could be loaded and fired using conventional Electronic Blasting Systems (EBS), while the lower section would be loaded with WG to be fired at a later date.

The design used 143 production holes and 54 perimeter holes. The same burn cut design was employed, featuring 76 mm blastholes and eight 102 mm reamers in a revolver configuration. For this design, all holes were loaded with bulk emulsion, with four decks of explosives in the perimeter and two decks in the production holes. Utilising the available blast vibration results, the site constants were updated, and the charge weights were maximised to adhere to the 100 mm/s limit.

In shaft blasting, the burn cut is a crucial

technique used to create an initial void or free face, which is essential for breaking rock effectively. This technique involves drilling a series of closely spaced holes in a pattern that is fired in a specific sequence to form the void. The significance of the burn cut lies in its ability to efficiently direct the energy from the explosive charges, minimising overbreak and ensuring the stability of the surrounding rock structure. Proper execution of the burn cut is vital for achieving optimal fragmentation and the intended depth of the blast.

A traditional burn cut employs a combination of ‘reamer’ holes that are left uncharged and surrounding charged blastholes designed to break into the reamers. This design creates the void into which the surrounding rock mass breaks as the blast fires. Since the concept required firing the blast as two separate shots, maintaining the integrity of the reamers in the lower section would be challenging. They could be blocked by mud, water and rock from the first blast, rendering them ineffective for the second blast. While methods for sealing the reamers at a specific depth for later recovery were considered, a second design concept without reamers was developed.

In the revised design, all the holes in the burn cut were 102 mm diameter, further increasing the charge weight and energy. The production holes outside the burn cut and the perimeter holes remained at 76 mm. The burn cut blastholes were loaded with four decks of bulk emulsion, two decks in each section of the blast. The burn cut holes were then timed in a sequence where the top deck of the centre hole would crater to the surface, creating an initial void into which the subsequent decks of explosives would fire. This design consisted of a combination of deck loading and timing to produce the required relief angles and void down to the bottom of the upper section of the blast. The lower section of the blast was timed in the same sequence and fired in the same way to create the necessary void once the upper section had been fully excavated.

A third burn cut design was also developed, combining the traditional design with reamers drilled to the bottom of the upper section firing conventionally, while the lower section was deck loaded and timed as previously described. Having these three design options enabled Orica and FGJV to engage in a considered discussion about which approach would yield the most successful outcome and maximise the shaft’s advance rate.

The concept of single-pass drilling and loading multiple blasts would triple the depth previously

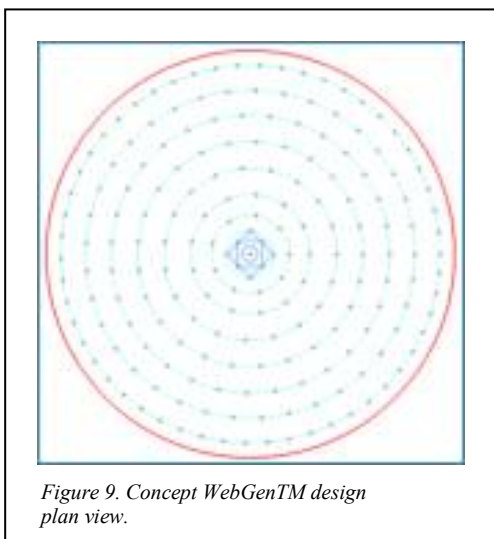


Figure 9. Concept WebGen™ design plan view.

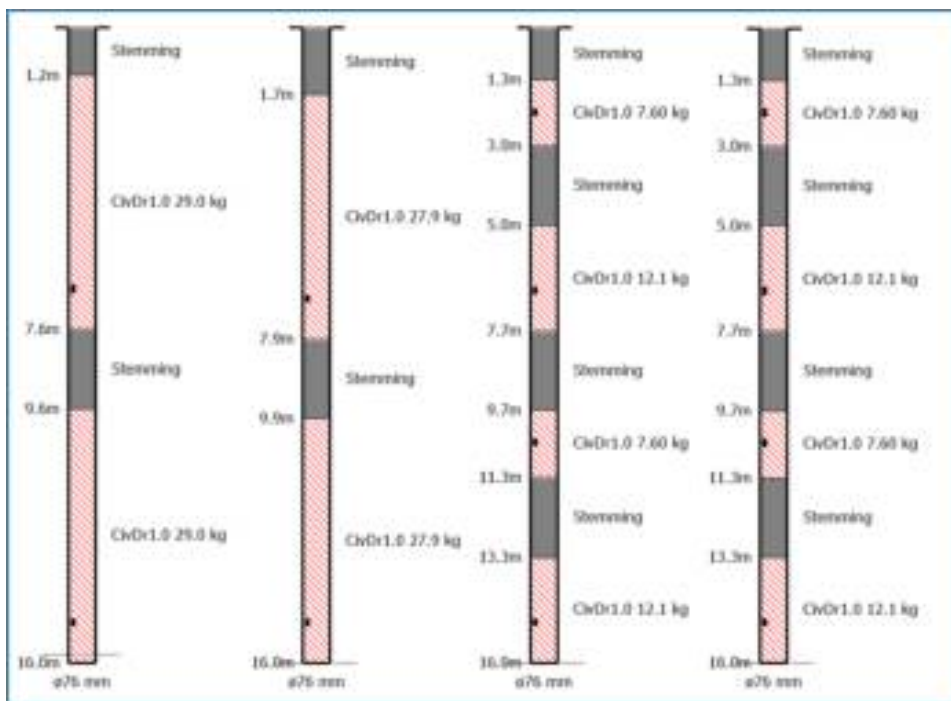


Figure 10. Conceptual design loading of cut, production, easer and perimeter holes.

achieved in the drill and blast cycle. This would significantly impact production cycle times by reducing the necessary mobilisations of the drill and loading crews, as well as the drill tramming and setup time, and the number of holes to be drilled. There would also be a threefold reduction in the crane time required to deploy and recover the drills, explosives loading equipment, and crews for the same volume of broken rock.

7 BLAST VIBRATION REVIEW

Blast vibration was originally limited to 100 mm/s PPV, with conservative charge weights and hole configurations. As part of FGJV's pursuit to review the blast vibration limit in consideration of permanent civil infrastructure, they engaged a third-party expert consultant to review the blast designs, vibration records, and proposed design changes, and compare them with published vibration performance benchmarks. The consultant visited the site to assess the operation and met with FGJV and Orica to discuss the blasting that had been conducted. After a positive meeting, the consultant gathered the relevant information for review and agreed that changes to the existing

constraints limiting the scale of blasting would enhance the fragmentation, diggability, and productivity of the operation while not subjecting permanent infrastructure to damage or harm. The consultant examined the pertinent aspects by reviewing available international literature and data collected from various other studies with which they were associated. These aspects were:

- The implications of the blast vibration on the shotcrete.
- The implications of the blast vibration on the grouted rock anchors.
- The expected zone of fracturing around a blasthole.
- An increase to the powder factor to improve the diggability of the blasted rock.
- An increase to the blast round length to improve productivity.

The review of the site data, along with information from other studies and accepted published material in the international literature, confirmed that blasting vibration would not compromise temporary ground support or impact the long-term design life of the shaft unless the vibration levels exceeded many hundreds of mm/s.

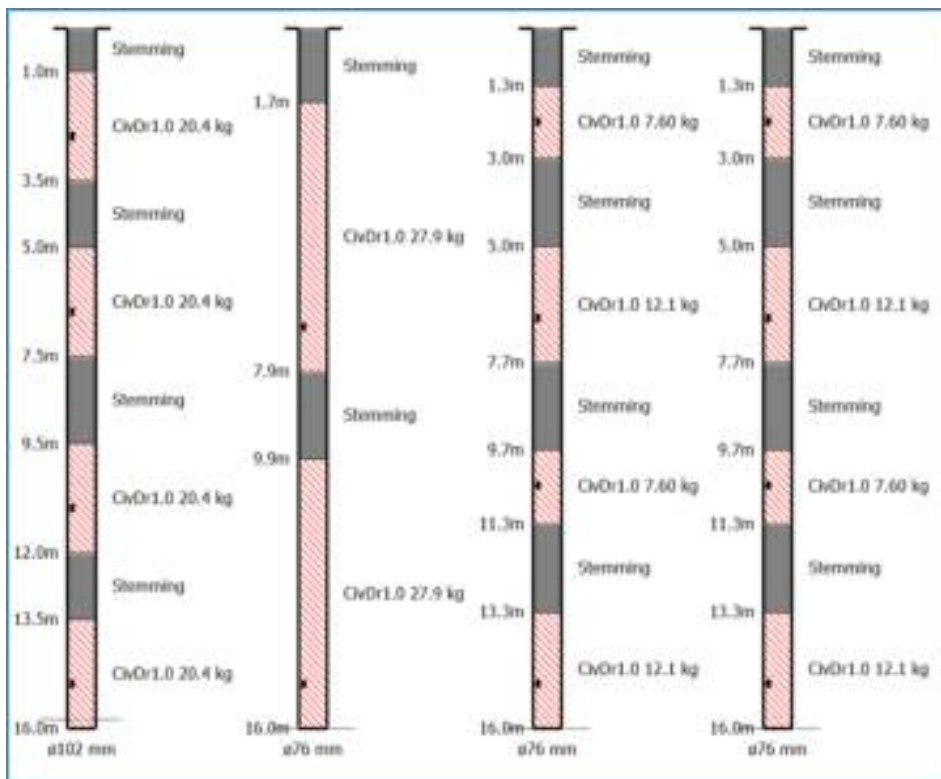


Figure 11. Conceptual loading of cut, production, easer and perimeter holes with no reamers used.

Controlled blasting that minimises vibration through adjustments to the blasthole diameter, explosive density, explosive length, and the stand-off distance would not affect the integrity of the ground support.

Based on this analysis and supported by Orica's blasting expertise, FGJV sought and obtained approval to raise the vibration limit to 250 mm/s PPV. The increase was sanctioned after thorough consideration of this study, along with evidence of controlled blasting and compliance with the required vibration limits. This adjustment facilitated more efficient blast designs with deeper cuts and reduced decked charging, directly improving cycle times without compromising safety.

8 BLAST DESIGN EVOLUTION

FGJV promptly requested a revised blast design down to 7.5 m deep, extending from the previous 5 m deep blasts. Orica recommended a change to 89 mm diameter blastholes now that the vibration limit had been increased. This would enable the

pattern to be expanded while maintaining the same powder factor, resulting in fewer holes and faster drilling and loading times. FGJV ordered the 89 mm consumables; however, due to supply lead

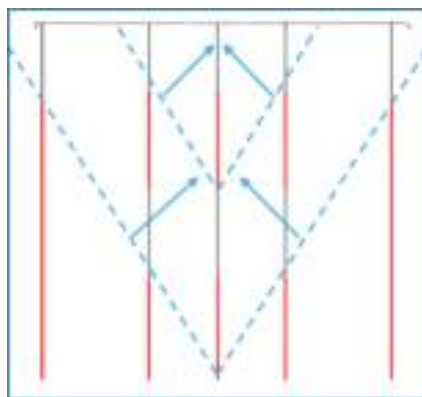


Figure 12. No Reamer burn cut cross-section showing the relief angles and direction of movement.

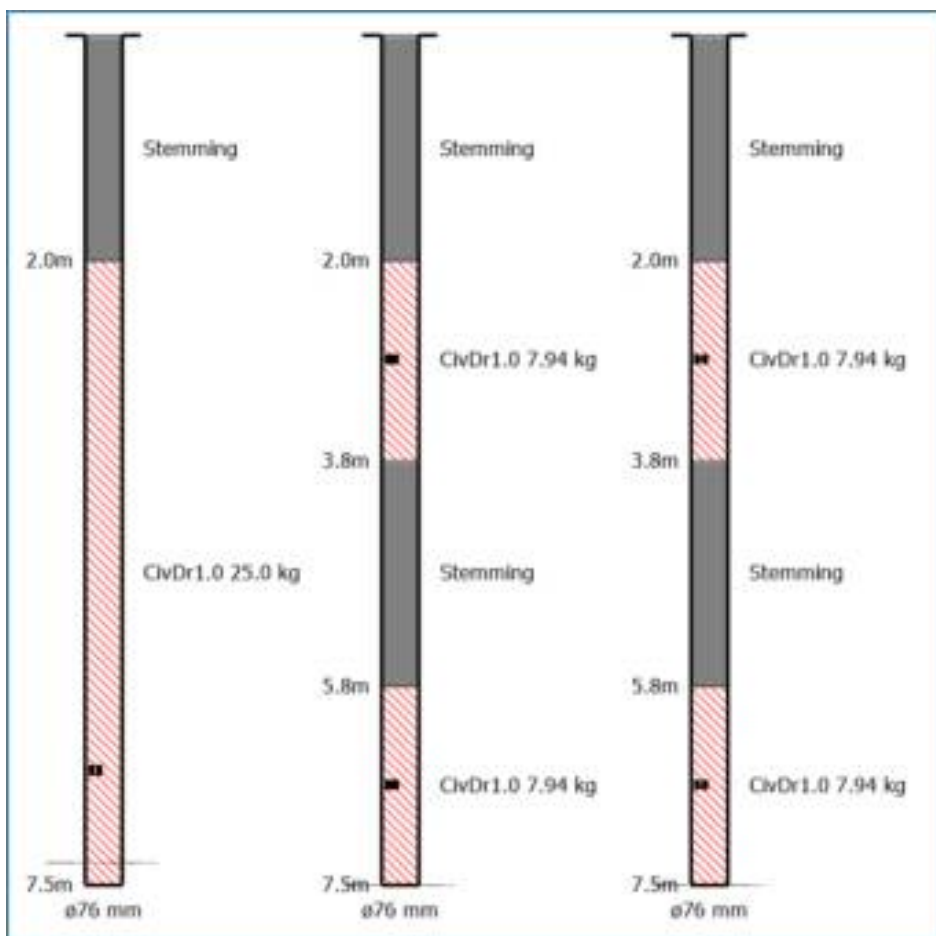


Figure 13. Level 55 design loading of cut, production, and perimeter holes.

times, the next few shots would need to use the 76 mm holes.

The earlier design was updated, slightly expanding the production pattern, increasing the spacing on the perimeter, and extending it to 7.5 m deep to blast Levels 55, 56 and 57. The design still used 76 mm holes, incorporated 163 production holes, and reduced the number of perimeter holes to 54. With the vibration limit raised, full column loading could now be achieved across much of the blast. In addition to the cut, the surrounding three rings were also designed to be fully loaded. The remainder of the design was deck loaded with increased charge weights. The strategy for managing the perimeter changed to a trim blast to fracture the rock and enable mechanical trimming to achieve the correct profile.

In-hole pressure testing for the WG system was also planned for this blast. Carbon resistor gauges (CRGs) placed in the blastholes at the primer positions measure the pressure experienced by the adjacent holes and from the explosive deck within the same hole firing. This measurement is crucial for understanding potential design advancements that can be made without exceeding the limits of the WG primer.

The Level 55 shot was successfully fired on the 6th of October 2024. Despite the increased MIC, the shot recorded only 80.14 mm/s vibration, well below the revised limit of 250 mm/s. This provided the project with confidence that further design improvements were now feasible.

Initial results from the shot looked promising, with good heave and fragmentation around the cut.

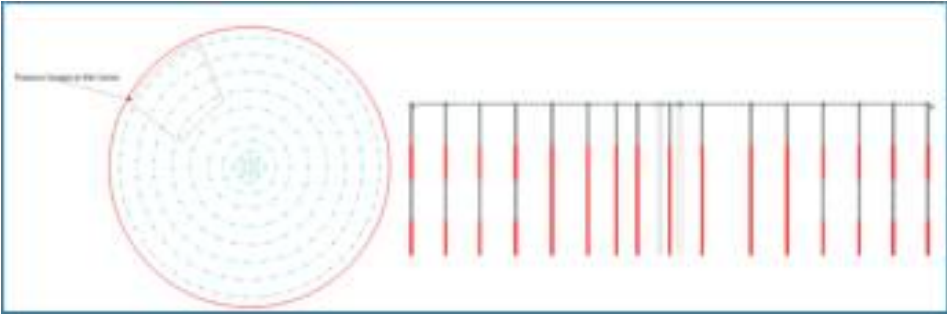


Figure 14. Level 55 shot plan and cross-section detailing deck loading and pressure gauge locations.

However, as excavation commenced, the top stemming zone from the decked holes was poorly fractured and required hammering. A review of the blast video and results showed little movement in the decked sections and moderate movement from the full columns during the blast. For this shot, the

stemming zone was increased to 2 m across the shot to help contain the energy from the increased charge and reduce the risk of damage to services. It was evident that this increase in stemming led to suboptimal results in the stemming zone. Once excavation proceeded below the stemming zone of

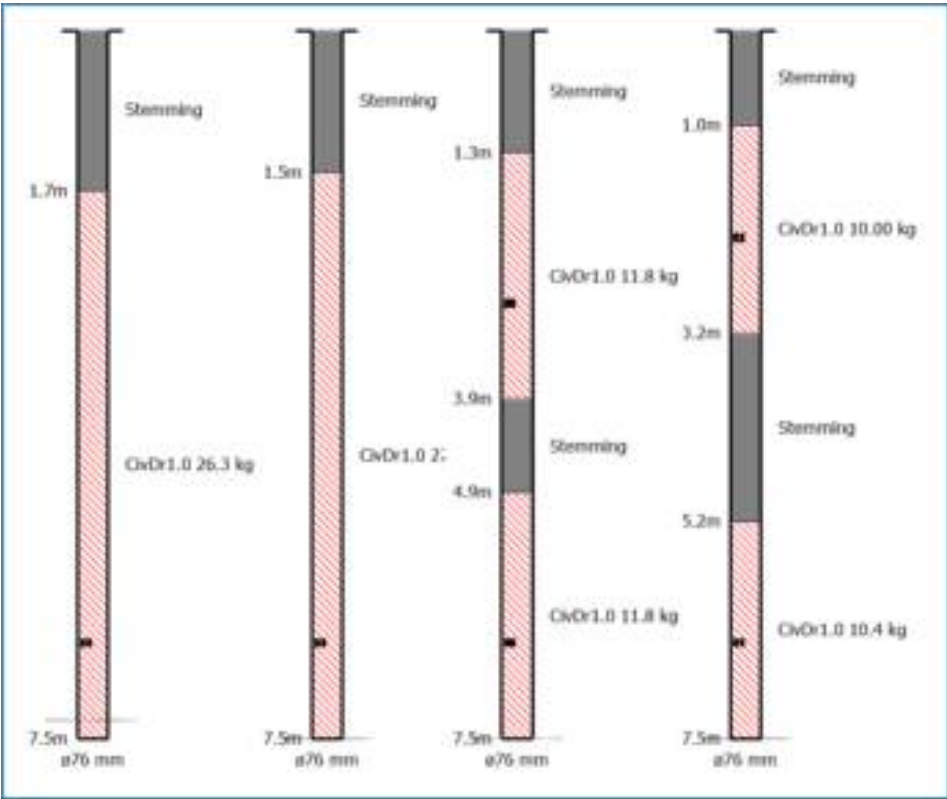


Figure 15. Levels 58 and 61 design loading of cut, production, easers and perimeter holes.

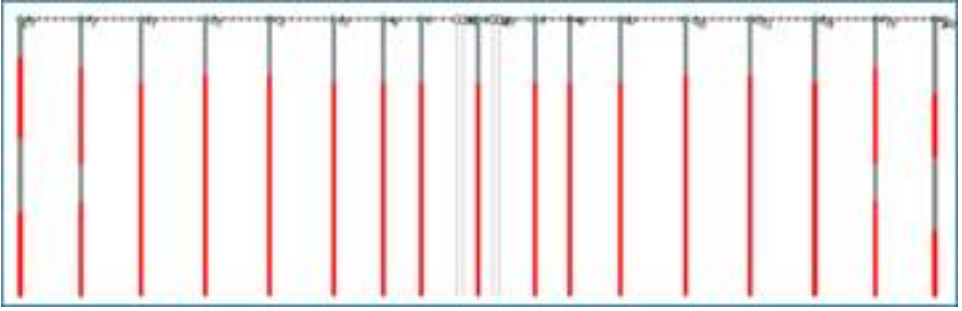


Figure 16. Level 58 and 61 cross-section detailing reduced deck loading.

the outer rings, fragmentation improved, allowing for easier excavation. With the transition from packaged explosives to emulsion in the perimeter holes, the holes were drilled vertically to offset the charges from the designed shaft profile and limit fracturing to the

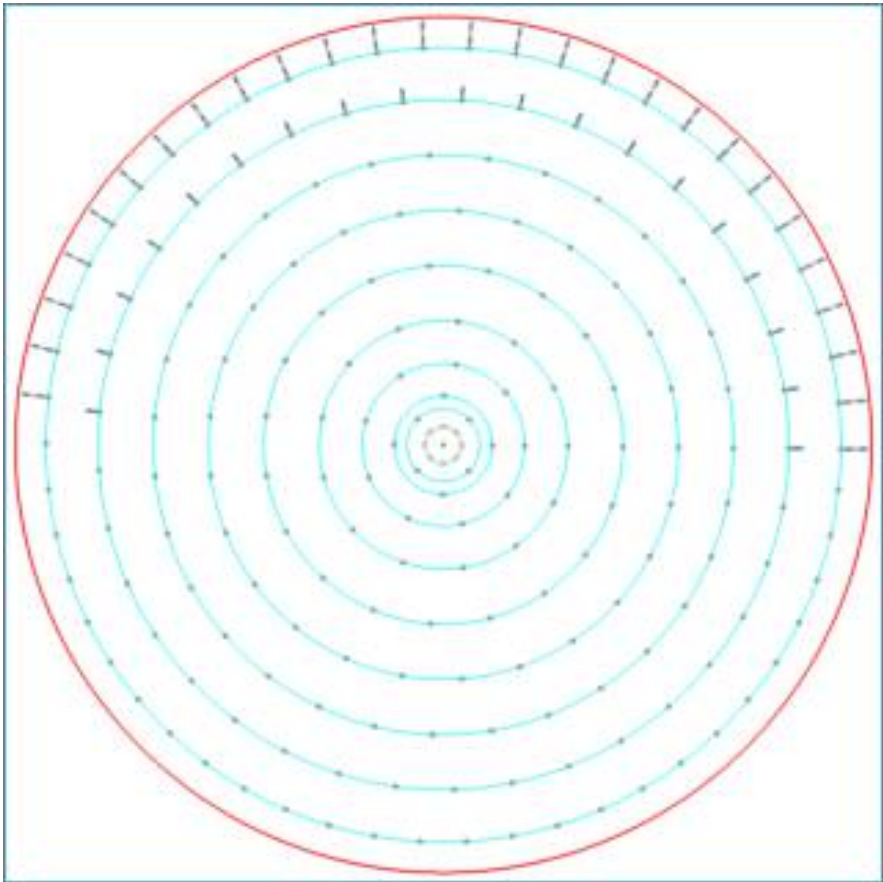


Figure 17. Level 64 and 67 design plan view detailing hole lookout in the northern half of the blast.

walls. When excavation reached the base of Level 57, the rock between the perimeter holes and the wall profile was fractured yet still intact. The area could be ripped, though it took longer than planned to excavate. FGJV and Orica discussed the outcome and agreed that a return to angled holes might be necessary in future blasts to reduce toe flare if the rock strength increased as the shaft deepened. Given the suboptimal fragmentation encountered in the stemming zone and at the toe flare at the base of the wall from the Level 55 shot, design changes were implemented for the next shot at Level 58.

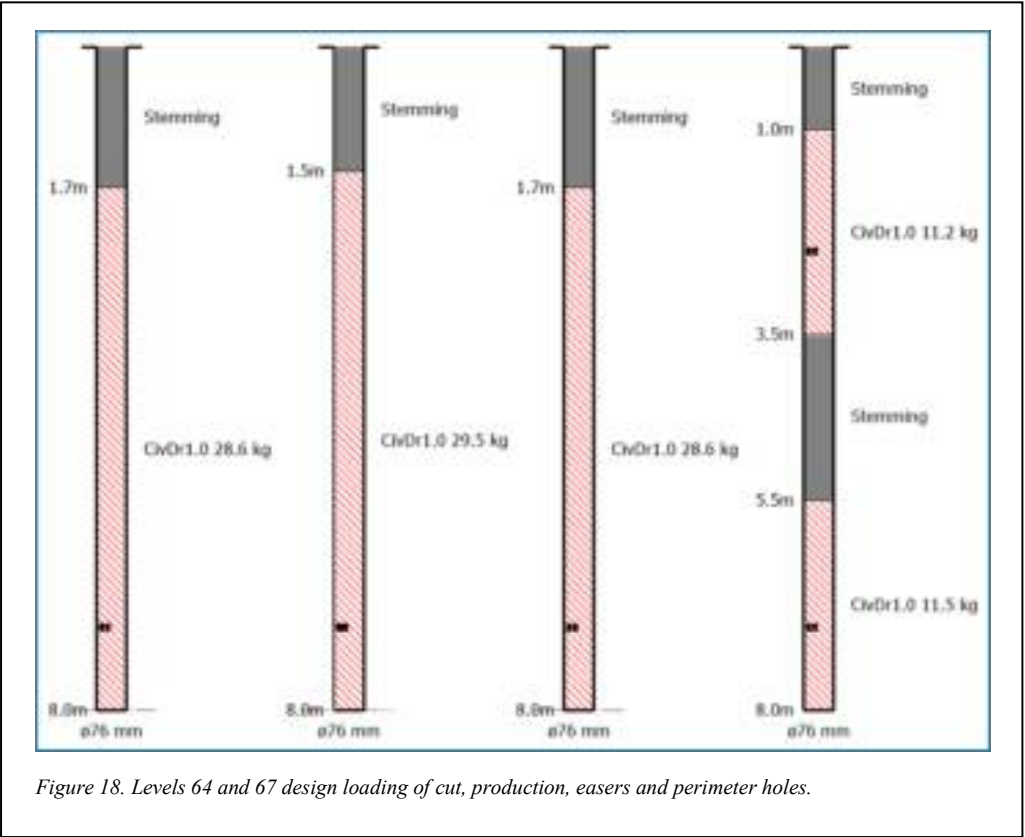
Throughout the following blasts, the design parameters were refined, leading to a dramatic improvement in the blast results. The shots remained designed to 7.5 m deep, and to address the poor fracturing in the stemming zone from the Level 55 blast, adjustments to the stemming heights for shots at Levels 58 and 61 were made as follows: the cut stem was reduced to 1.7 m, the production holes to 1.5 m, and the easers and perimeter holes to 1.3 m and 1.0 m respectively. Furthermore, all holes except for the easers and perimeter holes were designed to be loaded with a

full column of bulk emulsion. The deck in the easer holes was lowered to 1m, and these two decks were fired on the same delay. This resulted in excellent fragmentation and heave throughout the blasted muckpile.

As excavation reached the bottom of level 63, it became evident that there was some increased toe flare at the base of the wall around the northern half of the blast. For the shots at levels 64 and 67, the easer and perimeter holes were angled with a lookout in this area of 3 and 6 degrees respectively, and the easers were loaded with a full column of bulk emulsion. a 0.5 m Subdrill was also introduced across all holes in the pattern. The vibration results were kept under the limit of 250 mm/s by ensuring each hole was firing on its own delay. The design improvements made over the course of the previous four shots resulted in a complete turnaround in blasting performance. Fragmentation and heave were excellent, significantly improving production cycle times.

9 MAINTAINING SAFETY STANDARDS

FGJV maintained a rigorous commitment to the



safety of personnel and infrastructure throughout the drill and blast operation. Each blast campaign involved a systematic loading and pre-blast procedure to ensure the safety and protection of all personnel and shaft infrastructure. Services were intentionally kept at their maximum allowable distance from the blast face to minimise potential damage from fly rock or stemming ejection during the blast. Just prior to each blast electrical cables and flexible dewatering pipes were secured off the floor and fastened to the shaft walls. The personnel hoist was raised to the surface, cranes were rotated clear of the shaft, and ventilation systems were shut down before firing. All equipment was removed from the blast area, and a designated exclusion zone of 150 m from the collar of the shaft was enforced to guarantee workforce safety.

FGJV maintained strict standards for wall quality, overbreak control, and excavation accuracy throughout the blasting program. Each

blast was assessed against expected geometry and fragmentation, with adjustments made based on observed outcomes. As the shaft deepened and rock conditions became more competent, refinements were introduced, including increased subdrill lengths, modified stemming heights, and angling of perimeter and easer holes. These changes were progressively implemented to reduce toe flare, improve fragmentation consistency, and minimise profile overbreak.

Close collaboration with Orica facilitated the ongoing optimisation of blast designs while maintaining safety standards as the shaft excavation advanced. Because of the drill rig configuration, FGJV drillers needed to maintain a 1 m offset from the shaft wall, which usually left up to 500 mm of rock along the profile, necessitating trimming back to the design A-line using a hydraulic breaker and excavator. Progressive enhancements in wall condition,

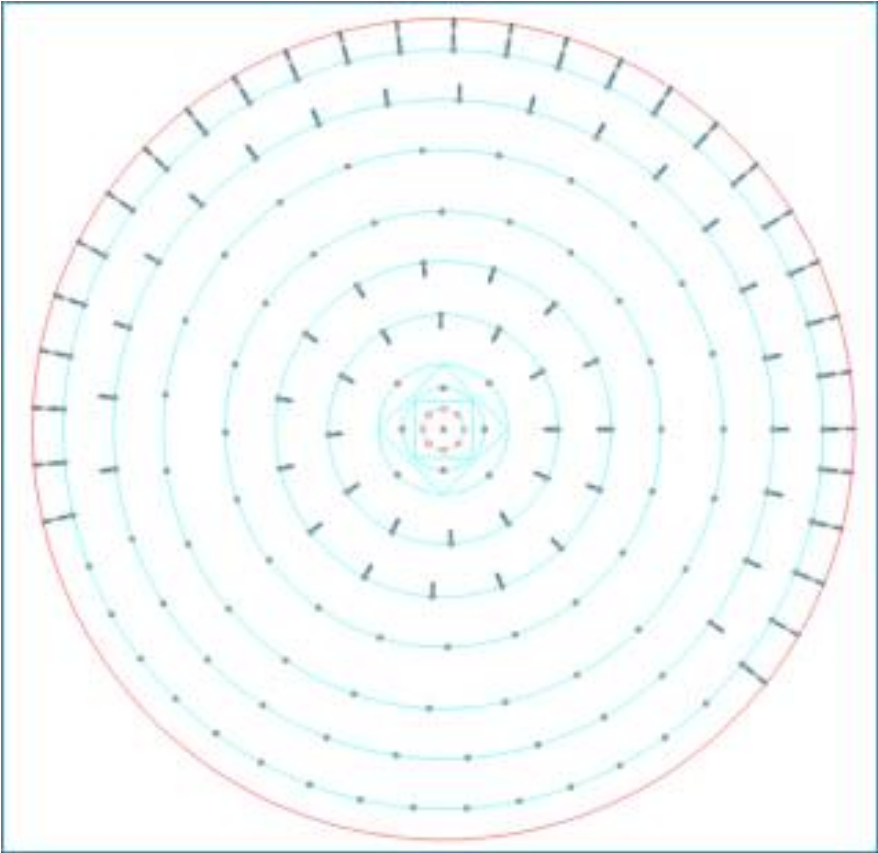


Figure 19. Level 73 design plan view detailing angled holes around the cut.

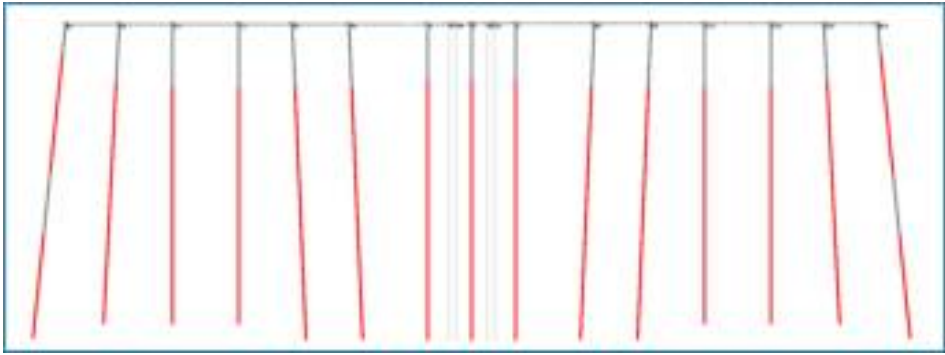


Figure 20. Level 73 Cross-section detailing angled holes and subdrill.

reduced overbreak, and consistent fragmentation outcomes validated the efficiency of this approach. FGJV's commitment to quality ensured that blasting and program efficiency did not compromise the integrity of the temporary ground support, thereby safeguarding personnel during the excavation phase and preserving the overall design life of the shaft.

10 A DESIGN STEP CHANGE

After several months, the 89 mm drilling consumables arrived at the Marica shaft. Additionally, approval was given to increase the unsupported wall height from 2.5 m to 3 m for the Excavation Support Class relevant to this area of the shaft. FGJV immediately requested a three-bench blast 9 m deep using the 89 mm holes. This signified a significant change in the design of the blasts. Employing 89 mm holes and expanding the pattern to maintain the powder factor led to a ~25 % reduction in the number of holes required for an ~18% increase in blasted volume.

The 9 m deep blast from Level 70 was designed with 89 mm holes and incorporated all the design improvements made to date. The shot consisted of 109 production holes and 45 perimeter holes. As with the previous shot, all blastholes except the perimeter holes were designed with full columns of bulk emulsion. The traditional burn cut, featuring eight reamers in a revolver pattern, was scaled up from 76 mm to 89 mm blastholes. The blast was fired on the 25th of March 2025, resulting in excellent heave and fragmentation, while the recorded vibration was 202.2 mm/s against the 250 mm/s limit.

As excavation reached the bottom of level 72 in the cut area of the blast, the final 0.5 m was

cracked but intact. It appeared that the blast design was nearing the limit of what it could achieve at this depth. In discussions with FGJV, Orica recommended increasing the subdrill in the cut area to 1.0 m. FGJV also requested an increase in subdrill for the perimeter holes. Additionally, the two rings around the burn cut were to be angled towards the cut at 2.5 degrees, and the pattern was adjusted slightly to maintain consistent toe burdens.

The shot at Level 73 was fired on the 8th of May 2025, producing good fragmentation and heave throughout the blasted muckpile. The initial raw recorded vibration was 286.3 mm/s, exceeding the 250 mm/s limit. Orica and the vibration monitor manufacturer analysed the blast video and vibration trace. They determined that the discrete spike in the vibration waveform indicated debris from the blast striking on or near the monitor, and a review of the blast video confirmed debris landing in the area at the appropriate time during the blast. The peak vibration from the trace, excluding this point, was 233.34 mm/s.

11 WIRELESS BLASTING REVIEW

After the successful blasting to a depth of 7.5 m with 76 mm diameter holes, Orica and FGJV reviewed the concept designs for single-pass drilling and loading of multiple blasts. While this would provide the project with flexibility and some advantages in drill and blast cycle time, ultimately, it would depend on whether the site could meet the required production targets to ensure the shaft is excavated on time.

With the vibration limit now set at 250 mm/s and improved advance rates achieved, the site was on track to reach the bottom of the shaft by the

required date. This was further reinforced by the introduction of 89 mm holes and an extension of depth to 9 m. The use of WG would have reduced crane lifts, drilling mobilisations, and service disconnections, all of which were time-consuming, labour-intensive, and high-risk activities. FGJV supported early-stage testing, including signal verification and in-hole pressure trials, and reviewed concept designs to assess feasibility.

However, when considering the strong results already achieved through bulk emulsion, 9 m triple-bench designs, and 89 mm drilling, the decision was made to defer WG at Marica. Although not implemented here, the groundwork completed positions the technology as a viable option for future phases of the Snowy 2.0 project, such as the downstream surge shaft, which is set to commence towards the end of 2025.

The WebGen™ design concepts remained valid, and while not adopted at the Marica shaft, this technology has facilitated the development of innovative mining methods to safely and efficiently improve production cycle times and advance rates.

12 OVERALL EFFICIENCIES

FGJV was required to maintain a clear set of production, efficiency, safety, and quality objectives throughout the excavation phase of the project to ensure the successful delivery of the upstream surge shaft. Central to these objectives was the necessity to complete the shaft excavation within tight project milestones while upholding safe and cost-effective methods. A key goal was to maximise the bench turnover rate by optimising blast designs to minimise the number of blast holes, thereby reducing drilling and loading times. Efficiency gains were also pursued through trial blasts of multiple benches in a single campaign, which further decreased the number of blasting cycles and the associated mobilisation efforts. This not only accelerated production rates but also directly contributed to cost reductions and enhanced overall safety by minimising the number of crane lifts, one of the highest-risk activities undertaken on the project.

To maintain progress on the development of the shaft, reducing cycle time and ensuring advance rate consistency became critical. Due to the increasing depth of the shaft, each stage of the bench excavation and support required significant setup time. The setup of the drill and blast stage included removing loose material on the shaft floor, lifting in drill rigs and consumables, setting

out the drill pattern, managing service protection, lowering materials such as stemming and explosives, lifting and placing blast mats, as well as constant dewatering of the shaft to manage groundwater inflows. A typical blasting cycle, from floor cleanup to the post-blast inspection, spanned 90 to 100 hours; therefore, reducing the number of these cycles through deeper and more efficient shots was essential for improving production rates, reducing costs, and limiting high-risk lifting activities.

Drilling campaigns encountered various constraints. Early attempts at blasting in less competent ground resulted in the collapse of blastholes, causing delays. Ground conditions varied significantly with depth, especially in the identified fault zone, leading to frequent hole blockages. High inflows of groundwater presented further challenges, requiring constant dewatering activities using a pump and fishtank setup alongside a series of inline pumps.

The first few blasts used two Sandvik Ranger DX800 drill rigs, but due to increasing the hole depth and available space an Epiroc T45 was introduced. This three-rig configuration significantly enhanced efficiency, enabling up to 25 m/hour per machine and substantially reducing overall drilling time. To further boost efficiency, drill patterns were carefully planned to allow for the simultaneous operation of all three rigs for as long as possible, before progressively scaling down to two and finally one as space became limited.

Further efficiencies were achieved by reducing blast mat usage. Following successful trial blasts, the team was able to decrease the number of blast mats used from full-face coverage to a minimal setup of four to six mats over the cut holes only. In addition to reducing blast mat coverage, the team also experimented with using accumulated groundwater inflows as a natural blast mat to suppress gas and flyrock ejection. By tracking the daily inflow rates, the team was able to halt dewatering activities ahead of the blast, during the final loading activities, and fill the shaft floor by up to half a metre by the time the shot was fired. These changes not only accelerated the blasting cycle but also improved safety outcomes and reduced impacts on shaft infrastructure and ground support.

Overall, the reduction in drilling and blasting cycles, achieved through optimised design, equipment upgrades, and operational planning, delivered significant gains in production, lowered project risk exposure, and was instrumental in

aligning shaft excavation progress with the project schedule.

13 FGJV LESSONS LEARNED

The excavation of the Marica upstream surge shaft presented a complex and evolving challenge, requiring FGJV to adapt its blasting strategies in response to varying ground conditions, increasing shaft depth, and production pressure. Throughout the project, FGJV transitioned from a cautious, low-vibration 2 m deep blast using packaged explosives to an ambitious and efficient 10 m deep, triple-bench design using fully charged 89 mm diameter holes. This advancement was not the result of a single breakthrough but rather stemmed from continuous improvement driven by on-the-ground experience, data analysis, and close collaboration with Orica.

A key learning was the value of iterative design based on observed outcomes from each blast. Essentially, each blast cycle informed the next; stemming heights were reduced, hole angles were introduced, perimeter strategies were adjusted, and project-wide standard design blast vibration limits were challenged and successfully increased. These refinements led to measurable improvements: fewer drill and blast campaigns, reduced crane lifts, improved fragmentation, shorter cycle times, and better wall conformity, which overall contributed to high-quality excavation and cost savings for FGJV. Safety was upheld throughout, and operational risk was minimised by the reduced frequency of high-risk setup activities.

These lessons now shape the planning and execution of the downstream surge shaft, which is scheduled to commence at the Lobs Hole site towards the end of 2025. Proven design elements, such as deeper blasts, bulk emulsion, the use of 89 mm blastholes, and decreased reliance on blast mats, can be implemented almost from the outset of excavation, saving both time and costs. FGJV's experience at Marica has set the project's benchmark for production-oriented, safety-focused shaft excavation, providing a launchpad for the downstream shaft team.

14 SUMMARY AND CONCLUSIONS

The Snowy 2.0 pumped-hydro expansion of the Snowy Mountains Hydroelectric Scheme in Australia is a significant engineering project that, once completed, is expected to provide clean and reliable energy storage and generation for the next 100 years. The upstream surge shaft Marica will

rank among the largest diameter operational shafts worldwide.

The Marica shaft blasting operations initially faced challenges due to ground conditions, which impacted the stability of blastholes. The first successful blast occurred at level 46, using careful design criteria to meet vibration limits. Subsequent blasts aimed to enhance advance rates and efficiency, with FGJV seeking to increase blast depth and explore new technologies to boost loading efficiencies. Orica and FGJV also successfully demonstrated that by executing blast designs in a considered and well-managed manner, the project could safely raise vibration limits to allow for improved blasting strategies while maintaining the integrity of the ground support.

The use of wireless technology, specifically the WebGen™ system, was explored to enable single-pass drilling and loading of multiple blasts, thereby reducing production cycle times. The developed concept designs showed that this technology would significantly impact production cycle times while safely maximising charge weights and complying with vibration constraints. This was accomplished by reducing the mobilisations of the drills and loading crews and cutting down drill tramming, setup time, and the number of holes drilled. Additionally, this approach would reduce the crane time needed to deploy and recover the drills, explosives loading equipment, and crews for the same volume of broken rock. While it wasn't adopted at the Marica shaft, this technology has facilitated the development of innovative mining methods to safely and efficiently enhance production cycle times and advance rates.

The evolution of blast designs at the Marica shaft involved increasing blast depths and transitioning to larger blastholes, which reduced the number of holes needed and increased the blasted volume. The project demonstrated improvements in fragmentation and heave. However, challenges remained in achieving optimal results in certain areas. Adjustments to design parameters, such as stemming heights and hole angles, were made to address these issues and enhance outcomes. Orica and FGJV ensured that incremental changes were implemented and reviewed to determine the best efficiencies. While there may be cases for changing everything at once, in blasting, it is crucial that improvements are made conservatively to identify which parameter best enhances blasting results.

Orica and FGJV's approach to the Snowy 2.0 project emphasised innovation, safety, and

efficiency. Exploring new technologies and maintaining iterative design processes helped meet production goals while maintaining safety standards. Lessons learned from the Marica shaft blasting campaign are being shared and applied to other parts of the project, demonstrating the value of collaboration and continuous improvement in complex engineering projects.

15 ACKNOWLEDGEMENTS

Orica wishes to thank the FGJV Marica Team, specifically Mitchell Parkinson, the co-author, Pierangelo Sorbelli, Peter Benson, Valentin Sugar, Endalkachew Kebede, Ashley Robinson, and Tony Carless. The work completed on the Marica shaft, along with Orica's efforts on the Snowy 2.0 project, could not have been achieved without the support of FGJV Directors Fabrizio Lazzarin, Sally Law, Antonio Animato, and Bruno Tirrizzi. The assistance of Snowy Hydro Limited's team is greatly appreciated, especially Richard Clarke.

Orica also wishes to thank Paul Ashbrook, Angus Ashbrook, and Emily Ashbrook. Orica's Snowy 2.0 team has worked tirelessly on the implementation of the project and the review of results, with special thanks to: Andy Kehoe, Chultemsuren Vangan, Ebenezer Atuahene, Ellie Dixon, Francis Haden, Mark Turner, Nathaniel Gill, Reuben Manyika, Steve Le Louarn, Tiago Rodrigues, Toni Lengenber, and Zac Miller. Lastly, many thanks to the broader Orica team of Mitch Smith, James Kirkpatrick, and Peter Ayres.

Any use of generative AI in this manuscript adheres to ethical guidelines for use and acknowledgement of generative AI in academic research. Each author has made a substantial contribution to the work, which has been thoroughly vetted for accuracy, and assumes responsibility for the integrity of their contributions.

REFERENCES

- Bowen, N. 2015. Next generation blasting – How wireless technology will change mining. *Paper presented to International Mining and Resources Conference (IMARC)* Melbourne 10-12 November.
- Heilig, J. September 2024. Vibration criterion review Marica shaft Snowy 2.0. 3rd party *Engineering Report commissioned by Future Generation Joint Venture.*
- Mallette, D., Lownds, M. & Goodridge, R. 2016. Wireless electronic blasting. In *Proceedings 42nd Annual Conference on Explosives and Blasting Technique* (International Society of Explosives Engineers: Las Vegas).
- Orica *Safe and Efficient Blasting Course Manual* 2021.
- Wicks, B. & Lovitt, M. 2017. A new era of blast initiation systems reducing safety risks, costs and enabling automation. In *Proceedings 9th World Conference on Explosives and Blasting*, (European Federation of Explosives Engineers, Stockholm).
- Wireless initiation - enabling new mining methods at Musselwhite. *Mining & Quarry World*, Volume 15, Issue 6, December 2018.
- www.futuregenerationjv.com.au/news/snowy-20-project-milestone-marica-surge-shaft-excavation-update Accessed 20/05/2025
- www.snowyhydro.com.au/snowy-20/about/%20Accessed%2020/05/2025
- www.snowyhydro.com.au/news/category/snowy-2-0/tunnelling/%20Accessed%2029/05/2025

RETURN TO CONTENTS PAGE

Paper or electronic records - ways to improve blast documentation and record-keeping

T.A. Davidsavor

Barr Engineering Co

R.E. Burnham

Lancer Insurance/Nobel Insurance

J.K. Ratliff

Cors & Bassett, LPA

C. Aimone-Martin

Aimone-Martin & Associates

ABSTRACT: Blasters face a long list of tasks to execute daily, from vehicle inspections to blast design and layout, hole loading and tie-in, securing the site, and finally, safely executing the blast. Good practices and, in most cases, regulations and requirements necessitate documenting their work, specifically their blast report. While some blasters use paper forms, others use electronic means to streamline the process. The authors have served as attorneys and expert witnesses in representing companies and individual blasters in numerous administrative and legal matters in which documentation of the blaster's work was both critical and detrimental to the effectiveness of defending their work. This paper discusses in a case-study manner the authors' observations concerning good blast report format, specific observations around electronic and paper report completion, record-keeping, seismograph setup and record-keeping, video and photo documentation, internal auditing, and other topics related to documenting the blast. The paper aims to serve as a discussion guide for blasters to improve their blast record-keeping process, develop better practices that effectively document their work, and allow them to defend their work should a claim arise concerning their operations. Blasting professionals must manage a complex set of tasks each day, from equipment checks and blast design to hole loading, securing the blast perimeter, and executing the shot safely. An integral part of this process is proper documentation, often a regulatory requirement across jurisdictions. Whether using paper-based systems or digital tools, accurate and thorough record-keeping is essential to ensure regulatory compliance, support forensic analysis, and protect against liability. Drawing on legal and expert witness experience across North America, this paper explores how quality blast documentation, particularly blast reports, can be improved through thoughtful structure, review processes, and technology. With examples and recommendations applicable to European practitioners, it aims to support safer, more defensible blasting operations across diverse regulatory environments.

1 INTRODUCTION

Accurate blast documentation is fundamental not just for compliance, but for understanding, learning from, and defending a blasting operation.

The 18th Edition of the ISEE Blaster's Handbook states that logs should be sufficiently detailed that another competent blaster could replicate the blast. In Europe, this level of documentation is likewise required in many national codes and EU-aligned

safety frameworks. Beyond technical compliance, clear records enable effective audits, facilitate communication with stakeholders, and serve as a critical reference point in legal disputes. Gaps in documentation, whether absent GPS data, poor diagramming, or ambiguous terminology, can create uncertainty and increase legal and regulatory exposure.

Maintaining blast records serves many crucial roles in preparing a blast, loading holes, tying in a shot, and documenting the impact of the blast on the surrounding environment and the overall effectiveness of the blast. The 18th Edition of the Blaster's Handbook highlights that blast logs "are critical for inventory tracking and quality control and are generally considered "eye witness" testimony by the courts as to the blast event that day" (Stier, Blaster's Handbook 18th Edition, 2008 p 633). Not only are blast records to provide a factual account of the preparation and execution of the blast, but they "should contain enough detailed information that another blaster could reasonably be able to reconstruct the blast at the same location in the near or distant future" (Stier, Blaster's Handbook 18th Edition, 2008 p 633). Not only should a blaster be able to reconstruct the blast, but the record must be sufficiently clear and correct so that an expert witness can easily explain it to a jury years later to convey what the blaster did and how the blast performed. Blasters often do all the 'right' things when preparing a blast. Unfortunately, the full description, detail, and record-keeping of the drilling of the holes, specific hole loading and timing, seismograph records, photos, video, and a description of the success of the blast are often not fully documented to demonstrate that the work was done to regulation and safely performed. This lack of detail allows third parties and attorneys to fill the documentation gap with speculation that may not be true. The blast log must accurately reflect the facts of the specific blast, as the blast log is the first document needed if some aspect of the blast did not perform as expected, as a record of the actual blast in case a claim is made, and as a starting point for subsequent blast design if the outcome was effective. Incomplete records do not allow efficient or effective defence or work completed, invite questions and speculation about the quality of the blaster's work, and generally look poorly upon blasters, even if they did the job correctly and safely.

Table 27.3 in the 18th Edition of the Blaster's Handbook lists the minimum information to be recorded in a blast report. This list may need to be

expanded based on the project, client, local regulations, state regulations, or national requirements. It does not include additional reporting needs for explosive inventory, storage, transportation, or disposition.

2 BLAST RECORDS

Blast records are maintained in paper (historic) and/or electronic formats. There are benefits and concerns to both methods of documentation. All modern blasters carry a mobile phone or tablet in their vehicle, which they may utilise to maintain electronic inventory control, record commercial driving logs, submit time and expenses, and communicate via electronic means with their clients and colleagues in the office. Historically, blasters recorded their blast reports on a paper form that was either company or client-specific and had labelled spaces to record critical blast information. With ruggedised field laptops, inexpensive tables, and smartphones, software developers preparing custom client-specific software, and commercial vendors selling commercial blast reporting products, blasters have many options that can streamline their blast record-keeping, interface with borehole measurement tools, automatically calculate inventories, predict ground vibration, and a myriad of other features that paper records could not do.

2.1 Benefits and challenges of paper versus electronic records.

Each of the methods for recording blast information has benefits and limitations. One benefit of paper records is that incorrectly entered data is less likely to carry over from one shot to the next. The authors have seen several cases where an incorrect data entry is propagated across subsequent reports or information from one blast is perpetuated into future records. When data is manually entered on paper, erroneous data carryover is less likely. In both Europe and North America, paper records are still common, especially in smaller or remote operations. However, digital solutions are increasingly prevalent, especially given the EU's emphasis on digitisation and sustainability. Table 1 summarises key advantages and disadvantages of each format for the European context. Table 1 summarises some identified benefits and limitations of paper and electronic documents. In any case, correct and complete records are necessary.

The European General Data Protection Regulation (GDPR) must also be considered when handling data on private properties or third-party structures, particularly with geolocation and imagery. Systems must be secure and compliant with data protection laws.

2.2 *Blasting regulations inconsistent with industry standards*

Much like the United States, Europe lacks a single uniform regulatory standard for blasting. Instead,

EU directives (like ATEX) provide broad frameworks, while individual Member States or local authorities define the specifics. A project in Germany will be subject to different blast vibration and air pressure thresholds than one in the UK or Sweden. Therefore, blasters must not only follow local regulatory thresholds (e.g. 133 dB air overpressure in one country versus 140 dB in another) but ensure blast logs reflect those standards explicitly. Blasting is a heavily regulated industry, and regulations vary in the US and Canada from state to state (or province to

Table 1. Comparison of electronic and paper records.

Electronic Records		Paper Records	
Benefits	Limitations	Benefits	Limitations
Records can be immediately stored and saved remotely	Increased risk of carrying erroneous data	Less likely to carry incorrect data from previous blasts	Records may be easily lost in field, truck, office
Forms can be prepopulated for client and jurisdiction	Potential loss of data during entry before data upload	Easier to diagram blast details (layout)	Takes more time to enter data manually
Quickly shared with others for immediate quality review	The loss of a tablet or device can limit the ability to record	Easily see if data is not entered into a field	Illegible entries make review challenging
Easily catalogued to review later for improvement	More challenging to detail unique blast rows/holes	Readily used in most field settings	Takes up storage space and can be challenging to find
Simplifies comparison of blast results	It may require internet service to operate properly	No significant training is required	Conflicting information can be easily recorded
Can use a quality algorithm to prevent data entry error	May provide inaccurate GPS location based on setting	Highly adaptable and added information can be recorded	No opportunity for an automated quality check
Drilling logs with location quickly electronically shared	“Garbage in, garbage out” simplifies input error	No risk of a dead battery or internet access outage	Cannot link site data to the report (vibration, drilling)
Can view the shot layout in live space as drilling/loading	Some templates limit input to limited input menus	Can capture additional dialogue more effectively	Are not connected to time and location for quality
Videos and photos can be GPS-tied into the report	Accessible only to those authorised to access	Accessible to regulators and others without software	Copying reduces legibility and clarity
Allows live-time product inventory update	Need specific software to collect data	Can be readily corrected if errors are noted in entry	Sometimes, it is not fully completed

province) and even within a state. Neither electronic records nor paper forms may automatically adjust for site-specific regulations. A blaster should contact the local authorities to determine the jurisdiction-specific regulations. Blast records must be adjusted to the local jurisdiction's requirements, and a blaster cannot assume that they rely on the same input. For example, the allowable air blast in Tennessee is 140 decibels, but Kentucky has 133 decibels. The City of Duluth, Minnesota, utilises a stepped function of vibration PPV vs. frequency that does not follow the RI 8507 Z-curve, whereas the joining municipalities default to the RI 8507 Z-curve.

2.3 Consistency between blast report and seismograph record

The blast report and seismograph records must show consistent details concerning the seismograph location, which is identified by address and description of placement or GPS location and the distance between the blast and monitoring locations. Monitoring reports should show the maximum charge weight per delay and scaled distances from the blast to each seismograph and report the peak velocity, zero-crossing frequency, and sound pressure amplitude, indicating compliance with limits imposed on the blasting operations. The blast report should duplicate this information. The seismograph records should utilise the same time convention as the blast so that the seismograph records accurately reflect the time of the blast. This is especially important when defending against vibration-related claims under Eurocode or DIN standards.

2.4 Legible and complete

Blast records not only 'tell the tale of the shot', but they also reflect on the blaster's professionalism. Confusion over the shot location, timing, pattern, deck placement, maximum charge per delay, and other critical information leaves the blast and blaster open to a credibility attack. Without legible, complete, correct, and non-conflicting blast records, it is almost impossible to recreate the shot for regression analysis purposes. If a blast damage claim ends up in court, illegible or incomplete blast records undermine the blaster's professionalism and the science of blasting, giving doubt to efforts to defend the blasting. The authors identified the following areas as frequent areas of inconsistency in reporting.

2.4.1 Dates and times

The time and date of the blast must match the date and time of any related electronic files, such as videos, photos, and seismograph records. The electronic system can significantly improve this, as time can be synchronised across several devices.

2.4.2 Timing map showing decks, hole locations, and orientation

Reports require both a plan view and a section view of hole types. The scaled plan view with a north arrow must show all holes laid out by rows labelled with the category of drill holes (i.e. trim, pre-split, buffer, production, etc.). Along the shot perimeter, it is best to label each row alphabetically and then numerically label the holes in the row to match the drill log. Next to each hole, label the detonation time resulting from the delay in the design. Section profile views of each loaded hole type must show, on average, the total drilled depth plus any sub-drill, the length of charges by product type, location(s) of boosters, and length and placement of inert stemming material. Hole types must match those identified on the plan view.

2.4.3 Crew information and BIC signature

The report must be signed by the blaster-in-charge (BIC) and include the printed name of each member of the hole-loading crew with the BIC clearly identified. This is often a regulatory requirement.

2.4.4 Drilling records

Drill records can be prepared in paper format or, more often, electronically. The information recorded for each hole by the driller must include any deviations from the expected drilling rate and hole conditions. If any unusual, unexpected, or adverse conditions are encountered during drilling, conditions must be sufficiently detailed to assist the blasting crew with appropriate loading. Such conditions include voids, fracture zones (loss of air or water), presence of water, hard or soft drilling zones, etc. The hole labelling scheme must match that used in the blast plan and report. The logs must be signed and dated by each driller, and to ensure effective transfer of knowledge, they must be countersigned by the BIC when the log is transferred to the blasting crew before hole loading. Such transfer of critical information is made easier when records are preserved in



Figure 1. Example of location recording error.

electronic format to improve the quality and completeness of crew communication.

2.4.5 Location of the blast

The precise location of each blast and the nearest structure(s) must be recorded and included on the blast report. The most effective way to accomplish this is to record the location using a GPS with latitude and longitude coordinates. Structure locations can easily be identified on Google Earth. Blast locations can be recorded in terms of the blast centre or, more preferably, to the nearest hole with all corners of the blast precisely located. However, in cases where blasting is close to structures, coordinates of the pattern corners are necessary to ensure the accuracy of measurements plotted within site propagation models. Maintaining a site regression 'K' factor and slope for both ground vibrations and sound pressures can be an invaluable tool for the blaster in planning and for predictions, as well as demonstrating that the blaster was within acceptable limits based on measurements at other locations.

The accuracy of GPS coordinates is affected by the method of recording from smartphone applications. The modern smartphone is antenna limited and is typically accurate to within a 4.9 m (16 ft.) to handheld GPS devices accurate to 3 m

(10 ft) in an open area with unobstructed satellite view. Each operator must understand the limitations of their GPS and consider this when recording location. Accuracy can be significantly affected by signal blockage near highwalls and buildings, atmospheric conditions, satellite geometry that continuously changes over time, and electronic interference. A GPS is always more accurate with horizontal location than vertical location, so areas with high relief should be verified for accuracy. More precise measurement is possible with a GPS tied to a local benchmark where sub-centimetre accuracy is available. To reduce data input error, consider using Universal Transverse Mercator (UTM) coordinates, as a simple grid system in metres is easily understood relative to a system incorporating decimal degrees (DD) or degrees, minutes, and seconds (DMS). In the UTM system, a unit is always a metre, whereas in DD or DMS, the linear dimension of a unit varies with latitude and longitude and is not intuitive. It is important to record the location of the blast and the seismograph as accurately as possible with available equipment. Many blasters using DD do not record sufficient decimals to allow future location. Figure 1 provides an example where the location of a blast and a seismograph are recorded in DD to various precision using a rounding convention and a

truncated convention. The location of a blast and a seismograph are recorded in Figure 1 to decimal degrees to 6 places. The locations R2 through R5 show the locations when rounded to the nearest number of decimals as indicated by the label (i.e. R2 is rounded to two decimal places, and T4 is truncated to 4 decimal places). As noted, the introduced error in location places the blast outside the mine, changes the direction of the blast, and significantly changes the distance to the blast.

In the example shown in Figure 1, the truncated entries at two decimal places the blast over 1,600 ft (500 m) to the north when rounded to two decimals and 2,300 ft (650 m) to the southeast when truncated at two decimals. At 45 degrees of latitude, one second of latitude is equal to 101 feet (30.9 m), and one second of longitude is 71.9 ft (21.9 m), so simply rounding DMS to the nearest second can easily result in an inaccuracy on the order of 30 to 60 feet (9 to 18 m). This consequential effect can be doubled if the location of a seismograph or home is rounded to the nearest second. Due to the rugged terrain of many European sites, GPS signal quality must be verified, and errors due to terrain shielding or atmospheric interference must be noted.

2.4.6 Location of the nearest structure

While the seismograph records must also record the location of the seismographs placed near the blast, regulatory requirements often require indicating the location of the nearest property or properties and identifying them by distance, direction, and type of structure. To clarify the location of these structures, the property address and GPS location allow them to be easily identified later. Furthermore, some jurisdictions require monitoring of utilities, public infrastructure, schools, or specific structures. In urban blasting, common in Europe's dense cities, this may include schools, listed buildings, or critical infrastructure. Photographic documentation of structural condition pre- and post-blast is advisable and often mandatory. The distance to these structures from the nearest blast hole should be reported to the nearest rounded foot within 100 feet (1/3 of a meter within 33 meters), within 10 feet at distances up to 500 feet (3 m within 150 m, and within 30 feet (10 m) for distances greater than 500 feet (150 m). The direction should be recorded using azimuth or one of the subcardinal directions (NE, SW, etc.).

2.4.7 Weather conditions

Details of weather conditions at the time of the blast are essential to explain any unusual off-site measurements of sound pressure and dust or fume that may occur from sources other than blasting. Such details should include temperature, wind speed and direction, cloud cover, and temperature inversion. The source of this information should be identified so those records can be verified. Excellent sources of accurate weather conditions can be found online at websites such as www.wunderground.com. European regulators increasingly focus on environmental impacts, dust, fumes, and noise. Weather records must include wind direction, temperature inversions, and cloud cover. These can be sourced from verified meteorological services or on-site instruments. Also, a brief note about the blast purpose (e.g. opening shot, trim shot, vibration-controlled blast) improves transparency and interpretation.

2.4.8 Reason for blast and performance

Sometimes, the most basic information is important but not obvious during the blast as the report is prepared. Was this blast an opening lift shot or a routine bench blast? Was it the first blast using a new explosive or initiation system? Did it do what it was supposed to do? A simple description will help recall the shot later and allow others to understand the outcome of the blast. A discussion of any misfire causes and how these misfires affected the blast can be helpful at a later time. Electronic report formats are sometimes more challenging to document these notes, but the details can easily be recorded on paper and photographed in the electronic system.

3 TIMELINE AND MAINTENANCE OF BLAST RECORDS

The regulatory requirements for storing blast records vary by jurisdiction. Blast records must be maintained to comply with local regulations. Regulations aside, at minimum, blast records should be maintained for the statute of limitations period in the governing jurisdiction where the blasting occurred. However, this standard can lead to confusion. Not only do limitation periods vary from state to state, but most states adopt a 'discovery rule' when applying a limitations period. The 'discovery rule' provides the time in which a claim must be brought does not begin to run until the claimant knows or should have known of the damage. As a result, a state with a

two-year statute of limitations may allow a claim for blasting that occurred five years before the lawsuit was filed. As a rule of thumb, blast records (and seismic data) should be maintained at least two years longer than the statute of limitations in the state where the blasting occurred. If blast records are digitised, as that storage space is not an issue, it makes sense to keep them even longer. Gone are the days when blast records may be saved in boxes in an on-site trailer or vehicle glove boxes. Blast records must be maintained in a safe and secure location. They should be maintained in electronic format or scanned and saved digitally in a cloud-based server. Alternatively, blast records should be saved on a computer with a backup hard drive.

4 CALCULATION OF HOW MUCH EXPLOSIVE CAN BE USED VERSUS HOW MUCH WAS USED

Blast plans usually incorporate loading quantities based on ideal hole diameter, drill length, and loading density (kg/m or lb/ft). However, non-ideal conditions can exist during hole loading, such as drill cuttings, rocks, and water entering holes, which may result in reduced charge weights. On the other hand, hole enlargements may occur without notice from worn drill bits, and drill steel wobble and overloading can occur. Bulk truck metering systems that are out of calibration can also contribute to inaccurate mixing and loading of products. Measuring the rise of explosives during loading by taping is critical to accurately record charge weights in each hole, either electronically or on the blast log by a blasting specialist assigned to this function. Additionally, the blaster should calculate the maximum charge per delay that keeps the blast within the regulatory limits for vibration, air overpressure, or a scaled-distance rule. The preparation of this calculation readily shows that the blaster was well within the allowable explosive charge per delay to meet the regulatory limits, allows a quick visual check of the hole loading to ensure all charges are within limits, and demonstrates that they loaded the holes under the charge weight that may cause vibration issues.

5 DEFINE WHAT N/A MEANS

A frequent and recurring problem with blast records is the use of the term 'N/A' usually in connection with seismograph information. The term always leads to confusion concerning whether a seismograph was used, a seismograph was used but did not trigger, or whether it

malfunctioned. A blaster should also include an indication of whether a seismograph was utilised with an explanation of the readings or, alternatively, why there was no recorded reading (e.g. 'No trigger') on the blast record.

6 REGULATOR ACCESSIBILITY

In addition to credibility issues and problems calculating data from a prior shot, an incomplete or inaccurate blast record can lead to regulatory issues. If uncorrectable, blast record-related compliance issues can lead to a citation. A citation becomes a public record easily accessed with an open records request from the regulatory agency. With a simple request, homeowners and their attorneys can obtain a summary of all blast citations. Unlike most personal data records, these citations identify the company and individual blaster, and the blaster may end up as a defendant in a lawsuit. The threat of litigation aside, blast records affect insurability and licensure. While technical in nature, citations related to missing or incomplete blast records reflect a failure to comply with blasting regulations. Multiple or ongoing compliance issues may impact a blaster's ability to secure insurance or employment.

7 REVIEW OF BLAST RECORDS

Memory loss over time is real, and attempting to complete the blast record later is often ineffective. Missing information on a blast record is correctable if timely discovered, but it may be impossible to recreate a shot if discovery is delayed. Data loss is fatal to a subsequent blast investigation. As pointed out above, missing information from a blast record not only affects the credibility of the blast but also severely undermines the blaster's credibility. Blasters should establish a procedure for a weekly or monthly review of blast records. At a minimum, a second set of eyes with similar or more experience and understanding of the project site should review the blast records for completeness, correctness, and legibility. These reasonable quality check tools include a plot of the blast extents as this quickly allows a review of the correct blast location relative to the site, plotting of the air overpressure and ground vibration data, as this identifies the correct seismograph locations, the maximum charge per delay, and the recorded ground vibrations, and finally, a review of the notes of the blast to evaluate if the performance of the blast was successful. Continuously updated plots of the blast extent and the vibration data often identify

the need for blast modification well ahead of any approach to regulatory limits. A review of the shot video and seismograph records provides a basic evaluation of the shot timing effectiveness and can lead to improvements in blast initiation timing or pattern layout.

8 SHOT VIDEOS

It is highly recommended that all shots be video recorded. Not only does video provide a good training tool for future shot adjustments, but it can also provide helpful information to defend a claim or lawsuit. The public perception of blasting stands in stark contrast to a well-designed shot. While it may be difficult to describe a controlled blast, a video showing minimal heave and limited vibrations is invaluable. Unless blast plans or regulations mandate otherwise, videos should be taken with the sound off or, alternatively, the video operator should be given strict instructions not to comment on the shot. Comments in response to a shot like 'holy cow', 'oh no', or worse are extremely unhelpful should the video end up in court. As with blast records, at a minimum, videos should be maintained for a period commensurate with a state's statute of limitations. Videos should be organised chronologically. Occasionally, video cameras malfunction, which can lead to claims that missing videos were intentionally deleted. If a video is lacking for a particular shot, an explanation should be made on the blast record. Additionally, if the camera's date and time are inaccurate, the video time should be recorded on the blast record. As with blast records, there should be a weekly or monthly verification of a corresponding video for each shot or an explanation for a missing video. Videos should be recorded on stable tripods from two vantage points ninety degrees from each other in full high definition or 4K format. Drones can also be very beneficial for observing rock movements that can be obscured in horizontal viewing. While mobile phone video is now generally of high quality, a handheld video often is unsteady, crops portions of the blast, or the operator watches the blast as the video frame drifts from the blast. Finally, handheld phones of blasters are often dirty and hazed, and foggy lenses from being in warm pockets are common. Photographs of the tied-in pattern, the erected seismographs, and the area after the shot was detonated, are very effective in understanding and documenting the setting. Visual records are powerful both for training and defence. All shots should be recorded with two video angles, and drone footage is highly recommended, especially

in large or inaccessible sites. Videos should be silent or free of casual commentary. Metadata must match the blast report's time and date. High-resolution imagery of pattern tie-ins, post-blast muck piles, and seismograph placements can clarify critical details.

9 SEISMOGRAPH RECORDS

While the blast report documents the conditions at the blast site, the blast preparation process and parameters, seismograph records are used to record vibrations at nearby structures, document compliance with regulations limiting vibration, and evaluate the blast performance. It is important to place the seismographs properly and document the seismograph(s) placement with a text description and photos.

9.1 Location of the seismograph(s)

It is important to describe the setup of the seismograph in the blast report. This should indicate that the unit was placed following the ISEE Field Practice Guidelines for Blasting Seismographs and, if unable, a detailed description of how and why the setup was modified. If project-specific or regulatory directives require the placement of the seismographs in a specific location or method, that should be documented. The units' precise locations should be described to detail the placement of the geophone and microphone and how the geophone was mounted, buried, sandbagged, or spiked. Geophone locations should also be recorded with a GPS, and the coordinates should be sufficiently precise to locate this spot in the future accurately. The method and model for the GPS coordinates must also be documented. Finally, the description should include the address of the property and a photograph of the setup, as well as an indication that the location was a temporary location utilising a portable seismograph or a permanent location with a stationary seismograph.

9.2 Digital seismograph records

Today's seismograph records are digital and easily stored for many years. Measurements are easily analysed using advanced software available from most seismograph manufacturers to prepare high-quality seismograph reports to demonstrate compliance with ground motion and air overpressure limits based on frequency. Therefore, it is imperative that frequency is recorded along with velocity. Most software can be used for advanced analyses, such as converting velocity in

terms of time to the frequency domain. Such FFT signal analysis can be helpful to blasters when selecting optimal delay timing patterns to offset natural frequencies inherent to geology. Similarly, derivations of velocity can be computed to show compliance regarding displacement or acceleration. Reports must include user-selected sample rate and velocity trigger amplitude. Too often, trigger amplitudes are not included in reports, or if selected too high, vibrations are not recorded. The absence of vibration measurements leaves no defence against damage claims. Preserving digital records allows advanced evaluation of zero-crossings to evaluate frequency compliance. It also allows for the correction of manually entered inaccurate data in blast reports (22 in/s instead of 2.2 in/s (559 mm/s instead of 55.9 mm/s)).

9.3 *Why was the seismograph moved?*

Often, a seismograph moves as the blasting operations progress across a site and other structures become closer to the blast. An arbitrary move can raise questions, and a simple explanation documented in the blast report at the time of the work is more accurate than memory.

10 COMPLAINTS AND NOTIFICATIONS

Homeowner notification and effective communication are the keys to reducing blast complaints. An automated phone/text system should be utilised for stationary blasting. Software and automated notifications can be utilised to determine and track notifications. All written communication should be maintained. When the communication is a formal response to a claim, certified mail should be used with return receipt confirmation, and the confirmation should be maintained. Communication should generally not include blast records or seismic data. Homeowners do not understand this information well, and engineers without blasting experience frequently misinterpret it. A complaint should always be recorded with when, where, who, and response details. A form is available for download on the ISEE website and identifies the information that should be documented. To allow a quick review in the future, all of these paper documents should be scanned and attached to the electronic blast report or physically attached to the blast report. The response to a homeowner's complaint should be directly related to the complaint. If the complaint concerns a failure to warn of a blast, an offer should be made to notify the homeowner of future

blast events. If the complaint concerns a damage claim, an investigation should be initiated. A failure to respond may lead to a lawsuit and the possibility of punitive damages. A complaint should also provide a good opportunity to review all relevant blast records.

Europe's high public awareness around blasting makes stakeholder management critical. Complaint logs should record who, when, where, and how each complaint was received and resolved. A structured, respectful response process, possibly using ISO 10002 guidance, can prevent escalation. Importantly, seismic data should not be casually shared with the public; such information can be easily misunderstood without context.

On occasion, a request may be made of the blaster to allow equipment or property to remain within the blast zone with the agreement that the party leaving the equipment (crusher, loader, conveyor, etc.) accepts the risk of damage to the equipment. In these instances, the agreement must be written and recorded, with both parties clearly articulating the risks and agreeing by signature to accept the transfer of liability. In many cases, damage has occurred, and the verbal agreement to accept the risk of damage is quickly forgotten or ignored when damage occurs. A signature from the management of the operation is necessary as the individuals agreeing to accept the risk may not have authority. This document should also be attached to the blast report for future reference.

11 EXAMPLES OF REPORTING ISSUES

The following examples highlight some of the authors' experiences evaluating blast reports from blasters across the USA. These examples are provided as tangible aids to aid blasters in improving reporting quality, not as punitive measures for the blasters who prepared them.

11.1 *Blast GPS coordinates located outside of a quarry*

A review of over 150 blast reports documenting the work completed by a blasting contractor at a quarry was performed by one of the authors to provide expert testimony on behalf of the blasting contractor. Upon plotting the provided GPS coordinates for the blasts performed over four years, twelve of the blasts were plotted outside the mined area, with one being located nearly a half-mile (800 m) to the north of any mined area. This challenged the site regression analysis, and a significant portion of data had to be dismissed to prepare a site model. Similarly, the location of

DIAGRAM OF SHOT
(Grid for Shot Number, Location, and Delay System)

Seismograph Information

Comments:

Location of shot?
Nearest structure?
Seismograph information?

Figure 3. Blast report example 2.

electronic blast reporting system without apparent notice of the actual values entered. As a result, the electronic files for the blasts were reviewed, and the actual maximum peak particle velocity was 0.22 in/s (5.5 mm/s), and the air overpressure was 119 dB. About 3% to 5% of manually entered data was erroneous in the author's approximation. When multiple entries were noted on a blast report, it added a challenge to understand what work was performed.

11.4 Varying Reporting Requirements

Blasters are mobile and go to the rock. In most cases, geology was not considered heavily in establishing geopolitical borders. While performing work, blasters face a patchwork of regulations conducting work as they move from one project location to another. In the example provided on the following page, the differences between state regulations in Kentucky and Pennsylvania are highlighted. Kentucky requires a sketch of the blast pattern, whereas Pennsylvania is prescriptive with weather documentation. Kentucky requires record retention for at least 5

years, while Pennsylvania only requires 3 years. Pennsylvania requires recording of the scaled distance, whereas Kentucky does not. While these differences are subtle, noncompliance violates regulation and may provide an avenue for discrediting the blaster. Some jurisdictional reporting is significantly different than the examples provided, and it is incumbent that the blaster know and understand this before beginning work at a site. Data retention periods vary across Europe. While some jurisdictions require retention for as little as three years, others extend this to ten or more. Given the 'discovery' rule common in civil law systems, keeping digital backups of blast logs, videos, and vibration records beyond the minimum statutory period is advisable. Centralised electronic records (compliant with ISO 27001 for data security) offer the best long-term solution.

12 CONCLUSION

European blasting professionals must navigate a diverse and evolving landscape of regulations, public expectations, and technical complexities. Whether using paper or electronic systems, blast

documentation must be clear, complete, and tailored to site and jurisdictional requirements. Electronic systems offer unmatched efficiency and auditability but must be implemented thoughtfully to avoid propagating errors. Ultimately, proper record-keeping is a critical component of professional, compliant, and defensible blasting operations, serving the operator, the public, and the industry at large.

Blast reports can be collected and stored electronically, but they must be complete, correct, and clear and meet the varying requirements for minimum recording and record-keeping in the jurisdiction where the work was completed. Blast reports are the blaster's first line of defence in case of a claim or the first place to look if a blast did not perform as desired. Customisable electronic records can be created for each project site and jurisdiction, and the entry of electronic reporting allows for rapid review by others, automated quality checks, and the importation of helpful information to plan and execute the blast. Blast records may be needed for a long time in the future, and the details recorded while loading the blast and the immediate reaction to the blast as recorded at the time of the blast will be the best memory. Finally, the auditing of blast reports can identify issues before an issue arises, and it is important to audit and verify input information to ensure an accurate report is prepared.

REFERENCES

Stiehr, J.F. 2011. *ISEE Blaster's Handbook, 18th Edition*. International Society of Explosives Engineers, Cleveland, OH, pp 633-635.

[RETURN TO CONTENTS PAGE](#)

The influence of mine water on blast design - Paper 2

T.N. Little

Blasting Geomechanics Pty Ltd, Perth, WA, Australia

ABSTRACT: This paper is the second in a series on blasting in water-affected ground, focusing on blast design adjustments required to accommodate mine water conditions. Its primary aim is to provide practical guidance to blast designers and drill-and-blast teams on managing water within and around the blast pattern. The paper introduces two tools developed by the author: a Blast Management Framework and a Blast Design Requirements (BDR) classification scheme. Adopting a 'best technical strategy' approach, the paper addresses blast design in large-scale surface mining operations where water presence poses significant challenges. Section 2 outlines a structured design process tailored for water-affected blast volumes. Section 3 presents recommendations for adjusting drill pattern geometry when blasting below the groundwater table (GWT), highlighting its role as a spatial engineering control critical to achieving primary blasting objectives. Section 4 discusses explosive selection and Section 5 deals with charge design modifications suitable for saturated ground, while Section 6 examines timing design changes, including the use of minimum response time (T_{min}) calculations and firing sequence strategies for wet conditions. Section 7 explores the elevated risk of explosive malfunction in water-affected blasting, detailing key contributing risk factors. Section 8 reviews the influence of water on blasting emissions, and Section 9 considers how water can be deliberately incorporated into a blast design. The paper concludes by summarising key blast management activities essential for implementing a best-technical-practice approach to blast design in saturated ground.

1 INTRODUCTION

1.1 Introduction

Blasting operations are unique in three important ways: they are potentially high consequence (high hazard but low residual risk), they have rapid design and implementation feedback cycles (need a cautious start and then accelerated learning), and they occur early in the value chain of extraction operations (can be used to leverage downstream processes). Because of these three unique and important features a blast management framework consisting of design management and

implementation management is indicated. Figure 1 illustrates the components of such a Blast Management Framework (BMF). It should be noted that only the Blast Design Management Process is addressed in this paper due to space restrictions.

Blast design management processes, involves: blast planning inputs (collecting blast design inputs, site conditions, any special conditions and site constraints) and establishing the blast design requirements (BDR) classification. The next stage involves, developing the blast design; undertaking design and risk analysis and obtaining the required approval and sign-offs.

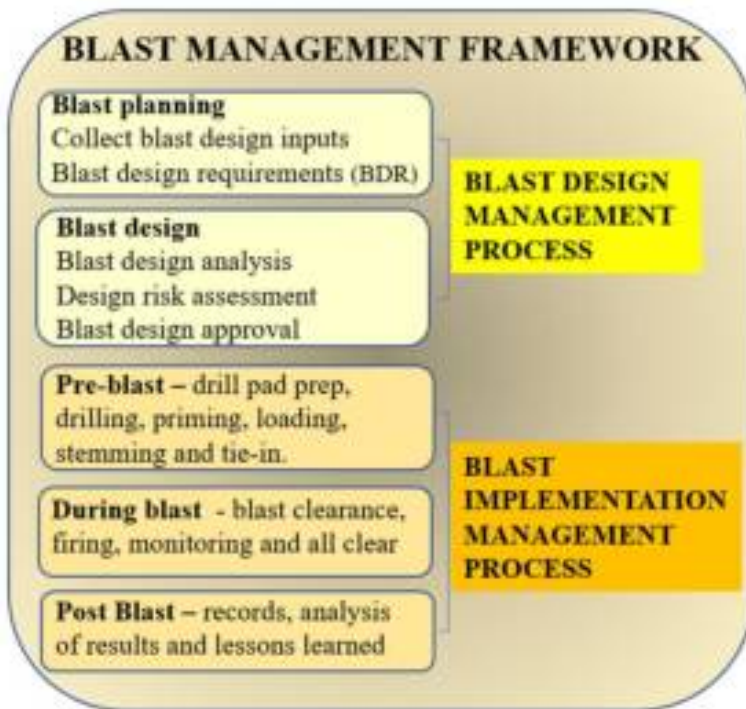


Figure 1. Components of a blast management framework (BMF) (Little 2017, 2022).

1.2 Blasting saturated ground

Water is a blasting parameter that is usually not explicitly considered in the blast design process except for the type of explosive used and potentially for using reduced timing (Slaughter 2020). Furthermore, based on the information in

Tables 1 and 2 water in the blast volume is controllable, but moisture in the ground may be uncontrollable. The presence of water in a blast pattern and hence blastholes, is controllable in the sense that mine dewatering and depressurisation can be planned and implemented. In addition, individual blastholes can be dewatered or other local dewatering can be used provided the inflow rate is not too high as in the case of artesian or dynamic water.

With regard to the presence of water (wetness) in blastholes being regarded as an uncontrollable

Table 1. Controllable blast design variables.

DRILL DESIGN	CHARGE DESIGN	BLASTING
Drillhole diameter	Type of explosives	Blast clearance
Drilled length	Energy of explosives	Firing system
Drilling pattern	Primer	Delay timing
Drilling accuracy	location(s)	Blasthole dewatering
	Design of charge	
	Charge lengths	
	Initiation accuracy and firing pattern	

Table 2. Uncontrollable design variable.

GEOLOGY / ROCK MASS	GROUND / SPECIAL CONDITIONS
Rock mass properties = intact rock properties + geological structures.	<i>In situ</i> rock stress,
Lithology = geometry of the rock texture/ fabric and properties	Reactive ground
	Temperature,
	Moisture content

parameter affecting the blasting outcomes, (Dehghani *et al.* 2019) suggests impacts on explosive in any of the following three ways:

- The wetness decreases the explosion temperature, thereby neutralising part of the energy released by the explosive material, which otherwise could be spent on rock fragmentation.
- The wetness/moisture tends to decompose the explosive material and produce nitric acid.
- The wetness/moisture may negatively affect the stability, sensitivity, detonation velocity, and strength of the AN-based explosives.

All of these impacts can be managed and also to counter these detrimental impacts, evidence is accumulating that water can have substantial benefits if marshalled in an innovative way, but this requires a detailed understanding of the mechanism involved and a best available technology approach.

1.3 Best Technical Solution Blast Strategy

The degree to which the drill design is changed to cater for the mine water conditions largely depends on the blasting strategy adopted (Little 2025). Three options were considered:

Option 1 - the traditional minimum unit cost for drill and blast strategy;

Option 2 - the best technical solution strategy; and

Option 3 - a standardised but overconservative strategy.

In this paper the best technical solution is the strategy adopted. However, it is conceded that

Option 3 may become more relevant as drill and blasting operations become more automated in the near future.

It is considered that the best technical solution for blasting in water affected ground will involve the following drill and blast adaptation or refinements in one form or another:

Data gathering: Geology, structural geology, hydrogeology site investigation and orebody knowledge.

Moisture and water detection using in-drill moisture sensors and blasthole dipping technologies.

Blast design and design analysis: Systematic blast design process based on risk analysis and that includes a design analysis step prior to releasing the design.

Pre-blast dewatering: When feasible, reducing water content before blasting to minimise energy absorption and pore pressure risks.

Explosive and initiation selection: Using water-resistant emulsions or slurry explosives in saturated conditions. Use best available electronic detonator systems.

Robust design for reliability: Based on blast-by-blast risk analysis and utilising best available technology. Optimising electronic detonator sequencing to manage pore pressure changes and energy distribution. Minimising sleep time between explosive loading and firing. Employing double priming.

Blast monitoring and modelling: Using sensors and/or numerical models to simulate and refine blast design parameter. This includes: Fragmentation - image based, Vibration



Figure 2. The Blast Design Requirement (BDR) classification scheme (Little 2017, 2022).

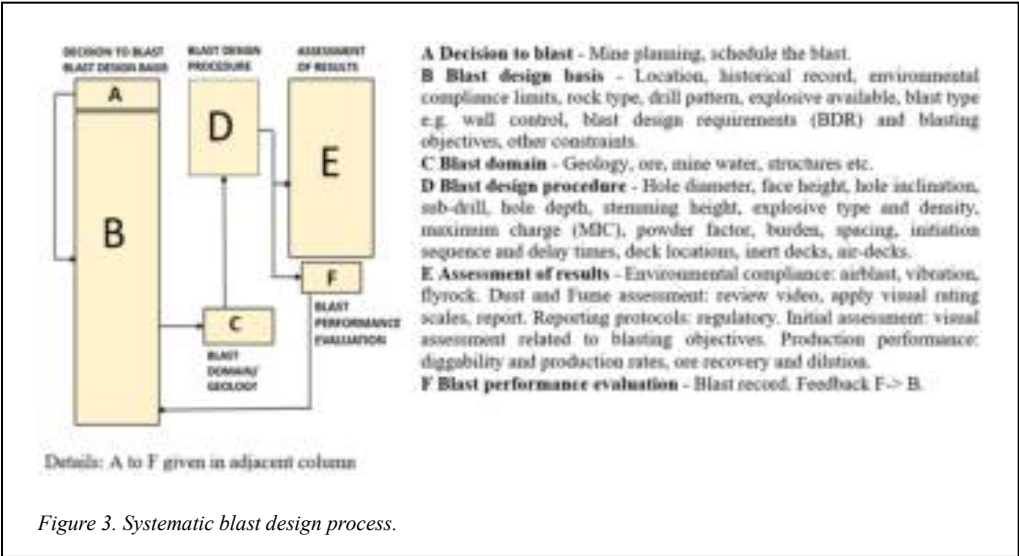


Figure 3. Systematic blast design process.

monitoring - predicts vibration impacts; supports proactive design adjustments; Energy output and detonation quality (pressure) - detects if VoD is compromised by water; validates explosive choice; Blast timing / performance Imaging - high-speed blast cameras to captures flame front, timing, and rock movement dynamics; useful in timing verification and sequence optimisation.

2 BLAST DESIGN AND DESIGN RISK MANAGEMENT

Blast design parameters need to be selected based on the blast design requirements (BDR), the blast type, the site geology (blasting domains), the mine design and plan, any site special conditions (of

particular interest here is all aspects of surface and groundwater) and any site constraints related to design, equipment, explosive and initiation types. Figure 2 shows the blast design elements. The blast design requirement (BDR) classification scheme is given in Figure 2. (Little 2017, 2022). In this context groundwater is considered to be a managerial blasting objective and in particular a special condition.

Developing a blast design should follow a systematic process. Figure 3 provides such a process as follows: a decision to blast, the blast design basis, blast domain, the blast design procedure, an assessment of results, and blast performance evaluation.

Figure 4 illustrates 9 water-related blast design

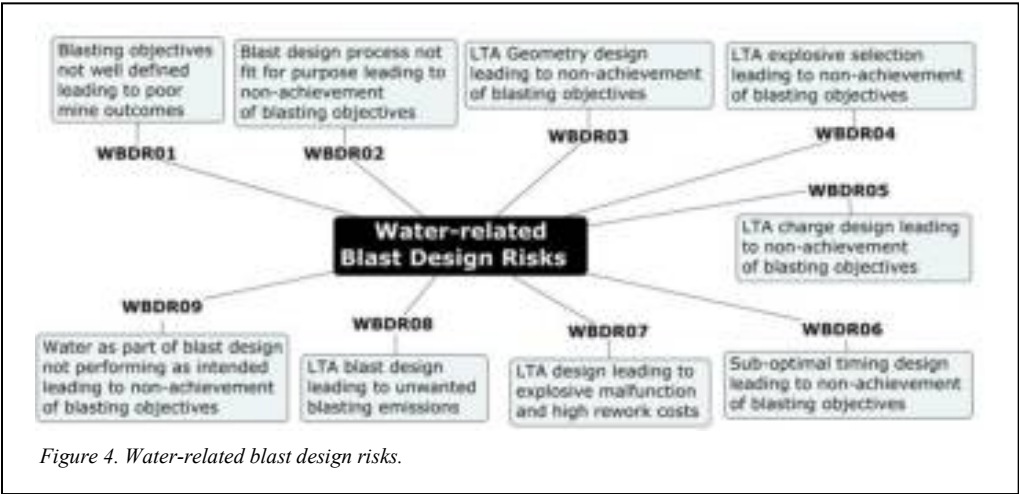


Figure 4. Water-related blast design risks.

risks and these are addressed in the respective section of the paper. For example, WBDR03 is addressed in Section 3 and WBDR09 is addressed in Section 9. The first two water-related risks are addressed in Section 1 and 2.

3 DRILL DESIGN

Drill pattern design is a critical spatial engineering control and serves as a primary communication tool between the blast designer and drill operators. It is a major determinant of the successful achievement of blasting objectives, including fragmentation, wall control, and muckpile movement. Drill design comprises several key parameters: burden, spacing, subdrill depth, stemming length, bench height, and blast size. Effective control of drilling quality, particularly in terms of collar position, hole deviation, and hole length, is essential for consistent blast outcomes.

A fundamental design question is: Where should each blasthole be located, and why? It is the author's view that the intended purpose of every blasthole should be understood by both the blast designer and the implementation teams. Blast patterns are commonly regular in form (square, rectangular, staggered, or triangular), but may also be asymmetric to better accommodate geological variability (Segui 2001).

Practical Calculation of Burden and Spacing: For initial design, Rustan (1990) proposed empirical formulas to estimate practical burden (B) based on blasthole diameter (d, in mm):

- Surface mining: $B = 18.8 \cdot d^{0.689}$
- Underground blasting: $B = 11.8 \cdot d^{0.630}$

Once the burden is estimated, the spacing-to-burden ratio (S/B) can be selected based on the rock mass, fragmentation goals, and expected drill deviations. These empirical equations provide a useful starting point when detailed knowledge of rock properties, explosive behaviour, or borehole deviation is limited. Adjustments should be made according to site-specific conditions (after Rustan, 1990):

- High rock density → Reduce burden.
- Large borehole deviation → Reduce burden.
- High explosive strength in hard rock → Increase burden.
- Low explosive strength in soft rock → Increase burden.
- Saturated ground → See adjusted guidelines below.

Blast Pattern Design in Saturated Ground: When blasting in saturated ground, burden and

spacing may require adjustments (typically $\pm 10 - 15\%$) to ensure safe and effective energy distribution.

Increase burden and spacing when water enhances energy transmission:

- Hydrodynamic enhancement in weak/jointed rock improves shockwave propagation.
- Water-filled fractures conduct energy and promote crack propagation.
- High-porosity, low-strength rock (e.g. sandstone, weathered rock): water reduces energy loss via dust generation.

Reduce burden and spacing when water diminishes blast effectiveness:

- Tight, low-porosity rock: water acts as a shock absorber, reducing energy transmission.
- Loose, saturated soils (e.g. sands, clays): increased pore pressure can cause instability or liquefaction.

Blast Design Adjustments for Explosive Reliability: Saturated ground increases the risk of explosive malfunction, such as sympathetic detonation or dead pressing. To mitigate this:

- Widen the pattern (increase burden/spacing) to reduce pressure overlap between holes.
- Alternatively, reduce blasthole diameter to decrease maximum instantaneous charge.
- Refer to Section 2.6 for additional risk management strategies.

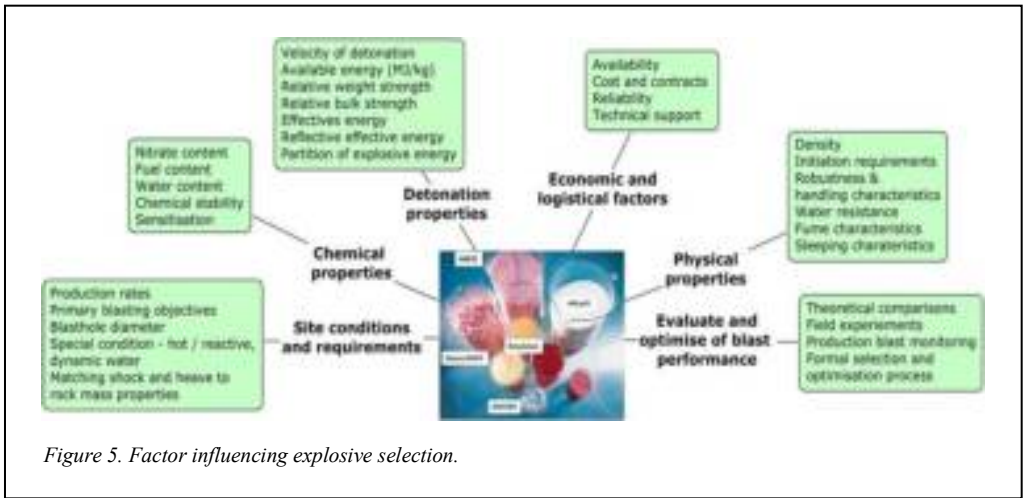
Stemming in Wet Blastholes: In wet ground, do not use drill cuttings for stemming. Instead:

- Use crushed aggregate, with particle size 10 - 25% of hole diameter.
- Increase stemming length by 10 - 20% to enhance confinement.
- Be aware of bridging or blockage risks during stemming placement in wet holes.

Stand-Off Distance for Wall Control: In saturated conditions, increase stand-off distances from final walls by 10–20%:

- Saturated rock masses often exhibit lower shear strength, increasing the risk of wall sloughing or damage.
- Additional spacing improves control and safety in wall control blasting.

Design to Minimise Sleep Time: In water-affected ground, reduce blast size if sleep time is a concern (e.g. risk of explosive degradation or leakage). Alternatively, consider:



- Lowering bench height, which shortens exposure time.
 - Staggering the blast into smaller sequences to control timing more effectively.
- Subdrill Adjustment in Wet Ground:* Subdrill (or subgrade) depth should be reduced by 10 - 30% in saturated ground compared to dry conditions. Reasons include:
- Water reduces confinement, allowing easier bottom breakage.
 - Over-drilling in wet ground can lead to overbreak, floor heave, or hole instability.
 - Proper subdrill control improves floor regularity and minimises unnecessary energy loss.
- Once the drill design is complete the blaster only has two remaining tools available, they are the charge design (Section 2.3) and timing design (Section 2.4).

4 EXPLOSIVE SELECTION

- The characteristics that influence the selection of explosives are summarised in Figure 5.
- Physical and chemical properties such as density, robustness and handling characteristics, water resistance, sleeping and fume characteristics.
 - Detonation properties such as sensitivity, explosive energy and its partition into shock and heave.
 - Economic and logistical factors such as cost, availability, and reliability and level of technical support provided by the supplier (Cameron *et al.* 1990, JKMR 1996).

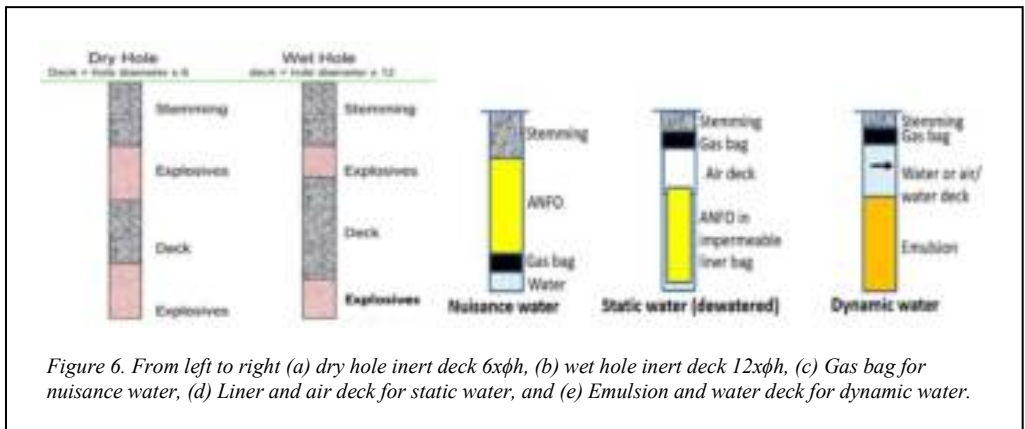
The primary ingredients of bulk commercial explosives are oxidiser (usually ammonium nitrate) and a fuel. Other ingredients are used to modify the physical and detonon properties of the explosive to match particular blasting, storage and handling requirements. These other ingredients include water, calcium nitrate, diluents, emulsifiers, high energy fuels, cross linkers, gums, gassing agents and sensitisers.

Some special liquid water conditions: The geological settings and associated water and sleep time restrictions that may also influence explosive selection or formulation. The special water conditions include: Geothermal water, Acidic Rock Drainage water, Acid Sulphate Soil water, Hypersaline water, and Hydrocarbon Reservoirs Fluid (multi-phase).

5 CHARGE DESIGN

Charge design involves selecting the explosive type and chemistry, the charge configuration (decoupling, priming, decking, stemming, charge length, base charge length) for each blasthole and selecting the in-hole initiation system type to be used, e.g. electronic.

Powder Factor in Saturated Ground: In saturated conditions, water absorbs part of the explosive energy and dampens shock wave propagation, reducing energy transfer to the rock. To counteract this energy loss, the powder factor is often increased by 20 - 30% compared to dry ground to maintain effective fragmentation. For example, if 0.5 kg/m³ is used under dry conditions, 0.6 - 0.65 kg/m³ may be required in saturated ground. However, there are exceptions:



- In weak, clay-rich or fractured ground, water may enhance fragmentation, and a lower powder factor may be appropriate to control overbreak and fines generation.
- Downers (2018) observed that in water-saturated, decoupled pre-split holes, a 20% reduction in powder factor was still effective, with acceptable wall quality - even when water surrounded all three explosive decks.

Field trials and rock mass characterisation are essential to validate powder factor adjustments.

Design of decoupled charges: Charge confinement in saturated ground is influenced by the degree of decoupling. This can be quantified in two ways:

- Decoupling Coefficient (K_d) = Hole diameter / Charge diameter.
- Coupling Ratio (R_c) $\cong$ (Charge diameter / Hole diameter)² = 1 / K_d^2 .

As K_d increases, R_c decreases quadratically, indicating reduced coupling efficiency. According to Ye *et al.* (2024), Water-Coupled Charges (WCBs) with K_d values between 1.26 and 1.71 resulted in 1.4 to 2.1 times larger rock failure volumes compared to air-coupled charges. The presence of water enhances the energy transmission into the rock by improving shock wave propagation and gas pressure coupling. The use of water as part of the blast design is further discussed in Section 8.

Design of priming: Primer location strongly influences the spatial distribution of explosive energy. Blair (2015) demonstrated that a detonator's position within a cylindrical charge affects both the stress directionality and amplitude:

- Detonation waves create non-uniform stress fields, with peak energy radiating in the direction of the wave front.

- The traditional bottom priming method may not always be optimal, especially in asymmetric or complex geometries.

In saturated ground double priming is strongly recommended, especially where product degradation or misfires are a concern. The ISEE handbook (2011, Ch. 19) supports this approach to improve initiation reliability in wet conditions.

Deck charge adjustments: Figure 6 illustrate the need to double the inert deck length in saturated ground this is done to ensure the explosive columns do not sympathetically detonate. When the borehole cannot be dewatered, loading may need to be modified to limit the pressure transferred to the borehole wall.

Design against stemming ejection: Effective stemming is critical to containing gas energy and reducing airblast, flyrock, and stemming ejection. In saturated ground:

- Water reduces the mechanical interlock of stemming particles.
- Increase stemming length by 10 - 20% to compensate for reduced confinement.
- Use clean, angular crushed aggregate with a diameter of 10 - 25% of the hole diameter.

This approach improves gas retention and maintains pressure within the blasthole.

Charge length adjustments: In saturated ground, the charge length may need to be increased to compensate for the reduced explosive performance, if explosive degradation is expected. This can be achieved by increasing the subdrill length. Alternatively, the use of water-resistant explosives or sealing techniques can help maintain the effectiveness of the blast.

Selecting the in-hole initiation system: For saturated ground conditions, electronic initiation systems are the preferred solution due to their:

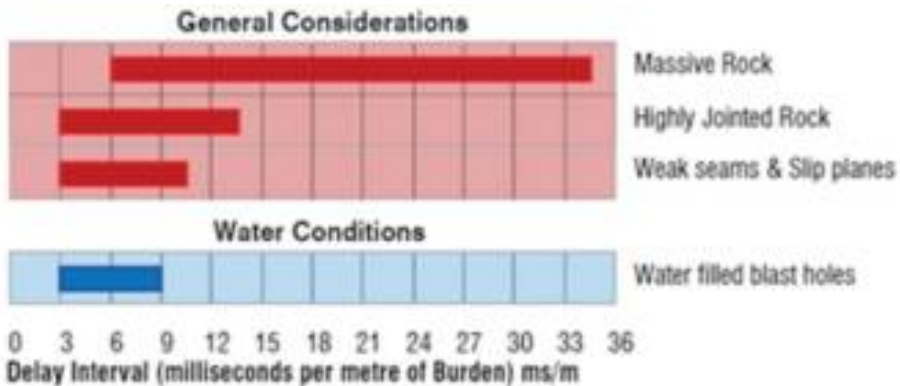


Figure 7. Suggested delay intervals between rows (Developed by John Floyd).

- Precision timing for better energy control and reduced risk of dead pressing.
- Resistance to water damage, increasing reliability.
- Flexibility in designing complex timing patterns.

In high-production or high-risk environments, electronic detonators are essential to achieving safe and effective blasting under wet conditions.

Once the charge design has been implemented the blaster has only one remaining tool available, that is the timing design which is discussed in the next section.

6 TIMING DESIGN AND BLASTING SEQUENCES

Optimising delay timing, both inter-hole and inter-row, has a demonstrable effect on key blast outcomes such as fragmentation, muckpile shape, backbreak control and ground vibration. In water-affected ground, pore-fluid interactions further modify the dynamic response of the rock mass, altering properties like stiffness, wave velocity, energy coupling and attenuation. These changes commonly reduce the minimum response time (T_{min}), the interval between detonation and the onset of significant rock movement, to values typically between < 5 and 10 ms/m.

Field Implementation: Use Inter-hole delay: $0.5 - 1.0 \times T_{min}$ and Inter-row delay: $1.5 - 3.0 \times T_{min}$.

- Measure site-specific V_p in both dry and saturated conditions.
- Compute T_{min} from your measured V_p (e.g. if $V_p = 2$ km/s, then $T_{min} \approx 0.5$ ms per metre).

- Select delays using the scaled ranges above, reducing delays in saturated weak rock by ~50 %, and in low-porosity hard rock by ~10 %.
- Validate fragmentation, muckpile heave and vibration; fine-tune toward the lower end of each range if over-break or excess PPV appears.

Design of blast timing: Timing design must consider the initiating systems, the likely interaction between blast holes, surrounding ground conditions (ore geology, waste geology and hydrogeology), the blast type and the desired blasting objectives (priorities). Minimum response time (T_{min}) can be used as a key input parameter in the design of appropriate inter-hole and inter-row delays under free face blasting conditions (Onederra *et al.* 2011).

Design of blast sequences: Some principles that apply:

- Delay used in practice Often increased (see summary for explanation).
- Pore pressure risk Use spaced timing (no < 5 ms delays).
- Charge configuration Can use decks with timed segments.
- Firing direction Always toward free face.
- Delay accuracy Use electronic detonators.
- Mixed wet/dry zones Time them as separate zones.

Summary: While water saturation theoretically shortens T_{min} by stiffening the rock and accelerating wave transmission, practical timing often requires longer delays to accommodate pore-pressure dissipation, prevent deck collapse and ensure consistent energy transfer.

7 RISK FACTORS FOR EXPLOSIVE MALFUNCTION

There are a number of explosive performance issues that can be severely exasperated by the presents of groundwater. The malfunction of an explosive charge refers to the tendency of explosive material and/or detonators, to work with reduced strength and/or at the wrong time, or not to work at all. Explosive malfunction impacts rock fragmentation, general safety and production. Therefore, the malfunction of an explosive and/or detonator in one borehole (acceptor) caused by the detonation of explosive charges in adjacent boreholes (donor) remains a serious risk and is one of the biggest problems in the blasting industry. In this paper explosive malfunction is taken to include: misfires, deflagration, sympathetic detonation, and dynamic pressure desensitisation of detonators and bulk AN-based explosives (dead pressing).

The EPC-UK risk assessment assumes bulk or package emulsion and uses 4 other risk factors singly or in combination (Farnfield et al. 2023):

- Distance between charges < 1 m → High risk.
- Distance 1 - 2 m, Water yes → High risk.
- Distance 1 - 2 m, Water no, Geology yes, Confined yes → High risk.
- Distance 2 - 3 m, Water yes, Geology yes→ High risk.
- Distance 2 - 3 m, Water yes, Geology no, Confined yes → High risk.
- Distance > 3 m, Water yes, Geology yes, Confined yes → High risk.
- Legend: Reduced distance higher risk, Water presence increased risk, Weak or broken ground increases risk, Over-confined hole increase risk.

The ORICA risk assessment uses 11 risk factors with the following being the criteria for high risk:

- Presence of water - Most blastholes are

wet or Heavy rain likely between loading and firing. Note the presence of water correlates very strongly with dynamic misfire probability.

- Site history - Confirmed cases in this geological domain using this blast design or This blast design has never been used before in this geological domain or ground conditions.
- Minimum distance - Distance between holes <= 30 x hole diameter.
- Jointing/structure - Conglomerate, or ground heavily jointed & presence of open bedding planes.
- Ground quality & strength - Voids, soft bands, conglomerate, broken ground or pre-conditioned ground likely or known in this location.
- Decking - Decking using drill cuttings or with < 30 x hole diameter between explosive decks.
- Blast confinement - Fully confined blast with no free face or Sinking cut, drop cut, box cut, trench.
- Blast powder or energy factor - Powder factor, >0.9 kg/bcm or Energy factor, >2.3 MJ/bcm.
- Multiple priming - Explosive decks are all single primed, all primers are placed at closest point to donor charge or later firing charge length is <2 m.
- Blasthole deviation, Redrills & Collar Accuracy - Collar markout imprecise or not controlled. Design vs. actual collar locations not reviewed prior to loading. No measurement or control of blasthole deviation. Top hammer drilling on benches more than 15m.
- Loading accuracy and charge lengths - Blasthole loading not monitored or controlled for overloading. Primer location in blasthole not controlled. In decking

Table 3. Surface markets misfire cases vs. charge separation distances (Orica 2021).

Distance Between Decks	<=15 x hole diameter	= 84% of cases
	15-30 x hole diameter	= 15% of cases
	>30 x hole diameter	<1% of cases

Distance Between Holes	<=30 x hole diameter	= 80% of cases
	30-35 x hole diameter	= 20% of cases
	>35 x hole diameter	No reported cases

applications, potential for gravel decks to be loaded less than design lengths.

Table 3 summarises misfire occurrence based on distances between blastholes and decks, using the hole diameter as the unit of measure. Distances at which dynamic pressure misfires have occurred vary significantly. The presence of water, path of transmission and localised geology may significantly alter distances required for pressures to reduce to levels to avoid misfires. Reduced separation distances may increase misfire risk. Acknowledging that suitable separation distances to prevent misfire are highly variable and location specific the information in Table 3 may be useful if quantitative risk analysis is being undertaken.

It is interesting to note that neither of the two risk assessment guidelines above make reference to timing. So, the question remains, what can be done with timing to reduce the likelihood of explosive malfunction in wet ground? Some guidance on timing is given here. Utilise high-precision electronic detonators to ensure that the delay sequences are maintained accurately. This precision minimises the chance of misfires or out-of-sequence detonations. Also implement carefully planned delay intervals between holes or blast stages. Staggering the energy release reduces the instantaneous load on the weak ground and helps dissipate the energy gradually, lowering the risk of a malfunction cascading into a larger failure.

8 DESIGN TO REDUCE BLASTING EMISSIONS

This section covers blast emission causes and

controls and in summarised in Figure 8.

Overall, using water (or blasting in saturated ground) can be highly beneficial in controlling flyrock, fumes, airblast/noise, and dust. However, careful management is required to prevent water contamination. In design terms, this means integrating methods to recover or treat water post-blast, as well as optimising charge loading, delay timing, and borehole water management to achieve the desired suppression effects without adverse environmental impact. With regard to fume the adoption of proactive fume mitigation management is suggested (Julian *et al.* 2015).

9 WATER AS PART OF THE BLAST DESIGN

A number of recent papers (mostly from China) have discussed introducing packaged water or utilising water present within the blast volume as part of the blast design. Figure 9 is an attempt to summarise the utilisation of water as part of the blast design as space does not allow further discussion on this important and innovative approach.

10 CONCLUSIONS

Based on the technical review conducted in this paper, the following conclusions are drawn to guide blast design in saturated ground conditions:

- Define the Blasting Design Requirements (BDRs) early in the process, including both managerial and technical blast objectives and measures.
- Select a clear blasting strategy, whether

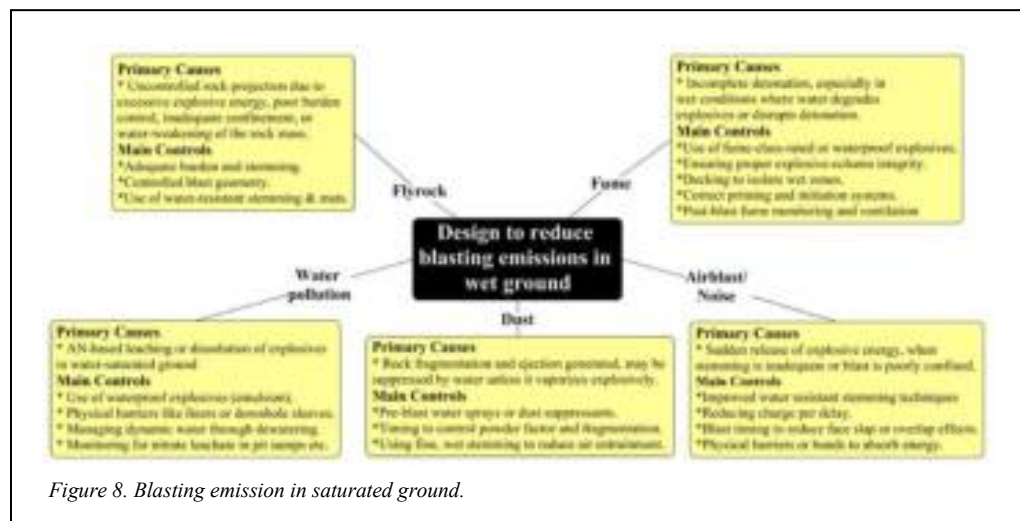


Figure 8. Blasting emission in saturated ground.

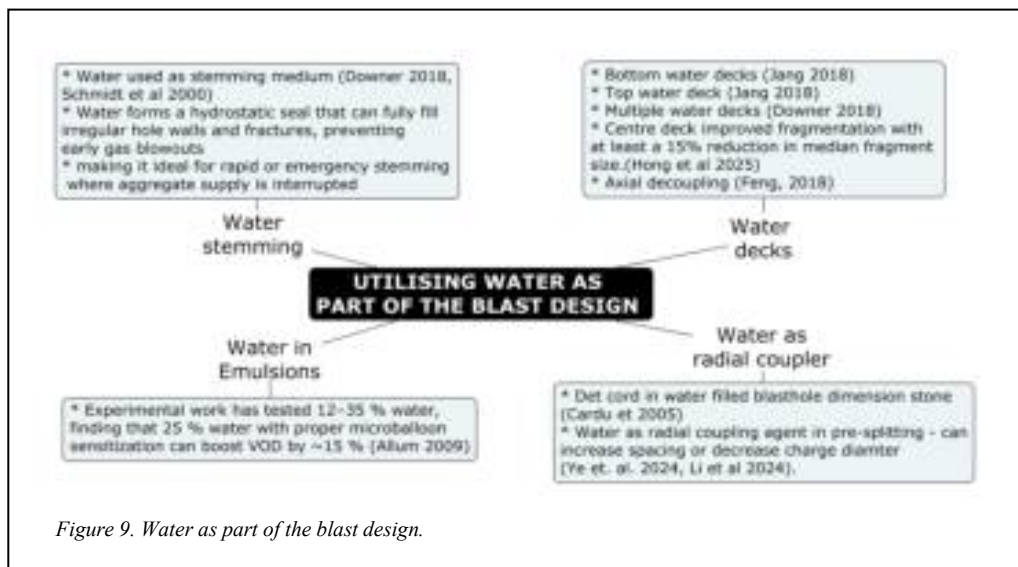


Figure 9. Water as part of the blast design.

a performance-optimised technical solution or a more conservative, standardised approach, and conduct a corresponding risk assessment to guide design choices.

- Validate the mine water conceptual model and integrate it into site-specific blast planning (Little 2025).
- Engage proactively with the mine water management team to align dewatering strategies (proactive or reactive) with blasting operations.
- Apply risk-informed, site-specific adjustments to drilling parameters such as burden, spacing, subdrill, and bench height, based on ground and hydrogeological conditions.
- Use robust, water-resistant emulsion explosives and reliable electronic initiation systems that maintain performance in wet conditions.
- Use risk-informed explosive charge design including emissions controls, misfires and explosive malfunction controls.
- Adjust blast timing and sequencing to manage dynamic rock-explosive-water interactions during detonation.
- Attempt to understand rock-explosive-water interaction and consider investigating and utilising water as part of the blast design.
- Use innovative but practical blast design implementation (to be published in Paper 3).
- Monitor critical aspects of the blast dynamics (to be published in Paper 4).

- Measurement of the blast results related to the blasting objectives and measures (Paper 4)
- Evaluate post-blast outcomes (wall and floor damage, stress changes, and hydrogeological changes (Paper 4).

Blasting in saturated ground presents unique challenges and potential opportunities that require coordinated planning, tailored design, and adaptive implementation strategies. A structured, risk-reward informed approach enhances both the safety and effectiveness of blasting operations under these special but specific site conditions.

REFERENCES

- Allum, J. 2009. A Study of the behaviour of emulsion explosives. This thesis is submitted in partial fulfilment of the requirements for the Degree of Ph. D. *Cranfield University*.
- Beale, G. 2013. Water and slope stability - the application of a new. *Slope Stability 2013* – Dight P M (ed). Australian Centre for Geomechanics, Perth, ISBN 978-0-9870937-5-2.
- Buffham, B. 2012. A few timing concepts. *Quarry*, February 2012.
- Blair, D. P. 2018. Vibration modelling and mechanisms for wall control blasting. *12th International Symposium on Rock Fragmentation by Blasting*, Luleå Sweden 11-13 June 2018.

- Cameron, A. & Grouchel, P. 1990. The effects of the quality of the bulk commercial explosives on the blast performance. *Proc. 3rd Int. Symp. on Rock Fragmentation by Blasting*, Brisbane, Aug. 26-31. pp335-343 (1990).
- Cardu, M., Patrucco, M., Lovera, E. & Michelotti, E. 2005. Quarrying by explosive and diamond wire in hard dimension stones. *Brighton Conference Proceedings 2005*, R. Holmberg *et al.* 2005 European Federation of Explosives Engineers, ISBN 0-9550290-0-7.
- Dehghani ,H., Babanouri, N., Alimohammadnia, F. & Kalhori, M. 2020. Blast-induced rock fragmentation in wet holes. *Journal of the Society Mining, Metallurgy & Exploration* (An Official International Peer-reviewed ISSN 2524-3462).
- Downer, 2018. Presplitting in wet ground - techniques and design considerations. *6th Australian ISEE Conference*, Perth, Australia, November 2018.
- Farnfield, R., Bermingham, L., Skinner, S. & Adderley, G. 2023. Shock and transient pressure damage to electronic detonators – a practical test. *EFEE's 12th World Conference on Explosives and Blasting*, Dublin, Ireland, 9-12 September 2023.
- Feng, X.D. 2018. An investigation into water decked blastholes using numerical modelling. A thesis submitted in partial fulfilment of the requirements for the award of *Bachelor of Engineering (Mining)*, UNSW.
- Little T N, 2017. Blast design requirements classification system for blast purposes. In proceeding of EFEE 9th World Conference on Explosive and Blasting, Stockholm, Sweden.
- Little, T N. 2022. Ground improvement blasting - further development of the BDR classification scheme. In *proceedings of EFEE 10th World Conference on Explosive and Blasting*, Maastricht, Netherlands.
- Little, T.N. 2025. The influence of mine water on site investigation, ground conditions and blast planning. *14th International Symp on Rock Fragmentation by Blasting*, Antalya, Turkiye, April 7-10, 2025.
- Onederra, I. & Esen, S. 2011. Selection of inter-hole and inter-row timing for surface blasting - an approach based on burden relief analysis. *Julius Kruttschnitt Mineral Research Centre -The University of Queensland*, Qld, Australia.
- ORICA, 2021. Detonator misfires due to dynamic pressure open cut & quarries. *Product Bulletin Australia Pacific Asia* - May 2021.
- Resource. Best practice guideline - prevention of explosive misfires in blasting applications. *Resources Safety & Health Queensland*, 2022.
- JKMRC, 1996. Open pit blast design - analysis and optimisation. Editor: A Scott. *Published by: Julius Kruttschnitt Mineral Research Centre*.
- Julian, L., Musunuri, A., Sellers, E.J. & Kanchibotla, S.S. 2015. Mitigation of post-blast fume in soft ground and wet conditions. *11th International Symp on Rock Fragmentation by Blasting*, Sydney, NSW, 24–26 August 2015.
- Segui, J.B. 2001. Asymmetric blasting: a rock mass dependent blast design method. *Proceedings of the EXPLO 2001 Conference*, Hunter Valley, NSW, October 2001, pp289-296 (The Australasian Institute of Mining and Metallurgy: Melbourne).
- Schmidt, M.F.C., Szucs, A. & Worsey, P. 2000. Use of water stemming for rock excavation and splitting. *Proceeding of International Society of Explosives Engineers Conference* 2000.
- Slaughter, S. 2020. Assessing water influence on drop cut blasting. *19th March, 2020* www.quarrymagazine.com.
- Ye, Z., Yang, J., Chen, M., Yao, C., Zhang, X., Ma, Y. & Zhou, C. 2024. Difference in rock-breaking characteristics between water-coupling blasting and air-coupling blasting. *International Journal of Rock Mechanics and Mining Sciences* Volume 183, November 2024.

A methodology to define drill and blast designs for drawbells at the Chuquicamata underground mine

A. Berg, P. Arriagada, R. Cortés & N. Jamett

Codelco - Chile

J. Henríquez, H. Cortés & D. Morales

Enaex - Chile

ABSTRACT: This paper presents a methodology applied at the Chuquicamata underground mine to define differentiated drill and blast designs for drawbells according to geotechnical units. The approach integrates vibration monitoring, real burden analysis, and drillhole deviation, supported by field testing, 3D scanning, and numerical simulations. Near-field vibration data were used to determine propagation (K) and attenuation (α) coefficients, enabling PPV modelling and calibration. Two PPV thresholds were identified: one to ensure effective excavation at the centre, and another to control breakage at the edges. Designs were corrected for drillhole geometry, burden alignment, and void constraints. In sectors with increased undercut-production height, multi-phase blasting and adjusted drawbell height were applied to prevent over confinement due to volume expansion. This integrated methodology improved breakage control, reduced structural damage, and enhanced operational continuity. It provides a validated framework for drawbell blasting design in large-scale caving mines with complex geotechnical and geometrical conditions.

1 INTRODUCTION

Chuquicamata underground mine represents a significant transition in Chilean mining, evolving from one of the world's largest open pit operations to a modern panel caving underground mine. This transformation entails not only massive infrastructure and cultural changes, but also critical technical challenges, particularly regarding the design and performance of drawbells, the essential elements initiating ore extraction (Brady & Brown 2006).

The proper design of drawbells has a direct impact on ore recovery, infrastructure stability, and overall operational continuity (Berg *et al.* 2024). In the early stages of the mine's operation,

a standard design approach was used, applying generic parameters for drilling and blasting regardless of varying rock mass conditions. However, field experience revealed significant deviations in blasting performance, propagation of caving, and unwanted damage in the Crown Pillar and extraction-level pillars (Jensen & Savigny 2000). All of this considering that a small percentage of the total explosive energy is effectively used for rock fragmentation (Sanchidrián *et al.* 2007).

To address this, a methodology was developed that integrates geotechnical characterisation, advanced blast monitoring, and *in situ* adjustment of design parameters. This paper presents a systematic approach to define differentiated drill

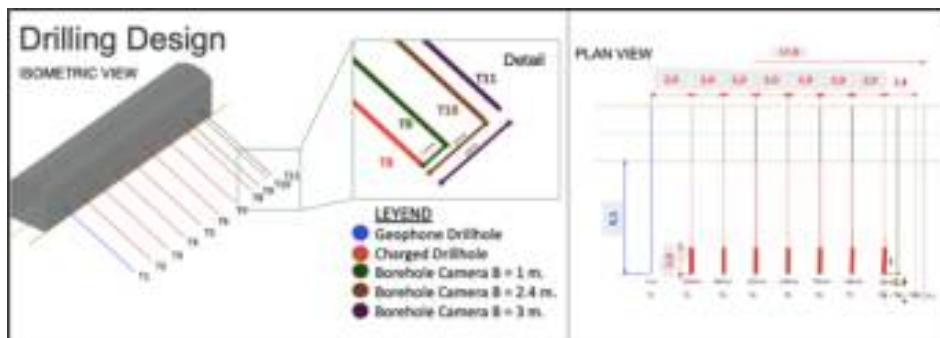


Figure 1. Cross Hole test drilling design.

and blast designs for drawbells according to the geotechnical characteristics of each sector, enabling more precise control over energy input, minimising vibration-induced damage (Gonzalez & Segarra 2014), and enhancing the overall efficiency of the caving process.

2 METHODOLOGY

The methodology developed for the Chuquicamata Underground Mine comprises six integrated stages: (1) geotechnical unit definition, (2) Cross Hole testing for rock mass response characterisation, (3) wave parameter estimation (Singh 2010), (4) vibration calibration and damage thresholds (McKenzie 2009), (5) design adjustments with geometry-based corrections and (6) volume constraints and multi-stage blasting strategy.

2.1 Geotechnical unit identification

The drawbell sectors were classified into Basic

Geotechnical Units (UGTBs) using the geotechnical characterisation previously developed by the Geology and Geomechanics departments, based on the latest approved geological and geotechnical plans. These UGTBs were defined considering rock quality designation (RQD), GSI values, joint spacing, and lithological variations. Each UGTB is associated with specific mechanical and structural behaviours that influence wave propagation, burden response, and susceptibility to damage

2.2 Cross hole testing

Cross Hole tests were conducted to characterise the propagation of explosive-induced waves under near-field conditions. Geophones were installed at the undercut level alongside a drill pattern that was partially loaded with explosives in order to capture the vibration waves generated by the charges.

The test design included:

- Point charges with predefined inter-delay times between blasts.

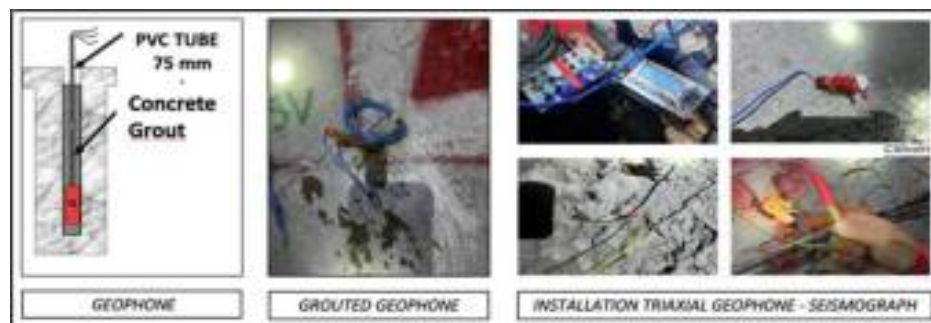


Figure 2. Installation of geophones and instrumentation.

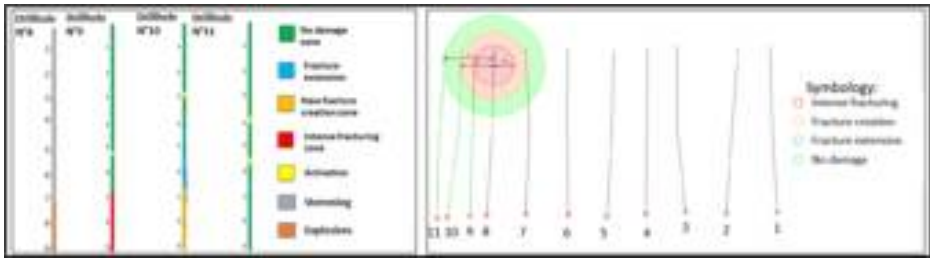


Figure 3. Damage analysis in observation drills.

- Instrumentation with geophones to record particle velocity and waveforms.
- Exploratory holes to allow visual inspection of damage to the rock mass.
- Empty holes to evaluate the attenuation behaviour of the wave across known gaps.

Using this information, the propagation coefficient K and attenuation coefficient α were derived through regression fitting of the empirical equation (1) (Siskind *et al.* 1980):

$$PPV = K \left(\frac{Q}{R^2} \right)^\alpha \quad (1)$$

Where:

PPV: Peak Particle Velocity
 Q: Explosive charge Kg
 R: Distance from the blast source
 K: Propagation coefficient
 α : Attenuation coefficient

Where PPV is the peak particle velocity, Q is the explosive charge, and R is the distance from the blast source. These coefficients are unique to each UGTB and provide a basis for predicting

vibration intensity and energy dissipation across the rock mass.

2.3 Wave Parameter Estimation

The vibration signals were successfully captured by the geophones, yielding seven distinct and isolated readings, which could be analysed independently. The waveform data allowed for the reconstruction of an elemental wave response for each initiation event.

Using the recorded particle velocity and charge geometry, a vibration model was developed by plotting the H&P factor (charge/distance²) against the measured PPV. The resulting curve showed a strong fit to a power regression, with a coefficient of determination $R^2 = 0.9805$.

From the vibration analysis, the following dimensionless coefficients were obtained:

$K = 705.68$: Wave propagation coefficient.

$\alpha = 0.5329$ Wave attenuation coefficient.

These parameters were then used to calibrate the PPV prediction model for each UGTB and to simulate realistic vibration propagation from

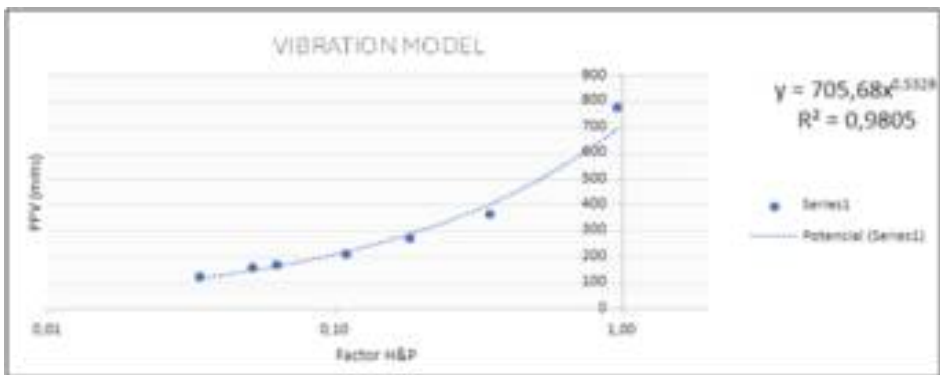


Figure 4. Geophone reading and vibrational model.

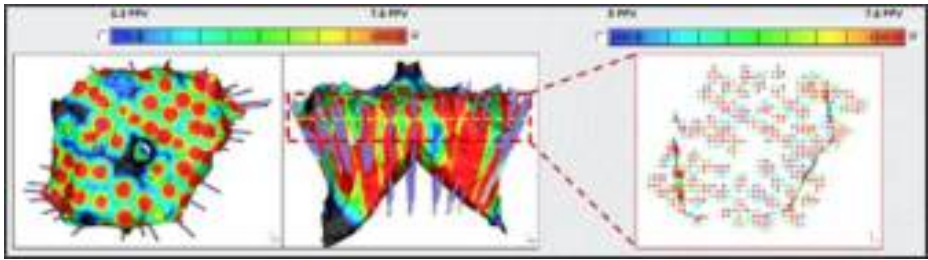


Figure 5. Vibration calibration and damage thresholds.

production blasts under site-specific geological conditions (Figure 4, Geophone waveform + PPV curve).

2.4 Vibration calibration and damage thresholds

An empirical calibration of vibration levels was performed using simulations in I-Blast, integrating real data from several drawbells excavated in the QIS geotechnical unit. These drawbells were selected because they had complete and reliable records of drilling parameters (including orientation and length), explosive loading, and post-blast excavation geometry obtained through 3D scanning.

Using this comprehensive dataset, the real burden model, explosive quantities, and timing sequences were input into the software. Simulations were then run to replicate the vibration field generated by each blast event.

Since the final excavation shape of these drawbells was accurately known, the vibration scaling in the model was iteratively adjusted to reflect energy interactions consistent with actual rock breakage. Special focus was placed on reproducing the interaction between blast holes

that successfully contributed to excavation both at the centre and the edges of the drawbell.

Vibration field calibration using I-Blast based on 3D-scanned excavation geometry of drawbells in UGTB QIS (Figure 5).

This calibration process led to the identification of two characteristic PPV values:

- PPV_centre: the minimum particle velocity required to initiate effective rock breakage and cave propagation at the central zone of the drawbell.
- PPV_edge: a slightly higher vibration threshold required at the peripheral zones, especially near the roof and lateral boundaries, to compensate for geometric complexity and energy dissipation.

Furthermore, the analysis revealed that the effective radius of interaction between charges that led to successful excavation was approximately 1.68 metres. This value closely matched the 1.77 metres predicted by the Near-Field Vibration Model Abacus at equivalent PPV levels, confirming the accuracy and applicability of the calibrated model.

This approach enabled blast designs to be fine-tuned both centrally and peripherally,

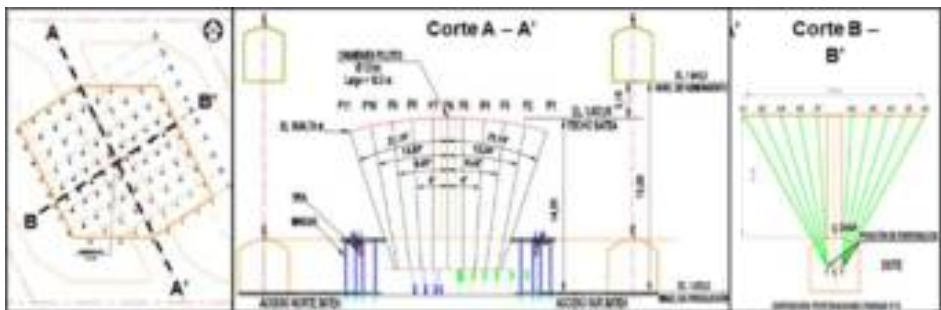


Figure 6. Plan view and longitudinal section, drawbell design MB N.



Figure 7. Longitudinal section, drawbells apex view.

improving excavation control and reducing the risk of underbreak or overbreak near critical structural zones

2.5 Design adjustments with geometry-based corrections.

In addition to adjustments based on vibration levels and explosive energy, the drawbell design also required correction based on geometric constraints, particularly at the apex zone.

The designs implemented in panels MB N2N3 and S2S3 (Figure 6) present a slight curvature when viewed in longitudinal section, but maintain a straight profile at the roof level.

This design reflects the geometric reality observed in construction and is consistent with the excavation profiles obtained from 3D scans of completed drawbells (Figure 7).

The APEX 0 reference marks the lowest constructible apex based on burden and geometry criteria. Here, the burden is defined as the shortest distance between a blast hole and the free face. The physical response of the rock mass to explosive detonation depends strongly on this condition.

When a compression wave propagates perpendicular to a free face and reaches that boundary (Figure 8), it reflects as a tension wave, generating the stress conditions necessary for fragmentation and displacement. However, if the upper portion of a blast hole lacks a perpendicular

free face (as in the top section of steep holes), the wave becomes trapped or ‘anchored’, producing fragmentation primarily by compression in a limited zone near the charge.

To counter this effect and ensure effective breakage, the trench geometry was redefined with a design criterion of perpendicularity to the free face, resulting in a naturally curved blast face. This geometry closely resembles the post-blast excavation shapes observed in 3D-scanned drawbells at Chuquicamata (Figure 9).

By integrating these geometric corrections into the drilling plan, the design ensures consistent burden conditions across the drawbell height, optimising energy use and improving excavation quality throughout the structure.

2.6 Volume constraints and multi-stage blasting strategy

One of the key design constraints in the drawbell blasting process was related to the available void volume for receiving the blasted material. In the N41 sector, the vertical distance between the production and undercut levels was increased from 18 metres (as in MB N2N3 and S2S3 sectors) to 22 metres. This design adjustment aimed to provide greater robustness to the Crown Pillar by increasing its thickness.

However, maintaining the same roof-to-undercut vertical distance while increasing the drawbell height led to a significant rise in the total

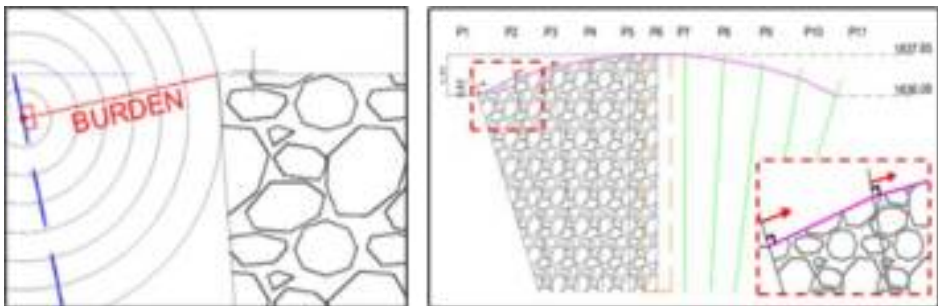


Figure 8. Perpendicularity of the burden.

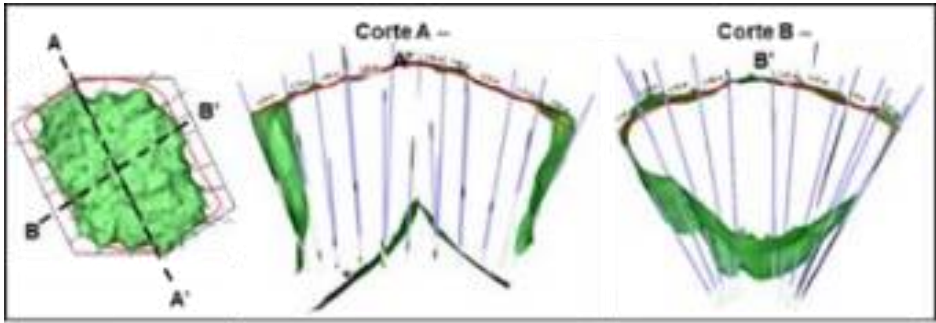


Figure 9. Longitudinal section and cross section of a typical drawbell.

excavation volume. The problem arises when this total spalled volume, due to rock expansion after blasting, is greater than the available void volume in the receiving system (chimney + trench + projection). This mismatch is caused by the difference between the *in situ* rock density ($\sim 2.6 \text{ t/m}^3$) and the post-blast loosened density ($\sim 2.0 \text{ t/m}^3$), which results in approximately 35% volume increase.

As a solution, a multi-stage blasting approach was adopted (Figure 10):

- Phase I is detonated first, followed by material extraction,
- Then Phase II is detonated once sufficient void space is available.

This volume-based adjustment ensures the

physical feasibility of the operation, preventing over-confinement, maintaining fragmentation quality, and protecting the integrity of the infrastructure above the drawbell.

2.7 Final Design Parameters and Drilling Summary.

Based on the vibration calibration, geometric corrections, and volumetric constraints described in previous sections, a final drill and blast design was established for the drawbells developed in the N41 sector of Chuquicamata underground mine. The design focuses on achieving efficient fragmentation while ensuring compatibility with available void volume, minimising the risk of confinement,

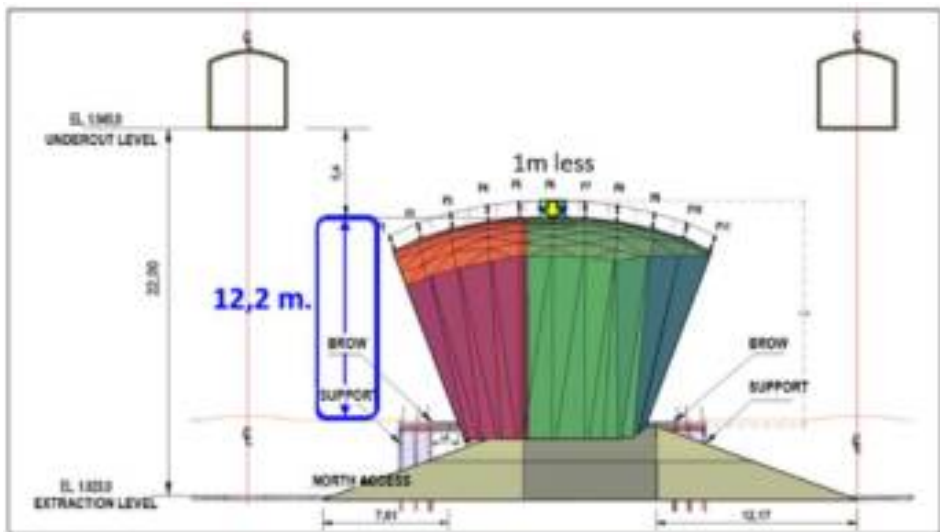


Figure 10. Multi-stage blasting.

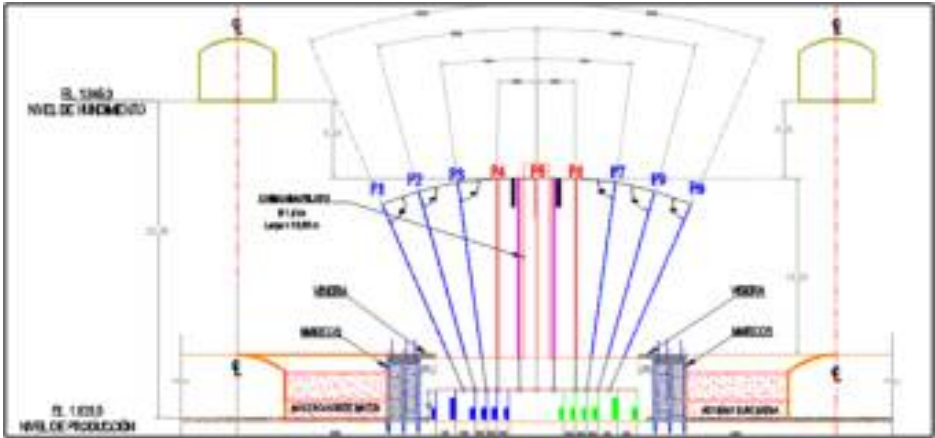


Figure 11a. Final drawbell design.

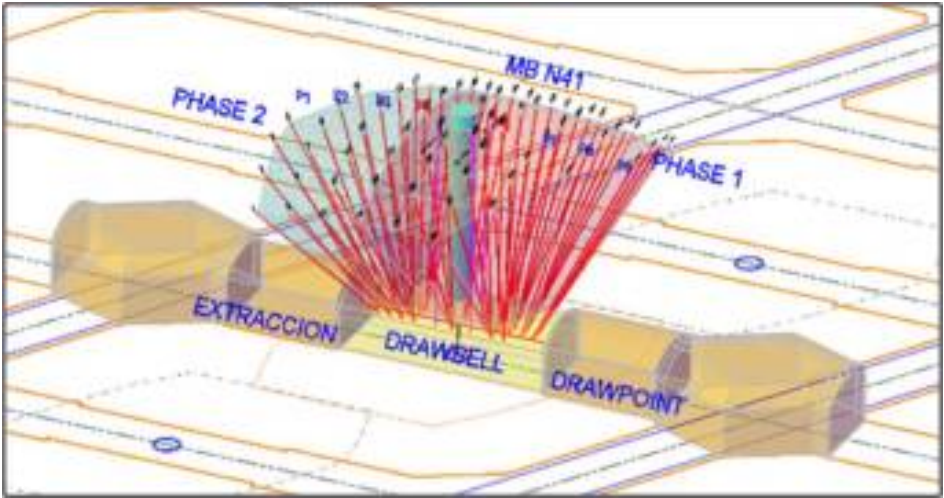


Figure 11b. Final drawbell design.

and preserving infrastructure integrity.

The final drill pattern consists of 66 blast holes with a diameter of 3 inches, distributed to conform to the optimal burden and spacing criteria derived from empirical calibration and burden simulation.

Key parameters of the final design are summarized in Table 1.

The burden range of 2.3 to 2.5 metres and spacing of 2.0 to 2.7 metres were selected to ensure wave interaction in both central and peripheral zones, as validated in Section 2.4. The

total drilled length corresponds to the complete trench design, and the perforation factor of 0.46 m.b./m³ confirms the balance between efficiency and fragmentation quality.

This configuration allows for controlled energy distribution across the height of the drawbell, aligned with the trench geometry and the available void volume. The drilling pattern supports a two-phase blast sequence, optimised for a final drawbell height of 12.2 metres, as discussed in Section 2.6

Table 1. Drilling summary -final design parameters.

Parameter	Unit	Value
Hole diameter	inches	3
Number of holes	units	66
Burden	m	2.3–2.5
Spacing at base	m	2.0–2.7
Total drilled length	m	815.6
Excavated volume	m ³	1,756.41
Area incorporated by trench	m ² / trench	640
Drilling factor	m.b. / m ³	0.46
Drilling intensity per incorporated area	m / m ²	1.27
Trench height	m	12.5

3 CONCLUSIONS

This study presented a methodology for designing drawbells in underground caving operations using geotechnical zoning, real burden analysis, vibration modelling, and geometric corrections. The main conclusions are as follows:

- Differentiating blast designs by geotechnical units allows site-specific adjustments to energy, vibration thresholds, and drill patterns, improving excavation efficiency and reducing structural risk.
- Cross hole testing and near-field monitoring enabled the calibration of wave propagation and attenuation parameters (k and α), critical for modelling ppv and defining breakage limits across UGTBS.
- The integration of 3d scan data, drillhole deviation, and explosive loading sequences into JKSimBlast allowed the identification of two vibration thresholds: one to ensure effective central breakage and another for peripheral detachment.
- The real burden analysis showed significant deviation from theoretical models, requiring the adaptation of designs based on actual collar positioning and hole trajectories.
- Geometric corrections based on apex 0 and the burden's perpendicularity to the free face helped align simulated designs with the observed post-blast excavation geometry.
- Operational feasibility was further constrained by the available void space. a single-phase blast of 13.2 m height exceeded the receiving volume, requiring a two-phase strategy and a final height reduction to 12.2 m to ensure safe fragmentation and material recovery.

The combined application of these techniques has led to safer, more efficient, and technically validated drawbell designs at Chuquicamata underground mine, providing a replicable framework for other large-scale caving operations.

REFERENCES

- Berg, A., Jamett, N. & Orellana, A. 2024. Application of burden analysis and drilling deviation control in drawbell blasting. *Internal technical report. Codelco Chuquicamata Underground Division.*
- Brady, B.H.G. & Brown, E.T. 2006. Rock mechanics for underground mining. *Dordrecht: Springer.*
- González, J. & Segarra, P. 2014. Vibrations induced by blasting in underground mining: Empirical models and near-field control. In *Fragblast 10*, New Delhi, India. Leiden: CRC Press.
- Jensen, H.M. & Savigny, W.W. 2000. Draw control and fragmentation in block caving mines. In Hustrulid, W. & Bullock, R. (eds), *Underground Mining Methods: Engineering Fundamentals and International Case Studies*: 595–604. Littleton: SME.
- McKenzie, C.K. 2009. Damage criteria for near-field blasting in massive rock masses. In *Proceedings of the 35th Annual Conference on Explosives and Blasting Technique*. Denver, CO: ISEE.
- Sanchidrián, J.A., López, L.M. & Segarra, P. 2007. Energy factors and efficiency in rock blasting. *International Journal of Rock Mechanics and Mining Sciences* 44(6): 775–786.
- Singh, P.K. 2010. Blast vibration assessment and its control – an overview. *International Journal of Rock Mechanics and Mining Sciences* 47(6): 919–931.
- Siskind, D., Stagg, M., Kopp, J. & Dowding, C. 1980. Structure response and damage produced by ground vibration from surface mine blasting. *Report of Investigations - United States, Bureau of Mines.*

[RETURN TO CONTENTS PAGE](#)

Impact of using water gel explosives (Aquagels) compared to heavy ANFO in open-pit mining operations

E. Núñez Recabal, R. Gil Greene & L. Manríquez

MAXAM Chile, Santiago, Chile

E. Tobar

Codelco Chile – División Gabriela Mistral

ABSTRACT: In open-pit mining, optimising unit processes significantly impacts operational efficiency and profitability. Drilling and blasting are fundamental stages in the extraction value chain, directly influencing material fragmentation and the performance of loading and hauling equipment. Heavy ANFO (HANFO) has historically been the most widely used blasting agent due to its low cost and ease of application. However, its fixed density and limited water resistance pose constraints in certain geological settings. This study evaluates the replacement of HANFO with an aquagel explosive across various lithologies in an open-pit mine. Based on HANFO's drilling patterns and loading rules, a six-month comparative analysis was conducted, yielding measurable results in operational efficiency and cost reduction. The findings indicate an average reduction of over 19% in the charge factor without compromising fragmentation or extraction equipment performance. Three case studies were analysed: Case 1 - Adjusting aquagel density to optimize released energy according to rock mass conditions; Case 2 - Evaluating variations in contour design geometry and optimising charge density based on sector lithology; Case 3 - Analysing different blasting configurations by modifying geometric parameters in contour and production designs to reduce drilling costs. Operational indicators validate the quality and safety of replacing HANFO with aquagel, demonstrating its technical and economic feasibility.

1 INTRODUCTION

In the context of open-pit mining, the drilling and blasting process is not only the first step in the rock extraction chain but also a critical determinant of the downstream operational performance. Proper blast design directly influences rock fragmentation, shovel productivity, crusher throughput, and overall mine-to-mill efficiency. Consequently, optimising explosive energy application is essential to improve operational continuity and reduce unit costs.

Heavy ANFO (HANFO) has traditionally been the preferred explosive in open-pit operations due to its low cost and ease of use. However, its limitations in density flexibility and poor performance in wet boreholes reduce its effectiveness under varying geotechnical conditions. In contrast, aquagel explosives offer the advantage of adjustable in-hole density and superior water resistance, enabling better control of energy distribution within the rock mass.

This paper presents the results of a six-month comparative study evaluating the substitution of

HANFO with aquagel in a large-scale copper mine in northern Chile. The analysis considers three specific cases in which the explosive loading strategy and blast design were adapted based on lithological variability and production needs. Key performance indicators included explosive consumption, charge factor, fragmentation quality, and productivity of loading equipment. The goal was to quantify the operational and economic benefits derived from switching to aquagel and demonstrate its viability as a flexible, high-performance blasting agent in challenging geological environments.

2 METHODOLOGY FOR ESTIMATING SAVINGS IN THE IMPLEMENTATION OF A VARIABLE DENSITY AQUAGEL EXPLOSIVE

The methodology used to estimate the savings associated with the implementation of a variable density aquagel explosive begins with the definition of a baseline. From this reference point, the savings are calculated based on three possible mechanisms: the reduction in explosive consumption (due to lower density), the increase in the drilling pattern (greater burden and spacing), or a combination of both. The analysis is conducted at the blast level, comparing the powder factor of each event against the one defined in the baseline.

If the powder factor of a blast is higher than the baseline, it is considered a discount from the savings (i.e. negative savings). Conversely, if the powder factor is lower, the difference is interpreted as positive savings. In cases where both powder factors are equal, it is assumed that there are no savings. The sum of these results, evaluated across all blasts executed during a defined period (e.g. one month), allows for the estimation of the total accumulated savings.

2.1 Definition of powder factor

In the Chilean context, the powder factor is defined as the ratio between the total mass of explosive used and the tonnage of the rock mass to be fragmented.

To calculate the powder factor for an individual hole, the total amount of explosive loaded (including the column charge and toe charge) is considered, along with the corresponding tonnage of rock, determined based on the burden, spacing, bench height, and rock density.

2.2 Definition of baseline

The baseline represents the technical reference point from which the effects of the aquagel implementation are measured. It is established at the time of switching explosives and must include at least the following parameters:

- Type of explosive and its density.
- Drilling pattern (burden, spacing, hole diameter, bench height, and toe charge).
- Stemming length per type of hole.
- Specific powder factor.
- Fragmentation indicators and other relevant KPIs.
- Classification of hole types: production, buffer, contour, among others.

The mine is divided into benches and phases. These, in turn, are subdivided into blasting units based on the characteristics of the rock. Each of these blasting units has a uniquely defined baseline.

2.3 Savings Calculation

The savings calculation is performed per hole type within each blast, comparing their characteristics against those defined in the baseline. The procedure includes:

- Defining the drilling pattern and loading rules for each hole type present in the blast.
- Summing the total amount of explosive used in the blast.
- Estimating the total tonnage of the rock mass affected by the blast.
- Calculating the overall powder factor for the blast and comparing it with the baseline.

The savings generated using aquagel can be classified into three categories:

- Optimisation of explosive energy, by adjusting density according to the characteristics of the rock mass.
- Expansion of the drilling pattern, increasing the volume of rock fragmented per hole.
- Combination of both strategies, i.e. reducing explosive density while increasing tonnage per hole.

2.4 Implementation of the savings model

The implementation of the model was carried out in different phases:

- Phase I: Direct substitution of the explosive.

In the initial stage, the drilling pattern, explosive density, and charge per hole defined in the baseline were maintained unchanged. The only

variable modified was the type of explosive, replacing the heavy ANFO with a variable density aquagel.

– Phase 2: Density adjustment.

Subsequently, adjustments to the aquagel's density were introduced under a progressive evaluation framework. Each change was validated based on the results obtained from the key performance indicators (KPIs) defined for the blast. If results remained within acceptable ranges, the change was approved.

– Phase 3: Expansion of the drilling pattern.

Once the density had been adjusted and if improvement opportunities were still identified (e.g. excessively fine fragmentation), the possibility of increasing the drilling pattern was evaluated. This change was also subject to validation based on empirical blast results.

– Alternative Phase: Only pattern expansion.

In cases where the characteristics of the rock mass did not allow for a reduction in explosive density, the feasibility of expanding the drilling pattern while maintaining constant density was directly assessed. As with previous phases, the approval of this change was contingent upon the performance observed in subsequent blasts.

Changes in density and/or drilling pattern were validated according to the results obtained. A direct comparison of technical data sheets between the two products to estimate theoretical energy differences was deemed invalid, as the explosives came from different suppliers and there is no standardized method for reporting the properties of industrial explosives. Such comparisons could lead to errors in calculations and incorrect conclusions.

3 CASE STUDIES

This study is based on data from a mining operation in northern Chile, a porphyry copper-type deposit covered by sterile gravels ranging from 10 to 80 metres in thickness. It is composed of lithological units such as volcanic sequences, granodiorites, diorites, and tonalitic porphyries. The dominant minerals are chalcopyrite and bornite, along with secondary sulphides such as chalcocite. The deposit exhibits zones of potassic, sericitic, and argillic alteration. Mineralisation includes oxides, primary sulphides, and mixed zones, whose distribution is controlled by lithological and structural factors of the rock mass.

Data from the first six months of the contract were analysed, a period during which the different phases of implementation and validation of a variable density aquagel explosive were carried

out. During these months, the comparative baseline and success indicators were defined. From the blasts executed during this period, three representative cases were selected to illustrate the migration from a heavy ANFO to a variable density explosive.

The geometric parameters and charging rules used for the calculations were: explosive type, hole type, burden, spacing, hole diameter, explosive density, bench height, stemming length, kilograms of explosive, and powder factor by hole type. Among these, the burden, spacing, explosive density, and stemming length were modified either individually or in combination.

Case 1 focuses on adjusting the explosive densities while maintaining the geometric parameters of the blast, with the objective of optimising energy distribution and reducing the powder factor by adapting the explosive density to each geological unit.

Case 2 proposes modifying the geometric parameters of the contour design and reducing the explosive density in certain areas to decrease effective energy and vibrations, thereby improving bench-berm stability and reducing the powder factor.

Case 3 aims to evaluate different blast configurations by adjusting both the geometric parameters and the explosive densities, in order to optimize the drilling and blasting designs according to the lithological/structural characteristics, which also leads to a reduction in the overall powder factor.

3.1 Case 1: Adjustment of Aquagel density

With the objective of optimising explosive consumption, the work focused on a production blast involving approximately 651,000 tonnes of rock to be fragmented, in an area predominantly composed of secondary granodiorite units with potassic alteration. The design of this blast was developed using the same drilling pattern defined in the baseline.

Figure 1 shows the distribution of densities for both Heavy ANFO and Aquagel within the polygon corresponding to Case 1, with the aim of optimising explosive consumption while ensuring fragmentation and extraction rates that meet the KPIs established by the client.

The densities used in buffer and production zones ranged from 0.9 to 1.15 g/cm³, which had a positive impact on the total blasting cost. This strategy resulted in a reduction of approximately 15% in the powder factor compared to the baseline.

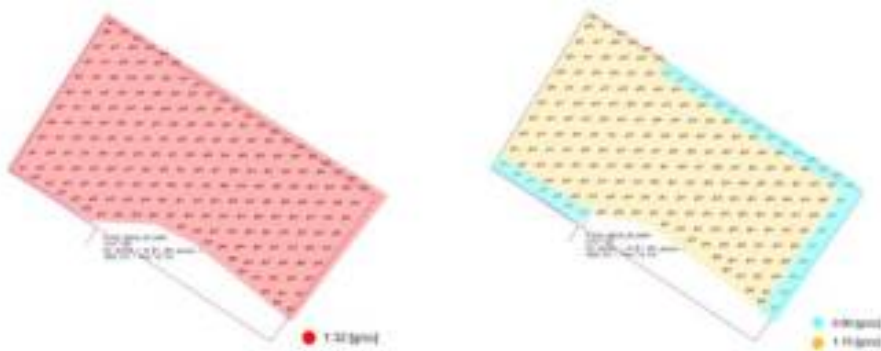


Figure 1. Comparison of densities used in case 1 of ANFO Heavy and Aquagel.

3.2 Case 2: Contour geometry optimisation

The main objective of this case was to analyse the influence of variations in the drilling pattern and the optimal charge density, adapted to the lithological characteristics of the sector.

The blast corresponded to a total of 344,450 tonnes of ore. For the savings calculation, the baseline defined for the sector was used as a reference. The following variables were modified: burden, spacing, explosive density, removal of an air deck, and reduction of stemming length. These modifications resulted in a 23.2% reduction in the powder factor compared to the conventional use of heavy ANFO.

The savings achieved were based on a 12% increase in the drilling pattern (in area), along with

density adjustments applied to the different hole types. Figure 2 graphically shows the density distribution comparing heavy ANFO with the variable-density aquagel explosive.

3.3 Case 3: Design adjustment and water resistance

This study comprehensively addressed the optimisation of the blasting process with the objective of reducing drilling costs by expanding the drill and blast pattern and varying charge densities in buffer and production zones, thereby improving overall process efficiency. Based on a detailed geological analysis, the blast design was adapted according to the specific physical quality characteristics of the rock mass. Figure 3

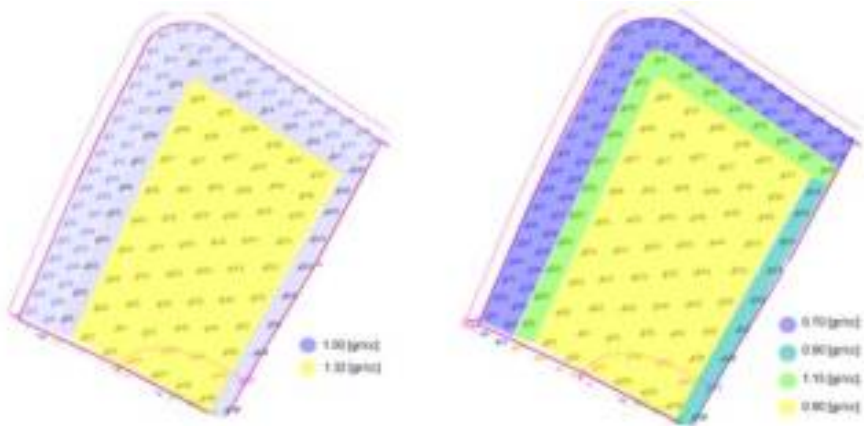


Figure 2. Comparison of densities used in case 2 of ANFO Heavy and Aquagel.

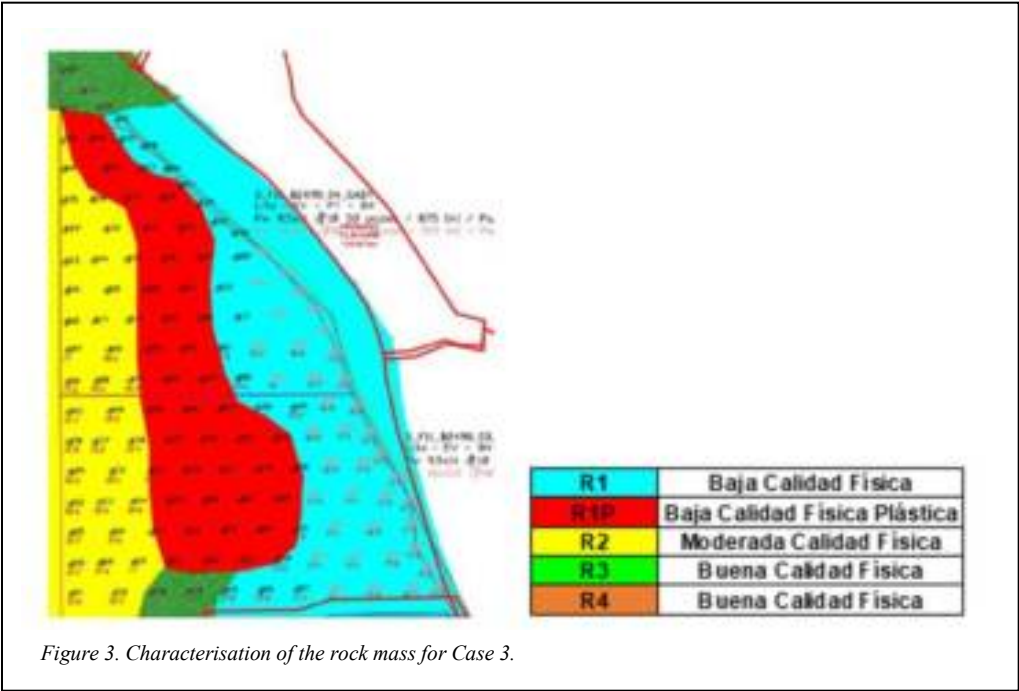


Figure 3. Characterisation of the rock mass for Case 3.

illustrates the subdivisions of blasting units based on rock properties throughout the blast polygon. In the analysed blast, a total of 638,000 tonnes of ore were removed. Of all the blast holes, approximately 15% showed the presence of water, with average water column lengths of around 7 metres (approximately half the bench height). The ability to use a variable-density aquagel explosive in water-bearing holes allows for the use of lower densities compared to those required when using heavy ANFO. This factor, combined with the expanded drilling pattern and density adjustments based on rock mass quality, resulted in a 23.4% reduction in the overall powder factor for the blast.

4 RESULTS AND DISCUSSION

During the first six months of the contract, the implementation of the aquagel-type explosive in blasting operations resulted in significant savings in explosive consumption. Compared to the use of heavy ANFO defined in the baseline, an average reduction of 19% in the powder factor was achieved over this period. Table 1 and Figure 4 show the monthly evolution of the savings percentage, calculated in relation to the powder factor established for each blasting unit in the baseline using conventional heavy ANFO.

In Month 1, the recorded savings were 3.9%, a significantly lower value compared to the

following months. This result is explained by the fact that during this period, Phase 1 of the implementation process was underway, which involved applying the aquagel under the same design conditions as the baseline, without modifications. The objective of this phase was to evaluate the explosive's performance under reference conditions and to develop improvement proposals for the subsequent phases: Phase 2, focused on adjusting charge densities by hole type, and Phase 3, aimed at expanding the drilling pattern.

Table 1. Results of fragmentation indicators.

KPI	Phase A		Phase B		
	Real	Target	Real	Target	
<2"	43.68	40	43.55	35	[%]
P ₈₀	6.91	8	8	N/A	[in]

In Month 2, Phase 2 began, during which adjustments to charge densities were made based on rock mass quality and hole type, allowing for savings of 22.5%.

The results remained stable over the following months: 21.8% in Month 3 and 21.9% in Month 4, reflecting the consolidation of the new blast design and proper operational execution of the proposed adjustments. In Month 5, a slight decrease in savings to 19.7% was observed, associated with

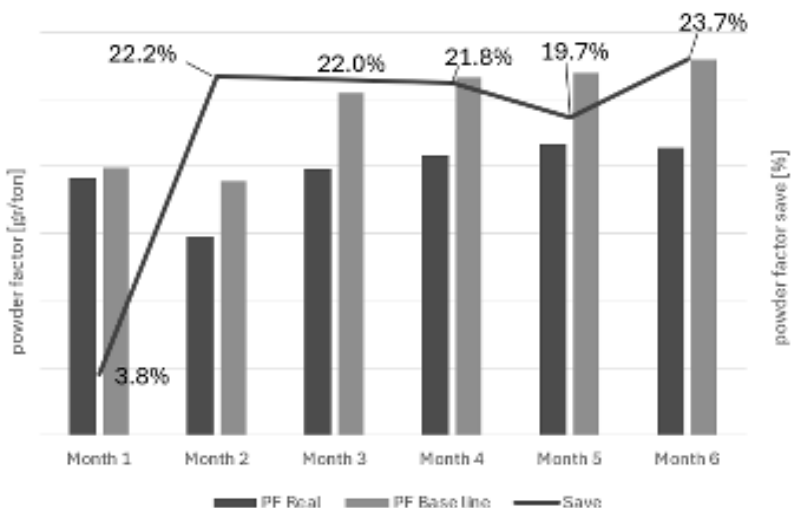


Figure 4. Comparison of real power factor with baseline.

geological variations and specific operational conditions in the area worked during that period.

Finally, in Month 6, the highest savings of the entire period were achieved, with a 23.3% reduction in the powder factor, validating the effectiveness of the implemented strategy, particularly the combination of density modulation and drilling pattern expansion based on rock mass quality.

Figure 4 shows a comparison between the powder factors defined in the baseline (with HANFO) and the actual powder factors achieved using aquagel, highlighting monthly differences and the overall improvement trend.

Fragmentation Results Regarding fragmentation, the success criterion was defined according to the operational phase of the deposit. For Phase A, the target was for at least 40% of the resulting material to be under 2 inches. For Phase B, the threshold was adjusted to 35% under 2 inches. As a complementary indicator, the 80% passing size of the particle size distribution (PSD) was expected to be below 8 inches in both phases. These targets remained within defined ranges during the analysis period, ensuring fragmentation quality for downstream processes.

Key Drivers of Savings The savings achieved in the powder factor were supported by several critical elements:

- Characteristics of the aquagel explosive, which offers an adjustable density range between 0.4 and 1.4 g/cc, adaptable to dry or wet conditions.

- Quality and depth of rock mass data, which allowed for a precise characterisation based on structural geological information from the bench floor plans under extraction, and field validation of that information.
- Data-driven decision-making, enabling the selection of optimal explosive densities by hole type and lithological unit.
- Operational discipline and field execution, ensuring the accurate implementation of the adaptive blast designs developed during planning.

Additional Savings When considering indirect benefits from aquagel implementation, such as reduced drilled metres due to pattern expansion, and lower usage of blasting accessories (detonators, boosters, air bags, etc.) the total savings can be broken down as follows:

- 73% of total savings correspond to the reduction in powder factor.
- 19% of savings result from reduced drilling.
- 8% of savings are associated with fewer blasting accessories used.

Outlook and Continuous Improvement It is understood that the versatility of aquagel, based on its wide available density range and energy profile, allows for optimisation of explosive consumption as more and higher-quality rock mass data become available. The next step to potentially increase powder factor savings involves leveraging information generated during drilling (MWD), which will enhance the geological floor plans that

describe rock mass characteristics and enable the creation of vertical profiles. This will make it possible to establish more detailed intra-hole density differentiation, thereby applying energy with greater precision according to the real conditions of the rock mass.

5 CONCLUSIONS

The implementation of a variable density aquagel explosive as a replacement for heavy ANFO has proven to be both technically and economically viable in open-pit mining operations with significant geological variability. Through a systematic analysis of three representative case studies and the monthly evolution of key performance indicators, the operational efficiency of aquagel was validated in terms of reduced powder factor, sustained fragmentation quality, and improved productivity of loading equipment.

The results obtained during the first six months of implementation show an average reduction of 19% in the powder factor, with savings reaching up to 23.3% under optimised application conditions. This performance was enabled by the adaptive use of the explosive, allowing density adjustments according to rock mass characteristics and the expansion of the drilling pattern without compromising blast effectiveness.

Furthermore, the progressive implementation of improvements across clearly defined phases, direct substitution, density adjustment, and drilling pattern expansion, allowed for controlled performance evaluation, minimised operational risks, and maximised technical learning. The resulting fragmentation quality remained within the operational KPIs, confirming the effectiveness of the energy modulation strategies applied.

Finally, the availability of detailed geological data, data-informed decision-making, and disciplined field execution were all critical factors in the success of this transition. Looking ahead, the integration of drilling-derived data (MWD) is expected to enable greater precision in intra-hole energy application, opening new opportunities for resource optimisation in mining operations.

REFERENCES

Cooper, P.W. 1996. *Explosives Engineering*. New York: Wiley-VCH.

International Society of Explosives Engineers. 2011. *Blasters' Handbook*, 18th edition. Cleveland: International Society of Explosives Engineers.

Persson, P.A., Holmberg, R. & Lee, J. 1994. *Rock Blasting and Explosives Engineering*. Boca Raton: CRC Press.

[RETURN TO CONTENTS PAGE](#)

Electronic initiation system applied in tunnel construction: focusing on the contour holes and decking mode

S. Kara & M. Laing

Hanwha, 9b Allée du Petit Bois Saint Georges/B, France

S.J. Lee, Y. Park & N. Hwang

Hanwha Corporation / Global, Cheonggyecheon-ro 86, Jung-gu, Seoul, Republic of Korea

ABSTRACT: This study conducts a comparative analysis of construction efficiency. In the first part we compared electronic and non-electric initiation systems, focusing on tunnel contour holes. Field blasting operations were executed across a variety of delays to identify the optimal delay time. The delay for the *in situ* rock mass was determined through regression analysis of blasting vibration in contour holes and for overbreak/underbreak. The blasting results with the electronic initiation system using the obtained optimal delay, applied to contour holes, were compared with the results of the non-electric initiation system. The overbreak and underbreak decreased by an average of 2.7 cm and 5.8 cm respectively, by using the electronic initiation system. Moreover, the vibration reduction rate in the region of contour holes was reduced by 11.9% based on the blasting vibration estimation equation proposed by the Ministry of Land, Infrastructure, and Transport (MOLIT 2006). Underground construction projects have become increasingly popular as cities grow in both size and density. This growth severely limits the amount of useable space for new projects on the surface, resulting in the decision to build them underground. This new policy however brings into focus the challenge of managing and reducing blasting related vibration due to the proximity of these projects to sensitive structures, utilities and foundations often present in urban environments. For this purpose, we have been required to manage the vibration by reducing the quantity of explosives used per round. Inspired by surface blasting techniques, we applied a decking mode to address and work around this challenge. We present in this paper the first attempt to blast a tunnel face using a decking method and reduce the charging per hole from 6 kg to 3 kg per hole and keep the same advance rate for this tunnel. The results are presented in the third part of this paper. In the second part we present the results of the decking mode in shaft. The use of the electronic initiation system in tunnel construction provides the users an effective tool to improve the overbreak/underbreak and keep the vibration at a low level with a large gain on the overall project schedule and budget. This is what we want to demonstrate in this paper. Keywords: electronic detonator, delay time, overbreak, underbreak, vibration, decking mode in tunnel.

1 INTRODUCTION

A non-electric detonator is a common type of detonator used in tunnel excavation, while the use of electronic detonators in recent years has also

been gaining popularity in complex tunnelling conditions (Fu *et al.* 2012). The delay time of an electronic detonator can be arbitrarily set up accurately according to in situ blasting conditions, from a minimum delay time of 1 ms to a

maximum delay time of 50,000 ms. Numerous studies have explored the benefits of precise delay time control, a notable advantage of electronic detonators, such as enhancing tunnel advance rates, changing tunnel designs through blasting vibration controls, and managing the excavation damage zone (EDZ) (Jung *et al.* 2023). Construction projects have increasingly utilised electronic detonators to mitigate damage to rock mass during the construction of intricate structures like dams and nuclear waste disposal facilities.

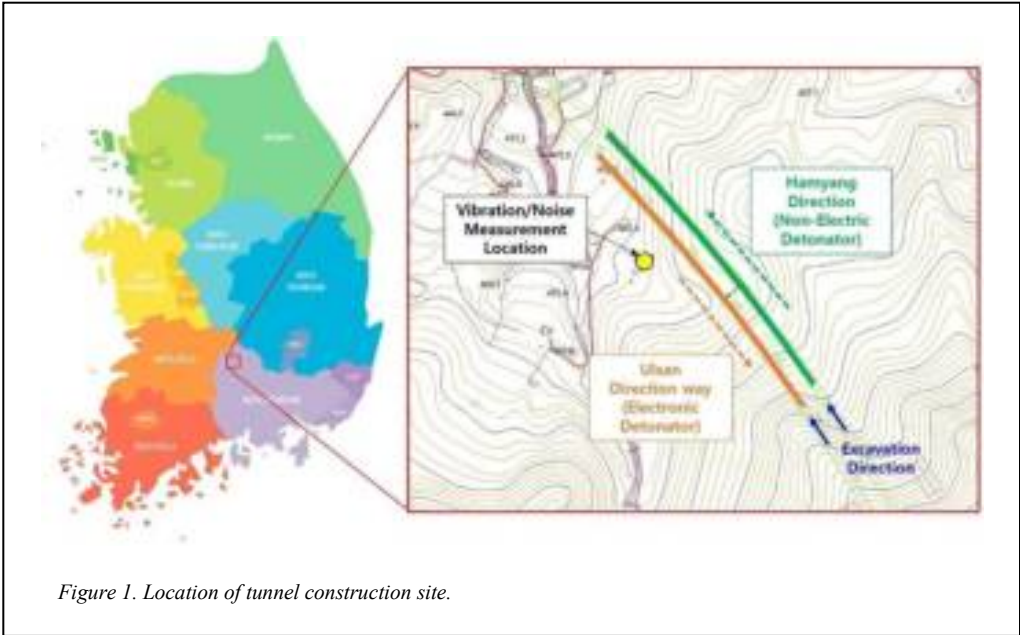
EDZ occurring during rock mass excavation is influenced by various factors, including excavation methodologies, geological conditions, and the conditions of the rock mass (Sata *et al.* 2000, Cai *et al.* 2005). The range of the EDZ is particularly contingent upon the blasting techniques and delay times of the detonator. Fu (2014) employed ultrasonic wave measurement methods to compare the range of EDZ when using electronic and non-electric detonators at a tunnel construction site. The findings from the research revealed that the range of EDZ was maximum 1.4 m for electronic detonators and 2.3 m for non-electric detonators. This study analysed the occurrences of overbreak/underbreak using displacement measurement method in fields applying both electronic and non-electric detonators. Also, it analysed the relationship between blasting vibration and the occurrences of overbreak/underbreak by analysing the peak particle velocity (PPV) of contour holes.

2 CONTOUR HOLES IMPROVEMENT

2.1 Field Blasting Operation

The test site was Zone 4 of the 14th national expressway construction project, extending across Hamyang and Changnyeong and located between Geochang and Hapcheon County in South Gyeongsang Province, South Korea. Figure 1 shows the location of the construction work and the direction of excavation. In this zone, two-lane parallel tunnels are currently under construction, employing the new Austrian tunnelling method (NATM). For the excavation work, the tunnel heading toward Ulsan utilised electronic detonators, while the one in the direction of Hamyang utilised non-electric detonators. Figure 2 shows the drilling and blasting patterns applied in the field, with the detailed specifications summarised in Table 1.

The field test was conducted in an area characterised by moderate rock, with 113 blasting tests utilising electronic detonators and 74 using non-electric detonators. The Hanwha Electronic Blasting System II (HEBS-II), an electronic blasting system developed by Hanwha Corporation, was employed for the tests (Figure 3). The HEBS-II system comprises a Blaster, Planner, Logger, Harness Wire, Connector, and HiTRONIC II detonator (Hanwha 2025).



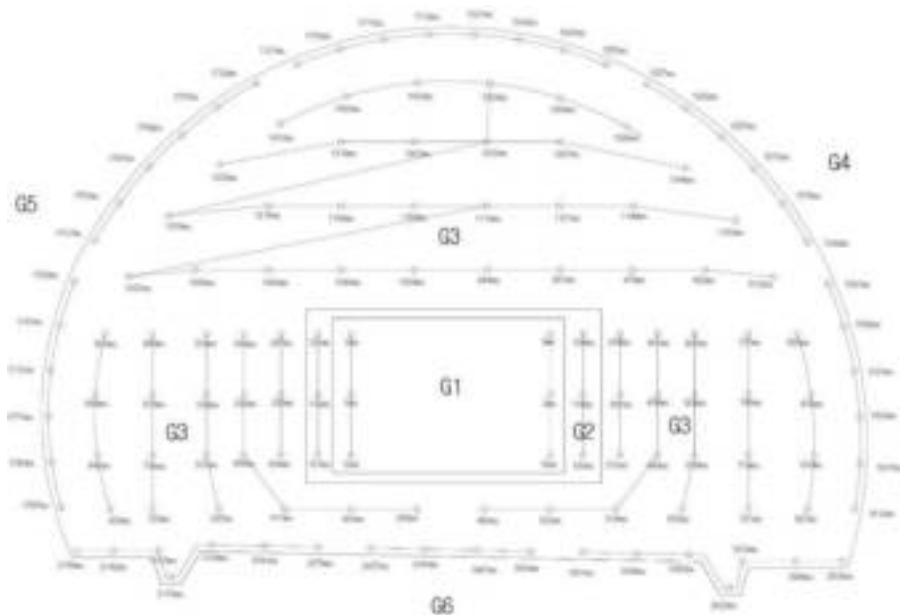


Figure 2a. Blasting design pattern of electronic detonator (delay time of contour hole is 5ms).

2.2 Results

2.2.1 Optimum delay time of contour hole using electronic detonator

This study analysed the correlation between overbreak/underbreak and blasting vibrations

according to the variation in delay times of contour holes. The delay times of contour holes were varied - 2 ms, 5 ms, 10 ms, 17 ms, and 20 ms. Contour lines of excavation sections were measured after each blasting operation to observe overbreak/underbreak. Regarding the PPVs of contour holes region, data related to contour

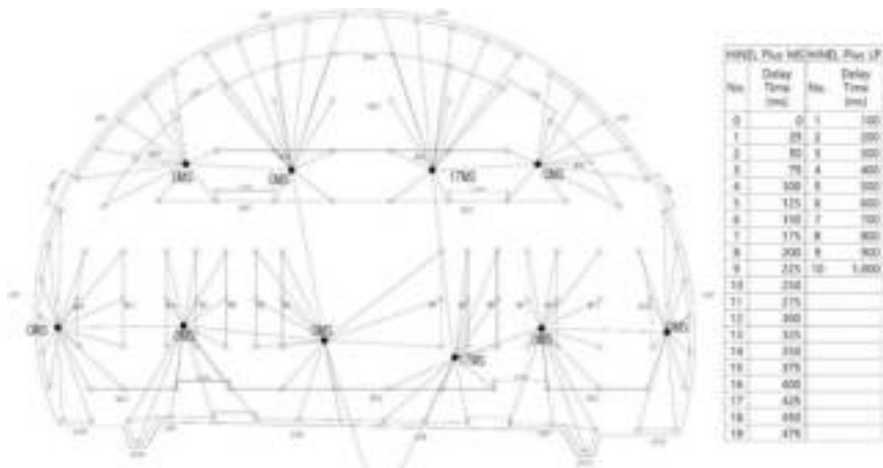


Figure 2b. Blasting design pattern of non-electric detonator.

Table 1. Blasting design pattern applied in tunnel blasting.

		Non-electric detonator	Electronic detonator
Center cut		V-cut	V-cut
Blast holes (ea)		134	134
Advance length (m)		3	3
Drilling length (m)		3.3	3.3
Diameter (mm)		45	45
Charge per hole (kg/hole)	Center	2.6	2.6
	Stope	2.4	2.4
	Contour	1.2	1.2
	Floor	2.4	2.4
Delay time between holes (ms)	Center	50	3
	Stope	25	17
	Contour	100	3/5/7/100
	Floor	100	17
Detonator		HiNEL Plus MS/LP	HiTRONIC II
Explosive		<u>NewMITE Plus II (Φ 32)</u>	<u>NewMITE Plus II (Φ32)</u>
		NewFINEX (Φ 17)	NewFINEX (Φ 17)
Specific charge (kg/m ³)		1.16	1.16
Specific drilling (m/m ³)		1.81	1.81

holes were isolated from the overall vibration measurement data. Table 2 summarises the PPVs, overbreak, and underbreak of contour holes region under each delay time condition.

Figure 2 shows the average values of PPVs under each delay time condition, along with measurements of overbreak/underbreak. The

correlation analysis between overbreak/underbreak and PPVs showed significant variance in result values likely due to rock conditions and other factors, but it still yielded meaningful results. It was observed that the regression curve from PPVs intersected with overbreak/underbreak data (Figure 4). As verified by the field test analysis, a delay



Figure 3. Electronic Blasting System II (HEBS II).

Table 2. PPVs and overbreak/underbreak developed with change of delay time.

Delay time (ms)	PPV (cm/sec)				Overbreak (m)				Underbreak (m)			
	Min.	Max.	Ave.	Std.	Min.	Max.	Ave.	Std.	Min.	Max.	Ave.	Std.
2	0.022	0.038	0.030	0.008	0.063	0.067	0.065	0.002	0.053	0.057	0.055	0.002
5	0.015	0.179	0.060	0.036	0.040	0.198	0.120	0.037	0.000	0.216	0.056	0.037
10	0.020	0.023	0.021	0.001	0.061	0.136	0.091	0.025	0.032	0.086	0.061	0.019
17	0.018	0.072	0.038	0.017	0.077	0.142	0.116	0.022	0.027	0.147	0.061	0.033
20	0.016	0.020	0.017	0.002	0.102	0.146	0.124	0.019	0.041	0.128	0.074	0.032

time of 10 ms not only decreases the occurrences of overbreak/underbreak but also effectively decreases blasting vibrations to contour holes.

2.2 Tunnel construction efficiency of non-electric detonator and electronic detonator

The tunnel construction efficiency for both electronic and non-electric detonators was assessed by analysing the results of test blasting using identical drilling patterns under similar rock mass conditions. For the blasting results with electronic detonators, test results from the application of a 10 ms delay time to contour holes were adopted.

Figure 4(a) shows the analysis of overbreak/underbreak on tunnel sections when electronic detonators were used. The results indicated that the excavation face was smoother and exhibited fewer instances of overbreak/underbreak compared to when non-

electric detonators were utilised. This study quantified overbreak/underbreaks by measuring the distance between each coordinate point and the designed excavation section including the lookout. Tables 3 and 4 list the average values of over/under-breaks recorded over six respective excavation sections. The average overbreak and underbreak were reduced by 2.7 cm and 5.8 cm, respectively, in comparison to non-electric detonators.

To analyse blasting vibration between the two methodologies, this research calculated the reduction rates of blasting vibrations in the contour holes region using the ground vibration equation (1) proposed by the MOLIT

$$v = 200 \left(\frac{D}{\sqrt{W}} \right)^{-1.6} \tag{1}$$

Where, v is the particle velocity (cm/s), D is the distance between the measuring point and the

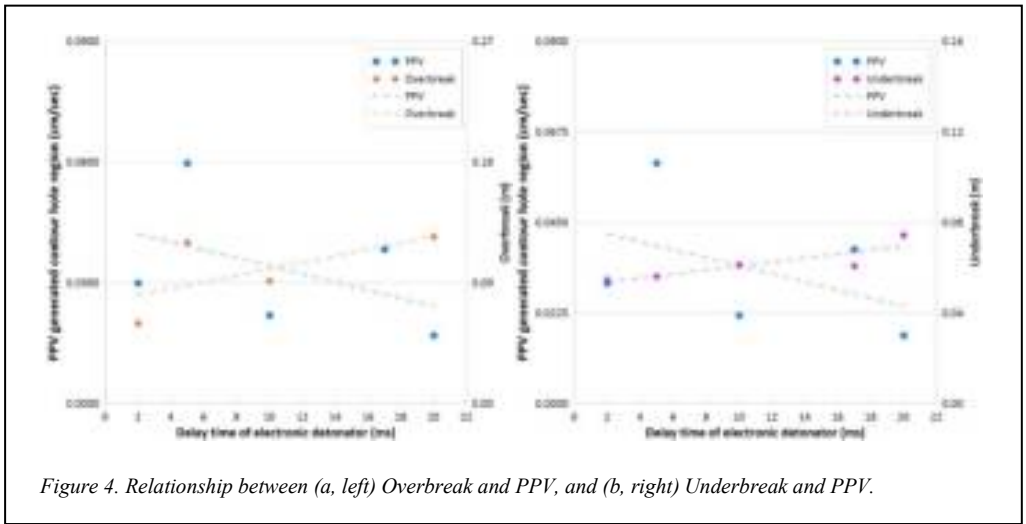
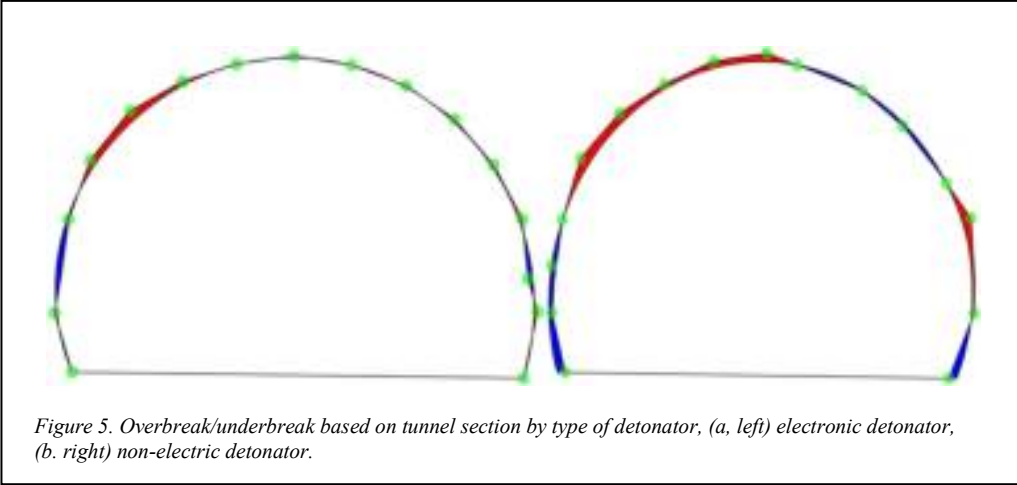


Figure 4. Relationship between (a, left) Overbreak and PPV, and (b, right) Underbreak and PPV.



explosion source (m), and W is the maximum charge weight per delay time (kg).

The vibration reduction rates of the contour holes region indicate that electronic detonators mitigate vibration by 68.94%, compared to 57.75% for non-electric detonators. As a result, the vibration reduction rate in the contour holes region was approximately 11.19% higher for electronic detonators, highlighting their greater effectiveness in terms of vibration control compared to non-electric detonators.

This variance in vibration reduction rates is attributed to the initiation errors inherent in each detonator. For non-electric detonators, the initiation error is 10%, and a broader timing interval correlates to a greater delay time error. In this study, blasting operations using non-electric detonators were configured to initiate five to six blasting holes concurrently, aiming for the creation of smooth blasting holes. However, as discussed, irregular detonations induced by the initiation error of non-electric detonators can lead to

vibration amplification due to constructive interference and irregular fracture planes. Conversely, electronic detonators can address these issues by precisely tuning the delay time, thereby enabling operators to design flexible delay times considering the characteristics of in-situ rock mass.

2.3 Discussion

This study analysed blasting vibrations and overbreak/underbreak in tunnel contour holes to compare the construction efficiency of electronic detonators and non-electric detonators.

A series of blasting tests using electronic detonators were conducted at delay times of 2 ms, 5 ms, 10 ms, 17 ms, and 20 ms to determine the most suitable delay time for contour holes in *in situ* rock mass. The findings indicated that longer delay times resulted in reduced blasting vibrations but led to a gradual increase in overbreak/underbreak. Regression analysis of the blasting vibration and

Table 3. Results of tunnel blasting using electronic detonator.

Test no.	Over break (m)	Under break (m)	Distance (m)	Charge per delay time (kg/delay)	PPV (cm/sec)	V (cm/sec)	Reduction Rate (%)
1	0.069	0.056	157	1.1	0.020	0.066	69.90
2	0.061	0.085	155	1.1	0.021	0.067	68.82
3	0.100	0.086	154	1.1	0.021	0.068	69.23
4	0.136	0.059	153	1.1	0.022	0.069	68.18
5	0.081	0.050	152	1.1	0.023	0.070	67.15
6	0.101	0.032	150	1.1	0.021	0.071	70.39
Ave.	0.091	0.061					68.94

Table 4. Results of tunnel blasting using non-electric detonator.

Test no.	Over break (m)	Under break (m)	Distance (m)	Charge per delay time (kg/delay)	PPV (cm/sec)	V (cm/sec)	Reduction Rate (%)
1	0.090	0.075	224	6	0.052	0.145	64.24
2	0.214	0.047	222	6	0.087	0.147	40.96
3	0.124	0.145	230	6	0.071	0.139	48.99
4	0.104	0.157	228	6	0.067	0.141	52.57
5	0.098	0.101	226	6	0.049	0.143	65.83
6	0.078	0.123	224	6	0.038	0.146	73.89
Ave.	0.118	0.108					57.75

overbreak/underbreak data indicated that 10 ms was the optimal delay time for the contour hole region.

The analysis of construction efficiency in the tunnel contour hole area with electronic and non-electric detonators revealed that when electronic detonators were used, overbreak decreased by an average of 2.7 cm, and underbreak decreased by an average of 5.8 cm compared to the results obtained with non-electric detonators. Furthermore, the study analysed the vibration reduction rates of both methods based on the ground vibration equations proposed by the MOLIT. When electronic detonators were used, the reduction in blasting vibrations in the contour hole area was approximately 11.19% greater than that observed with non-electric detonators.

As evident from the results of this study, electronic detonators are effective in suppressing overbreak/underbreak occurrences in the contour holes region and efficiently controlling blasting

vibrations through precise delay time management.

While the primary focus of this research was to compare the construction efficiency of electronic versus non-electric detonators in the contour holes region, notable improvements in tunnel advance rates and mean fragmentation size were also observed in an actual construction site.

Meanwhile, in tunnel construction sites, look-outs are typically designed to be approximately twice the size of the original design to prevent additional work processes such as blasting and excavation in the event of underbreak occurrences in the underbreak area. As analysed in this study, using electronic detonators with suitable delay times for the in-situ rock can not only reduce overbreak/underbreak occurrences but also create a smoother excavation face.

Moreover, since the designed Lookout can be accurately implemented in the field, it is expected that the amount of shotcrete reinforcement and



Figure 6. GTX project location and work site.

30m Blasting		
	Normal	Double Deck
Radius	15m	15m
Depth	1.0m	2.0m
Total Blasting	30 blast	15 blast
Total Period	90 days	60 days
Conclusion	1) Saving 15 times of Blasting 2) Saving 30 days of working	

❖ One Deck Blasting → Double Deck Blasting

- Double Deck: with HEBS II - 1st HEBS II, 2nd Non-EI Detonator
- GTX A Shaft Size (Diameter: 32 m, Depth 2.8 m, Total Depth: 16.8 m)
- : Total 14 Days of Working day has been reduced**

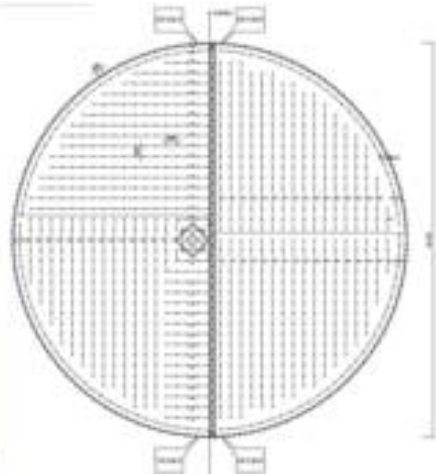


Figure 7. Project parameters and results.

working time affected by overbreak can be reduced.

3 DECKING MODE IN SHAFT BLAST

Blasting in urban environments often present difficult challenges in terms of keeping on schedule, vibration limitations and cost overruns. The use of programmable electronic detonators allows for the possibility to make substantial gains in the project schedule and also in the area in cost-savings.

The following project took place in Seoul (South Korea) for shaft excavation as a part of the GTX (Great Train eXpress) Project Figure 6.

In the first phase of the shaft excavation, material was blasted to a depth of 1m using a non-

electric initiation system, to respect the strict explosives charge limit at the surface. After the inclusion of an electronic initiation system, the excavation productivity was able to improve by a factor of 2, as 2 m sections could now be blasted.

The blast parameters are presented in Figure 7. The bottom deck was primed with an electronic detonator and the top deck was primed by non-electric detonator.

4 DECKING MODE IN TUNNEL BLASTING APPLICATION

For confidentiality reasons, the location of this project is not indicated in this paper.

This case study concerns tunnel blasting under a sensitive and heavily urbanised area where the

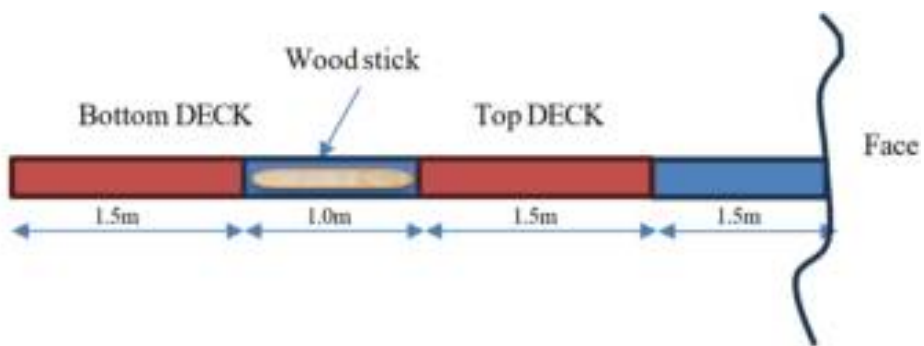


Figure 8. Hole charging.



Figure 9. Delay time applied in the first decked blast.

vibration limitations are very low, in addition to both the project budget and schedule being very tight. The objective of this work was to keep the same advance rate (5.5m per blast) and reduce the charge weight per hole to maintain a low vibration without extending the project schedule.

To achieve this objective, we decided to prove first that decked holes can be successfully implemented in a tunnel blast (proof of concept). Then the next step was to fine-tune the quantity of explosives per delay/deck and then determine the optimal timing design to keep the vibration under the requested limit.

For the first objective, we performed a blast outside of the sensitive area to measure the advance rate and evaluate the use of the decking mode. Two ways were proposed to perform this blast:

- 1st Proposal: keep the same procedure as

often practiced in surface blasting when decking is involved; Top Deck fires first and then the Bottom deck fires with an 8 ms interval. This means that both top Deck and Bottom Deck would fire before an adjacent hole would fire. The contour holes were blasted with one charge (no decks).

- 2nd Proposal: all Top Decks would be fired first and then all Bottom Decks would be fired afterwards, dividing the blast into two sections, fired within the same blast event.

To control the loading charge, we decided to use a cartridge explosive in Top Deck (1.5 m length), then separated the charges with a wooden stick of 1 m length and finally in the Bottom Deck, 1.5 m bulk emulsion explosive was used. See Figure 8 for a diagram of the charging concept.

The purpose of using the bulk emulsion



Figure 10. Blast result using the 1st Proposal (left), and wood stick separating the two charges (right).

explosive in Deck 2 is to maintain the fast-loading cycle.

Currently, and for the first trial, the client has decided to use the 1st proposal, whereby the decks are fired 8 ms apart before firing an adjacent hole. The blast was timed as shown below in Figure 9.

After the blast the advance rate was between 5.5 and 4.5 m (Figure 10). The 4.5 m advance was measured in some place close to the perimeter. The result was somewhat predictable as the 1st proposal does not provide much space for rock movement and burden relief.

5 CONCLUSION AND FUTURE WORK

Despite the often-mentioned increased price of the electronic initiation system compared with the non-electric initiation system, it has become very clear that the added value brought through the use of electronic initiation over the entire project can easily compensate for the increased unit-cost.

In the future, we plan to continue research and testing using the firing method explained in the 2nd Proposal, with the aim of successfully advancing 5.5 m across the entire blasting face and afterwards selecting the optimal millisecond delay to keep the vibration under the limit within the sensitive area.

The final goal is to continue developing new construction blasting methods using electronic detonators that enhance tunnel construction efficiency while minimising potential civil complaints.

6 ACKNOWLEDGEMENTS

This section is intended to acknowledge all of my colleagues in Seoul that assisted with the paper and the entities that agreed to conduct these studies.

REFERENCES

- Cai M. & Kaiser, P.K. 2005. Tunn. Undergr. *Space Technol.*, 20, 301-310 (2005).
- Fu, H., Shen, Z., Zhao, Y. & Wang, S. 2012. *Chin. J. Rock Mech. Rock Eng.*, 31, 5, 597-603 (2012).
- Fu, H., Wong, L.N.Y., Shen, Y., Shen, Z., Zhang, C. & Li, Y. 2014. *Rock Mech. Rock Eng.*, 47, 809-816 (2014).
- Hanwha 2025. Hanwha Corporation, 'Hanwha Electronic Blasting System',

<https://www.hanwhablastservices.com/ko/%EB%A9%94%EC%9D%B8/%EC%A0%9C%ED%92%88/electronic-blasting-system> (accessed:18-August 2025). (online).

- Jung, S.W., Song, J.H., Whang, N.S., Kim, H.G., Kim, N.S. & Lee, J.W. 2023 *Journal of Korean Tunnelling and Underground Space Association*, 25, 2, 157-173 (2023).

MOLIT 2006. Design and construction criteria of surface mine blasting for road works. *Ministry of Land, Infrastructure and Transport*, 2006.

- Sato, T., Kikuchi, T. & Sugihara, K. 2000. *Eng. Geol.*, 56, 97-108 (2000).

RETURN TO CONTENTS PAGE

3. Demolition blasting

Numerical simulations for enhanced precision and safety in structural blasting

A. Gerat & C. Rapps

Woelfel Engineering GmbH + Co. KG, Hoechberg, Germany

ABSTRACT: The growing number of building demolitions in densely built-up areas, coupled with complex load-bearing structures and the emergence of new construction materials demand greater precision in demolition planning and execution. Accurate predictions of impact areas, prevention of uncontrolled structural failures, and reduction of deviations in fall direction or mitigation of unstable equilibrium states are critical. These factors underscore the necessity of advanced, modern methodologies in demolition planning to ensure safety, efficiency, and reliability. Evolving from simplified multi-body systems and leveraging increasing computing capacities, numerical methods such as the finite element method are now being established in the demolition industry. This paper presents the application of numerical methods in blast demolition planning, validated through diverse case studies of advanced blasting strategies. The cases are further analysed using numerical simulations to provide deeper insights into their collapse behaviour and to highlight the advantages of simulation-aided planning.

1 BACKGROUND

Starting from a primarily experience-based origin of blast demolition planning of structures, factors such as limited space in densely built-up areas, complex load-bearing structures and the emergence of new construction materials or blasting strategies lead to the requirement of higher accuracy and precision in planning and execution.

First efforts to achieve higher precision in the planning stage led to the development of simplified multi-body systems which simulate the dynamic behavior for rigid-body models of structures. The continuous development of computing capacities allowed for further evolution of simulation models.

Modern simulation techniques, such as the finite element method, enable the analysis of global and local deformation of structures by

solving the equation of motion while ensuring the conservation of mass, momentum and energy. Enhanced by features such as nonlinear geometry, nonlinear material models and the use of advanced contact algorithms, the complex process of structural collapse can be simulated with high accuracy.

The advantages of simulations with a digital twin of the structure to be demolished are discussed below using several practical examples.

2 TILTING AND FOLDING OF CHIMNEYS

The blasting of chimneys requires more precise analysis methods and a way of predicting unwanted failure scenarios as the height of the structure increases. The typical tilting as a blasting strategy for chimneys reaches its limits due to the failure mechanism of “slump” where the hinge zones at the base of the chimney collapse due to

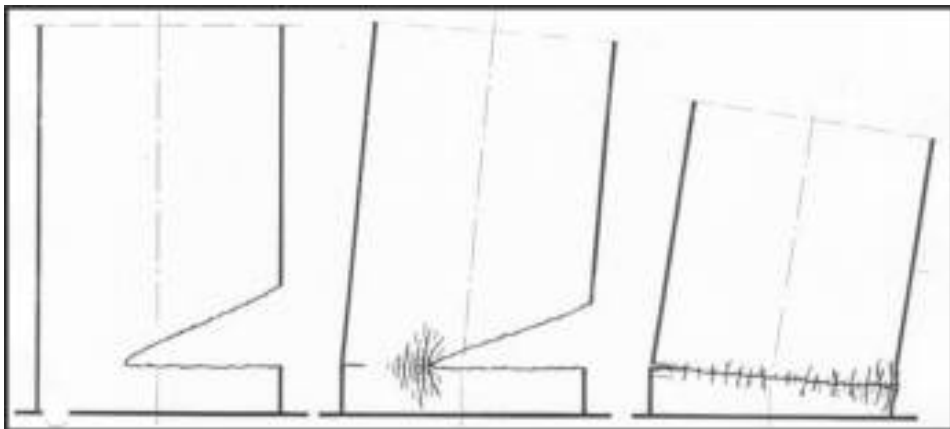


Figure 1. Schematic diagram of the failure mechanism 'slump'.

overload. Reasons for this failure mechanism can be inadequate blasting zones, inadequate material strength or insufficient cross-sectional area with consequences from deviations in fall direction to the stopping of the tilting process in an unstable equilibrium, see also Figure 1.

Because of additional effects like the dynamic increase of inertia forces or possible buckling of the weakened chimney shaft, it is not always possible to rule out this failure mechanism by hand calculations. This is why numerical simulations utilising nonlinear material models with

constitutive equations for damage and plasticity, nonlinear geometry for finite translation, rotation and strains and contact algorithms which allow for the consideration of friction or impact between structural parts, have established themselves in the planning of controlled demolition. The ability to predict critical failure mechanisms leads to the possibility of adjustments of the initial planning. If the adjustments do not lead to a satisfactory solution, numerical models allow for the analysis of advanced blasting strategies such as tilting on steel joints or blast folding.

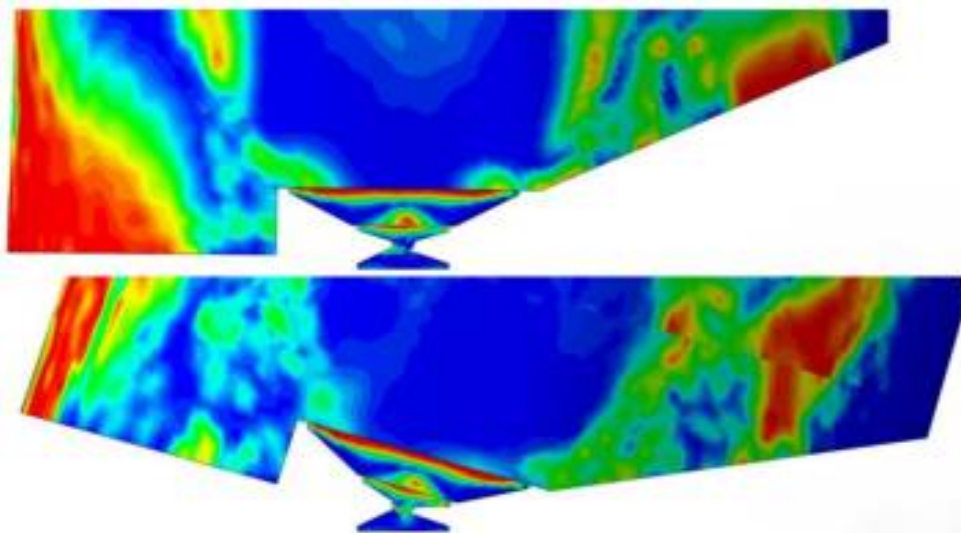


Figure 2. Analysis of tension (red) and compression (blue) zones during tilting on steel joints.



Figure 3. Simulation and real blasting of chimney blast folding in Ens Dorf (left) and Copenhagen (right).

The tilting on steel joints utilises massive steel joints which support the whole weight of the chimney shaft and guide it like hinges with minimal deviation in fall direction.

In cases where the tilting on steel joints is not feasible or if the surrounding area does not allow for tilting with the whole chimney height, blast folding provides another blasting strategy. By adding one or more blasting zones along the height of the structure, it is possible to fold the chimney with significant shorter impact length. Depending on the surroundings, the timing of each blasting zone can be varied until an optimal strategy is reached. The successful implementation and validation of simulation methods can be seen in the examples in Figure 3 and numerous successful demolition projects in recent years.

3 PROGNOSIS OF UNWANTED FAILURE MODES

During the blasting of the old north-east section of the Landeskroner Weiher viaduct, an unforeseen collapse mechanism occurred at the highest pier axes. Instead of the anticipated double blast folding, in which both segments were to lie exactly on top of each other, an early break occurred between the two segments, whereupon the upper segment slid along the lower segment and was rotated in the opposite direction, as shown in Figure 4.

Wölfel Engineering was then assigned to determine the reason for the unforeseen collapse behavior and to develop an optimised blasting strategy for the subsequent blasting of the



Figure 4. Overlay plot for the blasting of the northeast viaduct 'Landeskroner Weiher'.



Figure 5. Overlay plot for the blasting of the southwest viaduct 'Landeskroner Weiher'.

southwestern part of the structure. Simulations showed that the high slenderness of the piers and an interaction between superstructure, bearing and pier head led to a rotational restraint of the upper segment and consequently to constraining forces in the kinematic system, resulting in premature fracture between the segments, thus validating the ability to predict this failure mechanism through simulation.

After the first section was demolished, the new north-east section was built in its place. To avoid any risk to the new section during the blasting of the old south-western substructure, further simulations were carried out. Variant

investigations led to the concept of a triple blast folding, which was successfully carried out at the end of March 2025.

The agreement between simulation and reality can be seen in Figure 5.

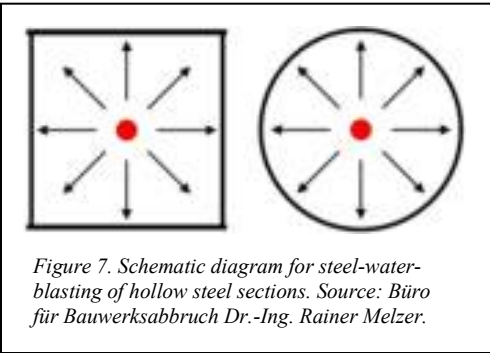
4 BLASTING OF THE DAMAGED RAHMEDE VIADUCT

In December 2021, a bridge inspection of the viaduct 'Rahmede' between the cities Hagen and Siegen using laser scan measurements showed deformation of the superstructure that required an immediate closure of the bridge for all traffic.



Figure 6. Overlay plot of simulation and reality for the viaduct 'Rahmede'.

After initial measures to stabilise the bridge, the analysis of possible variants led to the conclusion that further strengthening was not feasible and that the bridge had to be demolished by means of blasting.



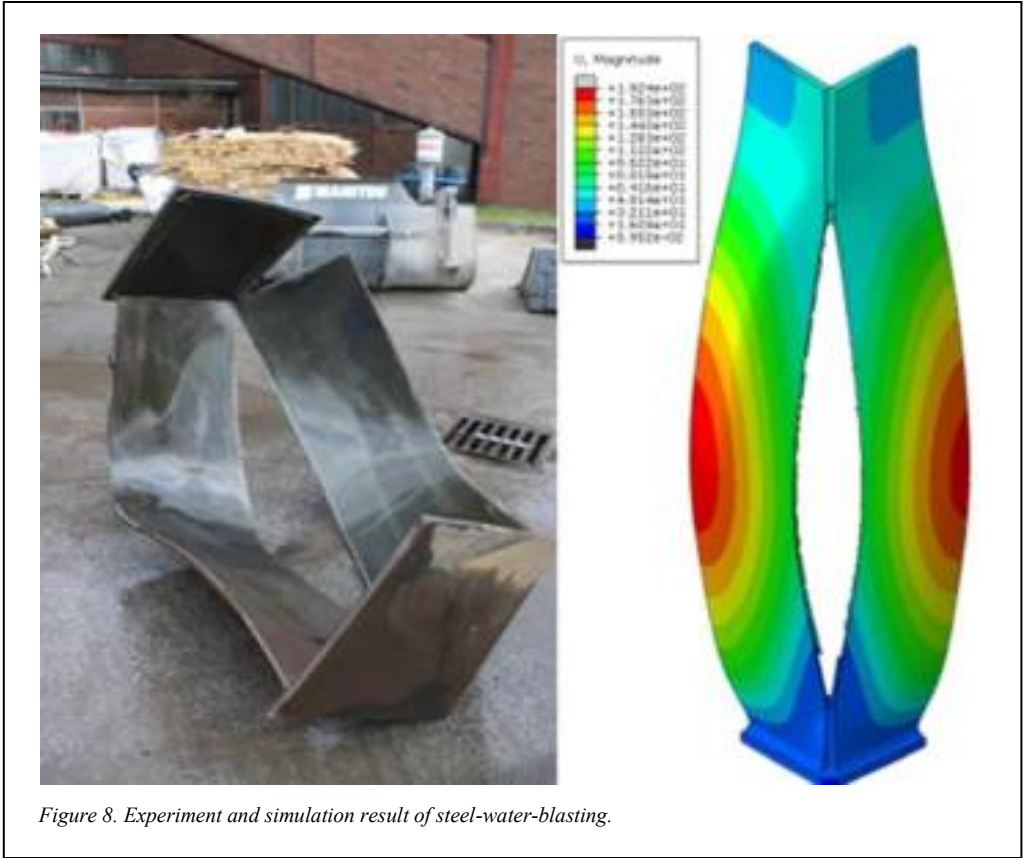
Wölfel Engineering was commissioned with the analysis of possible blasting strategies and the planning of the demolition. Simulations validated a double blast folding of all piers as the optimal blasting strategy. Because of the densely built-up

area with residential buildings, warehouses and an electroplating company directly next to the viaduct and the alpine-like terrain underneath the 70 m high viaduct, further investigations were necessary to predict the impact-area of the superstructure.

The effect of wind perpendicular to the superstructure during the free fall was studied using detailed models of the superstructure and terrain and enabled the precise planning of safety-measures for the adjacent area. The close agreement between the simulation and reality can be seen in Figure 6.

5 LOADING DENSITY DETERMINATION IN STEEL-WATER BLASTING

As an alternative to linear shaped charges or blasting with traditional charges, the method of steel-water blasting is a particularly efficient and economical method for the demolition of steel structures with hollow sections. Originating from the dismantling of hollow cast iron structures, this method utilises the nearly incompressible nature of



water as a transport mechanism for blast pressure. The hollow steel section is filled with water and loaded with detonating cord. After detonation, the blast pressure deforms the steel section from inside leading to high tensile forces and rupture of the section.

While this elegant blasting strategy allows efficient demolition with minimal charge quantities, the estimation of the loading density poses a complex task. If the loading density is too low, the charges cannot overcome the resistance of the steel section leading to a failed demolition attempt with the possibility of the structure remaining in an unstable equilibrium, posing high risks for demolition personnel and adjacent infrastructure. If the loading density is too high, there's a risk of accelerated sheet metal or shrapnel being slung away, thus endangering bystanders and the surroundings. This leads to a narrow spectrum of loading density that ensures the safe demolition of the structure.

This complex engineering problem poses another example for the use of simulation in explosive engineering. After numerous small and large-scale field tests it could be shown that carefully validated hydrocode simulations are able to model the interactions between explosives, water and steel under high pressure and strain-rates and to correctly predict the structural response, in turn enabling a safe charge quantity determination.

Following an extensive phase of accompanying research, the simulation method has already been successfully implemented several times in commercial projects.

6 CONCLUSION

Numerical simulations complement demolition planning and enable the safe planning and further development of blasting strategies and protective measures. Furthermore, the validation of the simulations and the visualisation of the blasting concept creates confidence among clients, public authorities, insurance companies and other stakeholders.

Several examples were used to show how simulations can predict undesirable collapse mechanisms during early stages of the planning process and offer the possibility of examining variants to determine optimal strategies and minimise risks during the following demolition.

The special case of steel-water blasting shows the latest developments in demolition engineering and highlights the strength of numerical analysis eliminating the need for costly in-situ experiments.

REFERENCES

- Melzer, R. & Schwarz G. 1996. Schornsteinsprengung mit Stahlkippgelenk. *Schweitzer Sprengtechnik, Vortrag, Dresdner Sprengschule, Heft 34.*
- Melzer, R. 2004. Vollraumsprengung hohler Bauwerkstützen aus Stahl. *SPRENGINFO, In Mitteilungsblatt des Deutschen Sprengverbandes e. V. 26.*

[RETURN TO CONTENTS PAGE](#)

Precision demolition: executing large-scale multi-structure power plant implosions

A. Konya, R. Gilbert & C. Moles

DetX™, Coweta, Oklahoma, USA

ABSTRACT: Large-scale, multi-structure power plant implosions in urban or infrastructure-adjacent environments represent one of the highest risk forms of high-consequence blasting™. These projects require advanced engineering, real-time adaptability, and precise control of structural sequencing, debris trajectories, and blast effects. This paper outlines the strategic framework necessary for executing such work, covering charge design, structural pre-weakening, environmental mitigation, and community impact minimisation. Using the Avon Lake Power Plant implosion as a case study, which was completed by DetX™ on December 19, 2024, we present the planning methodology, field innovations, and execution controls that enabled the simultaneous felling of boilers, stacks, and hybrid structures under tight spatial and political constraints. The goal is to establish best practices and critical considerations for demolition professionals managing high-risk, multi-structure events where failure is not an option.

1 INTRODUCTION TO PRECISION DEMOLITION

The explosive demolition of industrial structures has evolved from a tactical operation into a highly engineered discipline. Nowhere is this more apparent than in the emerging field of high-consequence blasting - projects where both the likelihood of risk and the magnitude of consequence are extreme. These include demolitions near populated areas, critical infrastructure, active utilities, or environmentally sensitive zones. In these cases, failure is not measured in dust or delay, but in death, litigation, or the catastrophic collapse of the wrong structure.

Power plants sit at the center of this shift. Originally constructed in remote industrial corridors, they're now surrounded by dense residential development, schools, and protected infrastructure. Their physical scale and construction as multi-level steel structures, hybrid

concrete assemblies, massive exhaust stacks further complicate the demolition equation. Mechanical demolition has become increasingly unviable: timelines stretch into years, crews are forced into structurally compromised buildings, and injuries continue to mount. Worse yet, as structural steel grows stronger and structures taller, traditional mechanical and manual methods struggle to keep pace.

Communities and developers are demanding a safer, faster, and cleaner outcome. Explosive demolition, when properly executed, offers all three. But this is not the same explosive demolition practiced decades ago. What's now required is a new methodology: Precision Demolition.

Precision Demolition is an engineering-first philosophy that applies rigorous design, sequencing, and modeling to the act of structural takedown. It is not about 'cut and kick' tactics or pre-weakening columns until a final nudge topples

a building. In fact, that approach is now dangerous and outdated. In Precision Demolition, structures are pre-cut - not pre-weakened - to preserve stability until the moment of initiation. Explosives are used not just for initiation but to carry out the actual cutting and collapse of steel elements. Every charge is placed with purpose and every sequence is calculated.

What differentiates Precision Demolition from traditional blasting is not the use of explosives, but the shift in objective. Traditional blasting was about achieving a result; in most cases this was getting the structure down. Blast engineering is about achieving that result while also delivering the right debris profile, maintaining environmental compliance, limiting vibration, protecting the fall zone, and preserving adjacent structures. This requires engineering principles and not just field knowledge.

This is especially critical in multi-structure engineered implosions, where buildings are not independent fall bodies. Instead, one structure must collapse into another, clearing space for a third structure to follow, in less than a few seconds. In Avon Lake, for example, a stack was centered between a boiler and a precipitator. The blast sequence had to prevent either from impacting the stack while also ensuring the stack fell before the second boiler dropped into the cleared space. Any deviation, by seconds or fall angles, would have caused structural interference and unpredictable collapse behavior.

In this paper, we present the Avon Lake Power Plant demolition as a case study in Precision Demolition. This was not just an implosion, it was a high-consequence, multi-structure event conducted in close proximity to public roadways, an active waterway, and residential neighborhoods. We explore the methodology, risks, charge design, and execution philosophy required to deliver a successful outcome. More importantly, we present a new framework for high-consequence demolition in the modern era where public tolerance for error is zero, and the engineering behind the blast must meet the complexity of the structures we bring down.

2 AVON LAKE POWER PLANT PROJECT OVERVIEW

The Avon Lake Power Plant was located along the southern shoreline of Lake Erie in Avon Lake, Ohio. Constructed in 1925 and expanded multiple times through the 20th century, the site ultimately consisted of over a dozen major industrial structures. At its peak, the facility represented one

of the most significant coal-fired power stations in the United States and was engineered using construction methods that spanned nearly five decades of industrial evolution. By the time it was decommissioned in 2021, the plant had outlived its economic and environmental viability and was targeted for full-site demolition to clear the area for redevelopment.

The goal of the explosive demolition was to safely and efficiently bring down the remaining large structures specifically two units which included four boilers, the precipitator, and two stacks (Stack 8 and 9) in a controlled sequence that minimised risk to the public, limited disruption to the surrounding community, and preserved critical surrounding infrastructure. The project was also tasked with achieving this in the shortest possible timeline, avoiding the long delays and manpower hazards associated with traditional mechanical demolition and the height of the structures. Due to community concerns, the number of blast events was further reduced from four to two, requiring a consolidation of multiple high-risk structures into one unified blast sequence. The first blast was a test blast, which was fired on July 31, 2024 which included boiler 9 and 10 of Unit 6-7. The second blast was the multi-structure blast of Boiler 11 and 12, the precipitator for unit 8, and two stacks (stack 8 and 9).

The challenges presented by this site were extensive. Structurally, each building was unique in age, design, and material composition as they were built across multiple time periods (see figure 2 for construction sequencing). The steel used in the precipitator differed in thickness and metallurgy from that used in the boilers, including i-beams with 6" flanges which could not be scarfed, resulting in flanges too large for traditionally available shape charges. The stacks were constructed of reinforced concrete, but one of them had been previously altered and featured a weakened back wall due to a large vehicular opening. Compounding the engineering difficulty, many of the plant's original drawings were missing or incomplete, requiring field verification, X-ray fluorescence analysis, and direct structural probing to determine failure points and load paths, including taking coupons of steel and testing them with linear shaped charges for penetration depth. These were not isolated buildings sitting in a field; they were interconnected structures, built into one another over time, with shared platforms, utility corridors, and adjacent fall paths.

The challenges presented by this site were extensive. Structurally, each building was unique



Figure 1. Avon Lake power plant prior to demolition.

in age, design, and material composition as they were built across multiple time periods (see figure 2 for construction sequencing). The steel used in the precipitator differed in thickness and metallurgy from that used in the boilers, including i-beams with 6" flanges which could not be scarfed, resulting in flanges too large for traditionally available shape charges. The stacks were constructed of reinforced concrete, but one of them had been previously altered and featured a weakened back wall due to a large vehicular opening. Compounding the engineering difficulty, many of the plant's original drawings were missing or incomplete, requiring field verification, X-ray fluorescence analysis, and direct structural probing to determine failure points and load paths, including taking coupons of steel and testing them with linear shaped charges for penetration depth. These were not isolated buildings sitting in a field; they were interconnected structures, built into one another over time, with shared platforms, utility corridors, and adjacent fall paths.

The risks on site were severe. To the north was Lake Erie and an active water intake system still serving the region. To the east, a major public roadway sat within dozens of feet of the drop zone. Residential neighborhoods surrounded the site in multiple directions, and several structures remained in place for telecommunication use,

including the original 1925 stack which had become a cell broadcast tower with active infrastructure installed prior to demolition. Any deviation in fall path, debris projection, or overpressure effect could result in major property damage, utility disruption, or loss of life. Structural collapse had to be contained within a narrow footprint that left no room for error. Vibration, dust, and blast noise were tightly regulated, and failure to comply with environmental and permitting thresholds could halt the project.

This made Avon Lake a clear example; not because of any one factor, but because every aspect of the project carried both elevated risk and unacceptable consequence. The proximity of civilians, the scale of the structures, the presence of infrastructure, and the nature of the demolition all combined into a single operation that demanded a zero-failure tolerance.

The requirement to have one main blast increased the difficulty, making this multi-structure which required structures such as the unit 9 boilers to drop into the area of the unit 8 boilers in one single event. This was not an implosion of a single building with a defined fall zone. Each structure was tied to others either physically or through fall sequence. The boiler and precipitator had to drop away from one another



Figure 2. Construction sequencing for Avon Lake.

while avoiding collision with the center-placed stack. That same stack had to fall before the second boiler dropped into the same space it occupied. The entire sequence required precision timing not just for individual structures but across interdependent collapses, where failure of one phase could disrupt every other.

Traditional demolition approaches, whether mechanical or old-model blasting, were incompatible with the complexity of the task. Pre-weakening the structure to the point of instability and ‘kicking the legs out’ would have resulted in uncontrolled collapse or premature failure. Likewise, treating each building as an isolated blast target would have demanded multiple blasts, increased cost, and prolonged community disturbance which was unacceptable to the local community council that refused to issue multiple blast permits.

This is where Precision Demolition becomes essential. The Avon Lake operation required an engineered approach at every level; from charge formulation to sequencing, from structural analysis to debris flow design. The structures were not weakened but pre-cut to maintain integrity until detonation. Explosives did not assist collapse, they executed it. Fall paths were modelled, not assumed. Vibration limits were met through analytical planning, not hope. And most critically, the entire program was designed not just to bring down a structure, but to bring down the right structures, in the right order, with controlled results, in a constrained environment, in a single

blast event. That is the definition of Precision Demolition.

The successful execution of a high-consequence, multi-structure blast of this scale required close coordination between specialised teams, each bringing discipline-specific expertise to the project. Trifecta Services Company served as the general demolition contractor, overseeing all mechanical cutting, site preparation, barricading, and environmental control measures. Trifecta also led all community engagement efforts, working closely with local officials and residents to navigate concerns and secure the approvals necessary to move forward with an explosive solution.

DetX acted as the blasting contractor responsible for all blast engineering, charge design, placement, and sequencing. The DetX team led the structural analysis and timing logic, developed the precision demolition plan, and executed the final blast event. Supporting this effort were Alford Technologies and Advanced Explosive Systems (AES), who supplied the custom-shaped charges and linear cutters required to sever high-strength steel members. Saul’s Seismic provided the vibration analysis and real-time monitoring needed to ensure the blast met regulatory compliance and community protection thresholds. Together, this team executed one of the most complex and tightly engineered power plant implosions in recent history, setting a new standard for urban-scale explosive demolition.



Figure 3. Avon Lake structure fall directions.

3 PRECISION DEMOLITION DESIGN AND RISK ASSESSMENT

Precision Demolition is a field methodology built on engineering principles. It involves the deliberate design of charge placement, timing, structural interaction, and debris control to achieve a controlled and predictable collapse. Unlike traditional approaches that rely heavily on pre-weakening and generalised charge layouts, Precision Demolition requires the structure to remain stable until the moment of detonation - placing full responsibility for failure sequencing and collapse behavior on the blast design itself.

At the core of this methodology is an understanding of risk and consequence. Risk is the probability of failure. Consequence is the severity of the outcome if failure occurs. High-consequence blasting involves scenarios where both are elevated; dense residential surroundings, critical infrastructure within the fall zone, limited exclusion zones, or structures interconnected in a way that one misstep could compromise the entire operation. In these environments, reducing risk is critical and a one-size-fits-all or traditional blasting approach cannot

be tolerated. The collapse has to be controlled, debris contained, and every stage of sequencing verified before initiation.

The design process begins with structural profiling which includes understanding each building's materials, support systems, fall direction options, and how it interacts with adjacent structures. Fall zone engineering is critical. Each trajectory must be modelled with the assumption that no structure is isolated, especially in multi-structure projects. The sequencing logic must control the collapse order and prevent mid-fall collisions, cross-structure interference, or rebound into protected zones.

At Avon Lake, this was especially critical. This approach started with a high-level design analysing each structure and the required fall direction and time to fall to ensure that it moved in the right manner to clear area for a subsequent structure. This is shown in Figure 3 with pre-planned fall directions for each structure to mitigate risk of the nearby infrastructure. The stack, precipitator, and boiler were physically interwoven. The stack sat between the boiler and the precipitator, with shared platforms and bracing across multiple spans. In that layout, if the boiler

or precipitator had started to fall before the stack initiated, they would have cut the stack off at mid-height, causing a lateral failure and likely scattering reinforced debris into the surrounding area, including the nearby waterway. Conversely, if the stack fell too early and intersected with either structure mid-collapse, it could have caused unplanned pile-up, structural ricochet, or redirected debris toward the public roadway. The entire sequence had to be modelled to milliseconds, not just for ignition timing, but for actual fall rate and progression.

Each cut had to be designed with an understanding of not only what would fail, but when and how it would fail. Structural stability was maintained through pre-cutting, not pre-weakening. This allowed crews to execute safe prep work while ensuring the structure remained intact until the exact moment of detonation. DetX designed the blast to handle all major structural severing, especially in critical vertical and lateral supports, so that the integrity of the structure was compromised only during the detonation itself. This eliminated the risk of unplanned collapse during prep and placed all structural transition under controlled sequencing.

Every decision was made with consequence in mind. Fall direction, sequence timing, stack lean rate, boiler drop zone, and even the footprint of the final debris pile were designed to meet both safety and community requirements. These outcomes cannot be achieved through traditional methods. Precision Demolition allows the blast engineer to work backwards from the end state - the exact location, orientation, and segmentation of the debris and then design the initiation logic, charge layout, and support cuts to achieve that result. It is not about bringing the building down. It is about bringing it down the right way, on schedule, under control, and with no room for error.

Risk assessment in Precision Demolition requires a deliberate evaluation of failure modes, site constraints, and third-party exposure. Unlike traditional demolition, where hazards are often mitigated by distance, slow progression, or redundancy in methods, high-consequence blasting involves little margin for error and compressed timelines. The goal is not simply to understand what might go wrong, but to quantify, prioritise, and engineer around those risks from the outset. This includes evaluating structural condition, site layout, environmental factors, and community impact. Each factor must be considered within the context of a single, controlled blast event.

Below are the top 10 risk categories that must be addressed in the design and execution of any precision demolition project:

- **Personnel Exposure During Prep:** Structural pre-cutting or barricading work conducted under or inside the structure presents life-threatening risk if sequencing or support assumptions fail. All measures must be taken to protect employees during critical cutting and cuts should be minimised and properly designed to maintain structural integrity.
- **Community and Political Safety:** Failure to control dust, sound, or timeline can create backlash from municipalities or local residents, especially in environments with tight public scrutiny or political oversight. However, the greatest risk comes from nearby community members and infrastructure. Blasts must be designed to protect community infrastructure and mitigate risk to damage. Furthermore, as persons want to see blasts, measures must be taken to secure the blast site – including thermal drones when blasting in the dark – and additional site security.
- **Structural Misfire or Collapse Interference:** Failure of one structure to collapse as modelled; whether early, late, or incomplete; can interfere with the fall path of others, potentially compromising the entire blast sequence. This can result in damage to nearby structures, explosive ordnance being left undetonated following the blast, hung structures, or weakened and cut structures still standing, requiring personnel to mitigate in extreme risk conditions.
- **Stack Base Failure or Rotation:** Minor deviation at the stack base can cause significant angular drift at height. Poor alignment or uneven support can lead to rotation, causing collapse outside the footprint. This becomes especially critical when stacks have opening or weaknesses including cold joints and different concrete conditions from multi-stage pours that can cause the stack to shift and fall into unplanned locations.
- **Uncontrolled Debris Projection:** Steel fragmentation or concrete breakout exceeding predicted fall zones can damage adjacent infrastructure or threaten exclusion zones, especially when charge types induce fragmentation. Debris Projection from shape charges or structural steel can also cut other

charges and create cutoffs or misfires from impacting nearby charges. Timing must be considered to mitigate these issues.

- **Unverified Structural Variability:** Changes in steel composition, thickness, or retrofits (especially undocumented) can alter how structures respond to charge effects or sequencing delays. Steel and concrete must be tested to ensure that they are uniform, or that the variability can be properly dealt with and accounted for in the charge design.
- **Residual Structural Stability Post-Blast:** Partial collapse or retention of unsupported segments creates significant safety hazards for re-entry and debris clearing operations. Charging and timing must be designed through the entire structure, not just kicking out the front rows, to ensure that the buildings properly fall apart and eliminate structural stability.
- **Environmental and Waterway Contamination:** Nearby waterways or protected zones can be impacted by airborne dust, debris migration, or vibration-induced sediment shifts, triggering environmental enforcement actions.
- **Vibration and Air Overpressure Exceedance:** Exceeding allowable limits can cause damage to utilities, buildings, or trigger legal liability. In tightly regulated zones, even marginal overpressure can shut down future work.
- **Equipment or Charge Failure:** Failure of detonators, cap-scatter in detonators, poor continuity, or improperly constructed charges can result in underperformance or asymmetrical collapse behaviour.

4 CHARGE DESIGN AND TIMING

Precision Demolition requires a fundamentally different approach to charge design than traditional explosive work. Charges are not simply selected to 'initiate failure' or provide a directional nudge, they are engineered to perform complete and reliable structural cutting. This means the charges must be overdesigned relative to the material's strength; not to compensate for poor placement or design, but to ensure complete severance under varying field conditions. Precision Demolition does not tolerate marginal cutting. Every charge must result in a clean, full-structure failure at the point of initiation.

Achieving this requires a detailed understanding of the materials being cut.

Structural steel, especially in legacy power plants, varies in grade, flange thickness, and yield strength. In some cases, undocumented steel retrofits introduce modern alloys into legacy framing, and concrete piers can contain hybrid reinforcement systems. For this reason, all critical structural elements must be tested in the field, both through non-destructive methods like X-ray fluorescence (XRF) for steel chemistry, and through concrete coring or coupon cutting for substructure behavior. Only once the material properties are verified can proper charge selection and configuration be completed.

Beyond charge type, the sequencing of detonation is equally critical. Structural collapse is a dynamic process, not a single moment in time. Each element must fail in a controlled sequence that not only causes collapse, but moves the structure in the required direction. This includes accounting for differential fall rates, secondary structure interaction, and recoil behavior in large components such as stacks and precipitators. Sequencing must be designed to eliminate or reduce the risk of rotation, bounce, or rebound. To achieve this, electronic detonators are used exclusively in Precision Demolition. These eliminate cap scatter and allow for millisecond-accurate timing between charge groups, reducing the potential for misfire and giving full control over the fall order of multi-structure systems.

At Avon Lake, this philosophy was applied in full. Field tests were performed on representative steel members and coupons taken from key structural zones. XRF analysis revealed that several of the support columns were composed of A572 Grade 50 steel with 6-inch flanges, beyond the effective capacity of standard linear charges. To overcome this, the team deployed Dioplex DX-5000 shaped charges provided by Alford Technologies, which were hand-packed onsite with C-4, by the DetX team, at controlled density and standoff. These charges were mounted in custom back-to-back configurations to ensure full penetration and severance of the vertical members.

Electronic detonators were used across all charge groups, enabling us to apply proprietary timing models refined for multi-structure implosions. This sequencing was essential in controlling the relationship between the boilers, precipitator, and centrally located stack. The stack had to initiate cleanly and begin descent before the adjacent structures could interfere. Sequencing was designed to allow the structure to fall into pre-cleared zones without interruption. The use of electronics eliminated uncertainty in delay timing

and provided the precision required to maintain trajectory control across a tightly confined site.

The end result was a blast sequence that did not just bring the structures down, it directed them, controlled them, and timed them to the exact parameters required by the site's constraints. Without engineered charge design and exact sequencing, the operation would have carried an unacceptable risk profile. With them, the collapse performed as modelled.

Additionally, quality assurance must be non-negotiable in Precision Demolition. With risk and consequence tightly coupled, any deviation in charge performance or timing can result in structural underperformance, uncontrolled collapse, or failure to initiate key elements. Every charge must function exactly as designed - on time, at full effect, and in the correct sequence.

To ensure this, all charges undergo a multi-layered QA process. Each charge is checked at installation, visually inspected, and independently verified by separate team members before final tie-in. Every initiation path, whether surface, trunkline, or electronic is tested for continuity and logged. No charge is tied in without double confirmation from two qualified personnel. Redundancy is built into all firing circuits. Dual initiation lines are used on critical charge groups to prevent single-point failure. Final system checks are performed with a full run-through of each detonator address, logging confirmation of proper programming, delay intervals, and firing signal integrity. This level of QA is not optional; it is what makes precision possible. The execution may last seconds, but it is the hours of inspection, verification, and cross-checking that ensure the structure behaves as modelled.

5 INNOVATIONS ENABLING SUCCESS

Several key innovations were deployed at Avon Lake to overcome both structural and environmental constraints. These were not auxiliary improvements, they were essential tools that allowed the project to succeed under high-consequence conditions and within the strict parameters imposed by the site. Each was selected based on a direct operational need and engineered to address specific risks that could not be solved using standard demolition practices:

- **Hand-Packed DX-5000 Shaped Charges** - The steel members at Avon Lake exceeded the cutting capability of commercially available U.S.-based charges. Support columns with 6-inch A572 Grade 50 flanges

required full severance to initiate structural collapse. The solution was to deploy Dioplex DX-5000 charges from Alford Technologies, which were hand-packed on site using C-4. These charges were configured in custom dual-sided applications to ensure penetration across the full flange width. Their deployment allowed for precision cutting of high-strength steel under tight standoff and constrained positioning.

- **Utilisation of Site Infrastructure as Blast Barricades** - Rather than importing external barricading materials, we repurposed structural elements from the plant itself, reinforced concrete walls, plate steel, and heavy foundation blocks, to serve as fragmentation shields and overpressure barriers. This reduced logistical complexity while providing proven mass and durability. Each element was modelled into the blast plan as part of a system-level containment design.
- **Engineered Mud Mats for Stack Impact Zones** - To control debris scatter and mitigate back-blast from the collapse of the two concrete stacks, engineered mud mats were constructed using layered fill, compacted earth, and geotextile fabrics. These mats absorbed impact energy, minimised plume generation, and confined debris within a modelled footprint. Their placement allowed for reduced fallout zones without compromising safety margins.
- **Thermal Drone Surveillance for Site Security** - Given the early morning blast time, which was selected to detonate the blast in the dark mitigating community disturbance, and urban proximity, the exclusion zone required full perimeter control in low-light conditions. Thermal drones were flown during the pre-blast window to identify and confirm heat signatures, ensuring that no personnel or bystanders breached the site boundary and maintained flight and active communications throughout the blast. Post-blast, drones were immediately redeployed to assess the debris field for any anomalies or residual thermal activity. This provided rapid feedback without the need for personnel to re-enter unstable areas.

Each of these innovations was applied based on necessity, not convenience, along with the other standards of Precision Demolition. They addressed hard problems: cutting steel that could not be conventionally severed, controlling debris in a site

boxed in by public infrastructure, and ensuring no margin for error in perimeter security. Collectively, they reinforced the core premise of Precision Demolition, that success in high-consequence blasting comes from engineering the execution, not reacting to the outcome.

6 PROJECT OUTCOMES AND LESSONS LEARNED

The Avon Lake Power Plant implosion was executed on December 19, 2024, at 5:30 AM. Within seconds, two boilers, a precipitator, and both concrete stacks collapsed in the planned sequence, within the modelled fall zones, and without deviation. All structures cleared their designed collapse paths, and no structural interference, lateral projection, or uncontrolled rebound was observed. The debris field matched predictions, enabling immediate mobilisation of post-blast operations without the need for emergency intervention or rework. No injuries occurred. No misfires were recorded.

From a performance standpoint, the project exceeded expectations. A demolition sequence originally scoped for four to five separate blasts was reduced to two, an initial test event and the final unified implosion. This reduced public disruption, simplified permitting, and significantly shortened the project timeline. All structural components fell within controlled zones, clearing the necessary footprint for redevelopment activities to proceed on schedule. Seismic and air overpressure readings remained within acceptable tolerances, and no damage to surrounding infrastructure was reported. More than 83,000 tons of ferrous material and over 4 million pounds of non-ferrous scrap were recovered, contributing to the overall efficiency of the site clearance effort.

What the Avon Lake project demonstrated, beyond technical success, was the clear viability of Precision Demolition as a field methodology for high-consequence work. The risks present on this site were not theoretical. Failure would have resulted in structural pile-up, debris into public roads, or uncontrolled collapse near an active water intake. Traditional demolition practices could not have solved those problems. What enabled success was a structured engineering approach: verify the material, design the charges, sequence the collapse, and build safety into every layer of execution.

The core lessons from this project are clear. Multi-structure implosions in constrained environments require blast engineers, not just

blasters. Sequence timing cannot be assumed; it must be modelled and confirmed. Structural preparation must prioritise stability, not convenience. And most critically, explosive demolition, when designed correctly, is not the riskier option. It is the controlled, compressed, and more predictable solution for sites where complexity is the norm, not the exception.

Avon Lake was not a lucky blast. It was a planned collapse, by design, by testing, and by control. That is what separates traditional demolition from Precision Demolition. And that is what the industry must move toward as public tolerance for risk disappears and structural complexity continues to rise.

7 CONCLUSION

Precision Demolition represents the next evolution in explosive structural takedown. It is a methodology rooted in engineering, built on verified modeling, and executed through controlled sequencing with zero tolerance for failure. Unlike traditional approaches that rely on pre-weakening or simplified assumptions, Precision Demolition requires a comprehensive understanding of structural behavior, charge performance, and environmental constraints. It delivers a final result that is not just effective, but controlled, by design, by timing, and by consequence management.

The core of this methodology lies in its ability to operate in high-consequence environments. These are sites where structural collapse must be exact, debris must stay within a defined boundary, and every aspect of public risk - vibration, overpressure, debris projection, and structural rebound - must be calculated and mitigated in advance. Precision Demolition does not reduce complexity; it engages with it directly, using an engineered framework to deliver predictable outcomes under conditions that leave no room for improvisation. As industrial structures become more complex and community tolerance for disruption decreases, the industry must move beyond outdated demolition tactics. Precision Demolition offers a proven path forward - one where safety, schedule, and control are not trade-offs, but deliverables. Through the use of Precision Demolition, we are able to better engineer and design blasts, including multi-structure blasts, to mitigate risks and make explosive demolition a safer and better solution to critical demolition.

8 ACKNOWLEDGEMENTS

The successful execution of the Avon Lake Power Plant demolition would not have been possible without the leadership, coordination, and operational discipline of Trifecta Services Company. As the general demolition contractor, Trifecta was responsible for all site preparation, barricading, mechanical cutting, and environmental control and spent several months on-site preparing the structure for a successful blast. They also led community engagement and navigated the political and regulatory landscape that enabled this high-consequence blast to proceed. Their ability to manage site logistics, integrate engineering input, and maintain a safe and efficient working environment across multiple disciplines was central to the outcome of this project. Furthermore, we would like to extend a thank you to Trifecta for trusting the DetX team to complete this blast in a high-consequence environment.

We would also like to acknowledge Alford Technologies for their support in providing Dioplex DX-5000 shaped charges and for their continued partnership in pushing the boundaries of charge capability for structural steel applications. Advanced Explosive Systems (AES) provided key charge components and support during the field loading process. Saul's Seismic played a critical role in vibration analysis, instrumentation deployment, and post-blast monitoring to ensure compliance and risk mitigation throughout the operation.

[RETURN TO CONTENTS PAGE](#)

Prediction of vibration levels from falling masses using DIN 4150-1 in practice

C.A. Andersen

NIRAS A/S, Allerød, Denmark

ABSTRACT: Accurate forecasting of vibration levels resulting from falling masses is critical in the field of demolition, particularly in urban environments where the potential for structural damage and human disturbance is significant. This paper explores the practical application of DIN 4150-1, a widely recognised standard for evaluating vibrations in buildings, to predict the vibration levels generated when heavy objects impact the ground during demolition activities. The study delves into the intricacies of obtaining the best estimates by considering factors such as soil properties, mass and shape of the falling object, drop height, and impact velocity. A comprehensive methodology is presented, which includes a historic study of measurements from previous demolition projects. This historical data provides valuable insights into the behaviour of vibrations in various urban settings and helps refine predictive models. Through a combination of theoretical analysis and empirical data, the paper aims to provide a robust framework for practitioners to accurately forecast vibration levels. Key aspects include the calibration of predictive models, the importance of site-specific parameters, and the integration of advanced monitoring techniques. The findings are intended to enhance the reliability of vibration predictions, thereby mitigating potential structural damage and ensuring compliance with regulatory standards. By focusing on the unique challenges posed by urban demolition, this paper contributes to the body of knowledge in vibration forecasting and offers practical solutions for engineers and demolition experts. The ultimate goal is to improve safety and minimise the impact of demolition activities on surrounding structures and communities.

1 INTRODUCTION

When demolishing any structure by blasting, it is inevitable that it will hit the ground. If the structure is tall and heavy, it follows from first principles that a large amount of energy will be transferred in the impact with the ground, where it will dissipate to the surrounding areas as vibrations. If other structures are nearby, they might be subject to damage due to these vibrations. It is therefore important to be able to forecast vibration levels from falling masses.

One method to predict the vibration level from falling masses on the surroundings is presented in

the German norm DIN 4150 Vibrations in buildings – Part 1: Predictions of vibration parameters. The method is presented in Section 2.

When using the method in practice, it becomes apparent, that the method is quite crude and can be improved upon by implementing a few simple alterations.

2 METHOD PRESENTED IN DIN 4150-1

DIN 4150-1 states that the vibration level and its corresponding frequency depends upon parameters such as mass, height, ground conditions, surface conditions, geometry and strength of the falling



Figure 1. Cement silo at Vordingborg, demolished 6 April 2018.

structure, and more. An equation to estimate the maximum vibration level is given as:

$$v_{max} = k \left(\frac{E}{E_0} \right)^{0.5} \left(\frac{R}{R_0} \right)^{-m} \quad (1)$$

Where:

- v_{max} is the maximum vibration level [mm/s]
- E is the energy of the falling object ($E = Gh$), [kJ]
- E_0 is a reference energy level of 1 kJ
- R is the distance from the point of impact to the point of interest [m]
- R_0 is a reference distance of 1 m
- k is an empirical constant [mm/s]
- m is an empirical constant [-].

The empirical constants k and m are site specific and should be determined beforehand, e.g. by dropping smaller masses at the site and measuring the outcome. This is however not always practically possible and values based on experience are therefore often used.

DIN 4150-1 also states that in most cases, the vibration level from the falling masses impacting the ground, will be greater than the vibration level from the blast of the demolition charges.

The current version of DIN 4150-1 is from 2022. In the previous version from 2001 the same formula appeared, however there was an obvious error in the formula, as the range, R was to be inserted in mm leading to either over-estimation of the vibration level or inconsistent values of the k - and m -constants.

3 COMPARISON TO MEASURED VIBRATION LEVELS

The author has had the opportunity to use the method on three separate occasions for demolition projects in the previous years.

3.1 Vordingborg cement silo

A cement silo at the port of Vordingborg in Denmark was demolished on 6 April 2018. The silo comprised of a cylindrical concrete container with a height of 52 m, a diameter of 17 m and an estimated mass of 2,533,000 kg. The silo is shown in Figure 1.

Vibration monitoring was carried out at five locations in the vicinity of the silo as shown in Figure 2 (left). The predicted vibration levels, using the method from DIN 4150-1 are presented

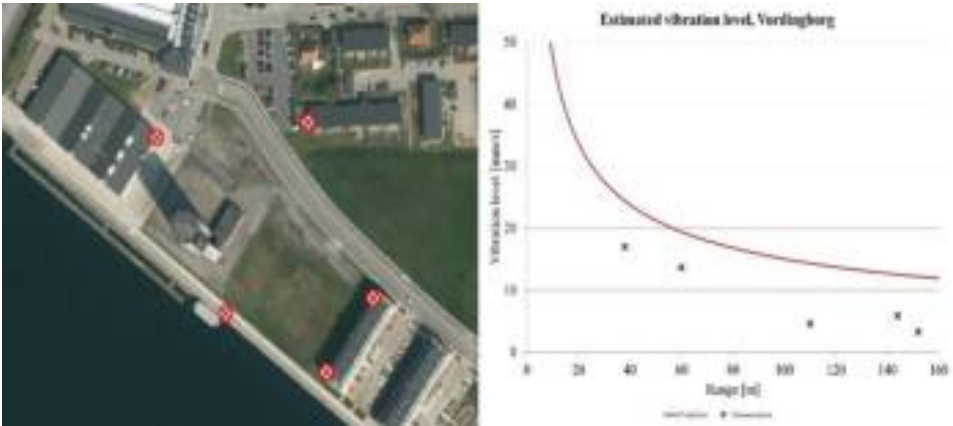


Figure 2. Vibration monitoring locations in Vordingborg (left) and prediction using DIN 4150-1 and measuring results (right).

in Figure 2 (right), along with the actual measuring results. The constants k and m were determined from measurements during a previous demolition at the same site.

As seen in Figure 2 (right), the measured vibration levels are quite a lot lower than the

predicted values. The prediction is off by between 42 to 271 %.

3.2 Amager Resource Centre

A 152 m concrete chimney was demolished in



Figure 3. The chimney at Amager Resource Centre demolished 31 October 2021.

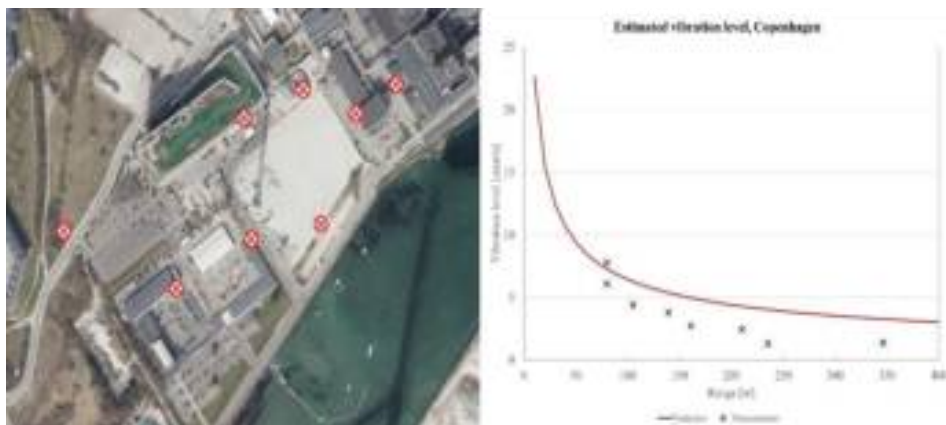


Figure 4. Vibration monitoring locations in Copenhagen (left) and prediction using DIN 4150-1 and measuring results (right).

Copenhagen on 31 October 2021. The chimney was shaped like a frustum of a cone on top of a revolved hyperbolic curve. The chimney had a diameter of 14.4 m at the base and 5.4 m at the top and had an estimated mass of approximately 2,101,000 kg. The centre of gravity was located

at a height of 42 m. The chimney is shown in Figure 3.

Vibration monitoring was carried out at eight locations in the vicinity of the chimney as shown in Figure 4 (left). The predicted vibration levels, using the method from DIN 4150-1 are presented



Figure 5. Chimney at Hedehusene, Denmark demolished 30 April 2023.

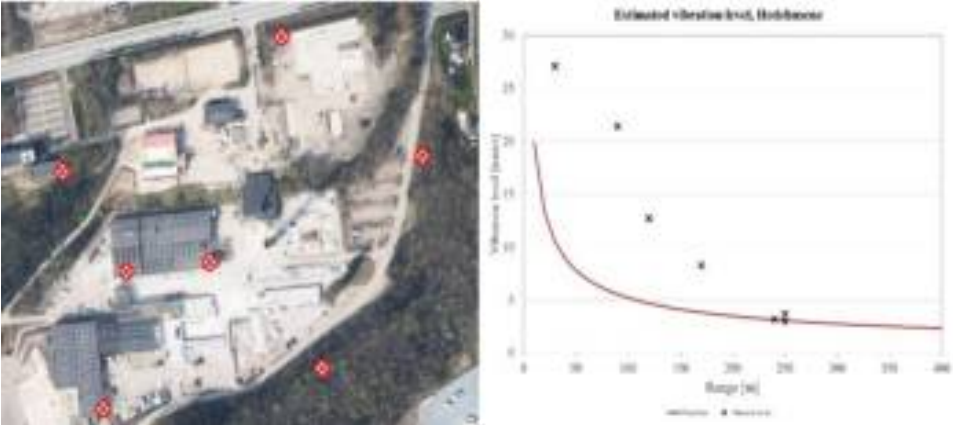


Figure 6. Vibration Monitoring locations in Hedehusene (left) and prediction using DIN 4150-1 and measuring results (right).

in Figure 4 (right), along with the actual measuring results. The constants k and m were determined through experience.

As seen in Figure 4 (right), the measured vibration levels are quite a lot lower than the predicted values. The prediction is off by between -7 % to +211 % with most measured values significantly below the predicted levels.

3.3 Hedehusene

The third example is a 90 m tall chimney in Hedehusene in Denmark, which was demolished on 30 April 2023. The chimney consisted of a cylindrical concrete shell, with a stepwise thickness of 20-30 cm and an external diameter of 7.3 m. The mass was estimated as 1,442,000 kg and the height of the centre of gravity was 44 m.

Vibration monitoring was carried out at seven locations in the vicinity of the chimney as shown in Figure 5 (left). The predicted vibration levels, using the method from DIN 4150-1 are presented in Figure 5 (right), along with the actual measuring results. The constants k and m were determined through experience.

As seen in Figure 6 (right), the prediction is quite good for some measuring locations, while the measured vibration levels are a lot higher than the predicted values for other measuring locations. The prediction is off by between -74 % to +2 %.

It is suspected that the reason higher vibration levels are measured in the near field, is due to the fact that the impact was on a concrete slab and that no sand cushion was employed.

3.4 Summary of presented test cases

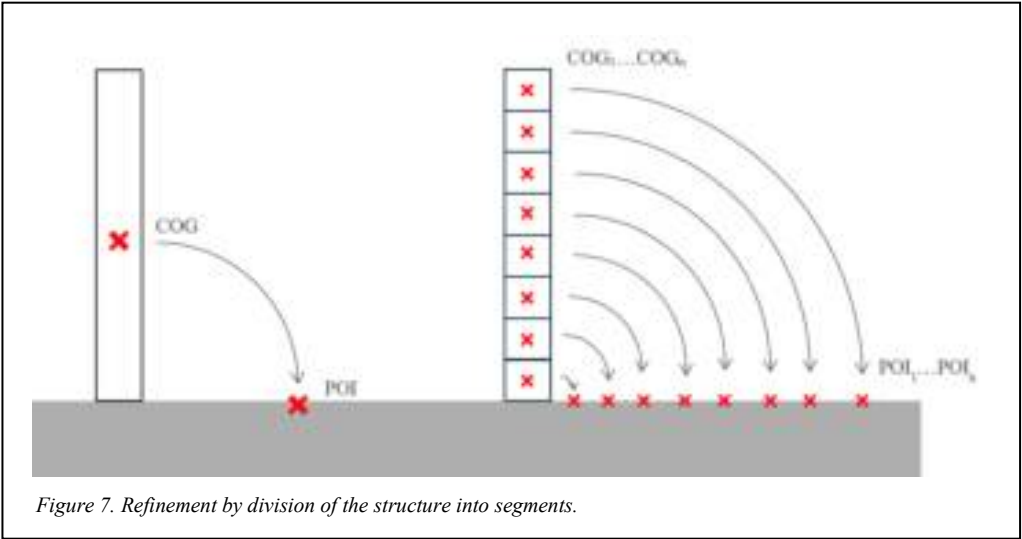
Of the 20 measurements presented in the previous sections, only three fall within 10 % of the predicted value, five fall within 25 %, 15 fall within 100 % and five are more than 100 % off. As the measurements represent the ‘true value’, the fault must be on the prediction.

There can be a number of reasons why the predictions are not accurate, some of them might include:

- The k and m constants are site specific and difficult to determine beforehand.
- The method does not specify which distance to use. Is it the distance from the foundation of the structure or the distance from the impact area? If it is from the impact area, is it from the impact of the centre of gravity or from the top of the structure.
- The entire structure will rarely hit the ground at the same time, e.g. structures of masonry which might break up during the fall or high-rise buildings which will hit the ground one floor at a time.
- The use of vibration mitigating measures like sand or rubble to cushion the fall of the structure can greatly influence the experienced vibration levels in the surrounding areas.

4 PROPOSED METHOD TO IMPROVE ACCURACY IN PRACTICE

Instead of observing the structure as a single structure with a ‘centre of gravity’ (COG) and counting the distance from the ‘point of impact’



(POI) of the COG, the structure can be divided up into segments, each with their own COG and POI as shown in Figure 7.

If the structure is uniform and homogenous, the segments can be of equal size, but if the structure is geometrically uneven, has varying densities, or the demolition plan intentionally breaks the structure into segments, the division into segments should reflect this. In the example presented in Figure 7, the structure is rigid and the collapse mechanism is similar to felling a tree.

Each segment will fall from a different height and thus have different potential energies and the distance from each POI to the measurement point will also differ. This leads to a predicted vibration level for each segment. The conservative way to summarise these predictions for a given measurement location, would be to add up the

contributions for each segment.

If the structure does not fall like a tree, but is expected to break up, a further refinement can be made by examining how many segments are expected to hit the ground at the same time.

4.1 Refined vibration forecast

The method is applied on the demolition of the chimney at Amager Resource Centre in 2021 (see section 3.2). The chimney was divided into 15 sections of 10 m each. In the preparation phase of the demolition, the vibration forecast was developed using the contribution from the four segments that provided the largest contributions. These vibration estimates were presented at the EFEE conference in Maastricht in 2022 (Andersen *et al.* 2022).

While post-processing the data for the

Table 1. Summary of measurements and predicted vibration levels for Amager Resource Centre.

Measuring point	Measured vibration level mm/s	Method as presented in DIN 4150-1		Divided into 15 segments, 2021 4 concurrent segments		Divided into 15 segments, 2025 2 concurrent segments	
		Prediction mm/s	Error %	Prediction mm/s	Error %	Prediction mm/s	Error %
1	4.40	6.3	43	8.6	95	4.4	0
2	3.80	5.4	42	7.9	108	4.0	5
3	2.75	5.0	81	6.7	144	3.3	20
4	1.30	4.0	211	5.3	308	2.7	108
5	6.10	7.3	19	10.7	75	5.4	-11
6	7.80	7.3	-7	11.6	49	5.5	-29
7	2.45	4.3	75	6.6	169	3.5	43
8	1.35	3.3	142	4.3	219	2.2	63

preparation of this paper, it has been found that doing the calculations for two concurrent segments impacting the ground, provides an estimate closer to the actually observed vibration levels. The revised estimates are presented in Table 1.

From the error values in Table 1, it can be seen that the ‘raw’ method, as presented in DIN 4150-1 is actually better than the estimate calculated by assuming four sections impact the ground concurrently, but also that it is possible to reduce the error significantly.

5 CONCLUSIONS

This study shows that it is possible to obtain more accurate predicted vibration levels from falling masses by refining the method presented in DIN 4150-1 instead of using the method as it is presented, however care should be taken to investigate the influence of how many segments the structure is divided into and how many segments will hit the ground at the same time.

Furthermore, the estimation of the constants k and m still presents a challenge, as they can only be determined retrospectively in most cases.

Lastly, the influence of sand or rubble to cushion the fall cannot be underestimated, as the prediction was too high in the two cases where cushions were in place (Vordingborg and Copenhagen, see Figure 1 and Figure 3) but too low in the case where no cushion was used (Hedehusene, see Figure 5).

6 ACKNOWLEDGEMENTS

Any use of generative AI in this manuscript adheres to ethical guidelines for use and acknowledgement of generative AI in academic research. The work has been thoroughly vetted for accuracy, and the author assumes responsibility for the integrity of the article. Generative AI has been used to assist in the formulation of the abstract for this article, but the actual article was written without the use of AI.

REFERENCES

- DIN 4150-1:2022 Vibrations in buildings – Part 1: Predictions of vibration parameters, *Deutsches Institut für Normung*, 2022
- S.P. Mann *et al.* 2024.Guidelines for ethical use and acknowledgement of large language models in academic writing. *Nature Machine Intelligence*, Volume 6: 1272-1274, November 2024.
- C.A. Andersen *et al.* 2022. Demolition and reuse of 152 m chimney at Amager Resource Centre, Copenhagen. *Proceeding from EFEE 11th World Conference on Explosives and Blasting*, Maastricht 2022.

[RETURN TO CONTENTS PAGE](#)

4. Drill and blast design methods and modelling

Influence of empty guideholes in single-blasthole experiments for tunnel blasting

C. Rodrigues San Miguel, C. Yi, N. Petropoulos¹ & D. Johansson

*Swebrec/Luleå University of Technology, Sweden
LKAB, Sweden¹*

ABSTRACT: Careful blasting is a controlled blasting technique that is often used to minimise the overbreak and damage to the remaining rock mass. Although decoupled charges in blastholes are common, the influence of empty guideholes in directing crack propagation has been less investigated. This study presents small-scale experiments using magnetite mortar cylinders with two and four empty guideholes around a central blasthole, charged with 0.28 kg/m³ of PETN with a 2.5 decoupling ratio. This work investigates the influence of the different guidehole configurations on crack development through ultra high-speed (UHS) photography and digital image correlation (DIC). The results indicate that the guideholes significantly influence the trajectories of the cracks, attracting their development and thus reducing the development of unwanted cracks. These findings provide insight into blast-induced cracks and contribute to knowledge for designing blasts where precise control of the cracks is critical.

Keywords: blast-induced crack propagation, blast damage, ultra high-speed photography, digital image correlation, contour blasting.

1 INTRODUCTION

Drill and blast designs in tunnelling involve the optimisation of variables to minimise the overbreak and blast-induced damage to the surrounding rock. Among existing design strategies, the use of empty drill holes, or so-called guideholes, is common in contour or cut blasting. The empty guideholes serve as a free face and are known to direct the propagation of cracks (Yue, *et al.* 2013). Although the empty guideholes are widely applied, their specific influence on the crack number, propagation direction, and final patterns are yet to be completely understood.

The general existing knowledge of empty guideholes comes from field experience and numerical modelling. However, the field cases are often affected by rock heterogeneity, which limits

the control of the dynamic interaction of the blast-induced cracks. Small-scale experiments are important for studying the blast-induced cracks under different blast designs. The advantage relies on controlling the experimental variables and observing the development of blast-induced cracks (Mohanti 1990). These results can be easily correlated to rock and explosive variables compared to field tests and therefore increase the understanding of crack behaviour. Overall, small-scale experiments have provided valuable information about directional blasting effect in the cracks of the empty guideholes around the blastholes and help to validate conceptual and numerical models based on fundamental principles (Mohanti 1990, Li *et al.* 2018).

Ultra high-speed (UHS) photography, in combination with the Digital Image Correlation

(DIC) method, is widely used in research to capture crack development in real time. These optical techniques enable the measurements of displacements and strains on sample surfaces (McCormick & Lord 2010). In the recent years, the implementation of this method in blasting experiments has provided unique insights into the response of the rock under the detonation of the explosives (Chi *et al.* 2019a, Chi *et al.* 2019b, Yang *et al.* 2021, Chi *et al.* 2024). These results can be used to assess the damage based on fracture intensity and fracture count density, defined by the length or number of cracks in a specific area. This approach enhances the analysis of crack propagation in blasting experiments and supports the interpretation of the damage under different blast designs, for example, how the configuration of blastholes and guideholes influences the resulting cracks.

Herein, small-scale blasting experiments conducted on rock-like cylinders with a single central blasthole and two different configurations of empty guideholes are presented. UHS photography and 2D-DIC methods were used to study crack propagation around the blasthole. The objective was to investigate the influence of

guideholes on crack propagation number, type, and directions, as well as to evaluate the damage level in the rock. The present results have potential applications in contour or cut blasting designs for tunnel or underground operations, where there is room for improving the efficiency of the design and minimising the damage.

2 MATERIAL & METHODS

2.1 Samples, Instrumentation, and Experiments Set up

Two magnetite mortar cylinders were designed and prepared to conduct blast experiments and implement the DIC method. The recipe and speckle patterning procedure followed were similar to R. San Miguel *et al.* (2024) and the design of the samples is shown in Figure 1. The samples had a central blasthole with diameter (ϕ_{bh}) 6 mm and either two or four guideholes distancing 50 mm between each other, with diameter (ϕ_{gh}) 10 mm and 50 mm length (L_{gh}). The blastholes were charged with detonation cord of 5 g/m strength due to its ease of measuring and controlling the diameter and explosive charge

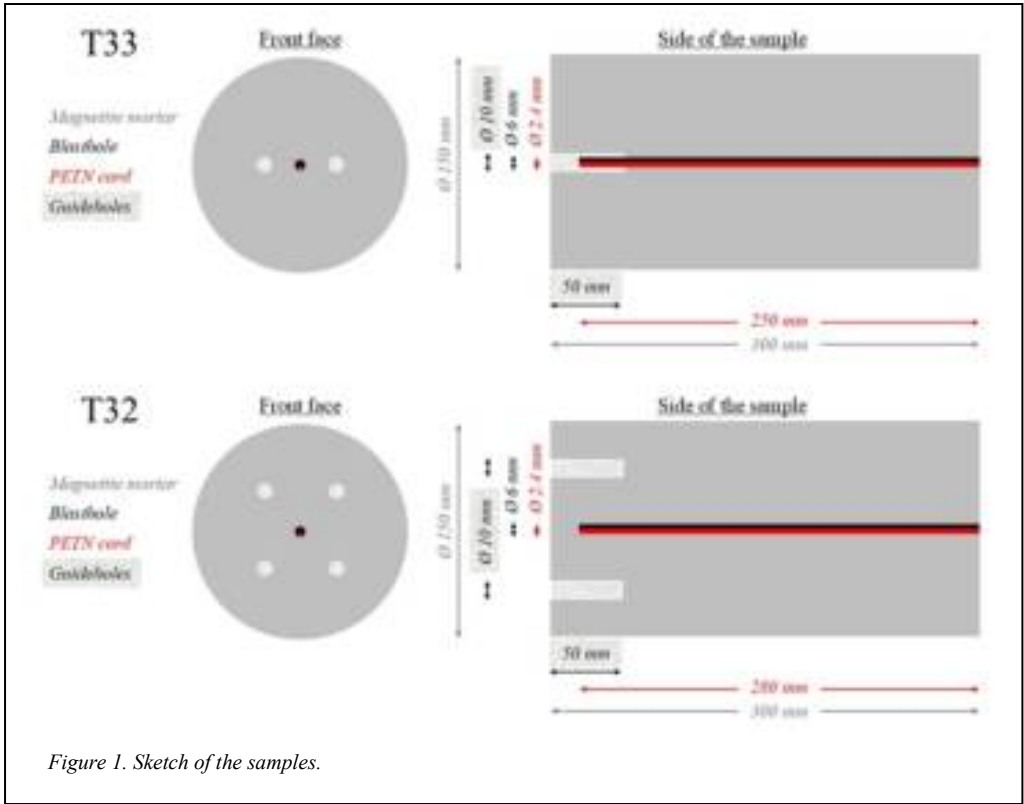


Figure 1. Sketch of the samples.

Table 1. Sample dimensions and explosive charge measurements.

Sample	Guideholes No.	M_0 [kg]	ϕ_0 [mm]	L_0 [mm]	L_{01}, L_{02} [mm]	M_1 [g]
T33	2	13.22	150	301	250	1.25
T32	4	13.20	150	299	274	1.37

Table 2. Equipment Configuration and Experimental Resolution.

Sample	FPS	Pixels	Image resolution [mm]	Time resolution [μ s]
T33	206349	256 x 256	0.59	4.83
T32	206349	256 x 256	0.59	4.83

FPS: Frames Per Second

concentration. The explosive charge diameter (ϕ_0) measured 2.40 mm resulting in a decoupling ratio (β) of 2.5. Velocity of the velocity of detonation (VoD) measurements of the detonation cord were conducted, and the velocity was 7248 m/s. A specific charge (q) of 0.28 kg/m³ or linear charge (q_l) of 0.005 kg/m remained constant in the tests. The final experimental measurements obtained from the study are compiled in Table 1.

Data collection was carried out using a Phantom 2512v ultra high-speed camera (UHSC), along with a data acquisition system and the Phantom Camera Control (PCC®) software. Each experiment involved the use of a sample, a set of four LED lights to provide high illumination, and the above-mentioned equipment. The experimental setup followed the configuration described in previous investigations (R. San Miguel, *et al.* 2024), and the resulting configuration is shown in Table 2.

The blast tests were conducted, and data was collected from the front surface of the cylinders. ZEISS CORRELATE® software was used to apply the DIC method, which enabled the exclusion of the empty guidehole areas from the analysis to avoid calculation errors.

2.2 Ultra High-Speed photography and Digital Image Correlation analysis

The collected data consisted of image sequences (frames) of the sample surface. For each test, 200 frames were extracted: 50 frames before detonation and 150 frames after. These sequences were visually analysed to determine the time of the

first gas ejection. The UHSC images were used to detect the developed cracks, classify them based on their type, and measure the damage through fracture intensity and fracture count density prior to the first gas ejection. The region of interest (ROI) in the DIC analysis was limited to the sample surface, excluding the guidehole areas. The DIC parameters were set as facet size 15 px and point distance 7 px, ensuring sufficient spatial resolution and over 50% subset overlap.

To assess the deformations and crack development, cumulative strain and displacement analyses were performed. The displacement thresholds were set as -0.295 mm and +0.295 mm, based on the image resolution of 0.59 mm per px. The coordinate system was centred in the blasthole, allowing the analysis of the rock movement in all directions. The strain threshold was set from 5 % to 10 %, as calculation errors were observed in the range of 0 - 3 % in the reference images, while cracks became distinguishable around 5 %. Crack development was analysed by examining areas with increasing strain across different time intervals between the reference frame and the moment of gas ejection.

3 RESULTS AND ANALYSIS

The UHSC images provide information on detonation, gas ejection times and blast-induced crack development on the sample's surface, especially on the number, and type of cracks (Table 3). The analysis consisted of a qualitative investigation of the crack patterns and behaviour characteristics of the sample under the established

Table 3. UHS photography results.

Sample	Detonation duration [μs]	First gas ejection [μs*]	Radial cracks (RC)	Branching cracks (BC)	Guidehole cracks (GC)	Inter-guidehole cracks (IghC)
T33	34.49	579.60	5	2	2	0
T32	37.80	280.14	0	1	4	1

*: Timestamp after detonation

blast designs up to the time of gas ejection from any crack. In T33, with four guideholes, the gases ejection time occurred at 280.14 μs and in T32, with two guideholes was detected at 579.60 μs after detonation. These results show a large difference, being 229.46 μs earlier in the case with more guideholes.

The cracks developed in the samples differed in type and directions. In the sample with two guideholes, a total of nine cracks were found in the sample surface, one crack going towards and through each guidehole after detonation (guidehole cracks) and five radial cracks (RC), two branching cracks (BC) before the gas ejection. In the case of four guideholes, four guidehole cracks developed from the blasthole after detonation and, one branching and one connecting the guideholes (inter-guidehole crack) developed before gas ejection. The DIC strain results provided valuable

information on when and where the cracks were developed without user bias in the sample surface. The results were divided into three stages: before detonation, mid-way to gas ejection and gas ejection time (Figure 2 and Figure 3).

The cracks demonstrated a symmetric path along the vertical axis of the sample which divides the sample into two halves (red-dashed lines in Figure 2 and Figure 3). Cracks propagating along the trajectory that intersects both, the blastholes and the guideholes were also observed in both study cases and are herein referred to as guidehole cracks (GC). Some cracks appeared to develop from another crack in the late stages of propagation and were classified as branching cracks. As a main difference in the results between the two tests, inter guidehole cracks (IghC) were found in the T32 study case connecting the guideholes. In T33, radial cracks were found to

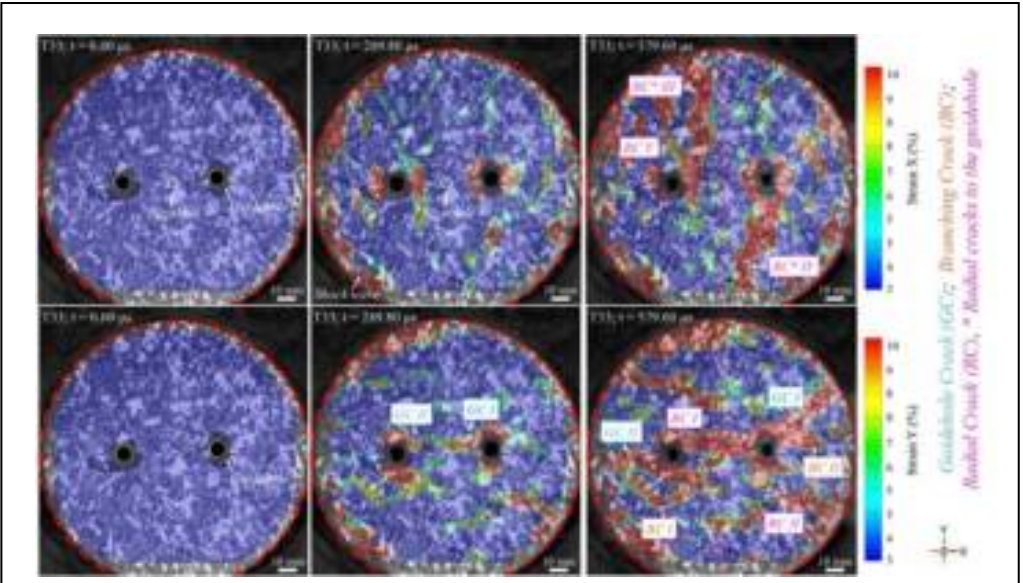
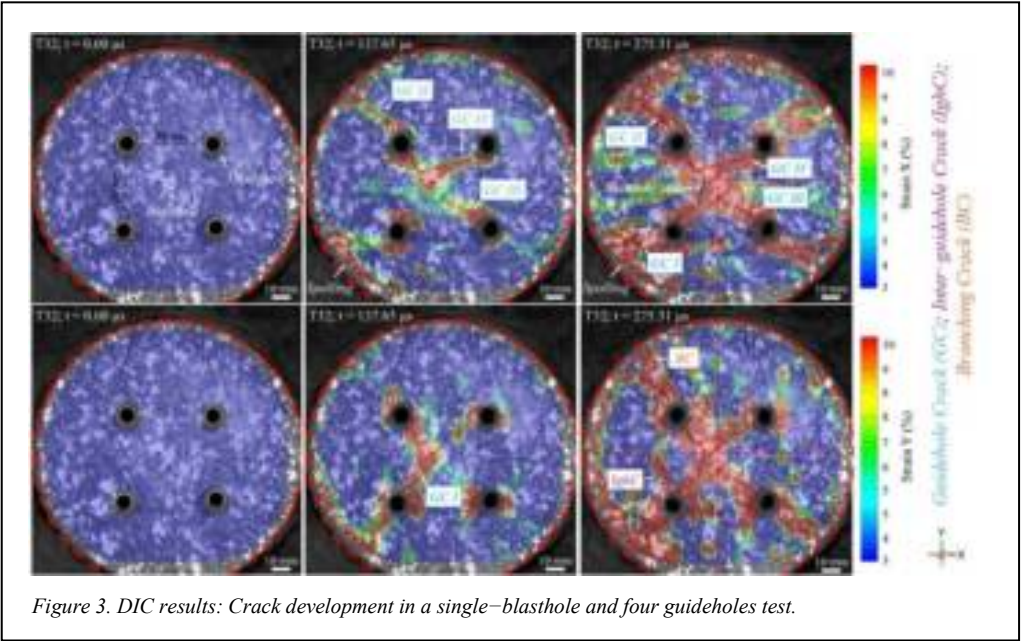


Figure 2. DIC results: Crack development in single-blasthole and two guideholes test.



connect the edges of the sample with the blasthole. Another type of cracks following the path from the guideholes towards the guideholes were found and were classified as well as special type of radial cracks (RC*). Deformations appeared at the edges of the sample and spalling cracks were recognised in the samples due to the reflection of stress waves at the boundary. The crack number and lengths were measured from the last time stamp before gas ejection and damage measurements as fracture intensity and fracture count density (Table 4). The results of the damage measurements were categorised into two groups: one including all the induced cracks, and another distinguishing between desired cracks. The desired cracks are the guidehole cracks, and the rest as unwanted cracks, considered as damaging the sample surface.

Taking the total induced cracks values of the damage measurements as the maximum or 100%

of damage, the results from the fracture intensity in the damaging cracks reveal to 74.30 % of the total induced cracks in the sample with two guideholes. On the other sample, with four guideholes, the damaging fracture intensity is 7.81 % of the total. These results differ greatly as the damage induced in the sample with two guideholes is 66.49 % higher than in the one with four guideholes. Overall, the damage measurement values increased with longer intervals between the end of the detonation and the first sign of gas ejection.

4 DISCUSSION

This study explored the influence of two and four empty guideholes on blast-induced cracks around a single-blasthole. UHS photography combined with the DIC method effectively captured data on

Table 4. Damage measurements.

Sample	ROI [mm ²]	Total induced cracks		Damaging cracks	
		P_{2I} [μm ⁻¹]	N_f [cracks/μm ²]	P_{2I} [μm ⁻¹]	N_f [cracks/μm ²]
T33	14131.95	31.50	636.85	23.41	495.33
T32	14994.29	19.17	400.15	1.50	66.69

crack development, their origin and the evolution of damage measurements after blasting.

A key observation was that an increased number of guideholes led to earlier gas ejection. The four guidehole sample showed the first gas ejection with 29.86 μs difference compared to the similar designed sample in R. San Miguel *et al.* (2025). In the two guidehole configuration, the first gas ejection was detected 159.60 μs earlier than in T23 of the previous work. This difference in the time of gas ejection suggests that the geometrical configuration of the empty guideholes plays a critical role in modifying the stress wave propagation and gas ejection behaviour.

In terms of crack propagation number, patterns and type of cracks, the present results are consistent with the trends reported in R. San Miguel *et al.* (2025). The presence of two guidehole cracks passing through the two empty guideholes in the sample T33, as well as T23 of the above-mentioned study confirms that the development of the cracks is directly influenced by the blasthole and guidehole configurations. These observations align with the findings of Li *et al.* (2018) and Tian *et al.* (2020), who concluded that the empty holes distance directly influences the outcome on directing the cracks. Furthermore, the four empty guidehole configuration in this study present a nearly identical crack pattern to the one observed in R. San Miguel *et al.* (2025), highlighting the consistency of the results. These results corroborate the findings from Chen *et al.* (2018) who conducted analytical and numerical modelling studies in setups with empty guideholes around a blasthole that demonstrated crack directionality and improved stress distribution in the rock.

DIC strain analysis enabled the classification of the cracks based on their propagation behaviour. Samples with symmetrical guidehole layouts consistently showed symmetry of the crack patterns. This symmetry supports the concept that a well-designed empty hole arrangement can improve the control of the crack direction and predictability of the blast-induced cracks. Moreover, the DIC data enabled the quantitative assessment of the blast-induced damage, revealing that increasing the number of guideholes reduced both fracture intensity and fracture count density. This indicates more controlled stress distribution and improved crack directionality, which is advantageous for contour blasting applications. Additionally, a trend was observed where longer delays between detonation and gas ejection are

correlated with higher damage measurements, suggesting a relationship between gas confinement duration and damage severity.

5 CONCLUSION AND FUTURE WORK

The small-scale blasting experiments involved the variation of the blast design in terms of empty guidehole configurations in single-blasthole samples, and led to the following findings:

- The present investigation confirms that the number and configuration of empty guideholes significantly influences the development and direction of the blast-induced cracks.
- Symmetrical layouts and optimal spacing enhance crack propagation control, reduce unwanted cracks or damage, and improve the blast design efficiency.
- The integration of UHS photography and DIC offers a powerful method for tracking dynamic process of crack propagation by blasting.

Future investigations could explore the use of the DIC method to measure the damage as a function of time. Although velocity measurements were not included in this study, as they demonstrated to provide less significant results on the crack development than the blastholes and guideholes configurations (R. San Miguel *et al.* 2025), they could still provide complementary data. Incorporating such results in future studies could support the development of statistical models. Properly designed empty guidehole configurations significantly improve the blasting outcomes by guiding the cracks and minimising the unwanted damage. Overall, these findings contribute to a better understanding of blast-induced damage across various scenarios for contour blasting.

6 ACKNOWLEDGEMENTS

This investigation was financed by Swedish Rock Engineering Research Foundation (BeFo) under project 427 and the Swedish Blasting Research Centre (Swebrec). The authors would like to thank LKAB Kiruna Mine for providing the test site.

REFERENCES

Chen, B., Liu, C., & Yang, J. (2018). Design and application of blasting parameters for

- presplitting hard roof with the aid of empty-hole effect. *Shock and Vibration*, Volume 2020, p. 16.
- Chi, L.Y., Xu, X., Zhang, Z. & Yang, J. 2024. Strain field development, fracturing, and gas ejection in decoupled charge blasting using granite cylinders. *Rock Mechanics and Rock Engineering*. Retrieved from <https://doi.org/10.1007/s00603-024-04079-y>
- Chi, L.Y., Zhang, Z., Aalberg, A. & Li, C. 2019a. Experimental investigation of blast-induced fractures in rock cylinders. *Rock Mechanics and Rock Engineering*, Volume 52, p. 2569–2584. Doi: 10.1007/s00603-019-01749-0
- Chi, L., Zhang, Z., Aalberg, A., Jun, Y., & Li, C. (2019b). Fracture processes in granite blocks under blast loading. *Rock Mechanics and Rock Engineering*, Volume 52, p. 853–868. Doi: 10.1007/s00603-018-1620-0
- Li, M., Zhu, Z., Liu, B., Zhou, L., & Dong, Y. (2018). Study of the effect of empty holes on propagating cracks under blasting loads. *International Journal of Rock Mechanics and Mining Sciences*, p. 186–194.
- McCormick, N., & Lord, J. (2010). Digital image correlation. *Materials Today*, 13(12), p. 52–54. Doi: [https://doi.org/10.1016/S1369-7021\(10\)70235-2](https://doi.org/10.1016/S1369-7021(10)70235-2)
- Mohanti, B. (1990). Explosion generated fractures in rock and rock-like materials. *Engineering Fracture Mechanics*, 35(4–5), p. 889–898.
- R. San Miguel, C., Petropoulos, N., Yi, C., & Johansson, D. (2024). Experimental investigation of blast-induced crack propagation based on digital image correlation analysis. *Shock and Vibration*, 2024(1). Doi: 10.1155/2024/4149322
- R. San Miguel, C., Petropoulos, N., Yi, C., & Johansson, D. (2025). Effects of guidehole configurations on crack propagation patterns in tunnel applications: Insights from digital image correlation and qualitative analyses. (U. Özer, A. Karadogan, U. Sahinoglu, M. Özyurt, & Z. Sertabipoglu Kayci, Eds.) *FRAGBLAST*, p. 953–962.
- Tian, G., & et al. (2020). Influence of charge hole spacing on the crack propagation behaviour under the effect of empty-hole directional blasting. *Advances in Civil Engineering*, p. 10.
- Yang, L., Aiyun, Y., Siyu, C., Shizheng, F., Chen, H., & Huanzhen, X. (2021). Model experimental study on the effects of in situ stresses on pre-splitting blasting damage and strain development. *International Journal of Rock Mechanics and Mining Sciences*, Volume 138. Doi: 10.1016/j.ijrmms.2020.104587
- Yue, Z., Yang, L., & Wang, Y. (2013). Experimental study of crack propagation in polymethyl methacrylate material with double holes under the directional controlled blasting. *Fatigue & Fracture of Engineering Materials & Structures*, 36(8), p. 699–833. Retrieved from <https://doi.org/10.1111/ffe.12049>

RETURN TO CONTENTS PAGE

Development of Scaled Damage Criteria for Blasting

J. Alvarez

Enaex EMTS Collahuasi Site, Chile

ABSTRACT: In a large-scale open-pit mining operation, continuous efforts are made to control damage to the slope walls. An essential tool in this effort is determining the PPVc (critical peak particle velocity) in areas of interest. By knowing this value and generating criteria, drilling and blasting designs can be established to reduce vibrations that affect wall construction. It is crucial to understand that the PPVc changes as different mining phases develop, especially due to exploitation on the benches, which leads to vertical inherited damage. Therefore, it is imperative to regularly update the PPVc based on field tests that provide accurate data. With this information, vibration damage criteria can be established, and drilling and blasting techniques can be designed to minimise the impact on the remaining rock mass. This document presents a methodology for generating scaled criteria that best represent the reality of the rock mass.

1 INTRODUCTION

The scaling of PPVc (critical peak particle velocity) is essential in the design and development of blasting engineering. To ensure the integrity of the walls in mining, it is vital to evaluate the effects of blasting and how they impact the lower benches. Explosions in the upper benches can significantly alter the characteristics of the lower bench, which, if not controlled, can result in the loss of crest or berm in that bench.

Given this challenge, the primary purpose of this study is to validate a methodology for developing drilling and blasting designs based on scaled damage criteria for the specific rock mass. For this purpose, field tests will be conducted in different locations, using traditional instrumentation to collect data on Vp and its behaviour on the wall. The collected information will be essential for developing the damage criteria of Hoek (Hoek & Brown 1988) and Vergara (Vergara 2016).

These criteria aim to provide a solid foundation and safe limits for creating designs that protect the integrity of the slope walls. Additionally, the results of this study are expected to offer a valuable contribution to the development of meshes and the establishment of safer and more efficient contour blasting practices.

In conclusion, the study seeks to improve drilling and blasting designs by understanding the impacts and effects of upper bench blasts on lower benches. Through this methodology, which includes practical tests and the development of Hoek & Vergara damage criteria, it aims to ensure safety and optimise wall construction in mining.

2 HISTORIC THEORETICAL BACKGROUND

2.1 Evolution from Hoek & Brown to Vergara

Over the years, various authors have developed and refined criteria, formulations, and tests. This evolution has allowed the transition from a model



Figure 1. Open-pit mine.

focused on intact rock to one that scales the rock mass, considering factors such as ‘deterioration’ due to structures, blasting, and other elements. These advancements aim to provide greater accuracy in the PPVc values used, with the goal of minimising the damage caused by vibrations to the wall, especially those generated by contour blasting. Below, we will review the definitions or formulations that are the most relevant.

2.2 *Scaling of PPVc - Rock mass altered by blasting*

Historically, the calculation of PPVc was based on elastic parameters derived from laboratory tests on intact rock. These initial models did not consider structural constraints or complete geotechnical characterisation of the rock mass. Thus, their practical application was limited to determining

the maximum peak velocity of the P-wave that intact rock could withstand. However, this situation did not reflect the real conditions faced by professionals daily. In reality, operations occur in an environment with constant physical alterations due to factors such as structural conditions and blasting.

Relying solely on these original models can lead to significant errors. In response to this limitation, a methodology has been developed for scaling PPVc for an altered rock mass, considering the modifications and alterations that distance the rock from its original intact condition.

2.3 *Therefore, the calculation of PPVc was modified.*

Equation 1 for Critical peak particle velocity by Hoek:

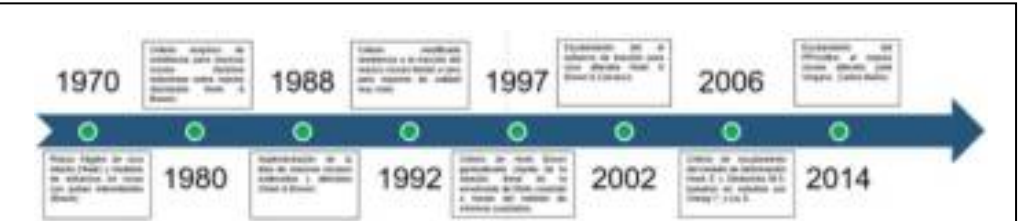


Figure 2. Timeline of the Evolution of Damage Criteria.

$$PPV_c = \frac{\sigma_t V_p}{E}$$

Where,

PPV_c = Critical peak particle velocity (mm/s)

σ_t = Tensile strength of intact rock (MPa)

V_p = P-wave velocity (m/s)

E = Young's modulus (MPa)

Later came Equation 2 for Scaled critical peak particle velocity.

$$PPV_{c_{mr}} = \frac{\sigma_{mr} V_{pmr}}{E_{mr}}$$

Where:

PPV_{c_{mr}} = Scaled Critical Peak Particle Velocity to altered rock mass (mm/s)

Σ_{mr} = Tensile strength scaled to altered rock mass (MPa)

V_{pmr} = P-wave velocity (m/s)

E_{mr} = Young's modulus scaled to altered rock mass (MPa)

The study of scaling the rock mass is based on research conducted by Sheorey (1997) and Bieniawski (2011). Both authors have proposed a formula based on RMR (Rock Mass Rating), suggesting greater stability of the rock mass.

Equation 3 for Scaled critical peak particle velocity by RMR.

$$PPV_{c_{mr}} = \frac{\sigma_t \cdot e^{\left(\frac{RMR-100}{27}\right)} \cdot V_p}{E_i \cdot e^{\left(\frac{RMR-100}{36}\right)}}$$

Where:

σ_t = Tensile stress (MPa)

RMR = Rock Mass Rating

E_i = Young's modulus (MPa)

V_{pmr} = P-wave velocity (m/s)

This established the following damage criterion in Table 1.

3 METHODOLOGY

3.1 V_{pmr} determination

The correct determination of V_{pmr} is crucial

to make sense of everything discussed earlier. Therefore, the following questions arise:

- Does V_p alter due to blasts from upper benches?
- Is V_p behaviour uniform along the entire wall?
- Is V_p identical at the crest and floor levels?
- Which V_p should be used, and does the choice matter?

To answer these questions, the methodology proposes placing geophones along the monitoring well during field tests. This setup allows for establishing two zones for analysis: one 'intact' and one 'altered', as exemplified in the following test description. Scaling formulas will be used for the analysis.

Table 1. Vergara Damage Criterion (2016).

Damage Criterion	Description of induced damage
Zona 1 > PPV _c (H&P)	The tensile strength of the intact rock is exceeded, causing the rupture of the rock matrix and the generation of new structures.
PPV _c > Zona 2 > PPV _{c_{mr}}	The tensile strength of the rock mass is exceeded, resulting in rupture within the rock mass (rock bridges), along with the opening of pre-existing structures.
Zona 3 < PPV _{c_{mr}}	No induced damage occurs in the rock mass due to blasting.
•PPV _c (H&P) = H&P Damage Criterion	
•PPV _{c_{mr}} = RMR Damage Criterion	

3.2 Test for obtaining V_p Phase 13 / UGT-PPK.

In the test conducted in Phase 13, corresponding to UGT PPK with characteristics of hard rock, a specific configuration was adopted for measurement. In the design of this test, three geophones were placed within a single well, distributed along its length, as illustrated in Figure 3. The strategic placement of these geophones aims to capture variations in measurements at different depths or points within the well, providing a detailed understanding of the properties and behaviours of the rock at the specific test location. The interpretation of these data will be essential for planning and executing future operations under similar conditions or in the same geographical location.

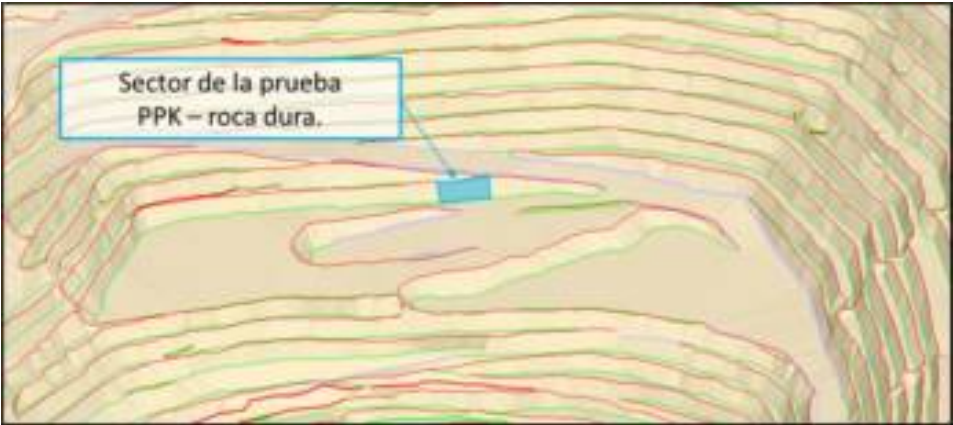


Figure 3. Test sector phase 13, UGT PPK, Hard Rock.

After conducting the test in Phase 13, UGT PPK with hard rock, it was confirmed that all geophones distributed along the well recorded data effectively and without failures. This indicates that the configuration and placement of the geophones were appropriate and that the devices operated correctly. Having valid records from all geophones is essential as it provides a comprehensive picture of the behaviour and properties of the rock mass at different depths within the well. This information is crucial for understanding the rock characteristics and making informed decisions regarding future operations or research in the area.

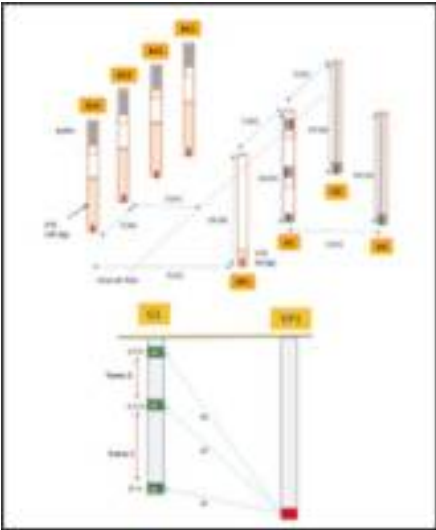


Figure 4. Diagram of 3 geophone test.

After analysing the data obtained from the geophones along the well, the team was able to segment the wellbore column into two distinct sections to calculate the P-wave velocity (V_p). These sections were selected based on the hypothesis that the rock mass would exhibit different behaviours depending on its depth.

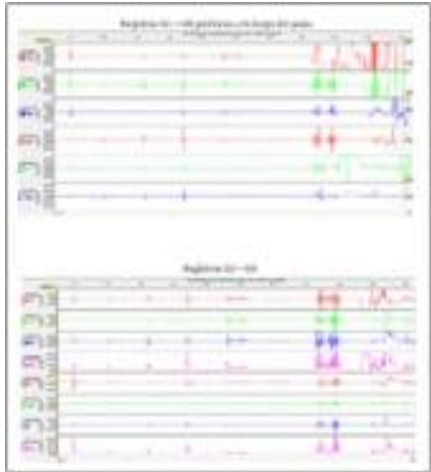
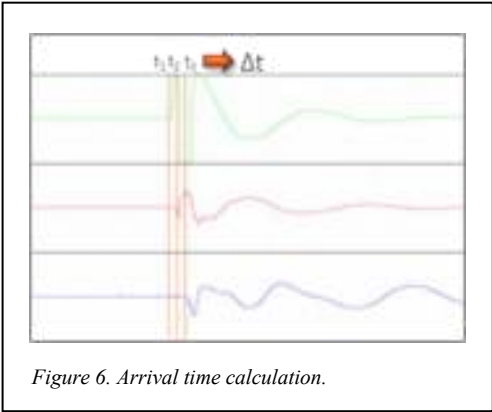


Figure 5. Recorded data.

The first section, closer to the bottom of the well, was considered representative of more 'intact' rock, less affected by previous activities such as blasting. Meanwhile, the second section, closer to the well collar, was expected to be more affected or 'altered' due to previous blasting in upper benches.



The purpose of this segmentation is to determine if there are indeed variations in Vp between these two sections, which could provide valuable information about the actual conditions of the rock mass and how these conditions might influence future blasting operations or overall stability in the area. These differences could also be crucial for adjusting techniques or procedures in drilling and blasting activities, aiming to maximise efficiency and minimise risks.

Table 2. Coordinates and distances.

	Norte	Este	Cota	Distancia
Vp	81348.59	30168.77	4178.38	
G1	81343.49	30168.97	4180.14	5.4
G2	81343.49	30168.97	4187.55	10.5
G3	81343.49	30168.97	4193.46	15.9

The significant difference in P-wave velocities (Vp) between both sections indicates significant variations in the quality and/or conditions of the rock mass along the wellbore. A higher Vp, as observed in Section 1, indicates more ‘intact’ and less fractured rock. In contrast, a lower Vp, such as recorded in Section 2, suggests a higher degree of fracturing or ‘alteration’, possibly due to the cumulative impact of previous blasting.

The presence of altered areas in the rock mass, indicated by reduced Vp, may have significant implications for the planning and execution of future blasting operations. These areas could be more susceptible to damage or instability during blasts, compromising bench stability and overall operational safety.

Table 3. Vp calculations.

Tramo 1	
Δ Distancia (m)	5.1
Δ tiempo (s)	0.0014
Vp (ARGM) m/s	3640
Tramo 2	
Δ Distancia (m)	5.4
Δ tiempo (s)	0.003
Vp (ARGM) m/s	1809

The criteria for ‘damage’ established based on Vp and UCS (Uniaxial Compressive Strength) provide a valuable framework for assessing and controlling the risk associated with blasting in the PPK/F13 area. Through this criterion, the team can establish safe vibration thresholds and optimise blasting techniques to mitigate the potential for damage to the rock mass.

The team obtained design criteria for both sectors, as shown in the summary figure.

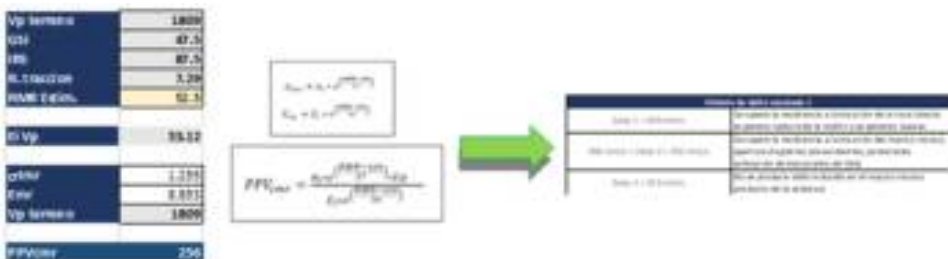
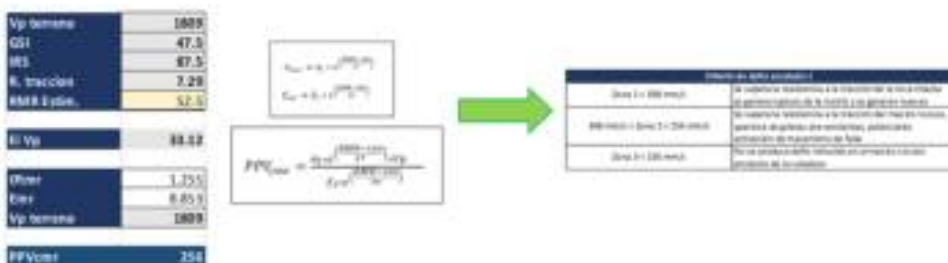
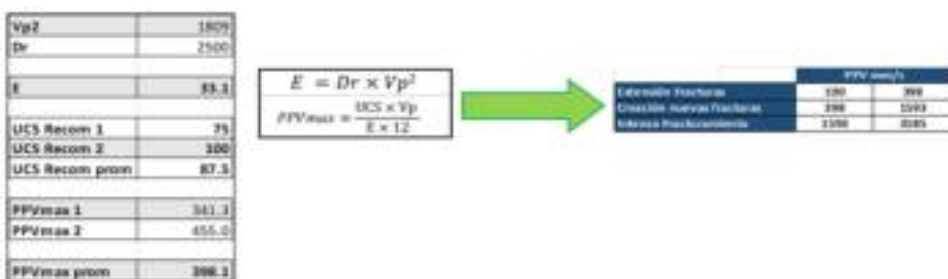
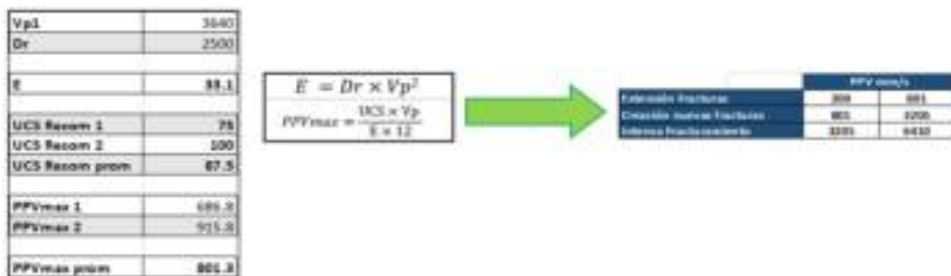
Based on the test results, the team has highlighted the following points:

- VP was successfully determined in two distinct sections, identifying areas of ‘intact’ and ‘altered’ rock.
- A classical ‘damage’ criterion was established.
- Scaling PPVc for both sections resulted in differentiated damage criteria. This will provide valuable guidance for adjusting designs and minimising damage to the walls from blasting.
- The damage criteria, as in previous analyses, vary concerning the PPVc values provided in laboratory studies. This underscores the need to adapt current drilling and blasting designs.

Given the positive results achieved using this methodology, testing continues in various sectors of the mine, showing consistently realistic results. This enables the development of different damage criteria, which are applied in the formulation of drilling and blasting designs.

4 RESULTS

With this new methodology of field testing and analysis, 8 tests have been conducted with positive



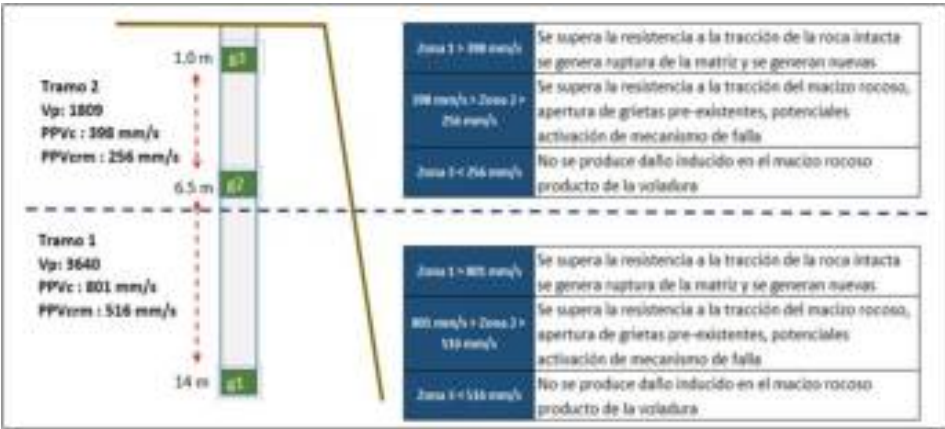


Figure 11. Final scaled damage criterion PPK - Phase 13.

Table 4. Summary of tests conducted with geophones along the well.

Zone	Prueba 1	Prueba 2	Prueba 3	Prueba 4	Prueba 5	Prueba 6	Prueba 7	Prueba 8
LOG	AR3M	PPK	PPK	RAPK+	AR3M	RAPK Alt	RAPK m	ANPK
GU	10 a 20	45 a 50	40 a 50	45 a 50	20 a 25	30 a 40	30 a 35	45 a 50
MM	20	53	50	53	28	40	38	60
RS	10 a 25	75 a 100	75 a 100	150 a 200	10 a 25	75 a 100	25 a 50	100 a 150
Separation zones	No	Si	Si	Si	Si	Si	Si	Si
Vp 1	2.029	3.640	4.688	4.792	2.615	3.339	2.917	4.164
Vp 2		1.829	1.458	1.767	1.777	1.581	1.136	1.364
PPV/c1	287	801	632	1.217	323	840	429	1.001
PPV/c2		358	193	486	109	398	167	328
PPV/cm1	137	516	302	784	114	482	240	692
PPV/cm2		256	122	287	56	228	94	226

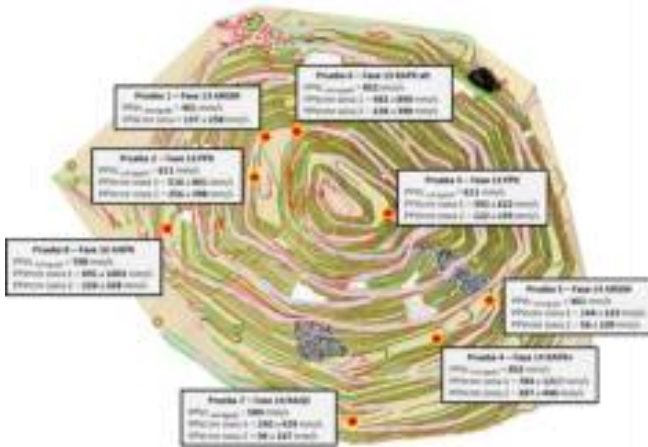


Figure 12. Test sectors and laboratory delivered PPVc values, and scaled PPVc ranges.

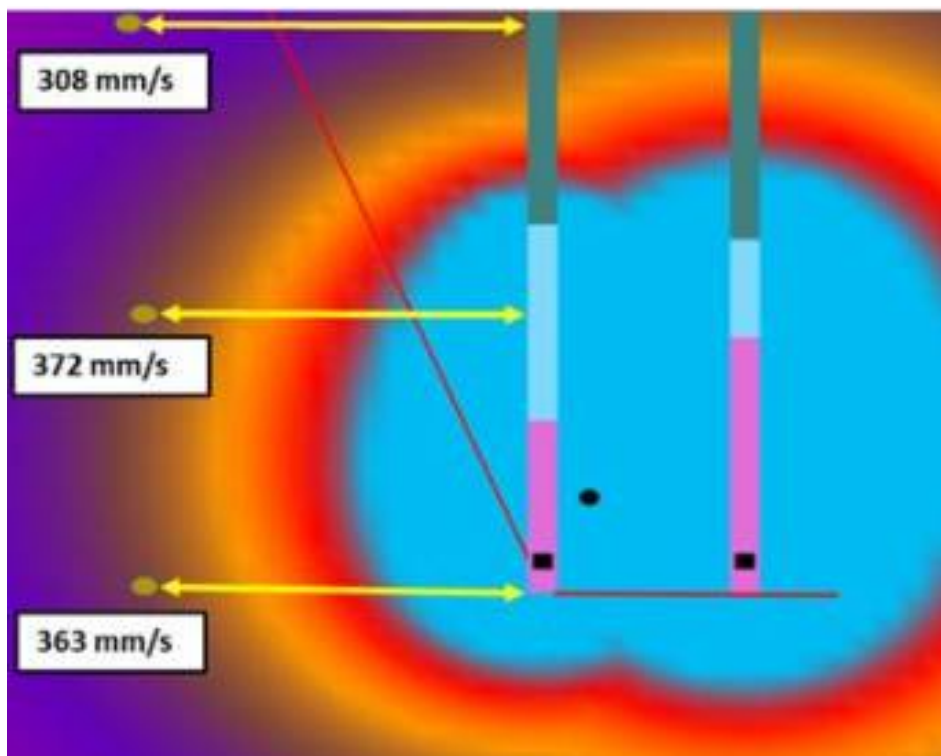


Figure 13. Vibration simulation.

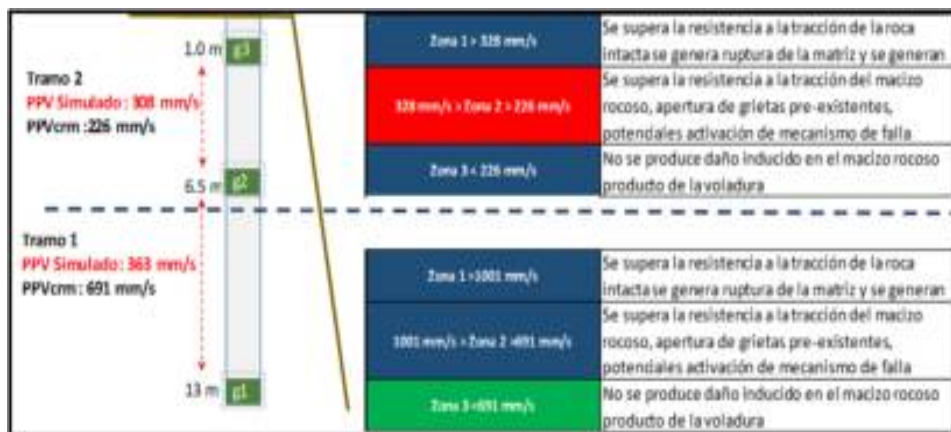


Figure 14. Application of scaled damage criterion.

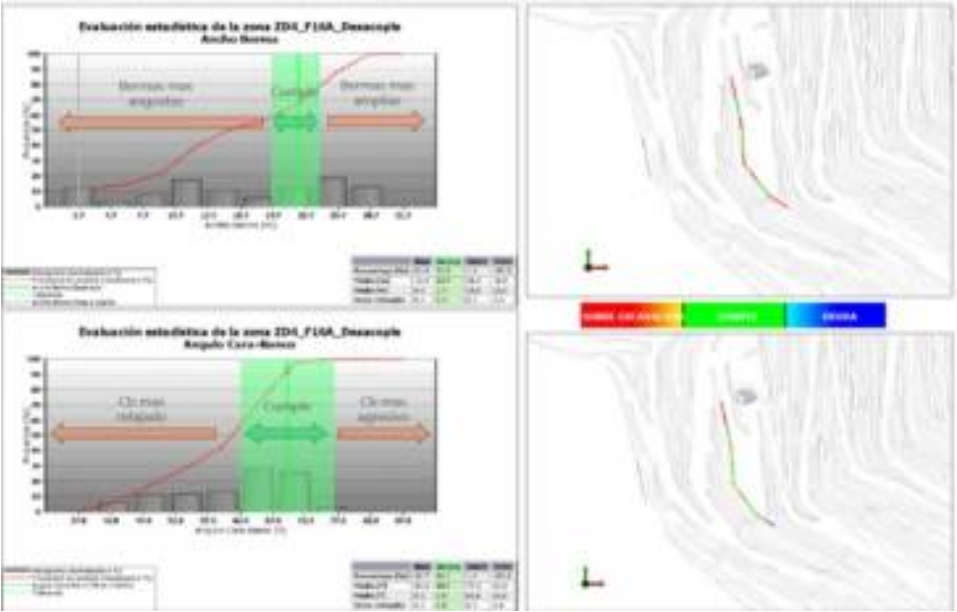


Figure 15. Bench width results.

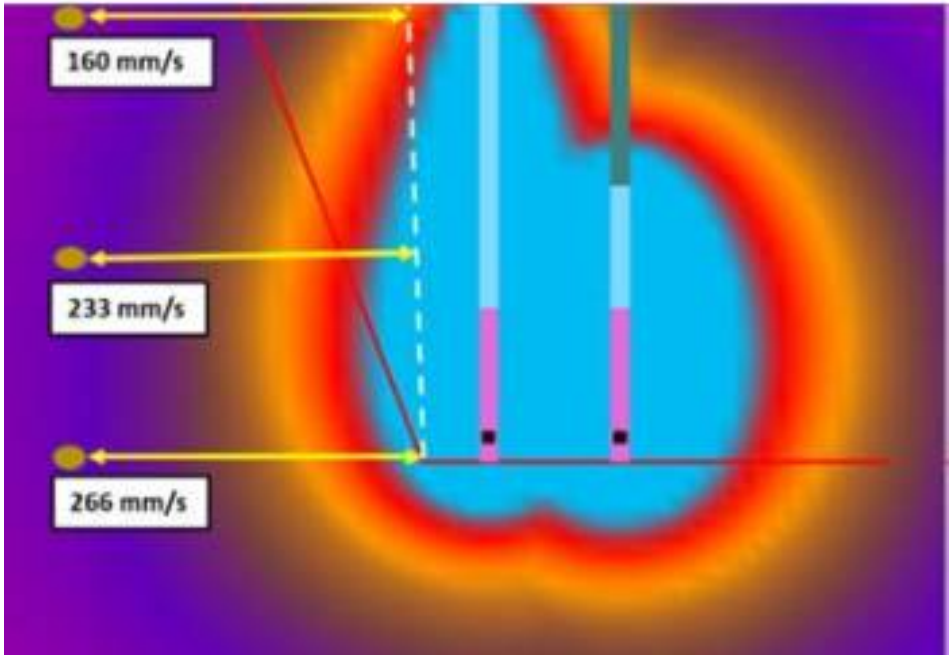


Figure 16. Vibration simulation.

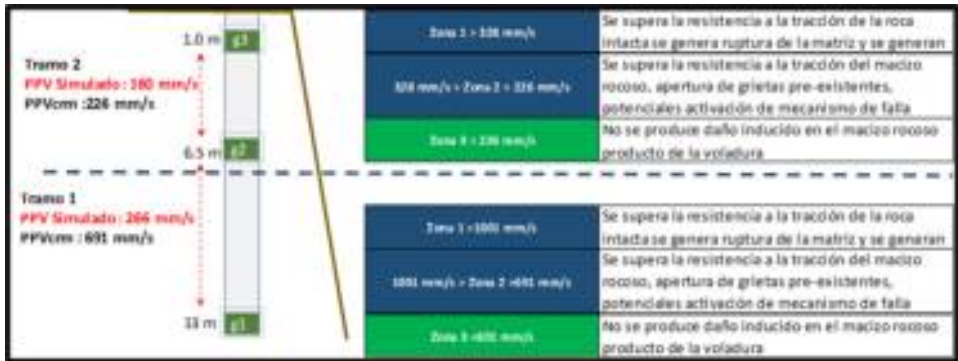


Figure 17. Application of scaled damage criterion.

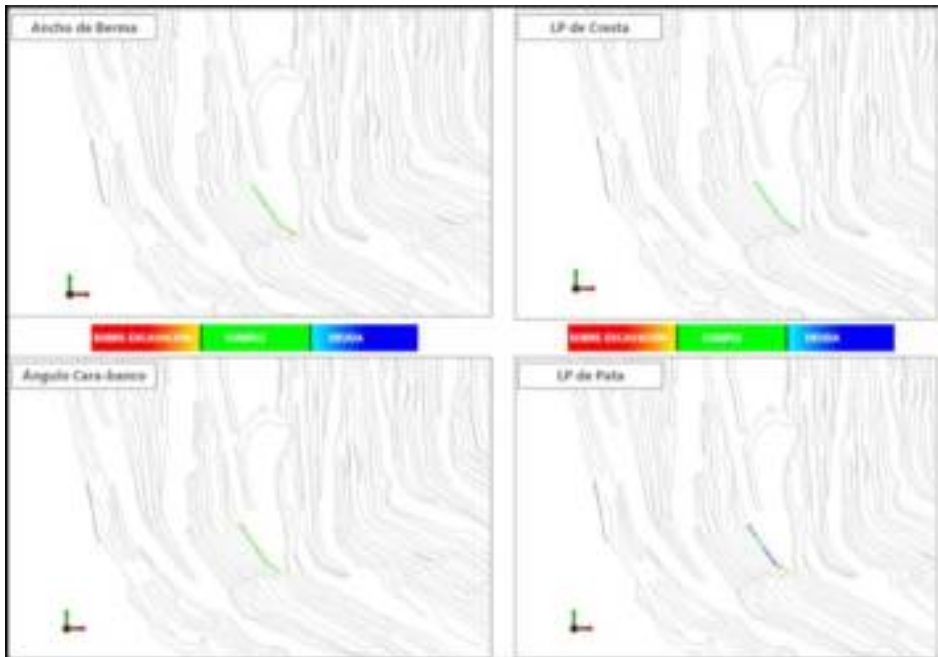


Figure 18. Bench width results with modified design.

and consistent results according to the sectors where the tests were performed. As shown in figures 12, the summary is presented in table 4.

As demonstrated with this methodology, both PPC and PPV_{cmr} were successfully determined, enabling the generation of scaled damage criteria for the various sectors where tests were conducted. This will facilitate the creation of designs that adhere to these criteria for lower benches, which must comply both at the crest and in the program line, as illustrated in the example.

4.1 Base Case - Offset 0 m.

Based on the near-field model obtained from the blasting, simulations were conducted to assess vibration levels 10 m inside the program line (criterion used). According to this, the vibration levels generated for Section 2 during blasting exceed the estimated threshold based on the methodology presented in figures 13 and 14.

Based on this, this design fails to meet the scaled damage criterion, as vibrations levels exceed the estimated thresholds in Section 2, potentially leading to bench loss. This aligns with the results shown in figure 15, highlighting poor bench width compliance. Therefore, multiple iterations are underway to generate compliant designs. One of the alternatives that meets the criteria is shown in figure 15.

4.2 Modified case by scaled damage criterion, Offset 1 m - buffer without stemming.

As an example, the configuration used is shown with a larger offset and leaving the buffer line without a stemming. This configuration fully complies with the scaled criterion, as reflected in the results, particularly improving bench width significantly.

The application of adjustments in both designs and loading configurations has resulted in an improvement in the trend of Factor of Condition and Design results, see figures 16 to 18.

5 DISCUSSION & CONCLUSIONS

Based on the field tests conducted by the team, it has been determined that through particle velocity measurements, there are zones of 'intact' and 'altered' rock within the wellbore column, which will eventually form the bench wall. This indicates that there are different maximum vibration levels that the rock mass can withstand before vibration-induced damage occurs. The determined P-wave velocities (V_p) in these zones have allowed the

team to obtain valuable results, establishing Peak Particle Velocity (PPV) thresholds both through traditional methodology and through scaling, and constructing scaled damage criteria.

These findings are crucial as they provide essential information for developing appropriate drilling and blasting designs in different sectors defined through the proposed methodology. By determining particle velocity in 'intact' and 'altered' rock zones, the team has identified how varied conditions of the rock mass are impacted by blasts from upper benches. The establishment of PPV thresholds provides a crucial tool for evaluating and controlling vibrations in the field, essential for minimising potential damage to bench walls and ensuring the integrity of the area.

The methodology for establishing scaled damage criteria developed by the team offers clear parameters based on more realistic terrain characteristics.

In summary, the proposed methodology for field tests and analysis has been validated as not only highly implementable, requiring only 2 to 3 additional wells and 4 geophones, but also significantly enhances the establishment of PPV thresholds and scaled damage criteria, which more accurately represent the terrain's reality. These findings have proven to be of great value for the formulation of drilling and blasting designs in each sector, moving away from laboratory-provided values that do not faithfully reflect the terrain's conditions. As a result, better designs and simulations have been implemented, minimising vibration-induced damage and ensuring safer and more efficient operations. The application and adjustment of these new damage criteria have led to an improvement in observed trends, both in the condition factor and design factor.

6 ACKNOWLEDGEMENTS

In the drafting of this report, the main contribution came from the Technical Team of Enaex Collahuasi Plant. This experience has reaffirmed the idea that teamwork always yields positive results. Although it may sound self-referential, the first acknowledgments are directed towards ourselves because, without the collaboration and mutual support among all team members, successfully completing this project would not have been feasible. Each member contributed from their specialty and position, adding value to the final document. The members of our valuable team are as follows: Jaritza Ramirez, Sebastian Navarro, Matias Leyton, Leonardo de Ferrari,

Rodrigo Echegaray, Aldo Abarzúa & Darwin Alfaro.

This team has been led by Jorge Alvarez, Technical Manager EMTS for Collahuasi. Each of them has demonstrated dedication, professionalism, and an undeniable commitment to the project. Through their contributions and collaboration, this report has gained life and relevance. Their passion and dedication have been essential for the success of this joint effort.

We would also like to thank the operations personnel at Enaex plant, represented by: Baltazar Marin, Erwin Montiel & Gonzalo Araya.

Lastly, our families, the cornerstone of our work. Total Thanks.

REFERENCES

Hoek, E. & Brown, E.T. 1988. The Hoek-Brown failure criterion - a 1988 update.

Vergara, J. 2016. Considerations on scaling of critical PPV to the rock mass. *In UMining* 2016. Santiago: Orica Latin America.

Accinelli Oliva, J. Unknown date. Analysis of P-wave velocity and its influence on damage generation due to geotechnical degradation.

Hewitt, C. 1955. Seismic velocities from surface measurements.

Estrada, L.A. 2015. Seismic Prospecting.

Chuquicamata Division of Codelco Chile. (1997). Estimation of rock mass strength in practice. *In Interdivisional Geotechnical Workshop*.

[RETURN TO CONTENTS PAGE](#)

Optimisation of blast design in complex geological environments via advanced digitalisation

S. Taoum, E. Coarita, O. Riahi & G. Lokou

Global Technical Solutions (GTS), EPC-Groupe, Paris, France

ABSTRACT: The efficiency and safety of blast operations in complex geological environments remain key challenges in modern mining. This paper presents a simulation framework that advances blast design by integrating detailed physical models of explosive behaviour and rock mechanics. The system captures essential blast-induced phenomena, including wave propagation, fragmentation, and structural impact. Advanced numerical models simulate shock wave propagation and dynamic fracture in heterogeneous media. The framework is calibrated through an optimisation process based on simulation results and field feedback. A real-time feedback mechanism enables continuous recalibration using sensor and geostatistical data, ensuring robust and responsive decision-making under geological uncertainty. This holistic approach paves the way for more predictable, efficient, and safer blasting practices, bridging the gap between theoretical modelling and operational needs. The framework holds promise for reducing environmental footprint and setting new standards for intelligent blast planning in the extractive industries.

1 INTRODUCTION

Blasting remains the primary method for rock breakage in surface and underground mining, as well as in large-scale civil engineering projects. However, achieving efficient, safe, and predictable blast outcomes remains a major challenge due to the intrinsic heterogeneity of geological formations. In complex rock masses, lithological variability, anisotropy, faulting, jointing, and stratification can significantly alter the propagation of shock waves and the fragmentation response of the material (Onederra, Furtney & Jang 2015, Sanchidrián *et al.* 2019). This geological variance introduces uncertainty into even the most carefully designed blast patterns, resulting in outcomes such as overbreak, ore dilution, underbreak, and structural instability.

A critical insight, often underestimated in empirical design, is that realistic geological layering and heterogeneity must be explicitly included in numerical models to enable accurate blast outcome prediction. Oversimplified or homogenous models tend to obscure critical behaviours such as wave focusing, reflection at interfaces, and preferential fracture propagation along weak planes (Ghasemi & Monjezi 2013). As digital geological modelling advances, it becomes possible to integrate data from core logging, geophysics, and remote sensing into 3D simulation environments, making realistic, site-specific simulations both feasible and impactful (Piergiorgio *et al.* 2020).

This paper introduces a high-fidelity, physics-based simulation framework for blast design that explicitly incorporates geological variability. The

model is based on finite element methods (FEM) for solving the dynamic wave equation in heterogeneous continua, coupled with the Chapman-Jouguet (CJ) theory for explosive detonation modelling. Unlike empirical tools that approximate fragmentation or energy dissipation, this approach resolves the governing partial differential equations that describe pressure, velocity, and stress fields within complex media (Zhou *et al.* 2020, Tsiampousi *et al.* 2017).

By integrating geological, explosive, and operational parameters into the same numerical model, the framework enables dynamic prediction of not only primary fragmentation but also geomechanical responses such as post-blast damaged zones. Furthermore, this predictive capacity provides a foundation for design optimisation, as simulation results can be combined with field measurements to adjust key blast design parameters (e.g., burden, spacing, delay timing) in a systematic and controlled manner. With a more extensive database, this could lead to the development of an iterative methodology to refine blast design based on both simulated and measured blast outcomes (Dai & Lu 2022).

The article is structured into three major components:

- Geological Modelling and Heterogeneity Representation. This section presents the methodology for constructing and integrating heterogeneous geological models into the simulation workflow, emphasising spatial resolution and property assignment strategies.
- Operational Blast Design and Physical Simulation. This part describes the finite element modelling of shock wave propagation, detonation physics based on

- Chapman-Jouguet conditions, and fracture mechanics within complex rock masses.
- Design Optimisation from Simulation Feedback. The final section presents a data-driven approach that combines simulation results with field measurements to adjust key blast parameters, such as burden, spacing, and delay timing, with the goal of maximising fragmentation efficiency while minimising both operational risks and costs.

In summary, this paper presents a comprehensive simulation and design framework that incorporates geological realism, improves physical accuracy, and bridges the gap between modelling and field practice. Such an approach is vital for next-generation blasting strategies that are grounded in predictive analysis and practical feedback.

2 ROCK MASS CHARACTERISATION

2.1 Rock mass characterisation

MWD (measurement while drilling) files obtained from drilling machine provide key parameters such as penetration rate, percussion, feed force, rotation and flushing pressures along the drilled hole at fairly regular intervals. This information, combined with the drilling machine’s radius and its rotation speed, is used to compute the specific energy required to break or excavate a unit volume of rock, following the approach proposed by Teale (1965). Existing correlations between RMR (Rock mass rating) and specific energy for certain types of rock allows to estimate the RMR parameter (Celada *et al.* 2009), which is then used to estimate other insightful parameters such as compressive strength, tensile strength, density and elastic Young’s modulus (Figure 1). For each

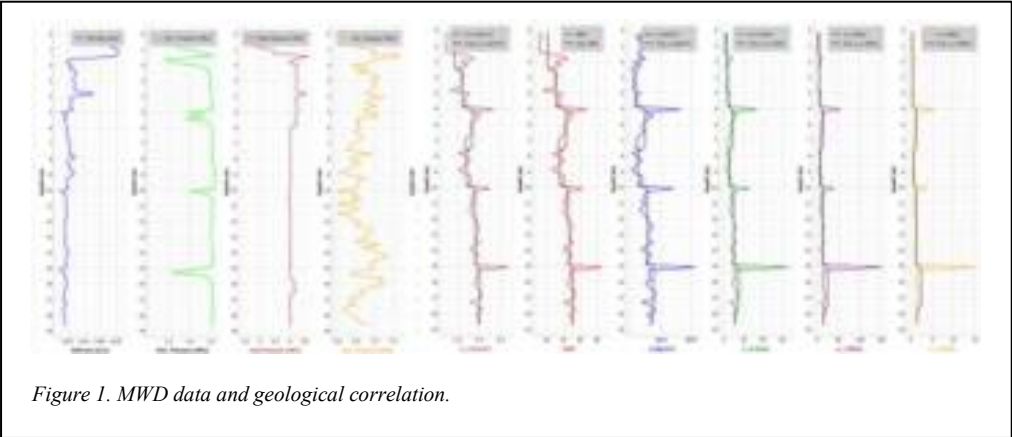


Figure 1. MWD data and geological correlation.



Figure 2. Interpretation of MWD results showing the estimated Young's elastic modulus.

measurement point, these parameters are computed and averaged over predefined depth interval, enabling the Characterisation of rock mass properties along the borehole, as shown in Figure 2.

Intact rock tests, geomechanical station data, and the interpretation of MWD analysis confirm that the analysed rock mass can be divided into two distinct layers. The upper layer, approximately 2 m deep, consists of a weaker rock mass affected by blast-induced damage from previous drilling operations, whereas the rock mass at greater depths is more competent. The dynamic mechanical properties of these rock masses are listed in Table 1.

2.2 Discontinuity sets

In the field, standing beneath a rock face is not

permitted due to the risk of rockfall. As a result, the classical methodology for measuring discontinuity orientation (Figure 3a) is not always feasible. To overcome this limitation, a numerical approach for discontinuity analysis has been developed. Using the point cloud of the area of interest, the algorithm takes as input several parameters, such as: the number of neighbouring points to consider; a planarity coefficient used to determine whether points lie on the same plane; and a minimum angular threshold to identify whether two detected planes are considered distinct. Based on a user-defined number of potential sets, the method identifies clusters, estimates the number of planes, and computes their normal vectors. From these vectors, the dip and dip direction of each plane are calculated and represented as poles in a stereographic projection.

Figure 3b shows the detected surfaces that

Table 1. Mechanical properties of the rock mass.

Property		Weak Limestone	Strong Limestone	Unit
Density	ρ	2600	2600	Kg/m^3
Poisson's ratio	ν	0.2	0.2	
Elastic modulus	E	14.8	24.9	GPa
Hoek-Brown parameter	s_c	70	70	MPa
Hoek-Brown parameter	m_b	2.6	4.2	
Hoek-Brown parameter	s	0.6	0.6	

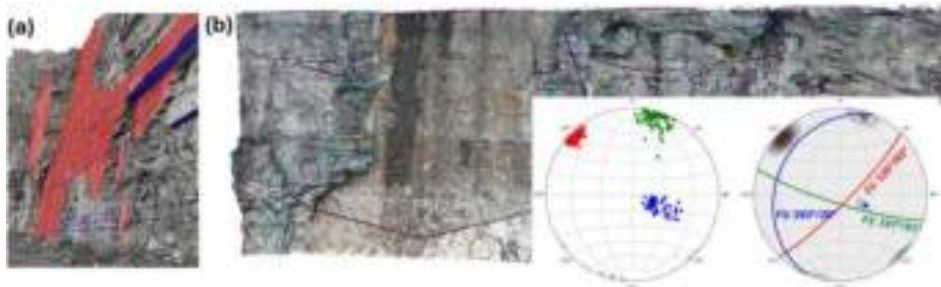


Figure 3. (a) Field-observed main discontinuity sets. (b) Orientation of discontinuity planes derived from point cloud analysis.

define the planes, along with stereonet displaying point densities that indicate the location of each set.

The numerical results indicate that the rock mass is characterised by three main sets of discontinuities, which are consistent with field observation. The first set is vertical and oriented $135^{\circ}/82^{\circ}$; the second set is sub-horizontal, with an orientation of $293^{\circ}/26^{\circ}$; and the third set is also vertical, oriented $197^{\circ}/81^{\circ}$.

2.3 Open discontinuities

Open discontinuities up to 20 cm wide were observed in the area prior to blasting, as shown in Figure 4. The orientations, $130^{\circ}/80^{\circ}$ and $129^{\circ}/82^{\circ}$, indicate that they belong to the first set. These open discontinuities are visible on the front wall; farther from the front wall, they appear relatively closed, as indicated by the dashed line.

3 BLAST DESIGN

The case study presented here is a blast in which the issue is an unintended large size of fragmentation Figure 5.

It is important to emphasise that the operation, from the first to the final step of the blast, was conducted without the use of modern digital or automated technologies.

Instead, both the site evaluation and the execution of the blast were carried out using conventional practices, relying on manual procedures and operator experience.

This context presents a valuable opportunity to assess the potential benefits of integrating advanced technologies into blasting operations. It allows for a more precise evaluation of how innovations such as data-driven design, real-time monitoring, and spatially accurate drilling can improve the efficiency, safety, and environmental

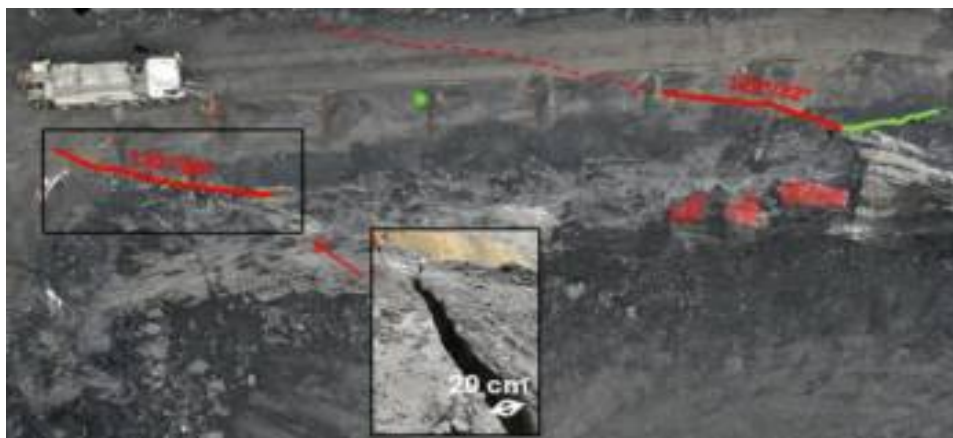


Figure 4. Open discontinuities observed in the front wall prior to blasting.



Figure 5. Illustration of large size fragmented rock.

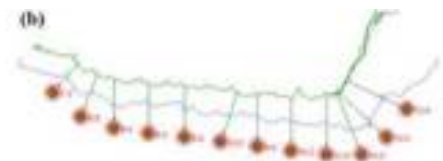
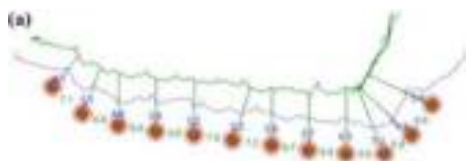


Figure 6. Illustration of blast design: (a) collar burden and spacing, and (b) toe spacing.

performance of blasting activities.

3.1 Operational implemented design – review

The technical team collected detailed information on-site in the pit to document the blast. For analysis purposes, the entire blast was subsequently reconstructed using a 3D digital blast design tool, where key parameters such as burden and spacing at both the collar and toe were examined, as illustrated in Figure 6.

Table 2. Explosive properties.

Product	ANFO	Emulsion
Density	0.9 (± 0.02)	1.22 (± 0.07)
Velocity of detonation (VOD) in confined steel	3500 m/s	4900 m/s
Total mass energy	3.8 MJ/Kg	3.0 MJ/Kg
Total volume energy	3 MJ/L	3.6 MJ/M
Detonation pressure	13.6 GPa	14.3 GPa

Blast holes were charged with ANFO (Ammonium Nitrate Fuel Oil) in dry and intact conditions, while water-bearing holes were loaded with water-resistant Emulsion. Table 2 contains the technical values for both used products.

The blast holes were charged using a tri-detonation configuration in wet conditions, incorporating a detonating cord to ensure continuous and reliable initiation. The charging conditions are shown in Figure 7.

Each hole was drilled to a target depth of 20.5 m, including 0.8 m of subdrilling. The explosive column had a diameter of 165 mm. Stemming lengths were set at 3.5 m for holes charged with ANFO and 4.5 m for those charged with emulsion.

3.2 Observations

Geological conditions, including pre-existing discontinuities, bedded rock formations, complex joint and fracture networks, and heterogeneous rock mass strength, play a critical role in governing the distribution of explosive energy during blasting. These features often lead to inefficient energy transmission, resulting in suboptimal fragmentation and reduced rock breakage efficiency. In addition, the lack of precision in energy placement, caused by

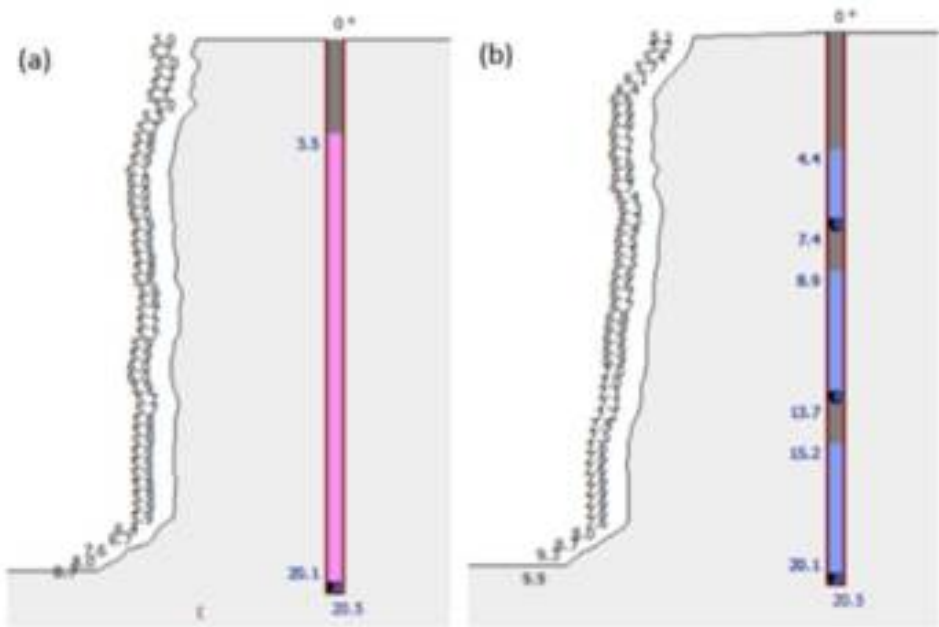


Figure 7. Charging conditions: (a) for dry holes using ANFO product, and (b) for wet holes using emulsion.



Figure 8. High-pressure gas escape through detonation phase resulting in energy loss.

the absence of drill hole geolocation, unchecked deviations, and insufficient QA/QC procedures, can significantly compromise blast performance.

Video footage of the detonation clearly reveals substantial energy losses: in 6 out of 12 holes, visible gas venting occurred during firing (Figure 8). This phenomenon, typically associated with stemming failure, results in the premature release of explosive gases and a corresponding reduction in energy transmitted to the surrounding rock mass. While stemming failure is often attributed to insufficient stemming length or inadequate confinement, it may also be linked to improper placement of detonating cord in the stemming, which seem to be the most effective explanation in this case.

4 GEOMECHANICAL MODEL

4.1 Geometry, geomechanical parameters and blast design

The geometry model consists of 291 645 tetrahedrons (Figure 9a). In the geomechanical model, two rock materials were considered, based on *in situ* measurements. Moreover, only open discontinuities were included in the modelling, as these are not expected to transfer stress across their surfaces. For the sake of simplicity, the mechanical properties assigned to these open discontinuities were assumed to be equivalent to those of air. The geometric configuration and the blasting sequences used in the modelling are those described in Section 3.

4.2 Elasto-damage model

The rock mass behaviour is modelled using an elasto-damage model based on the well-known Hoek-Brown criterion. The model includes a damage variable, d , which enables the macroscopic representation of microcrack evolution within the rock mass. When $d = 0$, the

material is in the purely elastic regime. If the stress state reaches the Hoek-Brown strength criterion, microcrack initiation is assumed to occur. Depending on the applied loading, the damage variable then increases progressively, up to a maximum value of $d = 1$, which corresponds to complete material failure. During the damage evolution, a minimum damage threshold ($0 < d_{min} < 1$) is considered, representing a state in which microcracks are present but not yet connected. This threshold defines the remaining, partially intact rock that may persist after blasting. Further details on this elasto-damage model can be found in Taoum & Coarita (2025).

4.3 Calibration of fragmentation based on numerical results

The damage level at the end of the simulation was mainly analysed in two zones: the back-break and the toe. In the back-break zone, the distance from the charge line to the back-break surface is approximately 1 m. This distance allows us to determine a minimum damage threshold of 0.45, indicating that rock with values below this threshold belongs to the remaining rock mass and is mechanically degraded. Post-blast observations of the front wall indicate that the back-break profile is influenced by vertical discontinuity sets, as the resulting fracture patterns align with their orientations.

In the toe zone, the simulation shows a damage extension of approximately 1.5 m below the blasthole bottom. This closely matches the depth of weak rock mass identified by MWD analysis, which is attributed to previous blasting operations. Since the explosive is initiated at the bottom of each blasthole, the damage propagates upward in a vertical direction. The inclination of the damage profile (approximately 50° with respect to the horizontal) is attributed to the superposition of stress waves generated by previously detonated

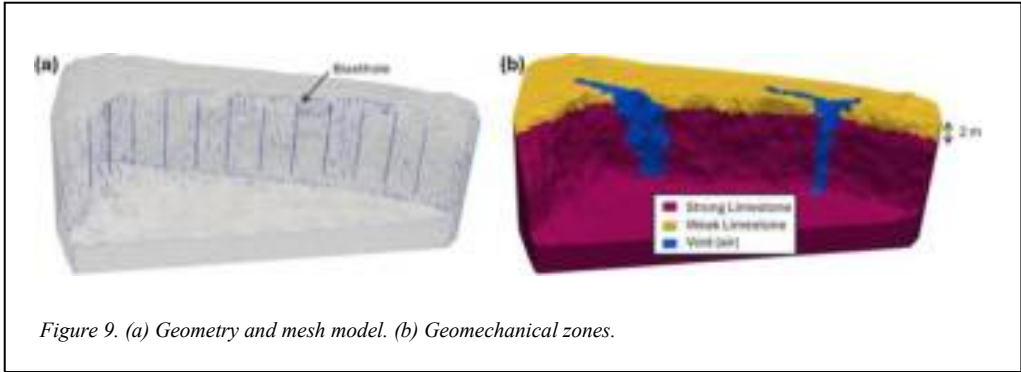


Figure 9. (a) Geometry and mesh model. (b) Geomechanical zones.

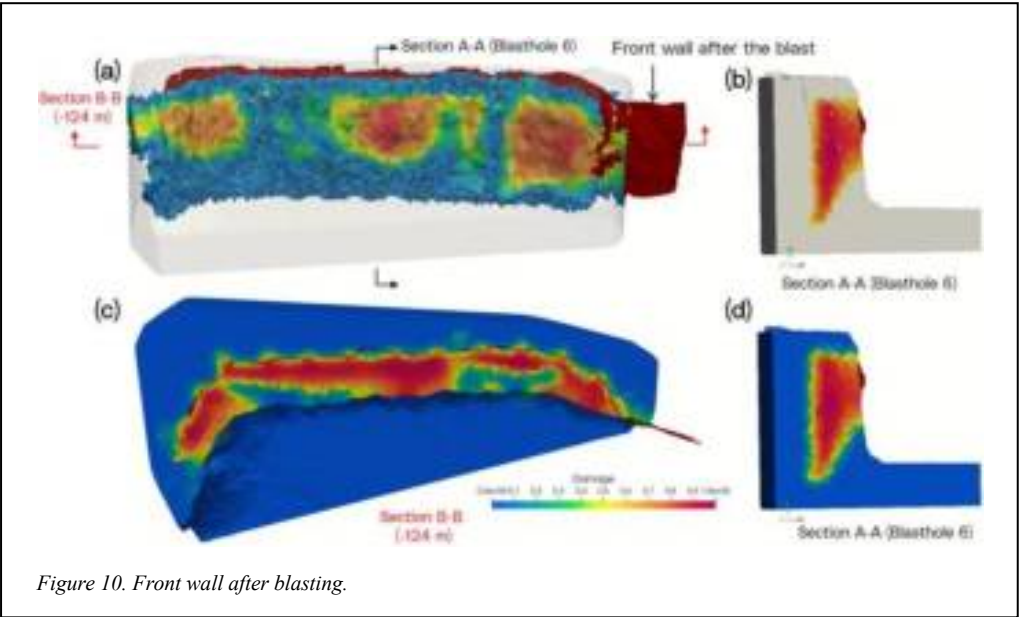


Figure 10. Front wall after blasting.

neighbouring blastholes with time delays. Numerically, this leaves an undamaged toe. Field observations confirm the absence of toe issues, as the rock mass tends to break along pre-existing horizontal or sub-horizontal discontinuities. Both this zone and the undamaged upper portion of the rock front are not initially fractured by the blast; instead, the initial fragmentation in these areas is governed by the presence of natural discontinuities within the rock mass.

It is important to note that the maximum damage reached a value of 0.98, with the highest concentration of damaged area located in the central portion of the charge line, where the material is considered continuous. On the lateral sides, the damage level is lower due to energy dissipation in areas with open discontinuities Figure 10.

With respect to fragmentation, the proposed methodology uses the damage predicted by the elasto-damage model to calculate a damage density, which depends on both the damage value and the volume of each finite element. This damage density is then correlated with the resulting fragmentation size after blasting. Fragmentation size is determined through machine vision analysis, as shown in Figures 11a & b, and Figure 11c presents the final calibrated results and demonstrates that the predicted and measured fragmentation curves are nearly identical.

5 IMPROVING ROCK FRAGMENTATION

5.1 Optimisation directions

Given the geological context of the site, a critical

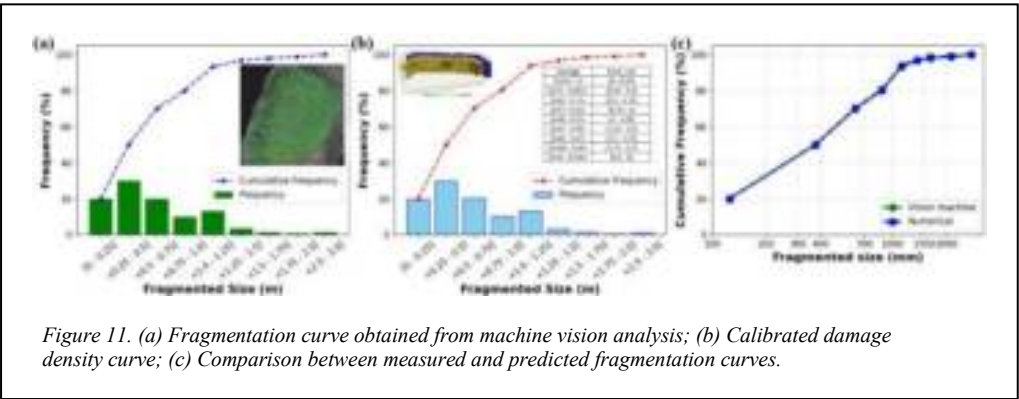


Figure 11. (a) Fragmentation curve obtained from machine vision analysis; (b) Calibrated damage density curve; (c) Comparison between measured and predicted fragmentation curves.

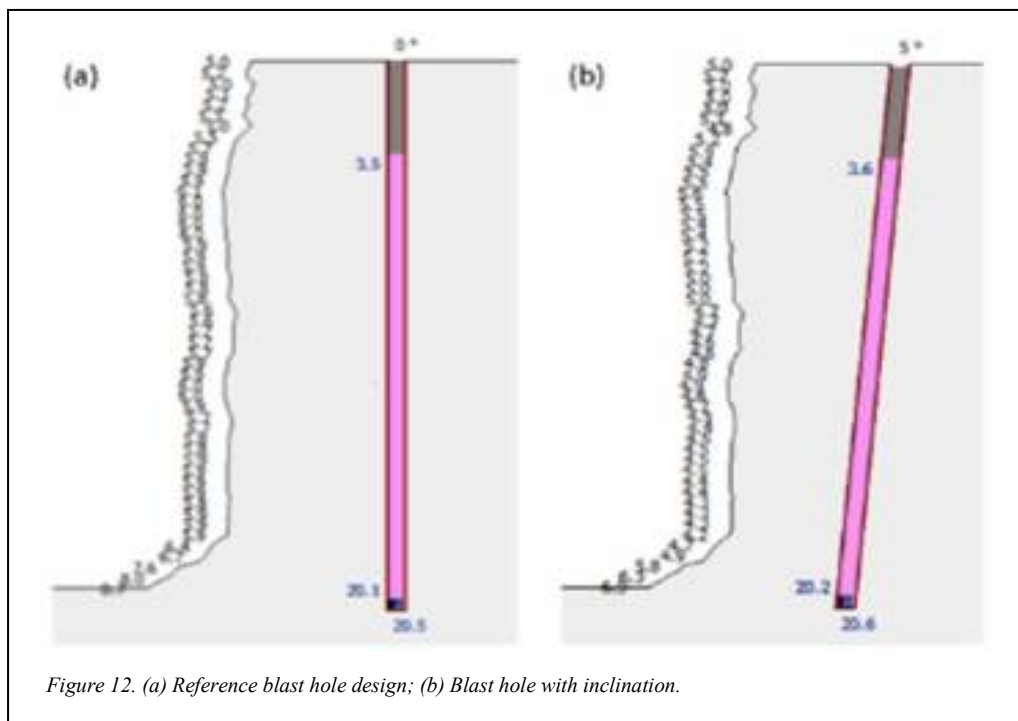


Figure 12. (a) Reference blast hole design; (b) Blast hole with inclination.

reassessment of key blast design parameters was done to optimise explosive energy use and enhance fragmentation outcomes. The parameters examined include burden and spacing - which may need to be reduced to account for lower rock confinement and limit energy dispersion - along with the distribution of explosives between column and bottom charges, the timing and sequencing of delays, and the initiation pattern, which must be adapted to the complexity of the rock mass.

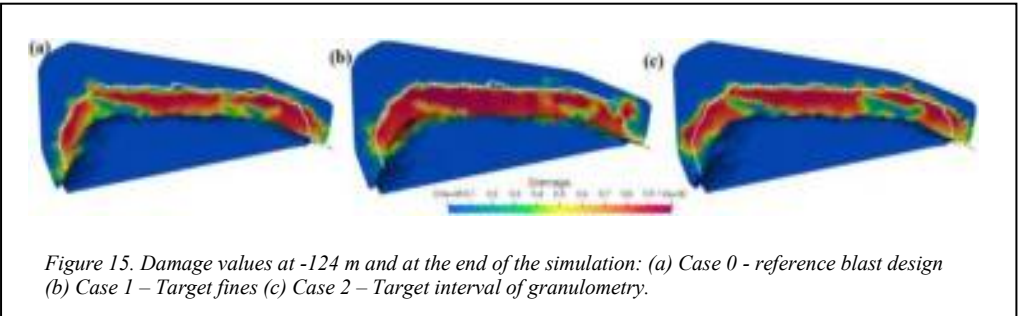
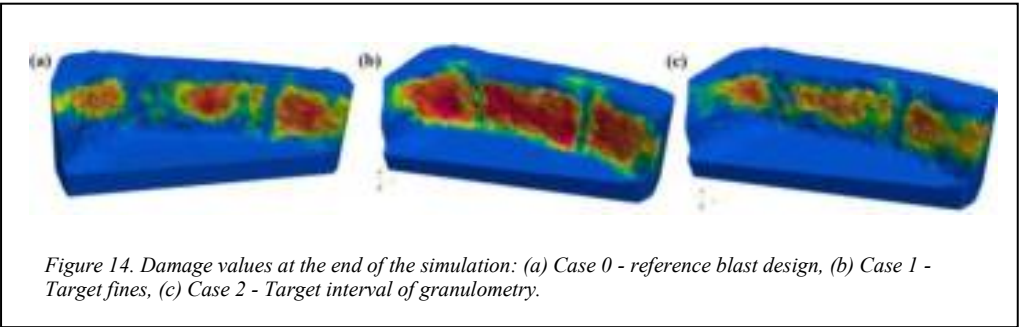
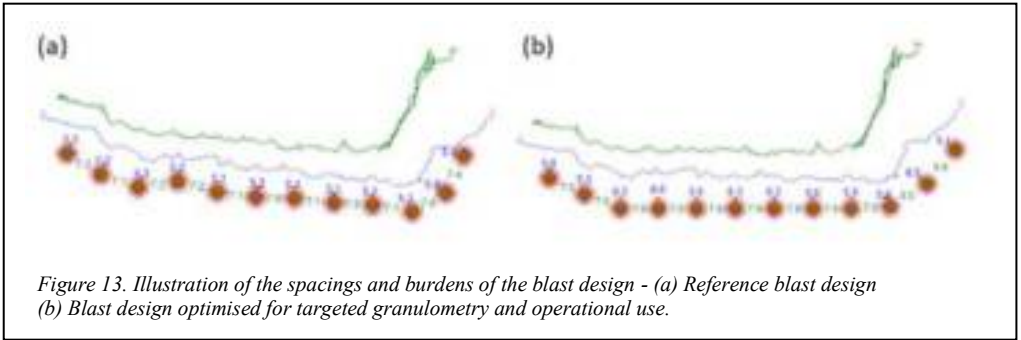
To systematically evaluate the influence of each variable, the study began with a reference blast configuration under controlled conditions: dry and intact blast holes using ANFO as the sole explosive. This baseline was selected to isolate the effects of design parameters from external factors such as water presence. The analysis focused exclusively on fragmentation results, with specific attention to variations in delay timing, blast geometry, hole inclination, and energy distribution. Other outcomes - such as safety, flyrock projections, or environmental impacts - were not considered and fall outside the scope of this study. Among the parameters tested, the use of uniform and consistent configurations emerged as a key factor in achieving controlled fragmentation. These findings are further detailed in the following design scenarios: Case 1 and Case 2.

5.2 Case 1: Target finer granulometry

The first design scenario involves a review of geometrical parameters, as well as the introduction of variable hole inclination to better match the energy requirements and achieve a more even distribution of explosive energy (Figure 12).

5.3 Case 2: Target quarry-need granulometry

In Case 2, the blasting operation was specifically designed to achieve a target granulometry range defined by the operational requirements of the quarry, namely within the interval of 320 mm to 600 mm. To meet this objective, particular attention was given to the precise design and placement of the explosive charges. This involved ensuring a more uniform spacing between the charges and adapting the charge distribution to account for the natural heterogeneity of the rock mass. A key strategy consisted of identifying geological discontinuities, as shown in Figure 9b, and avoiding charge placement directly within these zones or adjusting their positioning accordingly. In parallel, the design was optimised to remain practical and easy to implement in the field, ensuring that the proposed layout could be reliably executed by the blasting team without requiring overly complex adjustments on-site.



5.4 Damage results and fragmentation curves

Numerical damage results (Figure 14) show that the extent of the damage zone using the blast load described in Case 1 is greater than in both the reference Case 0 and Case 2. In Case 1, the damage appears to propagate further into the rock mass located in the front wall, with the damaged volume increasing from 6,754 m³ to 10,161 m³, representing a 50.4 % relative increase. In Case 2, the damaged volume is 8,070 m³, corresponding to a 19.5% increase compared to Case 0. Regarding the estimated damage values (Figure 15), the highest values are also observed in Case 1. A maximum damage value of 0.99 is obtained in Case 1, 0.97 in Case 2, and 0.98 in the reference case. These simulations also induce a similar damage depth beneath the blasthole,

approximately 1.5 m, at the toe. The back break profile likewise appears to be comparable across the cases. The damage density calculated from Cases 1 and 2 was then used to predict the resulting fragmentation size distribution expected under those conditions.

As shown in Figure 16, the fragmentation curves reflect these differences. Statistical analysis reveals that, in the fines-targeted blast, 90% of the material falls below 320 mm in size. Regarding the target granulometry specified by the customer, 26.5% of the material lies within the expected size range - this is the highest proportion among the three cases, compared to 24% for the reference blast and only 9% for the fines blast. These results demonstrate that achieving the desired fragmentation is possible, even in complex rock masses. Proper geomechanical Characterisation,

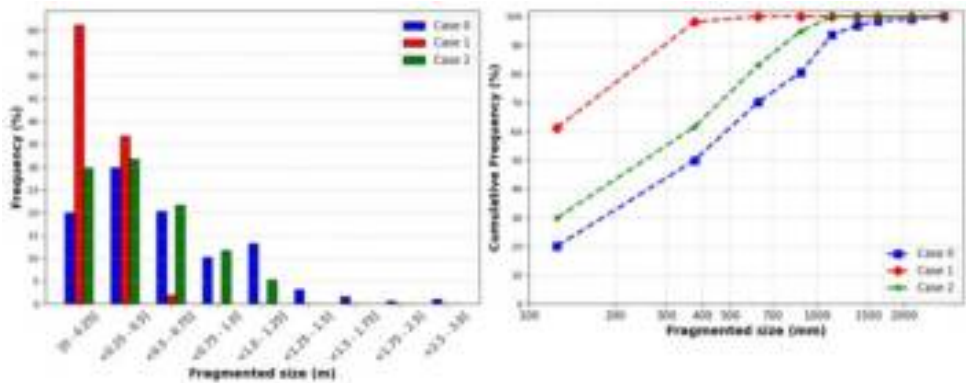


Figure 16. Comparison between fragmentation curves.

combined with a blast design adapted to the rock mass conditions, allows for optimised use of explosive energy and improved control of damage and the resulting fragmentation distribution.

6 CONCLUSION

This study presents a comprehensive simulation framework for the design and optimisation of blast operations in complex geological settings. By integrating high-resolution geological data, an elasto-damage numerical model, and quantitative field measurements, the framework demonstrates its ability to capture the multi-scale physical processes involved in explosive rock breakage from shock wave propagation to post-blast damage and fragmentation.

The application of this digital workflow to a real-world case study highlights several key outcomes:

- The incorporation of site-specific geomechanical parameters and open discontinuity mapping significantly improves the prediction of damage zones and fragmentation profiles.
- The use of a physically based elasto-damage model enables accurate spatial localisation of blast-induced fractures and provides a robust proxy for post-blast rock integrity.
- The correlation between simulation results and machine vision-based fragmentation measurements supports reliable calibration, reinforcing the predictive power of the approach.
- Moreover, the implementation of parametric optimisation strategies - adjusting burden, spacing, hole inclination, and delay timing -

demonstrates the feasibility of tailoring blast designs to target specific fragmentation objectives, whether fine granulometry or a customer-defined size range.

These results confirm the potential of digital twin approaches in blasting, where simulations are continuously refined through field data and used to drive proactive design choices. The proposed methodology not only enhances blast efficiency and safety but also lays the groundwork for more sustainable resource extraction practices by minimising energy waste and improving ore recovery.

Future work will focus on extending the framework's predictive capabilities by integrating real-time sensor feedback, drone-based fragmentation analytics, and reinforcement learning models to enable fully adaptive, autonomous blast design systems. Furthermore, expanding the application of this approach to underground mining, tunnelling, and other geotechnical domains could further enhance its relevance across the extractive industries.

REFERENCES

- Celada, B., Galera, J.M., Munoz, C. and Tardaguila, I. 2009. The use of the specific drilling energy for rock mass characterisation and tbm driving during tunnel construction. *ITA-AITES World Tunnel Congress*.
- Dai, Z. & Lu, C. 2022. Intelligent optimisation of blasting parameters in open-pit mines using hybrid machine learning and metaheuristic algorithms. *Engineering Geology*, 302, 106631. <https://doi.org/10.1016/j.enggeo.2021.106631>.

- Ghasemi, E. & Monjezi, M. 2013. A new approach to predict blast-induced ground vibration based on gene expression programming. *International Journal of Rock Mechanics and Mining Sciences*, 60, 75–81.
<https://doi.org/10.1016/j.ijrmms.2012.12.016>.
- Onederra, I., Furtney, J. & Jang, H. 2015. The role of rock mass characteristics in modelling blast-induced fragmentation. *Rock Mechanics and Rock Engineering*, 48(5), 1905–1921.
<https://doi.org/10.1007/s00603-014-0682-2>.
- Piergiorgio, R., Tutluoglu, L. & Ulusay, R. 2020. Advances in 3D geological modelling for rock mechanics and blasting simulation. *Journal of Geotechnical and Geoenvironmental Engineering*, 146(6), 04020043.
[https://doi.org/10.1061/\(ASCE\)GT.1943-5606.0002273](https://doi.org/10.1061/(ASCE)GT.1943-5606.0002273).
- Sanchidrián, J.A., Segarra, P., López, L.M. & Toraño, J. 2019. A review of empirical fragmentation models and their application in mine-to-mill optimisation. *Mining Technology*, 128(1), 1–14.
<https://doi.org/10.1080/25726668.2018.1552896>.
- Taoum, S. & Coarita, E. 2025. Prediction of open-pit rock fragmentation using a damage mechanical model, *14th International Symposium on Rock Fragmentation by Blasting (FRAGBLAST14)*, Türkiye, 359 – 367.
- Teale, R. 1965. The concept of specific energy in rock drilling. *International Journal of Rock Mechanics and Mining Sciences Geomechanics Abstracts*, 2(1):57–73.
- Tsiampousi, A., Augarde, C., Taborda, D.M.G. & Potts, D.M. 2017. A finite element framework for modelling large deformation problems in geotechnical engineering. *Computers and Geotechnics*, 82, 1–12.
<https://doi.org/10.1016/j.compgeo.2016.09.012>
- Zhou, Y., Yu, Z., He, C. & Li, C. 2020. Finite element modelling of blast-induced stress wave propagation in jointed rock masses. *Tunnelling and Underground Space Technology*, 104, 103514.
<https://doi.org/10.1016/j.tust.2020.103514>

RETURN TO CONTENTS PAGE

Impedance-based energy optimisation for improved fragmentation and NOx management

J. Navarro, L. Martínez, P. Vergara, P. Couceiro, F. Tapia & J. Blázquez

MAXAM

P. Ravanal, C. Astorga, J. Madrid & M. Galleguillos

Minera Escondida Ltda, BHP

ABSTRACT: Efficient energy transmission is critical for effective rock fragmentation in open-pit blasting. However, in zones with highly fractured or soft rock, mismatches in acoustic impedance between the explosive and the rock can lead to significant energy loss. These mismatches reduce fragmentation efficiency and, when energy input is increased to compensate, can trigger unintended side effects such as NOx fume generation. This paper presents a novel methodology that integrates the Fragment Size Energy Fan model with acoustic impedance theory to dynamically optimise explosive energy delivery. A redefined impedance formulation captures volumetric energy transfer and enables the calculation of a Rock Impedance Factor (RIF), calibrated through non-linear regression. To operate the model, rock impedance is estimated using the X-4Traits Model, a meta-learning algorithm that interprets Measurement While Drilling (MWD) data from both autonomous and manually operated drills to produce high-resolution, spatially distributed impedance maps. The methodology was calibrated and applied at Minera Escondida Limitada (MEL), using data from over 218,447 blastholes and 526 blasts. Two case studies illustrate its application: Expansion 1 demonstrates improved fragmentation consistency and a 10.7% reduction in powder factor by aligning explosive energy with local impedance variations; Expansion 2 highlights the model's utility in managing NOx emissions in fractured zones by optimising energy distribution and confinement without compromising fragmentation. This impedance-based framework provides a robust tool for improving fragmentation outcomes, optimising energy use, and enhancing control in geologically variable blasting environments.

1 INTRODUCTION

Blasting efficiency in open-pit mining relies on the effective transfer of explosive energy into the rock mass. In zones of highly fractured or very soft rock, mismatches in acoustic impedance between the explosive and the rock reduce energy coupling, leading to fragmentation issues such as excessive fines, oversize blocks, and uneven muckpile profiles. Although increased energy inputs are

often used to improve breakage, their effectiveness depends on how well the explosive energy aligns with the rock's mechanical properties. Without proper impedance matching, added energy may worsen results or trigger undesired effects such as NOx fume generation.

This study introduces a novel framework that quantifies the relationship between explosive energy and rock impedance by integrating field-derived rock mass characterisation with

fragmentation theory. A new mathematical formulation is proposed, combining the Fragment Size Energy Fan model (Sanchidrián *et al.* 2024) with acoustic impedance-based transmission theory. This integration enables a more realistic estimation of energy propagation and reflection at the explosive - rock interface, accounting for local geological variability.

To operate the model, rock impedance is estimated using the X-4Traits, which processes Measurement While Drilling (MWD) data from autonomous and manual drills via a meta-learning algorithm. This produces high-resolution maps of impedance variations, allowing for selective and optimised explosive loading.

The methodology was calibrated and implemented at Minera Escondida Limitada (MEL) using data from over 218,000 blastholes and 526 blasts. Fragmentation data was obtained through Split image analysis at the shovel face and drone-based muckpile assessments.

This paper presents two case studies that demonstrate the practical application of the model. In Expansion 1, the focus was on improving fragmentation performance across geologically variable zones by aligning explosive input with spatial impedance variations. In Expansion 2, the model was applied in a faulted and fractured rock mass to enhance fragmentation while also identifying conditions that led to increased NOx fume generation. In both cases, the impedance-based approach enabled more efficient energy distribution, resulting in improved fragmentation outcomes and better control over adverse blast effects in complex geological settings.

2 DATA OVERVIEW

MEL, a leading global producer of copper cathodes and concentrates, operates in a porphyry copper system enriched with molybdenum (Mo) and gold (Au). The lithology comprises feldspathic porphyry hosted in volcanic rocks, with dominant units including diorite, granodiorite, quartz porphyry, and andesite.

Rock excavation is performed through bench blasting with straight, multi-row patterns and 16.5 m deep blastholes. Six fully mechanised rotary rigs, four PV-351s and two PV-271s were used in the studied campaigns. Drilling was conducted in both manual and autonomous modes, with a preference for the latter to reduce human variability. Drill bits measured 270 mm and 311 mm in diameter.

Two complementary databases captured operational drilling data. The ‘MWD log’ recorded high-resolution (0.3 m) drilling parameters, including penetration rate (PR), feed force (FF), rotation force (RF), rotation speed (RS), flushing flow rate, and hole depth. The ‘MWD QA/QC’ database summarised average values per hole and included metadata such as drill mode, operator ID, drill bit specs, GPS coordinates, and hole type.

Table 1 summarises key data across 526 blasts and 218,447 blastholes, including design parameters, fragmentation measurements, and geological model information.

Operational performance data were gathered from twenty front-shovel excavators (see Table 1). The dispatch system tracked GPS coordinates (X, Y, Z), digging rates, and timestamps for each

Table 1. Data information.

Data set			Drilling		Blasting	Fragmentation	Terrain data
Expansion	Levels	No. Blast	No. of holes	Drilling mode ¹	No. of holes	No. Shovels	Block model
1	L1-L18	140	76,920	(M-45,383; A-31,537)	84,612	6	UCS RQD Lithological Mineralisation
2	L1-L14	234	92,068	(M-26,700; A-65,368)	121,601	5	
3	L1-L8	47	17,263	(M-9,150; A-8,113)	21,579	3	
4	L1-L5	17	3,397	(M-2,411; A-986)	4,076	1	
5	L1-L10	51	19,633	(M-9,620; A-10,013)	22,774	3	
6	L1-L7	37	9,196	(M-5,610; A-3,586)	11,771	2	
Total	62	526	218,447		266,413	20	

¹M - manual drilling; A - Automatic drilling; ² Data collected at every 5 to 10 min since 01/01/2023. ³ Data collected at every 15min since 01/01/2023.

Table 2. Average blast characteristics.

Type of blast	Burden (m/ft)	Spacing (m/ft)	Hole length (m/ft)	Stemming length (m/ft)	Volume (m ³ /ft ³)	Explosive Product*	Explosive (kg/lb)	Powder factor (g/t – lb/ft ³)
Production	9/29.5	11/39.4	16.5/541.3	5.5/18	1782 / 62930.7	RIOFLEX 7000	787/1735	258/30.3
Buffer	7/23	8/26,3	15/49.2	6/19.7	840 / 29664.3	RIOFLEX 7000	567/1250	

bucket interacting with the muckpile. Fourteen shovels were equipped with the Split system, which captured and analysed images during digging, rotation, and discharge to determine particle size distributions and passing percentages. These data were georeferenced to calculate hourly average fragmentation. For the six shovels without Split, daily drone flights analysed internal muckpile layers, ensuring full coverage. This combined approach enabled accurate fragmentation metrics across the entire fleet.

Explosive loading was carried out using MEMUs equipped with in-situ mechanical sensitisation and fully digital control. The explosive used was RIOFLEX 7000, a bulk watrergel consisting of 70% matrix and 30% ANFO, loaded at variable densities based on local rock properties. Table 2 presents the average blasting parameters used across the datasets.

To correlate blast design parameters with fragmentation outcomes, each borehole was assigned a representative value based on the average of measured variables within its influence zone, defined as half the burden and half the spacing. Figures 2 and 3 illustrate this correlation for Expansions 1 and 2 (Table 1). Boreholes are colour-coded by their corresponding X_{80} values from Split and drone analyses (Figure 1b), highlighting the spatial relationship between hole-level inputs and resulting fragmentation performance.

3 ROCK IMPEDANCE FACTOR (RIF) DEFINITION FOR OPTIMAL EXPLOSIVE ENERGY TRANSMISSION AND ROCK FRAGMENTATION

In recent years, various models have been developed to improve the accuracy of rock fragmentation prediction. Key contributions include those by Cunningham (1987, 2005), Djordjevic (1999), Chung and Katsabanis (2000), Kanchibotla (1999), and Sanchidrián and Ouchterlony (2017). This study adopts the Fragment Size Energy Fan model proposed by Sanchidrián and Ouchterlony (2023), which

provides a comprehensive formulation for estimating the entire sieving curve based on input energy variables, such as specific energy or powder factor. The model integrates parameters related to explosive energy application and resulting fragmentation distribution, enabling the estimation of particle size distributions and fragmentation efficiency across a wide range of blasting conditions:

$$\frac{x_p}{L_c} = x_0^* \left[\frac{E_0^* \left(\frac{L_c}{J_s} \right)^2}{E_0^*} \right]^{-\alpha(P)} \quad (1)$$

Where, $x_p = \frac{x_p}{L_c}$; x_p is particle size at the selected passing P ; L_c is a characteristic length of the blast initially estimated as $(\sqrt{HS})$, H is bench height and S is spacing between holes. For this study, L_c is estimated as $(\sqrt{TS})$ since stemming length T is a key variable influencing energy confinement in this operation; J_s is the joint spacing factor, calculated as $\min(S_j, B)$; S_j is joint spacing; B is burden; J_o the joint orientation factor; x_0^* is the new focus ordinate; $E = q \cdot e$, is energy powder factor (MPa); q is powder factor (kg/m³); e is explosive energy (MJ/kg); σ is rock mass strength $\sigma = \frac{\sigma_c}{2Y}$; σ_c is UCS (MPa); Y is Young's modulus (GPa); E_0^* is a reference energy constant. The time function f_{c2} is defined as:

$$f_{c2} = [\delta_1 + (1 - \delta_1 - \delta_2 \Pi_t) e^{-\delta_2 \Pi_t}]^{-1} \quad (2)$$

And the passing function $\alpha(P)$ is expressed as:

$$\alpha(P) = \alpha_{100} + (\alpha_p - \alpha_{100}) \left[\left(\frac{100}{P} \right)^{\beta} - 1 \right]^{1/\omega} \quad (3)$$

Where, $\Pi_t = \frac{c_p \Delta t}{x}$; c_p is P-wave velocity; Δt is in-row delay. The constants for the 13-parameter fit used in this study (Sanchidrián & Ouchterlony 2023) are: $\xi = 0.4182$, $\omega = 0.5021$, $x_0^* = 4281$, $E_0^* = 6.54E-07$, $\lambda = 1.527$, $\delta_1 = 0.7575$, $\delta_2 = 0.0342$, $\delta_3 = 0.0353$, $\beta = 0.5874$,

$$\alpha_{100} = 0.6067, \quad \alpha_p = 0.8405, \quad b = 1.6078, \\ \mu = 0.2775.$$

Equation 1 is reorganised into Equation 4 to consolidate all rock-related parameters. those intrinsic to the rock and not modifiable by blast design - on the left-hand side. This yields the Rock Factor (RF), which integrates joint spacing (J_s), joint orientation (J_o), and rock strength (σ) into a single term reflecting the fragmentation resistance of the rock mass:

$$1/2 J_s^2 (\sigma^2)^{0.5(P)} = RF = \frac{\alpha_p (E F_s L_s)^{0.5(P)}}{L_p \alpha_o E_o^{0.5(P)}} \quad (4)$$

As is common with automatic image-based analysis systems, Split experiences reduced accuracy when detecting fine particle sizes. To mitigate this, the X_{80} size is used as the primary indicator of fragmentation. X_{80} , the size below which 80% of material passes, provides a consistent performance metric aligned with the requirements of downstream crushing and comminution circuits. Accordingly, all references to $\alpha(P)$ in this study are related to the 80% passing size.

The efficiency of energy transfer during blasting is determined during the shock wave phase of detonation, which corresponds to the initial portion of the explosive's pressure volume (P-V) curve (Cooper 1996). According to the Lownds Energy Partition Model (Lownds 1996), only a fraction of the detonation energy contributes to rock fragmentation, while the remainder is lost as seismic, thermal, or gas expansion energy.

Explosives must be matched to rock conditions to optimise this energy transfer. High-velocity explosives like emulsions or watergels are more effective in competent rock, where most energy is consumed in shock wave propagation. Conversely, low-density explosives are better suited to fractured or soft rock, where fragmentation requires less shock energy, allowing more energy to support movement and heave.

From a wave mechanics standpoint, impedance mismatch between two media determines the proportion of energy transmitted (E_T) versus reflected (E_R) at their boundary, following the principle ($E_T + E_R = 1$). Analogously, in blasting, the explosive - rock interface behaves like the boundary between two materials with distinct acoustic impedances. The proportion of transmitted and reflected energy can be described as:

$$E_T = \frac{4z_r z_e}{(z_r + z_e)^2} ; E_R = \left(\frac{z_r - z_e}{z_r + z_e} \right)^2 \quad (5)$$

Where $z_r = \rho_r \cdot v_r$ is the acoustic impedance of the rock (ρ_r as rock density and v_r as wave velocity in the rock), and $z_e = \rho_e \cdot v_e$ is the impedance of the explosive medium, being ρ_e the explosive density and v_e the explosive velocity of detonation. The effective energy delivered to the rock becomes:

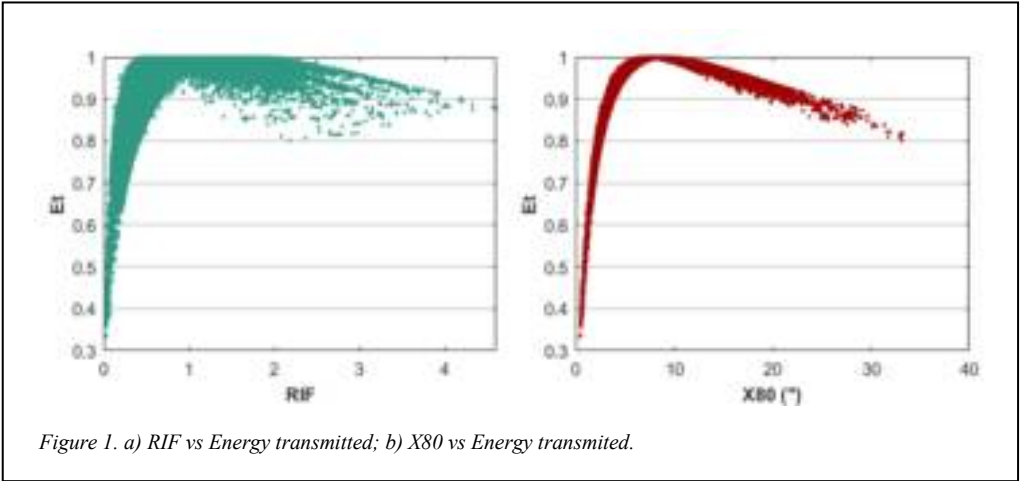
$$E_{eff} = E_T \cdot E = E_T \cdot q \cdot \theta \quad (6)$$

To adapt this model to volumetric blasting conditions, explosive impedance is redefined as $z_e = (\rho_e \cdot v_e) / V$, where V is the volume of rock to be blasted around the borehole. This redefinition makes the explosive impedance consistent with the rock-side impedance, which is also volume-based and derived from drilling data. It allows the impedance term to capture not only the shock-generating capacity of the explosive, but also the effective energy input available for fragmentation within the actual volume influenced by the blast. While the classical formulation $z_e = \rho_e \cdot v_e$ describes wave transmission across a unit area, it does not reflect the three-dimensional energy distribution relevant to breakage. By redefining impedance on a per-volume basis, the model can more accurately account for field-scale energy transfer efficiency, especially in conditions of poor coupling, such as soft rock, voids, or fractured zones, thereby improving the reliability of fragmentation predictions.

This adjustment ensures that the fragmentation model reflects the physical reality of energy transfer, particularly in zones of low coupling efficiency caused by soft rock or structural discontinuities. By incorporating the impedance-corrected energy into the Fragment Size Energy Fan model, the approach enables dynamic calibration of blast designs based on local rock behaviour. In this context, the variable z_r represents both the acoustic impedance and fragmentation resistance of the rock. Substituting equations (5) and (6) into equation (4), the fragmentation model becomes:

$$X_p = \frac{z_r L_p \alpha_o E_o^{0.5(P)}}{\left(E \frac{4z_r z_e}{(z_r + z_e)^2} F_s L_s \right)^{0.5(P)}} \quad (7)$$

To facilitate calibration, a power-law form is proposed:



$$\hat{E}_t = \frac{\sum_{i=1}^n \hat{E}_{t,i} \cdot \hat{R}_{iF} \cdot \hat{X}_{80,i}}{\left(\sum_{i=1}^n \hat{E}_{t,i} \cdot \hat{R}_{iF} \cdot \hat{X}_{80,i} + \sum_{i=1}^n \hat{E}_{t,i} \cdot \hat{R}_{iF} \cdot \hat{X}_{80,i} \right)} \quad (8)$$

A non-linear regression was implemented in MATLAB 2023b using the Levenberg–Marquardt algorithm. All estimated coefficients were statistically significant ($p < 0.0001$); however, due to internal know-how and confidentiality considerations, the specific values cannot be publicly disclosed. The model achieved an excellent fit with a coefficient of determination (R^2) of 0.99, consistent with its theoretical formulation. The calibrated term $\hat{E}_{t,i}$ is henceforth referred to as the Rock Impedance Factor (RIF), representing the adjusted ability of the rock mass to absorb and transmit explosive energy under field conditions.

Figure 1 illustrates the relationship between transmitted energy E_t and both the RIF (graph a) and X_{80} fragmentation (graph b). At low RIF and low X_{80} , energy transmission is inefficient, and fragmentation is excessively fine, indicating overcharging of soft ground. As z_e becomes better matched to the rock’s impedance, energy transmission increases and reaches a maximum,

corresponding to optimal fragmentation and energy efficiency.

Beyond this point, higher RIF and X_{80} values reflect undercharging of competent rock (transmitted energy begins to decline again), where the explosive shock is insufficient to achieve desired breakage. This reinforces the value of impedance matching, especially as the zone of maximum transmission aligns with the mine’s target X_{80} range (5–8 inches), depending on material type (sulfide, oxide, or waste).

Figures 2 and 3 in the case studies (Expansions 1 and 2) further illustrate the behaviour in 3D space, showing the interaction between RIF (Figure 2a), X_{80} (Figure 2b), explosive impedance Z_e (Figure 2c), energy powder factor E (Figure 2d), and transmitted energy E_t (Figure 2e). Together, these validate the model’s predictive capacity and its practical utility for dynamic blast design adjustment.

4 DRILL MONITORING DATA ANALYSIS: X-4TRAITS MODEL

To enhance drilling efficiency and data reliability, MEL is progressively automating its drill fleet,

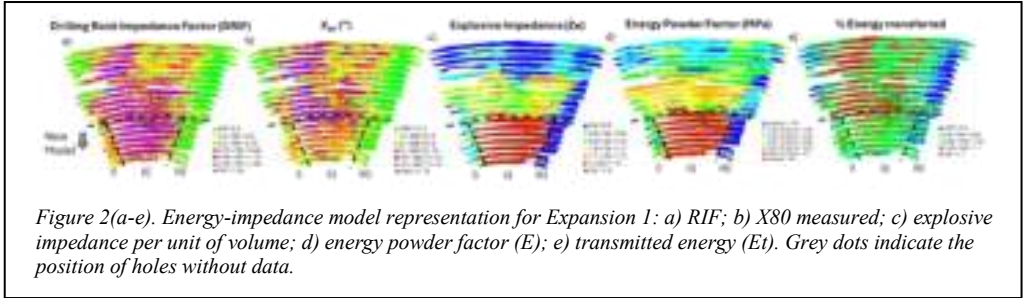


Figure 2(a-e). Energy-impedance model representation for Expansion 1: a) RIF; b) X_{80} measured; c) explosive impedance per unit of volume; d) energy powder factor (E); e) transmitted energy (E_t). Grey dots indicate the position of holes without data.

with two-thirds now operating autonomously. This shift requires rigorous maintenance to ensure calibration and operational readiness.

MWD (Measurement While Drilling) data accuracy depends not only on rock properties but also on external factors such as sensor calibration, hole depth, rig configuration, and drill performance (Schunnesson 1998, Navarro *et al.* 2021). These external factors introduce uncertainties that must be addressed to ensure accurate interpretation of rock-related parameters.

Due to recurring gaps in the MWD log database for a significant portion of blast holes, and to ensure consistency with other variables recorded per-hole, the analysis was based on average parameter values extracted from the QA/QC MWD database. This approach reduces the influence of hole depth on the data, which is not considered in the current correction model (see Navarro *et al.* 2021). Data normalisation for autonomous drilling included: (i) outlier removal, (ii) adjustment by rig setup, and (iii) correction by tricone bit type and rock hardness. Manual drilling data were further normalised by operator to reduce human variability (Navarro *et al.* 2021, 2024).

A Drilling Rock Impedance Factor (DRIF) was developed exclusively from drill monitoring data using the X-4Traits model, a machine learning framework that integrates ensemble regression and k-nearest neighbours (k-NN) regression. The approach employs a meta-model architecture that combines both algorithms with a 3D spatial function to improve prediction performance.

Key input variables - normalised penetration rate (PRNorm) and feed pressure (FPNorm) were derived using Principal Component Analysis (PCA), which reduces dimensionality while retaining the most relevant variance. The first principal component (PC1), representing the rock-drilling interaction (Schunnesson 1998, Peng *et al.* 2005, Ghosh *et al.* 2018, Navarro *et al.* 2021), was used as a primary predictor.

The ensemble regression captures generalised patterns using multiple weak learners, while the k-NN model incorporates spatial (X, Y, Z) coordinates to reflect local geological variation. Predictions from both models were combined using a linear regression meta-model, benefiting from the generalisation strength of ensembles and the spatial sensitivity of k-NN.

Model performance, evaluated using RMSE and accuracy, showed strong predictive ability. With a 70/30 training-testing split, the DRIF model achieved an RMSE of 2.13 (for values between 0 and 21) and an accuracy of 74.6%.

Figures 2 and 3 show the results for Expansions 1 and 2 (see Table 1) of the DRIF predictions from the X-4Traits meta-model (Graph a). The model predicts with high accuracy the rock traits showing the potential the X-4Traits model to have an accurate rock characterisation exclusively from drilling data.

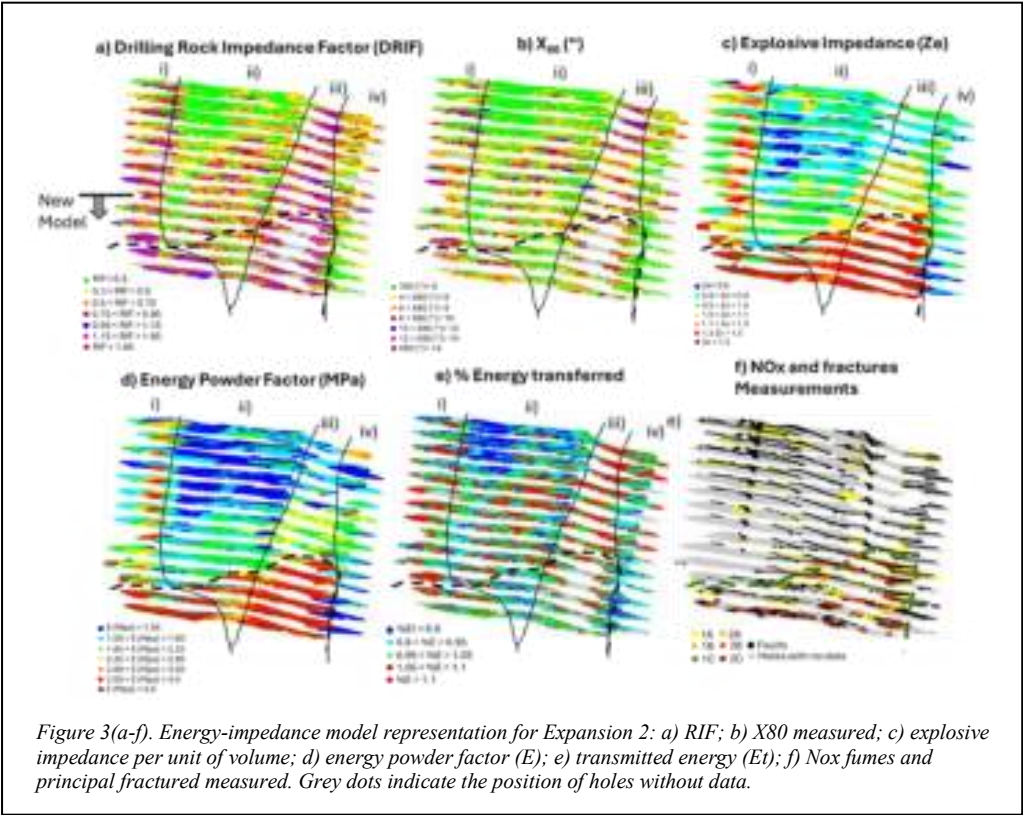
5 CASE STUDY 1: ENERGY-IMPEDANCE MATCHING TO IMPROVE FRAGMENTATION AND OPTIMISE POWDER FACTOR

The first case study applies the proposed impedance-based model in Expansion 1 (Table 1), which presented a complex geological configuration. In the central area (zone ii), where ore is concentrated, the rock exhibited high resistance to breakage, resulting in poor fragmentation. In contrast, the side zones (zones i and iii) consisted of softer material, generating excessive fines.

As shown in Figure 2, prior to model implementation (above the dashed black horizontal line), the applied explosive impedance (z_c) and energy powder factor (graphs c and d) were neither consistent nor aligned with the Drilling Rock Impedance Factor (DRIF) (graph a). This mismatch led to significant energy losses: over 20% of the energy was reflected or ineffectively absorbed in the soft zones (zones i and iii, indicated by dark blue in graph e), while the central area (zone ii) showed insufficient energy transfer (highlighted in red and brown), failing to meet the required X_{80} fragmentation targets (graph b) for both ore and waste.

This behaviour is further illustrated in the 3D visualisations of Figure 2, which show the relationships among DRIF (graph a), fragmentation (X_{80} , graph b), explosive impedance per unit volume (z_c , graph c), energy powder factor (graph d), and transmitted energy (E_t , graph e). These confirm the model's ability to predict energy transmission efficiency and guide the adjustment of explosive energy inputs under actual field conditions.

Following the implementation of DRIF as a control variable (graph a), the energy powder factor (graph d) was recalibrated across the bench to reflect local rock properties. This adjustment improved energy transmission in harder zones and reduced overcharging in softer zones, as evidenced by the shift toward more uniform green tones in graph e.



The result was a 16% reduction in coarse fragmentation within the ore zone (zone i, graph b) and a 10.7% decrease in powder factor in the waste zones (zones i and iii, graph d), demonstrating improved fragmentation uniformity and more efficient use of explosives.

6 CASE STUDY 2: MANAGING NOX FUMES AND ENERGY COUPLING IN FAULTED ROCK USING IMPEDANCE-BASED BLASTING

The second case study applies the impedance-based blasting model in Expansion 2 (Table 1, Figure 3), a geologically complex area where the rock mass becomes progressively harder with depth along both the eastern and western flanks (zones i and iii, where colors shift from red to brown in graph a). These areas are intersected by multiple major geological faults (black dots in graph f), creating challenging conditions for effective energy transmission and detonation control.

Prior to implementing the model (above the downward arrow in graph a), NOx fumes were

repeatedly reported in zones with varied energy powder factors (graph d), particularly along fractured regions. According to AEISG (2022), under ideal detonation conditions, ammonium nitrate-based explosives yield only nitrogen, water vapor, and carbon dioxide, all colourless gases. The presence of red or orange post-blast fumes indicates incomplete detonation or poor oxygen balance, typically resulting in the formation of nitric oxide (NO), which rapidly oxidises to nitrogen dioxide (NO₂) upon contact with air. Such NOx emissions typically result from fuel imbalance, desensitisation, or insufficient confinement. AEISG identifies six key contributors: (i) explosive formulation and QA/QC, (ii) geological conditions, (iii) blast design, (iv) explosive selection, (v) on-bench practices, and (vi) blast-hole contamination. Fume ratings based on the AEISG color-coded scale are shown in Figure 3, graph f.

In the eastern section (zone ii), where fault intersections were most prominent, DRIF analysis indicated high energy transmission requirements. However, the presence of fractures was not directly captured in the MWD logs due to data

gaps, as discussed earlier. As a result, the DRIF system reflected only fragmentation-related rock hardness and not structural discontinuities. Based on model recommendations, energy input (z_c and energy powder factor) was increased in these areas (highlighted by the black dashed line in graphs c and d). This adjustment led to: (i) improved energy transmission (graph e) and (ii) enhanced fragmentation (graph b), reflected in a 12.3% reduction in X_{80} .

However, this increase in powder factor also resulted in a rise in NOx fumes, particularly in the same fractured zones (see graph f, black dashed area). Four primary mechanisms were identified as potential contributors to this increase:

- Explosive Migration and Incomplete Detonation: Higher explosive density reduced gas bubble content in the matrix, making the product more sensitive to desensitisation if it migrated into fractures. Although the Rioflex product typically crosslinks in 10–20 minutes, premature migration could cause critical diameter loss, leading to incomplete detonation and elevated NOx generation.
- Shock Wave Reflection and Overpressure: Increased energy concentration, especially in areas with tightened pattern and high-density loading, may have caused shock waves to prematurely reach adjacent holes through connected fractures. This could lead to overpressure and reduced sensitivity in receiving charges, impairing detonation efficiency.
- Fracture-Induced Deconfinement: Shock wave propagation through fracture networks may have further opened the cracks, decreasing confinement and increasing the likelihood of incomplete detonation in those zones.
- Water in the Borehole: Where water was present, improper pumping practices, especially surface pumping or excessive hose retraction, may have trapped water within the explosive column, contaminating the matrix and reducing the effective critical diameter, ultimately resulting in unsteady detonation.

Additionally, where fractures intersected drill holes, local rock impedance was likely reduced. Studies (Boadu & Long 1996, Zhou *et al.* 2021, Azhari & Hassani 2013, Narváez-Campo *et al.* 2011) confirm that fracture density significantly reduces P-wave velocity, a proxy for impedance. Narváez-Campo *et al.* reported that increasing fracture frequency in core samples reduced P- and S-wave velocities by 20–30%, illustrating how

these zones can disrupt optimal energy transfer and detonation stability.

In response to the NOx increase, several corrective actions were implemented: i) a more energetic and sensitive product was used to counteract critical diameter limitations and mitigate overpressure sensitivity; ii) the new product achieved crosslinking in 10–20 seconds, reducing migration risk and minimising water contamination during pumping; iii) boreholes with identified fractures were loaded with lower density, improving energy coupling and reducing overpressure from adjacent holes; iv) timing was shortened in fracture-prone zones to promote better wave synchronisation and prevent interference from neighboring detonations. These modifications have resulted in a reduction in NOx emissions, improving both fragmentation outcomes and safety performance in this geologically sensitive zone.

7 CONCLUSIONS

This study introduces and validates a novel impedance-based framework for optimising explosive energy transmission in open-pit blasting. By integrating the Fragment Size Energy Fan model with acoustic impedance theory and calibrating it using a Rock Impedance Factor (RIF), the approach enables dynamic adjustment of explosive input according to local rock conditions. The use of the X-4Traits Model to estimate rock impedance from MWD data, leveraging meta-learning techniques, further enhances the model's applicability, enabling high-resolution mapping of mechanical rock behaviour across the bench.

Application of the model in two operational case studies demonstrated its capacity to improve fragmentation performance while minimising energy waste. In Expansion 1, aligning energy input with impedance variations led to a 16% reduction in coarse fragmentation and a 10.7% reduction in powder factor. In Expansion 2, the model enabled targeted energy increases in hard rock zones, resulting in improved breakage; however, it also revealed how energy overloading in fractured zones may trigger NOx fume emissions due to poor coupling and desensitisation mechanisms. Corrective actions based on these insights led to improved safety and performance.

Overall, the proposed methodology offers a robust, data-driven solution for blast design optimisation, enhancing both fragmentation efficiency and operational control in geologically complex environments.

8 ACKNOWLEDGEMENTS

The authors would like to thank to Minera Escondida Ltda for their valuable input sharing the data and the support provided in the daily work.

REFERENCES

- Chung, S. & Katsabanis, P. 2000. Fragmentation prediction using improved engineering formulae. *Fragblast: International Journal for Blasting and Fragmentation*, v.4 (3-4), pp. 198-207.
- Cooper, P.W. 1996. *Explosives engineering*. John Wiley & Sons.
- Cunningham, C.V.B. 1987. Fragmentation estimations and the Kuz-Ram model four years on. *Proceedings of the 2nd International Symposium on Rock Fragmentation by Blasting*, August, pp. 475-487.
- Cunningham, C.V.B. 2005. The Kuz-Ram fragmentation model 20 years on. *Proceedings of the European Federation of Explosives Engineers*, Brighton, England, pp.201-210.
- Ghosh, R., Gustafson, A. & Schunnesson, H. 2018. Development of a geo-mechanical model for chargeability assessment of borehole using drill monitoring technique. *International Journal of Rock Mechanics and Mining Sciences*, vol. 109, pp. 9-18.
- Lownds, C.M.A. 1996. A study of energy distribution and utilisation in bench blasting (*PhD thesis*). *University of Queensland*, Australia.
- Navarro, J., Seidl, T., Hartlieb, P., Sanchidrián, J.A., Segarra P., Couceiro, P., Schimek, P. & Godoy, C. 2021. Blastability and ore grade assessment from drill monitoring for open pit applications. *Rock Mechanics and Rock Engineering*, vol. 54, pp. 3209–3228.
- Navarro, J., Vergara, P., Couceiro, P., Blázquez, J., Tapia F., Ovalle, A. & Galleguillos, M. 2023. On the use of selective blast designs based on MWD technique to optimise downstream mining operations. A case study. *2024. 50th Conference ISEE*, Savannah, Georgia. vol. 54, pp. 3209–3228.
- Sanchidrián, J. & Ouchterlony, F. 2017. A distribution-free description of fragmentation by blasting based on dimensional analysis. *Rock Mechanics and Rock Engineering*, v.50 (4), pp. 781-806.
- Sanchidrián, J.A., Ouchterlony, F. 2023. Blast-fragmentation prediction derived from the fragment size-energy Fan Concept. *Rock Mech Rock Eng* 56, 8869–8889 (2023).
- Schunnesson, H. 1998. Rock characterisation using percussive drilling. *Int Journal of Rock Mechanics and Mining Sciences*, vol. 35, pp. 711-725.
- Peng, S.S., Tang, D., Sasaoka, T., Luo, Y., Finfinger, G. & Wilson, G. 2005. A method for quantitative void/fracture detection and estimation of rock strength for underground mine roof. *Proceedings of the 24th International Conference on Ground Control in Mining*, pp. 187-195.

[RETURN TO CONTENTS PAGE](#)

Techniques for quarrying dimension stones through the controlled use of explosives

M. Cardu & C. Todaro

DIATI-Politecnico, Torino, Italy

J. Seccatore

Universidad Catolica del Norte, Antofagasta, Chile

ABSTRACT: Explosives are a traditional, cost-effective method for extracting hard stones but are very seldom adopted in marble quarrying due to low efficiency and rock damage. ‘Splitting’ applies controlled blasting and precision drilling to detach large stone blocks with minimal fractures. This is achieved through precisely aligned drill holes, simultaneous detonation, and reduced charge density to guide fractures according to the wanted geometry. The technique requires careful execution, especially in anisotropic rock, where precise charge placement is crucial to maintaining block integrity. The paper reviews various implementation strategies in Italian quarries, examining geometric configurations, rock properties, and calculation methods to optimise efficiency and precision. It addresses challenges posed by different rock types and quarry layouts, offering practical solutions to improve splitting operations.

1 SPLITTING WITH EXPLOSIVES

A single fracture is initiated to isolate a specific rock volume while minimising additional damage by reducing impulsive forces and quasi-static pressure (Zhang & Zhao 2014). The most common approach involves *decoupling* the explosive charge from the borehole wall by using smaller-diameter cylindrical charges, thereby lowering the loading density.

Precisely aligned boreholes are drilled along the intended fracture plane and detonated simultaneously to achieve controlled breakage. In this setup, the explosive functions primarily as a pressurising agent rather than a pulverising one, eliminating the crushed zone typically formed around the holes.

This makes accurate fracture guidance particularly important, especially in anisotropic rock masses.

The mechanism of the charge action, illustrated in Figure 1, considers a row of charged boreholes and evaluates the failure conditions using a static model. Key parameters include explosion pressure, loading density, hole spacing, and the tensile strength of the rock. The fundamental condition for failure is given by $F = R$, where:

$$p \cdot \Phi = \sigma_t \cdot E \quad (1)$$

The most critical parameter in the design is the ratio between hole spacing and hole diameter. Accordingly, Equation [1] can be reformulated as:

$$p/\sigma_r > E/\Phi \quad (2)$$

The ratio p/σ_r can be considered constant for a given combination of explosive type, decoupling configuration, and rock properties. Specific consumption (or powder factor) is not well-suited for blast design, as it emphasises force balance rather than energy balance, focusing on active versus resisting forces. Instead of volume-based metrics, surface parameters are more relevant: the *active surface* (hole diameter $\times$ charge length) and the *resisting surface* (hole spacing $\times$ hole length) play a more critical role in blast performance.

Successful outcomes depend on the correct ratio between these surface areas, rather than the explosive-to-rock volume ratio.

Both hole diameter and spacing influence the effectiveness and cost of the blast. Specific consumption (expressed in g/m^2) generally decreases as the hole diameter is reduced, contributing to more efficient use of explosives and potentially lower operational costs.

The annular gap between the explosive charge and the borehole wall can be filled with various materials, such as air, water, or sand, depending on the desired effect and site conditions. Air results in a less confined charge and, therefore, requires a greater amount of explosive to achieve the same effect, whereas water typically offers the most practical and effective confinement solution in most blasting scenarios.

To ensure optimal blasting performance, all boreholes should be detonated simultaneously or with minimal delay; significant delays between detonations can reduce overall blast efficiency and control.

1.1 Fracture guidance

As is well known, when an explosive charge is placed far from a free surface, it tends to generate random radial fractures in the surrounding rock. By reducing the detonation pressure, for example through decreased loading density, this effect can be limited, resulting primarily in minor and temporary expansion of the borehole wall, as illustrated in Figures 2A and 2B. Detonating multiple, widely spaced coplanar holes produces a similar outcome, with scattered fracture patterns. However, as the spacing between the holes is reduced, the induced fractures begin to align between adjacent holes, leading to the formation of a more regular and continuous breakage surface, accompanied by some irregular cracking in the surrounding rock (Figure 2C). With further reduction in explosive pressure, the system can

produce a single, well-guided fracture that propagates cleanly along the row of holes, as shown in Figure 2D.

1.2 Conditions for the guidance

Under quasi-static loading conditions, which are favoured by low charge intensity and high levels of decoupling, the following design criteria must be met to ensure controlled fracturing:

- The distance from each borehole to the free surface must be greater than the spacing between holes to avoid the formation of unwanted cracks directed toward the free wall.
- Single-hole detonations should not generate significant tensile stresses, as excessive tensile loading can result in uncontrolled or premature fracturing (Liu *et al.* 2023).
- The stress fields produced by adjacent boreholes must interact constructively to induce failure specifically along the intended fracture alignment plane.

To satisfy these conditions, the simplified ‘static’ fracture model typically requires a decoupling ratio of at least 2 and a hole diameter-to-spacing ratio ranging between 1:6 and 1:12. In anisotropic rock masses, the effectiveness of fracture guidance is highly dependent on orientation: guidance tends to weaken when fractures must propagate across planes of schistosity but improves significantly when aligned with them.

1.3 The splitting technique

Two main fracture separation methods can be identified:

- *Presplitting for Blasting* is used to isolate rock volumes while preserving the integrity of the surrounding rock mass. It is commonly applied in civil and mining engineering projects.
- *Block Extraction* aims to isolate intact rock blocks for subsequent cutting and is typically employed in ornamental stone quarries (Majeed *et al.* 2019).

In both methods, charge sizing is governed by static principles, with key design parameters including the linear charge, drill hole diameter, and the spacing-to-diameter ratio. In contrast, burden and specific explosive consumption are generally less critical. Explosives are typically highly decoupled, and detonations are carried out simultaneously or with minimal delay to ensure effective fracture control.

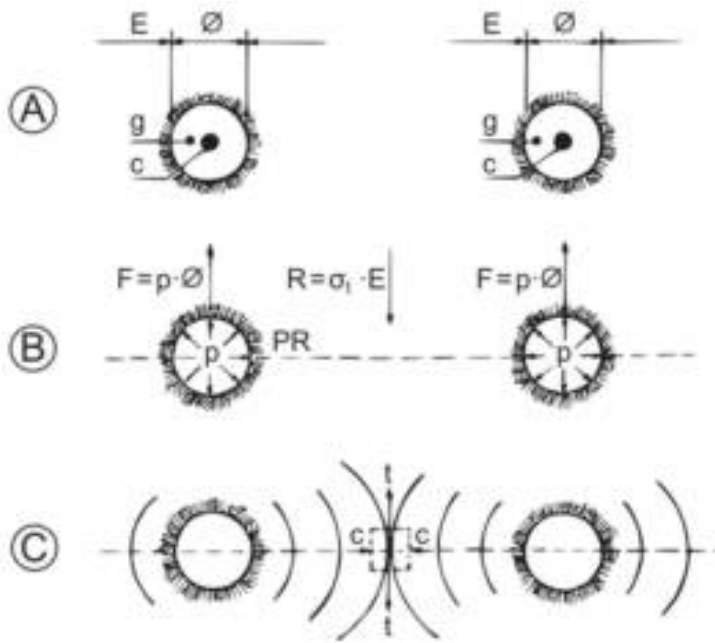


Figure 1. Splitting mechanism with parallel coplanar holes. A: Loaded holes for simultaneous blasting; Φ : hole diameter; E : spacing; g : gap (air, sand, or water). B: Explosion under static detachment theory. p : hole pressure; F : force; R : resistance; PR : break plane; σ_t : tensile strength based on load rate/duration. C: Alternative theory: fracture from indirect tensile stress (t) due to compressive shock front collisions (c) (Cardu et al. 2024).

Drill hole diameters are usually small. However, for larger boreholes (greater than 100 mm), standard cartridges, detonating cords, or diluted explosives are often employed. Presplitting designs are typically based on manufacturer recommendations (see Table 1) or empirical equations, such as the one proposed by Del Greco et al. (1983):

$$\left(\frac{\Phi_c}{\Phi_f}\right)^2 \cdot \frac{L_c}{L} \cdot \frac{\delta_c}{1000} \cdot p_s \cdot \left(\frac{\Phi_f}{E - \Phi_f}\right) = T \quad (3)$$

Where: Φ_c is the diameter of the cartridges (m), Φ_f is the diameter of the holes (m), δ_c is the density of the cartridges (kg/m³), p_s is the detonation pressure of the explosive (MPa), E is the spacing (m), L_c is the charged length (m), L is the hole length (m), and T is the tensile strength of the rock (MPa).

A detonation pressure of 20 MPa can be assumed to provide the wanted effect.

Different values for the detonation pressure are provided for different explosives, and the following can be assumed: 1200-1400 MPa (straight dynamites), 1000 MPa (ordinary dynamites), 900 MPa (nitrate/water gels), and 800 MPa (ANFO).

PETN cords with air, water, or sand stemming are typical in dimension stone extraction. Holes are closer and lightly charged. Conditions are uniform, so trial and error optimises sizing. For granite, using small holes (30 - 40 mm) and PETN cord in water, this formula applies:

$$P.F. = a + b \cdot \frac{S}{V} + c \cdot s \quad (4)$$

Where P.F. is the specific consumption of PETN (g/m³), S is the separation surface to be achieved (m²) - excluding natural discontinuities, as they are naturally exploited, if possible, in the blast design to facilitate the block separation, V is the block volume (m³), s is the desired block displacement (m), usually a few decimetres; a , b , c are empirical constants for granite, based on

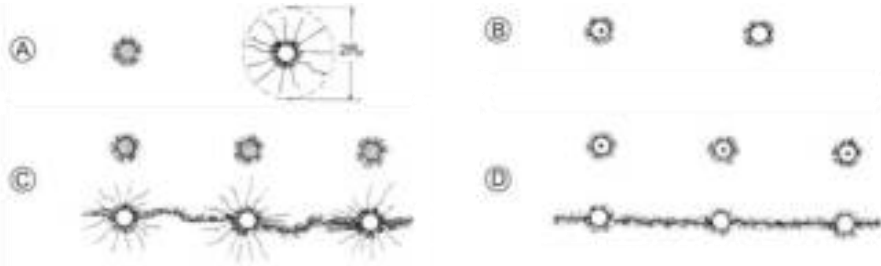


Figure 2. Splitting effects with coupled (A, C) and decoupled (B, D) charges. A: Coupled charges generate radial fractures up to RF, without impacting distant walls/holes. B: Decoupled charges under similar conditions cause no fracturing. C: Closely spaced coupled charges create continuous fractures via overlap. D: Aligned decoupled charges form a single splitting fracture (Cardu et al. 2024).

several case studies ($a = 10.5 \text{ g/m}^3$, $b = 26.5 \text{ g/m}^2$, $c = 28.7 \text{ g/m}^4$).

Once S , V , and s are known, compute P.F., then total PETN ($g = \text{P.F.} \times V$). Choose a cord charge (10 - 20 g/m), then determine the total length. Divide it by the number of strands per hole to determine the total drill length. Finally, divide S by the drill length to obtain hole spacing.

This method enables complete blast design. Similar formulas exist for other rocks. Figures 3 & 4 show examples of granite block separation using PETN cord and water.

The main drawbacks of this method stem from the use of explosives, which cause noise, vibrations, projections, induced fractures, and operational disruptions. Additionally, irregular cut surfaces can affect later processing, as smoother

surfaces improve the yield of slabs. Cutting with explosives results in 7 - 10% material loss per block, whereas diamond wire reduces waste to 2 - 2.5% (Cardu & Lovera 2002), a crucial factor for high-value materials. However, diamond wire alone is not yet viable for hard stones due to operational challenges and high costs. Figure 5 presents a combined approach that utilises explosives for vertical and horizontal cuts, and wire for lateral cuts, thereby enhancing production quality and efficiency with a modest cost increase. Further optimisation is needed to leverage diamond wire's precision and automation, alongside explosive splitting's reliability (Raza et al. 2023).

Although conducting a comprehensive economic comparison of different cutting methods

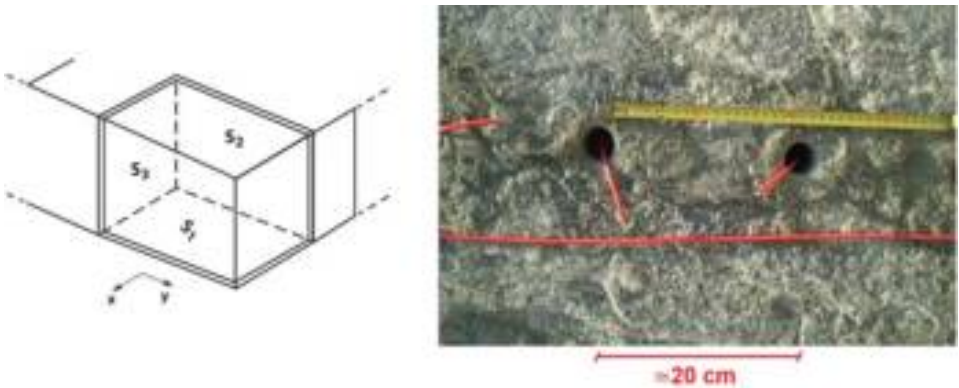


Figure 3. Scheme of the block production process. On the left, the primary block (around 1000 m³) is identified and separated by three splitting surfaces, one horizontal (S_1) and two vertical (S_2 and S_3); on the right, the technique adopted, i.e. strands of detonating cord into the holes and connection to the main cord for simultaneous blasting (Cardu & Todaro 2024).



Figure 4. On the left side, the blasted rock block is shown resting on a debris bed, positioned after the initial detonation. This large block is then divided into vertical slices, which are further separated along pre-induced vertical split planes. Each of these slices is subsequently laid on another debris bed to facilitate further handling. On the right side, the process continues as each vertical slice is further subdivided into smaller blocks of the desired volume, typically ranging from 2 to 6 m³, which are then ready for transport to the processing plant for cutting and finishing operations (Cardu & Todaro 2024).

remains a complex task, evaluating these techniques using objective metrics related to cost-efficiency and practical applicability is nonetheless valuable. To facilitate such an assessment, the following parameters can be considered:

- Volume of rock mechanically removed per unit of cutting area (expressed in m³/m²), reflecting material removal's effectiveness.
- Average cutting area productivity relative to machine power consumption (measured in m²/kWh), providing insight into energy efficiency.
- The typical power range of the machine used (in kW) indicates each method's operational scale and energy demand.
- Geometric limits of the cut, specifically the maximum height (or depth) h and length l of

Table 1. Data for sizing the charges in presplitting operations (Sandvik-Tamrock Corp., 1999).

Hole diameter	Charge	Pipe charge diameter	Pipe charge length	Hole spacing	Specific drilling	Cracked zone
mm	kg/m	mm	mm	m	m/m ²	m
32	0.13	22	380	0.45-0.70	2.22-1.43	0.4
32	0.21	25	1140	0.45-0.70	2.22-1.43	0.4
38	0.21	22	380	0.45-0.70	2.22-1.43	0.4
38	0.21	25	1140	0.45-0.70	2.22-1.43	0.4
51	0.38	32	1000	0.50-0.80	2.00-1.25	0.5
51	0.47	32	380	0.50-0.80	2.00-1.25	0.6
64	0.38	32	1000	0.50-0.70	2.00-1.43	0.5
64	0.47	32	380	0.50-0.70	2.00-1.43	0.6
64	0.55	25	1000	0.70-0.90	1.43-1.11	0.9
76	0.55	25	1000	0.70-0.90	1.43-1.11	0.9
76	0.71	40	380	0.60-0.90	1.67-1.11	1.1
89	0.9	32	1000	0.70-1.00	1.43-1.00	1.7
89	1.32	50	380	0.80-1.10	1.25-0.91	1.9
102	0.9	32	1000	0.70-1.00	1.43-1.00	1.7
102	1.32	50	380	0.80-1.10	1.25-0.91	1.9

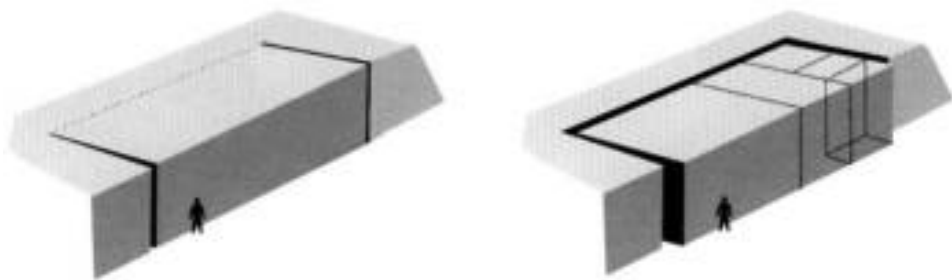


Figure 5. The cutting scheme in a granite bench with lateral cuts made using diamond wire. The back, horizontal, and sectioning cuts always use dynamic splitting (Cardu et al. 2005).

the achievable cut (in metres), define the method's physical constraints.

Table 2 presents indicative reference values for these parameters, offering a comparative overview for two representative rock types: a weak rock (limestone) and a hard rock (granite).

A standard borehole diameter typically ranges between 35 and 50 mm for splitting operations, with hole spacing generally set at approximately 10 times the diameter. Although using smaller diameters, increasing the spacing, or introducing notches along the cut walls can help reduce the volume of damaged rock, these modifications do not necessarily lead to better overall performance (Emad et al. 2022, Ahmadinejad 2009).

Net production is calculated solely based on the machine's effective working hours, excluding time spent on setup, repositioning, tool replacement, and borehole loading during splitting operations. Installed power serves as a general, though rough, indicator of energy cost, with typical power ratings specified for each type of machine. One notable advantage of splitting is that multiple machines can be operated simultaneously, offering flexibility not commonly available in other cutting methods.

The symbol ∞ (infinity) indicates scenarios in which there is no theoretical limit to the cutting span, although practical considerations such as machine repositioning become necessary as the operation progresses. For practical purposes, a cutting span of 8 metres is often considered a standard reference value. The product $b \cdot c$ represents the net productivity of a single, average-performing machine as a function of its installed power.

2 OPTIMISATION OF EXPLOITATION TECHNIQUES IN PORPHYRY QUARRIES

Trentino porphyry is a rhyolitic ignimbrite distinguished by pronounced anisotropy resulting from closely spaced natural joints, facilitating easy slab splitting. Less common discontinuities also influence the shape and size of the slabs.

While the rock mass exhibits anisotropic behaviour, the individual slabs are isotropic and highly resistant to further splitting. Due to its elevated quartz content, the porphyry is exceptionally strong (with uniaxial compressive strength, $C0C\_0C0$, ranging from 200 to 250 MPa) and highly abrasive.

Table 2. Comparison of possible rock detachment methods based on selected objective parameters. W.R. = Soft rocks; H.R. = Hard rocks.

PARAMETER	TECHNOLOGY					
	Splitting		Line Drilling		Diamond Wire	
	W.R.	H.R.	W.R.	H.R.	W.R.	H.R.
a , m ³ /m ²	0.005	0.005	0.05	0.05	0.005	0.005
b , m ² /kWh	0.3	0.15	0.03	0.015	0.2	0.05
c , kW	100	100	100	100	50	50
d , m	8- ∞	8- ∞	8- ∞	8- ∞	10-20	10-20
b·c , m ² /h	30	15	3	1.5	10	2.5

In situ, porphyry is classified into four types:

- Type I - Thin, uniform slabs measuring 1 to 4 cm in thickness.
- Type II - Thick slabs ranging from 15 to 20 cm.
- Type III - Wedge-shaped slabs with variable thickness.
- Type IV - Massive rock that does not form slabs.

Extraction efforts primarily focus on Types I through III, with Type I providing the highest yield and best product quality due to reduced waste generation.

2.1 Traditional exploitation techniques

Industrial-scale porphyry extraction began after World War I, primarily to supply material for road paving. The standard extraction method involved large chamber blasts at the base of rock faces, although horizontal or sub-horizontal blasts were occasionally used to detach overlying rock.

Once quarried, the material was manually sorted and processed into finished products, predominantly paving cubes, which accounted for approximately 90% of production. Yields varied between 20% and 50%, occasionally reaching up to 65%, with an average of around 30%. Annual output from the basin totalled several tens of thousands of tons of finished goods.

Until recently, extraction methods remained essentially unchanged, except for the gradual phasing out of chamber blasting. Mechanisation steadily increased with the introduction of drill rigs, loading shovels, dump trucks, and machine tools.

In 2000, the province had 93 active quarries producing 1.4 million tons annually, valued at €80.1 million, and employing 1,253 people. By 2010, the number of quarries had declined slightly to 91, with production falling to 1.048 million tons, valued at €52 million, and employment reduced to 933 workers.

This downward trend continued through 2015, with 82 quarries producing 708,000 tons, worth €37.8 million, and employing 621 workers. The latest data from 2021 indicate 61 operational quarries, with a total output of 639,000 tons and a market value of €37.3 million (a slight increase from €34.1 million in 2020), employing just 489 workers.

2.2 Blasting schemes (cautious blasting)

Today, most quarries employ controlled blasting techniques using parallel, horizontal boreholes to

minimise explosive consumption and reduce the occurrence of unwanted fractures within the rock mass.

Typical blasting parameters include:

- Drilling diameter: 50–90 mm
- Hole spacing: 1–2 m
- Bench height: 15–25 m
- Drilling depth: 5–10 m
- Explosive consumption: 50–100 g/m³

Dynamite-based explosives remain the standard choice, with initiation occurring at the borehole collar via instantaneous or micro-delayed electric detonators. Stemming material fills approximately 40 - 60% of the borehole length, optimising energy distribution and minimising overbreak.

ANFO has also been tested as a viable alternative to gelatinous explosives, yielding satisfactory results in some cases.

Charge design is based on a static or tensile failure model, where the critical factor is the load applied to the detachment surfaces rather than the specific amount of explosive consumed. The total force exerted by the explosive, acting orthogonally to the borehole plane, is calculated as the product of the explosion pressure and the cumulative cross-sectional area of the loaded segment (active surface). For effective fragmentation, this force must be equal to or greater than the resistance of the separation surfaces between the removable rock mass and the surrounding *in situ* formation.

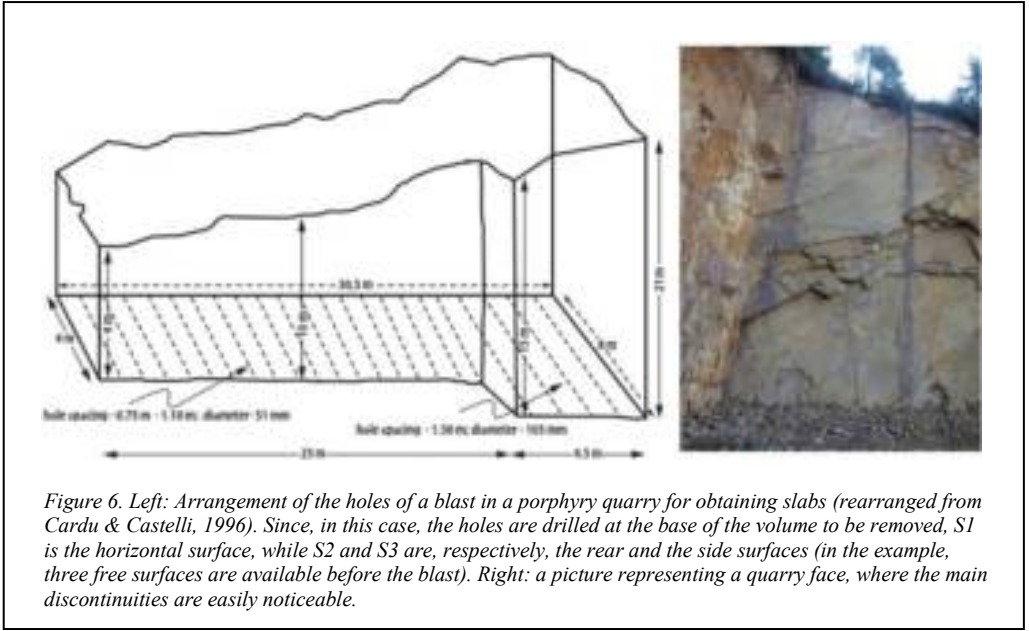
2.3 The 'static' sizing

The explosive force is assumed to act perpendicular to the borehole plane, facilitating the detachment of the target rock volume from the main rock mass (Peacock & Maxfield 2007). This detached volume, idealised as a parallelepiped monolith, is restrained by:

- The tensile strength acting on the charged plane, and
- The shear strength acting along the other detachment surfaces.

In bench blasting with vertical or near-vertical boreholes, these resisting forces predominantly act in horizontal or sub-horizontal directions, opposing the applied explosive force. Detachment is achieved once the explosive force exceeds the combined resistance of these surfaces.

This simplified model assumes negligible energy loss due to fragmentation, treating the blasted volume as remaining largely intact and breaking primarily along pre-existing joints. Therefore, it is most applicable to scenarios involving detachment without significant



fragmentation, such as the extraction of bedded rock slabs.

Figure 6 illustrates a horizontal borehole blast designed for porphyry extraction. In the case of vertical borehole blasting, the corresponding calculations involve:

- Determine the resisting force based on the geometry of the rock mass.
- Define the resisting surfaces as follows:
 - S₁: Tensile-resistant surface containing the charges.
 - S₂: Lower shear-resistant surface.
 - S₃: Lateral shear-resistant surface (it may be absent if the volume is isolated by natural discontinuities or prior excavation).

Next, multiply each surface area by its corresponding unit resistance:

- S₁ by the uniaxial tensile strength.
- S₂ by the shear strength under confining stress (approximated as lithostatic pressure).
- S₃ by the uniaxial shear strength.

Rock formations, especially those exploited for natural slab production, often exhibit mechanical anisotropy; therefore, unit resistance values vary depending on the orientation of each surface. After calculating the resisting forces on each surface, align their directional components and sum them to determine the total resisting force F_r .

The explosive is chosen based on its explosion pressure, p_e , considering the typical loading density.

For bulk density in the borehole, assume the following values:

- Dynamite: 1200 kg/m³
- AN-based explosives: 1000 kg/m³
- ANFO: 800 kg/m³

Dividing the total resisting force by the explosion pressure gives the active surface, S_a , defined as the area over which the explosive must act to produce the force required to balance the resisting force:

$$S_a = F_r / p_e \quad (5)$$

The active surface corresponds to the product of the hole diameter by the total charged length L_c :

$$L_c = S_a / \Phi \quad (6)$$

From the total charged length, the amount Q of explosive is obtained by multiplying the hole volume corresponding to that length by the bulk density of the explosive in the borehole:

$$Q = 0.785 \Phi^2 \cdot \delta \quad (7)$$

Where δ is the density of the selected explosive in the borehole. The total required borehole length L_t is obtained by increasing the total loaded length by the amount needed to accommodate the stemming. This can be achieved by setting an appropriate borehole utilisation coefficient, for

example, $\frac{3}{4}$ (with $\frac{1}{4}$ allocated for stemming and $\frac{3}{4}$ for the explosive). In this case, the result is:

$$L_t = L_c \cdot 4/3 \quad (8)$$

Other factors may also influence the additional length required to accommodate the stemming. By dividing S_1 by L_t , the spacing between boreholes (centre-to-centre) can be determined. The same method applies to blasts with horizontal boreholes, where the tensile-resistant surface corresponds to the base. In this case, the resisting force includes the weight of the blasted volume, and the vertical shear-resistant surfaces are subject to simple shear strength.

It's important to note that this sizing method doesn't account for optimal borehole spacing or specific explosive consumption. For instance, 10 boreholes with a 100 mm diameter, spaced 2 m apart, would consume four times as much explosive as 20 boreholes with a 50 mm diameter, spaced 1 m apart.

To ensure a uniform break surface, excessive borehole spacing should be avoided. The spacing E should not exceed approximately 20Φ , with values between 10Φ and 15Φ being more typical. This sizing approach follows the same principle used in splitting blasts. For isotropic rocks, and in the absence of test data, the following approximate values can be used:

- Tensile strength: ~ 0.1 of compressive strength.
- Simple shear strength: ~ 0.5 of compressive strength.
- Friction coefficient: ~ 0.5 .

3 CONCLUSIONS

This paper presents a comprehensive analysis of the application of dynamic splitting techniques in Italian quarries, combining theoretical foundations with practical implementations. It investigates a variety of geometric configurations employed in the splitting process and assesses their influence on the efficiency and effectiveness of rock fragmentation. The study further evaluates the performance of dynamic splitting across a broad spectrum of rock types, each characterised by distinct geomechanical properties, such as strength, elasticity, and fracture behaviour.

A central focus of the analysis is the set of challenges posed by different geological formations, including anisotropy, material heterogeneity, and the presence of pre-existing fractures, all of which can significantly affect the outcome of splitting operations. The paper also

considers the role of quarry design parameters, such as bench height, spacing and orientation of discontinuities, and overall extraction strategy, in determining the success of dynamic splitting techniques.

In addition, the study introduces a set of calculation methods developed and refined through empirical research and field experience. These methods aim to improve the splitting process's precision, control, and efficiency. They enable the optimisation of critical variables, including the positioning and orientation of applied forces, energy input, and the sequencing of operations, thereby promoting controlled and predictable rock breakage. By integrating these advanced approaches, the paper offers valuable insights intended to enhance productivity, cost-efficiency, and environmental sustainability within the quarrying sector.

REFERENCES

- Ahmadinejad, S. 2009. Application of stress analysis and fracture mechanics in controlled blasting for hard rock quarry mining. *Proceedings of the 1st Symposium of Dimension Stone*, Mahallat, Iran.
- Berta, G. 1990. Explosives: an engineering tool. *Italesplosivi*, Milano.
- Cardu, M. & Lovera, E. 2002. Optimising quarrying techniques and practices. *OSnet (Ornamental Stones Network)*, National Technical University of Athens, GR-157 80 Zografos, Athens, Greece.
- Cardu, M., Lovera, E., Michelotti, E. & Patrucco, M. 2005. Quarrying by explosive and diamond wire in hard dimensional stones. *Proceedings of the Third EFEE World Conference on Explosives and Blasting*, Brighton, R. Holmberg et al. Ed., ISBN 0-9550290-0-7, pp. 409-414.
- Cardu, M., Martinelli, D. & Todaro, C. 2024. Industrial explosives and their applications for rock excavation. *CRC Press, Taylor and Francis Group*, Boca Raton FL, 240 pp.
- Cardu, M. & Todaro, C. 2024. Vibration and stresses induced in the rock due to the splitting of dimension stones' exploitation. *Proceedings of ISRM European Rock Mechanics Symposium (EUROCK 2024)* New challenges

- in rock mechanics and rock engineering. July 15-19, Alicante, Spain.
- Del Greco, O., Fornaro, M., Mancini, R. & Patrucco, M. 1983. Profilatura dei fronti di cava nell'abbattimento con esplosivi: analisi di alcuni esempi notevoli. *Bollettino dell'Associazione Mineraria Subalpina*, 20(1-2), pp. 136-159.
- Emad, M.Z., Majeed, Y. & Rehman, G. 2022. Rock splitting techniques for reducing undesirable cracks and fissures in rock salt blocks. *J. S. Afr. Inst. Min. Metall.* vol.122 n.3 Johannesburg Mar. 2022.
- Liu, K., Qiu, T., Li, X., Zhang, X., Yang, J., Song, R. & Li, X. 2023. Deep rock blasting using decoupled charge with different coupling mediums. *International Journal of Geomechanics*, 23(8), p.04023112.
- Majeed, Y., Emad, M.Z., Rehman, G. & Arshad, M. 2019. Block extraction of Himalayan rock salt by applying conventional dimension stone quarrying techniques. *Journal of Mining Science*, vol. 55, no. 4. pp. 610-625.
- Peacock, D. & Maxfield, V. 2007. The Roman imperial quarries: survey and excavation at Mons Porphyrites 1994-1998, *Volume 2: The Excavations*.
- Raza, M.A., Raza, S., Khan, M.U., Emad, M.Z., Jalil, K. & Saki, S.A. 2023. Cost modelling for dimension stone quarry operations. *J. S. Afr. Inst. Min. Metall.* vol.123 n.11 Johannesburg.
- Sandvik - Tamrock Corp., 1999. *Rock Excavation Handbook for Civil Engineering*. Matti Heiniö Ed., Tampere, Finland, 364 pp.
- Zhang, Q. & Zhao, J. 2014. Quasi-static and dynamic fracture behaviour of rock materials: Phenomena and mechanisms. *International Journal of Fracture*, 189(1):1-32.

AI-integrated ore body movement modelling for enhanced blast outcomes

R. Sahu

Strayos, St. Louis, Missouri, USA

ABSTRACT: Accurate ore body movement prediction during blasting is pivotal for minimising dilution, ore loss, and material misclassification in open-pit mining. This paper presents a novel AI-based solution that integrates deep learning and a physics engine to capture the complex dynamics of blasted material more effectively than traditional blast-movement estimation. By leveraging standard geology, drilling, and survey data, our framework constructs a robust 3D post-blast block model that dynamically reallocates ore and waste zones at a high spatial resolution. Key innovations include a trained neural network that learns site-specific blast parameters (e.g. burden, timing, and rock characteristics) in combination with a physics-driven simulation to replicate particle flow and fragmentation behaviour throughout the muckpile. Iterative data validation and feedback loops enable continual refinement of the ore movement predictions. This dual-model approach provides operators with near-real-time insights into ore-waste boundaries, enabling smarter decisions for blast design, selective digging, and downstream processing.

1 INTRODUCTION

In open-pit mining, accurately predicting how ore and waste move during blasting is crucial for optimising operations. Blasting is a fundamental process for rock fragmentation prior to excavation. The primary objective of a blast design is the efficient distribution of explosive energy throughout a rock bench to achieve a desired degree of fragmentation. However, a significant and often underestimated consequence of blasting is the substantial movement and displacement of the rock mass. This movement causes the post-blast location of ore and waste boundaries to differ significantly from their pre-blast positions, rendering the original grade control models inaccurate for guiding shovel loading. Failure to account for this displacement is a primary driver of ore loss (where valuable ore is mistakenly sent to

the waste dump) and dilution (where waste material is mixed with ore, increasing processing costs and lowering head grade), which directly and negatively impact the economic performance of a mining enterprise.

The challenge is rooted in the nature of blast-induced movement, which is an inherently three-dimensional phenomenon characterised by extreme variability and a lack of a deterministic component. Studies have shown that horizontal movement within a single blast can vary by as much as $\pm 50\%$ from the average, and this variability is influenced by a complex interplay of factors including blast design, rock mass properties, and boundary conditions.

Over the years, several approaches have emerged to address this challenge. Direct measurement techniques, most notably the use of Blast Movement Monitors (BMMs) (Adam &

Thornton 2004), provide crucial *in situ* data by tracking the 3D displacement of sensors within the blast volume. This technology is recognised as a world standard for tracking ore movement and its use has proven to reduce ore loss and dilution.

However, the data provided is sparse, and relying solely on a few measurement points to define the movement of an entire block can be insufficient to capture the full field of complex, variable movement.

Physics-based modelling, particularly using the Discrete Element Method (DEM) (Furtney & Aglawe 2021), offers a way to simulate the dynamics of granular materials and fragmentation. These models can simulate the movement of millions of particles, accounting for interactions and muckpile formation. However, their predictive accuracy is highly sensitive to initial inputs, such as the geological characteristics of the bench, fracture networks, and the precise chemical energy release from explosives, many of which are subject to significant uncertainty. Discrepancies between purely physics-based models and measured reality can lead to significant economic losses.

More recently, machine learning (ML) models have been successfully applied to predict blast-induced rock movement. These models can learn complex, non-linear relationships from

historical blast data to predict movement with high accuracy. While powerful, these data-driven models do not inherently represent the underlying physics of particle interactions, swell, and gravity that govern the final muckpile shape (Fu *et al.* 2023).

This paper presents a hybrid framework that combines deep learning with a physics-based simulation engine also known as Physics Informed Neural Networks. Our approach leverages a trained neural network to interpret site-specific blast parameters and provide tailored energy inputs for a powerful 3D DEM solver. This solver then simulates the full dynamics of rock fragmentation and material flow to produce a high-resolution post-blast block model. By integrating the predictive power of AI with the explanatory power of physics, this framework provides a more robust and practical solution for predicting ore body movement, enabling mines to make smarter, data-driven decisions that enhance blast outcomes and improve the bottom line.

2 FRAMEWORK AND METHODOLOGY

The proposed AI-integrated solution is a multi-stage framework designed to simulate the 3D distribution of material following a blast. The general scheme, shown below, consists of an AI

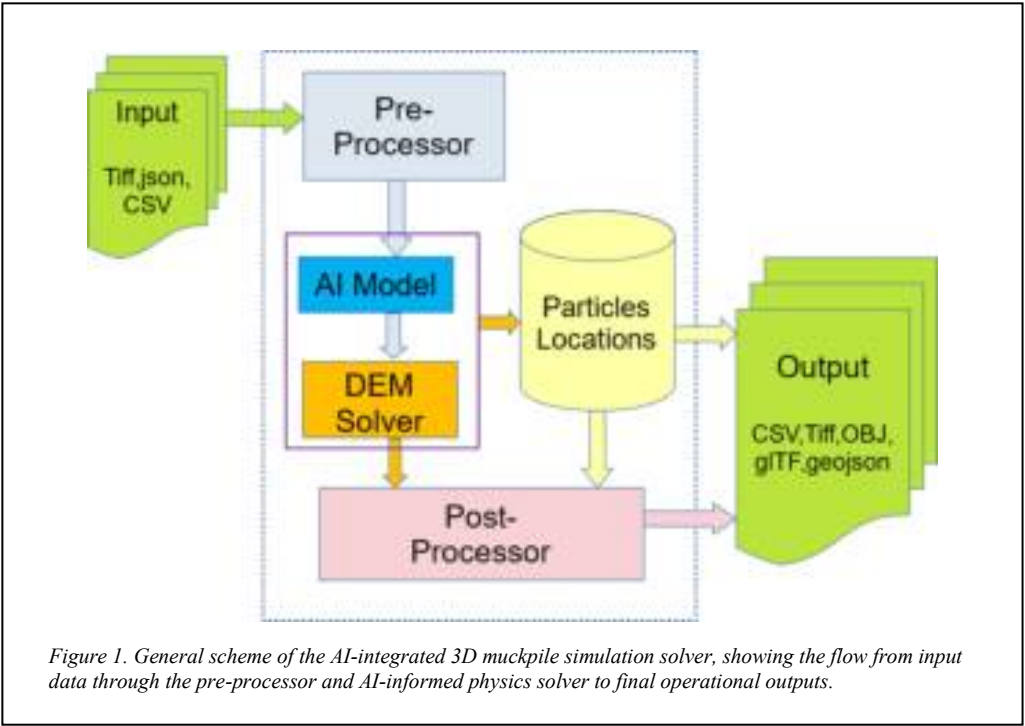


Figure 1. General scheme of the AI-integrated 3D muckpile simulation solver, showing the flow from input data through the pre-processor and AI-informed physics solver to final operational outputs.

model that processes blast design data, a core physics engine that simulates the blast dynamics, and a post-processor that generates actionable outputs for mine operations. This hybrid architecture addresses the computational challenges of full-scale blast simulation while maintaining the essential physics of rock movement and fragmentation. The system processes standard mining inputs including digital elevation models (DEM), drill hole distributions with timing and loading data, ground material properties, and geological block models containing mineral distribution data. These inputs feed into both the AI prediction engine and the physics simulation to generate comprehensive post-blast material redistribution maps.

The methodology is broken down into three principal components: the deep learning component for blast parameterisation, the physics-based simulation engine, and the post-processing stage for generating final models and reconciliation.

2.1 Deep learning component for blast parameterisation

A primary challenge in physics-based blast simulation is accurately defining the initial energy input from the explosive detonation. To overcome this, our framework employs a surrogate model in the form of an artificial neural network. The

purpose of this DL model is to translate readily available blast design parameters into the effective energy conditions required by the DEM simulation, bypassing the need for complex and uncertain first-principles detonation modelling.

The neural network is also trained on a synthetic dataset generated from a large suite of historical blast data or a series of more detailed, small-scale continuum models that capture detonation physics. The network features three hidden layers and is trained using a standard back-propagation algorithm. The network consists of three hidden layers with seven nodes each, utilising hyperbolic tangent activation functions and full connectivity between layers. The inputs to the network include a range of critical, site-specific features that influence blast outcomes:

- Geological Properties: Rock density, Uniaxial Compressive Strength (UCS), rock type.
- Blast Design Parameters: Explosive type, drill-hole diameter and locations, bench height, burden and spacing, and explosive charge length and timing.

The neural network predicts key parameters required for subsequent physics simulations, including equilibrium pressure, burden velocity, and crushed zone radius. This approach achieves burden velocity predictions with an R^2 value of 0.9497, with 92% of predictions falling within 2.5% of true values. The model provides rapid

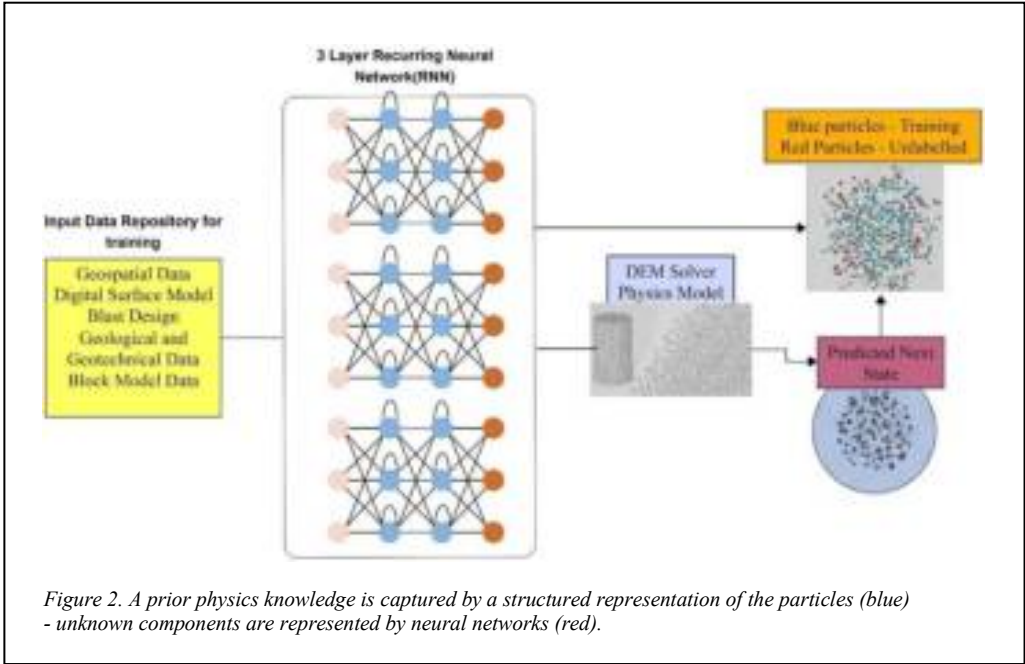
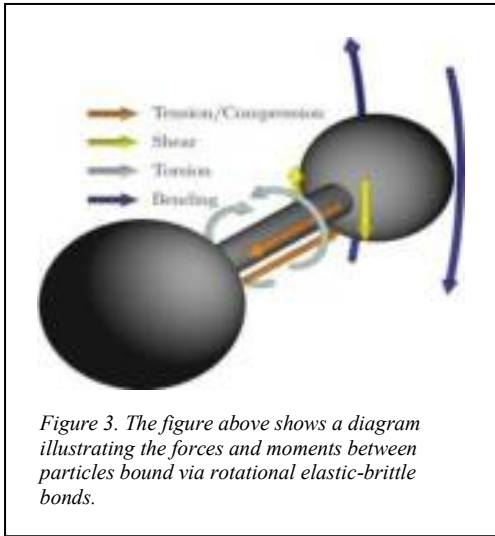


Figure 2. A prior physics knowledge is captured by a structured representation of the particles (blue) - unknown components are represented by neural networks (red).

parameter estimation, enabling near-real-time blast outcome predictions suitable for operational decision-making.

Training data generation utilises axisymmetric finite volume models representing cylindrical rock annuli with central explosive charges. These models capture the early-stage, near-field rock response during blasting, including both elastic and plastic deformation phases. The equilibrium pressure state achieved in these simulations serves as the initial condition for machine learning-informed burden movement calculations. The primary output of the trained neural network is the effective energy released per kilogram of explosive material and its spatio-temporal distribution, which serves as a direct input for the explosion model within the physics engine. This approach allows the system to learn the site-specific blast behaviour and provide a calibrated, realistic energy profile for each unique blast scenario.



2.2 Physics-based simulation engine

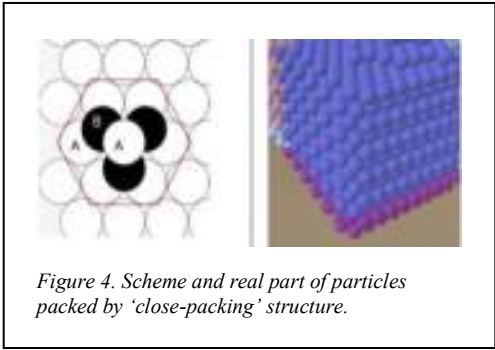
The physics simulation employs the discrete element method (DEM) to model rock mass behaviour as assemblies of spherical particles with defined interaction laws. This approach effectively captures the large deformation and fragmentation characteristics inherent in blast-induced rock movement while accounting for the discontinuous nature of fractured rock masses. DEM is a numerical method ideal for modelling the dynamics of granular and brittle materials by representing the rock mass as a collection of individual spherical particles. The simulation

process involves calculating forces on each particle and updating its position and velocity over discrete time steps using an explicit integration scheme.

Table 1. This table shows source data for the AI simulation solver with a combination of DEM.

Digital Elevation Model.	Geolocated DSM TIFF-file
Drill-holes distribution.	Array of 3D end-points for drill-holes positions.
Timing.	Each drill-hole has own explosion initialisation timestamp.
Loading.	Each drill-hole has own mass of explosion material.
Ground material properties.	Density, Tensile strength.
Explosion properties.	Effective energy release during explosion of 1 kilogram of explosion material.
Block-Model.	CSV-file containing the distribution of various mineral properties, the content of various ores and other characteristics.

The working domain represented as a collection of spherical particles, each of which can interact with neighbouring particles or other objects (e.g. impermeable surfaces) via force-displacement interaction laws. The numerical solution involves calculating the force acting on each particle at a given time, and then updating the particles kinematics using an explicit difference integration scheme (Weatherley *et al.* 2014).



Initial State and Material Properties Calibration: Before the simulation, the domain is populated with spherical particles arranged in a dense hexagonal lattice structure to represent an isotropic, intact rock mass. The mechanical behaviour of the rock is governed by the Mohr-Coulomb failure criterion, which requires

material properties of cohesion, internal friction angle, and density. Particles are connected by elastic-brittle bonds, and when the stress between two particles exceeds the failure criterion, the bond breaks, simulating rock fragmentation. Subsequent motion is governed by residual kinematics and frictional sliding between liberated particles (Cundall 1979).

Material properties require calibration against site-specific rock characteristics and blast performance data. The system accepts standard geotechnical parameters including density, tensile strength, and elastic modulus, converting these into appropriate DEM contact model parameters through established correlation relationships.

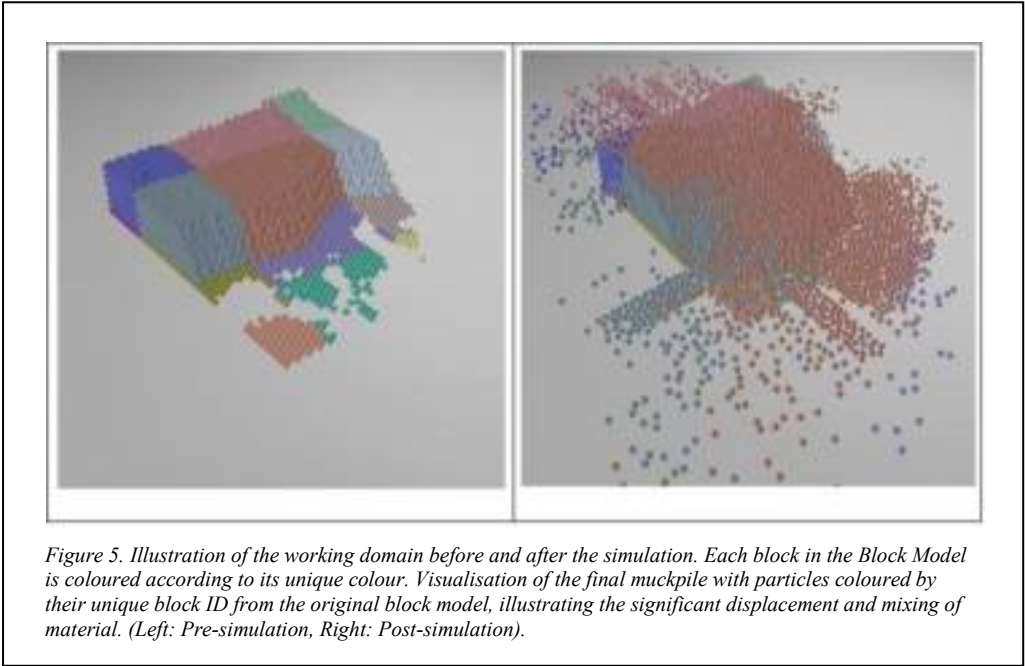
Table 2. Material properties table.

Material Properties	
c	Cohesion
ϕ	Friction internal angle
ρ	Density

Explosion Model: The energy parameters predicted by the AI model are integrated into the simulation. The solver calculates the impact of the explosion by redistributing the AI-predicted energy to particles located in the immediate vicinity of the explosive charges. This energy

transfer is proportional to the surface area of each particle ‘visible’ to the explosive charge segment, ensuring a realistic distribution where closer particles receive more energy. This energy input initiates the particle movement, fragmentation, and subsequent material flow.

Block Model Integration: Each particle within the simulation domain receives an assignment to specific geological blocks from the input block model, enabling tracking of ore and waste movement throughout the blasting process. This block assignment allows quantification of dilution, ore loss, and material misclassification resulting from blast-induced movement. The system maintains continuous tracking of particle positions and velocities throughout the simulation duration, storing this information until the system reaches kinetic equilibrium or predetermined time limits. Post-processing algorithms convert final particle distributions into updated block model configurations, accounting for the three-dimensional redistribution of ore and waste materials. Block model updates consider both direct particle displacement and probabilistic algorithms that account for fragmentation and mixing effects during the blasting process. This approach provides a realistic representation of grade dilution and tonnage redistribution while maintaining consistency with geological constraints (Yu *et al.* 2021).



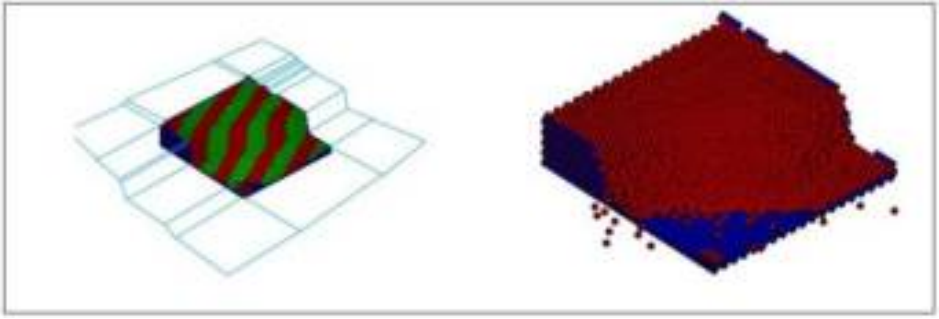


Figure 6. The snapshots show how the blast propagates according to the defined timing sequence, initiating fragmentation and heave that results in the formation of the final muckpile.

Boundary Conditions and Computational Domain: To ensure plausible results, side and bottom particles of the model are fixed to simulate the stable ground surrounding the blast volume, while an impermeable surface prevents particles from exiting the simulation domain unnaturally. All other particles are free to move under the influence of gravity, explosion forces, and inter-particle contacts. Simulation domains typically encompass bench dimensions of $84\text{ m} \times 24\text{ m} \times 12.5\text{ m}$ with particle assemblies containing approximately 400,000 spherical elements. Mean

particle diameter of 0.2 m with 0.5 relative deviation accounts for realistic fragmentation size distributions while maintaining computational tractability.

2.3 Post-processing and output generation

The simulation runs until the kinetic energy of the system dissipates and a stable, static muckpile is formed. The post-processor then converts the final 3D positions of all tagged particles into a range of useful outputs for mine planning and operations.

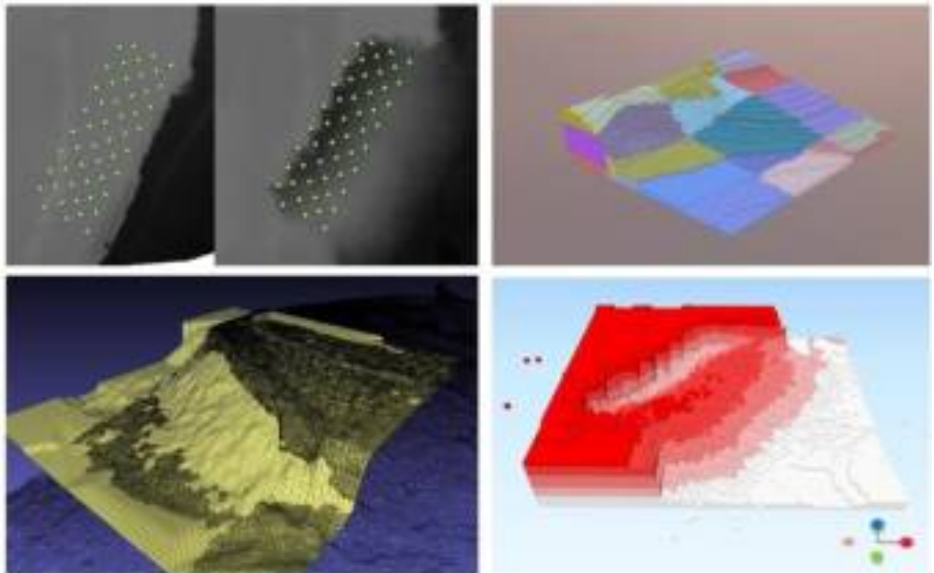


Figure 7. Final particle distribution coloured by a specific ore grade from the block model, revealing the post-blast ore-waste boundaries for improved grade control.

These outputs include:

- A new, high-resolution 3D post-blast block model with reallocated ore and waste zones.
- A geolocated Digital Elevation Model (DEM) of the final muckpile surface.
- 3D surface models (e.g. OBJ, GLTF) for visualisation.
- Voxels and Animation models into CSV files detailing the final coordinates and properties of each block for further analysis.

These outputs provide a comprehensive 3D understanding of the post-blast material distribution, enabling operators to move beyond 2D approximations and make more informed decisions.

3 CASE STUDY AND RESULTS

To demonstrate the capability of the AI-integrated framework, a simulation was performed for a typical open-pit bench blast. The input data for the physics engine included a geolocated Digital Elevation Model of the pre-blast terrain, the 3D coordinates and properties of all drill-holes, and a pre-blast ore block model. The drill-hole data specifies the mass of the explosive charge and the initiation timestamp for each hole, defining a

blasting scenario that proceeds from left to right across the bench.

The AI model processed these inputs along with site-calibrated material properties (Density: 3000 kg/m³, Tensile Strength: 2.5 MJ) to determine the effective explosion energy of 1.93 MJ/kg, which was then fed into the DEM solver.

Muckpile Formation Prediction: The system generates comprehensive three-dimensional muckpile formation predictions showing realistic material distribution patterns. Simulation results indicate that material proximity to free surfaces strongly influences final displacement, with surface materials experiencing greater lateral spread compared to deeper materials. Cross-sectional analysis reveals complex internal structure within muckpiles, with distinct zones of ore and waste materials creating opportunities for selective excavation. The system successfully predicts the formation of ore-rich zones that develop due to preferential movement patterns during blasting (Furtney *et al.* 2013).

Ore-Waste Boundary Evolution: Post-blast ore-waste boundaries show significant deviation from pre-blast configurations, with typical displacement ranges of 2 - 8 m depending on blast design and rock properties. The system accurately

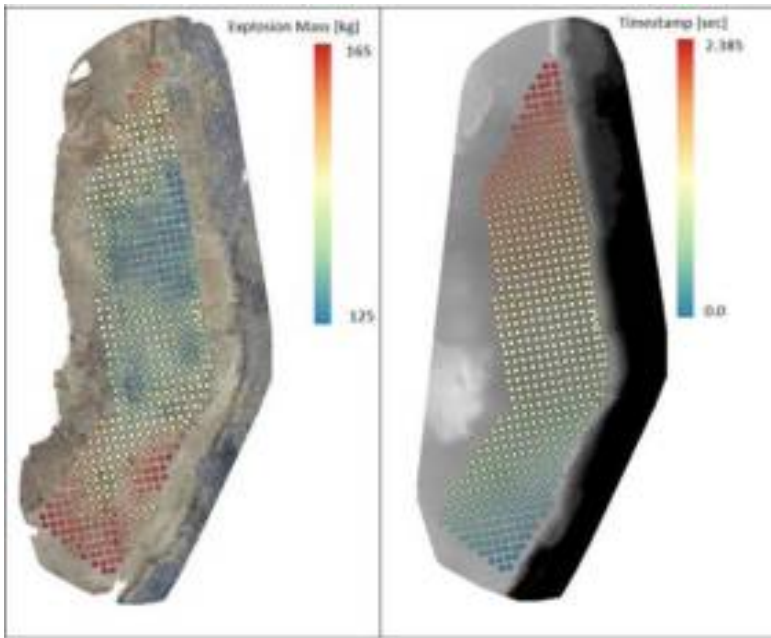


Figure 8. The images above show the orthographic and Digital Elevation Model images with distributed drill-holes, coloured by loaded explosion mass and blasting initiation timestamps.

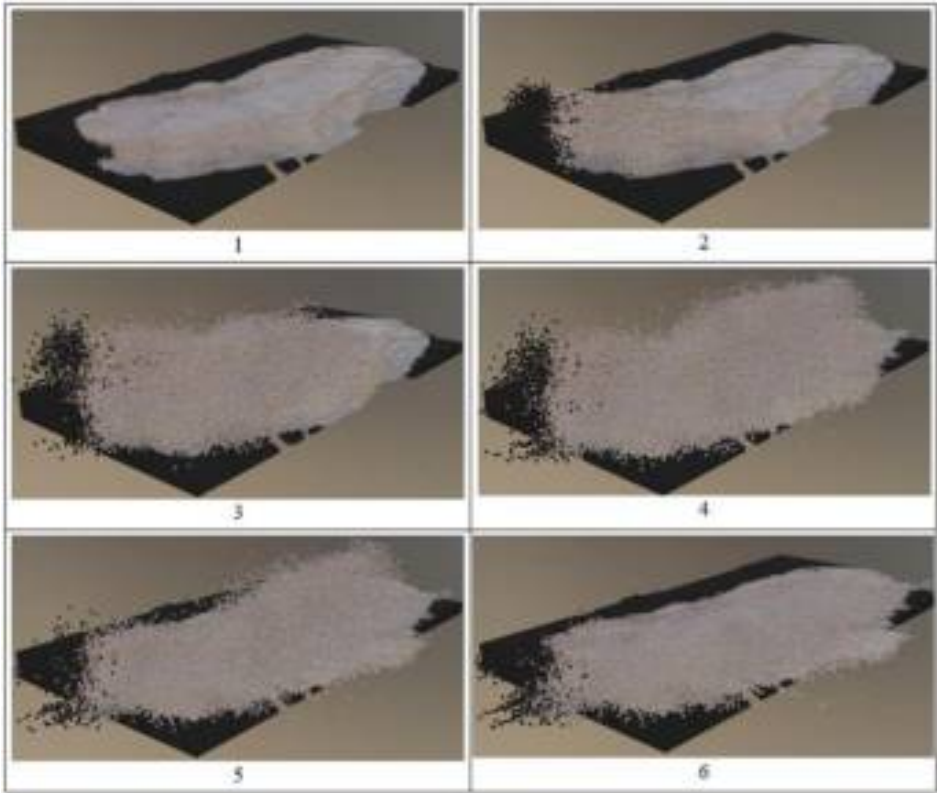


Figure 9. Time shots of the blast simulation results, showing the muckpile development as the explosion sequence progresses across the terrain.

captures these boundary migrations, enabling updated grade control decisions for excavation planning.

Dilution patterns follow complex three-dimensional distributions influenced by explosive placement, timing sequences, and local geological structures. The AI-integrated approach successfully identifies high-dilution zones that require modified excavation strategies or grade control adjustments. Material misclassification analysis reveals systematic patterns related to blast design parameters, providing feedback for future blast optimisation. Areas of significant ore-waste mixing correlate with specific geometric relationships between blast holes and geological boundaries.

The key result of the simulation is the final distribution of the ore and waste material. This visualisation clearly illustrates the significant movement and redistribution of material, with blocks that were originally distinct and separate

becoming intermingled.

By colouring the particles according to their grade content (Figure 9), the simulation provides a clear and actionable view of the post-blast ore-waste boundaries. The pre-blast polygons are no longer representative. The model reveals the new, complex boundaries that must be used to guide excavation. By utilising this corrected grade control model, operations can drastically improve outcomes. Studies using similar advanced modelling approaches have demonstrated the potential to decrease the ore loss rate by over 30% and the ore dilution rate by over 57% compared to using uncorrected pre-blast boundaries.

4 DISCUSSION AND FURTHER DEVELOPMENTS

4.1 Computational efficiency considerations

The hybrid AI-physics approach achieves

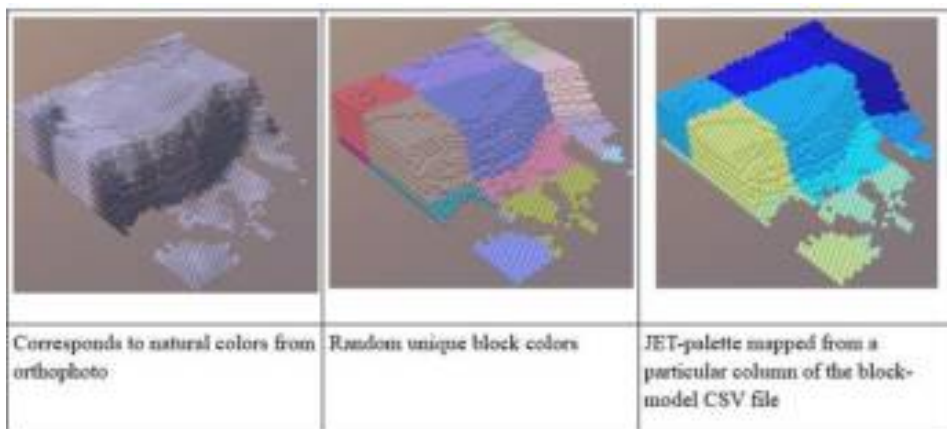


Figure 10. Simulated muckpile with particles coloured according to their original block model index.

significant computational savings compared to full-scale continuum blast simulations while retaining essential physics representation. Deep learning acceleration reduces parameter estimation times from hours to seconds, enabling practical implementation in operational environments. Scalability studies indicate linear computational cost scaling with problem size, making the approach viable for large-scale mining operations. Parallel processing implementations further enhance computational efficiency, with good scaling characteristics across multi-core and distributed computing platforms. Future developments focus on real-time implementation capabilities through advanced hardware acceleration and optimised algorithms. Graphics processing unit (GPU) implementations show promise for further computational speedup while maintaining solution accuracy.

The AI component effectively learns the complex, site-specific relationships between blast design and energy release, providing a practical way to initialise the simulation without relying on uncertain first-principles models of explosive chemistry. The DEM physics engine, in turn, robustly simulates the universal principles of momentum transfer, gravity, and frictional forces that govern particle movement and muckpile formation. This synergy creates a powerful predictive tool that is both adaptable to specific site conditions and grounded in physical reality. When compared with existing methods, the framework offers distinct advantages. While direct measurement systems like Blast movement measurement (BMM) are invaluable for providing

ground-truth data, hybrid AI physics model generates a complete 3D displacement field rather than a set of sparse vectors. Critically, BMM data can and should be used as a key input for the iterative validation and refinement of our AI model, creating a feedback loop that continually improves prediction accuracy, a concept aligned with the cost functions proposed by other researchers. Unlike purely data-driven ML models that predict displacement vectors in isolation, this integrated framework simulates the physical interactions, collisions, and constraints within the muckpile. This ensures that the final predicted muckpile shape, volume, and internal swell factor are physically realistic, avoiding the inaccuracies that can arise from simply translating pre-blast blocks based on interpolated vectors.

4.2 Integration with mine planning systems

The system architecture supports integration with existing mine planning and grade control software through standard data exchange formats. Block model updates utilise industry-standard formats compatible with major mining software packages. Automated workflows enable seamless integration into daily mining operations without disrupting existing procedures. The system processes standard geological, surveying, and blast design data already collected during normal mining activities. Future developments will include direct integration with autonomous mining equipment and real-time grade control systems. This integration would enable immediate adjustment of excavation strategies based on updated post-blast

ore-waste boundaries. This enables optimised, selective excavation that precisely separates ore and waste, maximising resource recovery and minimising the amount of barren material sent for costly processing. The result is a direct and positive impact on the mine's value chain, from improved reconciliation to increased profitability (Parker 2011).

4.3 Limitations and areas for improvement

The accuracy of the simulation is fundamentally dependent on the quality of the input data, including the geological block model and the surveyed drill-hole locations. The current implementation uses spherical particles, which is an approximation of the angular nature of fragmented rock, though the use of rolling resistance models helps mitigate this simplification. Future work will focus on integrating more complex particle shapes and incorporating detailed rock mass fracture data to further refine the fragmentation and movement models. The ultimate goal is to create a fully autonomous system where real-time monitoring data continuously calibrates the simulation, driving a cycle of perpetual optimisation for blast design and execution.

5 CONCLUSION

A coupled AI - physics framework markedly enhances blast-movement prediction, enabling data-driven blast optimisation and selective loading. Future work will incorporate hyperspectral drone scans for rock-property refinement and GPU acceleration for million-particle simulations. Combination of predictive neural network with a high-fidelity DEM physics engine can effectively capture the complex, three-dimensional dynamics of muckpile formation. The AI model learns site-specific blast characteristics to provide realistic energy inputs, while the physics engine simulates the subsequent fragmentation and particle flow to generate a robust and high-resolution post-blast block model. The primary contribution of this work is the demonstration of a practical and powerful hybrid approach that overcomes the individual limitations of purely data-driven or purely physics-based models. The case study illustrates the framework's ability to transform standard operational data into actionable intelligence, revealing the true post-blast distribution of ore and waste. This information is pivotal for optimising dig limits and improving grade control.

REFERENCES

- Adam, M. & Thornton, D. 2004. A new technology for measuring blast movement. In: *Innovative Mineral developments Symposium*. Sydney, NSW: AusIMM Sydney Branch.
- Cundall, P.A. & Strack. 1979. A discrete numerical model for granular assemblies.
- Fu, W., Furtney, J. & Valencia, J. 2023. Blast movement simulation through a hybrid approach of continuum, discontinuum, and machine learning modelling.
- Furtney, J.K., Sellers, E. & Onederra, I. 2013. Simple models for the complex process of rock blasting. *Rock Fragmentation by Blasting. The 10th International Symposium on Rock Fragmentation by Blasting*, 275-282.
- Furtney, J. & Aglawe, J. 2021. PFC3D Modelling of rock fragmentation by pressure pulse. In S. K. Saha & M. Mukherjee (Eds.), *Recent Advances in Computational Mechanics and Simulations* (Vol. 103, pp. 237-246). Springer Singapore.
- Parker, H.M. 2011. Reconciliation principles for the mining geologist. *Proceedings of the 6th International Mining Geology Conference*, Darwin, Australia.
- Weatherley, D., Hancock, W. *et al.* 2014. *ESyS-Particle Tutorial and User's Guide*. The University of Queensland.
- Yu, Z., Shi, X., Zhou, J., Gou, Y., Rao, D. & Huo, X. 2021. Machine-learning-aided determination of post-blast ore boundary for controlling ore loss and dilution. *Natural Resources Research*.

[RETURN TO CONTENTS PAGE](#)

Advanced edge blasting techniques and digital simulation to reduce material spillage and optimise mining efficiency

G.S. Huerta Valer

ORICA, Mauritania

B. Diallo

ORICA, EMEA

E. Thibault

KINROSS, Mauritania

ABSTRACT: In open-pit operations, maintaining ramp access during blasting is essential, especially when lower-level ramps are the sole route to active ore zones. This case study explores a series of controlled blasting trials conducted in a non-production test zone to evaluate edge blast designs aimed at minimising material spillage and preventing ramp obstruction. Four blast sequence designs, echelon, centre-lift, fishbone, and pre-split variations were tested under controlled conditions. Key parameters included drill accuracy, burden orientation, QA/QC practices, and delay timing. Post-blast assessments using drone-based topographic surveys quantified material displacement and spillage. To complement field trials, advanced digital movement simulations called OREPro™ 3D Predict were employed to predict rock movement for each design. The integration of simulation outputs with empirical data enabled a comprehensive evaluation of each strategy's effectiveness. Findings revealed that while certain designs, particularly those with optimised timing and hole configurations, reduced ramp spillage, variability persisted due to execution factors. The study demonstrates the value of combining field experimentation with digital modelling to enhance blast control, improve safety, and ensure uninterrupted ore access in sensitive pit zones.

1 INTRODUCTION

During the development of Phase 5 of the West Branch Pit, a critical operational challenge emerged due to the proximity of the blasting zone to the only access ramp leading to Phase 4, the mine's primary source of high-grade ore. The ramp, located directly beneath the edge of the planned blasts in Phase 5, is a

vital piece of infrastructure for maintaining a continuous ore supply to the plant.

Given the strategic importance of Phase 4 in meeting production targets, optimising edge blasting became a central focus for the Drill & Blast team. The objective was to ensure the structural stability of the ramp while maintaining operational efficiency.



Figure 1. Phase 5 of the WB pit, edge zone and affected ramp area.

Phase 5 primarily consists of waste material with occasional low-grade mineralised zones. In contrast, Phase 4, located deeper in the mine, contains low-, medium-, and high-grade ore and serves as the main source of ore for the stockpile that feeds the processing plant.

Conducting edge blasts without applying controls, alternative designs, or simulation support to predict outcomes could negatively impact ore supply to the stock piles, delay the mining schedule, and hinder the achievement of the mine's overall objectives.

2 TECHNICAL SOLUTION

To address the challenge of minimising spillage of blasted material onto the lower ramp, a series of controlled blast trials and design evaluations were conducted. A test area within the same mining phase, geologically and operationally consistent with the critical edge zone, was selected to allow for experimentation without disrupting ore hauling or access to Phase 4. This setup enabled the safe and effective testing of alternative blast configurations under representative conditions.

The initial phase of the project focused on optimising the initiation point (IP) location to reduce material displacement toward the ramp. Three configurations were tested: IP positioned far from the edge, IP close to the edge, and a centrally located IP, referred to as the 'Centre Lift'. In all designs, additional delays were introduced in the

two rows adjacent to the free face to create secondary relief surfaces. This sequencing allowed the edge rows to displace toward the centre of the blast pattern, thereby reducing the risk of material spilling over the ramp.

To support the design process, OREPro™ 3D Predict was integrated with i-kon™ III electronic detonators, enabling accurate simulation of blast-induced movement. This combination allowed for predictive modelling of displacement volumes and informed real-time adjustments to blast timing and geometry.

3 RESULTS AND DISCUSSION

In this section, all the results from the preliminary test blast phases carried out in an edge zone with consistent conditions are presented. Based on the outcomes, key learnings were applied to the directly impacted edge area. Using the data obtained, multiple blast simulations were developed to estimate movement and adjust the blasting sequences in order to minimize displacement. The use of OREPro™ 3D Predict was a key tool during these trials, as it allowed us to proactively adjust design parameters and achieve improved results.

- Trial 01 featured an IP located far from the edge, with a fully free-faced pattern. A delayed 'V-shape' timing was applied at the rear to reduce backbreak, and both back corners were angled inward to control material flow. Surveying data reported a

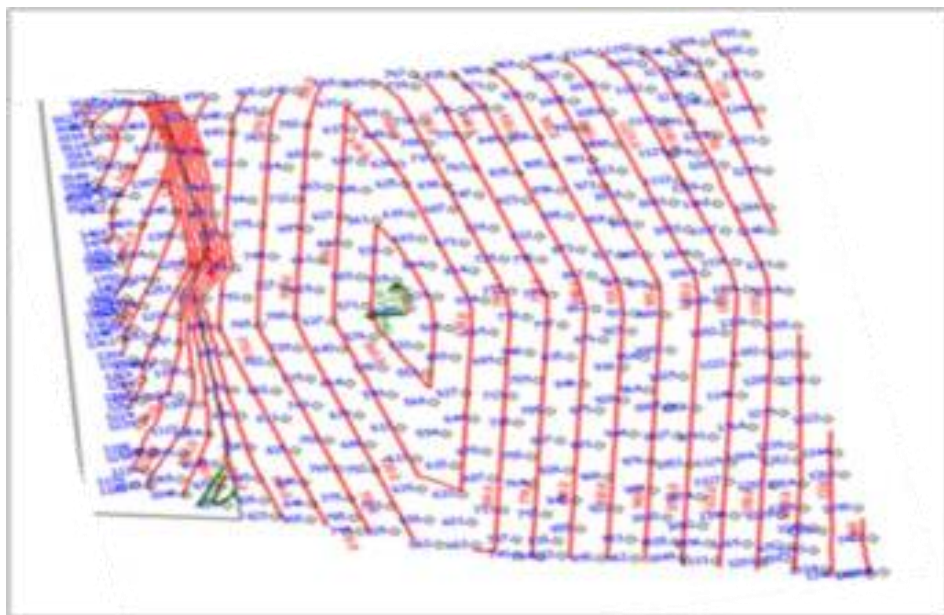


Figure 2. WB5 – 5020 - 8a centre lifted pattern isolines.

spill rate of 38 m³/m of free face, equivalent to 4.3%.

- Trial 02 placed the IP close to the edge, using a single free-face. A wide-angled ‘V-shape’ was implemented to enhance displacement and reduce confinement. The control line timing was aligned parallel to the edge to generate an echelon effect. This configuration achieved a reduced spill rate of 32 m³/m of free face, or 3.1%.
- Trial 03 the ‘Centre Lift’ positioned the IP at the centre of the pattern and divided the blast into two functional zones:
 - Zone 1 used a diamond-shaped lift to create a central void, drawing material from both sides. The lift was slightly offset toward the edge to maximise energy transfer.
 - Zone 2 consisted of three echelon-aligned rows designed to direct material toward the central void while minimising unnecessary displacement.

This configuration yielded the most favourable outcome, with a final spill rate of 26 m³/m of free face, or 2.1%.

All trials were conducted under consistent geological and operational conditions, ensuring reliable comparisons. The insights gained from

these tests were subsequently applied to the critical edge zone, supported by simulation-driven design refinements aimed at minimising spillage and improving overall blast performance.

Among all the blast patterns evaluated, Pattern WB5 - 5020 - 20 demonstrated the most effective control of material displacement, achieving the lowest recorded spillage at 19.5 m³/m of free face, equivalent to 1.00%. This configuration incorporated a ‘centre lift’ tie-up, a power trap row, and a tight timing sequence ranging from 70 ms to 2 ms, which collectively contributed to its superior performance in minimising edge spillage.

However, despite its success in reducing material displacement, this design parameter would not be adopted for official edge blasting operations. The primary limitation lies in its adverse impact on rock fragmentation quality, which is critical for downstream handling and processing. Additionally, the pattern does not fully leverage the availability of multiple free faces within the blast geometry, an important factor for optimising overall blast efficiency and fragmentation outcomes.

Following the completion of all trial blasts, the Orica technical team collaborated closely with the client to refine the most consistent and effective design, an echelon firing pattern at 45° with short

Table 1. Edge blast trials summary.

Date	Pattern	Status	Tie up	IP Location	Charge variation	Power trap row	Timing	Spillage (M3/M)	% Spillage	P80
6/2/24	5020 - 4	Baseline	Wide V	4th row from edge	None	None	35 ms - 75 ms	49	5.80%	n/a
10/2/24	5020 - 7	Baseline	Wide V	4th row from edge	None	None	35 ms - 75 ms	32	3.10%	256
16/02/24	5020 - 8a	Trial	Centre lift	8th row from edge	None	Yes	70 ms- 17 ms	26	2.10%	231
20/02/24	5020 - 8b	Trial	Wide V	14th Row from edge	None	None	105 ms - 17 ms	38	4.30%	246
24/02/24	5030 - 23	Trial	Wide V	12th Row from edge	Yes	None	70/50 ms 6ms	48	7.30%	307
10/3/24	5020 - 5	Trial	Echelon	8th Row from edge	None	Yes	50 ms- 50 ms	25.3	5%	308
21/03/24	5020 - 9	Trial	Centre Lift	9th Row from edge	None	Yes	-	40.5	1.60%	294
26/03/24	5020 - 20	Trial	Centre Lift	9th Row from edge	None	Yes	-	19.5	1.00%	354
11/4/24	5010 - 9	Trial	Wide V	8th Row from edge	None	Yes – 500ms	70 ms - 2 ms	52.3	2.4%	295
16/04/24	5010 - 10	Trial	Wide V	8th Row from edge	None	Yes – 500ms	70 ms - 2 ms	134.8		278
28/05/24	5000 - 1	Trial	Centre lift	8th Row from edge	None	None	67 ms – 17 ms	42	2.3%	n/a
13/06/24	5000 - 2	Trial	Wide V	7th Row from edge	None	None	74 ms - 5 ms	57	3.1%	252.2

diagonals, designed to optimise burden relief across the face. The intra row, hole-to-hole delay was set at 7 milliseconds, with row-to-row delays of 95 milliseconds, ensuring progressive energy release. The revised design extended the blast envelope toward the eastern boundary to incorporate additional material, thereby reducing the likelihood of repeated closing of the access ramp due to spillage events.

To validate and optimise the updated configuration, a series of simulations were conducted using OREPro™ 3D Predict. To accurately simulate blast-induced movement using OREPro™ 3D Predict, the following input data is essential:

- Bench and drill pattern geometry (including burden, spacing, and hole depth).
- Geological properties of the test area (rock type, density, fragmentation characteristics).
- Initiation Point (IP) location and blast design parameters.
- Timing sequence and delay intervals, especially in rows adjacent to the free face.
- Operational constraints, such as proximity to ramps and access roads.

- Conducting drone flights immediately after blasting to capture high-resolution topographic data of the muckpile.
- Generating a 3D surface model from drone imagery to quantify actual material displacement.

The simulation process involves designing the blast in OREPro™ 3D Predict using detailed input data, including drill geometry and timing. Post-blast drone surveys capture the muckpile topography, which is used to generate a 3D model and calculate actual displacement volumes. These results are compared with predicted movement to validate the design and refine future configurations, ensuring minimal spillage and optimised blast performance.

The first simulation was based on the planned blast design, incorporating the actual topography, drill hole layout, and charge distribution. The initial simulation provided an estimation of 27,587 m³ of material spillage into the ramp.

A second simulation was then performed using the as-drilled hole coordinates (X, Y) to account for deviations in drilling accuracy, including un-drilled holes and additional holes outside the original design.

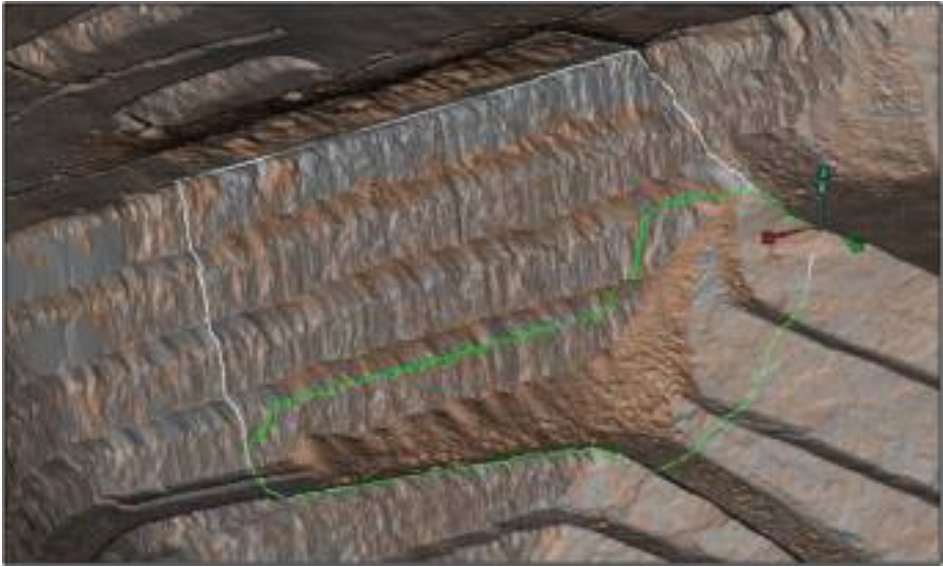


Figure 3. Post blast displacement survey view.

This second simulation proved critical in identifying discrepancies between the design and actual field conditions, particularly in the timing isolines. These variations had the potential to affect blast performance and material movement. As a result, targeted adjustments were made to the

initiation timing of specific holes in the edge zone to mitigate the impact of these deviations and ensure the blast behaved as intended. This simulation estimated a total spillage volume of 28,957 m³ into the ramp.

In this case study, three production blasts, WB5

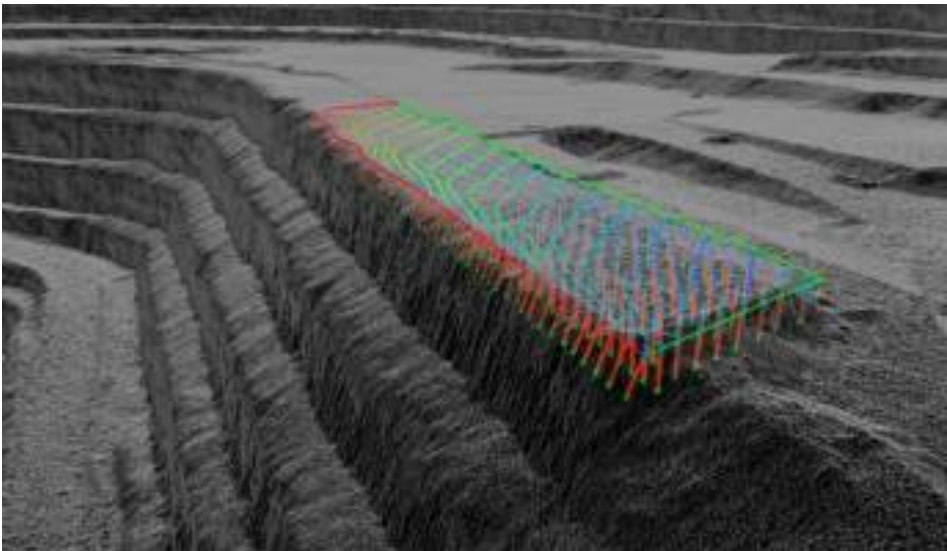


Figure 4. OREPro™ Predict simulation using the actual topography.

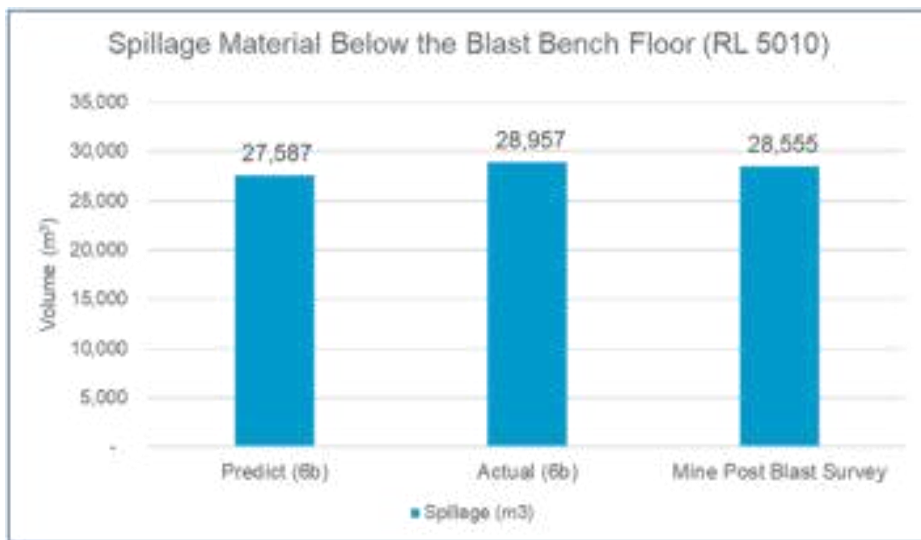


Figure 5. Proposed vs actual timing design.

– 5010 - 6B, WB5 – 5010 - 21, and WB5 – 5020 - 22 were evaluated to determine the most effective initiation timing for minimising spillage towards the lower ramp.

Blast ID 5010 - 6B, the first blast executed along the edge zone, was successfully carried out on 08/26/2024. For this blast, i-kon™ III electronic detonators were utilised, with the timing sequence directly imported from SHOTPlus™ to minimise the risk of human error during programming.

Following the blast, the mine’s survey team conducted a detailed post-blast mapping to quantify the volume of material displaced onto the ramp. The measured spillage volume was 28,555 m³. When compared to the predicted volume of 28,957 m³ from the OREPro™ 3D Predict simulation, the difference was approximately 402 m³, representing a variance of around 1.4%. This close correlation between predicted and actual outcomes validates the accuracy of the simulation model and the effectiveness of the adjusted blast design.

The subsequent blast, WB5 - 5010 - 21, executed on 09/01/2024, incorporated a more refined delay strategy to improve control over material displacement. Specifically, a +21 ms delay was applied to the penultimate row, while the final row was initiated 200 ms after the last production row, with 50 ms inter-hole delays. This

controlled timing sequence effectively confined explosive energy within the blast zone, reduced forward throw, and resulted in a noticeable reduction in spillage.

In contrast, the earlier blast, WB5 - 5010 - 6B, employed a simplified timing approach, extending the delay in the final two rows to 250 ms. While this configuration delivered good fragmentation and produced uniform face profiles, it lacked the finer control over edge displacement achieved in blast 21. The comparison highlights the importance of precise timing adjustments in managing blast-induced movement, particularly in sensitive edge zones.

The third blast, WB5 - 5010 - 22, carried out on 09/06/2024, followed a similar timing design to blast 21. However, it resulted in greater spillage. This discrepancy suggests that execution-related factors, such as timing inaccuracies or overlap, may have led to premature initiation of edge holes, reducing confinement and increasing material displacement toward the ramp.

Based on these observations, the timing configuration used in WB5 - 5010 - 21, featuring staged delays in the final two rows, has proven to be the most effective in controlling edge throw and minimising spillage. As such, it is recommended as the preferred approach for future blasts conducted near critical areas such as haul roads and access ramps.

Table 2. Edge blast trials results summary.

PATTERN ID	WB5 - 5010 - 6B	WB5 - 5010 - 21	WB5 - 5020 - 22
Simulation OREPro 3D post-blast.	27,587	11,156	25,190
Simulation surveyed post blast topography.	28,555	11,522	23,473
Difference.	~3%	~3%	~7%
Actual Result (Vol.).	11.1%	8.23%	10.6%
Actual Result (m3/m).	202.0%	90.72	117
QAQC Overall Score.	Average	Low	Low
Tie Up Description.	Echelon 45° starting at the free face. Lengthen the time in the last two rows towards the edge up to 250ms	Echelon 45° short diagonals, 7ms@holes, 95ms@row. Penultimate row +21ms, last row 200ms after last production hole 50ms@hole	Echelon 45° starting at the free face. 3 IP to keep initiation angle. Lengthen the time in the last two rows towards the edge up to 250ms
Fragmentation (P80).	222mm	220mm	232mm
Face Description.	Even, multiple edge holes	Uneven, Many gaps edge holes	Even. multiple edge holes

4 SUMMARY AND CONCLUSIONS

Based on historical data related to productivity and mining performance under ramp-constrained conditions, potential operational benefits were estimated using a consistent set of assumptions. Maintaining the same shovel productivity rate during ramp cleanup, a simulated scenario involving an uncontrolled blast predicted a spillage volume of 63,550 m³, which would require approximately six days to clear. In contrast, the actual blast resulted in a significantly lower spillage volume of 28,555 m³, which was cleaned up in less than three days.

This reduction in cleanup time not only accelerated the reopening of access to Phase 4, the mine's primary source of high-grade ore, but also contributed directly to improved ore delivery continuity. As a result, the operation-maintained alignment with production targets, demonstrating the tangible benefits of controlled blasting and optimised design execution.

REFERENCES

- Angeles, E. 2023. Blast movement modelling with Orepro™ 3D. *Orica Case Study*.
- Kifumbi, B. 2024. Increasing control of ore dilution to maximise material blending with Orepro™ 3D. *Orica Case Study*.

- EFEE 2019. SPH modelling of rock blasting: calibration from bench face movement data. *European Federation of Explosives Engineers Helsinki Conference Proceedings* 2019.
- Huang, M., Liu, Y. & Yang, Y. 2022. Edge detection of ore and rock on the surface of explosion pile based on improved Canny operator. USBM Report of Investigations 8506: 21-22
- Van Aswegen, C.V.B. & Cunningham, H. The estimation of fragmentation in blast muckpiles by means of standard photographs.

[RETURN TO CONTENTS PAGE](#)

5. Explosive detection for security

Findings on the development of efficient methods for neutralising improvised explosive devices

R. Laszlo, E. Ghicioi, C. Jitea & B. Simon

*National Institute for Research and Development in Mine Safety and Protection to Explosion
– INSEMEX Petrosani, Romania*

A. Nicola

University of Petrosani, Romania

ABSTRACT: Improvised Explosive Devices (IEDs) kill thousands every year, inflict grievous physical injuries, cause dire psychological harm and spread fear and disruption across affected communities. Their impact on security and stability is profound. Depending on the function that their users want them to play, IEDs will vary. They could be intended to carry out deliberate attacks on people and vehicles, or they could be made to destroy infrastructure and result in a large-scale death toll. Multiple configurations are always possible because the only thing limiting IED technology is the creativity of the person making or deploying the devices. The paper presents the research work and some of the features of a liquid-jet neutralisation device of the improvised explosive assembly and its testing program. The purpose of the tests was to verify the device's performance and adaptability for urban area interventions.

1 INTRODUCTION

1.1 A world-wide threat

An explosive weapon that is not conventional in nature and can be detonated in multiple ways is known as an improvised explosive device, or IED. Both citizens and troops are their objective. IEDs are becoming more and more significant in today's conflicts, and they will remain a component of the operational landscape for NATO military operations in the future (Reference 1). IEDs have a severe negative impact on the local populace and are one of the primary causes of troop casualties (References 2,3). The Alliance assists members and partners in building their own capabilities to lessen the hazards presented by IEDs, with a focus

on creating counter-measure technology, doctrine, education and training.

In recent years, IEDs have become the primary weapon for non-state armed groups across many conflicts. IED incidents have occurred in 66 countries and territories in the last three years, including in Africa, the Americas, Asia, and Europe. Eight countries saw over 1,000 civilian casualties of IEDs. The threat of IED attacks is a global problem. Cheap and relatively easy to construct, IEDs can be made anywhere from a wide range of materials - from everyday tools to commercial explosives used in construction and mining (References 4,5).

Depending on the function that their users want them to play, IEDs will vary. They could be intended to carry out deliberate attacks on people

and vehicles, or they could be made to destroy infrastructure and result in a large-scale death toll. Their position, level of destruction, and method of ‘delivery’ of the explosive device to the target all influence their role and intended impact (Reference 6). Multiple configurations are always possible because the only thing limiting IED technology is the creativity of the person making or deploying the devices. However, one design limitation has to do with the attackers’ inclination for close closeness to the victim. In certain situations, the attackers may decide to avoid discovery or cause themselves harm by staying far away from the equipment (Reference 7).

1.2 Operational considerations

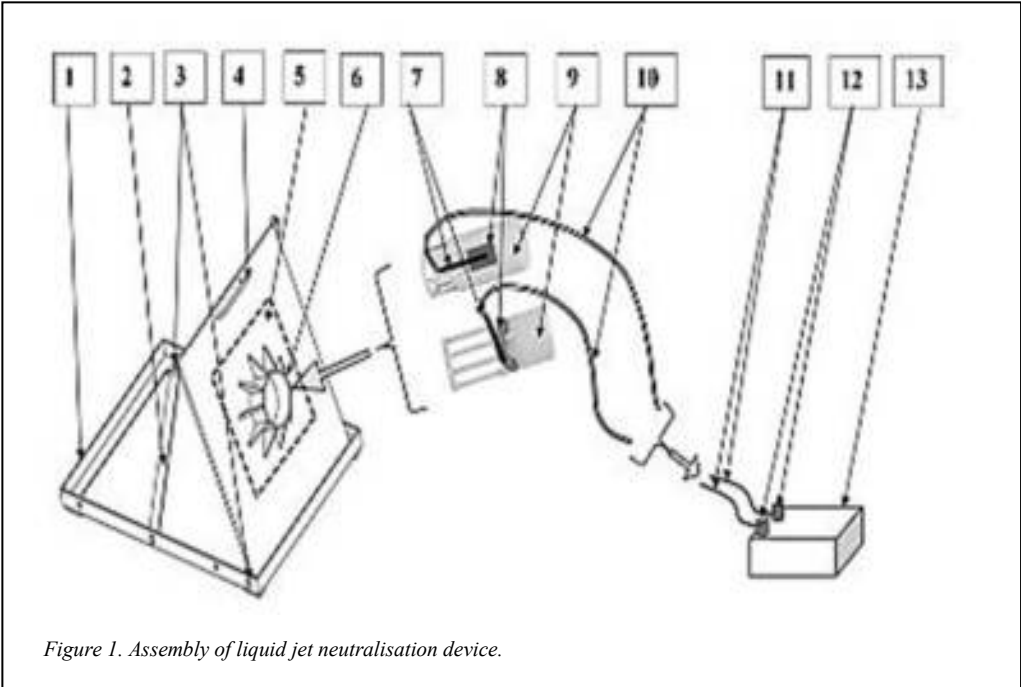
At present, at national and international level, there are several variants of water jet disruptors for intervention on improvised explosive devices DEI, in which the propulsion of water is ensured by initiating cartridges of special construction, inserted at the base of a metal pipe of similar length to a rifle, and the manipulation of the disruptor by the operator is done in the vicinity of the target - DEI.

There are several international patents dealing with the construction problem of water-based disruptors, e.g. Patent No.: US 9,322,625 B1, 2016, *Systems and methods for launching water*

from a disrupter cannon, Patent No.: US 7481,146 B2, 2009, *Device for disrupting improvised explosive devices (IEDS)*, in which the expulsion of water is along metal pipes, either truncated conical or cylindrical, and the thrust gases are generated by special cartridge explosive mixtures, with mechanical or electrical initiation.

In the course of 2023, a research team from National Institute for Research and Development in Mine Safety and Protection to Explosion - INSEMEX Romania perfected, patented, and tested the liquid-jet neutralisation device of the improvised explosive devices DNJLDEI (prototype for an improvised explosive device neutralisation device), which has the potential to be a valuable tool for urban interventions against terrorist attack.

These findings follow the results obtained in 2022 from the experimentation of a prototype de-neutralisation device for explosive devices DNDEI, which uses relatively small explosive remnants made of optimised metal and reusable water-receptacles. This paper presents some of the features of the device and it’s testing experimental program. The purpose of the DNJLDEI tests and experiments was to verify the workable capacity for urban area interventions. Tests were carried out for the two improved variants of plastic containers, presented in the previous paragraph, with measurements of the velocity of the projected



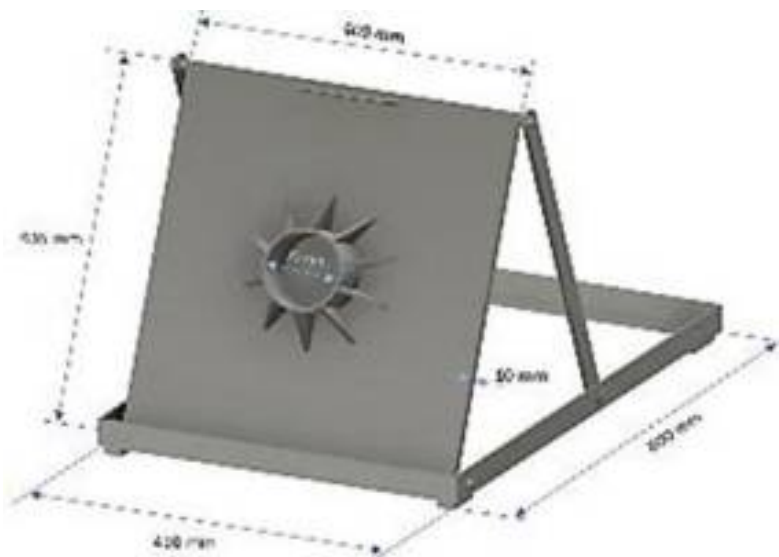


Figure 2. Neutralisation device base with adjustable frame.

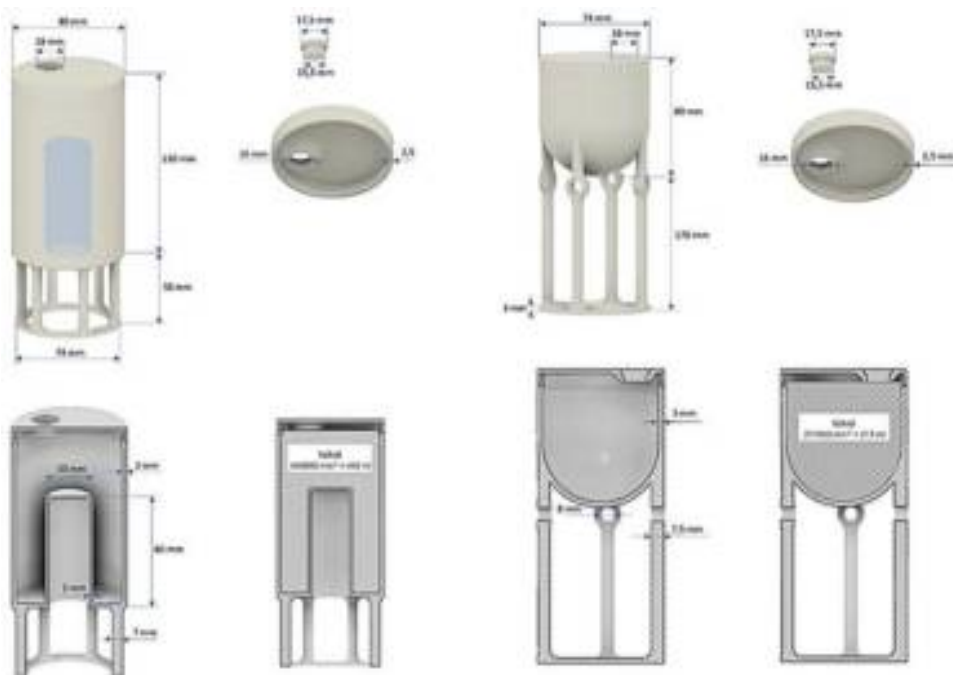


Figure 3. (Left) Long container with both flat ends. Figure 4 (Right) Short container with hemispherical end.

liquid jet (water) (References 8-11).

2 CONCEPT AND METHODS

Liquid jet neutralisation equipment for improvised explosive devices, consists of a set of parts that allows the propulsion of quantities of water at very high speeds, over 500 m/s, which pulverise the improvised explosive device, without detonating it.



Figure 5. Two constructive types of plastic container.

The assembly presented in Figure 1, consists of an adjustable metal frame (1), movable arms (2) and fixing screws (3) to various degrees of inclination (from horizontal to vertical working position), a reinforced metal base plate/mortar (5), provided with a carrying handle (4), and having a centrally positioned cylindrical plug (6) - reinforced around the perimeter and at the bottom (see details in Figure 2), for mounting the specially shaped plastic containers (9) - cylinder equipped with legs and fixing disc, with both flat closing surfaces (see details in Figure 3) or with one flat end and one hemispherical end (see details in Figure 4), removable containers, with a threaded cap in which there is placed a volume of liquid, as well as a small charge of plastic explosive (8), moulded on the lower surface of the compartment intended for it, into which an electric detonator (8) is inserted for initiation, and the detonator wires (10) are connected through a cable (11) to the terminals (12) of a blasting machine (13), and operated from a safe distance.

3 SITE TESTS AND RESULTS

3.1 Testing the ability of the water jet to penetrate a steel sheet

As mentioned before, two constructive variants of plastic containers with different dimensions and volumes have been perfected, which have been experimented and tested to study the speed and effect of the liquid jet. The testing of the neutralisation device was aimed at verifying the work capacity, which can be used in interventions in the urban area. Tests were carried out for the two improved versions of plastic containers (Figure 5), measuring the speed of the projected liquid jet (water). Also, a series of experimental tests were carried out with the devices to verify the ability of the water jet to penetrate a 0.4 mm thick steel plate, as well as to study how it affects an explosive cartridge (RIOPER G - 250 g, diameter 32 mm), considered in the testing activities as explosive-target-part of DEI (Figure 6).



Figure 6. Explosive placed on a steel plate.

Small plastic explosive charges (SEMTEX) were used, with a mass of 10 g and 20 g (Figure 7) respectively, initiated by an electric detonator (Figure 7), inserted in plastic containers (for the two types of containers with legs, cylindrical shape of approx. 450 ml, respectively with a hemispherical end-bottom of approx. 200 ml). A silicone wax was used to seal the lid thread and the filler plug of water containers (Figure 8).

The experimental program took place under controlled conditions, ensuring the safety of operators, with the provision of landmarks and markers necessary to calibrate the distances travelled by the liquid jet, to measure the



Figure 7. Preparation of SEMTEX explosive charges.



Figure 8. Preparation of plastic container (200 ml).

propulsion speed with the help of high-speed camera PHANTOM (Figure 9) and image processing software. The tests carried out had the following results:

- The use of plastic containers with a hemispherical part (filled with liquid-water with a volume of approx. 200 ml), together with a reduced load of 20 g SEMTEX (Figure 9), allows to obtain a central liquid jet, perpendicular to the mortar with a liquid jet velocity of more than 310 m/s at a distance of 300 mm, with a limited deformation capacity, without penetration of the steel sheet with a thickness of 0.4 mm (Figure 10), located at a distance of approx. 50 mm, and the explosive cartridge positioned on the sheet was fragmented and projected (vertically) at more than 10 m away (Figure 10).
- The use of plastic containers (filled with liquid with a volume of approx. 450 ml), together with a reduced load of 10 g SEMTEX, allows obtaining a central liquid jet, perpendicular to the mortar (Figure 11a (left)), with a maximum liquid jet velocity of 1080m/s (in the first 200 mm of liquid expulsion), and after 300 mm, the jet has a speed of over 560 m/s (Figure 11b (right)), with a limited deformation capacity, without penetration of the steel sheet, located at a distance of approx. 50 mm, and the explosive cartridge positioned on the sheet was fragmented and projected more than 10 m away (vertically).

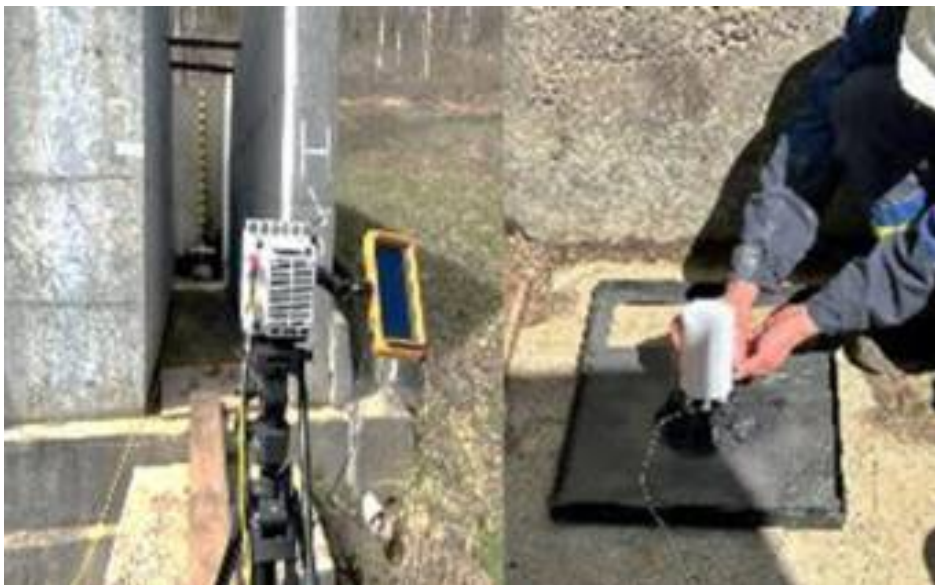


Figure 9. Preparation for blasting test.



Figure 10. Steel sheet and explosive after test.

a)

b)



Figure 11. Liquid jet velocity of a) (left) 1080 m/s at 200 mm and b) (right) 560 m/s at 300 mm.



Figure 12. Steel sheet and explosive after test using 20 g of explosives and 450 ml of liquid.



Figure 13. Improvised explosive device setup.



Figure 14. Field positioning of the neutralisation device against of IED.

When using a 20 g SEMTEX explosive charge, the sheet was perforated (Figure 12), and the explosive cartridge was fragmented and projected more than 10 m away (vertically).

3.1.1 Intermediary conclusions

The performance achieved with the plastic container with a liquid volume of approx. 450 ml is clearly superior to that achieved with the container with a hemispherical surface of approx. 200 ml. This can be explained, in addition to the smaller amount of liquid, by the fact that the reduced explosive charge is applied to a free/open surface, which produces a significant loss of energy, non-transferable to the liquid, resulting in a reduced recovery of the mechanical work generated by the combustion gases at detonation.

3.2 Testing liquid jet neutralisation device of improvised explosive bomb (suitcase)

The last stage of the testing program was called

the final functional and operational validation of the intervention module for DEI neutralisation. At this stage, an improvised explosive device was made (Figure 13), consisting of a rectangular suitcase, which was equipped with an explosive cartridge, an electric cable connected to the helmet jack of a mobile phone - as an electric signal generator, which could be operated/controlled remotely by mobile phone radio waves.

In Figure 14 the field positioning of the neutralisation device is shown in the vicinity of the suitcase. It was considered that the suitcase was inspected non-invasively/x-rayed and the location of the detonator was identified (marked on the suitcase with X -DET), so the neutralisation device was positioned so that the liquid jet acted directly on the detonator, in the present case being a mobile phone.

Following the use of the liquid jet neutralisation device on the suitcase equipped with an explosive cartridge, an electric cable connected to the headset jack of a mobile phone - considered



Figure 15. Perforation of the DEI case.



Figure 16. Destruction of the electrical circuit between the 'detonator' - mobile phone and explosive.

as a detonator, the following results were obtained:

- Perforation of the DEI case in front of the detonator (Figure 15).
- Destruction of the electrical circuit between the 'detonator' - mobile phone and the explosive cartridge without affecting the cartridge in the DEI type suitcase (Figure 16).
- Projection of the mobile phone detonator at over 25 m by the liquid jet from the neutralisation device.

The following is a sequence of representative frames of the use of the liquid jet neutralisation device, which allows direct observation of the effectiveness and precision with which the liquid jet expelled, destroyed and projected the IED detonator at a long distance and destroyed the electrical connection, without affecting explosive cartridges in the bomb suitcase (Figure 17).

3.2.1 Intermediary conclusions

As a result of the experiment carried out, it can be stated that the expulsion of the detonator from the target suitcase - IED (improvised explosive device), without affecting the explosive cartridge, validates the functionality, effectiveness and selectivity of the liquid jet neutralisation device, its use being easy, and the operation is carried out in safe conditions.

4 CONCLUSIONS

In the article was presented the results of testing a liquid-jet neutralisation equipment of an improvised explosive devices - IED (a prototype of a neutralisation device for improvised explosive

bombs), which has the potential to be a valuable tool for urban interventions against terrorist attack. The use of cylindrical plastic containers (filled with liquid with a volume of approx. 450 ml), together with an explosive load of 10 g of SEMTEX, allows obtaining a central liquid jet, perpendicular to the mortar, with a maximum liquid jet speed of 1080 m/s (in the first 200 mm of liquid ejection), and after passing 300 mm, the jet has a speed of over 560 m/s, with a limited deformation capacity, without penetrating a 0.4 mm thick steel plate, located at a distance of approx. 50 mm, and the explosive cartridge positioned on the sheet was fragmented and projected more than 10 m away (in the vertical plane).

When using a 20 g SEMTEX explosive charge, the sheet was perforated, and the explosive cartridge was fragmented and projected over 10 m away (vertically).

The last stage of experimental test program of was final verification of functionality of new prototype of IED neutralisation device, based on liquid-jet, the operational validation being performed upon a bomb briefcase, which was neutralised by accuracy destroying of detonator, without damages of explosive charge.

The device has the advantage of being configured on site, depending on the way the target is presented (briefcase, backpack, package, etc.), being able to use small explosive charges (approx. 10 – 20 g), with their placements in the place intended for a plastic containers of a special shape for the arrangement of the liquid (which can be water, water with dissolved salt or another kind of denser liquid), so as to obtain a more concentrated or more dispersed projected jet, at volumes of different liquids, between 200 ml and



Figure 17. Frames with the functioning of liquid jet neutralisation device.

600 ml. The initiation is performed from a safe distance for the operator of the device, after installation in the vicinity of the target.

REFERENCES

- Ref 1 – Daniel, C.G. & Frances, L.E. 2014. Improvised explosive devices. *Crisis and Emergency Management*, Routledge, 2014, pp. 120-124.
- Ref 2 – Wilkinson, A., Bevan, J. & Biddle, I. 2008. Improvised Explosive Devices (IEDs): An Introduction *Conventional Ammunition in Surplus*, Switzerland, 2008, pp. 136.
- Ref 3 – Graham, B. & Ricks, T.E. 2004. Munitions Issue Dwarfs the Big Picture. *Article in Washington Post*. 29 October, p. A01.
- Ref 4 – Beckett, A., Pelletier, P. & Mamczak, C. 2012. Multidisciplinary trauma team care in Kandahar, Afghanistan: current injury patterns and care practices. *Injury*, 2012, pp. 2072.
- Ref 5 - Slawinski, G., Swierczewski, M. & Malesa, P. 2017. Modelling and numerical analysis of explosion underneath the vehicle. *J. KONES* 2017, 24, 279–286.
- Ref 26 - Jamroziak, K., Bajkowski, M., Bocian, M., Polak, S., Magier, M., Kosobudzki, M., & Stepień, R. 2019. Ballistic head protection in the light of injury criteria in the case of the Wz.93 combat helmet. *Appl. Sci.* 2019, 9, 2702.
- Ref 7 - Ramasamy, A.M., Hill, A.M. & Clasper, J.C. 2009. Improvised explosive devices:

pathophysiology, injury profiles, and current medical management. *J. R. Army Med. Corps* 2009, 155, 265–272.

Ref 8 - Ghicioi, E. & Laszlo, R. 2023. Patented Request *CBI OSIM nr. a/00178/11.04.2023-DNJLDEI-INSEMEX*, 2023.

Ref 9 - *UEFISCDI project no 33 SOL/2021* - Complex simulator for the development, testing and validation of methods and means of response specific to intervention forces in the case of asymmetric threats and risks occurring in urban areas.

Ref 10 - *Core Program Project - INSEMEX-EXH2* - Increasing the national expertise capacity of explosions, fires, equipment in explosion-proof construction, explosive materials, technological processes, the environment, as well as the development of solutions to improve the level of safety and health at work specific to industrial applications endangered by atmospheres explosives, generated by the presence of hydrogen, 2022-2026.

Ref 11 - *PCDIEX – INSEMEX Explosives research* / development and testing site, substances flammable / toxic, anti-explosive equipment and personnel training intervention for toxic / explosive environments – PCDIEx.

[RETURN TO CONTENTS PAGE](#)

6. Health and safety in drilling and blasting

Prediction of flyrock distances using XGBoost machine learning method

Y. Ağan & T. Hüdaverdi

Istanbul Technical University, Istanbul, Türkiye

ABSTRACT: Blasting operations are frequently carried out in open-pit mining to excavate the rock. After initiation, the detonation reaction continues until the explosive material is consumed, resulting in rock fragmentation and displacement. In some cases, undesired environmental effects such as ground vibration, dust formation, airblast and flyrock are inevitable. Flyrock is relatively more critical in comparison to other environmental effects. It can cause fatal injuries and requires various precautions. Flyrock throw may be controlled by adjusting main blast parameters. Stemming distance, powder factor and burden are critical variables for flyrock distances. Additionally, artificial barriers may be replaced around the blast area to control flyrock. Prediction of flyrock distances is very important to create a safe zone free of personnel and equipment. Many researchers have proposed various equations to forecast flyrock distance. In recent years, artificial intelligence-based machine learning methods have been increasingly utilised for prediction. In this study, a predictive model was created using the extreme gradient boosting (XGBoost) regression algorithms. A total of 111 blasts were observed in an aggregate quarry located in Cendere Valley, Istanbul region. Seventy percent of data was used as training data set and 30% of it was used as test data. Selected input parameters of the prediction models are burden (B), spacing (S), bench height (H), stemming (T) and the powder factor (Pf). The output parameter is the flyrock severity which was determined based on the measured flyrock cases. The performance of the models was evaluated based on five different metrics. The calculated coefficient of determination (r^2) value is 0.951. The mean absolute error (MAE), root mean square error (RMSE), mean absolute scaled error (MASE) and root mean squared scale error (RMSSE) values are 5.386, 7.821, 0.172 and 0.220, respectively. In addition to the performance metrics, the impact of input parameters on the model's prediction performance was analysed using SHapley Additive exPlanations (SHAP) diagrams.

1 INTRODUCTION

In open-pit mining, blasting operations are carried out to break or loosen the rock. Through blasting, materials with suitable fragment sizes can be prepared at minimal cost. Blasting operations offer significant advantages in terms of production rate and cost-effectiveness. On the other hand, blasting can also lead to undesirable environmental impacts. These environmental effects are generally

categorised under four main headings: ground vibration, air blast, dust formation, and flyrock (Figure 1).

The factors causing these environmental effects can be examined in two groups. The first group includes controllable blast design parameters. These parameters consist of design variables such as spacing, hole diameter, burden, bench height, and stemming length, as well as the characteristics of the explosives used. The second group consists

of uncontrollable rock mass parameters, which are related to the physical, mechanical, and structural properties of the rock. Examples of these include rock strength, elastic modulus, density, number of joint sets, joint orientation, and joint spacing (Hudaverdi *et al.* 2011). Conducting various measurements during blasting operations provides significant benefits for predicting potential environmental impacts that may arise afterward. In this way, measures can be taken to reduce these effects in subsequent blasts. Among all the effects, ground vibration tends to spread over the largest area. It can cause discomfort to people living in nearby residential areas and may even result in material damage. Ground vibrations are largely dependent on the charge per delay and the distance from the blasting point (Monjezi *et al.* 2011). Using seismographs, ground vibration and air blast levels can be accurately measured.

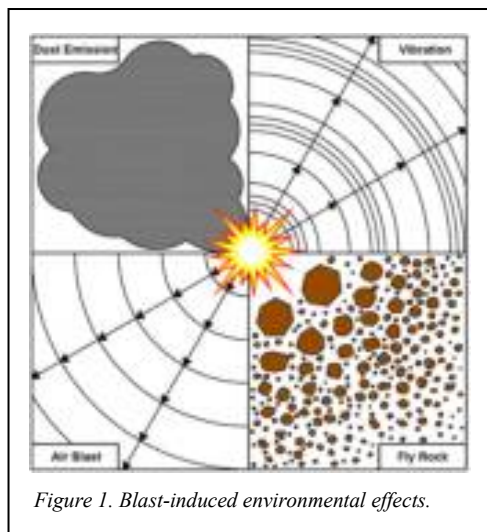


Figure 1. Blast-induced environmental effects.

Many researchers have attempted to predict vibration levels by developing models based on scaled distance (Duval & Petkof 1959, Langefors & Kihlstrom 1963, KG & Zienkiewicz 1968, Standard 1973, Roy 1991). Similarly, air blast, which is generated as a result of blasting, has also been predicted through empirical models by several researchers (Siskind *et al.* 1980, Hasanipناه *et al.* 2016, Bansah *et al.* 2016, Nguyen *et al.* 2020). Dust formation occurs in nearly every stage of an open-pit mining operation (Wang *et al.* 2025). During blasting, explosives produce energy that breaks the rock and generates a significant amount of dust. A dust cloud, often mixed with the smoke from explosives, spreads over a wide area (Alvarado *et al.* 2015).

Additionally, environmental factors such as temperature and wind make it more difficult to control or capture the dust (Vaupel *et al.* 2016). To reduce dust, pre-blast spraying is often used. In a study by Wang *et al.* (2025), the water infusion blasting technique was shown to significantly reduce dust formation. Flyrock can occur in various forms from the bench. Flyrock distances can be measured using tape surveying (chaining) after the blast or through high-speed camera recordings. In bench blasting, compared to other impacts, flyrock presents a serious risk of injury or even death to workers (Xiang *et al.* 2020). It may also cause significant damage to quarry equipment and nearby residential areas. Many researchers have developed mathematical, empirical and semi-empirical models to estimate flyrock distances. These models will be discussed in the following sections. Today, with the use of artificial intelligence-based algorithms, it is possible to build models capable of predicting flyrock distances. These models offer the advantage of solving multivariable and complex problems in relatively short times (Raina 2023).

In the second section of this paper, the study area and the collected data will be described. The third section will present the flyrock distance prediction models proposed by various researchers. In the fourth section, the literature studies will be summarised. The fifth section will focus on the developed machine learning method. Finally, the performance of the proposed model will be evaluated based on five different error metrics.

2 FIELD MEASUREMENTS

Site investigation was carried out in the Kemerburgaz Region of Istanbul. All field measurements were conducted in the Akdağlar Sandstone Quarry, which is located near five other neighboring quarries (Figure 1). Geologically, the area is part of the Trakya Formation which is primarily composed of sandstone layers. In addition to sandstone, thin layers of shale and limestone are also present. The Trakya Formation consists of Carboniferous-aged, fragmented sedimentary rocks. These sediments have undergone deformation, folding, and fracturing due to joint systems formed during the Hercynian and Alpine orogenic events. The sediment thickness generally ranges between 5 and 50 cm but can locally increase to 50–100 cm, particularly in sandstone-rich zones (Tugrul & Undul 2006).

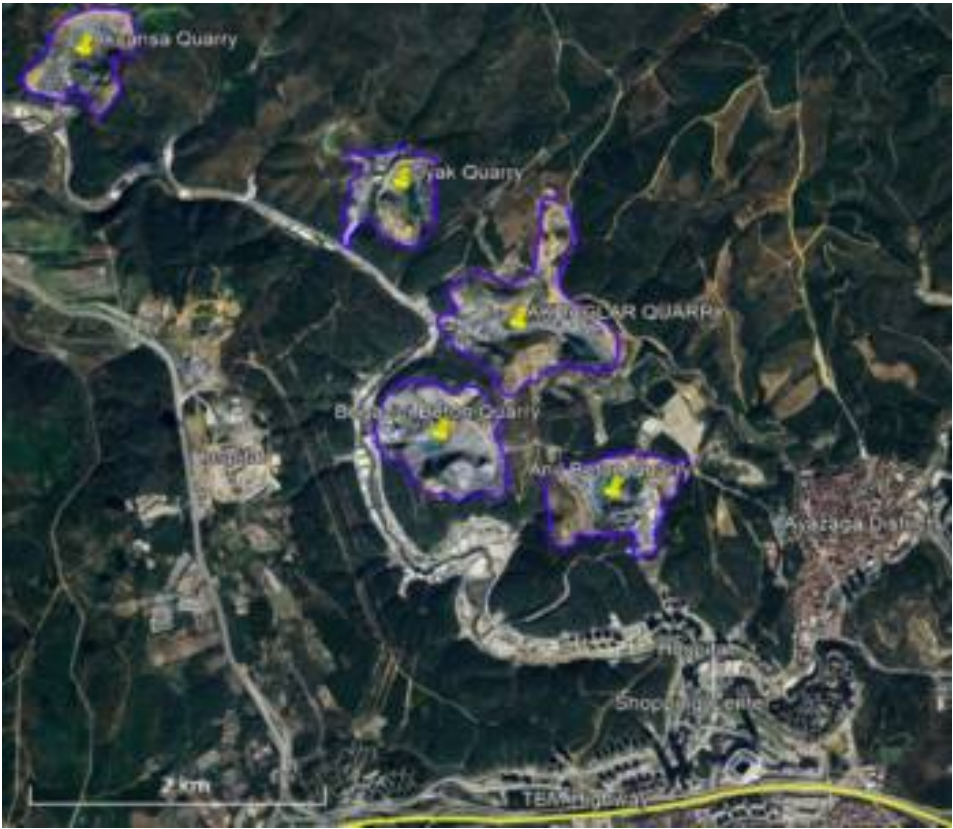


Figure 2. Satellite image of the mining site and surrounding settlements.

The diameter of the blast holes used in the quarry is 89 mm, and ANFO is used as the primary explosive. Both electric and Nonel detonators were employed in a staggered firing sequence for initiation. Various blasting parameters were measured on-site, including spacing (S), burden

(B), bench height (H), hole diameter (D), sub-drilling (U), stemming (T) and powder factor (Pf). In total, the model is created 5 input parameters: B, S, H, T, Pf. The output parameter is the flyrock distance (Fdis). Descriptive statistics of the measured data are presented in Table 1.

Table 1. Descriptive statistics for the measured data.

Variables	Min	Mean	Max	Std dev.	Skewness	Kurtosis
B (m)	1.8	2.243	2.5	0.211	-0.147	-1.113
S (m)	2.0	2.643	3.0	0.216	0.132	-0.286
H (m)	5.5	7.078	10.0	0.896	0.801	1.213
T (m)	2.0	2.43	3.0	0.294	0.366	-0.15
Pf (kg/m ³)	0.47	0.662	0.96	0.121	0.372	-0.759
Fdis (m)	28	79.234	142	35.74	0.243	-1.414



Figure 3. Bench view of the blasting area (a) and usage of the drone (b).

An example image of the benches where measurements were taken is presented in Figure 3a. After the blast, flyrock distances were measured using two handheld GPS devices and the classical tape measure method. Additionally, drone images were recorded before and after blasting for certain shots where the bench face was cluttered with rock debris (Figure 3b).

3 FLYROCK DISTANCE PREDICTION MODELS

Flyrock distances are among the most critical environmental factors resulting from blasting operations. Flyrock refers to the ejection of rock material from bench weaknesses as a result of blasting, caused by a combination of blast design parameters, explosive materials, and geological features. Gibson & St. George (2001) defined flyrock distance as the unexpected and undesirable projection of rock fragments that occurs during a blasting operation. By predicting flyrock distances, precautionary measures can be taken to mitigate this undesirable effect. Flyrock prediction equations can be classified into three main categories.

3.1 Mathematical models

In the study conducted by Lemesh & Pozdnyakov (1972), rock movements were monitored using high-speed cameras. According to this study, no movement was observed in the rock mass within 15–40 milliseconds after the blast, and afterward, the rock mass started to move at a velocity of

15–30 m/s. According to this study, flyrock distance was stated to be related to the throw velocity and time, and Equation 1 was formulated accordingly:

$$F_{dis} = v_0(t_0 - t_c) \quad (1)$$

where, F_{dis} represents the flyrock distance (m), v_0 is the initial velocity of the rock (m/s), t_0 is the time elapsed since the detonation of the explosive (s), and t_c is the time when the rock begins to move (s). Lundborg (1974) developed a theoretical relationship between the flyrock distance and the blast hole diameter (inches) for crater-type blasts (Equation 2).

$$F_{dis} = 260 \times d^{1.5} \quad (2)$$

Lundborg *et al.* (1975), in their study conducted in quarries, stated that the flyrock distance is related to the exit velocity and specific charge, and developed Equation 3. In this study, it was suggested that for blast hole diameters ranging from 75 to 200 mm, the maximum flyrock distance could range between 500 and 1000 metres.

$$F_{dis} = 137 P_f \times 255. P_f > 0.1 \quad (3)$$

where, F_{dis} represents the flyrock distance, and P_f is the specific charge (kg/m^3). In the kinematic equation developed by Roth (1979), it was stated that the flyrock distance is related to the gravitational acceleration and the angle of projection. F_{dis} is the flyrock distance (m), v_0 is the

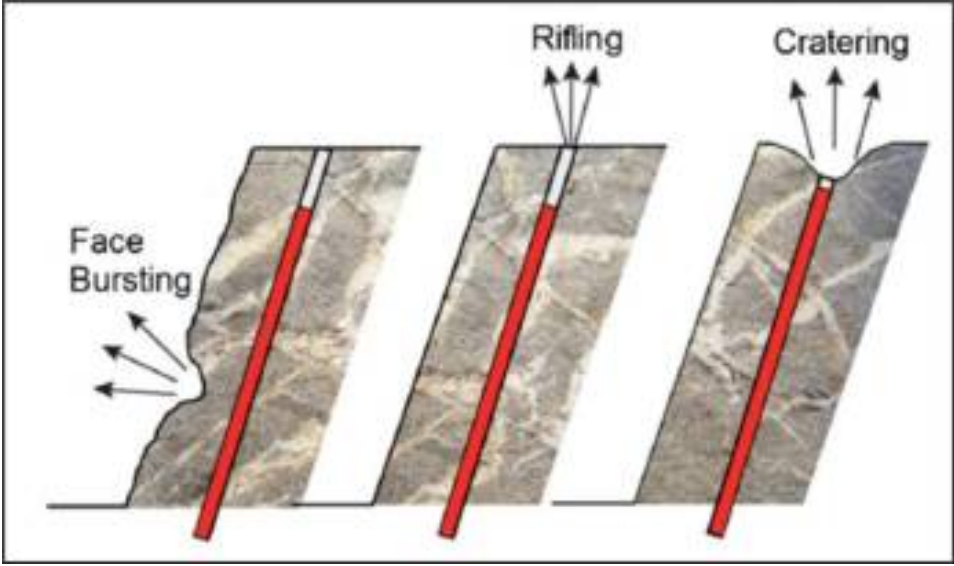


Figure 4. Mechanism of flyrock (Little 2007, Ye *et al.* 2021).

initial velocity of the ejected rock (m/s), θ is the flyrock projection angle, and g is the acceleration due to gravity (m/s²).

$$F_{dis} = v_0^2 \times \frac{\sin 2\theta}{g} \quad (4)$$

3.2 Empirical and semi-empirical flyrock models

To estimate flyrock distances, Gupta (1990) developed a semi-empirical relationship between stemming length and burden with flyrock distance (Equation 5).

$$\frac{T}{B} = 155.2 \times F_{dis}^{-1.87} \quad (5)$$

where, T represents the stemming length (m), B is the burden (m), and F_{dis} denotes the flyrock distance. In their study, Richard & Moore (2004) built upon the earlier work of Workman & Calder (1994) and classified flyrock events into three distinct types (Figure 4). The primary causes of flyrock throw during blasting operations are identified as face bursting, rifling, and cratering.

Face bursting occurs when there is a weakness zone in the bench face to be blasted. The high-temperature and high-pressure gases generated by the detonation exert pressure on these zones, forcing them to fracture. If sufficient resistance cannot be provided, part of the explosive energy is expended as flyrock and air blast (Hudaverdi &

Agan 2023). Stemming is applied at the collar of the blast hole after the main charge and aims to trap the explosive gases, thereby preventing energy loss. The appropriate size of the stemming material should be about one-tenth of the blast hole diameter (Olofsson 1990). Rifling occurs when the stemming is insufficient, missing, or made of unsuitable materials. In this case, the high-pressure gases generated by the explosion escape through the path of least resistance into the atmosphere (Ghasemi *et al.* 2012). Cratering occurs at the upper portion of the blast hole, where the rock structure is often fractured and cracked. The explosive gases penetrate these discontinuities, causing the upper part of the hole to expand and fragment, which leads to flyrock incidents. Prediction models for the types of flyrock mentioned above are given in Equations 6-8:

$$\text{Face burst: } F_{dis} = \frac{K}{g} \left[\frac{\sqrt{Q}}{B} \right]^{2.6} \quad (6)$$

$$\text{Rifling: } F_{dis} = \frac{K}{g} \left[\frac{\sqrt{Q}}{T} \right]^{2.6} \times \sin 2\alpha \quad (7)$$

$$\text{Cratering: } F_{dis} = \frac{K}{g} \left[\frac{\sqrt{Q}}{T} \right]^{2.6} \quad (8)$$

Here, α is the blasthole angle (degrees), F_{dis} is the flyrock distance (m), ql is the amount of

explosive (kg/m), B is the burden (m), T is the stemming length (m), g is the gravitational acceleration (m/s^2), and k is a constant. This study was carried out by observing face burst blasts. In addition to these models, various other prediction models have also been developed by many researchers in the literature.

3.3 The prediction of flyrock distance with artificial intelligence algorithms

In recent years, artificial intelligence techniques, which have been increasingly used across many disciplines, have also started to be applied for predicting flyrock distances—one of the blast-induced environmental effects. These techniques, known for their flexibility and high learning capabilities, can provide better prediction performance compared to traditional methods. High prediction accuracy can be achieved using methods such as Artificial Neural Networks (ANN), Support Vector Machines (SVM), Decision Trees (DT), and Random Forests (RF). Non-linear relationships can be easily modeled using these techniques. Calculations that are difficult and time-consuming to predict can be performed in a short amount of time.

During the modeling process, many independent parameters can be used. Moreover, these techniques are not limited to regression modeling—they can also be used for classification and optimisation purposes. In this way, the goal is to improve prediction accuracy.

4 LITERATURE REVIEW SUMMARY

In addition to the continued frequent use of empirical and semi-empirical techniques, many researchers today have focused on estimating flyrock distance using machine learning methods. Ghasemi *et al.* (2012) used an empirical approach combined with Monte Carlo simulation to predict flyrock distances. Their study utilised data from 150 blasting events and seven input parameters. Amini *et al.* (2012) developed Support Vector Machine (SVM) and Artificial Neural Network (ANN) models for the same purpose. The input parameters used included hole diameter, hole length, spacing, bench height, stemming, specific charge, and specific drilling. The performance of the models was compared based on the Root Mean Square Error (RMSE) criterion. Khandelwal & Manjei (2013) stated that traditional methods are insufficient for estimating flyrock distance, and artificial intelligence models yield higher performance. In their study, they developed an

SVM model using input parameters such as hole length, spacing, bench height, stemming, specific charge, and specific drilling. The output parameter was the flyrock distance ranging between 10 and 100 meters. Lu *et al.* (2020) employed Extreme Learning Machine (ELM) and ANN methods to predict flyrock. Their input parameters included spacing, bench height, specific charge, stemming, and rock density. In a more recent study, Hüdaverdi & Ağan (2023) developed a flyrock prediction equation based on measurements taken from an aggregate quarry. The measurements were conducted on low and medium-height benches with a hole diameter of 89 mm. The input parameters included the bench height to hole diameter ratio, in-situ block size of the rock (X_b), and specific charge (Pf).

5 MODEL DEVELOPMENT AND PERFORMANCE EVALUATION

In this study, a total of 111 blasting events were monitored. After each blast, flyrock distances were measured using GPS devices. For modeling purposes, 70% of the dataset (77 records) was allocated for training, while the remaining 30% (34 records) was used as the test set. In the model, B, S, H, T, and Pf were used as independent input variables, and the output variable was the flyrock distance (F_{dis}). The Extreme Gradient Boosting (XGBoost) technique was employed for modeling. This technique is based on the gradient boosting tree algorithm and is known to be effective in solving both regression and classification problems (Qiu *et al.* 2022). In building the model, 8 hyperparameters of the XGBoost algorithm were utilised. Hyperparameters are user-defined settings that guide the learning process of a machine learning model and can be adjusted prior to training. In this study, the model was trained using 200 estimators, and the RMSE value corresponding to this number of iterations was obtained. As the number of iterations increased, the RMSE value showed a decreasing trend (Figure 5).

After the training and testing phases, a coefficient of determination (R^2) plot was generated to evaluate the model's performance in predicting the test data. As shown in Figure 6, the R^2 value between the predicted and measured flyrock distances is approximately 0.95. During the training phase, the R^2 value was found to be around 0.98. The fact that the model achieves similar performance on both training and test datasets suggests that the risk of overfitting is low.

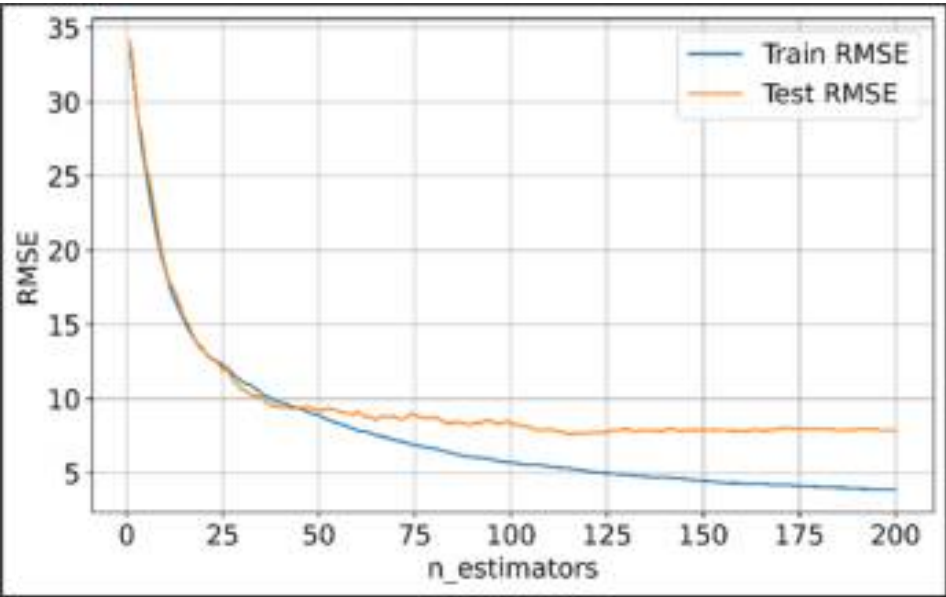


Figure 6. Training and test RMSE trends.

The model performance was evaluated not only using the R^2 value but also based on MAE, RMSE, MASE, and RMSSE criteria. During both the training and testing phases, the model's performance was assessed separately for each of these five metrics (Table 2).

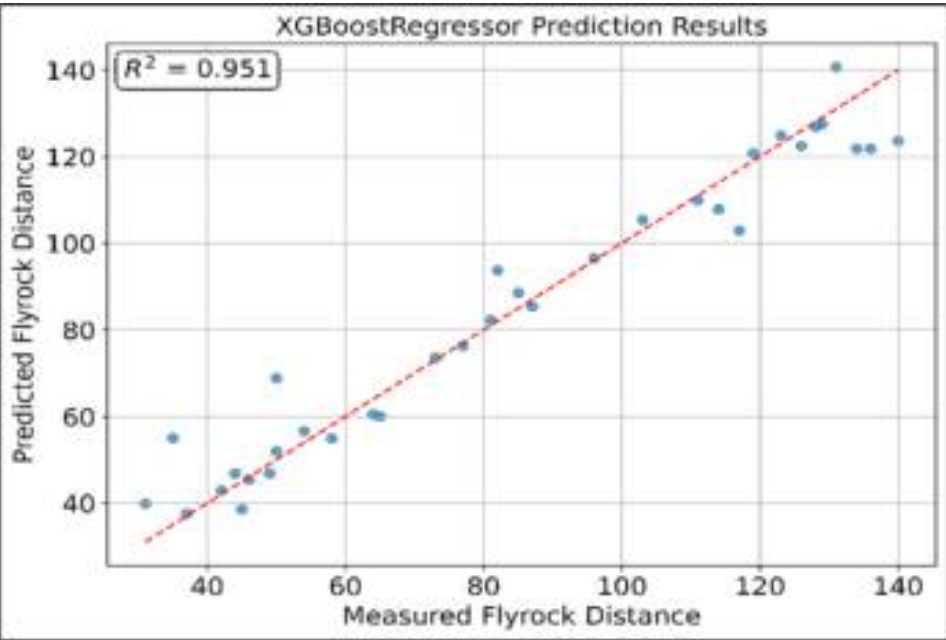


Figure 5. Measured vs predicted flyrock distance.

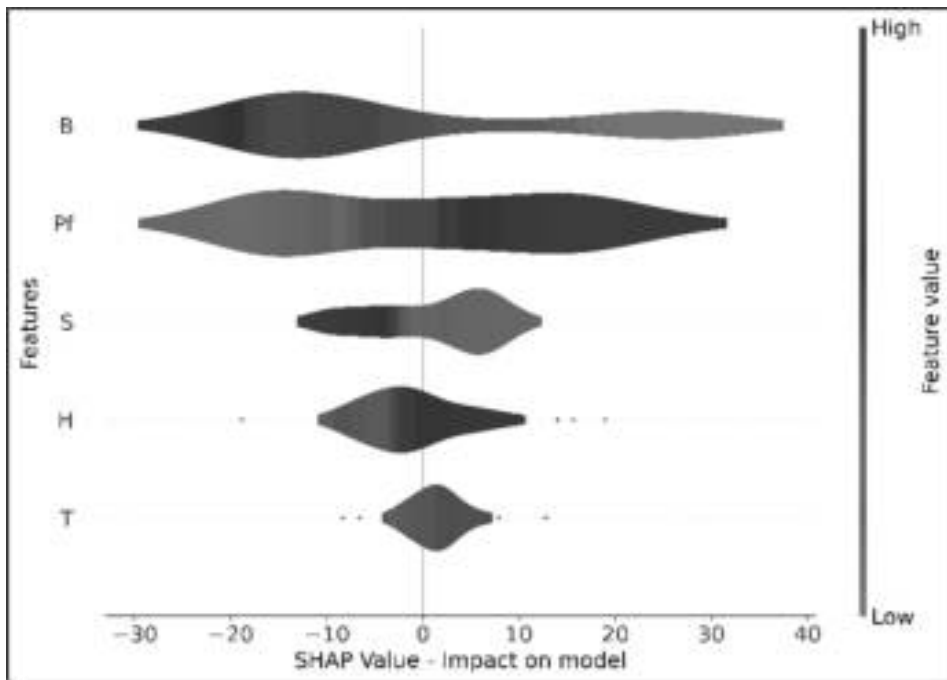


Figure 7. Interpretability of flyrock distance model.

Additionally, a SHAP diagram was generated in the model to illustrate the impact of each feature on the prediction (Figure 7). The SHAP diagram is of significant importance in terms of model interpretability. The horizontal axis shows the Shapley values, while the vertical axis displays the features. The color of the points represents the magnitude of the feature values, with dark red (darkest shades) indicating high values and dark blue (lighter shades) indicating low values. According to the diagram, the features that most

influence the model's predictions are burden and Pf. The features contributing the least to the predictions, as indicated by their narrow distribution, are S, H, and T.

6 SUMMARY AND CONCLUSION

In this study, flyrock distances were measured in a quarry located in the Kemerburgaz region of Istanbul. Classical models used for predicting flyrock distances were introduced. Additionally, a frequently used machine learning technique for predicting flyrock distances was highlighted, and a brief literature review was presented. Using the collected data, a model was developed with the XGBoost technique, which is one of the machine learning methods. To evaluate the performance of the developed model, five different error metrics were applied. In the test phase, the model's MAE, RMSE, MASE, and RMSSE values were 0.95, 5.38, 7.82, 0.17, and 0.20, respectively. The burden has the widest distribution regarding SHAP values in the SHAP diagram. High B values (red/darkest region) contribute to negative predictions by the model, while low values (blue/lighter region) support positive predictions. For the Pf feature, high values (red/dark region)

Table 2. The performance criteria of the XGBoost model.

Error Criteria	Train	Test
R^2	0.98	0.95
MAE	2.55	5.38
RMSE	3.88	7.82
MASE	0.08	0.17
RMSSE	0.11	0.20

are concentrated at positive SHAP values. High P_f values contribute to positive predictions, while low P_f values lead to negative predictions. The S feature also exhibits similar characteristics to B. The other features also have a significant impact on the model's output.

Machine learning techniques provide significant advantages in predicting flyrock distances. In this way, the safety of personnel and equipment can be ensured. Furthermore, by using various optimisation techniques, the boundaries of safe zones can be more precisely defined.

REFERENCES

- Alvarado, M., Gonzalez, F., Fletcher, A. & Doshi, A. 2015. Towards the development of a low-cost airborne sensing system to monitor dust particles after blasting at open-pit mine sites. *Sensors*, 15(8), 19667-19687.
- Amini, H., Gholami, R., Monjezi, M., Torabi, S. R. & Zadhesh, J. 2012. Evaluation of flyrock phenomenon due to blasting operation by support vector machine. *Neural Computing and Applications*, 21, 2077-2085.
- Bansah, N.K., Arko-Gyimah, K.J., Kansake, K., Dumakor-Dupey, B.A. 2016. Mitigating blast vibration impact. In *Proceedings of the 4th UMaT Biennial International Mining and Mineral Conference*, Tarkwa, Ghana, 3-6 August 2016; pp. 30-36.
- Duvall, W.I. & Petkof, B. 1959. Spherical propagation of explosion-generated strain pulses in rock. (No. 5481-5485). *US Department of the Interior, Bureau of Mines*.
- Ghasemi, E., Sari, M. & Ataei, M. 2012. Development of an empirical model for predicting the effects of controllable blasting parameters on flyrock distance in surface mines. *International Journal of Rock Mechanics and Mining Sciences*, 52, 163-170.
- Gibson, M.F.L. & St George, J.D. 2001. Implications of flyrock associated with blasting in urban areas. *AusIMM NZ Branch Conference*, 10p.
- Gupta, R.N. 1990. Surface blasting and its impact on environment. In R. K. Trivedy & M. P. Sinha (Eds.), *Impact of mining on environment* (pp. 23-24). Ashish Publishing House.
- Hasanipanah, M., Armaghani, D.J., Khamesi, H., Amnieh, H.B. & Ghoraba, S. 2016. Several non-linear models in estimating airoverpressure resulting from mine blasting. *Eng. Comput.* 2016, 32, 441-455.
- Hudaverdi, T. & Agan, Y. 2023. Application of intelligent models for flyrock prediction considering design parameters and bench face characteristics. *Mining, Metallurgy & Exploration*, 40(6), 2331-2347.
- Hudaverdi, T., Kulatilake, P.H.S.W. & Kuzu, C. 2011. Prediction of blast fragmentation using multivariate analysis procedures. *International Journal for Numerical and Analytical Methods in Geomechanics*, 35(12), 1318-1333.
- KG, S. & Zienkiewicz, O.C. 1968. *Rock mechanics in engineering practice* (pp. 21-124). John Wiley.
- Langefors, U. & Kihlstrom, B. 1963. *The Modern Technique of Rock Blasting*. New York: Wiley Publisher, Sf. 405.
- Lemesh, N. I. & Pozdnyakov, B. V. 1972. The kinematics of motion of ledge rock in the zone of fracture during blasting. *Soviet Mining*, 8(4), 388-391.
- Little T.N. 2007. Flyrock risk. In: *Proceedings of Expo 2007, AusIMM Wollongong*, Australia, pp 35-43.
- Manoj, K., & Monjezi, M. (2013). Prediction of flyrock in open pit blasting operation using machine learning method. *International Journal of Mining Science and Technology*, 23(3), 313-316.
- Monjezi, M. Ghafurikalajahi, M. & Bahrami, A. 2011. Prediction of blast-induced ground vibration using artificial neural networks. *Tunnelling and underground space technology*, 26(1), 46-50.
- Nguyen, H. & Bui, X.-N. 2020. Soft computing models for predicting blast-induced air over-pressure: A novel artificial intelligence approach. *Appl. Soft Comput.* 2020, 92, 106292.
- Raina, A.K. 2023. *Flyrock in Surface Mining: Origin, Prediction, and Control*. CRC Press.

- Roy, P.P. 1991. Vibration control in an opencast mine based on improved blast vibration predictors. *Mining Science and Technology*, 12(2), 157-165.
- Siskind, D.E., Stagg, M.S., Kopp, J.W. & Dowding, C.H. 1980. Structure response and damage produced by ground vibration from surface mine blasting; *US Department of the Interior, Bureau of Mines*: New York, NY, USA, 1980.
- Standard, I. 1973. Criteria for safety and design of structures subjected to under ground blast. *ISI, IS-6922*, (1973).
- Tugrul, A. & Undul, O. 2006. Engineering geological characteristics of Istanbul greywackes. In: *Proceedings of the 10th International Association for Engineering Geology and the Environment (IAEG) Congress Nottingham*, United Kingdom. Paper no. 395.
- Vaupel, K., Klenk, U. & Schmidt, E. 2016. Emissions from open pit mines-a challenge for air dispersion modeling. *Gefahrstoffe Reinhaltung der Luft*, 76(1-2), 14-18.
- Wang, Z.M., Zhou, W., Jiskani, I.M., Ding, X.H., Liu, Z.C., Qiao, Y. Z. & Luan, B. 2025. Dust reduction method based on water infusion blasting in open-pit mines: a step toward green mining. *Energy Sources, Part A: Recovery, Utilization, and Environmental Effects*, 47(1), 6029-6043.
- Ye, J., Koopialipoor, M., Zhou, J., Armaghani, D.J. & He, X. 2021. A novel combination of tree-based modeling and monte carlo simulation for assessing risk levels of flyrock induced by mine blasting. *Nat Resour Res* 30:225–243.
- Lu, X., Hasanipanah, M., Brindhadevi, K., Bakhshandeh Amnieh, H. & Khalafi, S. 2020. ORELM: a novel machine learning approach for prediction of flyrock in mine blasting. *Natural Resources Research*, 29, 641-654.
- Ghasemi, E., Sari, M. & Ataei, M. 2012. Development of an empirical model for predicting the effects of controllable blasting parameters on flyrock distance in surface mines. *International Journal of Rock Mechanics and Mining Sciences*, 52, 163-170.
- Qiu, Y., Zhou, J., Khandelwal, M., Yang, H., Yang, P. & Li, C. 2022. Performance evaluation of hybrid WOA-XGBoost, GWO-XGBoost and BO-XGBoost models to predict blast-induced ground vibration. *Engineering with Computers*, 38(Suppl 5), 4145-4162.

[RETURN TO CONTENTS PAGE](#)

A pathway to detonation of ammonium nitrate due to heat energy

G. Cavanough, A. Torrance & G. Stenzel

QMR, Brisbane Australia; Kilmore Consulting, Cooks Hill, Australia; Glencore Operations South Africa

ABSTRACT: Research conducted into reactive ground was not able to experimentally verify that reactive sulphide is a catalyst or contributor to the detonation of bulk explosives apart from being a source of heat from the exothermic chemical reaction of sulphide, contained in reactive ground, and nitrates contained in bulk explosive. It was demonstrated that ammonium nitrate (AN) emulsion (ANE) loaded into a lined hot blast hole will detonate due to heat alone. During the project a local incident occurred where a tanker of ANE detonated due to the resulting fire from a collision. The accepted risk of ANE detonating due to events such as this was assumed to be negligible due to the high-water content and lack of sensitisation. This incident motivated the authors to explore the role of heat energy in the causation of this detonation. A review of literature identified that the pathway to heat induced detonation of AN is essentially unknown with there being more than 80 proposed AN decomposition reactions. Experimental results and review of incidences of spontaneous detonation identified that: decomposition of ammonium nitrate always occurs before detonation, the heat source must be at the top of the explosive column for detonation to occur, and high levels of heat energy are required. A pathway to detonation was postulated whereby decomposition of ammonium nitrate results in ammonia. The ammonia is then cracked at high temperature to produce hydrogen. The reaction does not produce oxygen, so the hydrogen accumulates until exposed to air. Hydrogen in air has an autoignition temperature of approximately 600°C and ignition of the hydrogen leads to detonation of the AN. This pathway to detonation was experimentally verified by both detection of hydrogen and detonation.

1 BACKGROUND

Numerous larger-scale experiments have been conducted in an attempt achieve detonation of AN including:

- By small diameter impact of AN prill at temperatures between 180 to 249°C with a projectile traveling at 1500 m/sec with no resulting detonation (Van Dolah *et al.* 1966).
- AN with GMB sensitisation and coke at 180 to 270°C subject to 50 kg lead drop 11.5 m. The conclusion being that molten AN detonation is unlikely from impact in a collapsing building (Groothuizen & Heemskerk 1988).
- Paper lined steel drums of approximately 70 tonnes of pure AN heated by burning 5 tonnes of chopped wood and 10 tonnes of AN with no evidence of explosion.
- Medium scale drop hammer tests with no resulting detonation (Millot & Languy 2004) (refer to Table 1).
- Burns *et al.* 1953 did manage to detonate chemically pure ammonium nitrate (CPAN)

Table 1. Drop hammer impact tests (Shah 2022).

NO.	EQUIPMENT	TEST MATERIAL	OBSERVATION
1	Drop hammer in trough	Granular AN in a wood fire	No detonation and no self-sustaining decomposition
2	30 kg drop hammer at 7 m, in trough 11.7 m/s at impact	Mixture of AN (20 kg) + 5% microspheres at T>190°C	No detonation
3	30 kg drop hammer at 7 m, in trough	Mixture of AN (20 kg) + 5% microspheres + 12 % sawdust at T>190°C	No detonation Fire in sawdust
4	60 kg flat-bottomed drop hammer at 15 m, in trough. Speed about 17 m/s at impact	Mixture of 30 kg AN+ microspheres T>190°C	No detonation
5	60 kg flat-bottomed drop hammer at 15 m, in trough. Speed about 17 m/s at impact	Mixture of 30 kg AN microspheres + sawdust at T>190°C	No detonation Fire in sawdust
6	A 2 kg projectile from a tank explosion, V≈ 80 m/s	Mixture of AN+ microspheres + sawdust T>190°C (doubled test)	No detonation
7	A 2 kg projectile from a tank explosion, V≈ 150 m/s	Mixture of AN+ microspheres + sawdust T>190°C	Detonation

during tests conducted at pressures of 2600-3000 psi and temperatures of 114 to 344°C.

- Shah (2022) concluded fertiliser grade ammonium nitrate (FGAN) would require a substantial degree of confinement for detonation (see Table 1).

The University of Kingston Researchers did manage to detonate ammonium nitrate at 0.8 density at a temperature of 260°C on one occasion 100 % using a 7.2 kg slug to impact at 19 0 m/sec. (Bauer 2015).

Figure 1 shows a typical thermogravimetric analysis (TGA) study, which in this instance, was used to develop a proposed reduction in ignition temperature of ammonium nitrate due to rust (Oluwoye 2020).

TGA studies and isothermal test, similar to the method described in the ASIEG Code or Practice for Elevated Temperature and Reactive Ground, have identified the presence of sulphide lowering the ignition temperature, i.e. a rapid violent decomposition of AN (Anon 2020, Rumball 1991, Zhi-Xiang 2015).

Isothermal testing has been used to investigate additives to inhibit the reactions (Miron *et al.* 1982, Tunaley 1999).

Priyananda *et al.* 2014 concluded that the bubbles generated by the reaction of AN and pyrite reduce the thermal conductivity and hence increase the reaction temperature.

Gunawan 2009 treated ammonium nitrate as an ideal explosive and deduced that the presence of pyrite reduced the critical temperature.

The vented pipe test is used to test bulk explosives as part of the UN Recommendations on the Transportation of Dangerous Goods (2023). It has been identified that results from this test are not repeatable (Anon 2025). Kennedy (2003) conducted a Review of the Modified Vented Pipe Test and hypothesised the reactions and events within the pipe based on the temperature recordings.

Cavanough & Torrance (2025) identified that thermal conductivity model as described by Priyananda *et al.* (2014) is not supported by experimental data. The literature based on TGA,

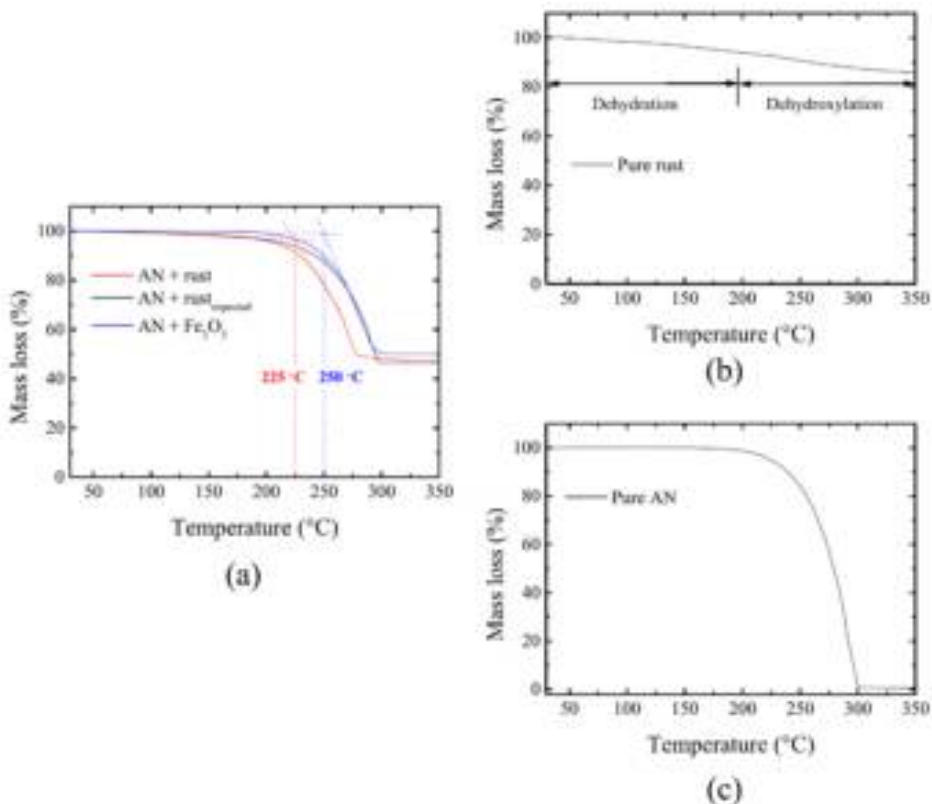


Figure 1. TGA profiles of the AN-rust mixture as compared to the expected (and AN-Fe₂O₃mixture) plots. Heating rate equals 20 °C/min (Oluwoye *et al.* 2020).

isothermal studies and the proposed reactions occurring the vented pipe test are based on a chemical pathway to the ignition temperature of AN under confinement.

Previous experiments have identified that ammonia and hydrogen are produced when AN decomposes (Babrauskas & Legget 2019, Dolah, R., Mason, C., Perzak, F., Hay, J. & Forshey, D. 1966)

The production of hydrogen from ammonia as the pressure increases (i.e. confinement) hydrogen conversion requires a higher temperature. (Trangwachirachai *et al.* 2024).

2 ROLE OF IGNITION TEMPERATURE & CONFINEMENT

Cavanough, Torrance & Stenzel (2024) reported the detonation of lined, unstemmed blast holes containing more than 750kg of inhibited emulsion in hot ground (refer to Figure 2). it was noted that

the spontaneous detonations occurred in a much shorter time period when the heat source was at the top of the hole. When the heat source was at the bottom of the hole, slumping and bubbling occurred before detonation. The mine site where this test was carried out had been deemed reactive hence the use of inhibited products. These experiments confirm that detonation is due to heat only and there is no requirement for contaminants, confinement, sulphide and impact to induce detonation.

These experiments also demonstrate that there is a continued risk when sites have an event that is due to hot ground being incorrectly attributed to reactive ground. The use of inhibited product will not prevent future similar incidents.

All previous studies using TGA, isothermal studies, bulk tests and the vented pipe tests have been used to deduce chemical equations to describe the path to detonation based on reactions at less than 350°C. This is logical as above this

temperature the ammonium nitrate has completely decomposed (refer to Figure 1).

It was not experimentally possible to identify an ignition temperature. When the bulk of ammonium nitrate exceeds a temperature of 200°C 'fume off' occurs and the entire mass decomposes to a white vapour.



Figure 2. Spontaneous detonation of inhibited emulsion in lined hot hole.

Experiments using ammonium nitrate mixed with reactive ground samples were raised to temperatures > 1000°C. There were no indications of detonation or deflagration. Therefore, it can be concluded that there is no ignition temperature, no requirement for confinement and no chemical pathway for explosive decomposition of ammonium nitrate.

3 SURFACE REACTIONS OF AN & ANE

Cavanough & Torrance (2025) proposed that detonation is due to ignition of hydrogen.

Experiments were conducted to establish if there is an ignition temperature and if detonations due to reactive ground are also due to hydrogen.

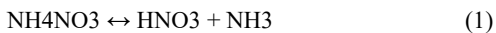
An empty pan was heated to a very high temperature and AN prill were slowly poured onto the hot surface and immediately decomposed.

A lid was placed on CPAN in a vessel which were then heated to a stage where loud boiling sounds occurred but no popping due to hydrogen ignition. The lid eventually was pushed off by

pressure build up and immediately the popping due to hydrogen ignition occurred due to contact with oxygen.

This indicates that the reaction does produce hydrogen and not oxygen. This is in agreement with the loaded blast holes that detonated in a shorter time when the heat source was higher in the explosive column.

A possible pathway to the hydrogen is proposed as ammonia cracking:



The experiment was repeated using HDAN and the reaction was quenched and the white material recovered and compared to the original HDAN.

Table 2 shows the HDAN and the white material are essentially ammonium nitrate with the white material being more acidic and having a waxy cleavage. The results of this experiment and previous experiments indicate the HDAN breaks into nitric acid (or breakdown products of nitric acid) and ammonium when subject to high levels of heat energy (i.e. the white material). The ammonia cracks producing hydrogen and nitrogen. On cooling ammonium nitrate is formed with increased acidity possibly due to the loss of ammonia.

This process was repeated ANE, with the ANE relative unaffected just below the reaction surface.

Both AN and ANE decomposition to hydrogen is due to the Leidenfrost effect whereby the gases at the surface decomposition thermally insulate the bulk of the mass.

4 ADDITIONAL ENERGIES FROM REACTIVE GROUND

Milled pyrite, (ammonium nitrate and fuel oil) ANFO and sulfuric acid were placed in a double insulated stainless-steel vessel with the outer layer removed at the bottom to allow heating with a hot plate. The hot plate temperature was set at 165°C. For all experiments using ANFO, the energy from the reaction resulted in the temperature reaching > 500°C, cracking off hydrogen (refer to Figure 3).

It was concluded that the pathway to detonation of ANFO for reactive ground is the same as for hot ground. Hydrogen produced by the decomposition of ammonia builds up and detonates at high temperature.

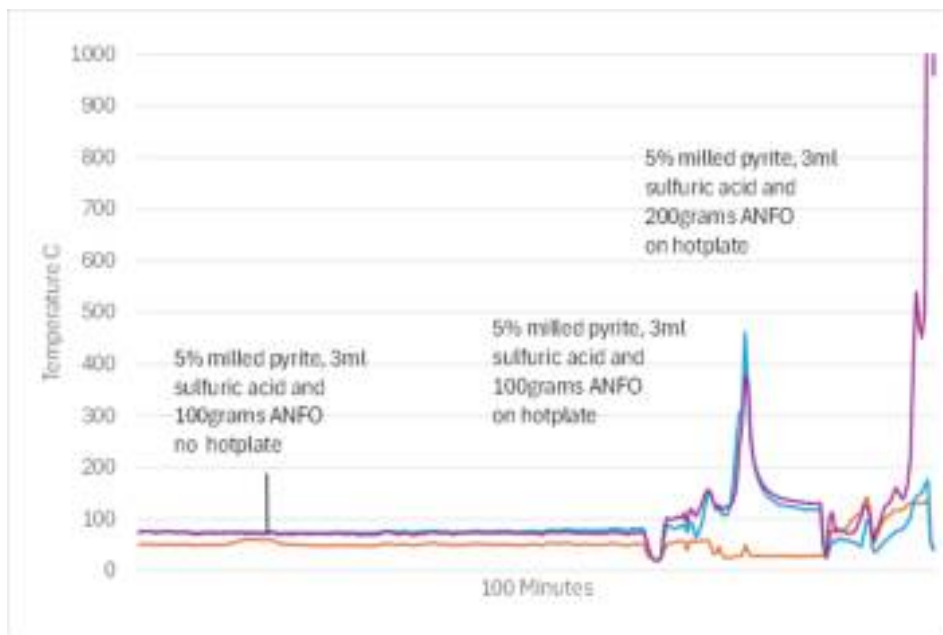


Figure 3. Continuous reaction - high temperature reaction when sample heated for 5 grams of milled pyrite increased for additional ANFO.

5 DETONATIONS

Stainless steel containers were attached to 2.7 and 3.5 m chimney sections. Each test used 1.5kg of

unsensitised ANE and was heated with gas heaters (refer to Figure 4).

The tests with 2 chimney sections resulted in a loud burst of flame and shock was observed



Figure 4. Heating arrangement.



Figure 5. Apparatus with 2 chimney sections gave flame.

travelling downwards (refer to Figure 5). Large amounts of gas were ejected from the lower chimney joins.



Figure 6. Apparatus with 4 chimney sections detonation.

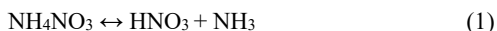
Two additional chimney sections were added to give an additional approximate 14 litres volume for the accumulation of hydrogen. Detonation did occur vaporising the stainless-steel vessel (refer to Figure 6).

Just after heating commenced for the second chimney tests a crack was observed in the stainless-steel vessel so the experiment was terminated. When the chimneys were disconnected, they were full of dense white gas and retained the gas on disassembling. It is possible the white gas acts like a cork and contains the hydrogen.

Examination of the recovered chimney sections indicates hydrogen detonation in the chimney and also shows deformation from a shock wave from the detonation of the ammonium nitrate (refer to Figure 7).

6 CONCLUSIONS

The pathway to detonation of ammonium nitrate due to heat is decomposition of ammonium nitrate to ammonia gas and nitric acid:



Ammonia has much higher vapour pressure than nitric acid and the melt self-acidifies increasing the heat of reaction with any sulphide at temperature $> 150^\circ\text{C}$.

Cracking off hydrogen from the ammonia gas due to temperature:



The ammonia gas cracks off hydrogen at approximately 500°C and the hydrogen accumulates under the dense white cloud produced by lower temperature decomposition products. Ignition and/or detonation of hydrogen occurs when sufficient hydrogen accumulates. This causes shock initiation of ammonium nitrate that has been sensitised by hydrogen gas and heat induced violent mixing (refer to Figure 10).

Low levels of sulphide reacting with nitrates at elevated temperature in a low thermal loss environment can provide sufficient heat to produce hydrogen. (Refer to Figure 9).

As a result of continuing reactive ground instances at mine sites the following risk matrices were developed to highlight that the QMR Reactive Ground test overcomes the inherent risk with using the AEISG Method:



Figure 7. Recovered chimney sections show hydrogen detonation of hydrogen in chimney and it appears there is deformation from the shock of the ammonium nitrate detonating.

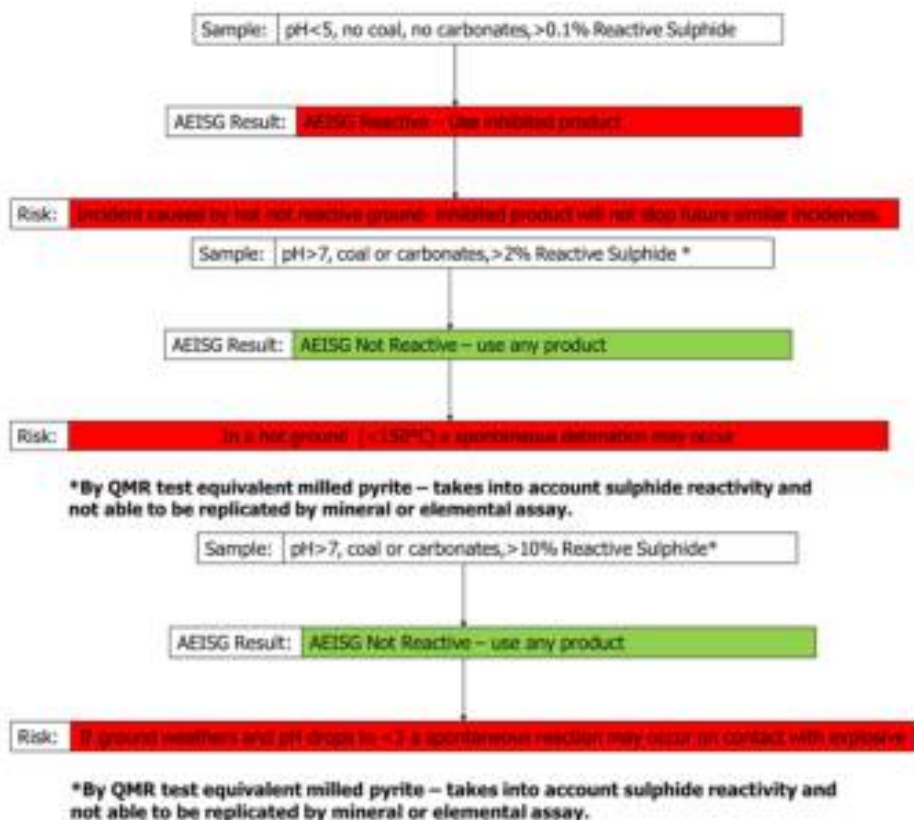


Figure 8. Summary.

Reactive and Hot Ground Risk Matrix			
pH <3	3 < pH < 7.5	pH > 7.5	
			<55 °C Hole Temperature
			55 °C < Hole Temperature < 100 °C
			100 °C < Hole Temperature
			500 °C < Hole Temperature
Including inhibited			Extreme Risk = Spontaneous Detonation of Bulk Product
Including inhibited			Extreme Risk = Spontaneous Detonation of Bulk Product if >2% reactive sulphide
			High Risk = Spontaneous Initiation of detonator if >5% reactive sulphide
			Moderate Risk = Weathering with generate acid if > 10% reactive sulphide
			Low Risk

Figure 9. Reactive and hot ground risk matrix.

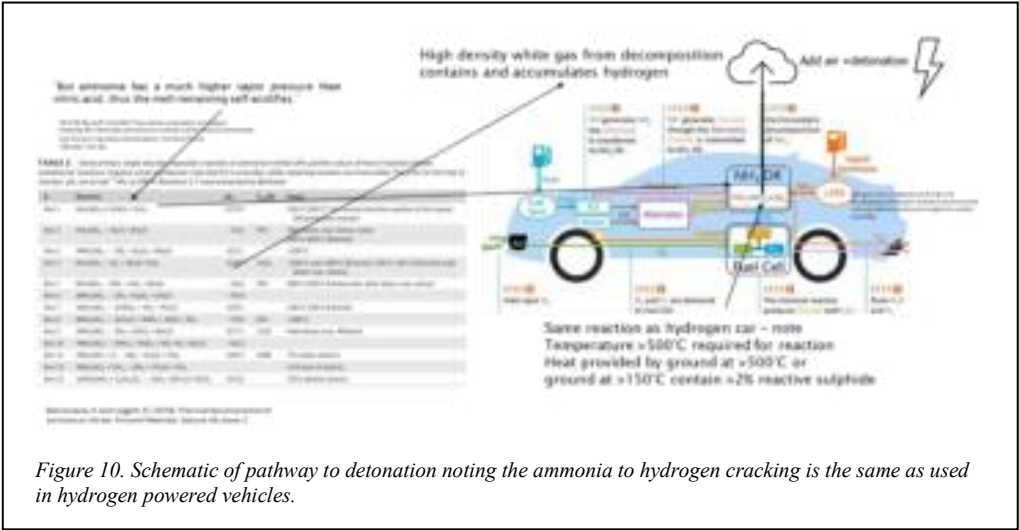


Figure 10. Schematic of pathway to detonation noting the ammonia to hydrogen cracking is the same as used in hydrogen powered vehicles.

REFERENCES

ARIA 2010. Fire inside a barn and explosion of fertiliser, 2 October 2003, Saint Romain-en-Jarez (Loire), France. *French Ministry for Sustainable Development - DGPR/SRT /SDRA/BARPI*, No. 25669.

Annon 2020. Code of practice elevated temperature and reactive ground. *Australia Explosives Industry and Safety Group Inc.* AEISG.

Anon 2025. Personal communications with participants of the *EPP Working Group of IGUS International Conference of Chief Inspectors of Explosives* Perth Australia March 2025.

Anon 2022. Weekly incident summary. *NSW Resources Regulator*. Week ending Friday 28 January 2022.

Babrauskas, V. & Legger D. 2019. Thermal decomposition of ammonium nitrate. *Fire and Materials* Wiley pp250-268.

Bauer, A. 2015. *SAFES Newsletter NO53* June 2015. Citing: Bauer, A. & King, A, 1979. Critical initiation parameters for molten ammonium nitrate. Kingston: Canadian Fertiliser Institute. And Shah, K. 2009. ANNA (ammonium nitrate and nitric acid). *Summary of the work at Queens University by Bauer King & Heater*. Little Rock AR: ANNA.

Briggs, T. & Kelso, I. 2001. Ammonium nitrate-sulphide reactivity at the Century Zn-Pb-Ag Mine, Northwest Queensland, Australia. *Explor. Mining Geol.*, Vol. 10, No. 3, 2001.

Brill, T.B., Brush P.J.& Patil D.G. 1993. Thermal decomposition of energetic materials 58. Chemistry of ammonium nitrate and ammonium dinitramide near the burning surface temperature. *Combust Flame*.1993;92:178-186.

Cavanough, G. 2023. A new method of reactive ground testing. *Cleveland: ISEE*.

Cavanough, G., Torrance, A. & Stenzel, G. 2025. The thermal basis for explosive decomposition with heat due to external heating or reactive ground. *Cleveland: ISEE*.

Chief Dangerous Goods Office 2022. Ammonium nitrate emulsion tanker trailer explosion. Incident Alert. *Government of Western Australia Department of Mines, Industry Regulation and Safety*.

Downs, G. 2014. Ammonium nitrate explodes during transport incident. Explosives safety alert no. 86 | 09 October 2014 | Version 1, *Queensland Department of Natural Resources and Mines*.

Elliot, M.A. 1948. Exothermal decomposition of mixtures containing ammonium nitrate. *United States Department of Interior Bureau of Mines*.

- Gunawan, R. & Zhang, D. 2009. Thermal stability and kinetics of decomposition of ammonium nitrate in the presence of pyrite. *Journal of Hazardous Material* 165 (pp. 751-758). Elsevier.
- Gunsinger, M.R., Ptacek, C.J., Blowes, D.W. & Jambor, J.L. 2006. Evaluation of long-term sulphide oxidation processes within pyrrhotite-rich tailoring, Lyn Lake, Manitoba. *Contaminant Hydrology* 83, 2 149-170.
- Han X., Freij S., Feng S., Gunwaan R., Zhang D. & Waters, P. 2005. A chemical characterisation of pyritic black shales in an iron ore mine. Fremantle, WA, 19 - 21 September 2005 *Iron Ore Conference*.
- Harries, G., Bellairs, P., Scott-Stewart, J. 1986. The reaction of ammonium nitrate with black pyritic shale at Mt Whaleback. *AusIMM/IE Large Open Pit Mining Conference* 141-146.
- Kennedy, B. & Tyson, N. 2001. Blasting in reactive ground. *Proceedings EXPLO 2001*, p55-61.
- Kiiski, H. 2009. Properties of ammonium nitrate based fertilisers. Dissertation for the Degree of Doctor Philosophiae. *University of Helsinki, Faculty of Science, Department of Chemistry Helsinki Finland*
- Littlefair, M., Rounsley, R., Beach, F., Jassini, D. & Pugh, G. 2003. Reactive ground and ammonium nitrate explosive interactions. *Fifth Large Open Pit Mining Conference*. Kalgoorlie WA 3-5 November.
- Lukaszewski, G. 1968. The reaction of ANFO explosives with mineral sulfides. *Proc.Aust.Inst. Min.Met* 228, December.
- Olson, D.B., Banks, M.L., Maxwell, D. & Schellhorn, J. 2002. Detonability of ammonium nitrate solutions. *Baker Engineering and Risk Consultants Inc, (unpublished)*.
- Marlair, J.G. & Kordek, M.A. 2005. Safety and security issues relating to low capacity storage of AN-based fertilisers, *J. Hazard. Materials*, A123 13-28.
- Mason, C.M., Ribovoch, J. & Van-Dolah, R.W. 1963. Studies of the bullet sensitivity of ammonium nitrate fuel oils mixtures. *United States Department of Interior Bureau of Mines*.
- Millot, S. & Languy, J.P. 2004. Safety guide for fertiliser storage facilities made (partially or totally) of wood. *Report bychnip. Source: UNIFA*.
- Miron, et al. 1983. *Report of investigations 8727* Reactivity of ANFO with pyrite containing weathering products, evaluation of additional inhibitors.
- Oluwoye, I., Mosallanejad, S., Soubans, G., Atarawneh, M., Gore, J. & Dlugogorski, B. 2020. Thermal decomposition of ammonium nitrate on rust surface: Risk of low-temperature fire. *Fire Safety Journal*.
- Priyananda, P., Djerdjef, A., Gore, J., Neto, C., Beattie, J. & Hawket, B. 2015. Premature detonation of an NH₄NO₃ emulsion in reactive ground. *Journal of Hazardous Materials*, 283 314 – 320.
- Proulx, R. & Scott, N. 2000. Drilling and blasting in hot and reactive ground conditions at Barrick Goldstrike's Meikle Mine. *International Society of Explosives Engineers 2000G Vol. 1 Drilling & blasting in hot and reactive ground conditions*, P 319.
- Ramirsz, M., Reyes, A. & Alarcon ,A. 2018. Improvement of safety practices for loading of ANFO-based explosives in zones with reactive rocks in Chile. *ISEE*.
- Rumball, J., Thornber, M. & Davidson, L. 1986. A study of the chemical reactions leading to spontaneous combustion of pyritic black shale at Mt Whaleback, Western Australia. *The AusIMM/IE Aust Newman Combined Group. Large Open Pit Mining Conference* October 1986.
- Schumann, R., Kawashima, J.L., Miller, S., Smart, R. & Stewart, W. 2009. Passivating surface layer formation on pyrite in neutral rock drainage. *Securing the Future and 8th ICARD*, June 22-26, Skelleftea Sweden.

- Shah, K.D. 2022. Ammonium nitrate & fertiliser safety. *Kishore D. Shah*.
- Tunaley, D. 1999. An emulsion explosive system for blasting in reactive volcanic conditions. *Proceedings Fragblast, South African Institute of Mining and Metallurgy*, p313-319.
- UN Recommendations on the Transportation of Dangerous Goods*, 23rd Revised Edition, 2023.
- Valenta, R., O'Sullivan, R. & Clark, A. 2019. Prediction of reactive ground using geoscientific data sets. *Geomin. Mine planning* 2019 August 7-9 Santiago Chile.
- Van Dolah, R., Mason, C., Perzak, F., Hay, J. & Forshey, D. 1966. Explosion hazards of ammonium nitrate under fire exposure. *U.S. Bureau of Mines Report of Investigations, No. 6773*.
- Watson, R.W., Hay, J.E. & Becker, K.R. 1974. Sensitivity of some ammonium nitrate-based explosive compositions. *U.S. Bureau of Mines Report of Investigations, No. 7840*.

[RETURN TO CONTENTS PAGE](#)

Blasting at 120°C (250°F) in opencast coal mining using thermal insulating blast hole sleeves

G.J. Stenzel

Glencore Coal South Africa

ABSTRACT: In South Africa, current safety limits restrict charging blast holes above 80°C (175°F), limiting productivity in heat-affected opencast blasting. This study details the development and operational deployment of polyvinyl chloride (PVC) blast hole sleeves by Glencore Coal to safely enable blasting up to 120°C (250°F). Laboratory and in-situ testing validated the sleeves' thermal insulation and containment of emulsion moisture, preventing premature detonation. The results led to an operational threshold of 120°C, reducing uncharged holes to under 2% and recovering previously un-blasted areas. The technique has enabled the safe blasting of over 158,000 hot holes and is now the safe blasting standard for charging and blasting hot holes.

1 BACKGROUND

As the mineable coal reserves in the Witbank Coalfield of South Africa continue to deplete, re-mining shallow, previously exploited underground bord and pillar reserves has become common practice. However, due to air ingress, fires have developed in these old workings, necessitating the blasting of hot overburdens and burning coal pillars. The challenge of blasting in high-temperature environments is well-documented, with industry guidelines emphasising the risks associated with charging explosives into hot holes (Coaltech 2022).

In South Africa, explosives suppliers have imposed a safety limit of 80°C (175°F) for charging blast holes. This limitation was imposed following several self-detonation incidents, including a fatal event in 2019. This limitation has hindered blasting operations, with up to 71% of drilled holes exceeding this temperature threshold on some blasting blocks and remaining uncharged.

Conventional cooling methods, including water and hole-cooling agents, have proven ineffective, often exacerbating the problem by generating steam, which further increases the hole temperature (SAIMM 2023).

With blasting restricted to a maximum of 80°C large areas of the strip mine Pit could not be charged and blasted and this required immediate action and an innovative solution or this pit would be closed.

2 DEVELOPMENT OF A HOT HOLE BLASTING SLEEVE

In September 2020 tests were conducted to use a polyvinyl chloride (PVC) welded sleeve for use in hot and venting holes in the range 80°C - 100°C. Testing was first done using water in hot and venting holes up to 157 °C to test the integrity of the sleeves.

The sleeves are made from polyvinyl chloride (PVC) as the primary material due to its superior



Figure 1. Fires in old underground workings (left & right). This burden must be blasted (G Stenzel 2019, 2020).

strength, insulation properties, and durability. PVC's inherent thermal resistance aligns with research on its application in high-temperature industrial settings, where it effectively mitigates temperature fluctuations and condensation formation (Research Gate 2023).

Hang tests were then conducted to ensure that the strength would be able to suspend the full load using water. (More viscous and SG 1 vs SG 0.95 for bulk explosives). It was found that the PVC welds failed at the bottom and thus a knot was made that solved the problem.

Tests showed that even over a period of 4 days the temperature inside the bag remained below 80°C while the outside rock temperature was at 157°C – the insulation and cooling effect of the water that was retained inside the sleeve.

Further tests were conducted to determine the capability of the PVC welded sleeve under extreme conditions and using a gas flame at 300°C water was boiled inside the PVC sleeve which

remained intact.

These experiments confirmed that the PVC sleeve was able to protect the bulk explosive from the heat and keep the temperature below 80°C and safe to blast.

These experiments showed that PVC was a suitable material and after testing with water tests were conducted by charging explosives up to 100°C. After 6 months and with blasting around 5000 hot holes up to 100°C the temperature was increased to 110°C in 2021. At this temperature we were able to charge and blast around 90% of all holes.

During 2022 further testing was conducted in a designated approved hot hole test site. The aim of these tests was to determine what the maximum safe temperature was at which holes could be charged when using PVC sleeves.

The sleeves had proven to be highly effective and a decision was made to patent this design as a protection from the supply market so that any

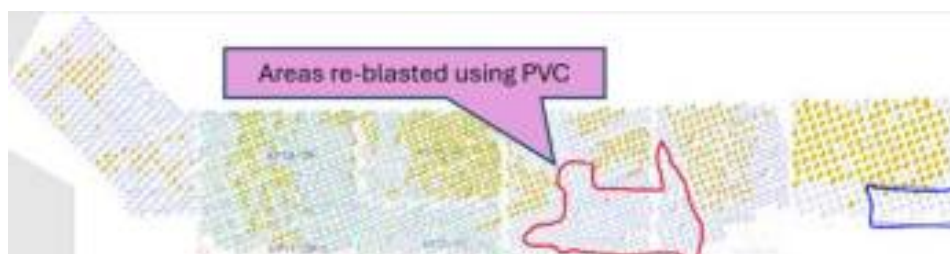


Figure 2. Showing blasting plans with up to 71% of holes in a block remaining un-blasted due to temperatures exceeding 80 °C.

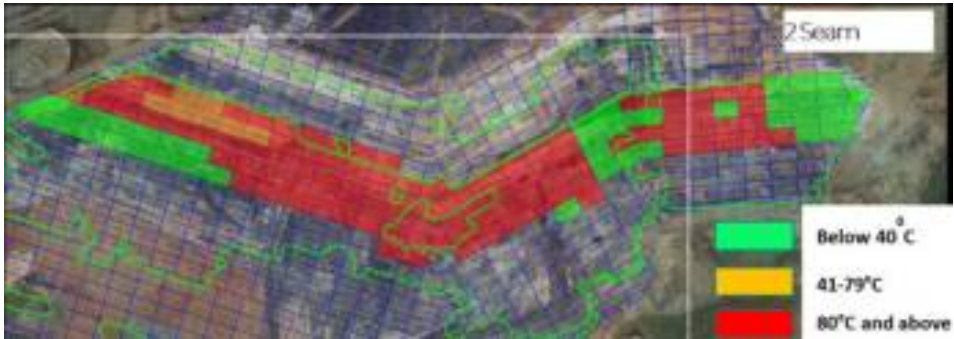


Figure 3. Showing the pit layout and heat map indicating large areas above 80°C.

mining company could use them and any supplier was able to manufacture to our specifications and thus there was available supply at the best price.

3 TESTING OF PVC SLEEVES TO DETERMINE THE MAXIMUM SAFE CHARGING TEMPERATURE

At the hot hole test site, holes with temperatures as high as 313°C were charged with bulk explosives inside the mine designed PVC sleeves. From these tests several very important findings became evident:

- The holes self-detonated when the bulk explosive temperature reached between 137°C - 141°C at the top of the explosive. In some cases, the bottom of the hole explosive temperatures exceeded 350°C inside the sleeves but only when the top of the explosive reached around 140°C did a detonation occur.
- The PVC sleeve has a thermal insulation effect of between 20°C to 60°C depending on the rock temperature where it is used.
- There is between 15-21% water in the emulsion and this protects the explosive as long as the water is kept inside the sleeve.
- When the water has boiled off (at 96°C in our mine) the explosive temperature rises rapidly and when the top of the explosive reaches around 140°C we get a detonation.
- After the water has boiled off, we always see a thick dense white smoke just prior to detonation.
- At a rock temperature of 158°C in an 18 m deep hole of 200 mm diameter it took 6 days and 22 hours to cook off the water and achieve self-detonation.
- An 18 m hole charged at 143°C was slept for 6 days and the explosives fired as normal. Figure 5 clearly shows the insulating properties of the PVC sleeve.



Figure 4. Images of PVC sleeve filled with water - strength tests, knot at bottom.

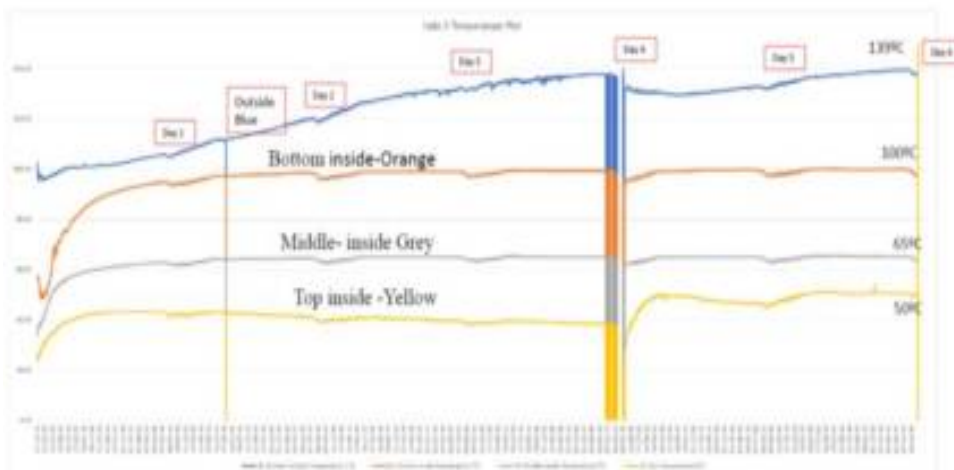


Figure 5. An 18 m x 200 mm diameter test hole charged with PVC sleeve at 143°C and slept for 6 days clearly showed the insulating properties of the PVC sleeve.

- When there is a ‘hot spot’ higher up in a hole that heats up the top of the explosive, the bottom remains cold and again as soon as the top temperature reaches around 140°C we get a self-detonation.

These tests allowed the mine to increase the maximum charging temperature in the PVC sleeves to 120 °C with the proviso that:

- Hole temperatures are measured using infra-red hot hole measuring devices that give the rock temperature accurately. (Thermocouple devices are not suitable as they give the air temperature inside the hole and do not detect ‘Hot spots’)
- A multipoint hot hole monitor is inserted into the sleeve that continuously measures the explosive temperature after charging at 5 m intervals from the bottom to the top of the sleeve.
- All holes are charged and blasted on the same shift, no sleepover blasts allowed.
- Holes are top primed with a booster and detonating cord, and timing is on surface.
- Holes are not stemmed so that it allows you to see the white smoke. Where infrastructure is in proximity of the blast, stemming is permitted under strict management supervision.
- A person is appointed to act as a ‘spotter’ to look for white smoke from charged holes and to look and listen for alarms from the Multipoint Blast eyes.

4 APPLICATION OF SLEEVES

As the most common source of heat is from the burning underground coal workings the PVC sleeve is inserted into the hole prior to charging and is suspended 1 m from the bottom of the hole. The sleeve is suspended using nylon ski rope through eyelets at the top of the sleeve and a stick is placed across the hole to keep it in place. The design of the sleeve is such that it exceeds the hole diameter by 30-40 mm and as soon as the explosive bulk is pumped into the sleeve, it is supported by the friction on the sides and not the ski rope. A typical schematic of a charged hole with PVC sleeve is depicted in Figure 6.

When there are underground coal pillars the holes are drilled to the floor of the pillar, but the sleeves are suspended at the burden/ coal interface and the pillar is blasted with the concussion. For holes above the roadways the holes are drilled to 2 m from the roof and sleeves are suspended a further 1 m from the bottom of the holes (Figure 7).

When the sleeve is charged a multi-point temperature measuring device cable (Multipoint Blast eye cable) is lowered into the sleeve and pushed down with the charging hose (Figure 9 left). This has sensors every 5 m and has a reader unit (Figure 9 right) that has both visual and audible alarms. When the boosters are dropped into the charged holes the reader units are recovered for reuse.

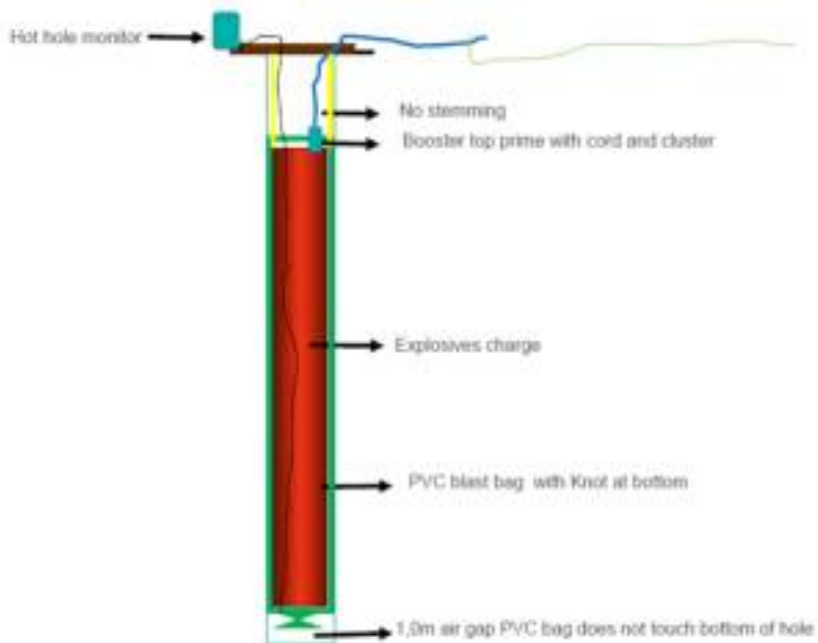


Figure 6. Schematic of a sleeved hole charged with bulk explosive and top primed.

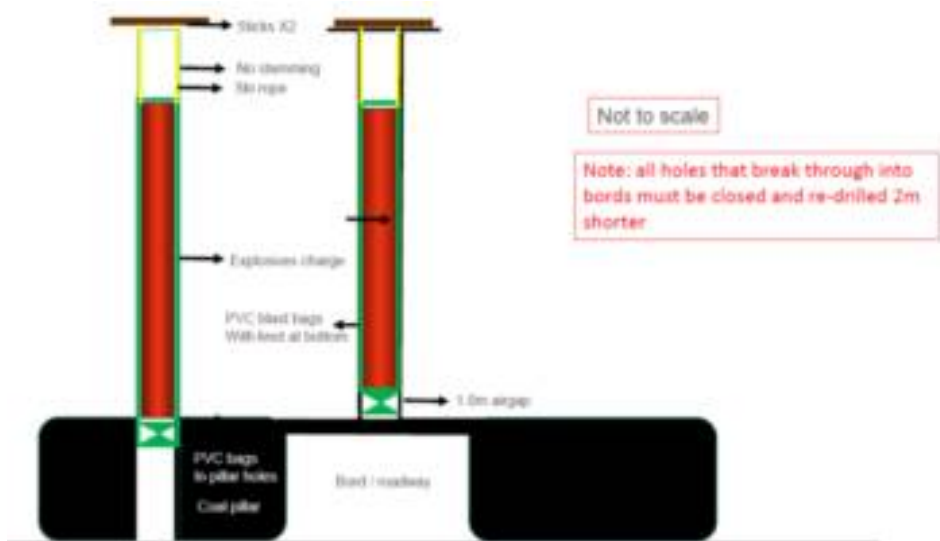


Figure 7. Shows how holes are charged when coal pillars and roadways/bords are present.



Figure 8. Photographs showing a sleeve being lowered into the hole and a sleeve ready to be charged Stenzel (2021, 2025).

5 RESULTS

Since the full introduction of the PVC hot hole blast sleeves in 2021 more than 158 000 holes between 80 °C -120°C have been safely charged and blasted. There are now less than 2% of holes that are not charged because the temperatures are above 120 °C. Based on our tests we will not be increasing the maximum above 120°C as we do not believe it is safe using our current explosive which is a 65% emulsion 35 % ANPP blend.

Tests with two other explosive manufacturers products have yielded the same results with detonations occurring when the top of explosive reaches a temperature of between 137°C-142°C.

We have been able to return to areas that could not be blasted as temperatures were too high and have redrilled and blasted these areas. (Figure 10).

Our blast plans indicate all charged and blasted holes and, as per Figure 11, we are now achieving around 98% of all drilled holes as shown.



Figure 9. Multipoint Blast Eye cable and reader unit used on all sleeved holes above 80°C (Stenzel 2025).

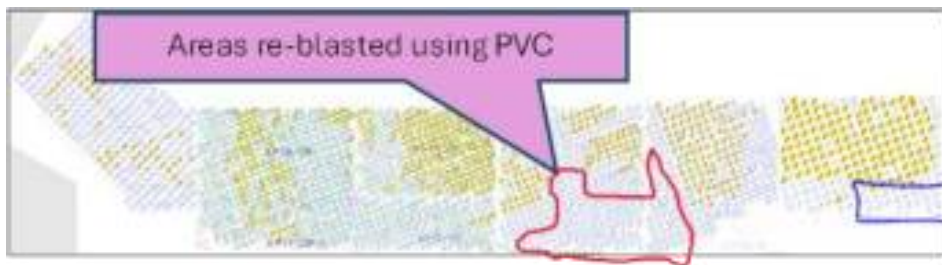


Figure 10. Many areas were redrilled and blasted using the PVC sleeves to allow mining to continue.

6 CONCLUSIONS

The use of PVC blast hole sleeves as designed and Patented by the mine has made it possible to safely charge and blast holes up to 120°C. The sleeves have also been used to charge breakthrough holes where the temperatures are below 80 °C which has seen a major saving on drilling costs. At the blast edges where there are often cracks and where explosive ‘runaways’ are seen these sleeves have also been successfully used to save on explosive costs and achieve excellent blast results.

7 ACKNOWLEDGEMENTS

This project was funded by Glencore Coal SA.

REFERENCES

- Anthony, D. & Grobelaar, J. 2009. Hot hole monitoring at Kleinkopje mine. *SAIMM Journal November*.
- AEISG. 2021. Code of practice: elevated temperature and reactive ground (Edition 4).



Figure 11. Example of a hot hole block (100 m x 60 m) where all holes have been charged (November 2024).

Australia Explosives Industry and Safety Group Inc. Retrieved from <https://www.aeiscg.org.au/publications>

Cavanough, G., Torrance, A. & Stenzel, G. 2025. The thermal basis for explosive decomposition with heat due to external heating or reactive ground. *Proceedings of the International Society of Explosives Engineers (ISEE)*, Cleveland.

Manas-Zloczower, I. 2013. Thermal and mechanical properties of PVC for industrial applications. *Journal of Polymer Engineering*, 33(7), 657–668.
<https://doi.org/10.1515/polyeng-2012-0118>

Mpofu, M. *et al.* 2023. Temperature measurements in blast holes. *SAIMM Journal*.

Stenzel, G.J. Mpofu, M. 2022. The management of hot holes in opencast coal mines. *Coaltech Report*.

Robertson, E.C. 1988. Thermal properties of rocks. *United States Geological Survey Bulletin* 88-129. Retrieved from <https://pubs.usgs.gov/bul/0888/report.pdf>

Meyer, R. 2007. Explosives (6th ed.). *Wiley-VCH*.

RETURN TO CONTENTS PAGE

Gaseous emissions of hydrogen peroxide emulsion explosives tested in an underground mine

A. Järvinen, H. Kuutti & M. Rumjantsev

IVTT Technical Research Centre of Finland, Espoo, Finland

V. Silvonen & T. Rönkkö

Aerosol Physics Laboratory, Physics Unit, Tampere University, Tampere, Finland

T. Halme

FORCIT Explosives, Laukaa, Finland

ABSTRACT: Ammonium nitrate (AN) based explosives produce problematic NO_x emissions. Hydrogen peroxide emulsion (HPE) explosives are developed to reduce these emissions. In this work, gaseous emissions from HPE and AN emulsion blasts were measured in an underground mine in the exit route of blasting fumes. Concentrations of CO, CO₂, NO, NO₂ and H₂O₂ were measured, and emission factors were calculated with carbon balance method based on CO₂ and CO concentrations and composition information of the explosives. Emission factors for CO were 16 L/kg for AN and 12 L/kg for HPE. HPE emission factors varied depending how long it was in the borehole before detonation. NO emission factor was 1.3 L/kg and NO₂ factor 0.3 L/kg for AN, while these factors were basically zero for HPE. H₂O₂ emission factor for HPE was 0.02 L/kg on average.

1 INTRODUCTION

Underground mines are a challenging work environment to maintain good air quality. Emissions of harmful gases and particles cannot be removed very efficiently with only ventilation. Thus, air quality can be improved by reducing emissions at their sources.

In the underground mining environment, use of explosives is a significant factor in workplace CO, NO and NO₂ concentrations. Traditional mining explosives contain nitro or nitrate compounds, which emit NO and NO₂. If explosives do not contain any nitrogen, the NO and NO₂ emissions should be minimised as they can be only formed in reactions with air, which often produces only low

amounts of these gases. Instead of nitro and nitrate compounds, hydrogen peroxide can be used as an oxygen-containing component in the hydrogen peroxide emulsion (HPE) explosives. Now, without nitro or nitrate compounds, the NO and NO₂ emissions should be minimised. Additionally, low CO emissions may be possible as the HPE does not suffer from CO-NO_x-tradeoff, where one emission can be reduced but the other increases depending on oxygen balance, typical for traditional explosives, for instance ammonium nitrate (AN) based (e.g. Rowland & Mainiero 2000, Nyberg *et al.* 2017).

On the downside, hydrogen peroxide itself is a very reactive and harmful substance and possesses a health risk if inhaled in addition to other



Figure 1. Emission measurement site location and distances to explosion locations.

exposure types. Thus, when HPE-type explosives are used, unreacted gaseous H_2O_2 may act as a harmful emission with similar concerns as NO_2 .

In this work, gaseous emissions of CO , NO , NO_2 and H_2O_2 from HPE and AN emulsion blastings were measured in an operational underground mine. Based on measurement results, emission factors for these harmful substances were calculated with carbon balance method.

2 METHODOLOGY

2.1 Measurement site

Gaseous emissions of explosives were measured in the underground tunnels of Outokumpu Kemi mine. The measurement site was located between 50 and 250 m from the detonation point, see Figure 1. The tunnel network had only one exit route, and exhaust fumes travelled along this when

fresh ventilation air was blown near the blasting site. The measurement site was in a branching tunnel and air to this site was blown from the exit route of fumes with an extra ventilation fan and short tube installed specially for these tests.

2.2 Measurement matrix

In total, 5 blasting events were measured. These contained two reference blasts with AN-based commercial emulsion explosive (Kemiitti 810, Forcitt Explosives) and 3 blasts with HPE. In the HPE tests, approximately 0.4% of the explosive mass was from nitrogen-based explosives. All the HPE tests were conducted with the same composition but in the first HPE blasting, the explosive was in the borehole approximately 3.5 h while in two others tests 1.5 h. The measurement matrix is presented in Table 1.

Table 1. Measurement matrix.

Test code	Explosive	Time in borehole (h)
AN1	Kemiitti 810	-
HPE1	HPE	3.5
HPE2	HPE	1.5
HPE3	HPE	1.5
AN2	Kemiitti 810	-

2.3 Instrumentation

Concentrations of CO, CO₂, NO and NO₂ were measured with a Sensor pack consisting of commercial sensors from different manufacturers and developed at VTT Technical Research Centre of Finland. CO₂ sensor in the pack is based on absorption of infra-red radiation while other sensors rely on electrochemical measurement principles. The Sensor pack was calibrated before the measurement campaign in laboratory conditions. Calibration gases from high pressure cylinders were diluted with nitrogen using a gas divider (S-2000-P, Environics Inc.). The calibration concentrations were in the similar range with expected concentrations in the blasting fumes.

In addition to sensor-based system, NO and NO₂ were measured with a dedicated chemiluminescence-based NO_x-analyser (Model T201, Teledyne API). The instrument measures principally NO but NO₂ is also detected when it is reduced to NO in the internal catalytic converter in the device. These two gases are measured by switching the sample route in the instrument. Unfortunately, NO_x analyser had technical issues and concentrations had step changes not seen in any other data. However, the analyser was able to confirm that NO and NO₂ concentrations were low when the sensor-based system was near its limit of detection. Also, the concentrations measured were relatively well in line with the Sensor pack.

H₂O₂ concentrations were measured with gas sensor system equipped with electrochemical sensor (Polytron 7000 and Model 6809705 H₂O₂ LC, Drägerwerk AG & Co. KGaA). The 4.-20 mA current signal produced by the sensor was logged using 1 kΩ resistor and a voltage logger. The maximum measurable concentration was 5 ppm with the applied configuration, and the detection limit was 0.1 ppm according to the manufacturer.

Sensor cross-correlation to other compounds than H₂O₂ were tested in laboratory conditions with CO, CO₂, NO, NO₂ and SO₂. These gases were obtained by diluting calibration gases from cylinders with filtered air using a gas divider (S-2000-P, Environics Inc.). The output of the H₂O₂ sensor was negative when subjected to NO₂. The sensor does not show negative concentrations reliably and the reading of the sensor was first increased with low concentration of SO₂, which provides positive cross-correlation and then output was reduced by adding NO₂. NO produced a small increase in the sensor output, but this was significantly smaller than the negative effect of NO₂ or positive of SO₂. Table 2 contains the results of the cross-correlation measurements.

Table 2. Cross-correlations of the H₂O₂ sensor with different gases.

Gas	Reference concentration (ppm)	Change in H ₂ O ₂ reading (ppm)	H ₂ O ₂ rdg. Gas conc. (ppm/ppm)
CO	99	0	0
CO ₂	3500	0	0
NO	12.2	0.25	0.020
NO ₂	2.4	-1.08	-0.450
SO ₂	3.4	1.32	0.403

2.4 Data-analysis methods

The emission factors were calculated using a carbon balance method, where measured carbon-based gases CO₂ and CO were used to estimate emissions per mass of explosive. A similar method, but relying only on CO₂, has been used for instance in determining emission factors for vehicles when CO emissions have been low (e.g. Canagaratna *et al.* 2004, Hallquist *et al.* 2013) and ships (e.g. Pirjola *et al.* 2014).

The volume-based emission factor EF_i is defined as volume of the emission gas V_i and mass of the explosive m_e with equation:

$$EF_i = \frac{V_i}{m_e} \quad (1)$$

Now the mass of the explosive is:

$$m_e = \frac{R_C \cdot M_C}{\rho_C} \quad (2)$$

Where n_C and M_C are the amount and the molar mass of carbon. The factor β_C is the molar fraction of carbon in the explosive.

By applying ideal gas law into Equation 1 and by taking Equation 2 into account, we get:

$$EF_i = \frac{RT}{p} \frac{n_i}{n_C M_C} \beta_C, \quad (3)$$

Where n_i is the molar amount of emission i .

The carbon balance method assumes that the amount of carbon is the same as the sum of amounts of CO and CO₂. Additionally, by converting amounts of gaseous components into concentrations C , we get:

$$EF_i = \frac{RT}{p M_C C_{CO} + C_{CO_2}} \beta_C, \quad (4)$$

This is the calculation function of the emission factors. Now the term containing concentrations can be solved from linear fits of xy-plot, where the sum of CO and CO₂ concentrations is in the x-axis.

3 RESULTS

3.1 Concentrations

Time series of gaseous concentrations of NO, NO₂, CO, CO₂ and H₂O₂ after each explosion are presented in Figure 2. NO and NO₂ concentrations were measured with Sensor pack. The time series have been plotted starting from the detonation and lasting 2 h. One should note that the measured concentrations are from the whole blasting events and contain also fumes from the booster charges.

The time series had relatively large differences. The duration from the detonation to the time when the peak of the plume arrived in measurement devices varied from a couple of minutes to more than an hour. The location of the blasting affected this duration. The shortest period was obtained for the nearest blasting, HPE2, and the longest for the AN2, which was the furthest away from the measurement site.

CO was detected from all blastings. HPE produced relatively sharp peaks with high concentrations and AN somewhat lower concentrations but for longer time. For HPE, the peak concentrations were between 120 and 210 ppm and for AN either 120 ppm (AN1) or 44 ppm (AN2). Lower CO peak concentration in case of AN2 is most likely resulting from widened and delayed peak, which has been mixed with larger volume of clean air.

CO₂ can be used to analyse mixing of the emission peaks with clean air as it is characteristic end product of optimal blasting event. CO₂ concentrations varied significantly between the blastings indicating that the explosion plume was diluted differently. For instance, HPE2 and HPE3 produced concentrations between 2200 and 2600 ppm but HPE1 only 1200 ppm. CO₂ concentrations of AN1 were at similar level with HPE1, and in case of AN2 peak was at less than 1000 ppm. When AN and HPE are compared, it should be noted that the carbon contents of these explosives were different.

NO concentrations were the highest for the AN blastings as expected. The peak concentrations were 10 ppm for AN1 and 3 ppm for AN2. The AN2 plume was delayed and widened compared to other blastings, which may reduce peak concentrations and may increase effects of chemical reactions affecting the measured concentrations. In case of the HPE2, which was located closest to the measurement site, the concentration of NO was 1.5 ppm at the highest and the peak was sharp, which increases the concentration compared to wide peaks. HPE1 and HPE3 produced only very low concentrations peaks.

NO₂ concentrations have similar profiles with NO but the peaks were lower, 3 ppm at maximum for AN1 and 0.5 ppm for AN2. In case of AN2, the NO₂ peaks appears later than NO, which may result from chemical reactions. The NO₂ was less than 0.2 ppm for all HPE tests.

H₂O₂ was measured at low levels, and the peak concentration was less than 0.6 ppm. The sensor had a characteristic concentration step directly from 0.0 to 0.1 ppm but provided higher precision above 0.1 ppm, which can be seen from the time series. The cross correlation with NO and NO₂ was corrected in the figure and this correction was small, in the order of 0.1 or 0.2 ppm due to very low NO and NO₂ concentrations.

3.2 Emission factors

In order to calculate the emission factors, concentrations are presented in an xy-plot, where x is the sum of CO and CO₂ and y the gas concentration for which the emission factor is calculated. Figure 3 presents these xy-charts for CO, NO, NO₂ and H₂O₂.

The CO concentrations are linear with the sum of CO and CO₂ (sum is dominated by CO₂) indicating that both these gases occur simultaneously, and the source of these gases is most likely the same over the time selected for the

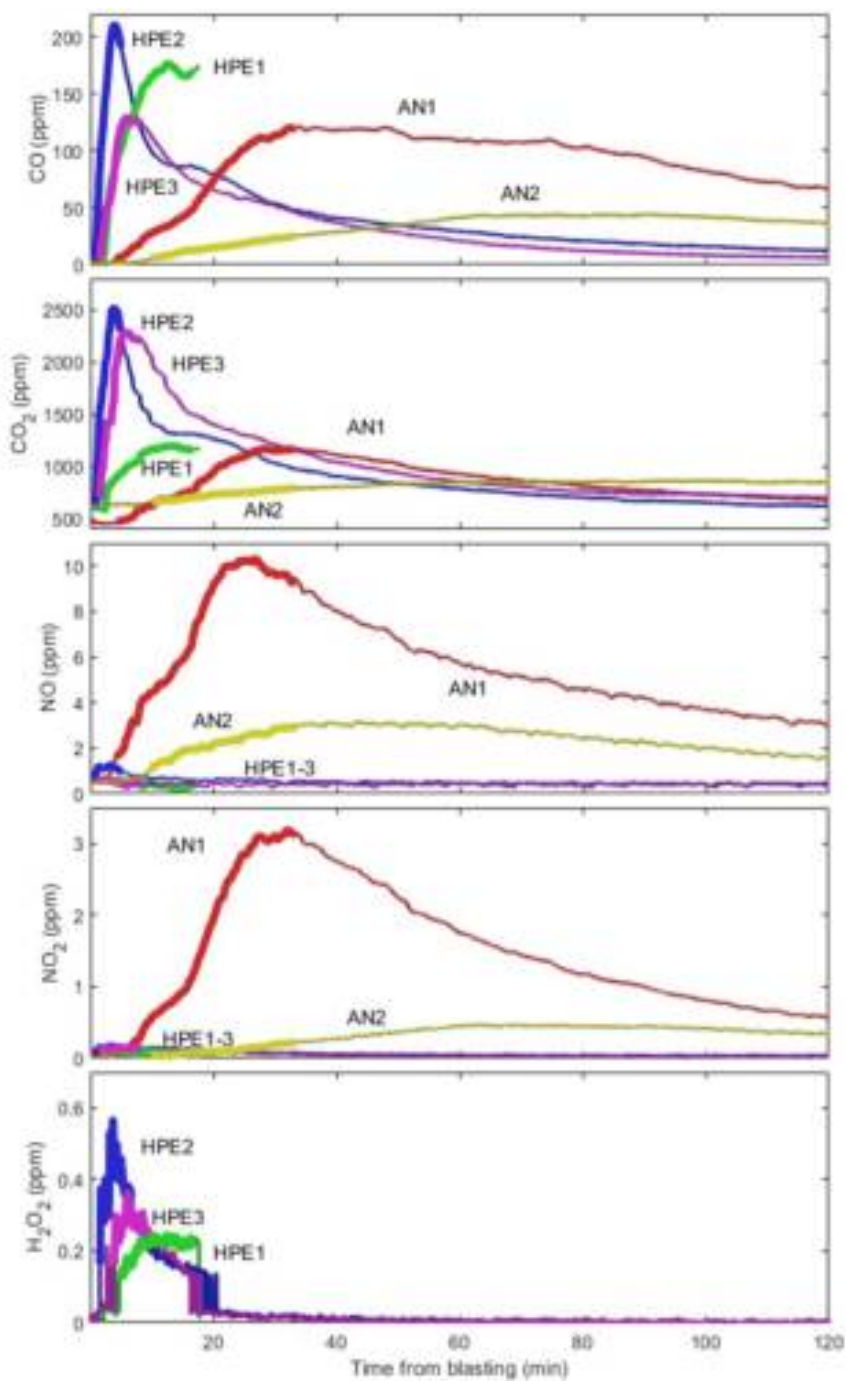


Figure 2 Time series of gas concentrations after the explosion occurring at time 0 min. The thicker and lighter sections of lines are presenting the data selected further into the emission factor analysis. H₂O₂ concentrations have been corrected for cross correlation with NO and NO₂.

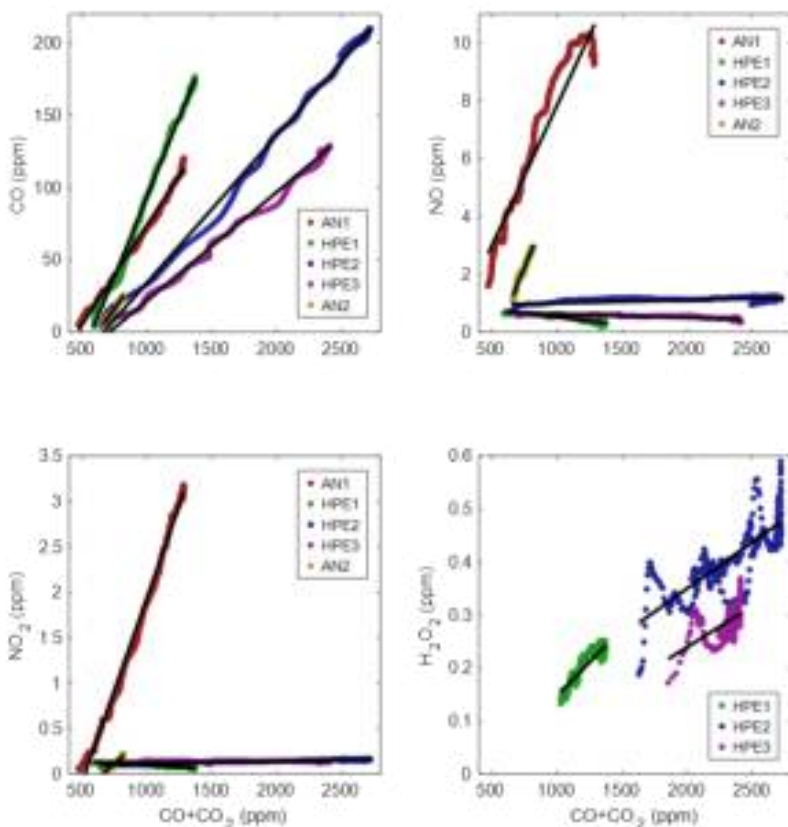


Figure 3. Concentrations of emission compounds as a function of carbon concentration, which is the sum of CO and CO₂. Also, the linear fits are presented to illustrate the emission factor calculation principle.

plots. The AN blastings have approximately the same slopes resulting in a similar emission factor for CO. The HPE1 slope is larger than the slopes of HPE2 and HPE3, which refers to a higher CO emission factor.

The slopes for NO are similar for two AN blastings. Especially in case of AN1, some non-linearity is seen, which most likely occurs due to chemical reactions reducing NO concentrations over time. Plume arrived at the measurement site slowly in case of AN blastings, which may increase effects of chemical reactions. In case of HPE, all tests produced almost horizontal fits, where CO and CO₂ are present but NO not, resulting in emission factors close to zero.

When studying NO₂, the slopes of AN differ from each other, resulting most likely from different plume arrival times. All HPE tests produced horizontal fits resulting in close to zero emission factors.

Gaseous H₂O₂ was also determined as function of sum of CO₂ and CO. Datapoints selected to this figure are not temporally the same as with other gases due to the sensor characteristic jump from 0 to 0.1 ppm. Only datapoints above this 0.1 ppm threshold have been selected. The datapoints are more scattered than with other gases but still somewhat linear relation is seen. HPE2 and HPE3 produce similar slopes and the slope of HPE1 test is somewhat larger indicating higher H₂O₂ emissions.

The emission factors calculated from slopes of Figure 3 according to Equation 4 are presented in Table 3. Emission factors for CO and H₂O₂ vary between HPE1 and other HPE tests, both emissions being higher in case of HPE1. The HPE1 was 3.5 h in the borehole, which gives more time for performance deteriorating chemical reactions compared to 1.5 h in case of HPE2 and HPE3. The latter two HPE tests

Table 3. Emission factors *L* per kg of explosive from 5 blasting events and average factors for AN and HPE explosives.

	CO			NO		NO ₂		H ₂ O ₂	
	(L/kgex)			(L/kgex)		(L/kgex)		(L/kgex)	
AN1	17.3	±	2.1	1.22	± 0.19	0.51	± 0.07		
HPE1	20.2	±	2.3	-0.05	± 0.04	-0.01	± 0.01	0.025	± 0.009
HPE2	9.4	±	1.1	0.01	± 0.01	0.00	± 0.01	0.016	± 0.004
HPE3	6.9	±	0.8	-0.01	± 0.01	0.00	± 0.01	0.015	± 0.005
AN2	15.5	±	2.5	1.38	± 0.37	0.15	± 0.07		
AN avg	16.4			1.30		0.33			
HPE avg	12.2			-0.02		0.00		0.019	

Note that *L* is defined in conditions: 1013.25 mbar and 273.15 K. The reported H₂O₂ emissions factors include corrections for NO and NO₂. Uncertainty values include fitting (95% limit) and approximate uncertainty of gas sensors. In case of HPE, some NO and NO₂ emission factors are below zero indicating that the concentrations have been decreasing after the blasting.

produced significantly lower CO emissions than AN tests.

AN reference blasting enable comparison to previous studies. For instance, Nyberg *et al.* (2017) and Kramarczyk *et al.* (2022) have conducted tests to determine emission factors for AN emulsion explosives. In both earlier studies, CO emissions were lower than in our study. For instance, Nyberg *et al.* (2017) determined CO of 8.6 and 3.4 L/kg for E682 in glass confiners (diameters 27 and 80 mm) and Kramarczyk *et al.* (2022) 4.11 L/kg for Emulinit 8L. Nyberg *et al.* (2017) determined somewhat higher NO_x emissions of 2.4 L/kg and 1.4 L/kg and Kramarczyk *et al.* (2022) somewhat lower with NO 0.51 L/kg and NO₂ 0.04 L/kg (same test as with CO). This difference could be related to different explosive but also to execution of the measurements as our tests were conducted in real mining environment and reference studies were conducted in a laboratory environment in a special blasting chamber. Tests in Nyberg *et al.* (2017) also indicate that the environment affects emissions as the confiner material was observed to affect the CO and NO_x emissions significantly with more emissions from mortar confinners than glass confinners.

A laboratory study by Polis *et al.* (2024) provides some comparison to HPE results. In their study CO emissions were 10-20 L/kg, NO 0.22-0.37 L/kg and NO₂ 0.03-0.05 L/kg. These reference results are in line with our study, only NO was somewhat larger in work by Polis *et al.* (2024).

4 CONCLUSIONS

Gaseous emissions of HPE and AN blastings were measured in an underground mine with tracer gas technique. HPE produced low emissions, NO and NO₂ emissions were drastically smaller than from AN blastings. Also, CO emissions were relatively low. Unreacted H₂O₂ was detected only in low concentrations.

Higher CO and H₂O₂ emissions were detected for test where HPE was 3.5 h in the borehole before detonation compared to 1.5 h. Emission difference suggests that the stability of HPE was not as good as with traditional compounds.

The measurement on H₂O₂ is not straightforward with electrochemical sensor systems. These sensors suffer from cross-correlation with other emissions gases. Especially NO₂ is problematic. For instance, with the sensor used in this study, it reduced H₂O₂ reading, which may result in underestimation of concentrations. It is recommended to analyse cross-correlations of the applied measurement method thoroughly.

5 ACKNOWLEDGEMENTS

This work was conducted in NEX-EL project funded by Business Finland and Finnish companies (Grant: 7806/31/2022). Authors acknowledge project partner Outokumpu Chrome for providing the test environment and support during the test activities.

REFERENCES

- Canagaratna, M.R., Jayne, J.T., Ghertner, D.A., Herndon, S., Shi, Q., Jimenez, J.L., Silva, P.J., Williams, P., Lanni, T., Drewnick, F., Demerjian K.L., Kolb, C.E. & Worsnop D.R. 2004. Chase studies of particulate emissions from in-use New York City vehicles. *Aerosol Science and Technology* 38:6, 555-573.
- Hallquist, Å.M., Jerksjö, M., Fallgren H., Westerlund J. & Sjödin Å. 2013. Particle and gaseous emissions from individual diesel and CNG buses. *Atmospheric Chemistry and Physics* 13, 5337–5350.
- Kramarczyk, B., Pytlik, M., Mertuszka, P., Jaszcz, K. & Jarosz, T. 2022. Novel sensitizing agent formulation for bulk emulsion explosives with improved energetic parameters. *Materials* 15, 900.
- Nyberg, U., Johansson, D. & Petropoulos, N. 2017. Toxic fumes from detonation of emulsion explosives. *Proceedings of the 9th EFEE World Conference on Explosives and Blasting*, Stockholm, Sweden.
- Pirjola, L., Pajunoja, A., Walden, J., Jakanen, J.P., Rönkkö, T., Kousa, A. & Koskentalo, T. 2014. Mobile measurements of ship emissions in two harbour areas in Finland. *Atmospheric Measurement Techniques* 7, 149–161.
- Polis, M., Stolarczyk, A., Szydło, K., Fabian, M., Pytlik, M., Lisiecka, B. & Jarosz, T. 2024. Comprehensive study on impact of hydrogen peroxide decomposition on the crucial parameters of OSM-type energetic materials. *Scientific Reports*, 14, 14093.
- Rowland – III, J.H. & Mainiero, R. 2000. Factors affecting ANFO fumes production. *Proceedings of the 26th Annual Conference on Explosives and Blasting Technique*, Anaheim, California.

[RETURN TO CONTENTS PAGE](#)

Perforating gun test series

J. Tatom & J. Romano

Analysis, Planning, Test, Research Inc. (APT), Huntsville, Alabama, USA

J. Hoffman

Institute of Makers of Explosives (IME), Washington, DC, USA

ABSTRACT: Since 2012, APT and the sponsors of IMESA[®]FR (Institute of Makers of Explosive Safety Assessment for Risk) have wanted to perform a jet perforating gun (JPG) test series. This is because the JPG model currently used in IMESA[®]FR is not based on test data; therefore, it is unknown exactly how accurate the current model is, but it is believed to be conservative. In 2024, the Bureau of Alcohol, Tobacco, Firearms and Explosives (ATF) sponsored the performance of a JPG test series with the goal of gathering the necessary test data to improve IMESA[®]FR's JPG model. This test series encompassed a total of eight tests. Upon completion of the test series, the results of the tests were compared to IMESA[®]FR's predictions to determine how accurate the model is and what adjustments, if any, would be needed. IMESA[®]FR Copyright 2025 USA Copyright © 2025 A-P-T Research, Inc.

1 BACKGROUND

1.1 Introduction

The Bureau of Alcohol, Tobacco, Firearms and Explosives (ATF) is the governmental organisation that regulates the storage of commercial explosives in the United States, including the storage of jet perforating guns (JPGs). JPGs are used by the oil and gas industry to assist in well completion. ATF (2021) regulates storage safety based on the American Table of Distances (ATD). However, ATF does accept variances based on quantitative risk assessments (QRA) using the IMESA[®]FR (Institute of Makers of Explosive Safety Assessment for Risk) tool (Tatom *et al.* 2018). IMESA[®]FR has a JPG model, however the

model is not based on JPG test data. Therefore, in 2024 ATF sponsored a series of JPG tests with the goal of collecting test data that could be used to improve the IMESA[®]FR JPG model.

1.2 What is a JPG?

JPGs are specialised tools that are used in the oil and gas industry to assist in well completion. Each JPG contains a series of conical shaped charges (perforators) which, when detonated, produce a hypervelocity jet that punches through the gun itself, the wellbore casing, and the geologic formation, allowing oil or gas to flow up the well. Given that these tools are meant to be used in deep wellbores, they are normally designed to operate under the high-pressure-fluid environments that

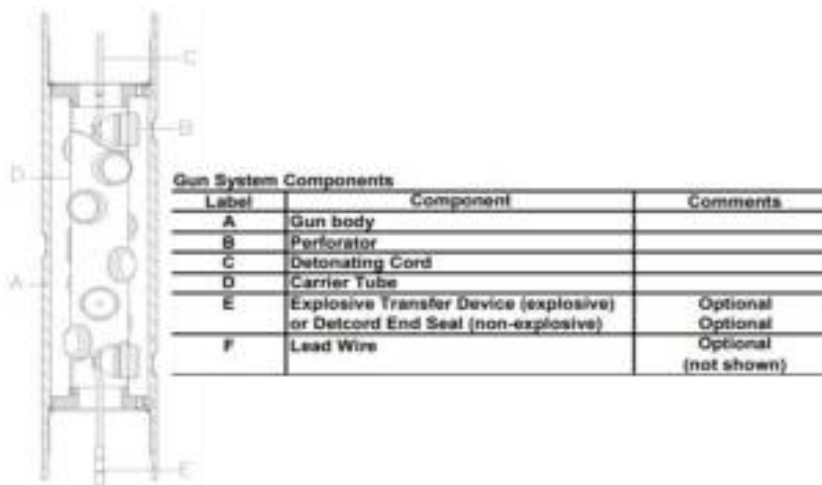


Figure 1. JPG components.

exist within oil and gas wells. While some JPGs are designed to function at atmospheric pressures, the goal of this test series was to observe how JPGs designed to function in high-pressure environments behave if they are detonated in atmospheric pressure. When utilised in standard operations, the outer shell of JPGs is engineered to remain in one piece; however, considering that the JPGs selected for testing are not rated for open air operations, it is not unreasonable to expect that the outer shells may rupture upon detonation. Figure 1 is an image provided by IME depicting the typical

JPG components (InstME 2017).

1.3 Model background

The current IMESA FR JPG model is based on data extrapolated from a model of a larger surrogate explosive article. The size of the surrogate item was chosen because at the time of IMESA FR's original release, about 20 years ago, JPGs were typically larger than they are today. The key testing parameters needed to update the existing model, in order of priority, are:

Table 1. Test matrix.

Test JPG	Dia.	Length	Initiation Height	Number of Items	JPG Orientation	NEW per Charge	Total Weight Gun + Charges
Test Company A 1	3.125 in	23 in	1 ft	Single	Horizontal	21.5 g	10,117 g
Test Company B 2	3.125 in	21 in	1 ft	Single	Horizontal	21.5 g	9,448 g
Test Company A 3	3.125 in	23 in	1 ft	Single	Horizontal	21.5 g	9,639 g
Test Company C 4	3.125 in	24.5 in	1 ft	Single	Horizontal	21.5 g	13,416 g
Test Company B 5	3.125 in	21 in	3 ft	Single	Horizontal	21.5 g	9,502 g
Test Company D 6	3.125 in	~ 15 in	1 ft	Single	Horizontal	23 g	10,467.8 g
Test Company B 7	3.125 in	21 in	Gun on Pallet	Single	Vertical	21.5 g	9,464 g
Test Company B 8	3.125 in	21 in	Gun on Pallet	Multiple (6)	Horizontal	21.5 g	56,936 g

Table 2. TP-21 mass bins (steel).

	Bin 1	Bin 2	Bin 3	Bin 4	Bin 5	Bin 6	Bin 7	Bin 8	Bin 9	Bin 10	Bin G
Upper Mass (g)	n/a	11,800	4,540	2,040	820	360	140	64	27	11	<6
Lower Mass (g)	>11,800	4,540	2,040	820	360	140	64	27	11	6	n/a

- JPG fragment range, bearing, and mass.
- Pressure versus distance.
- Fragment velocity.

2 PLANNING

When planning for this test series began, the goal was to test as many realistic configurations as possible. Due to limited time and funding, the test series had to be restricted to eight configurations. In addition to limiting the number of tests that would be performed, the types of tests had to be tailored according to what the testing range approved. In consultation with subject matter experts, range personnel, IME, and ATF, the test matrix shown in Table 1 was ultimately established to define the test series. The JPG specifications chosen represent 80-90% of those used in the US land market in the oil and gas industry around the time of these tests.

3 TESTING

This test series was conducted in the state of Texas and encompassed eight different tests that were performed in two phases, over the course of three weeks. All tests had six conical shape charges per gun, with a 60° phasing angle. Furthermore, for every horizontal test, the JPGs were oriented North-South with the sub facing south. The JPGs utilised in this test series were donated by four different manufacturers that have been denoted herein as Companies A, B, C, and D. Following the initiation of each test, fragment collection was done in accordance with Rev 2 of DDESB TP-21 (Swisdak *et al.* 2017).

The TP-21 mass bin categorisation method was used when analysing the debris catalogues. This is a method of categorising pieces of debris of similar size and mass into one of eleven bins. This makes predicting the potential behaviour and calculating the kinetic energy of fragments simpler since it can be done per mass bin rather than for each fragment. TP-21 provides a more in-depth description of this system and defines the upper

and lower limits for each mass bin as shown in Table 2.

3.1 Test 1

Test 1 took place on July 16, 2024, at 9:28 AM CDT (UTC -5). This test was unique since it was the only test that was performed with a non-threaded bulkhead installed on one end of the JPG. Figure 2 shows the setup for this test. It should be noted that all tests performed with a 1 ft elevation used the same setup.

Test 1 was the only test in the first phase of testing and therefore had a unique grid setup. Figure 3 depicts this search grid.



Figure 2. Test 1 setup.

3.1.1 Instrumentation results

For the sake of brevity in this paper, only high-level instrumentation results are contained herein,

as shown in the summary pressure gauge readings and fragment velocity parameters in Tables 3 & 4.

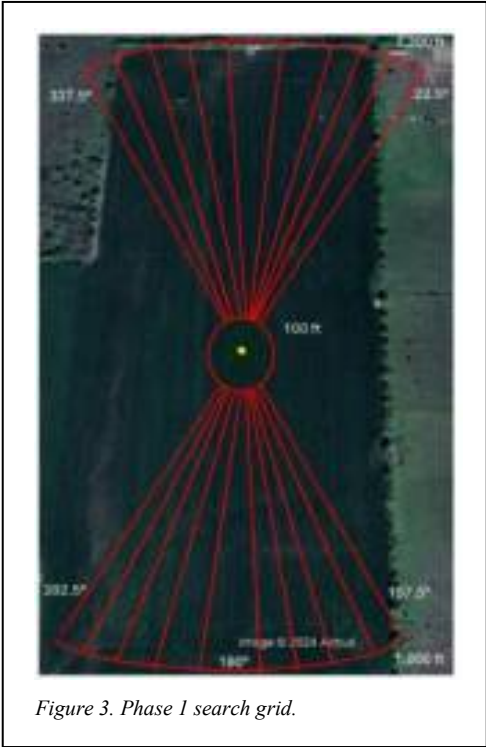


Figure 3. Phase 1 search grid.

3.1.2 Fragmentation results

In Test 1, 8,864.56 g or 87.62% of the initial mass was recovered. The bulkhead accounted for roughly 7% of the total mass; it exited the test area and was never recovered. Figure 4 shows all the recovered fragments.

Table 3. Test 1 pressure gauge readings.		
Distance (ft)	Northern Gauges (psi)	Western Gauges (psi)
5	n/a	3.34
10	3.7	1.92
20	1.32	n/a

In Test 1, a total of 25 fragments with a mass greater than 5 g were recovered, only four of which were recovered outside of 100 ft. The farthest recovered piece was found at 283.54 ft, along the 356.07° azimuth. The outer shell of the JPG was found at 16.71 ft, along the 150.59° azimuth. Figure 5 is a scatterplot of the recovered fragments.

Figure 6 depicts the fragment distribution by distance for Test 1 and follows a predictable trend with most of the fragments being found close-in, with the amplitude gradually decreasing as distance increases.



Figure 4. Test 1 recovered fragments.

Table 4. Test 1 fragment velocities.

Footage	No. of Fragments Tracked	Min. Velocity (ft/s)	Max. Velocity (ft/s)	Ave. Velocity (ft/s)
High-Speed	5	22.81	208.26	151.48
Standard	11	23.1	188.12	162.78
Drone	2	34.19	120.69	77.44

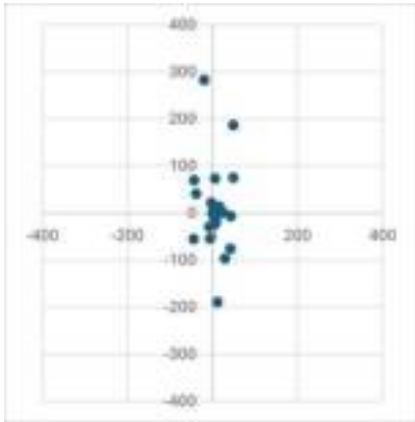


Figure 5. Test 1 scatterplot (units in feet).

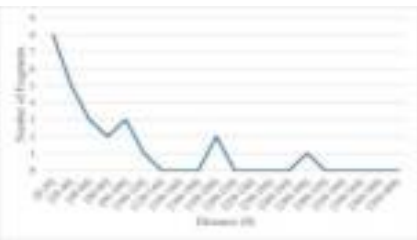


Figure 6. Test 1 fragments by distance.

Figure 7 and 8 depict the fragment distribution by azimuth in 5° bands, which is how the search grid was divided for this test (shown in Figure 3). Figure 7 shows the fragments that were above 5 g and found past 100 ft, while Figure 8 shows all the fragments above 5 g regardless of distance. It should be noted that in this test no fragments were found past 100 ft and within 0° - 15°. The shaded areas represent the azimuths where most fragments were expected to land: 0°±10° and 180°±10°.

Figure 9 shows how many fragments are in each mass bin. Most fragments fall in Bin 10. The Bin 2 entry is the outer shell of the JPG, and the Bin 6 entries are two large pieces of the charge carrier.

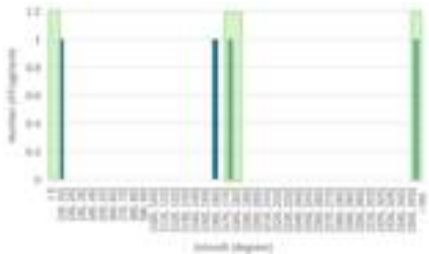


Figure 7. Test 1 fragments by azimuth past 100 ft.

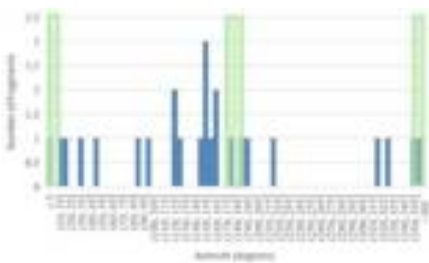


Figure 8. Test 1 fragments by azimuth.

3.2 Test 2

Test 2 took place on August 13, 2024, at 10:14 AM CDT (UTC -5). This test had the same setup as Test 1, as shown in Figure 2. The gun in this test was unintentionally offset 5° clockwise of the intended axis, as the gun was aligned with the grid markers rather than true North-South.

Test 2 and all following tests were performed in the second phase of testing. The second test phase utilised the search grid shown in Figure 10.

3.2.1 Instrumentation results

The pressure gauge readings and fragment velocity are summarised in Table 5 and Table 6.

3.2.2 Fragmentation results

In Test 2, 8,654 g or 91.6% of the initial mass was recovered. Figure 11 shows all the recovered fragments.

Table 5. Test 2 pressure gauge readings.

Distance (ft)	Northern Gauges (psi)	Western Gauges (psi)
5	12.91	3.77
10	7.29	3.88
20	n/a	1.38

Table 6. Test 2 fragment velocities.

Footage	No. of Fragments Tracked	Min. Velocity (ft/s)	Max. Velocity (ft/s)	Ave. Velocity (ft/s)
High Speed	4	181.59	226.29	187.81
Standard	6	170.5	248.88	186.07
Drone	2	107.34	152.42	129.88

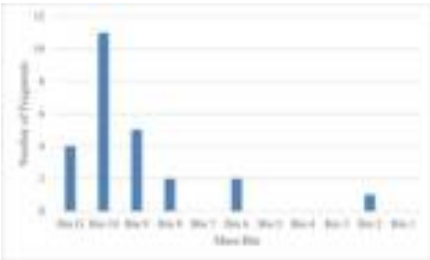


Figure 9. Test 1 fragments per mass bin.

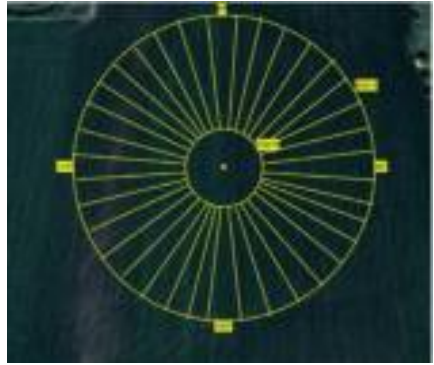


Figure 10. Phase 2 search grid.

In Test 2, a total of 34 fragments with a mass greater than 5 g were recovered, of which 21 were recovered outside of 100 ft. The farthest of these pieces was found at 300.97 ft, along the 10.61° azimuth. The outer shell of the JPG was found at 109.80 ft, along the 180.22° azimuth. Figure 12 is a scatterplot of the recovered fragments; the red line roughly represents the gun's alignment.

Figure 13 shows the fragment distribution by distance for Test 2. As stated previously, most fragments were recovered past 100 ft and the peak was in the 100 - 120 ft range.

Figure 14 and Figure 15 depict the fragment distribution by azimuth in 10° bands, which is how the search grid was divided for this test (shown in Figure 10). Figure 14 shows the fragments that were above 5 g and found past 100 ft, while Figure 15 shows all the fragments above 5 g regardless of distance. The shaded areas represent the azimuths,



Figure 11. Test 2 recovered fragments.

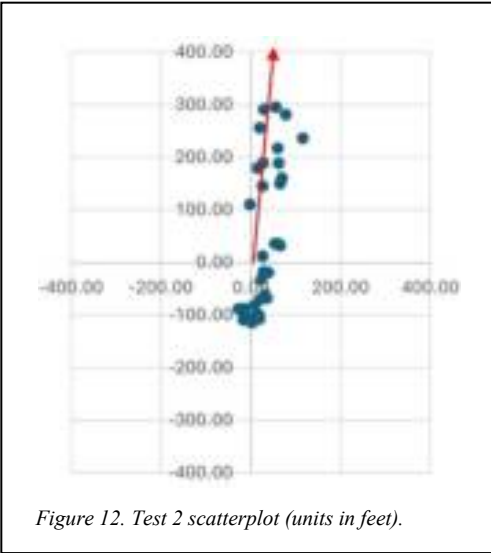


Figure 12. Test 2 scatterplot (units in feet).

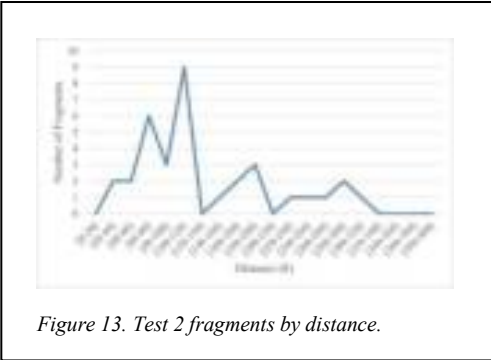


Figure 13. Test 2 fragments by distance.

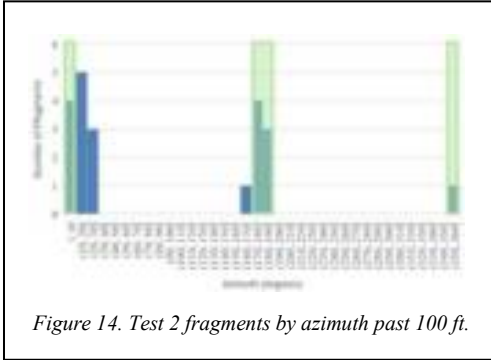


Figure 14. Test 2 fragments by azimuth past 100 ft.

where most fragments were expected to land: $0^{\circ}\pm10^{\circ}$ and $180^{\circ}\pm10^{\circ}$.

Figure 16 shows how many fragments are in each mass bin. Most fragments fall in Bin 10. Like Test 1, the Bin 2 entry is the outer shell of the JPG.

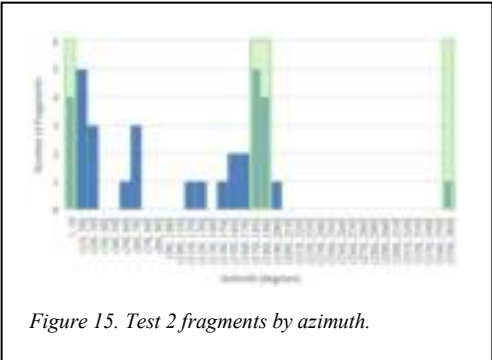


Figure 15. Test 2 fragments by azimuth.

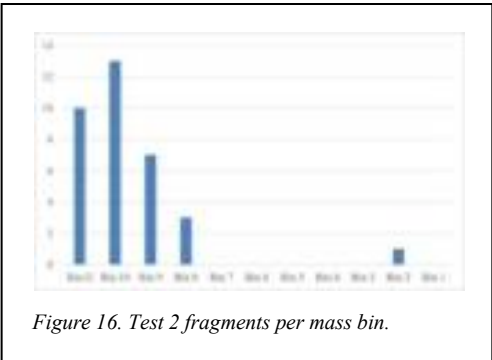


Figure 16. Test 2 fragments per mass bin.

Table 7. Test 3 pressure gauge readings.

Distance (ft)	Northern Gauges (psi)	Western Gauges (psi)
5	6.49	6.24
10	4.68	3.66
20	n/a	2.32

Table 8. Test 3 fragment velocities.

Footage	Num. of Fragments Tracked	Min. Velocity (ft/s)	Max. Velocity (ft/s)	Aver. Velocity (ft/s)
High Speed	3	51.56	115.38	93.06
Standard	8	41.25	283.8	166.4
Drone	3	77.53	155.86	123.77

3.3 Test 3

Test 3 took place on August 14, 2024, at 10:47 AM CDT (UTC -5). This test had the same setup as Test 1, as shown in Figure 2.



Figure 17. Test 3 recovered fragments.

3.3.1 Instrumentation results

The pressure gauge readings and fragment velocity are summarised in Table 7 and Table 8.

3.3.2 Fragmentation results

In Test 3, 9,193 g or 95.4% of the initial mass was recovered. Figure 17 shows all the recovered fragments.

In Test 3, a total of 51 fragments with a mass greater than 5 g were recovered, of which 19 were recovered outside of 100 ft. The farthest piece was found at 336.90 ft, along the 172.33° azimuth. The outer shell of the JPG was found at 2.28 ft, along the 1.33° azimuth. Figure 18 is a scatterplot of the recovered fragments.

Figure 19 depicts the fragment distribution by distance for Test 3 and follows a trend like Figure 6.

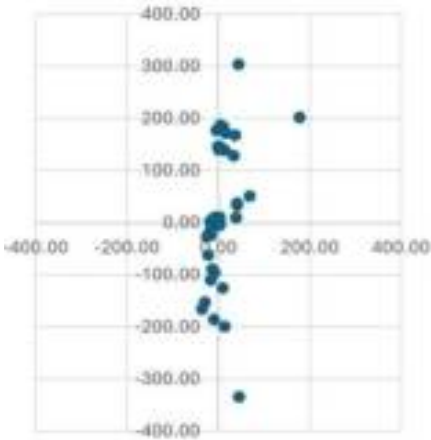


Figure 18. Test 3 scatterplot (units in feet).

Figure 20 and Figure 21 depict the fragment distribution by azimuth in 10° bands. Figure 20 shows the fragments that were above 5 g and found past 100 ft, while Figure 21 shows all the fragments above 5 g regardless of distance. The shaded areas represent the azimuths where most fragments were expected to land: 0°±10° and 180°±10°.

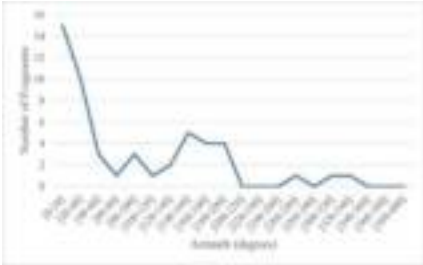


Figure 19. Test 3 fragments by distance.

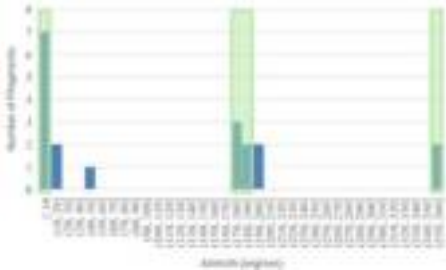


Figure 20. Test 3 fragments by azimuth past 100 ft.

Figure 22 shows how many fragments are in each mass bin. Most fragments fall in Bin 10. Like Test 1, the Bin 2 entry is the outer shell of the JPG,

and the Bin 6 entries are some large pieces of the charge carrier.

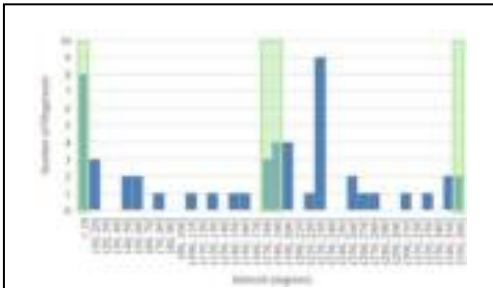


Figure 21. Test 3 fragments by azimuth.

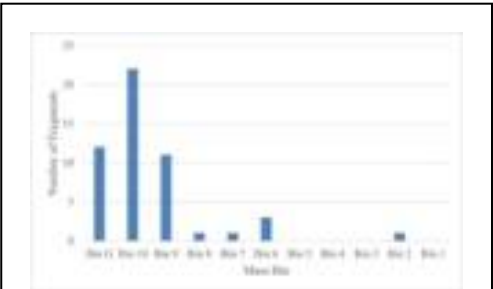


Figure 22. Test 3 fragments per mass bin.

3.4 Test 4

Test 4 took place on August 15, 2024, at 10:16 AM CDT (UTC -5). This test had the same setup as Test 1, as shown in Figure 2.

3.4.1 Instrumentation Results

The pressure gauge readings and fragment velocity are summarised in Table 9 and Table 10.

Table 9. Test 4 pressure gauge readings.

Distance (ft)	Northern Gauges (psi)	Western Gauges (psi)
5	8.34	3.19
10	6.02	2.76
20	n/a	1.6

Table 10. Test 4 fragment velocities.

Footage	No. of Fragments Tracked	Min. Velocity (ft/s)	Max. Velocity (ft/s)	Ave. Velocity (ft/s)
High Speed	4	52.53	292.16	169.4
Standard	8	66.2	307.69	197.87
Drone	2	82.5	261.3	171.9

3.4.2 Fragmentation Results

In Test 4, 12,835.4 g or 95.7% of the initial mass was recovered. Figure 23 shows all the recovered fragments.

In Test 4, a total of 44 fragments with a mass greater than 5 g were recovered, of which 16 were recovered outside of 100 ft. The farthest piece was found at 334.69 ft, along the 27.13° azimuth. The outer shell was found at 78.89 ft, along the 190° azimuth. Figure 24 is a scatterplot of the recovered fragments.

Figure 25 depicts the fragment distribution by distance for Test 4 and follows a trend like Figure 6.

Figure 26 and Figure 27 depict the fragment distribution by azimuth in 10° bands. Figure 26 shows the fragments that were above 5 g and,



Figure 23. Test 4 recovered fragments.

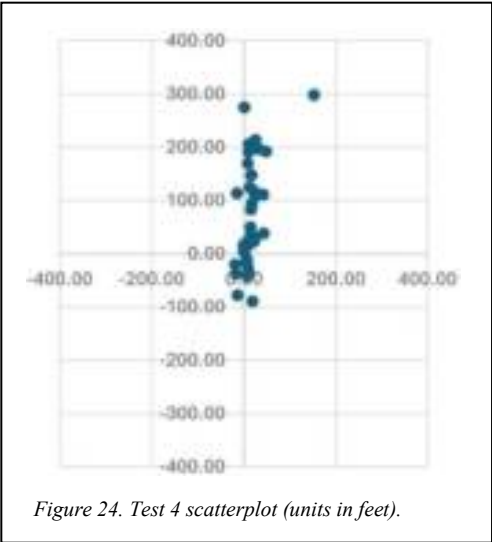


Figure 24. Test 4 scatterplot (units in feet).

found past 100 ft, while Figure 27 shows all the fragments above 5 g regardless of distance. The shaded areas represent the azimuths where most fragments were expected to land: $0^{\circ}\pm10^{\circ}$ and $180^{\circ}\pm10^{\circ}$.

Figure 28 shows how many fragments are in each mass bin. Most fragments fall in Bin 10. The shell of the JPG in this test had enough mass to qualify for Bin 1.

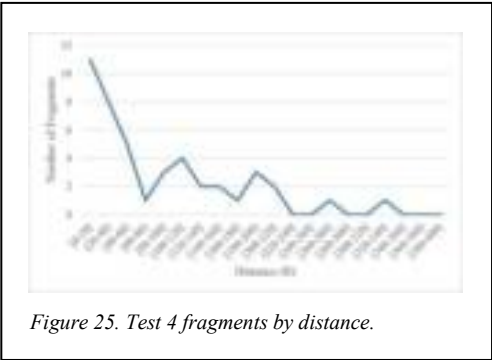


Figure 25. Test 4 fragments by distance.

3.5 Test 5

Test 5 took place on August 16, 2024, at 10:14 AM CDT (UTC -5). For this test, the JPG was positioned at a height of 3 ft above ground level, and was held in place by a test stand as shown in Figure 29.

3.5.1 Instrumentation results

The pressure gauge readings and fragment velocity are summarised in Table 11 and Table 12.

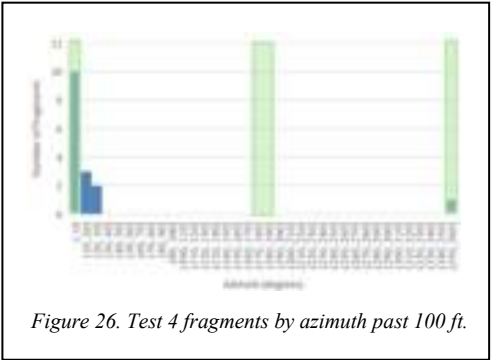


Figure 26. Test 4 fragments by azimuth past 100 ft.

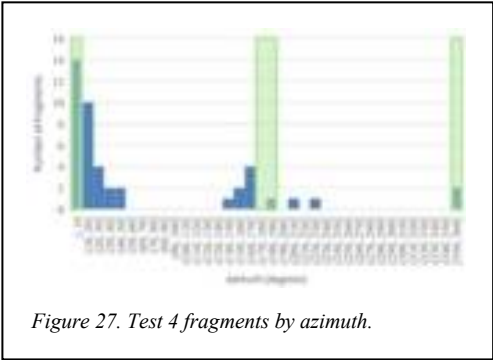


Figure 27. Test 4 fragments by azimuth.

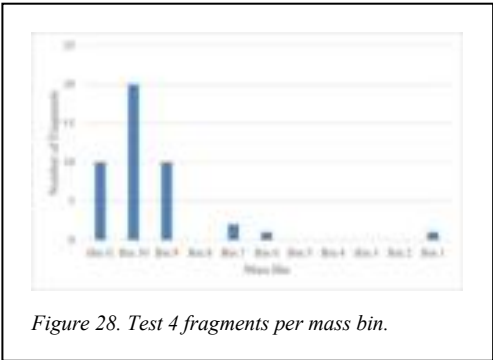


Figure 28. Test 4 fragments per mass bin.

Table 11. Test 5 pressure gauge readings.

Distance (ft)	Northern Gauges (psi)	Western Gauges (psi)
5	6.56	4.31
10	4.31	2.76
20	n/a	2.07

3.5.2 Fragmentation results

In Test 5, 9,083 g or 95.6% of the initial mass was recovered. Figure 30 shows all the recovered fragments.



Figure 29. Test 5 setup.

In Test 5, a total of 57 fragments with a mass greater than 5 g were recovered, of which 16 were recovered outside of 100 ft. The farthest piece was found at 287.71 ft, along the 14.45° azimuth.

The outer shell was found at 159.56 ft, along the 184.71° azimuth. Figure 31 is a scatterplot of the recovered fragments.



Figure 30. Test 5 recovered fragments.

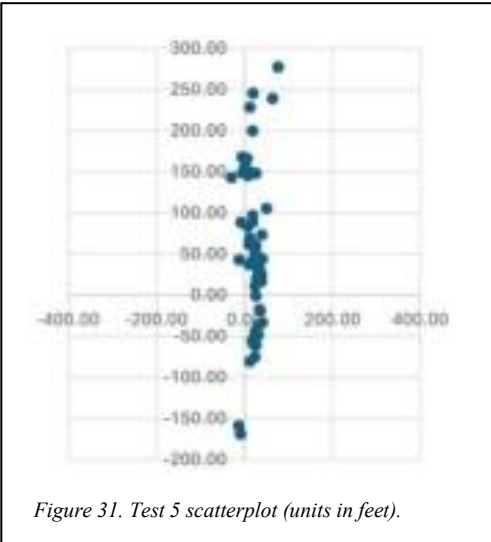


Figure 31. Test 5 scatterplot (units in feet).

Figure 32 depicts the fragment distribution by distance for Test 5 and follows a trend like Figure 13, with the peak amplitude outside the 0 – 20 ft range.

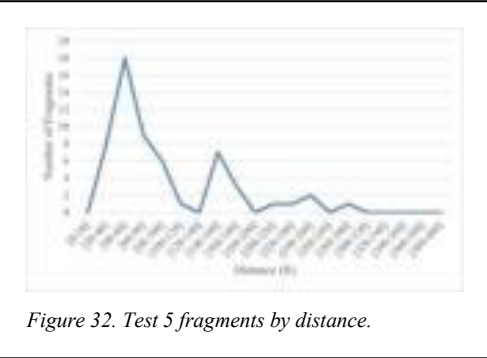


Figure 32. Test 5 fragments by distance.

Figure 33 and Figure 34 depict the fragment distribution by azimuth in 10° bands. Figure 33 shows the fragments that were above 5 g and found past 100 ft, while Figure 34 shows all the fragments above 5 g regardless of distance. The shaded areas represent the azimuths where most fragments were expected to land: 0°±10° and 180°±10°.

Figure 35 shows how many fragments are in each mass bin. Most fragments fall in Bin 10. Like most of the previous tests, the Bin 2 entry is the outer shell of the JPG.

3.6 Test 6

Test 6 took place on August 19, 2024, at 9:58 AM CDT (UTC -5). This test had the same setup as Test 1, as shown in Figure 2.

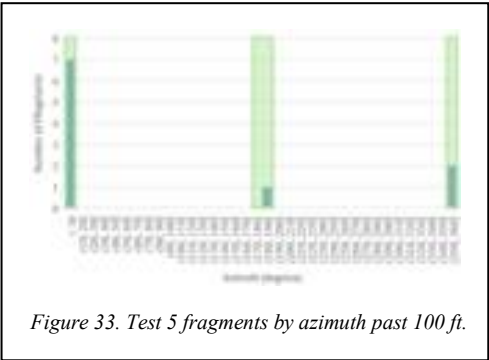


Figure 33. Test 5 fragments by azimuth past 100 ft.

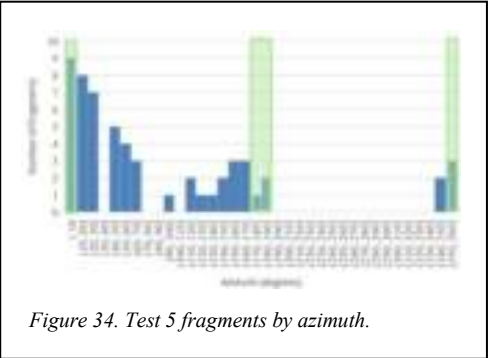


Figure 34. Test 5 fragments by azimuth.

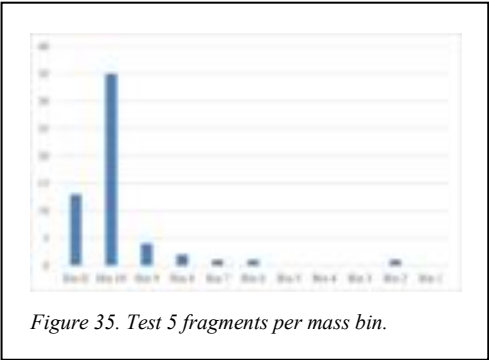


Figure 35. Test 5 fragments per mass bin.

3.6.1 Instrumentation Results

The pressure gauge readings and fragment velocity are summarised in Table 13 and Table 14.

Table 12. Test 5 fragment velocities.				
Footage	No. of Fragments Tracked	Min. Velocity (ft/s)	Max. Velocity (ft/s)	Ave. Velocity (ft/s)
High Speed	5	159.6	276.32	201.06
Standard	7	93.23	288.57	174.82
Drone	n/a	n/a	n/a	n/a



Figure 36. Test 6 recovered fragments.

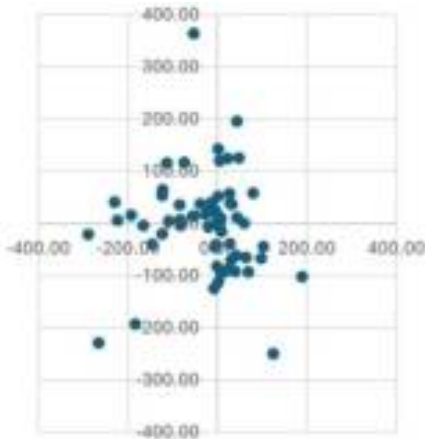


Figure 37. Test 6 scatterplot (units in feet).

Table 13. Test 6 pressure gauge readings.

Distance (ft)	Northern Gauges (psi)	Western Gauges (psi)
5	9.46	8.56
10	3.73	9.9
20	n/a	1.85

3.6.2 Fragmentation Results

In Test 6, 10,134 g or 96.8% of the initial mass was recovered. Figure 36 shows all the recovered fragments.

In Test 6, a total of 62 fragments with a mass greater than 5 g were recovered, of which 28 were

recovered outside of 100 ft. The farthest piece was found at 367.89 ft, along the 351.70° azimuth. In this test, the outer shell of the gun broke into three pieces. The exact range and bearing of these pieces are listed in Table 15.

Table 14. Test 6 fragment velocities.

Footage	No. of Fragments Tracked	Min. Velocity (ft/s)	Max. Velocity (ft/s)	Ave. Velocity (ft/s)
High Speed	n/a	n/a	n/a	n/a
Standard	n/a	n/a	n/a	n/a
Drone	5	56.4	203.03	119.45

Table 15. Test 6 JPG fragments.

	Fragment 1	Fragment 2	Fragment 3
Range (ft)	31.60	74.87	100.48
Bearing (°)	301.89	145.63	156.63

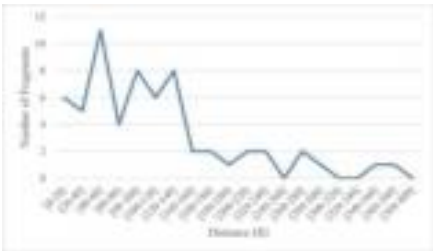


Figure 38. Test 6 fragment by distance.

Figure 37 is a scatterplot of the recovered fragments.

Figure 38 depicts the fragment distribution by distance for Test 6 and demonstrates a different trend than previous tests. The peak amplitude occurs outside of the 0 - 20 ft range even though there is a non-zero amplitude at that point.

Figure 39 and Figure 40 depict the fragment distribution by azimuth in 10° bands. Figure 39 shows the fragments that were above 5 g and found past 100 ft, while Figure 40 shows all the fragments above 5 g regardless of distance. The shaded areas represent the azimuths where most fragments were expected to land: 0°±10° and 180°±10°.

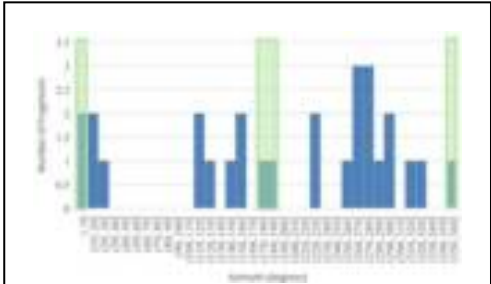


Figure 39. Test 6 fragments by azimuth past 100 ft.

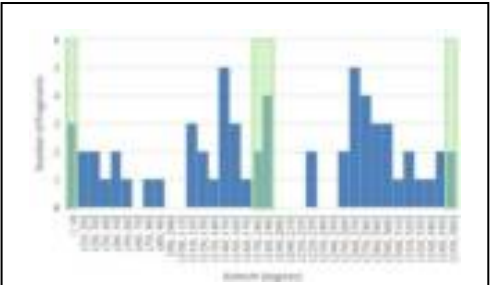


Figure 40. Test 6 fragments by azimuth.

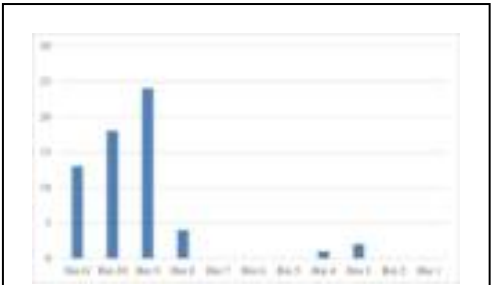


Figure 41. Test 6 fragments per mass bin.

Figure 41 shows how many fragments are in each mass bin. Most fragments fall in Bin 9. The fragments of the JPG shell are the entries for Bins 3 and 4.

3.7 Test 7

Test 7 took place on August 20, 2024, at 10:00 AM CDT (UTC -5). For this test, the JPG was positioned vertically, on a pallet at ground level, with the sub facing downwards into the pallet as shown in Figure 42.

3.7.1 Instrumentation Results

The pressure gauge readings and fragment velocity are summarised in Table 16 and Table 17.

Table 16. Test 7 pressure gauge readings.

Distance (ft)	Northern Gauges (psi)	Western Gauges (psi)
5	3.52	4.64
10	2.94	3.12
20	n/a	1.49

Table 17. Test 7 fragment velocities.

Footage	No. of Fragments Tracked	Min. Velocity (ft/s)	Max. Velocity (ft/s)	Ave. Velocity (ft/s)
High Speed	3	65.72	180.44	130.99
Standard	n/a	n/a	n/a	n/a
Drone	3	82.52	160.2	111.51

3.7.2 Fragmentation Results

In Test 7, 9,145 g or 96.3% of the initial mass was recovered. Figure 43 shows all the recovered fragments.

In Test 7, a total of 35 fragments with a mass greater than 5 g were recovered, of which 14 were recovered outside of 100 ft. The farthest piece was found at 297.96 ft, along the 311.96° azimuth. The outer shell was found at 1.69 ft, along the 277.26° azimuth. Figure 44 is a scatterplot of the recovered fragments.

Figure 45 depicts the fragment distribution by distance for Test 7 and follows a trend like Figure 38.

Figure 46 and Figure 47 depict the fragment distribution by azimuth in 10° bands. Figure 46 shows the fragments that were above 5 g and found past 100 ft, while Figure 47 shows all the



Figure 42. Test 7 setup.



Figure 43. Test 7 recovered fragments.

fragments above 5 g regardless of distance. The shaded areas represent the azimuths where most fragments were expected to land: $0^\circ \pm 10^\circ$ and $180^\circ \pm 10^\circ$.

Figure 48 shows how many fragments are in each mass bin. Most fragments fall in Bin 10. Like most previous tests, the Bin 2 entry is the outer shell of the JPG.

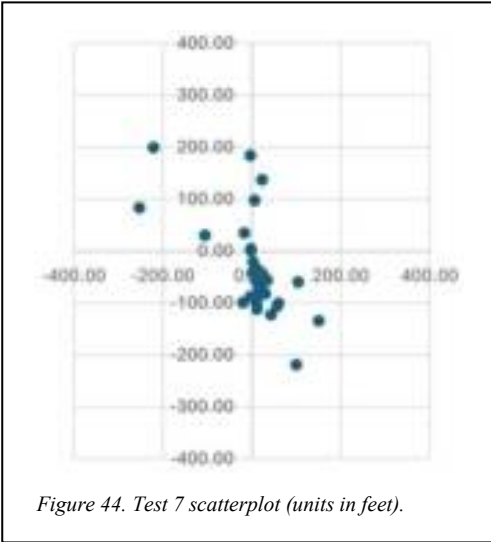


Figure 44. Test 7 scatterplot (units in feet).

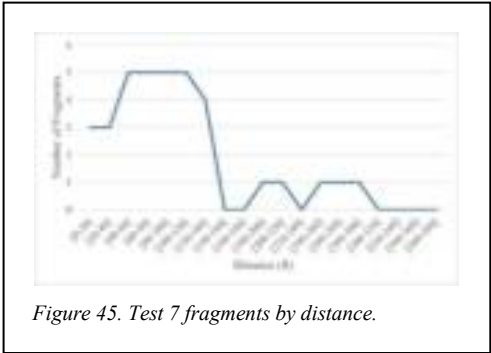


Figure 45. Test 7 fragments by distance.

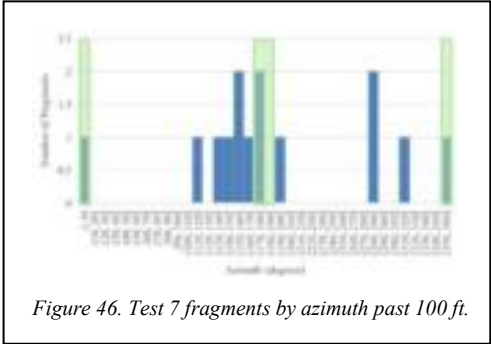


Figure 46. Test 7 fragments by azimuth past 100 ft.

3.8 Test 8

Test 8 took place on August 21, 2024, at 10:45 AM CDT (UTC -5). For this test, the six JPGs were positioned horizontally, on a pallet at ground level, in a 3-2-1 stack as shown in Figure 49.

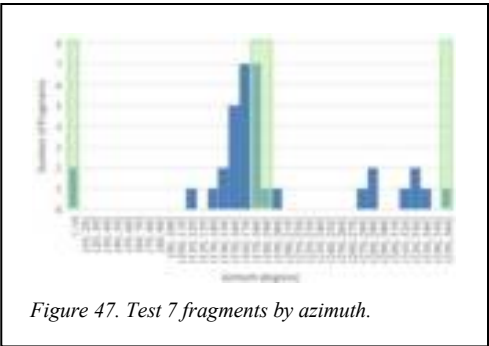


Figure 47. Test 7 fragments by azimuth.

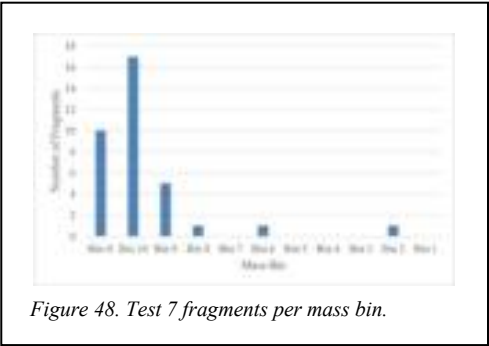


Figure 48. Test 7 fragments per mass bin.

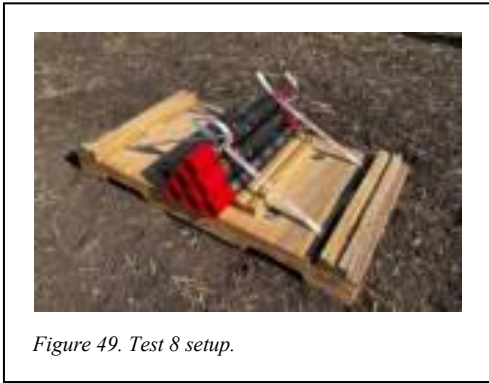


Figure 49. Test 8 setup.

3.8.1 Instrumentation results

The pressure gauge readings and fragment velocity are summarised in Table 18 and Table 19.

Table 18. Test 8 pressure gauge readings.		
Distance (ft)	Northern Gauges (psi)	Western Gauges (psi)
5	23.57	18.67
10	10.73	4.75
20	n/a	1.96



Figure 50. Test 8 recovered fragments.

3.8.2 Fragmentation results

In Test 8, 53,521 g or 94% of the initial mass was recovered. Figure 50 shows all the recovered fragments.

Table 19. Test 8 fragment velocities.

Footage	No. of Fragments Tracked	Min. Velocity (ft/s)	Max. Velocity (ft/s)	Ave. Velocity (ft/s)
High Speed	9	112.87	318.62	168.56
Standard	n/a	n/a	n/a	n/a
Drone	11	97.19	268.49	173.07

In Test 8, a total of 266 fragments with a mass greater than 5 g were recovered, of which 150

were recovered outside of 100 ft. The farthest piece was found at 393.94 ft, along the 351.71° azimuth. The exact range and bearing of each gun's outer shell are listed in Table 20.

Figure 51 is a scatterplot of the recovered fragments.

Figure 52 depicts the fragment distribution by distance for Test 8 and follows a trend like Figure 38.

Table 20. Test 8 JPG locations.

	Shell 1	Shell 2	Shell 3	Shell 4	Shell 5	Shell 6
Range (ft)	119.07	98.02	93.89	327.76	244.49	273.77
Bearing (°)	209.81	184.62	136.20	214.48	138.20	181.45

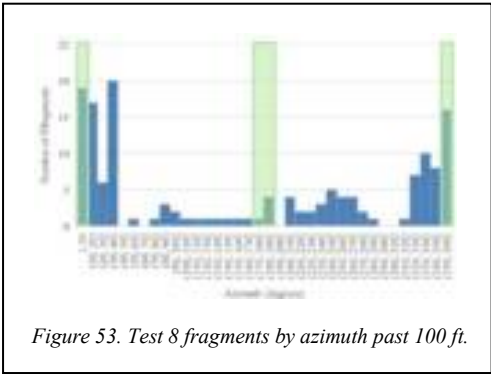
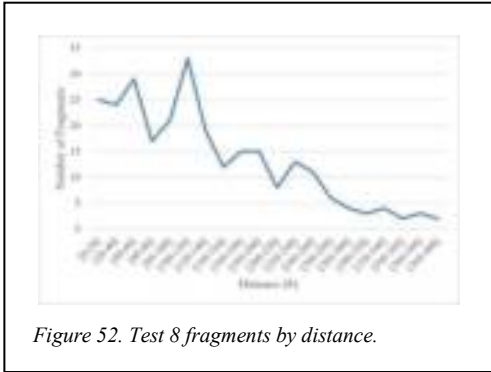
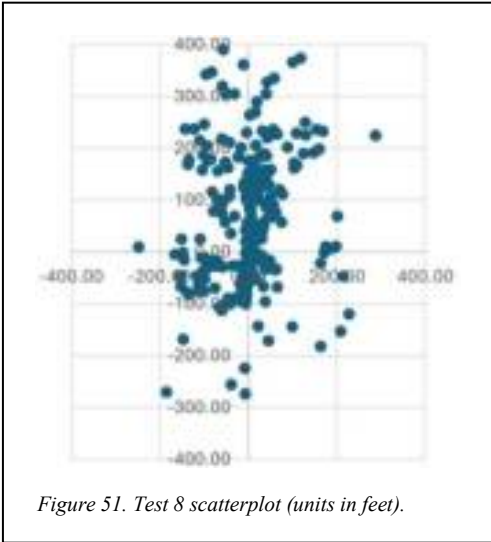
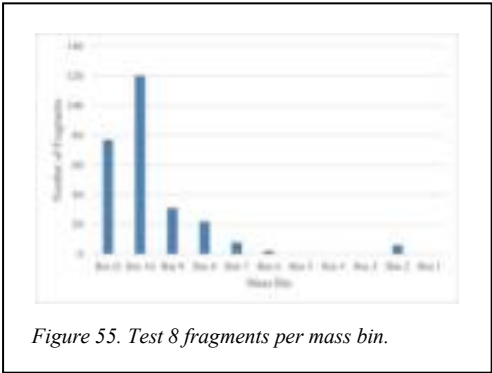
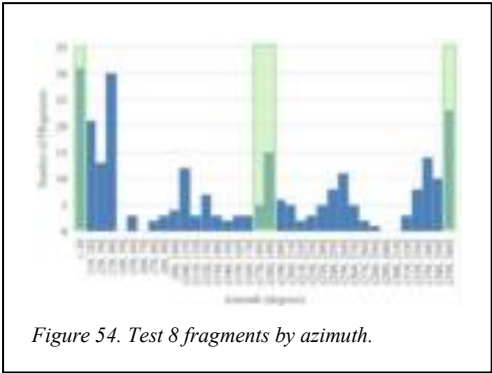


Figure 53 and Figure 54 depict the fragment distribution by azimuth in 10° bands. Figure 53 shows the fragments that were above 5 g and found past 100 ft, while Figure 54 shows all the fragments above 5 g regardless of distance. The shaded areas represent the azimuths where most

fragments were expected to land: $0^\circ \pm 10^\circ$ and $180^\circ \pm 10^\circ$.

Figure 55 shows how many fragments are in each mass bin. Most fragments fall in Bin 10. All the JPGs' outer shells fall in Bin 2.



4 MODEL COMPARISON

As stated in Section 1, the current IMESA FR JPG model is not based on empirical test data. Therefore, it is unknown how accurate the model is. To assess this, the test mass distribution and debris density were compared to the predicted values from IMESA FR. While these comparisons were made for every test, only a brief comparison between Test 8 and IMESA FR is covered in this paper.

Table 21 was made to compare the number of fragments recovered in each mass bin versus what IMESA FR predicts.

Table 22 shows the comparison of the debris density values that were calculated from the debris catalogs within five bands. These calculated values are compared to the IMESA FR predicted debris density at the midpoint of each band.

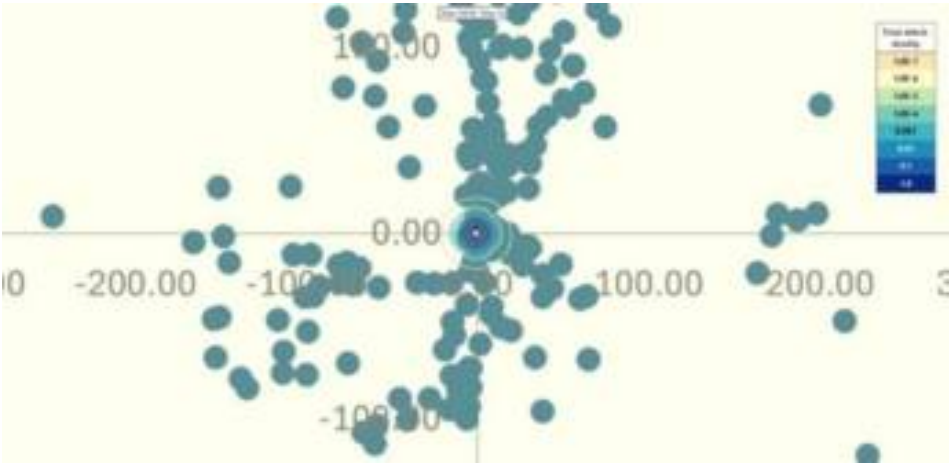


Figure 56. Debris density contour comparison (close-in).

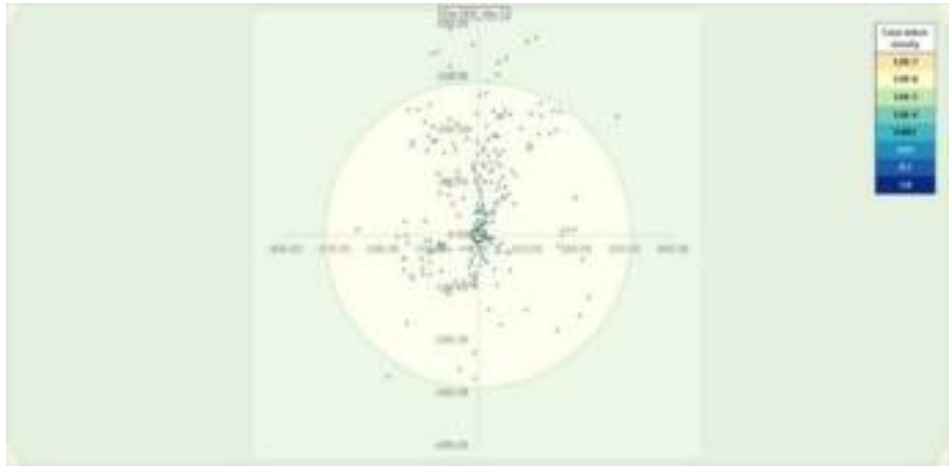


Figure 57. Debris density contour comparison.

Table 21. Mass distribution comparison.

Total Number of Fragments											
	Bin 1	Bin 2	Bin 3	Bin 4	Bin 5	Bin 6	Bin 7	Bin 8	Bin 9	Bin 10	
Test 8	0	6	0	0	0	2	8	22	31	120	
Model el	0	0	0	0	0	0.58	2.77	6.41	11.5		

Table 22. Debris density comparison.

	0 - 20 ft	20 - 100 ft	100 - 200 ft	200 - 300 ft	300 - 400 ft
Test	3.04E-03	2.20E-04	8.00E-05	2.51E-05	5.12E-06
Model	7.89E-03	2.03E-06	4.17E-06	7.52E-06	1.07E-05

To accompany Table 22, Figure 56 and Figure 57 were created to compare the Test 8 scatterplot to the total debris density contour that IMESA FR creates.

Table 22 and the accompanying figures show that in most cases, IMESA FR is underpredicting the debris density. However, IMESA FR is overpredicting the debris density in the 300-400 ft band. Furthermore, while it cannot be seen in the images, IMESA FR is predicting extremely low debris densities as far out as 1,800 ft. While no fragments were recovered past 400 ft in Test 8, that does not mean that IMESA FR is wrong in predicting that there is a probability that fragments could travel that far. For example, in Test 1 (a single item test) it was confirmed that the bulkhead made it past 1,300 ft and was never recovered, so it is possible for a fragment to travel that far, even if the likelihood of it happening is extremely low, as IMESA FR predicts.

5 FUTURE PLANS

As stated in Section 2, the original plan for this test series had more tests envisioned but had to be limited to eight tests due to time, funding, and range constraints. Therefore, it was determined that more testing will need to be done in the future. Specifically, testing needs to be done to observe the effects of different gun sizes, phasing angles, number of shots-per-foot, and total number of charges, as all these parameters may influence the behavior of the JPG during detonation. In addition to the remaining parameters that need to be tested, it is desirable to perform further testing to fully capture the behaviour of the bulkhead, since that was not accomplished in Test 1.

6 SUMMARY AND CONCLUSIONS

To summarise, the primary goal of this test series was to collect empirical data that could be used to improve IMESA FR, especially the range, bearing, and mass of the JPG fragments. The test results will be used to improve the existing IMESA FR JPG model. Originally, it was believed that the results from this test series would be used to reduce conservatism in the model. However, upon comparing the test results to the model, it was determined that the model was conservative in some areas but non-conservative in other aspects. Overall, this test series has provided a significant amount of data that can be used to begin the process of looking into possible improvements of the model. However, there are numerous additional parameters that still need to be tested.

REFERENCES

- Bureau of Alcohol, Tobacco, Firearms and Explosives (ATF). Table of Distances*, 26 October 2021.
- Institute of Makers of Explosives (IME) 2017. Guide to obtaining DOT approval of jet perforating guns using AESC/IME perforating gun specifications Version 02*, 1 September 2017.
- Swisdak, M.M., Tatom, J.W. & Conway, R.T. 2017. Procedures for the collection, analysis, and interpretation of explosion-produced debris - Revision 2, *DDESB Technical Paper 21*, 30 November 2017.
- Tatom, J., Evans, B., Hoffman, J., Fritz, C., Duncan, M. & Robinson, M. 2018. IMESA FR Overview. In *National Defense Industrial Association (NDIA), International Explosives Safety Symposium & Exposition*, San Diego, CA, August 2018.

[RETURN TO CONTENTS PAGE](#)

7. Innovation and technical development

An empirical study on the estimation of detonation pressure by separation distance of electronic detonators using the underwater explosion impact

H. Kim, S. Kim, J. Choi & Y. Lee

Hanwha/Global Corporation, Seoul, South Korea

S. Oh, J. Choi & B. Choi

Korea Institute of Geoscience and Mineral Resources, Daejeon, South Korea

ABSTRACT: With the increasing demand for mining operations in the mining industry, the importance of electronic detonators, which offer advantages in terms of safety, precision, and efficiency has been growing. Unlike conventional detonators, electronic detonators, controlled by internal circuit boards, are well known to be highly susceptible to dynamic pressure. However, a standardised testing method for evaluating the impact resistance of electronic detonators has not yet been established. This study established a foundation for evaluating the dynamic pressure resistance of electronic detonators based on the Underwater Explosion Impact System (UEIS). During the experiments, the peak pressure at varying distances from the explosion source was measured, and the detonation behaviours in the underwater condition was observed by high-speed camera system. As a result, experiential expression on the maximum pressure was suggested according to the separation distance. These results were compared with numerical simulation results obtained using ANSYS LS-DYNA software.

1 INTRODUCTION

With the growing demand for mining operations and the increasing use of underground structures, the importance of electronic detonators offering advantages in safety, precision of delay times, and efficiency during blasting has been rising. Unlike conventional detonators, electronic detonators controlled by internal circuit boards are known to be highly susceptible to dynamic pressure, especially from adjacent blast holes. Although several studies have evaluated their resistance to dynamic pressure using mechanical impact devices and underwater explosion (UNDEX) systems, a standardised testing methodology has yet to be

clearly established. Moreover, existing empirical models and measurement data largely originate from military applications or large-scale environments, limiting their applicability in supporting standardised evaluation methods.

Previous studies have examined underwater explosion dynamics and pressure measurement techniques under conditions. Zhuang *et al.* (2019) investigated the dynamic behaviour of slender explosive charges in underwater conditions, providing insight into shape-dependent pressure properties. Sheng *et al.* (2021) analysed the effects of bubble jet impact loading on submerged structures, while Chambers *et al.* (2000) focused on the pressure response of deformable surfaces

subjected to underwater explosions. Moon *et al.* (2017) addressed challenges in sensor calibration and uncertainty quantification in underwater pressure measurements, and Zhuang *et al.* (2020) further evaluated the structural damage mechanisms of reinforced concrete columns exposed to underwater explosions. More recently, Farnfield *et al.* (2024) examined the effects of underwater shock and gas pressure on electronic detonators, identifying primary failure modes. While these studies have significantly contributed to understanding various aspects of underwater explosion phenomena, it often lacks comprehensive validation results necessary for establishing standardised experimental systems in controlled environments.

To overcome these limitations, this study conducted empirical research based on experimental measurements and numerical simulations to establish a foundation for evaluating the dynamic pressure resistance of electronic detonators under underwater explosion conditions. A designed Underwater Explosion Impact System (UEIS) was developed to conduct controlled water explosive experiments using the HITRONIC II electronic detonator, enabling accurate pressure measurement and high-speed visualisation of detonation behaviour across various stand-off distances. Based on the acquired data, an empirical equation was derived to estimate peak detonation pressure as a function of distance. Furthermore, two-dimensional axisymmetric simulations were performed using the LS-DYNA MMALE method, with model calibration conducted through adjustment of the JWL equation of state parameters. As a result, comparative analysis between experimental and simulation results confirmed high agreement in pressure magnitudes and waveform properties. Through this combined approach, the study aims to contribute to the establishment of standardised methodologies for evaluating electronic detonator performance in confined underwater environments.

2 EXPERIMENTAL SETUP

2.1 Underwater explosive impact system

To evaluate the resistance of an electronic detonator to dynamic pressure under underwater conditions, an Underwater Explosion Impact System (UEIS) was developed, consisting of an explosion chamber, a dynamic pressure source and a measurement system.

During chamber development, cubic, cylindrical and rectangular geometries were considered to optimise structural integrity, measurement efficiency and fabrication ease. Although cylindrical chambers offer stress distribution stability, a cubic design was selected for its simpler fabrication and better internal accessibility. Final specifications were based on the required pressure measurement range for deriving empirical stand-off distance-pressure relationships. One-dimensional wave propagation theory was applied to simplify shock wave analysis, where the shock velocity C_s , depends on the initial sound speed in water C_0 , and the pressure differential, as expressed as such Equation (1) (Landau *et al.* 2008):

$$C_s = C_0 \left(1 + \frac{\gamma + 1}{2\gamma} \frac{P}{P_0} \right)^{1/2} \quad (1)$$

Where: C_s is the shock wave velocity, C_0 is the sound speed in water under isothermal conditions, γ is the specific heat ratio of water (approximately 7.15), P is the shock pressure and P_0 is the hydrostatic pressure.

Based on one-dimensional wave propagation theory, the time t required for a shock wave to reach a target distance d in water is given by Equation (2):

$$t = \frac{d}{C_s} \quad (2)$$

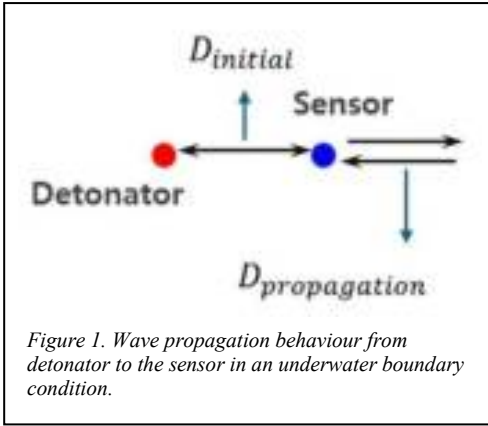
Assuming the boundary conditions illustrated in Figure 1, the time T_1 for a shock wave to travel from the detonator to the sensor (distance d_1) and the time T_2 for the wave to travel from the sensor to the boundary (distance d_2), are expressed by Equations (2) and (3), respectively:

$$t_1 = \frac{d_1}{C_s} \quad (3)$$

$$t_2 = \frac{d_2}{C_s} \quad (4)$$

Accordingly, in a confined underwater experimental setup, the time T_1 required to fully measure the pressure waveform after the shock wave reaches the sensor and the time T_2 for the reflected wave to return to the sensor from the boundary, are defined by Equations (5) and (6) respectively:

$$T_1 = t_1 + t_{wave} \quad (5)$$



$$T_2 = t_1 + 2t_2 \quad (6)$$

Where: T_{wave} represents the single-wave duration of the detonation pressure waveform.

Consequently, to prevent the interference of shock waves in a confined underwater experimental setup with boundary constraints, the condition described by Equation (7) must be satisfied:

$$T_1 \geq T_2 \quad (7)$$

In general, several factors influence the duration of underwater shock waves, including explosive energy, stand-off distance, medium

properties, attenuation mechanisms, and wave reflection or superposition. However, due to the presence of uncertain parameters and complex interdependencies, theoretical evaluation of this duration remains challenging. Therefore, in this study, the value of T_{wave} wave was empirically set to 0.4 ms, based on previously reported measurements from similar explosive configurations (Choi *et al.* 2017, Hamashima *et al.* 2010, Ma *et al.* 2021, Moon *et al.* 2017, Nowak *et al.* 2023, Zhuang *et al.* 2019, Zhuang *et al.* 2020).

Based on the superposition analysis, the 1000 mm cubic chamber was selected as the optimal configuration, balancing measurement reliability and experimental efficiency. Figure 2 illustrates the shock wave superposition ratio (%) at each stand-off distance, calculated based on the analysis of overlapping waveforms.

The underwater explosion chamber consists of four main components, including an upper frame for securing the detonator and sensors and an acrylic observation section. The range of stand-off distances for shock wave measurement, one of the primary experimental targets, was defined as less than 500 mm, based on the preceding shock wave superposition analysis, and the chamber was designed in a cubic form (1.0 m × 1.0 m × 1.0 m) for spatial efficiency. SUS304 stainless steel was used for durability under repeated underwater detonations. A PC sensor holder was also

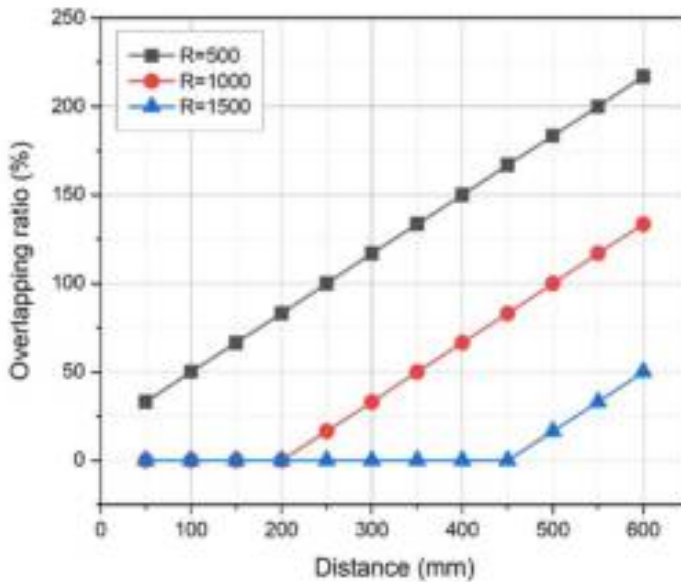


Figure 2. Results of shock wave ratio estimation by stand-off distance and chamber specification.

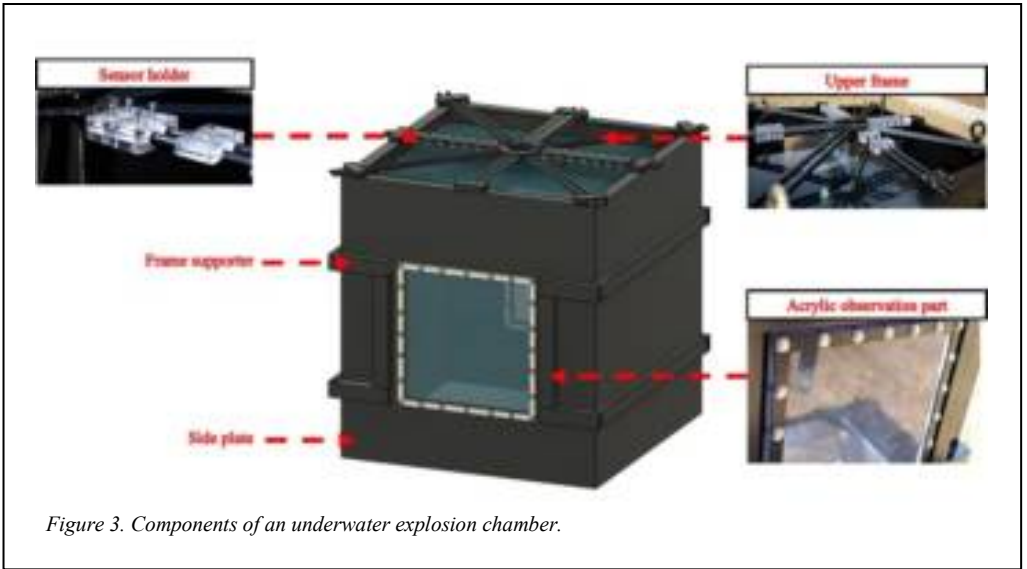


Figure 3. Components of an underwater explosion chamber.

implemented to ensure precise control of stand-off distances. Figure 3 shows the completed underwater explosion chamber of UEIS.

2.2 Underwater explosion pressure measurement

To derive an empirical equation for detonation pressure as a function of stand-off distance, a series of underwater detonation pressure measurements was conducted. The experimental setup consisted of two primary subsystems, namely the underwater explosion system and the data measurement system.

For high-speed visual observation, a Phantom VEO E310L high-speed camera was employed. The camera settings, including resolution, focal

length and frame rate, were appropriately adjusted to ensure clear visualisation of the detonation behaviour. It was also measured with the pressure acquisition equipment via a central controller, enabling simultaneous recording of both visual and pressure data.

Pressure signals generated by the detonation were acquired using a Lecroy WaveSurfer 4024 oscilloscope, which offers a minimum sampling rate of 2 GS/s to accurately capture the rapid transient shock wave signals. A dummy detonator was directly connected to the oscilloscope to serve as the trigger source, ensuring precise timing of signal acquisition. Pressure sensors were connected through dedicated amplifiers before being interfaced with the oscilloscope. Figure 4



Figure 4. Equipment used to measure underwater detonations (a) Lecroy Wavesufer 4024 (b) Phantom VEO E310L.

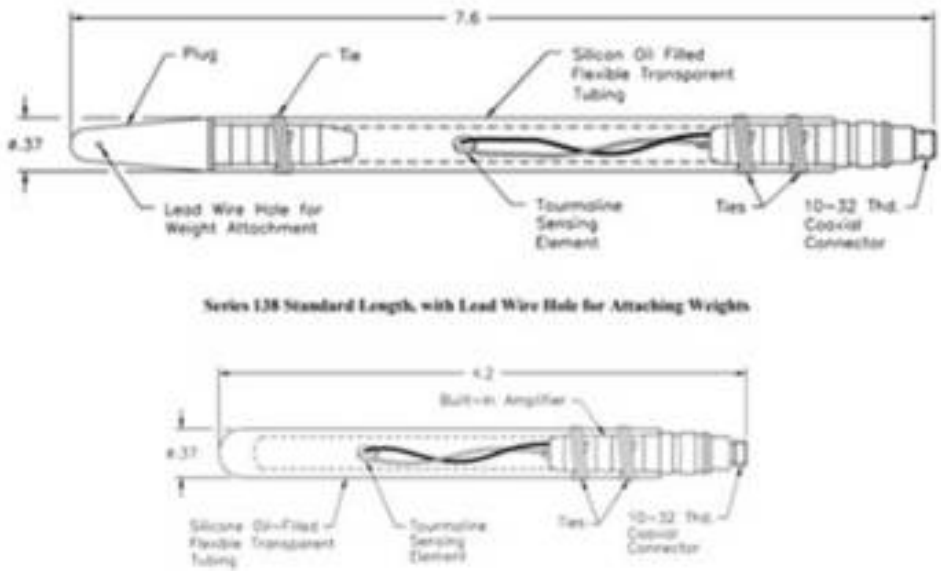


Figure 5. Diagram of PCB 138A series from PCB Piezotronics.

shows the oscilloscope and high-speed camera used in the experiment.

To measure detonation pressure in water, a fluid dynamic pressure sensor based on tourmaline

technology, the PCB 138A series from PCB Piezotronics, was employed. This piezoelectric sensor has demonstrated validated sensitivity and reliability in previous studies, and its maximum

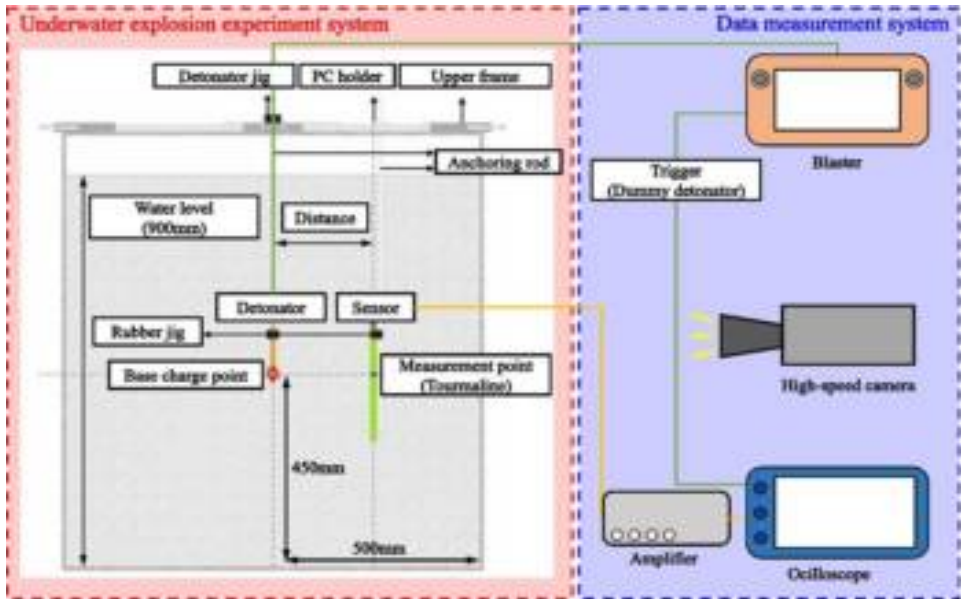


Figure 6. Schematic diagram of the underwater explosion pressure measurement setup.

measurable pressure of approximately 344 MPa makes it highly suitable for capturing high-intensity shock waves generated by underwater detonations (Figure 5).

Table 1. The number of tests at each stand-off distance.

Distance (mm)	Point case
40	4
45	6
75	10
100	11
125	9
150	12
175	12
200	16
225	16
250	15
275	15
300	16
325	16
350	17
375	16
400	27
435	15

During the underwater explosion tests, the chamber was filled with water up to approximately 900 mm from the bottom surface. This water level was consistently maintained throughout all experiments to ensure repeatability and measurement consistency. Pressure sensors were then installed at predefined stand-off distances by attaching them to rubber jigs mounted on vertical support rods. In the same method, the electronic detonator, which served as the explosive source, was fixed to a rubber jig connected to the central ring structure positioned at the centre of the upper frame. Once the installation positions were determined, the support rods are lowered so that both the base charge point of the detonator and the sensing point of the piezoelectric pressure sensor are aligned at a water depth of 450 mm. The electronic detonator used in this study was the Hitronic II model, which was connected to the HEBS II blaster for initiation. Figure 6 shows a schematic diagram of the underwater explosion pressure measurement setup.

2.3 Experimental condition

In this study, to derive an empirical equation for detonation pressure as a function of stand-off distance, a total of 248 underwater explosion tests were conducted. The tests covered a wide range of stand-off distances, from 40mm to 435mm, encompassing both near-field and far-field regions. For each distance, repeated trials were performed to ensure statistical consistency and data reliability. Table 1 summarises the number of tests conducted at each stand-off distance.

During the experiments, the Phantom VEO E310L high-speed camera was configured with a resolution of 512 × 384 pixels and a frame rate of 15,000 frames per second. A 50 mm focal length lens was selected to balance field of view and magnification, enabling optimal observation of the detonation behaviour and the formation of air bubbles. The exposure time was set to the minimum value to clearly capture the transient phenomena.

3 EXPERIMENTAL RESULTS

3.1 Detonation pressure analysis

To analyse the time-pressure history of underwater shock waves generated by detonator initiation, representative experimental results at each stand-off distance were selected and compared based on their detonation pressure-time curves. Figure 7 presents the measured pressure-time histories for various stand-off distances, plotted on a same time axis, clearly showing an attenuation trend in which the peak detonation pressure gradually decreases with increasing stand-off distance. This tendency is consistent with theoretical and empirical models, confirming that the measured results fall within a physically valid range. The waveforms exhibited similar patterns across all stand-off distances, characterised by a rapid pressure rise immediately after detonation followed by a gradual decay. Notably, the pressure rise rate during the initial phase was approximately 28,896.55 GPa/s, while the decay rate during the falling phase was 12,804.80 GPa/s, both of which are comparable to typical underwater explosion profiles.

3.2 Bubble behaviour analysis

Underwater explosion of explosives or pyrotechnic materials generates a spherical gas bubble that undergoes pulsating behaviour due to the interaction between internal gas pressure and

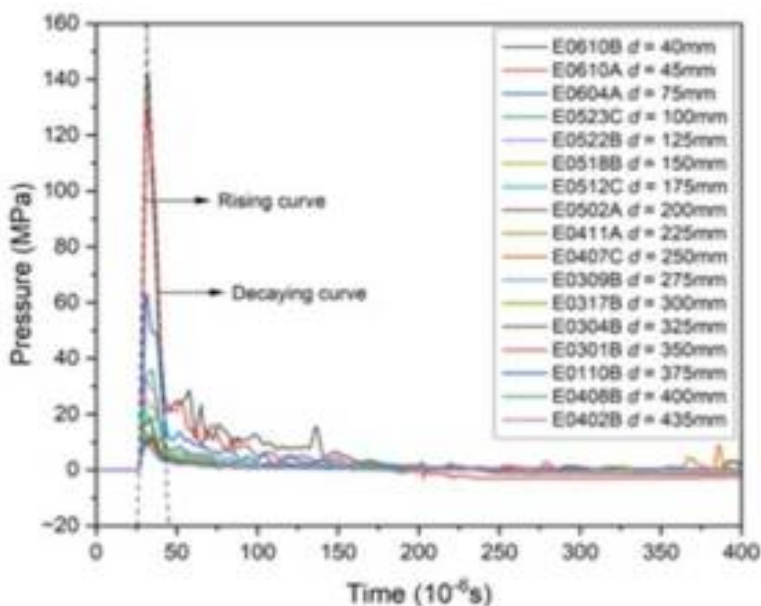


Figure 7. Pressure-time history curves at various stand-off distances.

external hydrostatic pressure. In particular, the contraction phases can generate secondary and tertiary shock waves, making it essential to analyse bubble behaviours to capture pressure properties over an extended time window and ensure data reliability.

Figure 8 shows high-speed camera frames capturing the primary collapse and pulsating phenomena observed in this study. The bubble expanded rapidly within a few hundred microseconds after explosion, with expansion continuing for 10 - 15 ms. The primary shock

wave was generated at 200 - 300 μ s and reached the chamber boundary at 400 - 500 μ s, indicating the completion of the main energy transfer. The bubble then expanded to its maximum size between 10,000 and 15,000 μ s, collapsed around 25,000 μ s, and re-expanded due to residual internal energy.

3.3 Determination of and empirical equation for explosion pressure prediction

In underwater explosion environments, explosion



Figure 8. Representative bubble dynamics captured from underwater detonation of an electronic detonator.

Table 2. Regression results of detonation pressure estimation equations based on mathematical models.

Model type	Empirical equation	Coefficient of determination	RMSE (MPa)
Power-law	$P(R) = 8653.63R^{-1.154}$	0.962	5.03
Exponential	$P(R) = 70.4984e^{-0.0026R}$	0.607	16.20
Modified power-law	$P(R) = \frac{3196.12}{(R - 17.62)^{0.9888}}$	0.984	3.25
Scaled model	$P = 312568.66 \left(\frac{W^{1/3}}{R} \right)^{1.2724}$	0.980	3.70

pressure exhibits a nonlinear attenuation behaviour according to stand-off distance. To quantitatively predict this behaviour, various empirical and semi-theoretical models have been developed. Typically, such predictive models are derived through regression analysis based on experimental data. The most commonly used equations can be categorised into several types: (1) power-law models based on classical studies such as those by Cole (1948) and Swisdak (1978), (2) exponential models, which are particularly effective in the near-field region where rapid attenuation occurs, (3) modified power-law models that incorporate distance-correction or logarithmic terms into power-law expressions, and (4) accumulation-type models developed through dimensional analysis.

In this study, regression analysis was performed based on the four mathematical models described above to derive an empirical prediction equation fitted to the experimental data.

Table 2 presents the derived empirical equations and the reliability analysis results for the four mathematical models, where R represents the stand-off distance (mm), P is the peak detonation pressure and W refers to the charge weight (kg). The power-law and exponential models showed relatively low coefficients of determination, with R² values of 0.962 and 0.607, respectively, indicating limited agreement with the experimental data. In contrast, the modified power-law model, which introduces a distance correction term c, more accurately captured the nonlinear attenuation

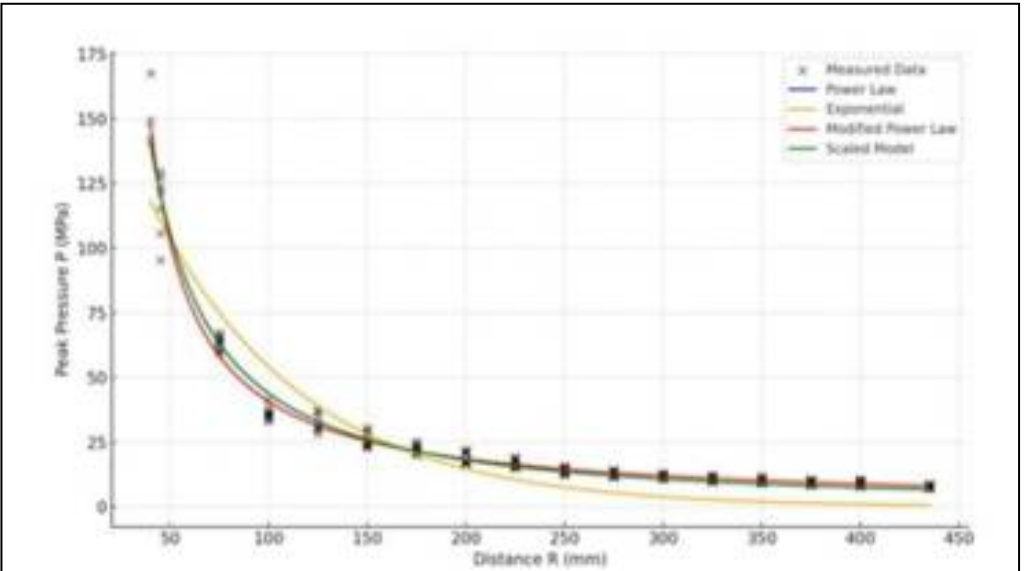


Figure 9. Comparison of regression results for detonation pressure estimation using four mathematical models.

behaviour of detonation pressure. As also noted in previous studies by Swisdak (1978) and Yuan *et al.* (2019), the inclusion of this correction term can enhance both the regression fit and the physical interpretability of the model. However, in this analysis, the derived value of c was -17.62 , which may raise concerns regarding physical realism. The scaled model, on the other hand, exhibited a high coefficient of determination ($R^2 = 0.98$) and a low root mean square error ($RMSE = 3.7$ MPa), demonstrating both quantitative reliability and physical consistency. Therefore, the scaled model is considered a suitable predictive tool for evaluating stand-off-distance-based shock resistance performance. Figure 9 presents a comparison of regression results for detonation pressure estimation based on different mathematical models.

4 COMPARISON WITH NUMERICAL ANALYSIS RESULTS

LS-DYNA, developed by the ANSYS Corporation, was employed in this study to simulate underwater explosion (UNDEX) and evaluated detonation pressure as a function of stand-off distance. In the development of electronic detonators, a key challenge lies in understanding the behaviour of internal components such as circuit boards and fuseheads under dynamic loading. Due to the extreme pressure, the extremely short response time and the presence of a protective shell, direct experimental observation of internal structural behaviour is virtually impossible. Consequently, numerical simulations offer a valuable alternative, enabling detailed analysis of the mechanical response of internal materials under dynamic loading. These simulations provide essential insights into shock propagation mechanisms and

play a crucial role in guiding the design optimisation of detonator structures.

To simulate underwater explosion phenomena involving small-scale explosives such as detonators, the Multi-Material Arbitrary Lagrangian-Eulerian (MMALE) method was employed. The MMALE approach was selected for its ability to represent multiple materials including explosives, water and air within a single eulerian mesh. This feature makes it particularly suitable for addressing future problems involving the dynamic structural response of electronic detonators and fluid-structure interactions induced by shock waves.

Table 3. Material keyword *Mat_Null defined in air and water material.

	Density (kg/m ³)	Pressure cut off (Pa)
Air	1.225	0
Water	998	0

Tables 3, 4 and 5 present the material properties used for air and water in the simulation. Both fluids were modelled using the *MAT_NULL keyword, which allows the mechanical response to be governed entirely by an equation of state (EOS) without internal strength considerations. For air, *EOS_LINEAR_POLYNOMIAL was adopted to represent its relatively simple and linear compressibility properties, providing computational efficiency while maintaining physical accuracy. In contrast, water was modelled using *EOS_GRUNEISEN to account for its highly nonlinear compressibility under dynamic loading conditions.

Table 4. EOS keyword *EOS_Linear Polynomial defined in air material.

	C_0 (Pa)	C_1 (Pa)	C_2 (Pa)	C_3 (Pa)	C_4 (Pa)	C_5 (Pa)	C_6 (Pa)	E_0 (Pa)	V_0
Air	0	0	0	0	0.4	0.4	0	245000	1

Table 5. EOS keyword *EOS_Gruneisen defined in water material.

	C (m/s)	S_1	S_2	S_3	γ_0	A	A (Pa)	V
Water	1647	1.921	0	0	0	0	0	1

Table 6. Material keyword **Mat_High_Explosive_Burn* defined in explosive material.

	Density (kg/m ³)	Detonation velocity (m/s)	C – J pressure (GPa)
PETN	1500	7450	22

Table 7. EOS keyword **EOS_JWL* defined in explosive material.

	A (GPa)	B (GPa)	R ₁	R ₂	ω	E ₀ (GPa)	V ₀
PETN	625.3	23.29	5.25	1.6	0.28	6200	1

Tables 6 and 7 summarise the material properties assigned to the explosive used in the underwater explosion simulation. The explosive was modelled using the **MAT_HIGH_EXPLOSIVE* material model in conjunction with the **EOS_JWL* equation of state, enabling accurate representation of the thermodynamic and pressure-driven behaviour of condensed explosives. The properties were based on PETN (Pentaerythritol tetranitrate), a widely used high-performance explosive known for its high detonation velocity and brisance. In this study, the initial internal energy parameter E_0 in the JWL equation was systematically reduced to calibrate simulation results against experimental data. As E_0 can vary depending on factors such as material purity, particle morphology, and measurement conditions. Although typically derived from empirical data (Urtiew *et al.* 1991) or tests like cylinder expansion, the JWL parameters including E_0 are subject to empirical refinement. Accordingly, this calibration enhances simulation accuracy without compromising the physical validity of the detonation model.

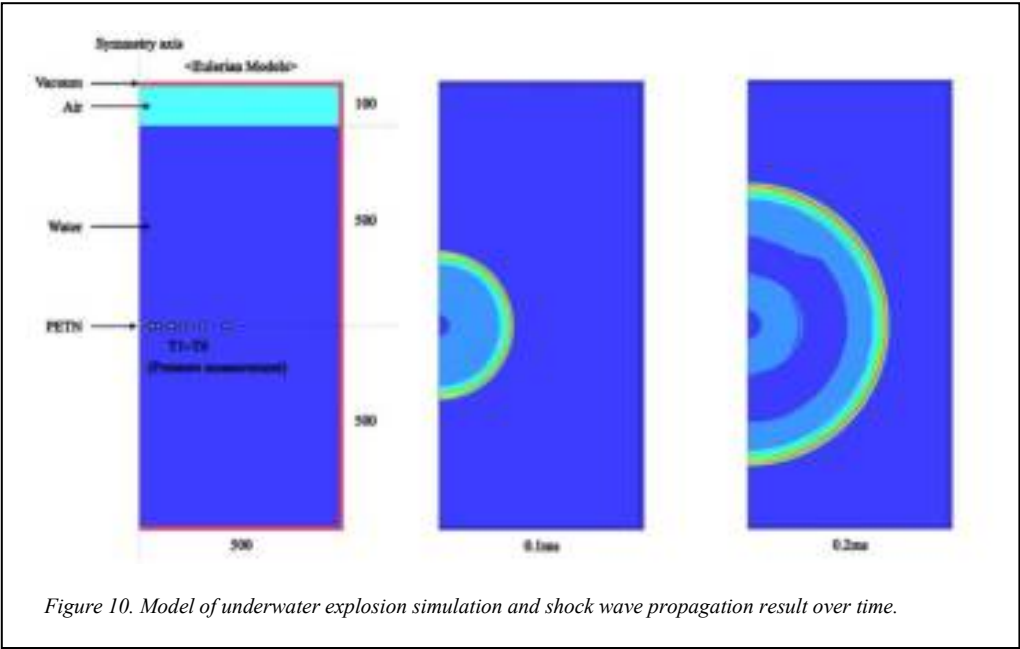
In this study, a two-dimensional axisymmetric simulation approach was adopted to evaluate material properties, define the ALE domain, and perform mesh size sensitivity analyses. Among the various factors influencing the accuracy and computational efficiency of ALE simulations, mesh configuration plays a particularly critical role. In ALE-based analyses, coarse meshes tend to distort the propagation of physical quantities between elements, whereas fine meshes significantly increase the computation. To address the trade-off between accuracy and computational efficiency, a series of simulations was performed

using quad-element models with mesh sizes ranging from 1 mm to 10 mm. The explosive was represented as a cylindrical geometry with a radius of 4 mm and a height of 10 mm, reflecting the actual physical configuration. Based on this parametric mesh sensitivity study, the optimal mesh size was determined, and the final simulation conditions are summarised in Table 8.

Table. 8 Summary of mesh characteristics and explosive properties in the finalized simulation.

	Conditions
Mesh shape	Quad
Mesh size	1mm x 1mm
Explosive Material	PETN
Explosive shape	ϕ 8mm x H10mm (Cylinder)

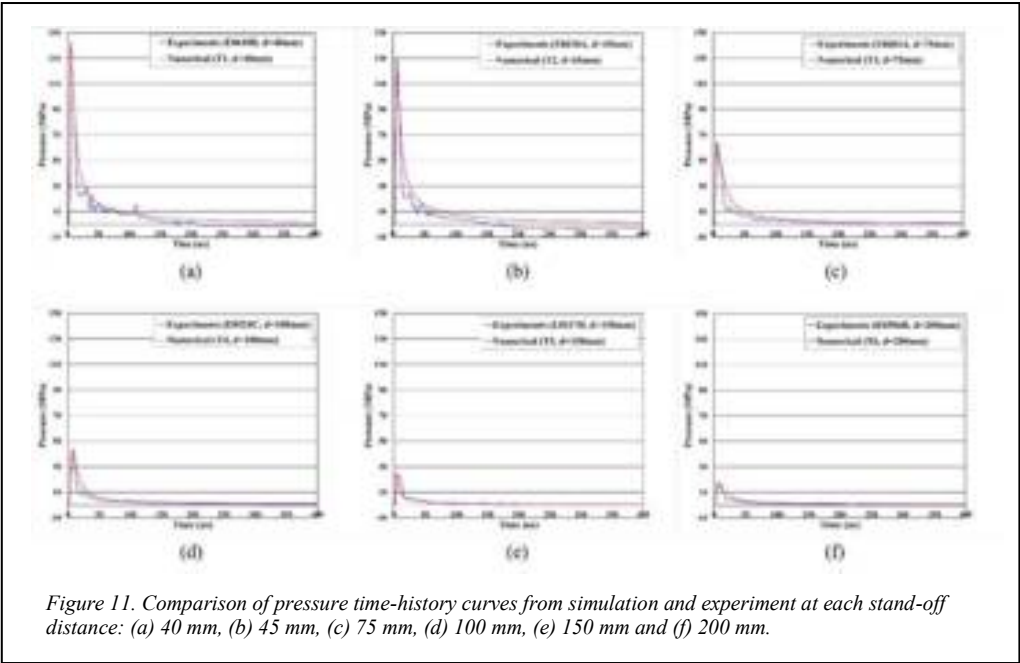
The geometric configuration of the MMALE simulation model is illustrated in Figure 11. This model consists of four distinct regions: explosive, water, air and a vacuum zone, which serves as a non-reflecting boundary. To closely replicate the experimental setup, a cylindrical PETN charge with a radius of 4 mm and a height of 10 mm is placed at the origin. The surrounding water domain extends 500 mm in width and 1000 mm in height, with the uppermost 100 mm above the water surface filled with air. The **INITIAL_VOLUME_FRACTION_GEOMETRY* keyword is employed to define these fluid regions. Explosive is initiated at the



origin using the *INITIAL_DETONATION keyword to simulate the underwater explosion phenomena. To validate the simulation model, the pressure results obtained from the simulation were compared with experimental measurements. Pressure sensors labeled T1 through T6 were positioned at standoff distances of 40 mm, 45 mm,

75 mm, 100 mm, 150 mm and 200 mm, respectively, to record the resulting pressure data.

The resulting pressure distribution from the underwater explosion is shown on Figure 10. Upon explosion, a spherical compression wave is generated and rapidly propagates outward toward the boundaries of the model. Following this initial



shock wave, the formation and expansion of a spherical gas bubble near the charge center is observed, reproducing primary features of the underwater explosion behaviour.

Table. 9 Comparison of peak pressures between experimental Scaled Model and numerical results.

Peak pressure results (MPa)			
Stand-off distance (mm)	Experimental (Scaled Model)	Numerical	Percentage error (%)
40	140.45	142.74	1.63
45	120.90	122.46	1.29
75	63.12	63.81	1.10
100	43.77	44.24	1.08
150	25.26	25.26	3.33
200	18.12	17.43	3.79

Figure 11 presents a comparison between the simulated pressure responses and representative experimental measurements at each sensor location. The simulated pressure profiles replicated the primary features observed in the experiments, including the initial sharp pressure rise, the subsequent nonlinear attenuation behaviour and the peak pressure amplitudes. In particular, the numerical results consistently captured the rapid pressure increase immediately following explosion, the nonlinear decay in the pressure tail and the overall waveform shape across all sensor positions. The simulated pressure-time histories exhibited agreement with experimental data, with peak pressure magnitudes and waveform features showing high similarity. To validate the simulated peak pressure values, a comparative analysis was conducted against the empirically derived Scaled Model introduced earlier. As shown in Table 9, the peak pressures obtained from simulation and experiment at stand-off distances within 200 mm exhibited a percentage error of less than 5%. Such accuracy demonstrates the reliability and scalability of the simulation for analysing the dynamic structural response of electronic detonators in future close-range underwater explosion scenarios.

5 CONCLUSIONS

This study conducted an empirical investigation to evaluate the dynamic pressure resistance of

electronic detonators under underwater explosion conditions. A custom-designed Underwater Explosion Impact System (UEIS) was developed to enable controlled experiments, allowing for precise measurement of detonation pressure and visualisation of detonation behaviour across various stand-off distances. A total of 248 tests were performed, and a scaled empirical equation was derived to predict peak pressure as a function of distance. To extend and reinforce these results, two-dimensional axisymmetric simulations were carried out using the MMALE method in LS-DYNA. The simulation model was constructed incorporating the explosive charge, surrounding water and air domains, as well as appropriate boundary conditions especially. Calibration of the JWL equation of state particularly through systematic adjustment of the initial internal energy parameter enhanced the agreement between numerical predictions and experimental observations. As a result, comparative analysis revealed strong correlation between the simulated and measured pressure-time histories, with peak pressure discrepancies remaining within 5% in the near-field region. The combined methodology established in this study comprising empirical pressure modeling, calibrated simulation, and validated pressure-time histories provides a foundation for the standardised evaluation of electronic detonators in confined underwater environments. However, future research should focus on extending the simulation by incorporating fluid-structure interaction (FSI) techniques to analyse the dynamic response of internal components, along with experimental validation to ensure model reliability.

6 ACKNOWLEDGEMENT

This research was supported by the project titled "Development of an Underwater Explosion Chamber for Electronic Detonators and Estimation Formula for Underwater Detonation Pressure Based on Standoff Distance" (Project Code: IP2024-039), funded by Hanwha Corporation. We would like to express our sincere gratitude to Hanwha Corporation for their financial support.

REFERENCES

Chambers, J.P., Holmes, D.G. & Plohr, B.J. 2000. Pressure measurements on a deforming surface in response to an underwater explosion. *Shock and Vibration*, 7(2), 101–112.

- Choi, G., Kim, Y. & Lee, J. 2017. Underwater explosion experiment using pentolite. - 206 - *Journal of the Korean Society of Safety*, 32(5), 101–106.
- Cole, R.H. 1948. Underwater explosions, Princeton, NJ: *Princeton Univ. Press*.
- Farnfield, R., Bermingham, L. & Adderley, G. 2024. Shock and transient gas pressure damage to electronic detonators. *The Journal of Explosives Engineering*, 41 (5), 28–36.
- Hamashima, H., Shibuta, M., Nishimura, Y. & Itoh, S. 2010. Behaviour of bubble pulse in food processing using underwater shock wave. *International Journal of Multiphysics*, 4(2), 113–124.
- Huang, L.L., Ma, H.H., Shen, Z.W., Chen, H.J., Wang, Y.X. & Chen, J.P. 2021. Effects of content of titanium fibre on detonation characteristics and mechanical properties of RDX explosive. *Journal of Energetic Materials*, 40(4).
- Landau, L.D. & Lifshitz, E.M. 1987. *Fluid Mechanics. 2nd Edition*, Pergamon Press, Oxford.
- Moon, H., Choi, J. & Kim, Y. 2017. Assessment on shock pressure acquisition from underwater explosion using uncertainty of measurement. *Measurement*, 112, 1–10.
- Nowak, B., Kwasek, A., Szymczyk, A. & Cudny, M. 2023. Small-scale underwater explosion in shallow-water tank. *International Journal of Impact Engineering*, 180, 104487.
- Sheng, Z., Zhang, W., Chen, S. & Xu, Z. 2021. Exploration of array measurement technology for bubble jet load induced by underwater explosion. *Chinese Journal of Applied Mechanics*, 38(6), 1339–1347.
- Swisdak, M.M. 1978, Explosion effects and properties: Part II—Explosion effects in water. *NSWC/WOL TR 75-116*, Naval Surface Weapons Centre, Silver Spring, MD.
- Urtiew, P.A. & Hayes, B. 1991. Parametric study of the dynamic JWL-EOS for detonation products. *Combustion, Explosion and Shock Waves*, 27(4), pp.505–514.
- Zhuang, M., Zhao, X., Zhang, X., Li, J., & Ma, G. 2019. Underwater explosion of slender explosives: Directional effects of shock waves and structure responses. *International Journal of Impact Engineering*, 131, 103922.
- Zhuang, Y., Liu, J., Jiang, L. & Tang, Y. 2020. Experimental investigation on dynamic response and damage models of circular RC columns subjected to underwater explosions. *International Journal of Impact Engineering*, 141, 103574.

[RETURN TO CONTENTS PAGE](#)

Wireless blasting solutions in European mining operations

J.A. Kreivi

WebGen™ Orica, Finland

J.S.C. Marijnissen

WebGen™ EMEA, Orica, United Kingdom

ABSTRACT: Through Rock Wireless (TRW) blasting has opened new possibilities to approach blasting for mineral extraction. WebGen™, wireless blasting technology, communicates through rock, air, and water to initiate blasts safely and reliably. Using pre-charging techniques, it is possible to recover ore that otherwise would be sterilised in a stope or bench. Since wireless technology was introduced into the mining industry about 10 years ago many new mining methods have been developed, improving safety in hazardous areas and improving recovery and efficiency. The journey from traditional wired blasting systems to a wireless initiation system has been an eventful and exciting route. As wireless blasting becomes more and more accepted in the mining industry, it is exciting to present examples of wireless blasting in Europe. Several mining operations, both underground and surface, have worked together with Orica to introduce new mining solutions into their operations using the WebGen™ wireless blasting system. This paper will present examples for applications and benefits of TRW initiation in Europe. TRW initiation and new adapted mining methods have improved the ore recovery, efficiency and safety significantly by:

- Reducing exposure of people in hazardous areas in underground and surface mining operations.
- Improving ore recovery and stope reconciliation.
- Reducing ore loss and dilution using the T-drive elimination method combined with retreat mining.
- Providing flexibility in mine scheduling by turning loaded blasts into access roads.

1 WIRELESS ENHANCED MINING SOLUTIONS

Since the introduction of wireless initiation of blasts about 10 years ago many wireless enhanced mining methods have been developed. Each of the methods could be developed due to the physical removal of the wire of the detonator. With the physical limitations removed, blast engineers are not limited to the established ways of blast design. This is valid for both underground and surface applications. Starting often as an idea that seemed

to be impossible, drilling and blasting engineers worldwide worked together with Orica engineers to develop new approaches to extract ore more efficiently. Wireless, innovative, but always honouring sound blast engineering principles.

2 WEBGEN™ WIRELESS BLASTING SYSTEM

The WebGen™ wireless initiation system has two main components (Figure 1):

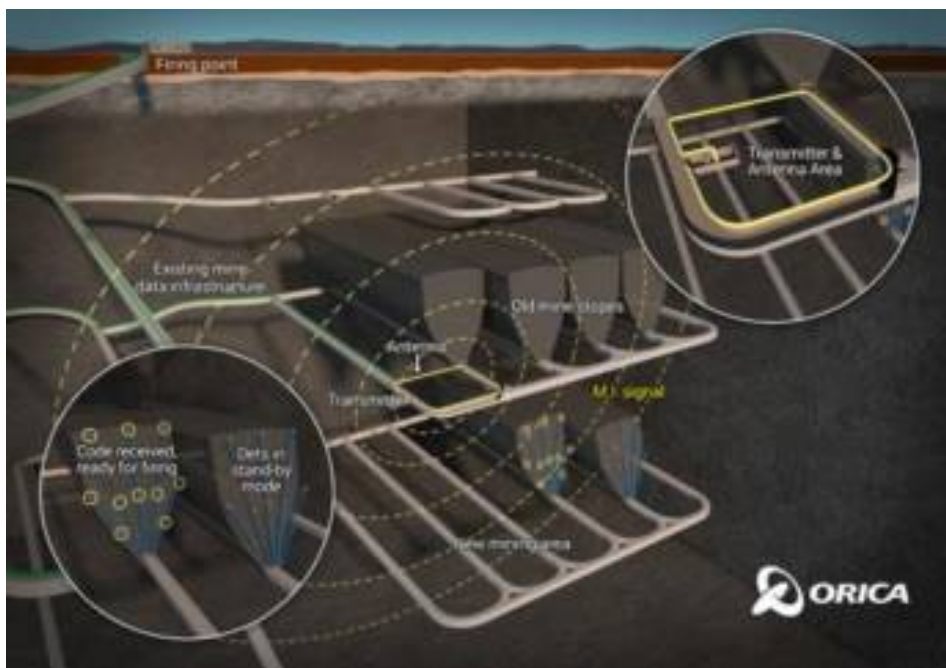


Figure 1. WebGen™ uses low frequency magnetic induction waves to communicate with primers.

- The wireless primer, which is consumed in the blast.
- The firing equipment that sends low frequency magnetic induction signals (1,800 Hz) to the wireless primers.

During the charging phase of the blast the wireless primers are encoded with arm and fire codes and a delay time. Primers can get a delay from 1ms to 30,000ms in increments of 1ms. Primers can sleep in the blasthole for up to 60 days, however the bulk explosive sleep time may be a greater constraint, especially in reactive ground.

Primers are assigned to a specific group of primers, known as the Blast Identification (BID) group. Primers in a specific BID will all get the same arm and fire code and respond to the same firing command. A blast can be divided into multiple BIDs for flexibility. BIDs can then be fired individually, or up to ten BIDs can be merged and fired together. This aspect makes wireless blasting highly flexible and the blasting sequence can be changed depending on the mining situation. In conventional wired electronic blasting systems, the number of primers is limited due to the resistance and capacitance of the wires. As there are no wires present with wireless primers this

physical constraint is removed and the number of primers in each BID and each blast is unlimited.

3 WIRELESS ENABLED DRILL & BLAST DESIGN PROCESS

Wireless blasting enables many new blasting and mining methods because:

- Blastholes can be pre-charged.
- After charging no further physical presence is required at the blasthole.
- Blasts can be fired remotely, at short notice, and in any order.

Pre-charging means loading and sleeping blastholes days or weeks in advance of the planned firing time and then continuing other activities on and around the sleeping blastholes.

The WebGen™ enhanced drill and design process starts with a concept design that is developed by the drill & blast engineers of the mine, Orica or jointly. This concept will be discussed with the mine team and Orica's Wireless Blasting Specialists.

In conventional mining the blasting direction is either longitudinal or transverse. The orebody and stope shape dictates if the blasting will be longitudinal or transverse. Using wireless

initiation, the BIDs can be designed in such way that you can use longitudinal and transverse blasting in the same stope and even in the same blast, something that is not possible using conventional initiation systems

The new design approach requires that the blast engineers think differently, and it is important to establish all ‘what ifs’ that can happen when a wireless design is applied. Therefore, Orica encourages customer to use a peer review system where the mine and the Wireless Blasting Specialist discuss all risks associated with the wireless blast. After the local team review, the drill and blast design it is separately reviewed by Orica’s WebGen™ Design Centre based in Canada. This gives a second, independent set of eyes on the design to ensure that all contingencies are captured.

During drilling of the blastholes a concept blast design will be developed assigning the explosives charge per blast hole, blast timing, optimal BID charging sequence and blast schedule. Upon completion of the drilling, the blast holes will be checked to establish the designed vs actual drilling and decide if any modifications are needed to the blast design. At this stage it can be decided on any actions to be taken to ensure the best blast results.

Orica uses magnetic induction (MI) signal

surveys to ensure reliable signal strength to wireless blasts. Signal surveys are done before and after drilling, and sometimes immediately before firing.

4 THROUGH THE ROCK WIRELESS (TRW) IN EUROPE

Orica’s Through Rock Wireless enhanced mining methods are being used in both underground mining and surface mining operations throughout the European mining industry. Great successes have been achieved in:

- Reducing exposure of mining crews in hazardous areas, e.g. stope brows and high walls.
- Improving ore recovery and reducing dilution.
- Reducing overbreak and underbreak in production blasting in underground and surface mining.
- Reducing lateral development in underground mining operations.
- Improving mining cycle time.
- Introducing better flexibility into mine schedules.

Following are three case studies of wireless blasting in European mining operations; at Outokumpu’s Kemi Mine in Finland, DPM

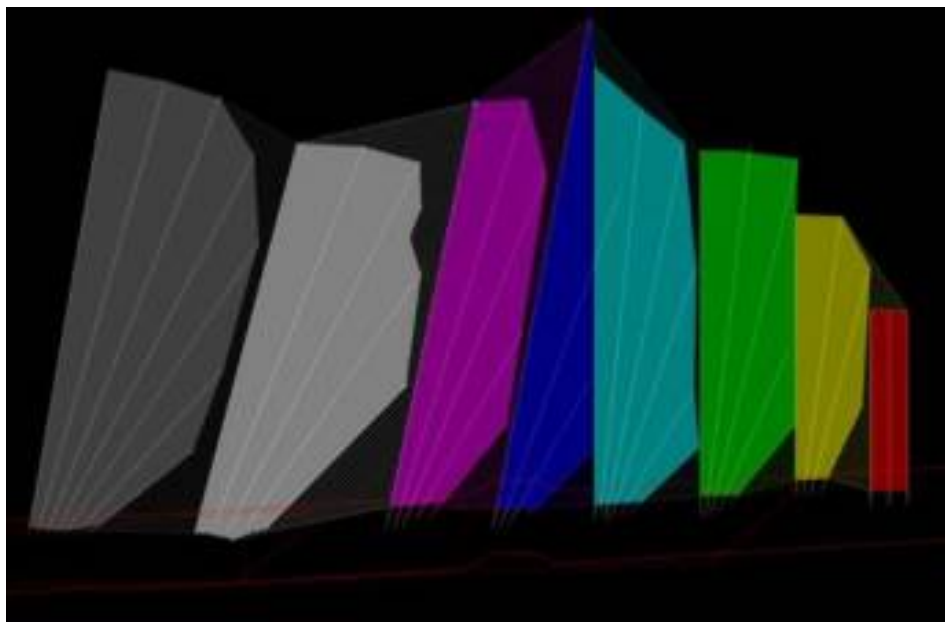
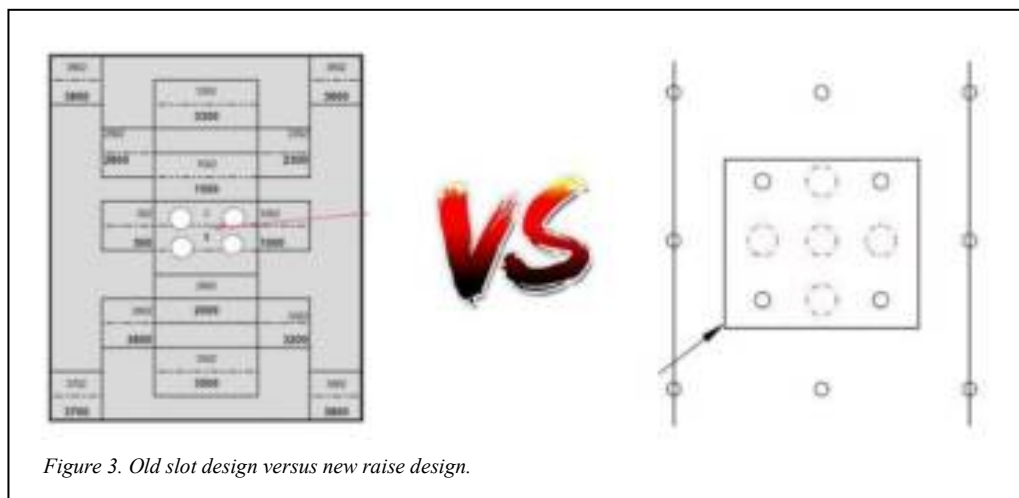


Figure 2. Fanned drilling and blasting in the crown pillar and ore sterilisation.



Chelopech Mine in Bulgaria, and Boliden's Kevitsa Mine in Finland.

4.1 Outokumpu Kemi mine, Finland

Improved ore recovery and stope reconciliation in the crown pillar at Outokumpu Kemi Mine, Finland.

When blasting with conventional initiation systems, Outokumpu Kemi mine experienced significant ore recovery challenges and high exposure of mining crew in hazardous underground areas. The mine reached out to Orica to jointly find a solution to improve the ore recovery, while reducing the exposure of the mining crew to the hazardous brow area of the crown pillar stopes.

Traditionally the mine would open the stope with a raise and slot blast. After opening the slot, the stope was divided in blast blocks using a fan design (Figure 2). Using this method the mine suffered significant ore loss in the top corners of each blast. The fan design left 'wedges' of

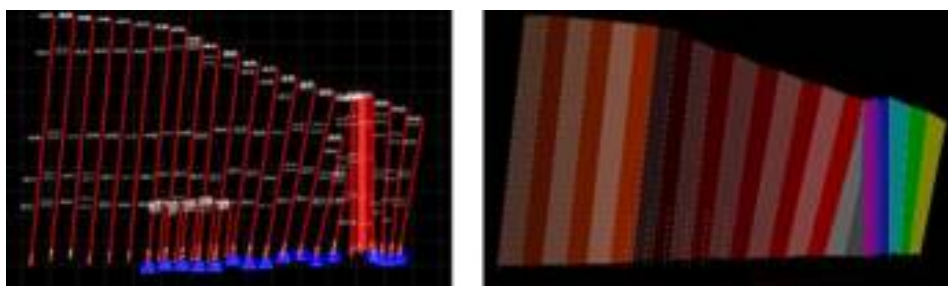
sterilised ore that were not recoverable.

The wireless initiation system made it possible to blast the crown pillar differently. The solution was implemented in two stages:

- Improving blasting results of the stope opening (slot).
- Improving the drill and blast design of the remainder of the stope by pre-charging.

The first stage of the solution focussed on improving the stope opening. The mine struggled to blast the raise and slot to the desired height. A redesign of the slot (Figure 3) proved highly efficient, achieving a slot height of 30 m.

The second stage focussed on improving the rest of the stope. The slot opening was positioned in a more suitable location, with pre-charged rings behind the slot to be blasted after the slot. (Figure 4). The remainder of the stope was re-designed using parallel rings of blastholes. The parallel blast ring design left no undrilled wedges of ore. The whole stope was pre-charged with wireless primers. This would have been impossible with a



conventional initiation system due to the risk of misfires due to wire cut-offs.

Pre-charging reduced the exposure of the mining crew to the hazardous area of an open raise or stope brow. The crew charged the whole stope in one campaign and did not have to return to charge and connect each blast. Blasting could now be done when planned or when necessary, without time consuming re-entries of the stope. This made the whole stope cycle safer and faster.

The results of the wireless enabled mining method exceeded expectations. The stope:

- exceeded budget ore recovery by 92%
- exceeded budgeted stope recovery by 80%
- Budgeted waste rock handling reduced by 40%
- significantly reduced exposure of the mining crew to hazardous areas.

4.2 DPM Chelopech mine in Bulgaria

Reducing lateral development and improving ore recovery and dilution by eliminating the slot T-drive and use of Reverse Throw Retreat (RTR) mining at DPM Chelopech Mine, Bulgaria.

The mine is a technology driven mine always looking for innovative solutions to reduce ore loss and dilution. It requested an evaluation of the WebGen™ wireless initiation system to demonstrate potential new mining methods and benefits. Several wireless enhanced mining methods were discussed and a wireless enhanced conceptual design was developed. One of the concepts to be demonstrated was T-drive elimination combined with Reverse Throw Retreat (RTR) mining.

Conventionally, the slot is placed at the end of the stope by developing a T-drive and a raise, into which the slot is opened in several blasts. When the slot is open the stope is mined by retreating towards the access point. With conventional wired systems, this sequence cannot be changed.

The wireless enhanced T-Drive elimination method has several advantages compared to the conventional sequence. The slot can be put anywhere in the stope, and eliminating the T-drive saves the cost and time of lateral development. By combining the Reverse Throw Retreat method, stope cycle time is reduced because less remote mucking is required, and lateral ore drive development is reduced by recovering ore behind the slot with pre-charged rings.

In the evaluation of wireless enhanced mining methods at Chelopech, two T-drive elimination stopes were demonstrated, one with down holes and one with up-holes. The principle of the T-drive elimination method is similar for down-holes and up-holes. The best location of the slot is determined based on the stope shape. The raise is blasted with a wired electronic initiation system. The rest of the slot is designed in multiple BIDs for flexibility, to ensure good blast results. Cavity survey results are used to decide which BIDs will be merged for the next blast.

The cavity surveys of the down-hole slot in Figure 5 show a funnel being created by the opening blasts. The funnel directs the broken material towards the loading point (drawpoint) in the ore drive. The loader only has to make lateral movements and does not have to clean-up around corners. This increases the loading rate.

Due to the stope shape the left corner of the stope was not blasted with the slot to protect the stope entrance at the top. This side was fired as the last BID to clean the corner and finalise the stope.

The rest of the stope was mined using the Reverse Throw Retreat (RTR) mining method. RTR is a wireless enhanced mining method where blind up-holes behind the slot are pre-charged in predetermined BIDs. The design creates a funnel and casts broken ore back towards the drawpoint. This increases the loading rate as the broken material is loaded from a single point, and less remote mucking is required. Lastly the wings in

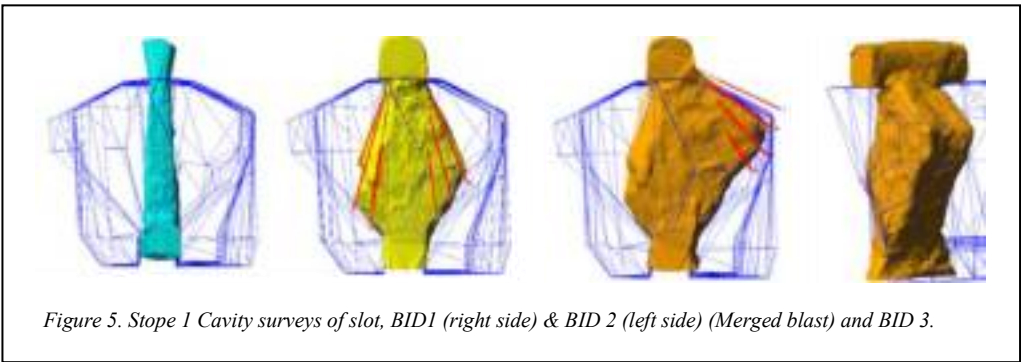


Figure 5. Stope 1 Cavity surveys of slot, BID1 (right side) & BID 2 (left side) (Merged blast) and BID 3.

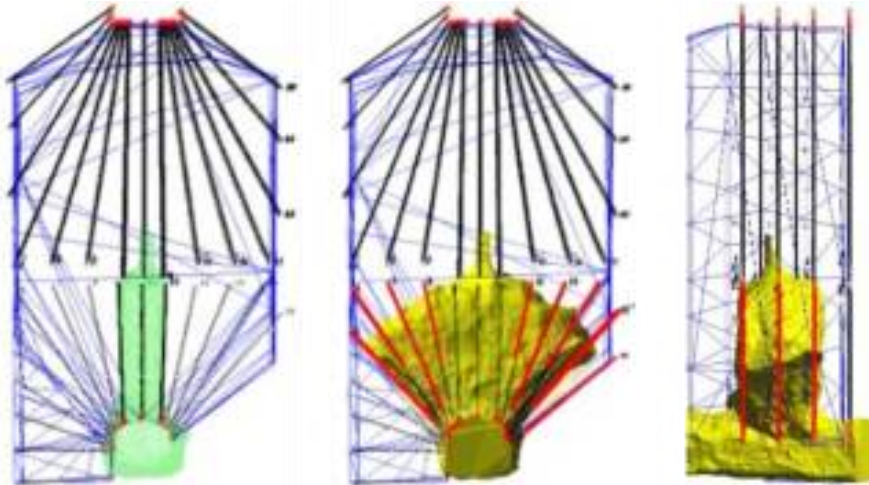


Figure 6. Stope 2 - CMS bottom slot (WebGen™) – BID1, BID 2, BID 3 & BID 4 merged.

the bottom corners of the stope are blasted. The wings are drilled with lifter holes that will be blasted upwards and sideways. The broken material will be blasted towards the middle of the stope and the stope can be cleaned out easily.

Stope 2 was designed and blasted using a hybrid method. The bottom of the slot was blasted using WebGen™ while the top of the slot was blasted using conventional electronic detonators. Figure 6 shows that an arch formed at the top of

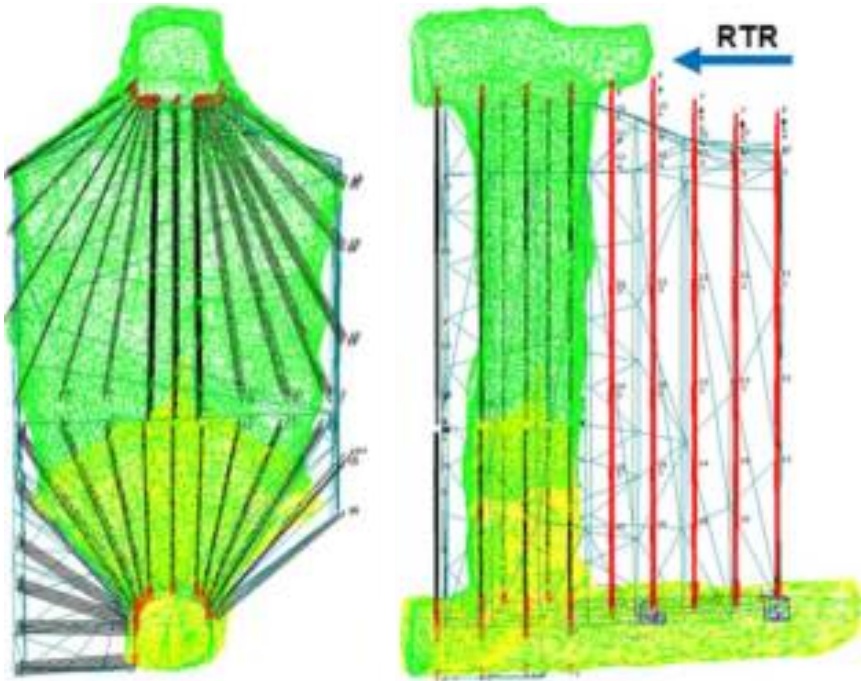


Figure 7. Stope 2 - slot fully open.

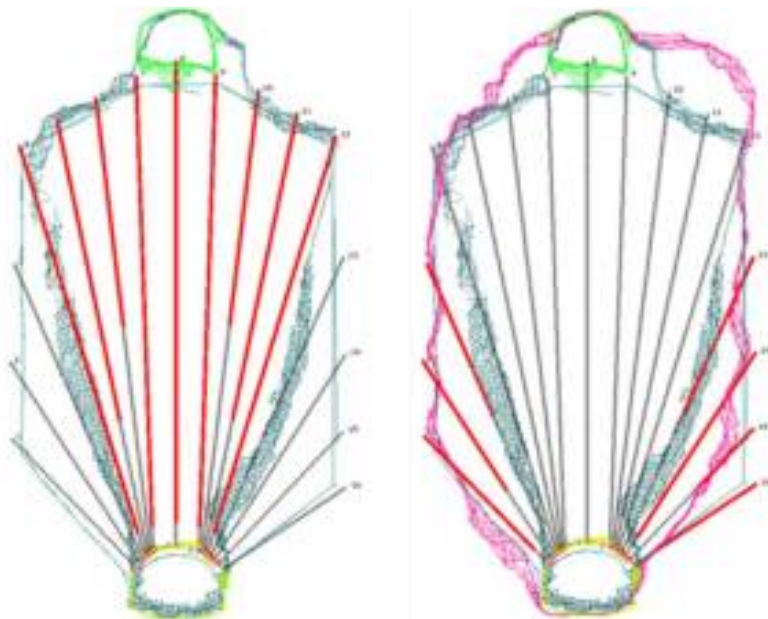


Figure 8. Cavity survey after BID 6 (left) & after BID 8 & BID 9 (merged right).

the uphole slot. This was expected and is not underbreak. As there was access at the top it was

decided to extend the top-down holes to compensate for the arch. In this way data could be

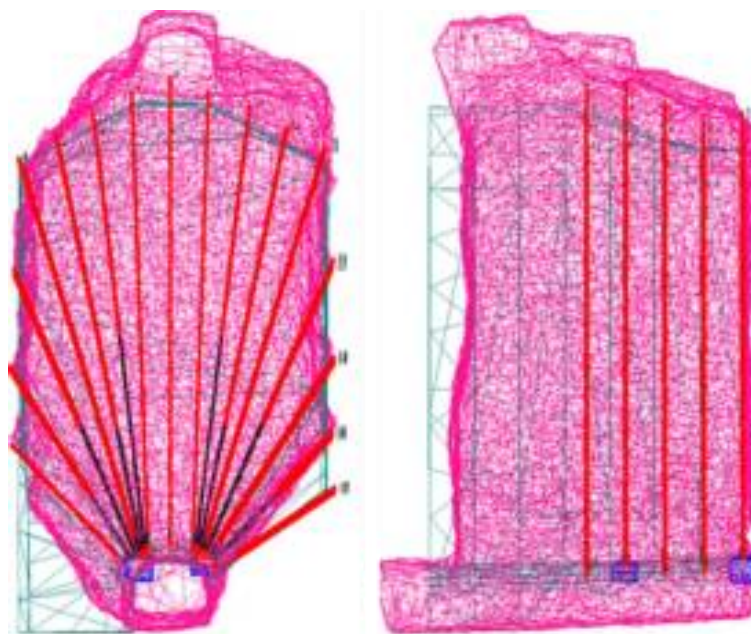


Figure 9. Stope 2 - Final cavity survey of the wireless enhanced blasted stope.

gathered about the arch for future designs.

After the slot was successfully opened, the rest of the stope was mined by RTR of the pre-charged rings (Figure 7). The cavity survey (Figure 8) clearly shows the funnel guiding broken material to the drawpoint in the ore drive for faster loading of blasted rock.

Figure 9 shows that the stope design was achieved. A last-minute design change meant the last ring of the stope was not charged and fired. This made it difficult to recover the last of the broken ore, and this material appears as underbreak in the bottom left corner. Some limited overbreak occurred in the top of the stope due to a fault.

The evaluation of wireless enhanced mining methods was a great success, achieving:

- Ore loss reduced by 5% compared to the mine average.
- Dilution reduced by 4.3% compared to the mine average.
- Loading rate improved by 30%.
- Lateral development reduced by 33%.
- Proof of concept of six wireless enhanced mining methods.

4.3 Boliden Kevitsa Mine in Finland

Reducing hazardous exposure of mining crew and Mining Scheduling Flexibility (MSF) at Boliden Kevitsa Mine, Finland.

This mine approached Orica and Forcit to work jointly to evaluate the potential benefits of wireless enhanced methods for their surface mining operation. Objectives for the evaluation were:

- Demonstrate how wireless blasting can mitigate risk and operational impacts from wall instability caused by melting ice on the high wall (Figure 10).
- Demonstrate how charged wireless blasts can be used as access roads for production equipment.
- Demonstrate how wireless blasting can control dust to comply with environmental restrictions.

A blast block next to the southern high wall of the pit was selected and divided into two sections:

- The first part was designed using conventional electronic detonators (EBS).
- The second part under the hazardous high wall was designed to use WebGen™



Figure 10. Ice hanging on the pit high wall creates a hazard.

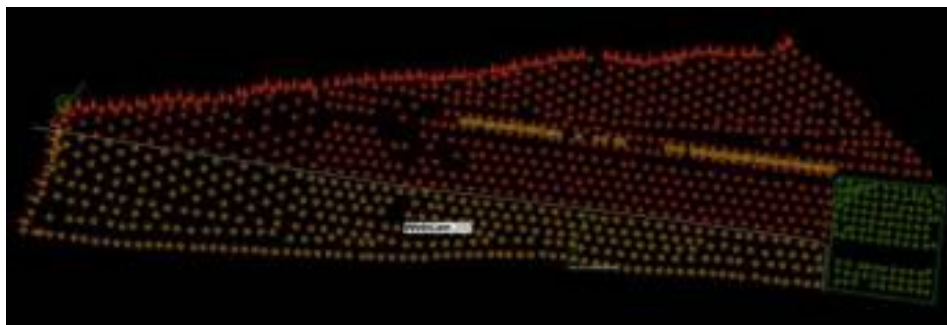


Figure 11. Blast block drilling design (red & green - conventional EBS, brown - WebGen™).

(Figure 11).

The conventional EBS section consisted of 843 blast holes and 1,479 primers. This area was charged before the section designed for wireless primers, which consisted of 353 blastholes and 635 primers. Due to warmer than expected weather a spotter was used to observe the high wall during charging of the WebGen™ primers. On average, 45 primers were encoded and primed per hour (23 blastholes). WebGen™ charging was completed during three day-shifts.

After charging and stemming, a wheel loader was used to grade a road over the charged wireless blastholes to give better access for the blast crew to connect the wired electronic blast. This road also provided access for a water truck to spray water across the bench as a dust prevention measure. By grading the bench, the wheel loader removed most of the drill cuttings and large rock material that could get airborne during blasting (Figure 12).

Driving the wheel loader over the charged blast demonstrated that this technique can be used throughout the mine. Charged blast blocks can be transformed into haul roads and bench access points, creating scheduling flexibility for all related parts of the mining cycle.

During the pre-blast check it was noticed that the icy high wall had been active overnight, resulting in large ice and rock fragments falling onto the charged bench. Were there wires, they would certainly have been cut. Repairing the wires would have caused significant delays and required mining crew to enter a very hazardous area, with risk of falling rocks and ice.

The conventional wired electronic blast was fired first, approximately 1 hour before WebGen™ blast. After the post blast check the WebGen™ blast was successfully blasted without any delays. It is not possible to fire blasts separately and

sequentially in this way using only wired systems, because flyrock from the first blast will damage the surface wires on the second blast.

Traditionally, when blasts taken next to each other the first two rows of the second blast are not drilled before firing the first blast. After mining the first blast, drilling equipment has to be sent back in to complete the drilling. The first row will need many offset angled drill holes to compensate for overbreak of the previous blast and to avoid toes at the bottom of the bench. Using wireless initiation, no additional angled holes will be needed to drill as the blastholes will be pre-charged. The clear and sharp division between the conventional blasted bench and the pre-charged wireless bench proved that using this hybrid mining method the time consuming off-set blastholes can be avoided, improving the overall cycle time of the bench.

In summary the evaluation of wireless enhanced blasting at the mine was highly successful achieving:

- No misfires.
- Reduced time working on foot under the hazardous high wall.
- Reduced charging time per blast hole.
- Faster bench cycle time.
- No need for spotters during stemming loading.
- Improved access for the water truck to spray the bench for dust control.
- Improved and faster access to adjacent conventional wired blast patterns.

There is still greater potential for operational efficiency by exploiting the mine schedule flexibility (MSF) techniques that wireless primers enable. Ultimately, haul routes can be established over loaded blasts to maintain fleet productivity in parts of the mine constrained by available bench area.

5 WIRELESS ENHANCED MINING - THE FUTURE

There are many wireless enhanced mining methods that improve safety and mining production underground and on the surface. The examples presented here are just some of them. Wireless blasting enables mining engineers to approach mine planning differently by removing many of the constraints imposed by conventional wired initiation systems. An idea that seemed impossible before is suddenly possible. All you need is imagination and make sure you follow sound engineering principles. Wireless blasting is a revolutionary game changer in the blasting industry. The wireless blasting future is bright.

6 ACKNOWLEDGEMENTS

Our thanks go to the drilling and blasting engineering teams at Boliden Kevitsa Mine, Outokumpu Kemi Mine and DPM Chelopech Mine. Also to Sergei Mihalev, Technical Services Superintendent, Chelopech Mine, Bulgaria and Stefan Popov, Drilling & Blasting Engineer, Chelopech Mine, Bulgaria.

[RETURN TO CONTENTS PAGE](#)

Development of gasless lead-free time-delay compositions for electric and non-electric detonators

M. Gerlich, W. Trzciński & M. Hara

NITROERG S.A. Bieruń, Poland

M. Gerlich

Military University of Technology, Warsaw, Poland

ABSTRACT: The widespread use of explosives drives the continuous improvement of blasting agents, including time-delay detonators. One of the key challenges in this field is enhancing the precision and reliability of detonators while minimising their environmental impact. A crucial step toward this goal is the development of safer, lead-free, and gasless time-delay compositions that provide consistent performance regardless of external conditions. This study focuses on the methods of designing modern delay compositions, including W/CuO, Fe/BaO₂, and silicon-based systems (Si/Bi₂O₃, Si/Sb₂O₃). Unlike traditional pyrotechnic compositions, these mixtures react primarily in the solid phase, reducing sensitivity to pressure fluctuations inside the detonator. Such a combustion mechanism contributes to greater accuracy and reliability in blasting operations, directly impacting both safety and the effectiveness of blasting work. To evaluate their potential for industrial implementation, these compositions were analysed in terms of their combustion rate, thermal stability, and reaction products. Advanced instrumental techniques, including Differential Thermal Analysis (DTA) and X-ray Diffraction (XRD), confirmed their gasless nature, while Scanning Electron Microscopy (SEM) and Energy Dispersive Spectroscopy (EDS) provided insight into the morphology and elemental composition of combustion residues. A particularly important aspect of this research is the drive toward environmentally friendly alternatives to traditional lead-based compositions. Eliminating hazardous substances such as lead oxides from detonator formulations aligns with global efforts to reduce the ecological footprint of explosive materials while maintaining or even improving operational efficiency. This presentation aims to describe the methods used in the design of delay compositions and highlight the critical role of detonators in ensuring both safety and the effectiveness of blasting operations. The findings contribute to the advancement of safer and more sustainable blasting solutions, offering an alternative to conventional time-delay compositions that enhances both performance and environmental responsibility.

1 INTRODUCTION

Time-delay detonators are essential components in blasting operations, where precise timing between successive explosions improves rock fragmentation, reduces ground vibrations, and enhances overall safety (Monjezi *et al.* 2014, Xu *et*

al. 2021, Xu *et al.* 2019). The effectiveness of these systems depends largely on the stability and accuracy of the pyrotechnic delay compositions they contain.

Traditional delay compositions often rely on toxic substances - particularly lead-based oxidants, which pose environmental and occupational safety

risks. Additionally, these mixtures tend to generate significant amounts of gas during combustion, making their performance more sensitive to pressure fluctuations and less consistent under varying conditions (Focke *et al.* 2019). To overcome these limitations, recent research (including the authors' own studies) has focused on the development of gasless and lead-free compositions, aiming for cleaner and more reliable alternatives.

Gasless delay compositions are defined primarily by their reaction mechanism: the oxidiser reacts directly with the fuel in the condensed phase, without first decomposing to release gaseous oxygen. This solid–solid or solid–liquid interaction ensures low sensitivity to pressure changes inside the detonator and contributes to stable, predictable burn rates. Although some authors describe gasless systems as generating less than 10 cm³ of gas per gram of pyrotechnic composition (Charsley *et al.* 1980), this volumetric criterion does not fully capture the underlying condensed-phase nature of the reactions.

This review summarises and expands a series of studies conducted by the authors on various time-delay compositions, such as Fe/BaO₂ (Gerlich *et al.* 2020), Si/Bi₂O₃ (Gerlich *et al.* 2025), Si/Sb₂O₃, Si/Pb₃O₄, W/CuO (Gerlich *et al.* 2023) and Sb/KMnO₄ (Gerlich *et al.* 2022). Special attention is given to the experimental techniques used to investigate these systems and how they contribute to a deeper understanding of combustion mechanisms, thermal behaviour, and the factors affecting delay precision.

The compositions were evaluated using differential thermal analysis (DTA), thermogravimetry (TG), X-ray diffraction (XRD)

of combustion residues, bomb calorimetry, and thermochemical modelling of adiabatic combustion temperatures. These methods allowed for a detailed analysis of ignition thresholds, reaction pathways, thermal stability, and energy output, forming a comprehensive framework for assessing and optimising delay compositions.

2 MATERIALS AND METHODS

2.1 Materials

The pyrotechnic delay compositions investigated in this study were based on selected combinations of fuels and oxidisers designed to promote gasless or semi-gasless combustion. The following systems were examined:

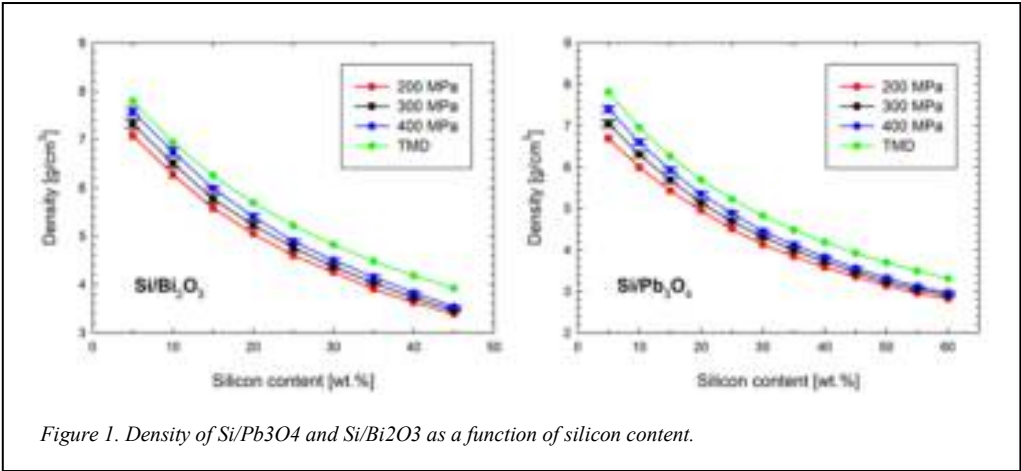
- W/CuO (tungsten copper(II) oxide).
- Fe/BaO₂ (iron/barium peroxide).
- Si/Bi₂O₃ (silicon/bismuth(III) oxide).
- Si/Sb₂O₃ (silicon/antimony(III) oxide).
- Sb/KMnO₄ (antimony/potassium manganate(VII)).

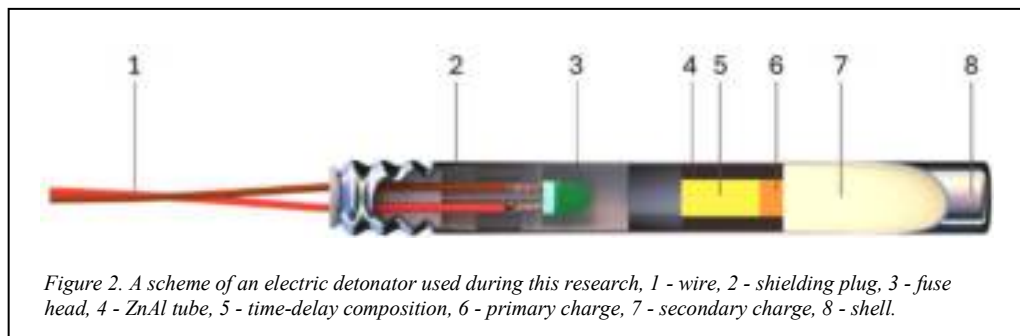
All powders used were of analytical grade with purities ≥99%. The particle size distributions were determined using a laser diffraction analyser (FRITSCH ANALYSETTE 22 MicroTec plus).

2.2 Sample preparation

The powdered components were mixed using a TURBULA® 3D shaker mixer for 6 hours to ensure homogeneous distribution. In the case of the Fe/BaO₂ system, BaO₂ was pre-dried at 60 °C for 24 hours to eliminate moisture. For Sb/KMnO₄, all components were granulated to the 0.2 - 0.8 mm range prior to loading.

Pyrotechnic columns were formed by pressing





the mixtures into metal (ZnAl) tubes: Pressures of 200, 300, and 400 MPa were applied using a hydraulic press equipped with a strain gauge. Final pyrotechnic columns were typically 20 - 30 mm in height, depending on the composition and tube. To ensure loading accuracy, each ZnAl tube was weighed before and after filling with the time-delayed mixture. Knowing the weight of the pressed mixture and the dimensions of the ZnAl tube (and thus the volume of the prepared mixture), the density of the mixture as a function of silicon content and pressing pressure was determined. The density dependence of the tested mixtures on silicon content is shown in Figure 1.

2.3 Burn rate measurement

Burn rate was evaluated by incorporating the compositions into electric detonators (as shown in Figure 2), equipped with a fuse head and PETN as a secondary charge. Delay time was measured as the interval between electrical ignition and shock wave detection using a microphone ohmmeter setup. Results were corrected for signal propagation time and igniter delay.

Each test was repeated at least five times per composition and compaction pressure. Burn rates were calculated from known column lengths and measured combustion times.

2.4 Thermal analysis

Thermal stability and ignition behaviour were investigated using:

- Differential Thermal Analysis (DTA).
- Thermogravimetric Analysis (TGA).

Analysis was conducted in an argon atmosphere with a heating rate of 10 °C/min up to 1100 °C. Samples (~3 mg) were tested in Al₂O₃ crucibles using a Setaram Labsys system. Onset temperatures, exothermic peaks, and mass changes were recorded to characterise ignition and reaction steps.

2.5 Combustion product analysis (XRD, SEM, EDS)

Solid combustion residues were collected post-reaction and analysed using:

- X-ray Diffraction (XRD) with a Rigaku SmartLab 3 kW diffractometer, Cu-Kα source, θ-2θ geometry, 2θ range 10° - 90°, step 0.02°.
- Scanning Electron Microscopy (SEM) and Energy Dispersive Spectroscopy (EDS) using a CrossBeam 540 system (Carl Zeiss) for surface morphology and elemental composition.

This allowed the identification of crystalline phases and confirmation of solid-state or gasless reaction pathways.

2.6 Heat of combustion

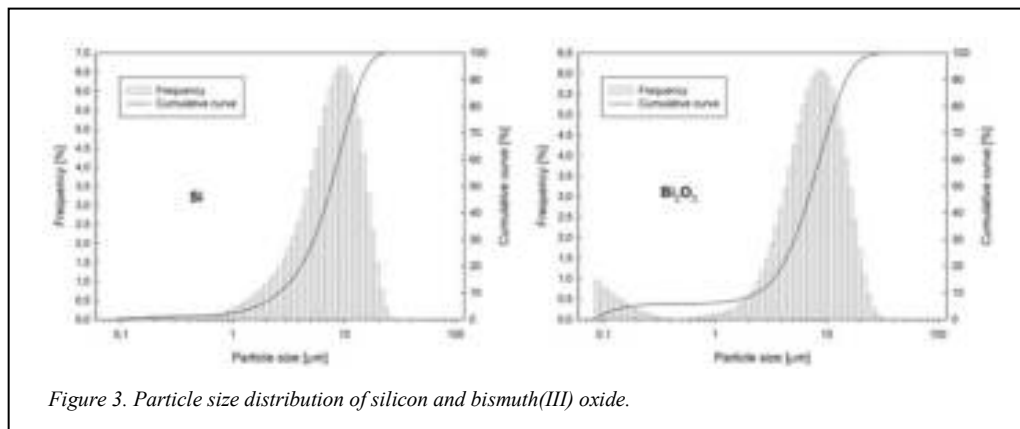
The gross heat of combustion was determined using a KL-12 oxygen bomb calorimeter. Approximately 15 g of each unpressed sample was placed in a 4 mL quartz crucible inside a steel bomb filled with argon at 2 MPa. Ignition was initiated using a Kanthal resistance wire. Water bath temperature changes were recorded to calculate combustion enthalpy (J/g).

2.7 Adiabatic temperature calculation

Using experimentally determined heats of combustion and assumed reaction stoichiometry, adiabatic combustion temperatures (T_a) were calculated by solving the energy balance:

$$Q_{exp} = \sum_{i=1}^n x_i \left\{ \int_{T_{ref}}^{T_a} [C_{p,i}(T)]_i dT + (Q_m)_i + \int_{T_{ref}}^{T_a} [C_{p,i}(T)]_i dT \right\} \quad (1)$$

Specific heat data and enthalpies of formation were obtained from the JANAF thermochemical



tables (Chase 1998). Polynomial approximations for $C_p(T)$ were used for accurate integration across temperature ranges.

3 RESULTS AND DISCUSSION

One of the most important parameters of time-delay composition components is their grain characteristics. It is indicated that the decrease in the average particle diameter increases the combustion rate due to the increase in the contact surface of the components and the elimination of the diffusion hindrance (Kalombo *et al.* 2007). Example of volume-based particle size distributions of used components are shown in Figure 3.

The analysis of above characteristics indicate that the components used in this study has narrow grain characteristics, which is crucial in terms of accuracy of the detonators.

Table 1. Grain size diameters and density (Chase 1998) of used components.

Powder	TMD [g/cm ³]	D[4,3] [μm]	D99 [μm]
Silicon	2.33	7.0	16.0
Tungsten	19.25	10.5	25.6
Iron	7.87	7.9	13.2
Antimony	6.68	9.3	34.0
Antimony (III) oxide	5.20	23.8	49.3
Bismuth (III) oxide	8.90	8.3	15.3
Lead (II, IV) oxide	8.30	8.2	14.1
Copper (II) oxide	6.31	24.3	68.8
Barium peroxide	4.96	8.4	13.7
Potassium manganate (VII)	2.70	5.3	17.0

De Brouckere mean diameters (D[4,3]) along with the diameter of a particle size distribution curve where 99% of the particles are smaller than that size (D99) and TMD (theoretical maximum density) is shown in Table 1.

From the applicational point of view, the most crucial parameter in developing new time-delay elements is the burn rate of the pyrotechnic mixture. That parameter depends mostly on two factors: the quantitative composition of the mixture and the elaboration pressure. The need to understand the characteristics of combustion seems obvious. Only in this way is the technologist able to determine optimal parameters for both mixture preparation and elaboration.

The study of the effect of compaction pressure and fuel content on the combustion rate not only allows us to define desirable parameters, but also reveals much about the pros and cons of a given composition, as well as the reaction mechanism itself. A good example of that is the Si/Pb₃O₄ composition, whose dependence of burn rate on silicon content under varying loading pressures is presented in Figure 4.

It can be seen that an initial increase in compaction pressure (and consequently, mixture density) leads to a decrease in combustion velocity, a trend typically observed in gas-generating mixtures. However, for compositions with a silicon content exceeding 35%, this tendency reverses, with higher combustion velocities observed for mixtures of greater density. This shift suggests a transition in the combustion mechanism from a gas-generating to a condensed-phase process. The most likely explanation is a reduction in combustion temperature as the composition moves further from the stoichiometric ratio. As a result, the system lacks sufficient energy to decompose the

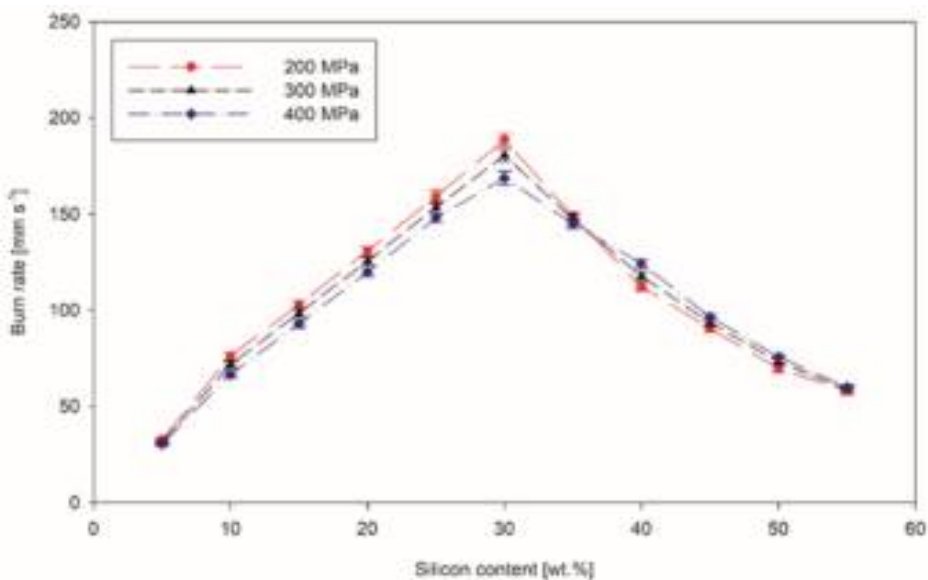


Figure 4. Influence of silicon content and loading pressure on the burn rate of Si/Pb₃O₄ mixture.

oxidiser and release gaseous oxygen, leading to a fundamental change in the reaction dynamics. There is additional crucial information that can be drawn from Figure 4. It is clear that industry should take into account the possibility of slight

differences in elaboration pressure (and, consequently, the density of the composition, which depends not only on the pressing force). That is why it is advised to use compositions which show the smallest dependence on

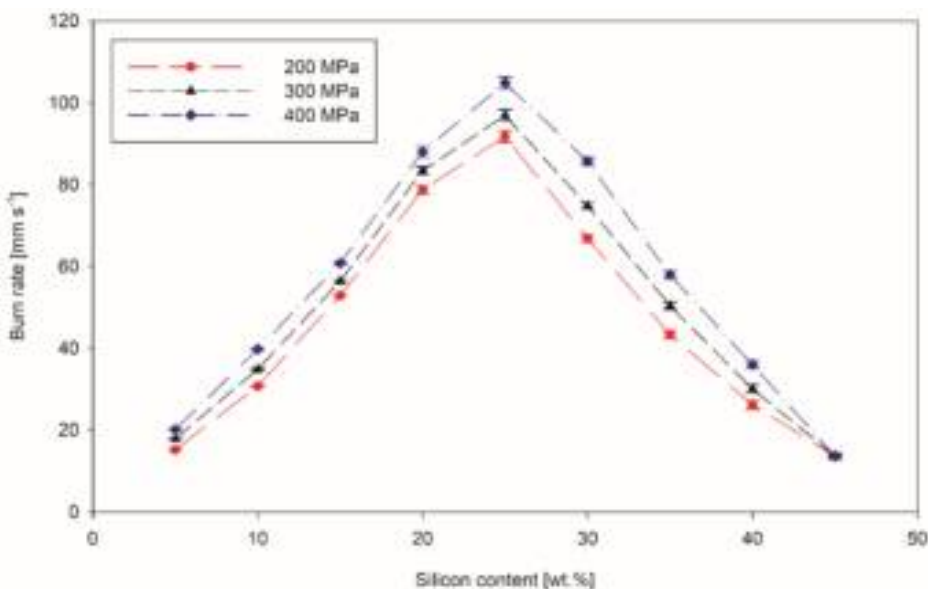


Figure 5. Influence of silicon content and loading pressure on the burn rate of Si/Sb₂O₃ mixture.

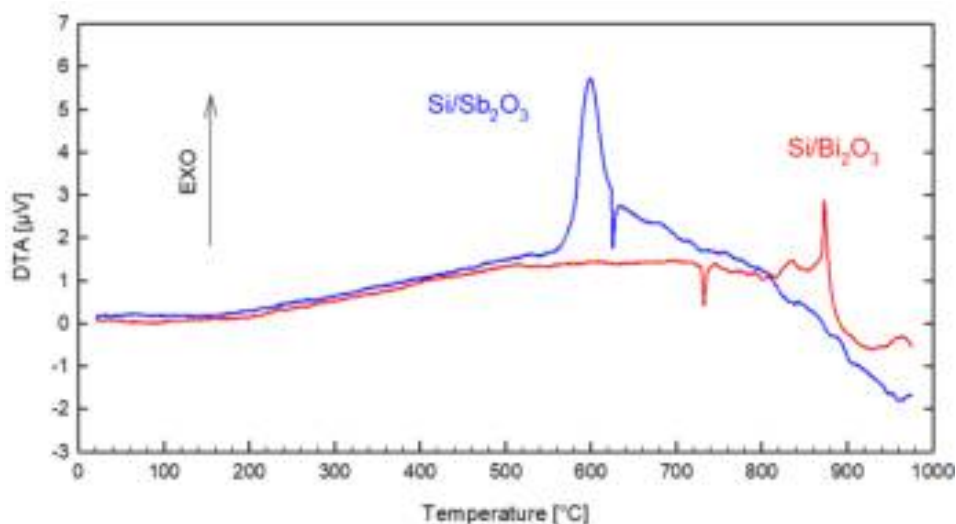


Figure 6. Thermograms for Si/Sb₂O₃ and Si/Bi₂O₃ mixtures with 20 wt.% silicon (temperature rate 10 K/min, argon 50 ml/min).

elaboration pressure (density). For Si/Pb₃O₄, these are compositions containing 5% and 55% silicon. The above is an example of how significant information can be drawn from these simple tests. A good example of a gasless composition is Si/Bi₂O₃. The effect of compaction pressure and fuel content on the combustion rate of this composition is shown in Figure 5.

The analysis of the influence of the loading pressure on the dependence of burn rate on silicon content indicates that the burn rate increases with elaboration pressure. This means that, across the entire range of fuel content, the reaction between silicon and bismuth(III) oxide is a solid-state reaction. Additionally, this composition exhibits a high burn rate, achieving around 100 mm/s for a composition containing 25% Si.

As shown, the dependence of burn rate on elaboration pressure offers important insight into the reaction mechanism, which affects the accuracy of the delay time. However, there is a more powerful tool to determine the thermochemistry of the combustion - thermal analysis. It allows us to check whether there is a thermal decomposition of the oxidant prior to the ignition of a specific time-delay composition. Figure 6 shows thermograms of Si/Bi₂O₃ and Si/Sb₂O₃.

Figure 6 presents DTA curves for Si/Sb₂O₃ and Si/Bi₂O₃ mixtures, each containing 20 wt.% of silicon. The DTA curve for the Si/Sb₂O₃

system reveals an exothermic effect starting around 560 °C, reaching a peak at 600 °C. According to reference (Rugunanan 1993), for a mixture with 40 wt.% silicon, a similar effect appears in the 500 - 540 °C range. Literature data indicate that Sb₂O₃ undergoes a solid-to-liquid phase transition at 928 °C and vaporises at 1729 °C, forming Sb₄O₆ dimers (Barin 1995). Therefore, the observed effect on the thermogram is not due to thermal decomposition of the oxidiser nor any phase change. Likewise, silicon does not experience any polymorphic or phase transformations under atmospheric pressure up to 1685 °C. This suggests that the exothermic signal results from a reaction between the components occurring at the solid-solid interface, confirming the gasless nature of the reaction in the Si/Sb₂O₃ system.

For the Si/Bi₂O₃ mixture, two significant thermal events can be observed on the DTA curve. The first, an endothermic effect, begins at approximately 730 °C and is attributed to the polymorphic transformation of α-Bi₂O₃ into β-Bi₂O₃ (Qiu *et al.* 2011). The second, an exothermic effect, occurs at 820 °C and is considered to result from a reaction between the two components. Since there is no associated phase transition or thermal decomposition, this reaction between silicon and bismuth oxide is also assumed to proceed in the solid state.

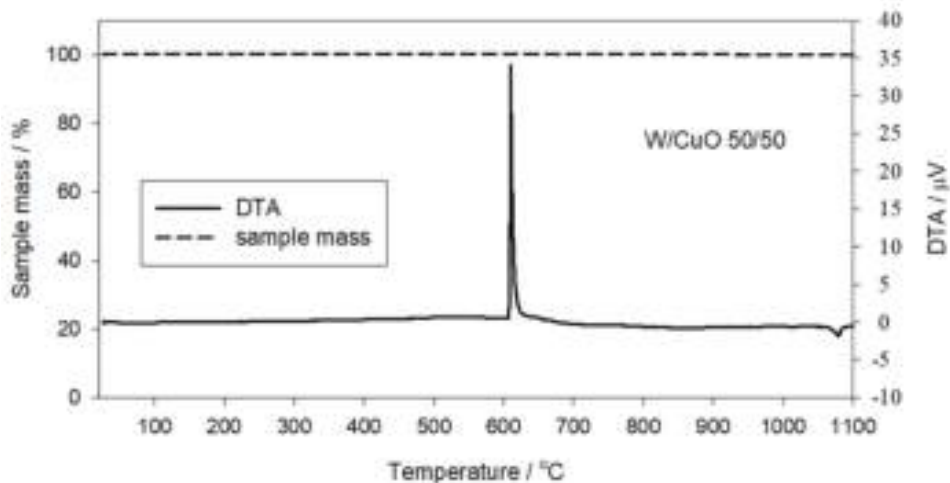


Figure 7. Thermogram of W/CuO with 50 wt.% tungsten (temperature rate 10 K/min, argon atmosphere).

A similar thermogram is obtained for the W/CuO composition (Figure 7). Two effects are visible on the DTA curve: the first large exothermic effect starts at about 606 °C (combustion of the mixture), and the second, much weaker effect, starts at about 1065 °C (thermal decomposition of trace amounts of unreacted CuO). The analysis of the TG curve indicates the lack of gaseous combustion products that would cause a decrease in the mass of the sample after the initiation of combustion at about

606 °C.

Similar characteristics can be seen for the Fe/BaO₂ (50% Fe) composition (Figure 8), where the onset of the main exothermic effect occurs at about 480 °C.

To determine whether the reaction occurs in the condensed phase or at the gas/solid or gas/liquid interface, it is crucial to compare this thermogram with that of the oxidant (BaO₂) (Figure 9).

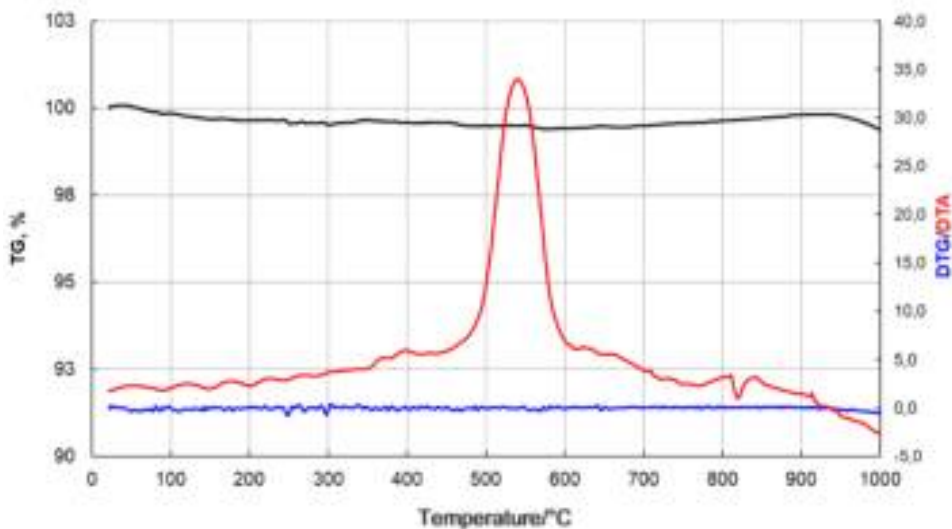


Figure 8. Thermogram of Fe/BaO₂ with 50 wt.% iron (temperature rate 10 K/min, argon atmosphere).

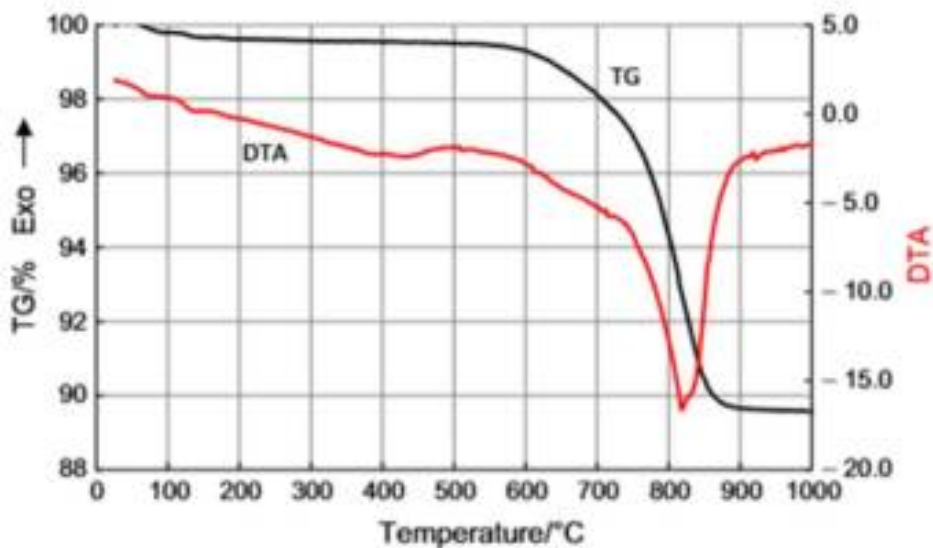


Figure 9. Thermogram of BaO2 (temperature rate 10 K/min, argon atmosphere).

Comparison of Figures 8 and 9 indicates that iron and barium peroxide react prior to the beginning of the thermal decomposition of BaO2. However, it is not desirable for the oxidant to have such a low decomposition temperature, as it increases diffusion-controlled mechanisms.

Thermal analysis is the most significant method in the first step of developing new time-delay compositions, as it offers deep insight into the mechanism of combustion.

Another parameter that affects the use of a specific composition is the heat of combustion.

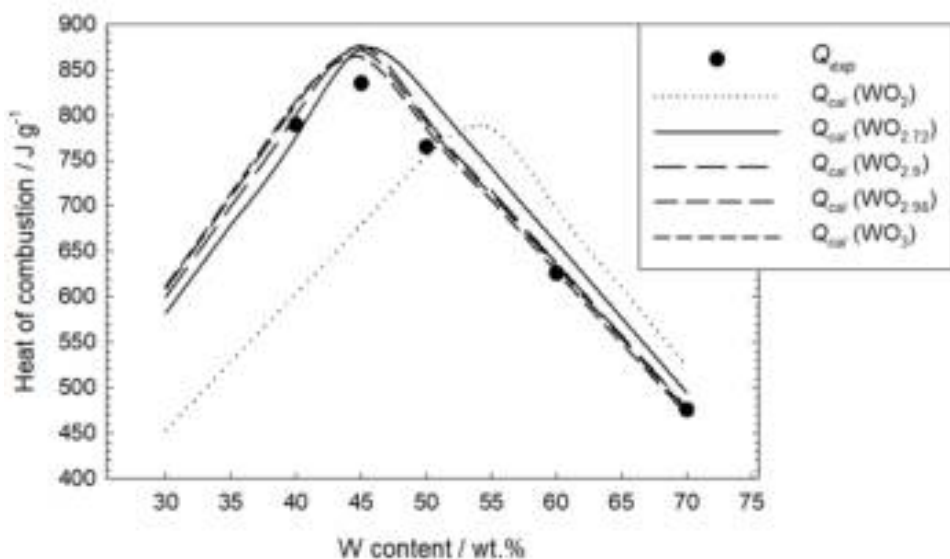


Figure 10. Comparison of the experimental heat of combustion of the tested mixtures with the heats estimated theoretically for the assumed tungsten oxides in the combustion products.

Table 2. The level of the tungsten oxidation has an impact on the stoichiometric amount of fuel, which for the below reaction is 53.6%; 45.9%; 44.3%; 43.8% and 43.5% respectively.

$W + 2\text{ CuO} \rightarrow \text{WO}_2 + 2\text{ Cu}$	$\Delta H_r = -809.4\text{ J/g}$	(3)
$W + 2.72\text{ CuO} \rightarrow \text{WO}_{2.72} + 2.72\text{ Cu}$	$\Delta H_r = -891.2\text{ J/g}$	(4)
$W + 2.9\text{ CuO} \rightarrow \text{WO}_{2.9} + 2.9\text{ Cu}$	$\Delta H_r = -886.5\text{ J/g}$	(5)
$W + 2.96\text{ CuO} \rightarrow \text{WO}_{2.96} + 2.96\text{ Cu}$	$\Delta H_r = -889.6\text{ J/g}$	(6)
$W + 3\text{ CuO} \rightarrow \text{WO}_3 + 3\text{ Cu}$	$\Delta H_r = -887.0\text{ J/g}$	(7)

Depending on the material of the casing, various compositions will exhibit different critical diameters, which also depend on the quantity of energy released during combustion. Figure 10 presents the heat of combustion of the W/CuO composition (as a function of tungsten content).

This method also allows for preliminary

prediction of combustion products. As shown above, even for a complicated system such as this, the heat of combustion corresponds well with the theoretically calculated heat of combustion for the assumed products, in accordance with reactions 3 - 7. The enthalpy of the reaction was calculated using Hess's law (2).

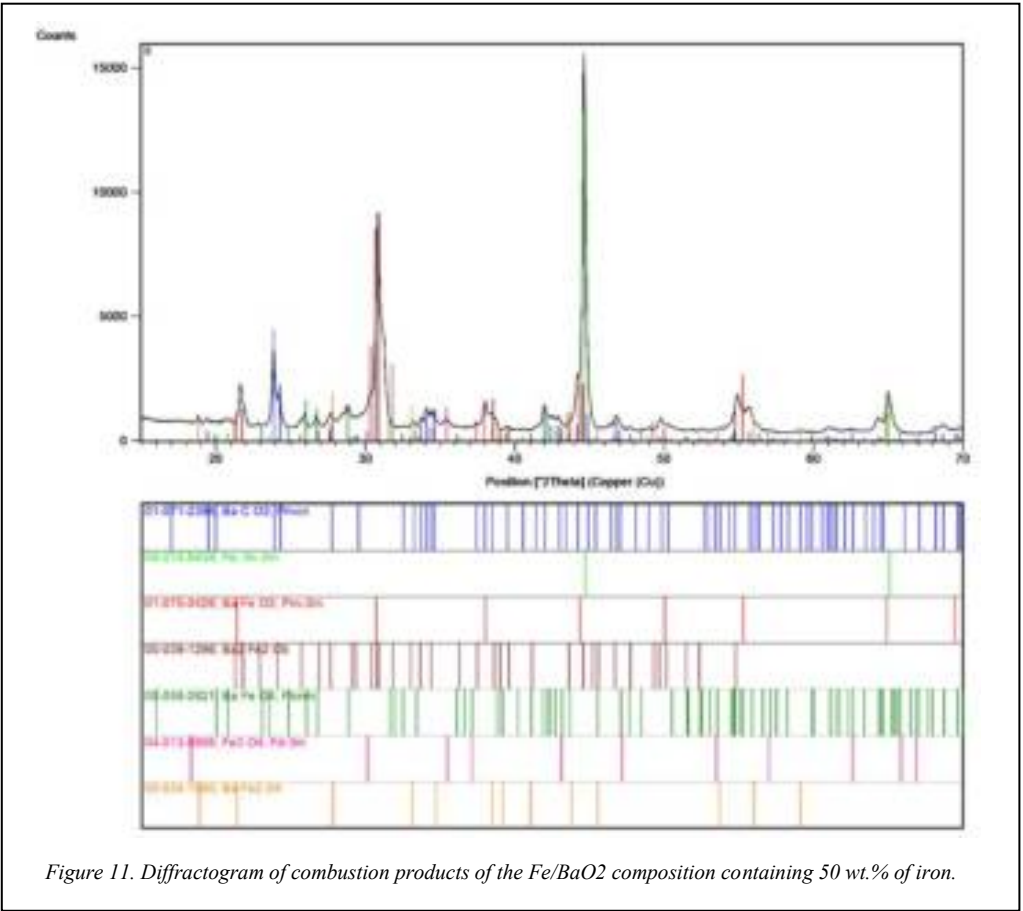


Figure 11. Diffractogram of combustion products of the Fe/BaO2 composition containing 50 wt.% of iron.

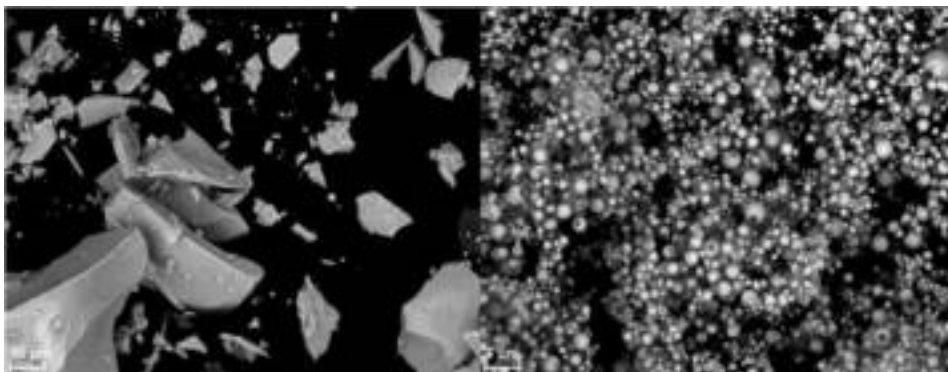


Figure 12. (Left) SEM images of Si/Pb₃O₄ (5% Si) combustion products.

Figure 13. (Right) SEM images of Si/Pb₃O₄ (10% Si) combustion products.

$$\Delta H_r = \sum_{i=1}^n n_i (\Delta H_f^0)_{T_r} - \sum_{j=1}^m n_j (\Delta H_f^0)_{T_r} \quad (2)$$

Assumptions made on the basis of heat of combustion can be verified by a method complementary to thermal analysis, i.e. X-ray diffraction. In the case of crystalline products, it is possible to distinguish XRD patterns of the

products, as shown in Figure 11.

Results from XRD analysis, along with SEM of the combustion products, give us robust feedback on the qualitative composition and morphology of the products. Depending on the oxygen balance, both the qualitative composition of the product and its morphology can differ. SEM images of the

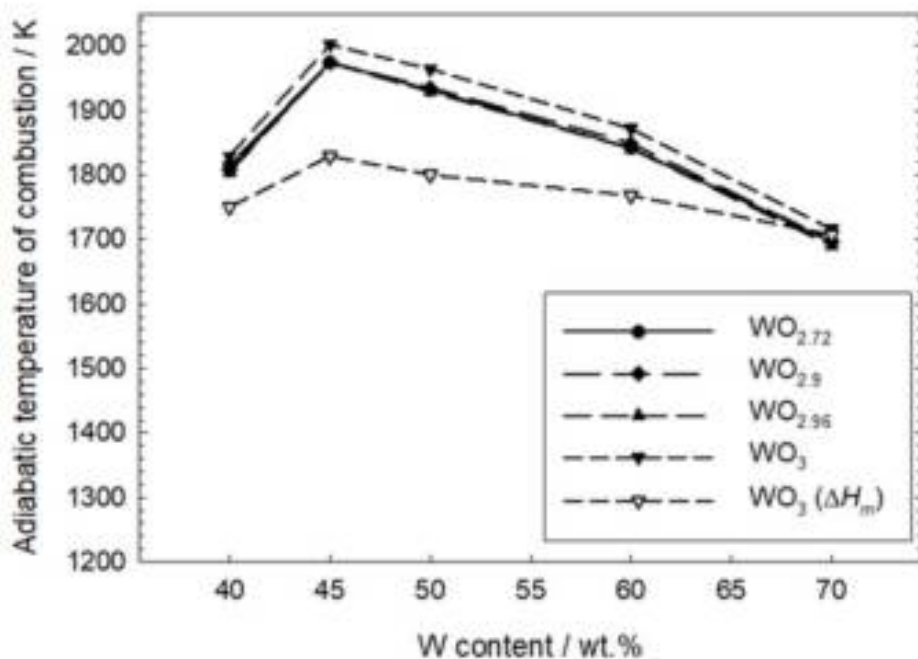


Figure 14. Adiabatic combustion temperatures estimated from the experimental heat of combustion of the tested W/CuO mixtures.

combustion products of Si/Pb3O4 are presented in Figures 12 and 13.

As seen above, for some compositions even slight change in the fuel content can affect the structure of the combustion products. It is especially important in case of gaseous compositions (as for Si/Pb3O4), because the compactness of the combustion products affects the permeability of gaseous products (including evaporated metals) which affect pressure change and in the end burn rate.

All of the above data make it possible to determine the adiabatic combustion temperature, taking into account the heat of combustion, the composition of the combustion products, and their specific heat capacities. The value of the adiabatic combustion temperature of the W/CuO mixture as a function of tungsten content is presented in Figure 14.

Among the four tested mixtures, the mixture containing 45 wt.% of tungsten has the highest combustion temperature. This temperature is 2000 K assuming solid WO₃ in the combustion products. The temperatures of mixtures with WO_{2.72}, WO_{2.9} or WO_{2.96} combustion products are close to each other. Taking the WO₃ melting into account definitely reduces the calculated combustion temperatures. In the actual process, the combustion temperatures should be within the estimated range.

4 CONCLUSIONS

The presented study compiles and expands upon the authors' original experimental investigations into the combustion behaviour of selected gasless and low-gas pyrotechnic delay compositions. A particular focus has been placed on understanding how key formulation and processing parameters (such as grain size, fuel content, and compaction pressure) influence burn rate and reaction mechanism, which in turn directly affect the accuracy and stability of time-delay detonators.

Systematic studies of compaction pressure and fuel content have shown that changes in these parameters can alter not only the rate but also the mechanism of combustion. For example, in the Si/Pb₃O₄ system, a shift from gas-generating to condensed-phase combustion is observed at higher silicon contents and compaction densities. These findings highlight the sensitivity of certain compositions to processing conditions, suggesting that formulations with minimal pressure dependence are preferable for practical applications.

Through the use of thermal analysis (DTA/TG), insights were gained into the onset temperatures and nature of thermal events, allowing the identification of gasless reaction pathways in systems such as Si/Bi₂O₃ and Si/Sb₂O₃. This technique proved critical in distinguishing between thermal decomposition, phase transitions, and actual redox reactions occurring in the solid state.

Complementary bomb calorimetry and thermochemical modelling further allowed estimation of reaction enthalpies, confirming proposed reaction schemes and enabling predictions of combustion performance. X-ray diffraction (XRD) and SEM imaging of combustion residues validated these assumptions by revealing phase composition and morphological characteristics of the reaction products.

Taken together, these methods constitute a robust experimental framework for the rational design of time-delay compositions. By combining kinetic, thermal, structural, and morphological data, they enable a deeper understanding of the combustion behaviour of pyrotechnic systems, supporting the development of delay elements with improved reliability, reproducibility, and environmental safety.

The findings discussed in this review build upon and expand the authors' previously published work, providing an integrated perspective on how analytical techniques contribute to optimising formulation and performance of gasless and lead-free pyrotechnic delays.

REFERENCES

- Monjezi, M., Mohamadi, H.A., Barati, B. & Khandelwal, M. 2014. Application of soft computing in predicting rock fragmentation to reduce environmental blasting side effects. *Arab J Geosci* 7: 505-511.
- Xu, S., Chen, T., Liu, J., Zhang, C. & Chen, Z. 2021. Blasting vibration control using an improved artificial neural network in the Ashele copper mine. *Shock Vib*.
- Xu, S., Li, Y., Liu, J. & Zhang, F. 2019. Optimisation of blasting parameters for an underground mine through prediction of blasting vibration. *J Vib Control* 25: 1585-1595.
- Focke, W.W., Tichapondwa, S.M., Montgomery, Y.C., Grobler, J.M. & Kalombo, M.L. 2019. Review of gasless pyrotechnic time delays. *Propellants Explos Pyrotech* 44: 55-93.

- Charsley, E.L., Chen, C.H., Boddington, T., Laye, P.G. & Pude, J.R.G. 1980. Differential thermal analysis and temperature profile analysis of pyrotechnic delay systems: ternary mixtures of silicon, boron and potassium dichromate. *Thermochim Acta* 35: 141-152.
- Gerlich, M. & Wojewódka, A. 2020. Study of gasless compositions used in time-delay elements. *J Therm Anal Calorim* 139: 3473-3479.
- Gerlich, M., Trzciński, W.A. & Hara, M. 2025. Combustion characteristics and thermochemistry of selected silicon-based compositions for time-delay detonators. *Materials* 18: 1456.
- Gerlich, M., Hara, M. & Trzciński, W.A. 2023. Tungsten and copper (II) oxide mixtures as gasless time-delay compositions for mining detonators. *Materials* 16: 3797.
- Gerlich, M., Trzciński, W.A. & Hara, M. 2022. Some aspects of the burning process of antimony and potassium manganate(VII) compositions. *Materials* 15: 4736.
- Chase, M.W. 1998. NIST-JANAF *Thermochemical Tables, 4th edition*. New York: American Chemical Society and American Institute of Physics for NIST.
- Kalombo, L., Del Fabbro, O., Conradie, C. & Focke, W.W. 2007. Sb_6O_{13} and Bi_2O_3 as oxidants for Si in pyrotechnic time delay compositions. *Propellants Explos Pyrotech*.
- Rugunanan, A.A. & Brown, M.E. 1993. Combustion of binary and ternary silicon/oxidant pyrotechnic systems, part II: Binary systems with Sb_2O_3 and KNO_3 as oxidants. *Combust Sci Technol* 95(1-6): 85-99.
- Barin, I. 1995. *Thermochemical Data of Pure Substances, 3rd edition*. Weinheim: VCH.
- Qiu, Y., Yaug, M., Fan, H., Zuo, Y., Shao, Y., Xu, Y., Yang, X. & Yang, S. 2011. Phase transitions of α - and β - Bi_2O_3 nanowires. *Mater Lett* 65(4): 780-782.

HMX based electronic initiation system for blasting in extremely reactive ground

K.M. Dufresne

DynaEnergetics, Europe GmbH, Troisdorf, German

ABSTRACT: Blasting in geothermally high temperature ground and/or extremely reactive ground is an increasing common challenge for surface and underground mines. Many current PETN based non-electric and electronic detonators have manufacturer published in-hole temperature limits ranging from 55 °C to 65 °C (131 °F to 150 °F). To overcome limitations of PETN based initiation systems, mines are known to load blastholes with a qualified high temperature bulk emulsion that are top primed with a cartridge of temperature qualified detonator sensitive emulsion plus an RDX detonating cord downline. A mining company with a site characterised by geothermally high temperature ground plus highly reactive ground employed a blasting system like the one described above asked DynaEnergetics (DEs) to develop an electronic initiation system for application in geothermally high temperature ground and/or extremely reactive ground conditions. The DynaEnergetics IGNEO high temperature electronic initiation system is rated for in-hole use in ground up to 150 °C (302 °F) for 48 hours. (DMC 2020).

1 TRADITIONAL BLASTING METHODS IN EXTREMELY REACTIVE GROUND

Reactive ground is a term used in the mining industry to describe the condition where an exothermic chemical reaction can occur between sulphide bearing rock and blast holes loaded with nitrate-based explosives. Reactive ground conditions increase the risk of explosives deflagration and/or unplanned detonations. (Scovira 2015)

Blasting in geothermally high temperature ground and/or extremely reactive ground is an increasing common challenge for surface and underground mines. Many current PETN based non-electric and electronic detonators have manufacturer published in-hole temperature limits ranging from 55 °C to 65 °C (131 °F to 150 °F). When confronting blasthole temperatures above the manufacturer published limits, operations may

request site-specific guidance and qualification for use. The advised safe loading window for in-hole PETN based detonators is often too short to be practicable for production blasting.

To overcome limitations of PETN based initiation systems, surface mines with high temperature ground and/or reactive ground are known to load blastholes with a qualified high temperature bulk emulsion that is top primed with a cartridge of temperature qualified detonator sensitive emulsion plus an RDX detonating cord downline (typically 8 g/m, 40 gr/ft), (Pieterse & Small 2022). The primer assembly of emulsion cartridge and detonating cord is sometimes used in conjunction with a thermally resistant plastic shell. The exposure time of this primer assembly is typically qualified for 8 hours at a maximum temperature of 150 °C (302 °F).

As there is no in-hole delay detonator in this system, blasthole timing is accomplished with



Figure 1. IGNEO HT150 electronic detonator and booster charge assembly.

surface delays. Although this system is safe, top priming and surface delaying does usually negatively influence floor level quality, muckpile fragmentation uniformity, and top size. Additionally, product safe loading windows times reduce blast block size and increase blasting frequency.

2 DYNAENERGETICS IGNEO HIGH TEMPERATURE ELECTRONIC INITIATION SYSTEM

Newcrest Mining Limited desires to improve blasting performance at the Lihir Mine in Papua New Guinea by moving to an in-hole delay initiation system that can withstand the geothermally high temperature ground and extremely reactive ground characteristic to the mine. The Lihir Mine is in the crater of a dormant volcano and is the hottest open cut gold mine in the world. In mid-2017, Newcrest approached DEs to develop a very high temperature electronic initiation system that would enable in-hole timing delays as opposed to surface timing delays. Since this time, DEs has developed an HMX (High Melting Explosive) based in-hole electronic initiation system specifically for blasting in geothermally hot and/or extremely reactive ground conditions. Restrictions due to the pandemic paused field evaluations of the system at the Lihir site, and this work is anticipated to resume in 2024.

DEs is a provider of well-completion and well de-commissioning solutions to the global oil and gas industry. DEs integrated perforating systems are recognised by the unconventional oil and gas sector as being safe, efficient, and reliable. DEs perforating systems include an intrinsically safe

initiation system that is immune to voltage, stray current, and radio frequency energy. Key components of the system include an igniter, surface tester, and firing panel. Electronic detonators for oil field applications can withstand temperatures up to 260 °C (500 °F) for a limited period of time.

DEs has adapted their seismic perforation system technology to a very high temperature electronic initiation system designed for the mining industry. The IGNEO system can withstand in-hole temperatures up to 150 °C (302°F) for up to 48 hours. It is intrinsically safe from stray current, stray voltage, and radio frequency energy. In the field, the system handles very much like other electronic initiation systems.

The foundation of the system is the HT150 high temperature electronic detonator as shown in Figure 1. The detonator has an HMX base charge that is encapsulated in a copper shell with an anti-corrosion coating. HMX has a decomposition temperature of 287 °C (549 °F) as compared to PETN with a decomposition temperature of 202 °C (396 °F). The downline is double coated thermally resistant wire with a harness wire surface connector clip. The detonator is pressure rated to 3 Bar. The time delay is user programable from 1ms to 1500ms in 1ms intervals. The detonators are Class 1.4B and can be shipped by air.

The system also has a unique booster charge to complete the primer assembly. The business end of the booster charge is made of an encapsulated 20+ gram HMX perforating shaped charge. The detonator and booster charge assembly measures 6.5 cm x 23.0 cm (2.6 in x 9.1 in). The booster charges are Class 1.4D and suitable for air cargo

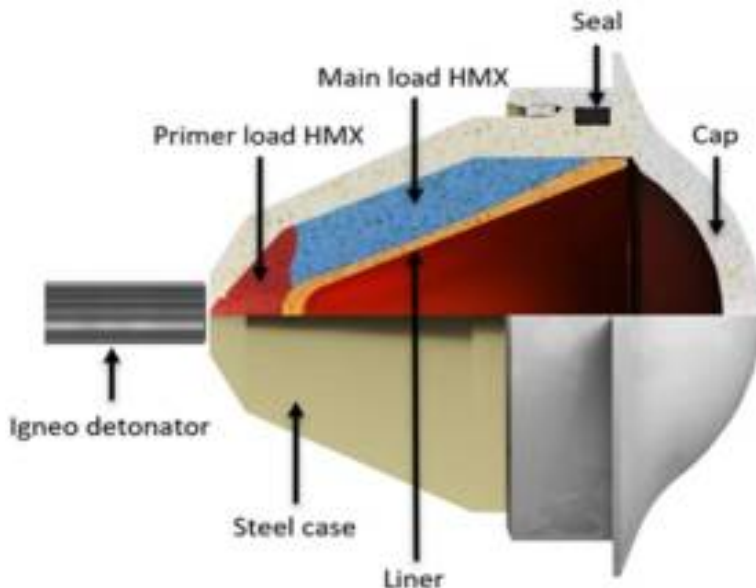


Figure 2. IGNEO booster charge, cross section.

shipment. See Figure 1. IGNEO HT150 Electronic Detonator and Booster Charge Assembly, Figure 2 – IGNEO Booster Charge – Cross Section.

The shape charge generates a high pressure, high velocity jet that reliably detonates bulk emulsion charges, including those with high percentages of urea inhibition to combat reactive ground.

The figure shows a perforating jet entered a concrete column target. The modelling software we use is from AutoDyn employing the Ansys Workbench module. Basically, the original 3-D AutoCAD model is imported from Autodesk Inventor into the modelling software. Certain material parameters are set and then a simulation is run based on either the Lagrange or Euler

mathematical equations.

See Figure 3 - Shape Charge Detonation Computer Model.

The programming and testing device is intrinsically safe, and the housing is made of aluminium and designed to withstand challenging operating conditions. Detonator programming is done after hooking up the detonator to the harness line. Detonator timing may be assigned by the programming device or by a computer-generated blast design file. Up to 50 detonators may be tested while connected to the same harness line. See Figure 4 IGNEO Programming and Testing Panel.

The firing panel is intrinsically safe and has an intuitive user interface. The firing panel can



Figure 3. Shape charge detonation computer model.



Figure 4. IGNEO programming and testing device.

control up to 1000 detonators and has wired or wireless trigger capability. See Figure 5 - IGNEO Firing Panel.

3 APPLICATIONS FOR HMX BASED HIGH TEMPERATURE INITIATION SYSTEM

High percentage urea inhibited nitrated based

emulsions that are qualified for geothermally hot and/or reactive blasting applications up to 150 °C (302 °F) are available from several manufacturers. In the case of reactive ground, the safe in-hole residence time for inhibited emulsions is temperature and time exposure dependent and determined by reactive ground versus nitrate based explosive isothermal testing. While high



Figure 5. IGNEO firing panel.

temperature emulsions have been available for over 25 years, the choice of initiation systems to use with these products has been limited.

In the USA, unrestricted sleep time of in-hole detonators with PETN base charges is limited to ground temperatures 65.5°C (150°F) or below. In the USA, with explosive manufacturer qualification and MSHA variance, a blasting system consisting of an in-hole PETN based non-electric detonator, packaged detonator sensitive emulsion, and inhibited bulk emulsion may be used be up to 80 °C (175 °F) for 4 hours. In the USA and Canada, under qualification and approval, in-hole PETN based electronic detonators have been used underground in ground temperatures up to 85 °C (185 °F) for 4 hours. Other countries have similar high temperature and/or reactive ground blasting qualifications and regulations and should be confirmed before use.

When blasting in hot and/or reactive ground with temperatures greater than 80 °C to 85 °C (175 °F to 185 °F), the global go-to initiation system for

priming has been the RDX detonating cord and high temperature emulsion cartridge emulsion previously described.

With the introduction of an electronic HMX detonator and HMX primer, it is now possible to pair this initiation system with a qualified high temperature emulsion, for an in-hole delayed blasting system rated for ground temperatures up to 150 °C (302 °F) for 48 hours. In-hole bottom initiation as compared to top priming and surface delaying is known to improve muckpile fragmentation uniformity and reduce oversize. For blasting in ground temperatures from 65.5 °C to 85 °C (150 °F to 185°F), the HMX electronic initiation can expand the blast window from 4 hours to 48 hours. This allows operations to significantly increase blast block size and/or reduce blasting frequency.

Numerous mining operations can benefit from the utilisation of an HMX based initiation system. Underground precious and base metal mines with geothermally hot ground may not have to shut down development headings for days at a time to

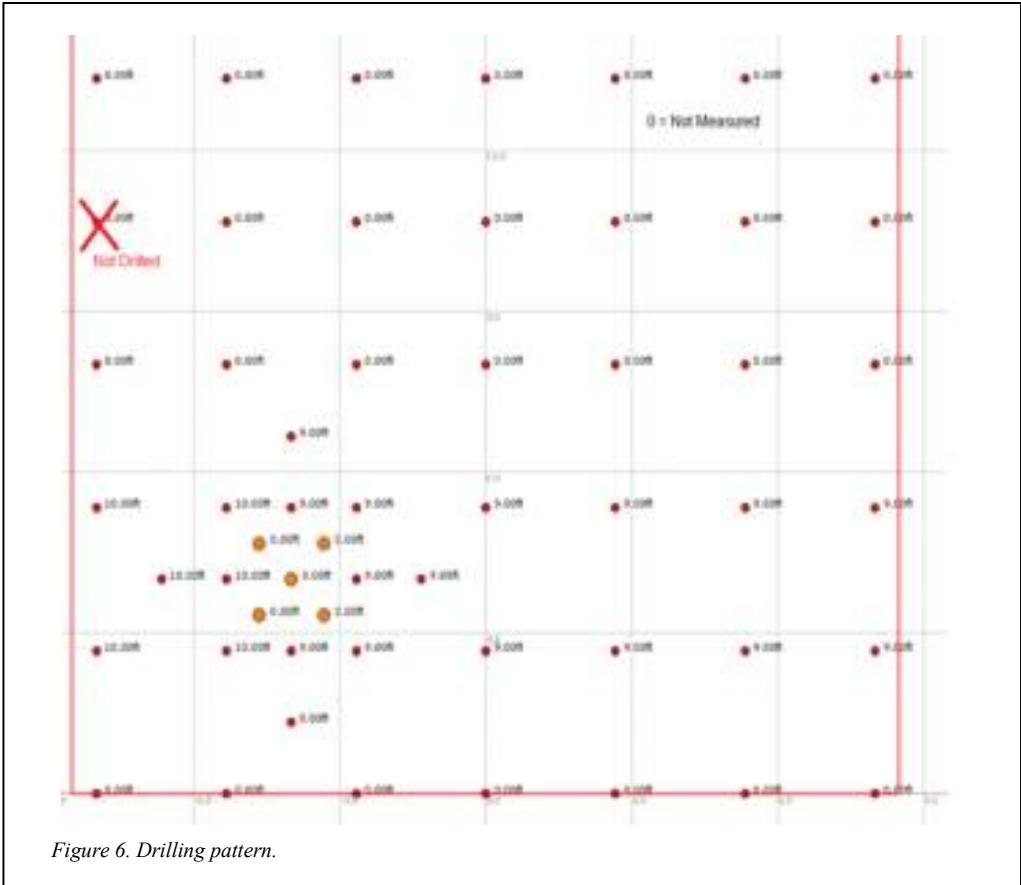


Figure 6. Drilling pattern.



Figure 7. Recorded in-hole temperature.

allow them to cool down before blasting. Surface operations characterised by high temperature reactive ground may be more able to blast through unexpected zones of extremely reactive, high temperature ground without resorting to mechanical breakage. Surface coal operations with reactive hot overburden and/or burning coal seams now have an initiation system option that permits safer and improved blasting of hot blocks.

4 CAMPAIGN BLASTING

In western USA, two underground gold mines encounter very high temperatures in their development heading, between 165 and 200 F, and decided to use the IGNEO system for the first time. Demonstration blasts were initiated successfully (Proulx & Scovira 2000).

Nevada Gold Mine - Meikle underground Operation

Application - The underground mine regularly sees in-hole temperatures well above 200 °F, hence the need for the demonstration shots. The mine had determined two development drives to test the detonators. The first location had hole temps averaging 260 °F but due to the average depth of holes (4 ft versus 10 ft that was drilled) it was decided to not use the detonators in this location due to very poor ground condition. The

2nd location had a much lower average in-hole temperature but slightly better drilling quality than the first location.

Drilling - The face was 25 ft wide by 17 ft high. The decision was made to neck down the heading to 17 ft wide in order to have the same drill pattern that was already proven successful with 1.5 seconds of delay. Drilling was a 6 x7 pattern with 32 inch spacing. Planned drill length 10 feet. Only 6 of the 21 measured holes were to depth. 2" holes :49 holes. 4" relief holes: 5 holes drilled as shown in Figure 6.

Hole temperature - Hole temperatures were recorded from a single hole from each quadrant. The recorded temperatures were 171 °F, 157 °F, 164 °F and 168 °F.

Loading - The primer consisted of an IGNEO detonator inserted into a stick of Pyromex Primers were pushed to the toe of the hole with the loading hose and the holes were loaded with Subtek Eclipse emulsion. Many of the holes were collapsed making it difficult to get the loading hose to the toe of the hole. Load lengths varied anywhere from 3-10 ft, averaging about 7 ft.

Timing - Typical development rounds are fired anywhere from 7 to 9 seconds. For the demonstration shot, a modified timing plan had to be used due to the 1.5 second limitation of the system.

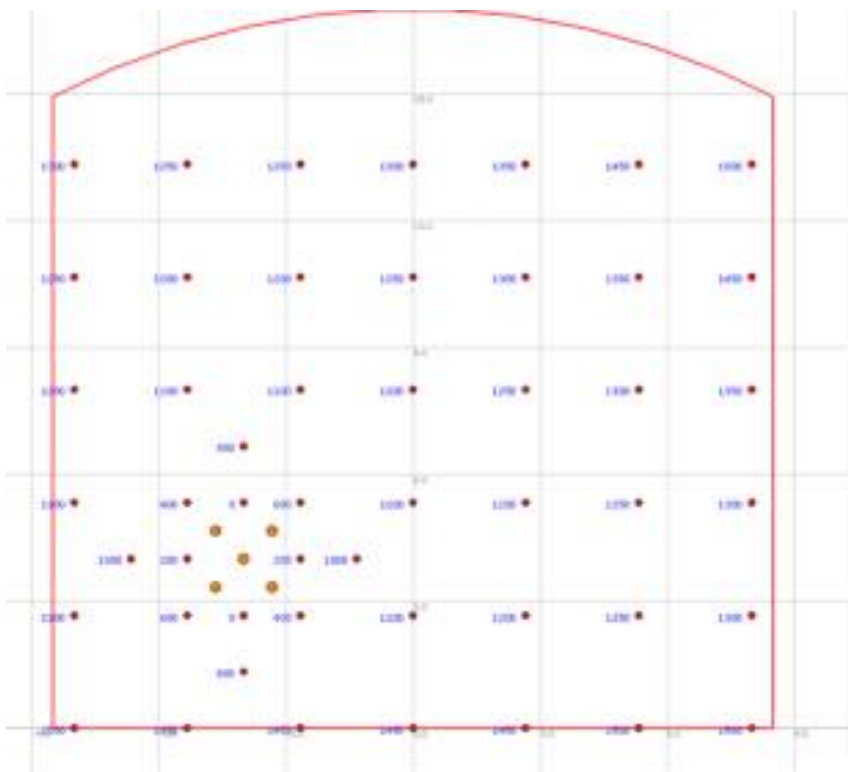


Figure 8. Timing pattern.

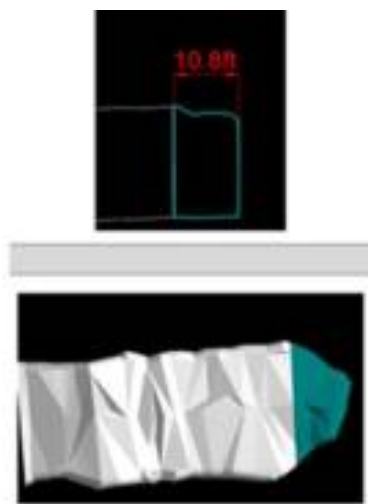


Figure 9. Results.

1000 ms were used in the burn to allow time for the material to clear. The remainder of the time was used to shoot the remainder of holes in 50 and 100 ms increments as shown in Figure 8.

Results - Based on initial laser measurement and final survey the round pulled under 11 ft at the deepest part. Results were consistent with the hole measurements taken. The holes on the left that were measured were drilled to depth whereas holes on the right that were measured were all short as shown in Figure 9.

5 CONCLUSIONS AND FUTURE WORK

An HMX based electronic initiation system has been developed and introduced to the mining industry that has the potential to improve blast performance and efficiencies in operations with high temperature ground and/or extremely reactive ground.

When combined with a qualified high temperature emulsion, the HMX initiation system enables blasting in ground temperatures up to 150 °C (302 °F) for 48 hours. This extended loading window enables blasting of larger blocks and/or decreased blasting frequency.

The HMX based electronic initiation system permits bottom in-hole delaying in ground temperatures above 80 °C (175 °F). As compared to top priming and surface delaying, in-hole bottom delaying is known to improve floor quality, muckpile fragmentation uniformity and reduce oversize.

Field evaluation of the HMX initiation system at the Lihir Mine is continuing.

Work on the next generation of the HMX initiation system is underway and will bring with it expanded application range and flexibility.

Demonstration blasts have taken place at 2 underground gold mines in western USA and results will be presented at the conference in September.

6 ACKNOWLEDGMENTS

The author expresses thanks to D. Scott Scovira, Global Manager Blasting Science, BME Explosives, for guidance and assistance in preparing this paper.

REFERENCES

DMC Press Release, 06 August 2020. DynaeEnergetics introduces IGNEO™ Intrinsically Safe™ initiating system for high-temperature mining applications.

Pieterse, D. & Small, G. 2022. Reactive ground blast management in a South Africa open pit zinc mine. *Proceedings of the Forty-Eighth Conference on Explosives and Blasting Technique, International Society of Explosives Engineers*, Cleveland, Ohio.

Proulx, R.P. & Scovira, D.S. 2000. Drilling and blasting in hot and reactive ground conditions at Barrick Goldstrike's Mickle mine. *Proceedings of the Twenty-Sixth Conference on Explosives and Blasting Technique, International Society of Explosives Engineers*, Cleveland, Ohio, pp. 319-331.

Scovira, D.S. 2015. Ammonium nitrate (AN) based explosives and reactive ground awareness. *CIM Convention*, Montreal, Quebec.

[RETURN TO CONTENTS PAGE](#)

Eliminating the TNT risk for opencast boosters

G.J. Stenzel

Glencore Coal, Johannesburg, South Africa

R.P.G. Nel

AECI Mining Explosives, Johannesburg, South Africa

ABSTRACT: The increasing global shortage of Trinitrotoluene (TNT) has highlighted challenges in the sourcing and application of traditional explosive boosters, particularly in the mining sector. TNT has long been a key component in booster formulations; however, its production is concentrated in regions such as China and Russia, where geopolitical factors have contributed to supply chain disruptions. Additionally, the inherent risks associated with TNT and Pentaerythritol tetranitrate (PETN) in handling, storage, and transportation necessitate the exploration of alternative formulations. This study examines the development and implementation of a two-component booster by AECI that eliminates the use of TNT and PETN. The booster, which has been deployed at Glencore opencast coal mining operations in South Africa, is manufactured using locally sourced raw materials that are non-explosive in their base form. This design presents several advantages: **safety enhancements** - the removal of explosive precursor materials reduces handling and transportation risks; **regulatory compliance** - the classification of the raw materials as non-explosive simplifies logistical and regulatory requirements; **environmental considerations** - reduced reliance on imported materials lowers the carbon footprint associated with transportation and supply chain logistics; **economic and operational benefits** - the potential for on-site manufacturing at mining operations enables local enterprise development and job creation, contributing to regional economic sustainability. Furthermore, the booster incorporates a sealed shell design, which minimises exposure of the components to elevated temperatures in blast holes, thereby enhancing safety and performance reliability. This paper discusses the formulation, manufacturing process, and field implementation of the TNT-free booster, evaluating its implications for mining safety, regulatory frameworks, and supply chain resilience. The findings suggest that adopting non-explosive raw materials in booster technology presents a viable solution to mitigate the risks associated with traditional TNT-based systems while offering economic and environmental benefits. The use of these new-generation boosters has made hot hole charging safer and they have been embraced by the staff on the benches at Glencore Coal in particular when charging hot holes.

1 INTRODUCTION

The global shortage of Trinitrotoluene (TNT) has created significant challenges for the mining industry, particularly in the sourcing and

application of traditional boosters. This shortage is driven by a combination of escalating military demand and supply chain disruptions. With limited production concentrated in a few countries and commercial users forced to compete with defence

sectors, prices have surged, and availability, coupled with supply chain challenges, has become increasingly unpredictable. This, coupled with the traditional restrictions in shipping raw materials with explosive properties, necessitated a review.

Against this backdrop, AECl identified the need to develop an innovative alternative to PETN/TNT based boosters that not only addresses supply chain and safety concerns but also performs reliably in demanding field conditions, including hot-hole and geothermal blasting environments. These high-temperature conditions are common in opencast coal mining operations and pose serious safety and performance risks for conventional boosters, which may degrade or fail under extreme thermal conditions.

2 ADDRESSING HIGH-TEMPERATURE BLAST HOLE CONDITIONS

To further enhance the thermal resilience of the formulation in high temperature blasting conditions, AECl developed a heat-resistant shell as an alternative to standard polypropylene casings, specifically for use in applications where

prolonged exposure to elevated temperatures is a concern.

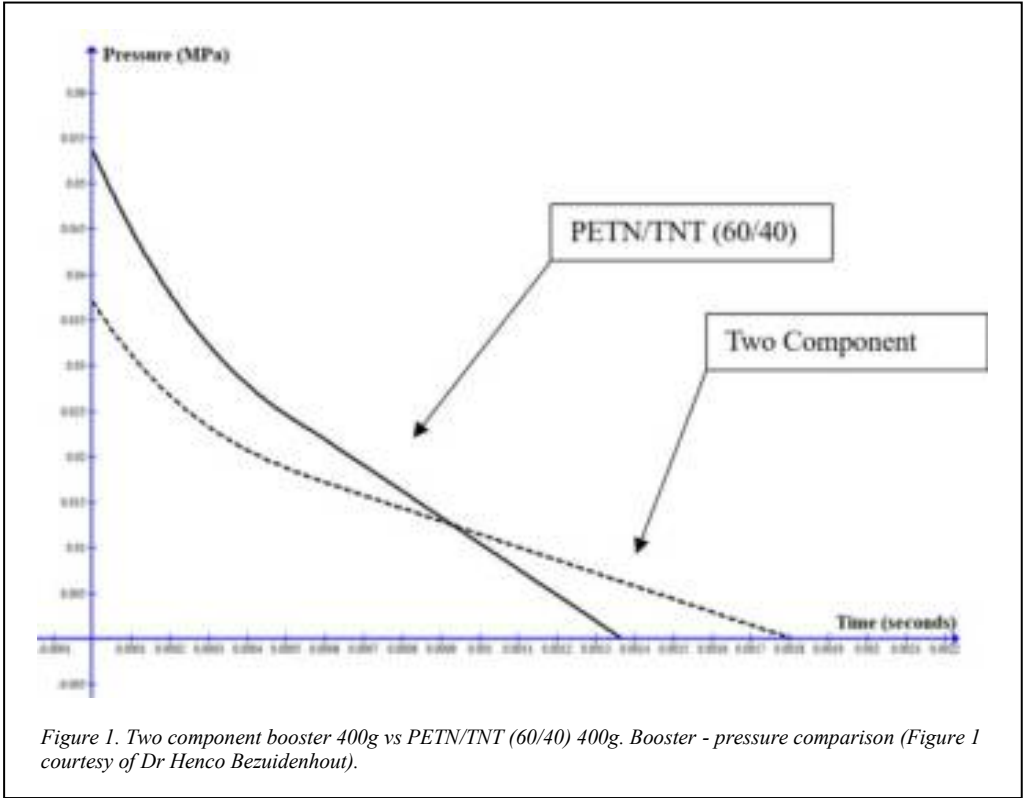
The redesigned shell features a screw-fit cap that forms a robust mechanical and waterproof seal, which successfully passed a 96-hour, 3-bar pressure pot test with no evidence of water ingress. This shell innovation, coupled with the non-explosive two-component formulation, offers a significant performance advantage over traditional TNT-based boosters.

Given that TNT melts at just 80.2 °C, conventional cast or moulded boosters are at risk of softening, leaking, or failing in hot-hole conditions. In contrast, the two-component booster system provides approximately 68% greater thermal tolerance before the risk of deflagration, making it far more reliable in extreme thermal environments.

Following laboratory validation, the high-temperature booster was subjected to rigorous pickup sensitivity testing using:

- Electronic detonators.
- Shocktube (8D strength) detonators.
- Double-strand 8 g/m detonating cord/fuse.

In all configurations, the boosters detonated reliably and produced consistent indentations on



mild steel witness plates, confirming strong and stable energy output under elevated temperature and humidity conditions. Figure 1 is a representation of the energy output of the two-component boosters vs traditional PETN/TNT Boosters.

The detonation impulse in air 3 m from origin (at stable detonation - P_{ej}) is approximately 13% lower than for Pentolite at a diameter of 58 mm. It is important to note that detonation impulse is a critical parameter, and the relatively small difference of approximately 13% between the two component booster and the PETN/TNT (60/40) booster is sufficient to ensure that the booster delivers adequate energy to reliably initiate bulk emulsion explosives.

Below is the formula for determining detonation impulse, where there is an element of time included.

$$I = \int P(t) dt$$

Where:

I = impulse (N·s or Pa·s)

$P(t)$ = pressure as a function of time

t = duration over which the pressure is applied.

3 FURTHER TESTING

The following tests were completed on the new two component boosters to prove intrinsic safety in handling, transport, and storage, to demonstrate equivalent performance to legacy PETN/TNT boosters.

Impact testing - Purpose: To assess whether the formulated booster becomes sensitive to impact after activation or sensitisation, and to ensure the non-explosive precursors remain safe during manufacturing and transport.

Friction testing - Similar to impact testing, this assesses the propensity of the final booster to ignite or detonate under shear or rubbing forces and confirms safe handling pre-activation and checks the post-activation formulation for potential friction sensitivity.

Electrostatic discharge testing - To determine whether the activated booster (or raw material, if applicable) can be ignited or initiated by static electricity and demonstrate that the raw materials are ESD-inert, and the activated compound remains no more ESD-sensitive than standard PETN/TNT boosters.

Booster/detonators interface testing - To assess if the booster can be reliably initiated by standard detonators (e.g. No. 8 blasting caps, electric or electronic variants). This is a critical performance metric and involves measuring initiation reliability, observing detonation velocity

buildup, and verifying symmetry of the shockwave transmission.



Figure 2. Large scale gap test setup (picture courtesy of Dr Henco Bezuidenhout).

Booster/detonating cord interface testing - to validate the ability of the booster to accept and propagate a detonation from standard detonating cord, cords of various strengths (e.g. 10-40 grams/metre).

Booster with various emulsions interface testing - Initiation threshold and performance across different explosive types (emulsions, ANFO, water gels) and whether the booster meets or exceeds benchmarks in terms of brisance and detonation pressure.

Water ingress testing (at different pressures) - to ensure the booster's casing and internals remain sealed and dry, even when submerged under pressure.

Drop testing - to simulate accidental drops during storage, transport, or emplacement. Confirms the booster doesn't go off, degrade, or crack under realistic handling conditions and ensures the mechanical integrity of the casing and stability of internal materials.

Hi temperature testing (shell integrity higher than 100°C) - to check whether the booster maintains structural and chemical integrity at elevated temperatures. Verifies that no melting, deformation, or thermal degradation occurs and that no premature reactions are triggered, especially during transport.

Large-scale gap test - To measure the shock sensitivity and power of the activated booster. A standard test for assessing a booster's ability to transmit a detonation across a gap (usually made of inert material like PMMA). Refer to Figure 2.

4 FIELD TESTING

This was the most essential component of the product introduction and resulted in many

iterations of the final product. Before commercial introduction, AECI tested 84,463 units across various African sites to ensure consistent performance and operational safety.

Field testing was essential to confirm that the booster can reliably initiate bulk explosives when deployed in actual blastholes and that the product would reliably withstand the harsh hot-hole conditions experienced at the Glencore South African coal mines. It also verified consistent performance across a range of practical variables, including different blasthole depths, diameters, and charge configurations. Additionally, the field testing verified the booster's effectiveness under diverse geological conditions where initiation challenges are difficult to simulate in laboratory settings.

At Glencore Coal's South African operations, where chargeable blasthole temperatures can reach up to 120°C, this innovation, in combination with bulk explosives developments, has enabled safe charging in conditions previously considered too hazardous.

Since implementation, over 1,833,322 units have been fired without incident at Glencore's South African coal mines.

In addition to its proven reliability, operational



Figure 3. Application at Glencore Coal South Africa (picture courtesy of Gerhard Stenzel).

crews have shown a clear preference for the booster's shell design, which allows for easy and secure detonator placement, as well as the convenient attachment of weights when required, enhancing its practicality in varied field applications.

5 REDUCTION IN HANDLING AND TRANSPORTATION RISKS

Removing TNT and PETN from booster formulations offers significant safety advantages, starting with a reduced risk of accidental detonation. These materials are highly sensitive to shock, friction, and heat, making them hazardous during storage, handling, and transportation. Their elimination substantially lowers the likelihood of unintended explosions, enhancing safety across the supply chain.

There is also an eased regulatory burden. Explosives are subject to strict controls under both national and international frameworks, such as the UN Recommendations on the Transport of Dangerous Goods. Switching to non-explosive precursors simplifies compliance, reducing the administrative workload and logistical constraints typically involved in handling and moving Class 1 hazardous materials.

As a result of this risk reduction, operations may also benefit from lower insurance costs as premiums and liabilities are tied to the severity of potential incidents. With non-explosive materials, the likelihood and consequences of catastrophic events are greatly diminished.

The use of safer, non-explosive materials also enhances supply chain stability. These materials are easier and less costly to transport and store, particularly in remote or logistically challenging regions. This improves reliability and helps maintain production schedules even during disruptions.

6 EASE IN REQUIRED REGULATORY COMPLIANCE.

Utilising non-explosive raw materials classified under international regulations, such as those from the UN, IATA/ICAO, ADR, and IMO significantly streamlines logistics by exempting materials from stringent UN Class 1 explosive requirements. This eliminates the need for specialised packaging, hazard placards, transport permits, and vehicle restrictions, simplifying both the documentation and handling process.

Air transport benefits as non-explosive materials avoid the complex compliance demands

imposed on explosives by IATA/ICAO regulations. This is particularly valuable for servicing remote operations where quick, reliable delivery is essential. Similarly, road and sea transport are more efficient, as non-explosive materials bypass strict IMDG and ADR requirements, reducing costs and the risk of logistical delays.

Administrative overheads are also reduced. Unlike explosives, non-hazardous materials do not require licenses, dedicated storage, or frequent inspections. This lowers regulatory burdens and simplifies compliance management across the supply chain.

The overall impact results in improvements in operational efficiency, safety and supply chain flexibility.

7 ENVIRONMENTAL CONSIDERATIONS

Using locally sourced, non-explosive raw materials significantly reduces transportation-related greenhouse gas (GHG) emissions. Locally manufactured raw materials minimise carbon output and help reduce scope 3 emissions indirectly generated by supply chains. This aligns mining operations and suppliers with broader sustainability objectives.

Environmental compliance is another major advantage. Countries like South Africa are increasingly enacting stringent climate policies aligned with frameworks such as the Paris Agreement, Nationally Determined Contributions (NDCs), and corporate GHG reporting standards like ISO 14064 and the Greenhouse Gas Protocol. Lowering transport emissions helps mining companies and their suppliers meet these regulations and supports environmental, social, and governance (ESG) reporting and carbon disclosure requirements.

Non-explosive materials further reduce environmental risk during transport by eliminating hazards such as chemical spills, fires, or contamination.

Lastly, sourcing materials locally (near-shoring) contributes to economic sustainability by supporting regional suppliers and reducing dependency on global freight, an essential pillar of sustainable development.

8 ON-SITE MANUFACTURING CAPABILITY

An added benefit of the product is that remote on-site manufacturing of boosters (where economies of scale allow) through modularised

systems provide additional risk mitigation and security of supply. It enables just-in-time production, minimising the need to store large volumes of hazardous final materials and reducing the financial risks associated with supply chain disruptions.

This localised approach enables job creation and skills development within mining communities.

Additionally, on-site production stimulates local enterprise development. The need for raw materials, packaging, maintenance and logistics services boosts the demand for support industries.

On-site or near-site manufacturing reduces exposure to geopolitical risks and international shipping delays, which is particularly relevant for critical materials like TNT.

Furthermore, local manufacturing aligns with ESG and corporate social responsibility (CSR) goals. It provides tangible evidence of a company's contribution to local communities, improving its social license to operate and overall ESG performance.

Finally, manufacturing boosters on demand increase operational flexibility.

9 CONCLUSION

The elimination of TNT and PETN in booster formulations marks a significant advancement in the safety, sustainability, and operational efficiency of blasting practices in the mining sector. By leveraging non-explosive, locally sourced raw materials and introducing a thermally resilient, sealed-shell booster design, AECI has addressed critical challenges posed by high-temperature blast conditions, stringent explosives regulations, and volatile global supply chains.

Extensive laboratory testing, followed by large-scale field deployment across African sites, has validated the booster's reliability, safety, and performance under real-world conditions.

Beyond safety and technical robustness, the innovation supports broader strategic goals. It simplifies regulatory compliance, reduces greenhouse gas emissions, and improves supply chain resilience.

Furthermore, the potential for on-site manufacturing enables cost-effective, just-in-time production, while fostering job creation, local enterprise development, and alignment with ESG and CSR principles.

In summary, the TNT free booster offers a scalable, safer, and more sustainable alternative to traditional systems.

10 ACKNOWLEDGEMENTS

Thanks go to: Dr Henco Bezuidenhout (Technical Manager AECI Mining, Modderfontein, South Africa), Gerhard Stenzel (Coauthor and General Manager, Opencast Technology, Glencore Coal, South Africa), Ronald Huggins (Product Manager, Initiating Systems & Boosters, AECI Mining, South Africa) and Hendrik Botha (Global Portfolio Manager: Explosives, AECI Mining, South Africa).

REFERENCES

- AECI Mining 2024. Case study: decentralised booster production and its socioeconomic impacts. *Internal Report*.
- AECI Mining 2024. Innovation in sustainable explosives: field performance and supply chain adaptation. *Internal Technical Report*.
- AECI Mining 2024. Two component booster formulation evaluation. *Internal Technical Report*.
- Glencore. 2023. *Sustainability Report: Community Investment and Supplier Development*.
- ICMM 2021. Enhancing mining's contribution to the sustainable development goals. *International Council on Mining and Metals*.
- South African Department of Trade, Industry and Competition (DTIC) 2023. Industrial policy action plan.
- South African Department of Forestry, Fisheries and the Environment (DFFE) 2021. National Climate Change Adaptation Strategy.

[RETURN TO CONTENTS PAGE](#)

The Shock-Wave Laboratory at the TU Bergakademie Freiberg

T. Schlothauer, K. Keller, E. Kroke & G. Heide

TU Bergakademie Freiberg, Freiberg High Pressure Research Centre

M. Kilo

Fraunhofer Institut für Silikatforschung, Wuerzburg

ABSTRACT: This paper describes the subterranean shock wave lab at the TU Bergakademie, its technical and legal basis and the consequences for practical experimental work. It includes examples of practical testing and reproduction of experiments showing the possibilities for material testing and material research. This work is aimed at practitioners from companies which work frequently with explosives and who are looking for an opportunity for testing material behaviour. With its high charge mass of 20 kg per shot and different charge geometries together with the possibility of using a wide range of high explosives the shock wave laboratory is a place to perform experiments with high efficiency. Additionally, the TU Bergakademie Freiberg has a large portfolio of physical and chemical characterisation for samples of all kinds.

1 THE SUBTERRANEAN SHOCK-WAVE LABORATORY AS PART OF THE FREIBERG HIGH-PRESSURE RE-SEARCH CENTRE

1.1 Short historical overview

In December 2006, at the TU Bergakademie Freiberg (TUBAF), the Dr. Erich-Krüger Foundation was formed and in 2007 the first Krueger Research Centre at the TUBAF was established. This was the Freiberg High-Pressure Research Centre (FHP) as a research network between different faculties of this University. Currently it includes a high-pressure facility laboratory with Multi Anvil Presses (MAP), a unit for material testing, synthesis and research under intense dynamic loading, the Shock-Wave Laboratory (SWL) and, recently added, a Diamond Anvil Cell (DAC).

TUBAF was founded in 1765 and has a long history as a teaching and research unit for geosciences, engineering and mining. The research and teaching mine is part of the education program (Figure 1) for geoscientists and mining engineers.

This mine, named 'Reiche Zeche', is regulated by German Mining Law, and includes all kinds of mining work - weathering, drilling and also blasting, including a storage room for the explosives. This will significantly reduce the approval procedure for blasting works.

The work to construct the laboratory unit with active ventilation and control room (Schlothauer 2016) was completely performed by workers from the mine itself without external companies. Simultaneously, blasting tests were performed to control the usability of the new systems. The first official approval of the subterranean shock wave laboratory was in 2020, currently it has a working permission until 2032. Before gaining its

permission, an expert opinion was prepared by the Federal Institut for Material Testing and Research Berlin (BAM Berlin). This opinion also contains security requirements concerning ventilation, ridge securing, fixtures for breaking the shock wave and additional security areas.

We have only one approving authority, the Saxonian Mining Office, this simplifies special authorisation procedures for blasting experiments. Because the lab stands under German Mining Law, the kind of experiments, explosives and ignition were written into the operational plan and, if it is permitted by the authority, all experiments are possible at all times without any additional contact with the authority. Currently up to 8 experiments per working day are possible, depending on charge mass and preparation time. Because the laboratory is an integrated part of the mine additional security measures like evacuation plan, storage rooms for explosives and its conditions and protection, weather management of the detonation products and also access possibilities are given in its main operational plan and additionally in its special operational plan for blasting works.



Figure 1. Above-ground units with the winding tower of the research and teaching mine 'Reiche Zeche'.

The work to construct the laboratory unit with active ventilation and control room (Schlothauer 2016) was completely performed by workers from the mine itself without external companies. Simultaneously, blasting tests were performed to control the usability of the new systems. The first official approval of the subterranean shock wave laboratory was in 2020, currently it has a working permission until 2032. Before gaining its permission, an expert opinion was prepared by the Federal Institut for Material Testing and Research Berlin (BAM Berlin). This opinion also contains security requirements concerning ventilation, ridge

securing, fixtures for breaking the shock wave and additional security areas.

We have only one approving authority, the Saxonian Mining Office, this simplifies special authorisation procedures for blasting experiments. Because the lab stands under German Mining Law, the kind of experiments, explosives and ignition were written into the operational plan and, if it is permitted by the authority, all experiments are possible at all times without any additional contact with the authority. Currently up to 8 experiments per working day are possible, depending on charge mass and preparation time. Because the laboratory is an integrated part of the mine additional security measures like evacuation plan, storage rooms for explosives and its conditions and protection, weather management of the detonation products and also access possibilities are given in its main operational plan and additionally in its special operational plan for blasting works.

1.2 Description of the Shock Wave Laboratory

Because this laboratory is open for civil researchers worldwide it will be described here in detail. The laboratory is located at the first underground level of the mine at a depth of approximately 150 m. It is positioned deep below surface and for this reason limited access guarantees high security concerning blasting works itself and also for the storage room (maximum capacity 500 kg explosives). Additionally, experimental blasting works in the lab have no impact for uninvolved persons as might occur in other locations such as in quarries. The laboratory has permissions for a maximum charge mass of 20 kg of high explosives (HE) per shot, calculated for plastified Hexogen (1,3,5-trinitro-1,3,5-triazacyclohexane (Klapötke 2007), Research Department Explosive, RDX). The following explosives are part of the operational plan:

- Plastified explosives, based on PETN (pentaerythritol tetranitrate), RDX and HMX (High Melting Explosive (Ezzati 2018), octahydro-1,3,5,7-tetranitro-1,3,5,7-tetrazocine (Smith 1999), Octogen).
- Gelatinous ammonium nitrate-based explosives.
- Emulsion explosives.
- Watertel explosives.
- Powdered ammonium nitrate explosives without oil.
- Detonation cords.

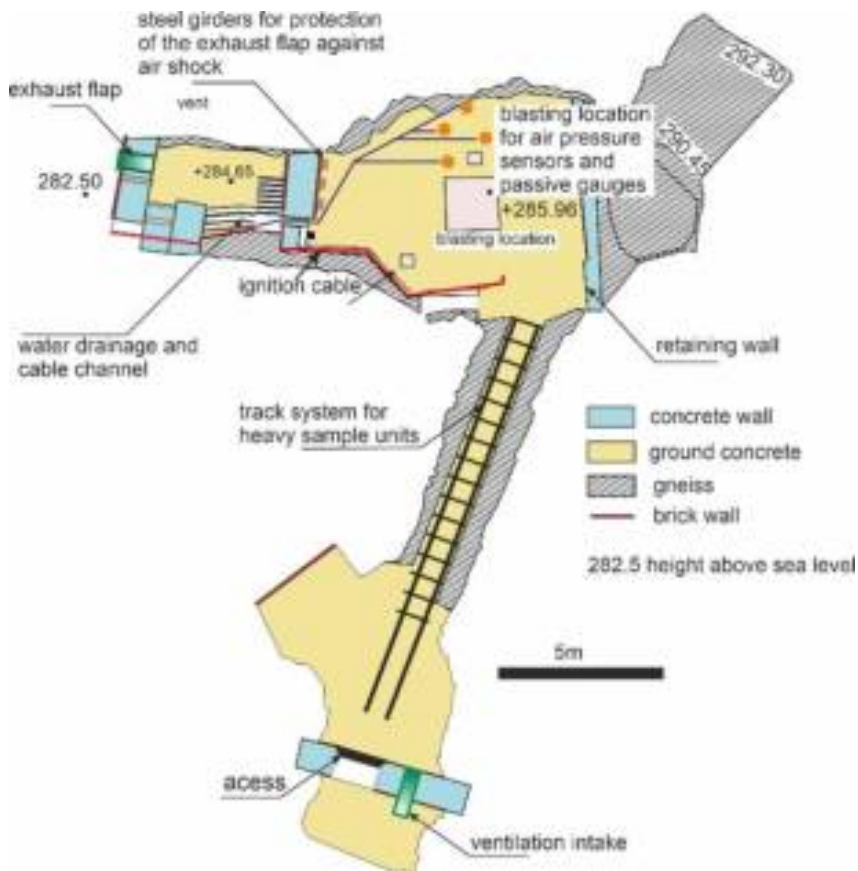


Figure 2. Underground layout of the laboratory.

- TNT (trinitrotoluol) only as exception by its very negative oxygen balance
- CompB (60% RDX, 39% TNT, 1% wax (Klapötke 2009)) only as exception by its very negative oxygen balance.

The maximum charge mass is independent from the used explosive, despite its different TNT equivalents (Goel 2015). Because the lab is only accessed through the winding tower (Figure 1) of the main shaft its size parameters, of 0.75 x 1.5 m (maximum load 2000 kg), are the limiting factor for the size of any parts planned for shock experiments. For larger setups an assembly in the laboratory itself will be required.

Underground, any heavy samples can be moved to the laboratory by train. Because the electric voltage in the blasting room is limited to 40 V, all high voltage equipment there is only temporary until the electric detonator is in the room or

engines, powered by compressed air, are in use. Additionally, it is necessary to avoid permanent 220 V installation in the blasting room due the high humidity and free water surfaces in the mine. A general overview about the laboratory is given in Figure 2. The blasting location is filled with sand to avoid undesired shock reflections from the reinforced concrete on the ground.

The ventilation has a volume flow of 110 m³/min, sufficient to clean the lab from the gaseous detonation products CO and NO_x in 45 minutes (for 2000g HE), measured with Draeger hand sensors near the ignition place.

Except the floor, which was made from 30 cm reinforced concrete, the walls and roof consist of gneiss, a stable and self-sustaining rock (elastic modulus from 55 GPa to 80 GPa, static, uniaxial compressive strength of 80 MPa to 120 MPa, volume of the chamber 211.25 m³). The ceiling is



Figure 3. View into the measurement and ignition room.

secured by reinforcement mats. All parts with openings to other parts of the mine are closed also with reinforced heavy concrete and clad with bricks.

The second part of the shock wave laboratory is the control and ignition room, safety tested already by the detonation of 40 kg Eurodyn 2000 in the blasting chamber (Figure 3).

It contains the place for ignition of the charges (electric detonators, U-detonator, 450 mA no-fire current during 5 min, moment ignitors, commercial capacitor blasting machine, standard ignition cable, one detonator per shot). Because this room contains different cables for measurements too, it is noteworthy that ignition and measurement cables are separated.

The measurement cables must be as short as possible and for this reason the connection to the location near the exhaust flap (Figure 4), where the measurement units are positioned, is performed through three drilling holes. From this location the temporary measurement cables connect the close surroundings of the charge with the electronic devices and this is connected with the measurement room by LAN.

For security reasons during firing all personnel are only allowed in the firing end measurement room.

The measurement and ignition room is equipped with LAN, two 8 channel oscilloscopes (60 MHz and 20 MHz), equipment for shorting pin measurements (pins from Dynasen inc), and possibilities for air pressure sensors (Tadini 2018)



Figure 4. Measuring equipment mounted near the vent, shown is a Tranet Oscilloscope with 8 channels.

for different air pressures (Kistler, 350 bar, 70 bar, 35 bar, one pencil probe 6233A). As a separate possibility for special experiments, evacuation of sample units is possible ($p_{\min}$ ca. 10^{-2} mbar).

For the preparation of sample holder units, sensor holders and other equipment, required for shock experiments, our own mechanical workshop is available. For data reduction and following calculations and also for the required EoS-calculations in-house self-developed MatLab codes were used.



Figure 5. Four sheets of pure Nickel before shot with distance holders. The fifth sheet is absent to show the positioning of the distance sticks.

2 TYPES OF EXPERIMENTS PERFORMED

2.1 Explosive welding / cladding

These experiments were performed with common commercial explosives with low detonation velocity (Lysak 2007). Charge masses vary between 4000 and 6000 g with sample sizes of 600*150*6 mm. Sample materials were black steel



Figure 6. Explosive welding-charge preparation with 6000 g emulsion (Senatel Powerfag), booster on the right side, still without electric detonator.

and pure Ni-metal, sheet number between 2 and 7 (Figure 5).

As booster 250 g of Eurodyn 2000 were used (Figure 6). All welding experiments were performed in a sand bed (thickness approx. 30 cm, additionally the samples were covered with sand before ignition.

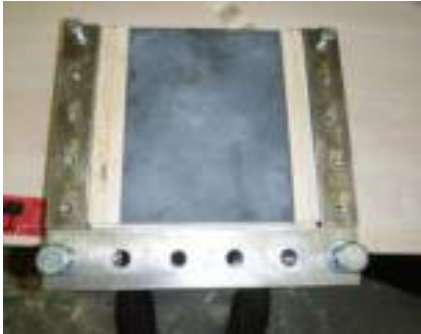


Figure 7. Sample holder for larger rock samples under dynamic loading before shock. Sample size: 250*250 mm, rock sample here Devonian shale.

After shock the metal sheets shows an intense material connection with the typical wave formed structural pattern (Carpenter 1975), also the cladding of 7 sheets is possible.

2.2 Dynamic testing of natural rocks and cylindrical charges

During mining and blasting processes the success of work depends often on the properties of rocks itself (Jug 2017). Unfortunately, it is very difficult to obtain rock samples which allows the complete



Figure 8. Cylindrical charge, load 4666 g ANFO, Booster: plastified PETN.

investigation of rock behaviour under well-defined pressure conditions during blasting to calculate the pressure-temperature data (Ahrens 1995). To overcome these problems a special sample holder for rock samples (250*250*34-45) mm was developed (Figure 7), shown with an example of devonian shale.

All metal parts are made of black steel. The general principle of the experiment is the acceleration of a metal plate (black steel, 260*50*5 mm) by the detonation of high explosives. During collision with the rock sample a shock wave inside the rock will be induced. To obtain a planar collision a system after DeCarli (1981) was used. Because all kinds of explosives may be used as described above a maximum collision velocity up to 4 km/s is reachable. To avoid disturbing shocks inside the sample from the outer parts of the sample holder all parts are additionally encapsulated in plywood, the so called 'dynamic impedance correction' (Schlothauer 2016). It is visible that the rock sample is completely recoverable, if the recovery follows the shot after short time, after disassembly the crack

propagation through the sample is still visible *in situ*.

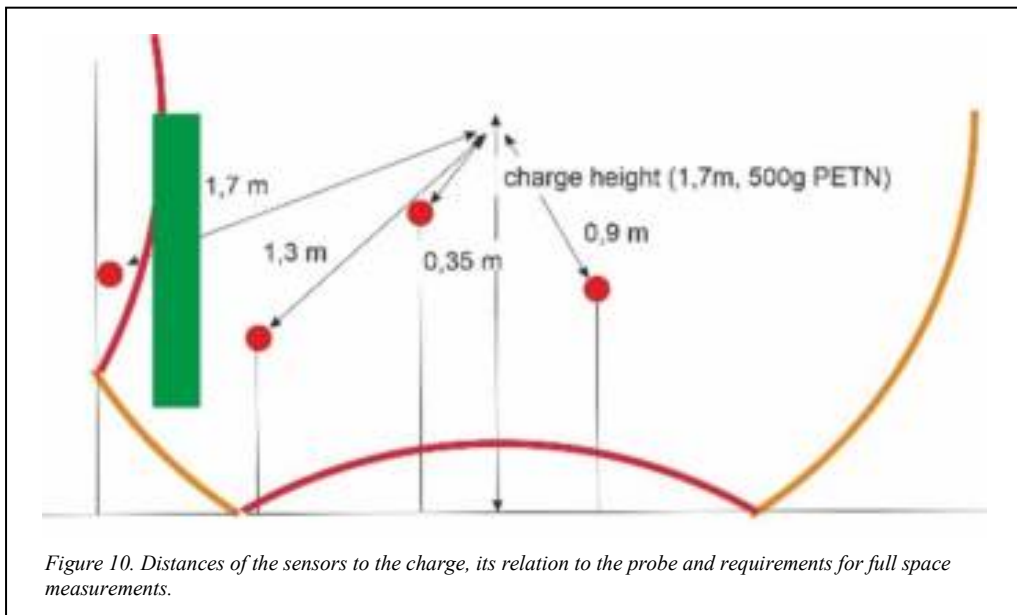
It is also possible to work with cylindrical samples like drill cores, because cylindrical charges were already tested frequently. (Figure 8). Direct tests of rocks in contact with explosives are also possible (Grady 1980).

2.3 *Dynamic tests of composite materials with passive gauges and velocity measurements*

For materials usable for security applications practical parameters like deformation and reduction of the strength of the shock wave in air by the material are very important to calculate its possibilities for protection. To measure the degree of deformation not in all cases high-cost time resolved velocity measurements like Velocity Interferometer System for Any Reflector (VISAR), Photon Doppler velocimetry (PDV) or high-speed cameras are strictly required, in a few cases also passive gauges or cantilevers (van Netten 1997) may be used. This concerns also direct surroundings of the charge (Kisters 2016).



Figure 9. Sample holder for stability tests with passive gauges (material: lead) with rails for moving heavy samples in the lab itself (see also Figure 3).



They consist of a material with low strength against plastic deformation, e.g. lead or tin-lead alloys. These gauges in the example are located behind the back of the sample perpendicular to its surface and after elastic deformation of the workpiece the reduction of its length stands for the degree of deformation of it during shock (Figure 9).

To ensure the validity of the results obtained in security tests only plastified PETN is in use for this kind of work. To determine the velocities of the workpiece in detail measurements with shortening pins (Dynasen Inc.) are possible.

2.4 Air pressure and velocity measurements - experimental setup

This type of experiment is of interest for the testing of materials usable for security applications (e.g. shock damping, protection assemblages). To measure the attenuation of the shock, a special arrangement in the SWL was constructed (Figures 10 & 11).

The piezoelectric air pressure sensors were mounted on steel holders with fixed distances related to the position of the charge and the ground of the lab. The measurements were performed in the so called 'full space regime' (Sochet 2018). In this case it is required, that the shock wave reaches the sensors before any reflections from the ground or other obstacles will take place (Figure 10).

This is given by a sufficient height-distance ratio between sensors, ground and charge position.

The first pressure measurement is given by the pencil probe (hydrostatic pressure alone), this marks simultaneously the starting point of the velocity measurement. The first sensor for the complete pressure is located between pencil probe and the (imaginary) sample surface. Because the shock is assumed as spherical and the height of the sensor is large enough to exclude reflections from the ground, Sensor 2 marks the distance between charge and sample surface despite the fact that it is not located directly in front of the sample (Figure 11).

Sensor 3 is located behind the sample at the wall of the lab with a distance between the rear of the sample and sensor surface to avoid a direct collision between metal coverage of the sample and sensor. However, this distance must be small enough to exclude the measurement of attenuated shocks by the rim of the sample (Figure 12).

The explosive used in all cases was plastified PETN as the detonation of PETN is well described in literature (Kubota 2010, Ree 1984). The charges were spherical, the charge masses varied between 500 and 2000 g. The distances between surface of the charge and sensor were measured using a laser rangefinder. The signals from the sensors were recorded with the TraNET FE 408 DP (Figure 4).

The sample itself consists of small beads which were made on glass foam at the Insitut für Glas und Glastechnologie (TUBAF), wrapped in glass fibre fabric, manufactured at the Saechsische Textilinstitut (STFI) Chemnitz.

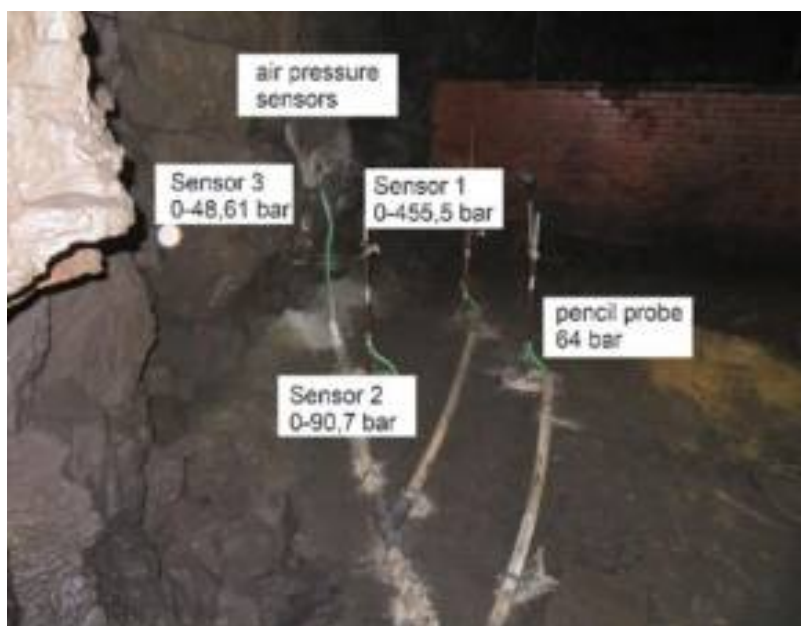


Figure 11. Positioning and pressure range of the used air pressure sensors in the SWL with the measurement cables in tubes mounted on the ground.



Figure 12. The complete cage mounted in front of Sensor 3.

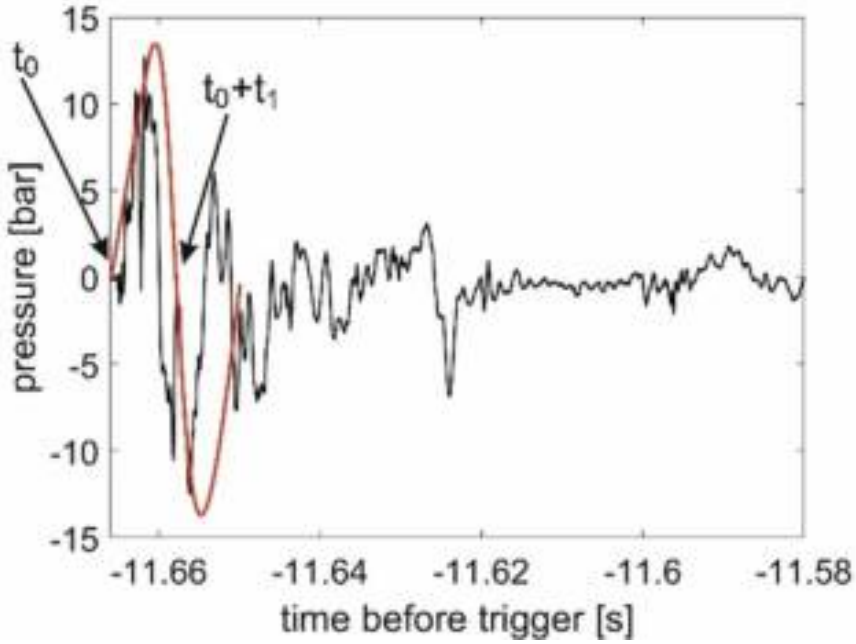


Figure 13. Parameters of Equation 1 in relation to an experimental measured curve, the red line shows the schematic positioning of the first sine function.

2.5 Air pressure measurements- comments from the physical point of view.

It is not the main aim of this paper to describe the damping effects of different materials in detail, rather, some experimental results will be shown which have emerged during the practical work and which are often not taken into account by the usual theories. (e.g. Sochet 2018). An important parameter for possible damage of materials is the impulse I , which is calculable with:

$$I = \int_{t_0}^{t_0+t_1} p(t) dt \quad (1)$$

The complete first impulse I is defined as the first sine function, as visible in the $p=f(t)$ -plot (Figure 13) and this first pulse is commonly the criterion for possible damages. This impulse marks the value for the complete energy input into the workpiece in dependence of time (unit $N*s$) as the summarised integral of pressure over a defined time.

It is important to take the impulse from Equation 1 as a vectorised parameter into account, the first part is positive (damage by compression), the second part is negative (damage by extension).

Despite the fact that commonly the first pulse was used to calculate the damage potential from a detonating charge (Figure 14) it may be helpful to take a look to the complete measured signals (Figure 15).

It is clearly visible that the second pulse is much larger and will contain a larger damage

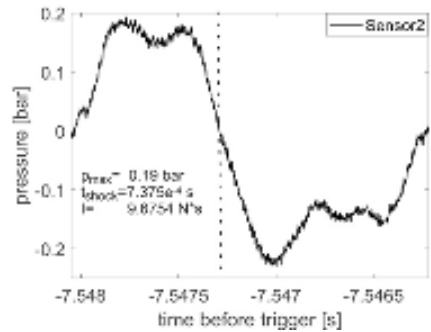


Figure 14. First positive pulse from a detonating charge (PETN 500g, plastified, spherical charge, distance 1.391 m) with parameters maximum pressure, pulse duration and pulse energy.

potential too compared with the first positive pulse from Figure 14, as shown in Figure 16 and in this case a reduction of the investigation to the first pulse alone to estimate the damage potential is not sufficient.

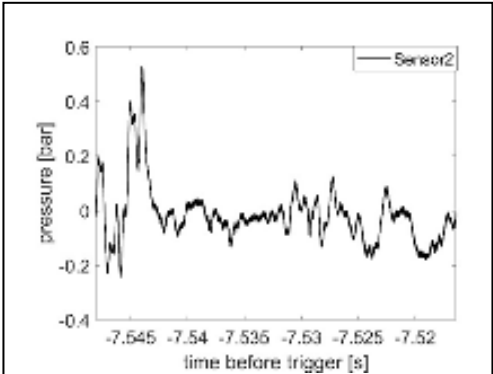


Figure 15. Air pressure measurement from Figure 14 with expanded time axis.

The maximum of the first pulse and the area under its time dependent curve (Equation 1) and additionally the equivalent parameters of this second pulse depends on the surroundings of the sensor and additionally on shock interferences in air and because its calculations are very difficult an experimental verification of this parameter is required. Possibly gaseous detonation products also play a role.

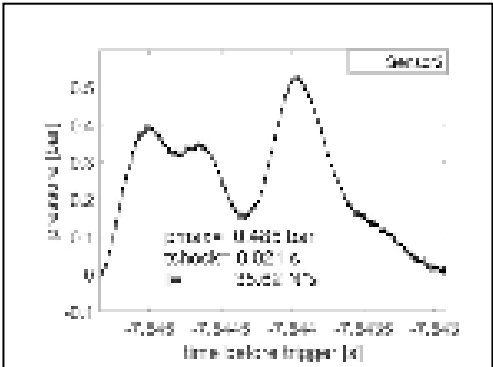


Figure 16. Pressure and impulse parameters of the second positive pulse, experimental parameters as given in Figure 14.

An additional problem is the first positive pulse itself. It is mathematically defined as the area between the beginning of the positive slope of the curve to its first intersection with the x-axis, where the pressure as function of time at first tends to

zero and becomes later negative (Equation 1). This is given in figure 17.

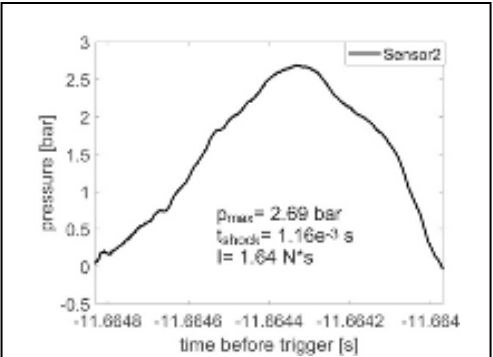


Figure 17. First positive pulse with its data, explosive: PETN, plastified, charge mass: 2000 g, sensor distance 1.136 m.

This simple physical picture becomes more complicated if the time will be changed again (Figure 18).

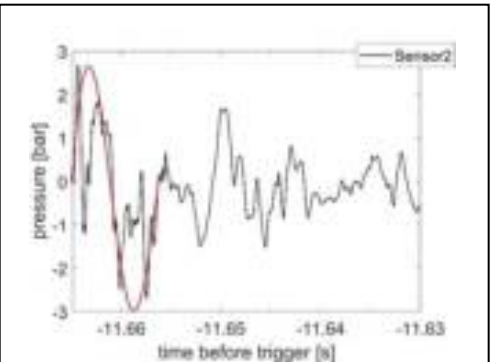


Figure 18. Complete first pulse with marked first sine function. Experimental parameters as given in Figure. 17.

Now it is visible that the first positive pulse is divided into two parts with a small negative section between them (Figure 19) and the first sine function must cover both parts of them. The first part of this pulse is the shock wave itself and the second part is defined by the energy of gaseous detonation products. Because they are often neglected in calculations of shock intensities, in many case its influence for damage processes are in these cases underestimated. Its slightly larger parameters compared with the first part of the impulse are also given in Figure 19. This effect may increase if commercial mining explosives with its higher specific swath volume at the same

charge mass compared with plastified explosives were used.

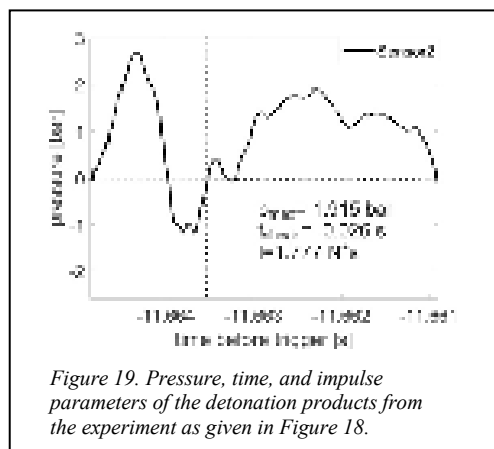


Figure 19. Pressure, time, and impulse parameters of the detonation products from the experiment as given in Figure 18.

3 CONCLUSIONS

The shock wave lab at the TU Bergakademie Freiberg has great potential for research and material testing and it is open for many kinds of civil research. The focus is on material behaviour under intense dynamic loading, air pressure measurements, failure analysis, tests of natural rocks and concrete (including steel-reinforced concrete) and more. Because the Saxonian Mining Office is the single regulatory authority, and the blasting work is based on the proposed working plan alone, the experimental work is possible without any additional authorisation. In the case of new experiment requirements, it is possible to add them quickly. Additionally, staff at the mine and the university workshop are able to build and supply to laboratory users additional tools and specific sample holder requirements, massively simplifying the needs of researchers.

4 ACKNOWLEDGEMENTS

The build-up of the laboratory itself until its final permission was founded by the Dr. Erich-Krüger-Foundation. The air pressure measurements and the required equipment was founded by the IGF-project 21729 BR.

REFERENCES

Ahrens, T.J. & Johnson, M.L. 1995. Shock wave data for rocks. In: T.J. Ahrens (Hg.): *Mineral Physics & Crystallography: A Handbook of Physical Constants*. Washington D.C., pp. 35–44.

Ahrens, T.J. & Johnson, M.L. 1995. Shock wave data for minerals. In: T.J. Ahrens (Hg.): *Mineral Physics & Crystallography: A Handbook of Physical Constants*. Washington D.C., pp. 143–185.

Carpenter, S.H. & Wittman, R.H. 1975. Explosion welding. In: *Annual Review of Materials Science* 5, pp. 177–199.

DeCarli, P.S. & Meyers, M.A. 1981. Design of uniaxial strain shock recovery experiments. In M.A. Meyers und L.E. Murr (eds.): *Shock Waves and High Strain-Rate Phenomena in Metals: Concepts and Applications*. pp. 341–373 New York: Plenum Press.

Goel, M.D. 2015. Blast: characteristics, loading and computation - an overview. In: Vasant Matsagar (Hg.): *Advances in Structural Engineering*. New Delhi: Springer India, pp. 417–434.

Grady, D.E., Kipp, M.E. & Smith, C.S. 1980. Explosive fracture studies on oil shale. In: *Society of Petroleum Engineers Journal* 20 (05), pp. 349–356.

Jug, J., Strelec, S., Gazdek, M. & Kavur, B. 2017. Fragment size distribution of blasted rock mass. In: *IOP Conf. Ser.: Earth Environ. Sci.* 95, pp. 42013.

Kisters, T., Kuder, J. & Nau, S. 2016. Autonomous gauge for blast impulse determination close to explosive charges. In: *Shock Waves* 26 (2), pp. 117–127.

Klapötke, T.M. 2007. High energy density materials. Berlin, Heidelberg: Springer (*Structure and bonding*, 125).

Klapötke, T.M. 2009. Chemie der hochoergetischen Materialien. 1. Auflage. Berlin: de Gruyter.

Kubota, S., Saburi, T., Ogata, Y. & Nagayama, K. 2010. Numerical simulations of detonation phenomena in PETN by systematic equation of state for detonation products. In: *Sci. Tech. Energetic Materials* 71 (2), pp. 44–50.

Mehdi, E., Roya, A., Tahereh, B., MirKamali, E.S. & Behnam, F. 2018. The energetic check of derived boron nitride nano- cage with (HMX) in different conditions of temperature by

- density functional theory method. In: *International Journal of New Chemistry* 5 (1).
- Lysak, V.I., Kuz'min, S.V. & Bagmutov, V.P. 2007. Physical foundations and fields of practical application of explosive welding of metals. In: *Russ. J. Non-ferrous Metals* 48 (6), pp. 461–471.
- Schlothauer, T. 2016. *Aufbau des Schockwellenlabors im Lehr- und Forschungsbergwerk 'Reiche Zeche' der TU Bergakademie Freiberg und die Entwicklung von dynamischen Höchstdruck-synthesemethoden*. Dissertation. Dissertation. Freiberg: TU Bergakademie Freiberg, Institut für Mineralogie.
- Ree, F.H. 1984. A statistical mechanical theory of chemically reacting multiphase mixtures: application to the detonation properties of PETN. In: *J. Chem. Phys.* 81 (3), pp. 1251–1263.
- Smith, G.D. & Bharadwaj, R.K. 1999. Quantum chemistry based force field for simulations of HMX. In: *J. Phys. Chem. B* 103 (18), pp. 3570–3575.
- Sochet, I. 2018. Chapter 1: Introduction. In: Isabelle Sochet (Hg.): *Blast effects. Physical properties of shock waves*. Cham: Springer International Publishing, pp. 1–15.
- Tadini, P., Gault, K. & Sochet, I. 2018. Chapter 4: Shockwave overpressure measurement: comparison between two piezoelectric sensor materials. In: Isabelle Sochet (Hg.): *Blast Effects. Physical Properties of Shock Waves*. Cham: Springer International Publishing, pp. 57–72.
- van Netten, A.A. & Dewey, J.M. 1997. A study of blast wave loading on cantilevers. In: *Shock Waves* 7 (3), pp. 175–190.

Pioneering transformation of explosive practices in Indonesia: overcoming geological heterogeneity with dual salt emulsion technology

M. Arbi & H. Prasetya

Kemitraan MNK BME (KMB), Jakarta, Indonesia

ABSTRACT: KMB, as one of the blasting service companies in Indonesia, introduced Dual Salt Emulsion (DSE) products as a transformative solution to optimise the use of ammonium nitrate (AN). DSE explosives made with ammonium nitrate (AN) and calcium nitrate (CN) overcome the hurdles of insufficient domestic AN production and strict AN importation regulation to meet the total explosives demand for the mining industry. The introduction of CN to DSE formulations is a breakthrough that reduces dependence on a limited AN supply and thus loosens up the supply chain and reduces total cost. The implementation of DSE was recently demonstrated at one of the KMB sites in Central Kalimantan and the use of CN has been legally approved by the Indonesian Ministry of Energy and Mineral Resources (ESDM). The use of DSE provided the mine site with improvements to digging time and fragmentation, plus environmental benefits. The use of DSE was fully carried out in accordance with regulatory standards and seamlessly integrated throughout mining operations.

1 INTRODUCTION

In the operational process of blasting, ammonium nitrate (AN) remains a fundamental component for the formulation of bulk explosives. However, in recent years, AN supply in Indonesia has become increasingly limited due to several factors such as import limitations, market fluctuations and insufficient domestic AN production. This can disrupt mining operations, especially during peak periods. These challenges are one of the reasons for the need for alternative solutions to avoid price volatility and logistical difficulties to maintain performance while supporting sustainability goals.

Dual Salt Emulsion (DSE) is an innovation that is expected to break into the market in Indonesia by combining calcium nitrate (CN) into an emulsion matrix containing ammonium nitrate

(AN) and is expected to overcome issues of AN availability. By optimising AN-based emulsions using CN, DSE not only reduces dependence on limited AN supply but also supports operational cost efficiency and ease of transportation if DSE transportation is required at remote area sites that do not have a plant. DSE has been successfully applied to one of the KMB sites in Central Kalimantan which shows its effectiveness in increasing efficiency and technical capabilities in blasting activities in actual field conditions.

DSE has been demonstrated to support consistent blasting performance and can be adjusted to meet user needs. DSE's resistance to the crystallisation process is one of its advantages compared to single salt emulsion formulations. Stability is one of the advantages of DSE which can prevent separation between the oxidiser phase

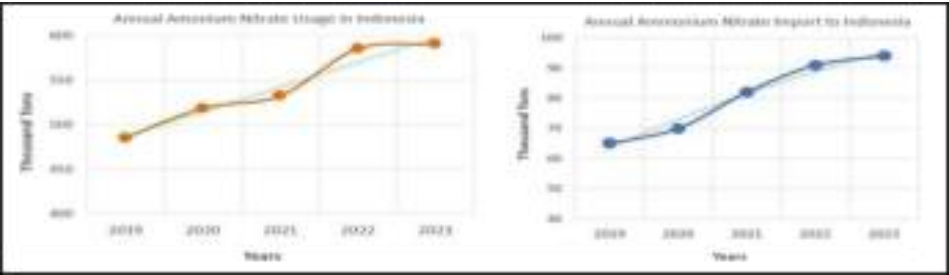


Figure 1. Trends in ammonium nitrate (AN) consumption and import volume in Indonesia from 2019 to 2023 (BPS 2023).

and the fuel phase during the transportation stage, long-term storage, and can withstand mechanical stress such as during the pumping process. The sustainability of DSE can be utilised for the shelf life of emulsion, especially in mining environments that do not allow for direct production and are in locations that are difficult to reach logistically. Due to the stability of the DSE formula, blasts may be slept longer so that production levels can be achieved. In addition, field observations show that DSE tends to produce lower levels of post-explosion NOx smoke compared to single salt emulsion, which contributes to better environmental and occupational safety results (Scovira 2024).

Data on the use and import of Ammonium Nitrate (AN) in Indonesia from 2019 to 2023 shows a significant increase. As seen in Figure 1, consumption increased from around 500,000 tons in 2019 to more than 550,000 tons in 2023. The increase in mining industry activity and the increasing need for explosives are reflected in this increase. AN import also increased from around 70,000 tons in 2019 to almost 100,000 tons in

2023, indicating Indonesia's increasing dependence on AN supply from abroad.

From June 2024 to November 2024 there were fluctuations in coal prices and these are shown in Figure 2. Fluctuations in coal prices required the company to be able to face market fluctuations, so efficiency was needed to achieve effectiveness in the blasting process. The use of CN as a raw ingredient in explosives presented itself as one of the steps to maintain stability and effectiveness when there are fluctuations in coal prices.

The use of CN at one of the KMB locations in Central Kalimantan is a strategic step to support operational continuity, cost efficiency, and compliance with existing regulations. With the Company's decision to use CN in DSE, the Company can optimise resource usage while reducing the risk of fluctuations in AN supply. Before this time, the use of CN in DSE was not approved in Indonesia, and this encouraged KMB to conduct evaluations and technical studies showing results and benefits that are in line with user expectations to support CN approval in Indonesia.

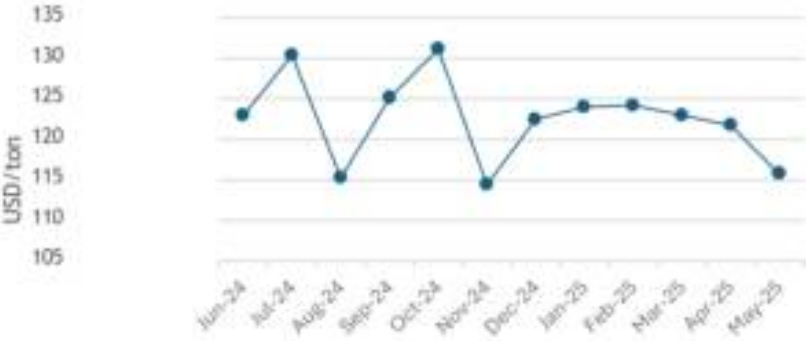


Figure 2. Indonesia reference coal price chart (ESDM, 2025).

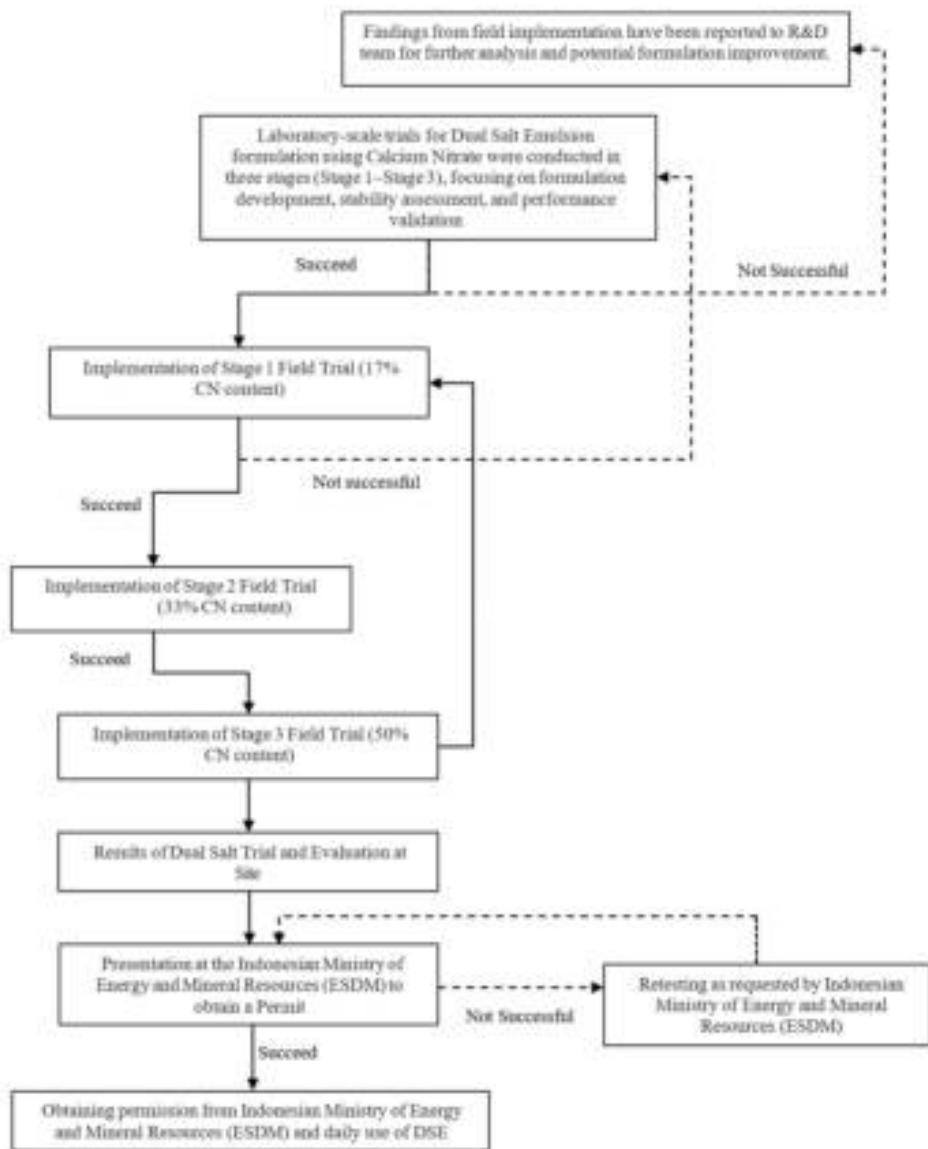


Figure 3. DSE research diagram.

2 METHODOLOGY

In the implementation testing of Dual Salt Emulsion (DSE) at the site, various observations were conducted to ensure the accuracy and relevance of the data obtained. These observations included a comprehensive review of Ammonium Nitrate (AN) usage in Indonesia, encompassing

import volumes and applicable regulations, to assess how DSE can reduce dependence on AN. Additionally, site maps, explosive specifications, types of explosives used, blasting geometry, and other supporting data were collected.

The study on DSE implementation involved both primary and secondary data collection to provide a comprehensive overview of the

effectiveness of DSE usage in the mining area. The performance of DSE explosives was compared with conventional explosives. The combination of these data sources enabled a thorough and in-depth analysis to ensure that DSE implementation complies with sound mining engineering principles and relevant regulations.

In the DSE testing, data was collected in two main stages, namely lab-scale testing and field-testing. In lab testing, a series of experiments was carried out to test and ask whether the formula applied for DSE is effective. This stage mainly looked at how stable DSE is over a certain period. In the field-testing stage, which involved full-scale blasting to see the performance of DSE to realise rock breakage with the desired fragmentation and excavation time. During this stage, some data was collected, such as rock fragmentation, excavation time, and detonation velocity (VOD) values, to assess how effective the explosives work. See Figure 3.

3 REGULATORY FRAMEWORK

Technical requirements related to blasting are regulated in Indonesia in the Regulation of the Minister of Energy and Mineral Resources No. 1827 K/30/MEM/2018 concerning Good Mining Practices. In this regulation, the requirements cover aspects related to Occupational Health and Safety, permits and approvals, material descriptions and testing protocols are regulated so that the blasting and mining processes can be carried out properly. Regarding the use of CN in Indonesia, based on current national regulations, CN is not classified as an explosive. Some legal references that support this are: Ministry of

Finance Regulation No. 26/PMK.010/2022, which assigns Harmonised System (HS) codes: CN is listed under HS code 31026000 (fertiliser), while AN falls under 31023000 (commercial explosives).

- Ministry of Trade Regulation No. 25 of 2022, which revises import policies.
- Ministry of Defence Regulation No. 5 of 2016, concerning the development of the explosives industry.
- Ministerial Decree of Energy and Mineral Resources No. 1827 K/30/MEM/2018, related to technical standards in mining.
- National Police Regulation No. 17 of 2017, covering licensing, security, and control of commercial explosives.

In addition, in Presidential Decree No. 125 of 1999, CN does not meet the definition of explosives contained in the regulation because it does not have the characteristics of an explosive in its pure form. Regulation of the Chief of Police No. 2 of 2008, which regulates the supervision of commercial explosives, does not list CN among oxidising agents such as nitric acid, sulfuric acid, or ammonium nitrate. Based on these regulations, CN can be treated as a standard chemical that is soluble in water and is not classified as an explosive and hazardous or toxic waste. As an illustration, Table 1 shows a summary of regulations governing the use of CN from various countries and organisations.

The use of CN has been regulated in various countries as specific guidelines for each country. Although CN is generally not classified as an explosive in general, its oxidising properties are a concern so that care must be taken when handling it. The risk increases when CN encounters flammable materials or is exposed to high heat. To

Table 1. Regulations on the use of CN in various countries.

Country/Agency	Regulation	Regulation status on calcium nitrate (CN)
Canada	Explosives Regulation, 2013 (SOR/2013-211)	Listed as a restricted component
European Union	Regulation (EU) 2019/1148	Explosive precursor subject to reporting
United Kingdom	The Control of Explosives Precursors and Poisons Regulation 2023	Chemical subject to reporting
Hong Kong	UN No. 1454 Dangerous Goods	Classified as dangerous goods under specific conditions
India	Explosives Act, 1884 and Ammonium Nitrate Rules, 2012	Not classified as explosive
Indonesia	National Police Regulation No. 2 of 2008	Not listed as explosive material
ECHA (EU)	CLP (EC) No 1272/2008	Not classified as explosive
NITE (Japan)	UNRTDG	Oxidizer but not explosive
OSHA (USA)	Hazardous Materials Standard 1910.109	Not classified as explosive



Figure 4. Dual salt emulsion manufacturing process on a lab-scale.

prevent potential hazards, safe handling and proper storage in accordance with safety standards.

4 IMPLEMENTATION OF TRIALS AND DATA COLLECTION

4.1 DSE Formula 1 - Formula 3 lab-scale testing

Before conducting field tests with DSE, laboratory tests were conducted to ensure the formulation met the required performance standards, in terms of crystallisation point and density. Crystal points needed to be checked to help determine the thermal stability of the formulation during storage and handling in the field. Density was analysed to ensure the emulsion was within the ideal range because density is a picture of the overall performance of the emulsion. All tests became a strong technical foundation to verify the quality, stability and ability of the emulsion to be tested and later tested on a larger scale in the field. See Figure 4.

Table 2. Lab-scale dual salt emulsion test results.

Formula	CN Content (%)
1	17
2	33
3	50

The laboratory tests mainly focused on two critical parameters: crystallisation behaviour and

the density after gassing of the emulsion formulations. Table 2 provides a summary of the laboratory-scale test results for three different formulations (Formula 1 to Formula 3), highlighting the differences in crystallisation points and measured densities across each composition.

Monitoring of the emulsion was carried out for seven days to understand the stability of DSE over time. Every day DSE was monitored to check the consistency and uniformity of the product. The purpose of the check was to see how the physical properties of the emulsion changed over time to identify the stability of the emulsion. See Figure 5.

The results from the seven-day observation are summarised in Table 3, providing important data to evaluate the long-term stability and suitability of the DSE for use in mining operations.

Evaluation was carried out in the lab-scale testing process, and it was done to obtain evidence and develop strong confidence to be able to carry out the next process, namely testing on a field-scale. After it was found that the testing on a lab-scale met the requirements, then the field stage testing was carried out in three stages according to the CN formula content.

4.2 DSE Staging Full Scale (Field) Testing

Field testing was conducted on the three formulas that had been lab tested. This test was done to ensure that any changes to the formula produced results that met the quality standards. In addition,

Table 3. Visual check of three types of formulas used.

Date	CN				Stability Test			
	%	D1	D2	D3	D4	D5	D6	D7
27(?) July 2024	17	✓	✓	✓	✓	✓	✓	✓
10 July 2024	33	✓	✓	✓	✓	✓	✓	✓
15 July 2024	50	✓	✓	✓	✓	✓	✓	✓

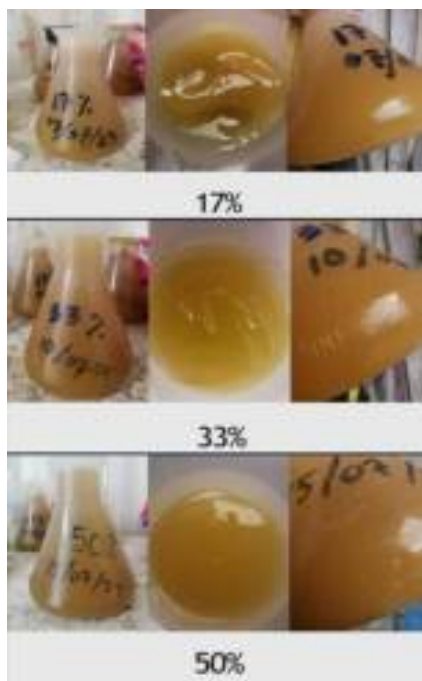


Figure 5. DSE stability check on Formula 1 - Formula 3.

this testing also aimed to ensure that there are no unwanted deviations, such as the appearance of excessive fumes or smoke. By conducting repeated tests on each formula, the consistency of each formula in real field conditions was verified. The target for blasting fragmentation results is ≤ 50 cm with a percentage of 80%.

Stage 1 of Dual Salt Emulsion (DSE) testing served as the initial stage of DSE testing to

determine the performance of DSE in field-scale testing. Detonation Velocity (VOD) measurements were carried out to assess the performance of the explosives. As shown in Figure 6 the VOD recorded for Stage 1 was 5,397 m/s, a value considered adequate to effectively break rocks on site. VOD measurements were carried out at each stage to determine the performance of the explosives. In Stage 2 of DSE testing, nine blasting trials were carried out. In Stage 2, Detonation Velocity (VOD) measurements were carried out and are shown in Figure 7. The VOD value obtained was 5,206 m/s, which is sufficient to effectively break rocks on site.

To further ensure the overall performance of the explosives at the test site, digging time was conducted at Stage 1 and Stage 2. This data provided an important measure of the target productivity of the excavation and helped evaluate the effectiveness of the blasting. Digging time was taken during the testing of Stage 1 and Stage 2, and then from the overall results it was stated that Stage 1 and Stage 2 were positive and could be carried out to the next stage (Stage 2 testing was carried out after Stage 1 testing was declared positive and carried out in stages). In Stage 1 using 17% CN, the average time was 10.67 seconds (from 3 blasting events), and in Stage 2 using 33% CN, the average digging time was 10.64 seconds (from 9 blasting events). From Stage 1 and Stage 2, the digging time was still below 11 seconds so that it met the established standards.

In both stages, DSE worked consistently with the established standards and did not show significant variations in blasting results, including blasting effects and energy distribution. This indicated that the DSE formulation was stable and

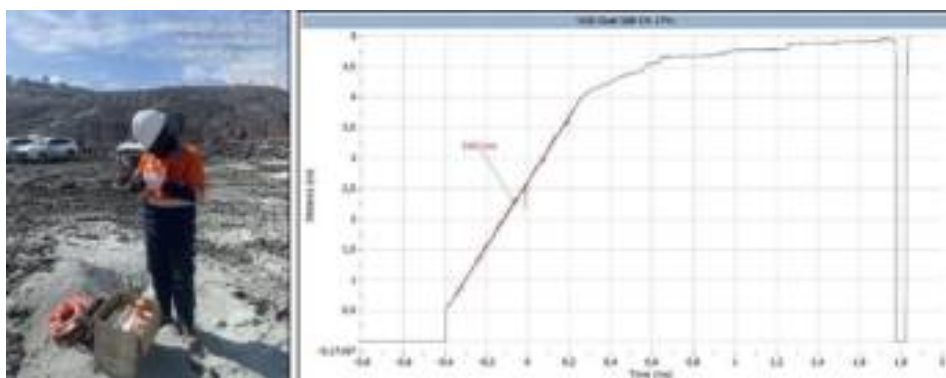


Figure 6. VOD data capture process at Stage 1.

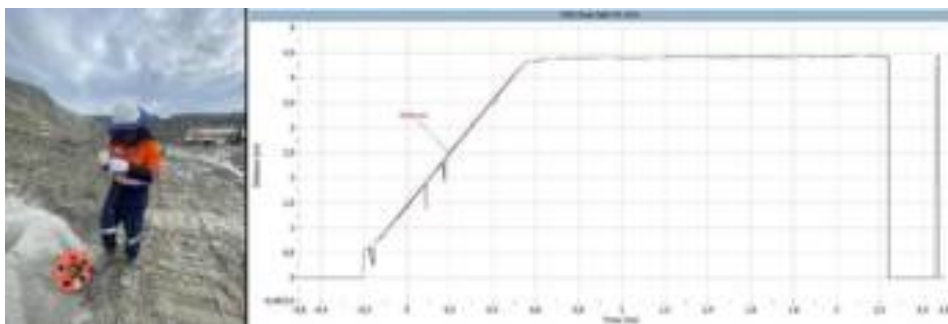


Figure 7. VOD data capture process at Stage 2.

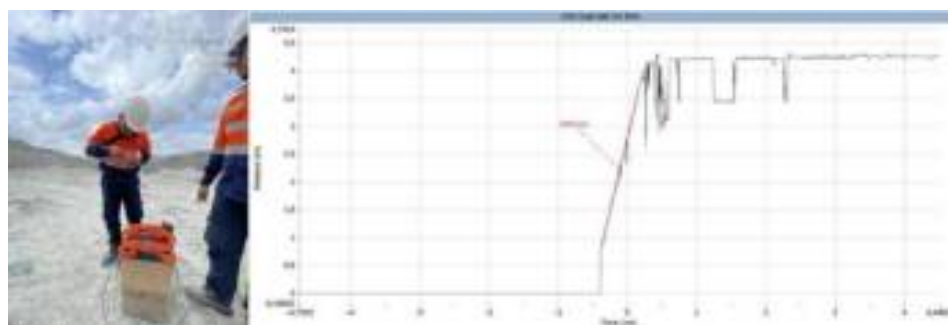


Figure 8. VOD Data Capture Process at Stage 3.

reliable in performing optimally. The next step taken to verify the blasting results was to analyse the rock fragmentation to assess the distribution of the fragments produced. In Stage 1, the average fragmentation was 89.97% (from 3 blasting events), while in Stage 2, the average fragmentation was 94.87%. In Stage 1 and Stage 2, the average fragmentation reached the target fragmentation of ≤ 50 cm at least 80%.

When Stages 1 and 2 criteria were met, it proceeded to Stage 3 with CN usage of 50%. Stage 3 production of the Dual Salt Emulsion (DSE) was conducted between July 16–17 and July 24–30, 2024. At the start of the Stage 3 trial, Velocity of Detonation (VOD) measurements were taken to assess the emulsion's performance, as illustrated in Figure 8 the recorded VOD was 5,245 m/s, which is more than sufficient to effectively break the material at the site. The target for blasting fragmentation was set at ≤ 50 cm, with at least 80% of the fragments meeting this specification.

Testing in Stage 3 was done by checking

digging time and fragmentation as one of the requirements was to check the performance of explosives. For eight blasts, the average digging time was 10.58 seconds, and the average fragmentation was 91.68%, with a standard digging time of 11 seconds and fragmentation at 80%, the stage 3 test for DSE meets these standards as illustrated in Figure 9.

When all DSE Stage 1 to Stage 3 tests were completed, the results were reviewed and presented to the customer where the KMB site conducted DSE testing. After that, a submission to the Indonesian Ministry of Energy and Mineral Resources (ESDM) was made to obtain permission for use of CN in DSE for blasting activities.

ESDM requested repeat testing for DSE with a 50% CN formula to increase confidence in the use of DSE as an explosive. A repeat of DSE testing was carried out and a presentation made again, ESDM officially granted permission to use CN as DSE as an explosive.

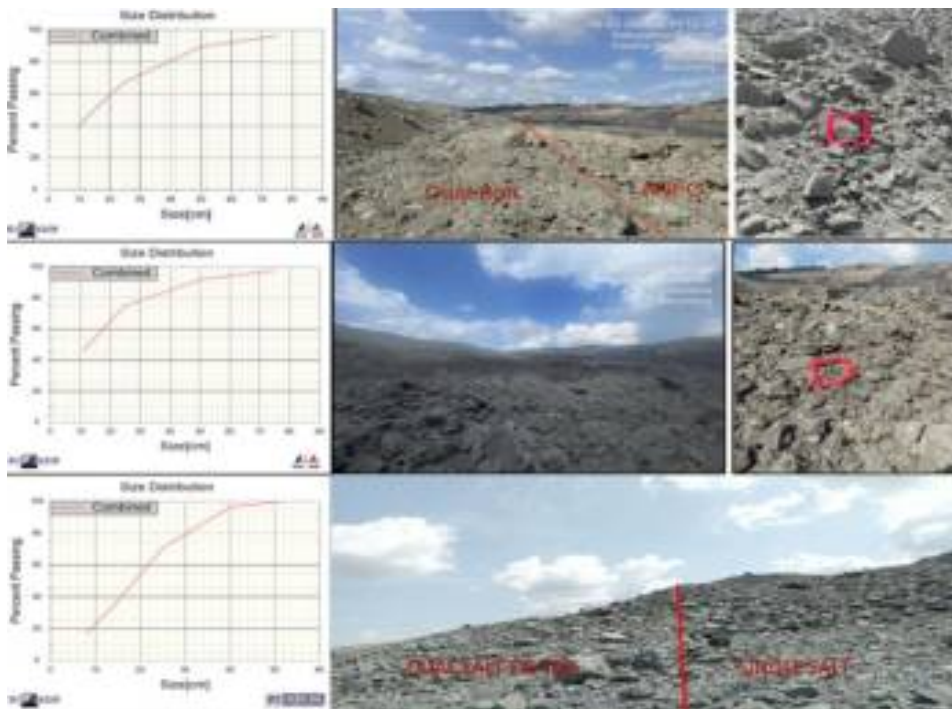


Figure 9. Fragment size distribution in some tests in Stage 3.

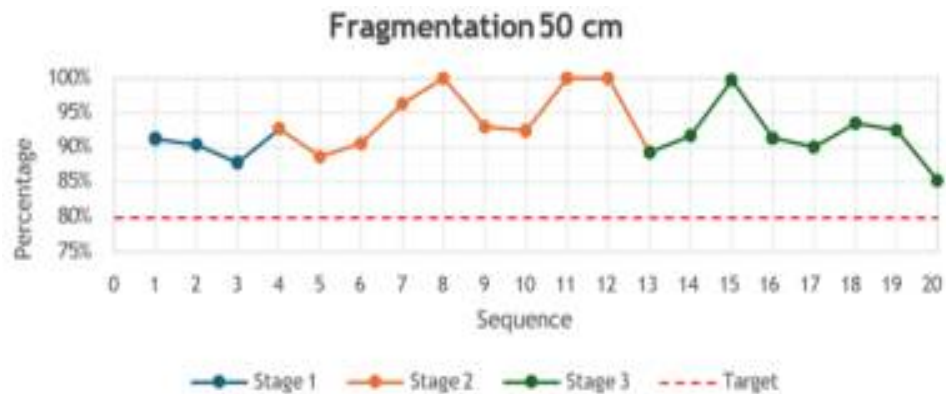


Figure 10. Summary of actual fragmentation results vs target fragmentation 50 cm $\leq$ 80%.

5 DATA ANALYSIS

5.1 Summary of trial dual salt emulsion test results Stage 1 – Stage 3

In this section, a comprehensive analysis of DSE performance is presented based on the results of laboratory testing, Stage 1 and Stage 3. The

parameters focused on are the average digging time, detonation velocity (VoD), fragmentation results and overall visual observation of the blasting results. Each parameter is examined to provide a detailed description and explanation to the performance of the emulsion in its actual field application.

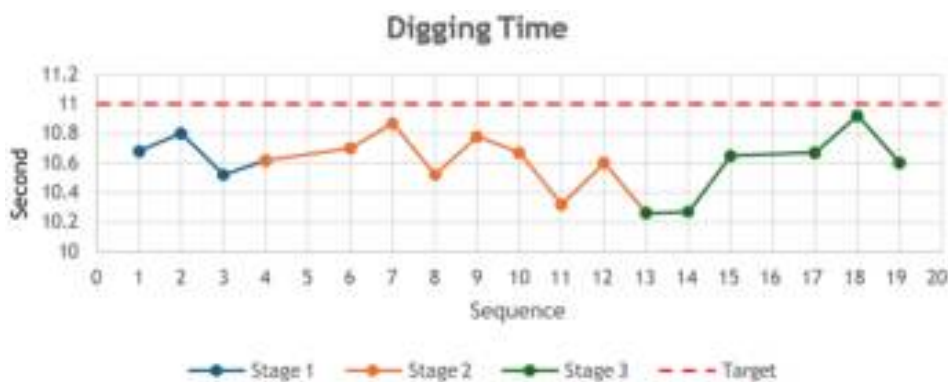


Figure 11. Summary digging time actual vs target 11 second.

Table 4. Summary of trial dual salt emulsion test results Stage 1 – Stage 3.

Trial	Date	CN	Lithology	Holes	PF (kg/bcm)	Fragmentation ≤ 50cm	Fumes	VOD (m/s)	Digging Time (s)
1	08-Jul-24	17%	Metasediment	66	0.81	91.33%	No	5,397	10.68
2	09-Jul-24	17%	Metasediment	31	0.81	90.42%	No	-	10.8
3	10-Jul-24	17%	Sandstone	72	0.64	87.87%	No	-	10.52
4	11-Jul-24	33%	Sandstone	43	0.81	92.69%	No	5,206	10.62
5	12-Jul-24	33%	Sandstone	36	0.99	88.69%	No	-	TBD
6	13-Jul-24	33%	Sandstone	86	0.78	90.64%	No	-	10.7
7	18-Jul-24	33%	Quarry	92	0.79	96.33%	No	-	10.87
8	20-Jul-24	33%	Sandstone	157	0.49	100.00%	No	-	10.52
9	20-Jul-24	33%	Sandstone	92	0.79	93.04%	No	-	10.78
10-1	22-Jul-24	33%	Sandstone	48	0.60	92.47%	No	5,065	10.67
10-2	23-Jul-24	33%	Sandstone	31	0.50	100.00%	No	-	10.32
11	31-Jul-24	33%	Quarry	70	0.91	100.00%	No	-	10.6
12	16-Jul-24	50%	Sandstone	123	0.67	89.27%	No	5,245	10.26
13	17-Jul-24	50%	Sandstone	73	0.80	91.70%	No	-	10.27
14	24-Jul-24	50%	Quarry	87	0.75	99.76%	No	-	10.65
15	25-Jul-24	50%	Sandstone-Siltstone	69	1.00	91.36%	No	-	TBD
16	26-Jul-24	50%	Sandstone-Siltstone	71	0.83	90.08%	No	-	10.67
17	28-Jul-24	50%	Sandstone	60	0.85	93.54%	No	-	10.7
18	29-Jul-24	50%	Sandstone	81	0.66	92.49%	No	-	10.92
19	30-Jul-24	50%	Sandstone	80	0.82	85.24%	No	-	10.6
Max				157	1.00	100.00%	-	5,397	10.92
Min				31	0.49	85.24%	-	5,065	10.26
Average				72	0.77	92.85%	-	5,228	10.62

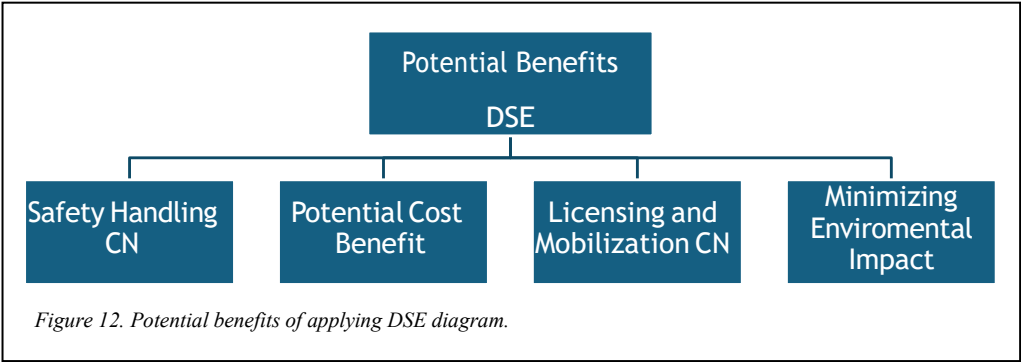


Table 4 above summarises key parameters obtained from the Dual Salt Emulsion (DSE) trials conducted from Stage 1 to Stage 3 at the site. The results are outlined as follows:

- Velocity of Detonation (VoD): Based on VoD data from Stage 1 to Stage 3 testing, the DSE VoD ranges from 5200 to 5400 m/s. In Stage 1 with 17% CN, the VoD reached 5397 m/s. Then in Stage 2 with 33% CN, the VoD reached 5206 m/s, indicating that the DSE performance was consistent. For Stage 3 with 50% CN. The VoD reached a value of 5245 m/s, which is lower than Stage 1 but still within the acceptable range when compared to single-salt emulsion. Based on the VoD values from the three stages, it shows the consistency of the use of CN to be applied to the blasting process.
- Digging Time Evaluation: Digging time assessment was carried out to assess how efficient explosives are in breaking rocks and can be excavated according to the standards that have been applied. For single salt emulsion, the standard digging time ranged from 10.27 to 10.51 seconds (L1) and 9.57 to 10.07 seconds (L2), with an overall average of 9.57 to 10.51 seconds. In Stage 1, using 17% calcium nitrate (CN), DSE showed improved performance, with L1 times ranging from 9.20 to 10.13 seconds, L2 from 9.51 to 10.18 seconds, and an average of 9.82 seconds, all within the expected range. Consistent results can also be observed in stages 2 and 3. so that all stages have good consistency in terms of digging time.
- Fragmentation Results: Rock fragmentation is another important indicator of performance. For particle sizes between 1 and 20 cm. single salt emulsion typically

achieves fragmentation rates ranging from 21% to 34%. In Stage 1 with 17% calcium nitrate (CN). fragmentation ranged between 25% and 34%. Both Stage 2 (33% CN) and Stage 3 (50% CN) also met the expected fragmentation ranges, varying from 15% to 47% depending on particle size. These results align with single salt emulsion standards, confirming that the Dual Salt formulation delivered the desired fragmentation performance in the field.

Overall, the results across all testing stages confirm that the Dual Salt Emulsion (DSE) delivers consistent performance and meets the expected standards. Significant improvements were noted in key parameters such as Velocity of Detonation (VoD) and blast fragmentation. All findings are in line with single salt emulsion benchmarks, demonstrating that the new Dual Salt Emulsion formulation was reliable and effective for practical field use.

5.2 Potential benefits of applying dual salt emulsion (DSE)

5.2.1 Safety handling of calcium nitrate (CN)

The DSE formulation, in addition to providing economic benefits, also provides significant benefits in operational safety. When viewed from the dissolution temperature. where the single salt emulsion method requires a dissolution temperature of up to 72°C, the DSE only requires a temperature of around 65°C. The lower boiling point of CN not only reduces energy requirements but also improves work safety in the emulsion plant. Lower temperature handling can reduce dangerous risks such as hazardous thermal degradation of raw materials. Relatively low temperatures also reduce the operational pressure of equipment, thereby reducing the risk of leakage or failure in operations that can cause accidents.

5.2.2 Potential cost benefit

The use of DSE has great potential for cost benefits from the DSE manufacturing process compared to the manufacture of single salt emulsion. In testing Stage 1 to Stage 3, careful observation of the DSE manufacturing process obtained the potential for annual energy cost reduction of 30 to 40% to produce DSE compared to single salt emulsion. In addition, there is a cost difference between CN and AN. With a CN mixture of 50%, production costs can be reduced due to the difference in raw material prices, especially when there is fluctuation in AN price in the Indonesian market.

5.2.3 Licensing and mobilisation of calcium nitrate (CN)

The use of DSE also provides advantages in terms of licensing and logistics requirements. The use of CN, where CN is classified as fertiliser by many authorities, including Indonesia, provides simpler licensing compared to materials categorised as explosives. Its use is categorised as fertiliser, so it does not require complicated safety procedures. The licensing process can also be done faster and simpler and does not require strict supervision of explosives regulatory materials, thereby reducing the risk of delays in shipping and licensing. CN storage requirements are also simpler because they do not require high-security storage facilities. CN can be stored in standard warehouses with basic security measures so that it can reduce overall costs, eliminating the need for expensive special storage infrastructure. CN can be transported through general distribution channels without requiring special permits for dangerous explosives and does not require security escorts or strict transportation protocols.

5.2.4 Minimising environmental impact

Reduction of floc waste and reduction of Hot Water Boiler (HWP) emissions are a benefit of using DSE. The first thing is related to floc waste, where the waste is emulsion waste that includes all by-products produced during the production process that are not included in the final process. Among the most challenging types of waste is colloidal waste formed in the disperse phase, which can have a negative impact on the quality of the emulsion and reduce production efficiency. Residual AN in coating materials commonly called floc, can pose several safety hazards and can have an impact on the quality of the emulsion. Some single salt emulsion production can produce

floc residues that can cause defects in the emulsion structure, and additional costs for handling flocs that can have an impact on worker health risks. In contrast, DSE production produces zero floc, which marks a significant improvement in environmental and operational safety.

6 SUMMARY AND CONCLUSION

Compared to single-salt emulsion explosives, DSE shows increased stability, and this has an impact on increasing safety during storage and transportation. Several stages of testing were carried out to determine the most optimum CN ratio added to DSE, and it was found that the 50% ratio was the most optimum in terms of its superior performance in terms of lab and field testing. After going through several stages of testing and licensing through the Ministry of Energy and Mineral Resources, one of the KMB sites in Central Kalimantan received permission to use CN for the blasting process, and this became the breakthrough in the use of CN for explosives in Indonesia. The use of DSE can provide benefits for the company because it reduces dependence on the use of AN, especially during peak periods. These benefits can reduce the supply chain and support sustainable operations.

Convenience is also obtained in terms of logistics and licensing for CN procedures. In Indonesia, CN is classified as a fertiliser and not an explosive, simplifying the regulatory and transportation process. In addition, the use of DSE provides several benefits, namely in terms of handling safety, potential cost benefits, easier licensing and mobilisation of CN, and minimising environmental impacts. After going through several processes KMB succeeded in getting permission from ESDM to use DSE daily in the blasting process.

REFERENCES

- Badan Pusat Statistik 2024. Ammonium nitrate by country of origin. 2017–2023.
- Canada 2013. Explosives Regulations. 2013. SOR/2013-211. Registered November 27, 2013, under the Explosives Act.
- Scovira, D.S. & Gustavsson, T. 2024. Breaking the nitrate-based explosives greenhouse - the dawn of production scale hydrogen peroxide emulsion (HPE) for industrial blasting. International Society of Explosives Engineers (ISEE).

- European Parliament & Council of the European Union. (2019). Regulation (EU) 2019/1148 on the marketing and use of explosives precursors. amending Regulation (EC) No 1907/2006 and repealing Regulation (EU) No 98/2013. Official Journal of the European Union. L 186. 1–20.
- Government of India 1989. Manufacture, Storage and Import of Hazardous Chemicals Rules. 1989. Han, Z. (2016). Thermal stability studies of ammonium nitrate (Doctoral dissertation).
- Indonesian National Police 2008. Peraturan Kepala Kepolisian Negara Republik Indonesia Nomor 2 Tahun 2008 tentang Pengawasan dan Pengendalian Bahan Peledak Komersial.
- Indonesian National Police 2017. Peraturan Kepala Kepolisian Negara Republik Indonesia Nomor 17 Tahun 2017 tentang Perizinan. Pengamanan. Pengawasan. dan Pengendalian Bahan Peledak Komersial.
- Kaljuvee, T. Edro, E. & Kuusik. R. 2008. Influence of lime-containing additives on the thermal behaviour of ammonium nitrate. Journal of thermal analysis and calorimetry. 92(1). 215-221.
- Ministry of Defense. Republic of Indonesia 2016. Peraturan Menteri Pertahanan Republik Indonesia Nomor 5 Tahun 2016 tentang Pembinaan Industri Bahan Peledak Nasional.
- Ministry of Energy and Mineral Resources 2025. Coal benchmark price chart. Directorate General of Mineral and Coal.
- Ministry of Energy and Mineral Resources. Republic of Indonesia 2018. Keputusan Menteri Energi dan Sumber Daya Mineral Republik Indonesia Nomor 1827 K/30/MEM/2018 tentang Pedoman Pelaksanaan Kaidah Teknik Pertambangan yang Baik.
- Ministry of Finance. Republic of Indonesia 2022. Peraturan Menteri Keuangan Republik Indonesia Nomor 26/PMK.010/2022 tentang Penetapan Sistem Klasifikasi Barang dan Pembebanan Tarif Bea Masuk atas Barang Impor.
- Ministry of Trade. Republic of Indonesia 2022. Peraturan Menteri Perdagangan Nomor 25 Tahun 2022 tentang Perubahan atas Peraturan Menteri Perdagangan tentang Kebijakan dan Pengaturan Impor.
- National Institute of Technology and Evaluation (NITE). (n.d.). Chemical substance information: calcium nitrate (MITI No. 1-379). J-CHECK - Japan Existing Chemical Data Base. Retrieved May 27. 2025.
- President of the Republic of Indonesia 1999. Keputusan Presiden Republik Indonesia Nomor 125 Tahun 1999 tentang Bahan Peledak.
- United Kingdom 2023. The control of explosives precursors and poisons regulations 2023 (Statutory Instrument 2023 No. 63). Made on January 23. 2023; in force from October 1. 2023.

[RETURN TO CONTENTS PAGE](#)

Health, safety and environmental considerations in use of HPE explosives

T. Halme

Oy Forcit Ab, Finland

ABSTRACT: Hydrogen peroxide (HP) has been the main nitrogen-free alternative to ammonium nitrate (AN) in recent explosives development. This new application as a raw material for bulk explosives in mining and quarrying brings up new aspects regarding occupational health, safety and environment. Exposure of workers to HP is very different from stationary industrial process facilities. From an environmental point of view this application for HP is also something new. HP is directly toxic to aquatic life, so releases into the environment must be noted even if HP decomposes into water and oxygen. HP is a very reactive chemical so it's expectable that there's some reaction products from reactions with minerals. These potential issues must be studied and clarified before large scale utilisation of HP based formulations. In this presentation our experimental observations are summarised giving some answers to these questions and some findings from literature also reviewed.

1 HYDROGEN PEROXIDE (HP) AND HYDROGEN PEROXIDE EMULSION (HPE)

HP is a powerful oxidiser and widely used in process industry. Technical grade HP is usually available up to 60 w-% solution. Properties of aqueous HP solutions are well known. HP is a very reactive chemical, much more reactive than AN. Contamination usually leads to fast exothermic decomposition of HP. This reaction produces large volume of oxygen gas and vapourised water leading to pressure build-up or even to explosion. In this connection it must be mentioned that sometimes the term oxygenated water is misleadingly used in parallel with HP. This is of course a serious mistake. HP is a chemical compound and oxygenated water is a mixture of oxygen and water. These two having completely different chemical characteristics and risks.

Explosive compositions based on HP can be formulated in many ways. In this context the focus is on mixtures with fuels, not molecular explosives even if HP is easily capable of forming those too. Typical mixtures with fuel are 1) absorbed in solid material, 2) water gel and 3) water-in-oil emulsion. The last one (HPE) being the most obvious because continuous fuel phase protects HP droplets from contamination to some degree.

Technically speaking usability and performance of HPE is comparable to common ANE products with some exceptions. HPE viscosity and pumpability can be adjusted so that standard ANE loading technology can be converted for HPE. Detailed risk analysis is of course at the centre of the conversion. Energy output of HPE is calculated and compared to standard ANE products. Basically, hydrocarbons in fuel phase are oxidised by oxygen released from HP. Based on this principle a relation can be made between concentration of HP and ideal energy output of

HPE. For example, oxygen balanced HPE made of 50% HP solution energy is ca 2.4 MJ/kg and in case of 60% HP ca. 3.7 MJ/kg calculated with Explo 5.0 software. Standard ANE like Forcit's Kemiitti 810 has 2.9 MJ/kg. Forcit has tested performance of Kemiitti HPE in development blasts. Fragmentation and advance was at least as good as with ANE and measured VOD was 4400...5300 m/s in Ø51 mm horizontal holes. HP concentration used in this test was 56%.

Technical differences to ANE come mainly from lower density range of HPE. Ungassed HPE has density of 1.15-1.20 g/cc depending on HP concentration. Typical density window for gassed HPE is 0.8-1.0 g/cc. So, in water containing vertical boreholes there's a risk of getting part of the explosive column loose and floating on top. Another feature with the lower density window may be a limitation in vertical hole depth. Gassed product compresses itself to higher density in the bottom of borehole and if density gets too high initiation sensitivity is probably lost. Theoretically estimated critical column height is about 20 m but experimental verification by VOD measurements is needed to confirm the estimate.

2 OCCUPATIONAL HEALTH

According to ECHA (European Chemicals Agency) long term (8h) exposure limit to HP in air is 1ppm and short term (15min) exposure limit is 2ppm. Some of the first symptoms of exposure to HP vapor are itching and burning feeling on skin, airways and eyes. Later dizziness. Repeating long term exposure can lead even to permanent lung damage. Study shows that fine particulate matter helps HP entering deeper in lungs causing more severe damage. This may be a relevant factor in mining environment.

During our pilot and field scale testing HP concentration in air has been monitored continuously with Dräger X-am 5100 sensor. This sensor is capable of measuring HP concentration up to 20 ppm. The readings have been repeatedly over 2 ppm whenever HP or HPE is handled. We haven't observed any differences if HP was 50% or 60% concentration. The measured values are presented in the table 1.

We have observed that reactivity evolves and spreads very fast through the whole mass of HPE. When the first signs of vapour cloud and bubbling appear the reactivity spreads through whole mass in minutes. Probably this is due to sharp temperature increase (up to ~100°C), strong convection and chemically aggressive radicals

released from decomposing HP. The high concentration of HP in vapour is caused by increased vapour pressure of HP in boiling HPE but also strong convection of oxygen gas and vapourised water that carries HP as an aerosol. Whenever HP sensor is brought close to vapour cloud the readings immediately jump over the scale. Yet another detail increasing potential consequences from exposure to fumes from reactive HPE is that with some reactants oxygen released from decomposing HP can be in excited molecular state, so-called singlet-state. This singlet-oxygen is classified as a reactive oxygen species which can damage DNA i.e. shows carcinogenic effects.

Table 1. Measured HP concentrations in workplace air.

Measurement point	HP concentration in air, ppm
Refilling loading unit tank, hatch open, in 1m radius	10-15
Refilling loading unit tank, hatch open, in 5m radius	1-2
During early stage of loading tunnel round	0-0.3
After ca 30min from start of loading tunnel round	1.0-1.5
In vapor cloud coming from reactive HPE	>20

Contamination is an everyday risk in loading with bulk products. Both ANE and HPE have similar greasy and sticky consistency. From a long experience using ANE products in industry we have learned how easily stains of emulsion are spreading all around. ANE contamination is an annoyance without any severe consequences. In the case of HPE the contamination and spreading takes place in a similar way but the consequences can be more serious. The mildest one is getting skin contact causing some itching and a stinging/burning pain. Potentially an infection of damaged skin later. In more serious cases HPE gets in the eye causing permanent damage. Contaminated working clothes and other combustible materials can set on fire spontaneously. We have seen HPE spreading for example on car seats and detonator boxes. During the years of our experimental work with HPE we have had two cases where organic material used to clean HP/HPE stains have ignited spontaneously.

Forcit's solution to these risks at this stage of development is proper PPE, very careful working methods and keeping volumes at a manageable level. The key points are protecting breathing, preventing skin contact and avoiding any spreading of HPE. A motorised full-face mask with suitable gas filter, chemical resistant coveralls, long-sleeved chemical resistant gloves and rubber boots (leg sleeves on boots) is seen as an adequate set of PPE.

Forcit has arranged three campaigns of blast fume measurements for Kemiitti HPE product. Two of them in 35 m³ blast chamber with 1 kg charges and one in an underground mine. Tests proved the obvious result that no NO_x exist in blast fumes. But from an occupational health point of view the concentration of CO has been comparable to levels gotten from ANE product. Also, some ppm of HP was detected. This means that proper venting after HPE blast is as important as is in case of ANE. HPE formulation can be optimised so that CO production is minimal, this is one development goal for future work.

The blast chamber results were presented at EFEE 2023 in Dublin by LTU.

The full-scale test in underground mine was done in co-operation with VTT (Technical Research Centre of Finland Ltd) in 2024 and the results will be presented in EFEE 2025 by VTT.

3 SAFETY

HPE contains more than 90 weight-% HP solution hence HPE has similar potential for reactivity as HP. W/O structure hinders the reactivity because water immiscible fuel phase is continuous. But as already mentioned the emulsion structure doesn't remove reactivity. Reaction releases large volume of oxygen gas and water vapour and a lot of energy is released which heats up HPE accelerating the reaction further. According to our observations reactive HPE gets very slippery so it may fall out from downward inclined holes very easily causing a risk of heavy contamination for workers and surroundings and of course affects negatively to blast result. Upholes being the extreme case in this respect.

Compatibility with other explosive components requires risk assessment and test data. The industry has gotten used to compatibility of common materials and components with ANE. In case of HPE the chemistry is different and especially in case of thermal runaway reaction it's critical to know whether detonators and primers/boosters commonly used are compatible with HP. In worst case reactivity in HPE in

combination with other explosive components can lead to premature detonation. A special case of compatibility issues is capability of HP forming initial explosives with certain chemicals. Reaction with acetone or hexamine proceeds easily and products are triacetone triperoxide and hexamethylene triperoxide diamine respectively. These are very powerful and very sensitive compounds. Risk analysis should cover this kind of factors too.

Another aspect of risk management is to understand the nature of HPE from the viewpoint of classification. It may be tempting to make a direct comparison to ANE. But by definition UN3375 isn't usable for HPE. According to UN Manual of Tests and Criteria point 20.2.1b mixtures of oxidising substances which contain 5.0% or more combustible organic substances shall be subjected to the classification procedure for self-reactive substances. Testing and analysis to possibly include HPE in the framework of ADR is future work. All this is on a condition that HPE doesn't change during transport or storage. Vented packages and tanks are required for HP because it decomposes spontaneously and oxygen gas must be released without pressure build-up. In case of HP solution oxygen gas bubbles can escape from liquid. In the case of viscous HPE the oxygen bubbles can't escape. This means that there's a possibility that HPE becomes spontaneously transformed into explosive. HPE may be classified initially as non-explosive but after a certain time it spontaneously transforms into an explosive. Time for this transformation is dependent on temperature, formulation details and potential contamination. In some early tests for gassed HPE we used this phenomenon to make samples for VOD measurements. HP grade without special stabilisers was used to make HPE samples in plastic pipes. Pipes were filled with fresh HPE and then density was followed on daily basis. After three days we had a cap-sensitive HPE. With improved stabilisers in HP this process can be slowed down significantly. Spontaneous gassing is observed and reported elsewhere too. For example, in Australian tests sensitivity to detonation was observed after 29 days. Limiting transport and storage temperature and batch age control may help with this issue. But from the experience of ANE handling we know that there's always layers of older material in tank walls, corners and pipes. So, there's always a risk on existence of older potentially sensitised material in an HPE system. A common estimate for decomposition of HP is less than 1% in a year. In Figure 1 some estimated

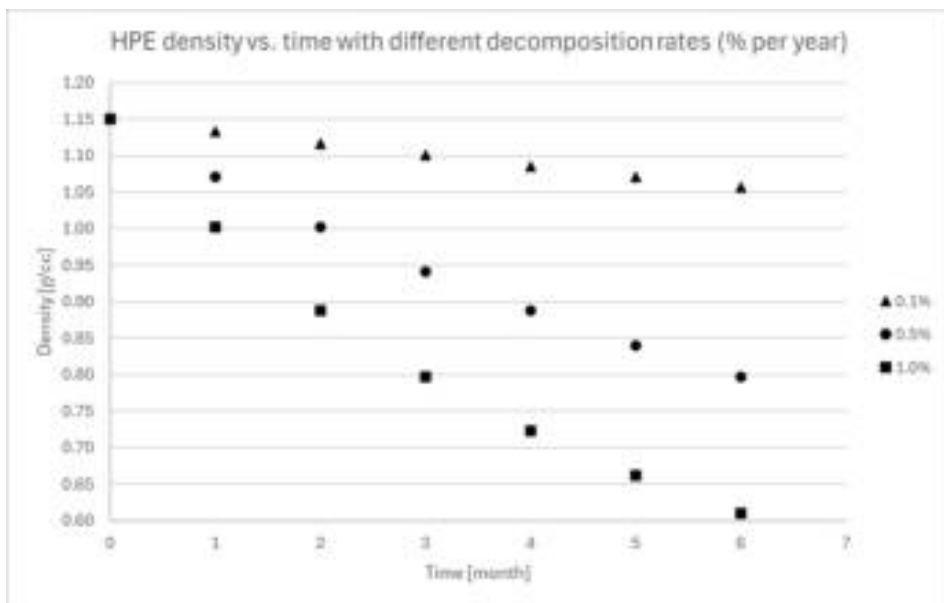


Figure 1. Estimated density of HPE with different annual decomposition rates.

densities vs. time are shown with different decomposition rates for HP. In density below 1.05 g/cc HPE is detonable for sure. To keep classification and risk management under control the supplier should provide test data showing that spontaneous sensitisation doesn't occur if HPE is to be transported or stored as a non-explosive material.

Finally, there's a question of waste handling. From experience we know that in bulk emulsion system it's very hard to avoid waste completely. In mine conditions mineral dust is everywhere causing contamination. Contaminated HPE can start a thermal runaway decomposition at any moment. This means that firstly creating any HPE waste must be avoided and if waste is created it must be converted to a safe (unreactive) form immediately. No storage of HPE waste. There's no simple solution to this issue but at least it must be acknowledged and understood before entering any practical scale loading. Demulsification may be the first step to reduce risk level. At least fuel and oxidiser are separated then. Still concentrated HP solution remains and as an aqueous liquid it is even more reactive with contaminants.

4 ENVIRONMENTAL IMPACT

HP is marketed as a green chemical because decomposition products are oxygen and water. In

use as a raw material for bulk explosive the release scenario should be re-evaluated. In form of w/o emulsion HP is protected against external factors to some degree. Many studies show that 5 - 20% of bulk explosives used stay unreacted in muck after blast. This would create a significant source for HP into the environment. Depending on further handling of rock material and mine water varying amounts of HP can enter surrounding waters. According to recent studies HP is toxic to aquatic life. There are no results showing the consequences of large-scale use of HPE to surrounding waters so studies are needed to avoid any negative surprises.

When HP/HPE reacts with minerals some metals in water soluble form can be expected. In Table 2 analysis results of two tests are shown. One sample was drill cuttings from pyrite mineral (sample Py) and the other was drill cuttings from granite-based mineral (sample Gr). Py-0 = pyrite sample washed with water, Py-HPE = pyrite sample reacted with HPE and washed with water. Gr-0 = granite sample washed with water, Gr-HPE = granite sample reacted with HPE and washed with water. Pyrite sample was very reactive with HPE. In just minutes after contact a violent fast spreading decomposition of HPE took place. In contrast with granite reaction was very slow and just notable after strong mixing. In the case of the

Table 2. Metal concentrations ppm in water.

Sample	Al	Ca	Cd	Co	Cu	Fe	K	Mg	Mn	Na	Ni	Pb	S	Zn
Py-0	452	516	9	4	1363	1147		1253	103	4	1	52	5528	3360
Py-HPE	539	538	6	4	1472	3955		926	73	3	1	56	6506	2388
Gr-0		468					71	50		1335			122	
Gr-HPE		196					39	21		514	1		56	

Pyrite sample the filtered water was blue-green when the sample wasn't in contact with HPE and the sample that reacted with HPE had deep brown colour in water. This may be due to different oxidation states of iron. The metal analysis from filtered water samples was done with ICP so it can't make any distinction between different oxidation states of the metals. The goal of this test was to see whether something new appears in reaction with HPE. The results show that only in case of granite nickel appears in the analysis results when sample had reacted with HPE. All the other metals existed in water soluble form before reacting with HPE. The tests arrangement wasn't very accurate so more tests are needed for estimating actual potential for metal releases.

As a powerful oxidiser HP is capable of transforming oxidation states of metals. One case of special concern is chromium. In a slightly alkaline environment HP can oxidise harmless Cr(III) into very toxic Cr(VI). Spray concrete, filler used in underground production areas and limestone create this kind of alkaline environment. Test results are lacking but chromate is such a highly toxic pollutant that this topic needs to be studied thoroughly before large scale use of HPE especially in areas where chromium containing minerals exist.

5 CONCLUSIONS

HPE has shown promising results in field tests from a blasting technical point of view. The main challenge is safety. A new level of personal protection is needed. Working methods needs to be developed to avoid spreading and contamination. The ultimate solution will be mechanised loading where the operator is located in the cabin of a loading unit.

From a material safety point of view, reactivity of HP/HPE and consequent effects of it and spontaneous sensitisation of HPE are the main challenges. One solution to transport and storage risks is on-site production integrated with a loading unit.

Environmental issues require more studies before large scale utilisation of HPE. The risks are potential releases and toxic effects of HP in surrounding waters, products from reaction of HP with minerals and ground, especially potential oxidation of harmless chromium (III) into very toxic Cr(VI).

REFERENCES

- Agnez-Lima, L.F., Melo, J.T.A., Silva, A.E., Oliveira, A.H.S., Timoteo, A.R.S., Lima-Bessa, K.M., Martinez, G.M., Medeiros, M.H.G., Di Mascio, P., Galhardo, R.S. & Menck, C.F.M. 2012. DNA damage by singlet oxygen and cellular protective mechanisms, Mutation Research/Reviews. In *Mutation Research*, Vol 751, Issue 1, Jul-Sep 2012, pp. 15-28.
- Bailey, V. 2017. Characterisation of hydrogen peroxide based explosives and ventilation modelling to quantify re-entry times in underground development blasting, 2017, BEng Thesis Mining, The University Of Queensland, Australia.
- Fang, J., Samuelsen, O.B., Strand, Ø. & Jansen, H. 2018. Acute toxic effects of hydrogen peroxide, used for salmon lice treatment, on the survival of polychaetes *Capitella* sp. and *Ophryotrocha* spp. *Aquaculture Environment Interactions*, Vol. 10, pp. 363-368, 2018.
- Hydrogen peroxide safety & handling, Solvay Chemicals, Inc.
<https://www.solvay.com/sites/g/files/srpend616/files/2019-10/H2O2%20Safety%20and%20Handling%20of%20Hydrogen%20Peroxide.pdf>.
- Kettle, A.J. & Sellers, E. A novel chemically-stabilised hydrogen peroxide/hydrated oil-based emulsion-type explosive with competent blasting characteristics for a

nitrogen-free mining legacy: Results Of United Nations Classification Tests Of Explosives For Authorisation Submission, 12-ISEE22-Full-paper-Emulsion-type-hydrogen-peroxide-explosives-for-nitrate-free-mining-legacy-AJK01.pdf.

Liang., J., Huang, X., Yan, J., Li, Y., Zhao, Z., Liu, Y., Ye, J. & Wei, Y. 2021. A review of the formation of Cr(VI) via Cr(III) oxidation in soils and groundwater. *Science of the Total environment* 774 (2021) 145762.

Manfra, A., Sharma, J. & Kilburn, J. Inhalation pneumonitis caused by Nebulised hydrogen peroxide. *Cureus Open Access Case Report*, DOI: 10.7759/cureus.3811n.

Morio, L.A., Hooper, K.A., Brittingham, J., Li, T-H., Gordon, R.E., Turpin, B.J. & Laskin, D.L. 2001. Tissue injury following inhalation of fine particulate matter and hydrogen peroxide is associated with altered production of inflammatory mediators and antioxidants by Alveolar Macrophages, *Toxicology and Applied Pharmacology*, Vol 177, Issue 3, 15 December2001, Pages 188-199.

Roseth, R., Olsen, R.C.J., Sverdrup, E.M., Pedersen R., Bergheim, H., Gulyaeva, K., Roaldkvam, E. & Harstad, A.O. 2025. Tunnel driving med HYPEXBIO. Pilotstudie av utvasking av hydrogenperoksid fra sprengstein og tunnel, Nibio rapport, ol. 11, Nr. 72, 2025 (In Norwegian).

<https://echa.europa.eu/de/registration-dossier/-/registered-dossier/15701/6/2/10>.

[https://www.dni.gov/files/NCTC/documents/jcat/fIRSTresponderstoolbox/78--NCTC-DHS-FBI---Triacetone-Triperoxide-\(TATP\)-.pdf](https://www.dni.gov/files/NCTC/documents/jcat/fIRSTresponderstoolbox/78--NCTC-DHS-FBI---Triacetone-Triperoxide-(TATP)-.pdf).

https://www.dni.gov/files/NCTC/documents/jcat/fIRSTresponderstoolbox/104s_-_First_Responders_Toolbox_-_Hexamethylene_Triperoxide_Diamine_HMT_D-survey.pdf.

RETURN TO CONTENTS PAGE

Prilled hydrogen peroxide mining explosive

M. Araos

STX Technologies, Brisbane, Australia

ABSTRACT: Hydrogen peroxide (HP)-based explosives have been present in the mining industry for the last 6-10 years. The development of hydrogen peroxide-based explosives was primarily driven by the need to eliminate after-blasting NO_x fumes produced by current ammonium nitrate-based mining explosives. Later, researchers discovered that HP-based explosives could also eliminate the nitrate leaching that AN-based explosives produced. Currently, a company is offering a commercial HP-based emulsion-type mining explosive in the Nordic countries. There is also an HP-based watergel-type mining explosive that has been thoroughly studied in the literature. The technology of these products is similar to the ammonium nitrate-based counterparts, emulsions and watergels. However, AN-based mining explosives also include a product known as ANFO, which is safe, simple to manufacture, cost-effective, and straightforward to handle. The product is prepared by mixing AN prill and fuel oil. It is in this area where the HP-based technology still presents a gap - there is no prilled-type HP-based explosive so far that can offer the same characteristics of ANFO. This document will present the development of an HP-based prilled-type explosive (formula and detonation properties). Two components make up the product: an HP solution and an absorbent polymer. The densities of the product varied between 0.90 and 1.00 g/ml. The explosive product detonated at a velocity between 2,600 m/s and 3,600 m/s in 100 mm inner diameter PVC pipes (unconfined conditions). A hypothesis is put forward to explain the detonation mechanism of the prilled HP. This breakthrough development paves the way for HP-based explosive technology variants that can streamline field operations and enable compliance with environmental standards, among other advantages.

Keywords: Detonation properties, environmentally friendly explosives, hydrogen peroxide, non-ideal explosives.

1 INTRODUCTION

Most of the current explosives for mining blasting activities are based on ammonium nitrate (ANFO, emulsions, and watergels). However, in 2012, a mining explosive based on hydrogen peroxide (HP) was patented (Araos 2012). Then, in the following years, plenty of literature has been published about it (Araos & Onederra 2015 & 2017, Gustavsson *et al.* 2025). Those publications address the properties of HP-based mining

explosives as watergel and, to a lesser extent, as emulsions.

In 2024, the research on HP-based products unveiled a new category, a sphere-like HP explosive (or prilled-HP). An absorbent material (sphere-shaped) was added to the HP solution. Upon absorption, the final HP-based product resembled prilled AN.

The lab work focused on understanding the swelling/absorption of HP by the absorbent material (which also served as fuel for the

Table 1. Formulas for the prilled-HP explosive.

Component	HP70-A	HP60-A	HP50-A	HP70-AG	HP60-AG	HP50-AG
HP (wt.%)	56.00	49.44	42.50	55.09	48.72	41.9
Water from HP (wt.%)	24.00	21.18	18.19	23.61	20.87	17.93
Water added (wt.%)		11.78	24.31		11.61	23.97
Glycerine / gum (wt.%)				19.30	16.79	14.20
Shaped material	20.00	17.60	15.00	2.00	2.00	1.99
Density (g/mL)	0.91	0.89	0.88	1.01	0.96	0.92
OB	0.05	0.12	0.12	0.14	0.14	0.05

composition), absorption time, use of the absorbent material with other fuels, etc. The second stage involved the measurement of the detonation properties of the prilled HP in a blasting range site. Three parameters were assessed during the detonation properties tests, concentration of HP, a sole fuel (the absorbent material), and a fuel that was prepared by a combination of the absorbent material and glycerine. Overall, the VODs measured for the prilled-HP were similar to those of ANFO, or even higher. A further component, safety of product, was not included at this stage.

The focus of this study is primarily on the chemical preparation and physical detonation characteristics of the prilled HP product, while safety considerations will be addressed in

subsequent research phases. This approach allows for a comprehensive understanding of the product's performance before evaluating its safety implications.

2 EXPERIMENTAL

2.1 Formula

This work used an explosive based on both a hydrogen peroxide solution and a powder absorbent material (sphere-like, with a diameter below 0.3 mm) that absorbed the HP solution and formed a multitude of spheres (or beads). The same absorbent-shaped material was also used as a fuel for the explosive composition. The initial raw material was HP 70% w/w. Explosive composition



Figure 1. Estimated density of HPE with different annual decomposition rates.

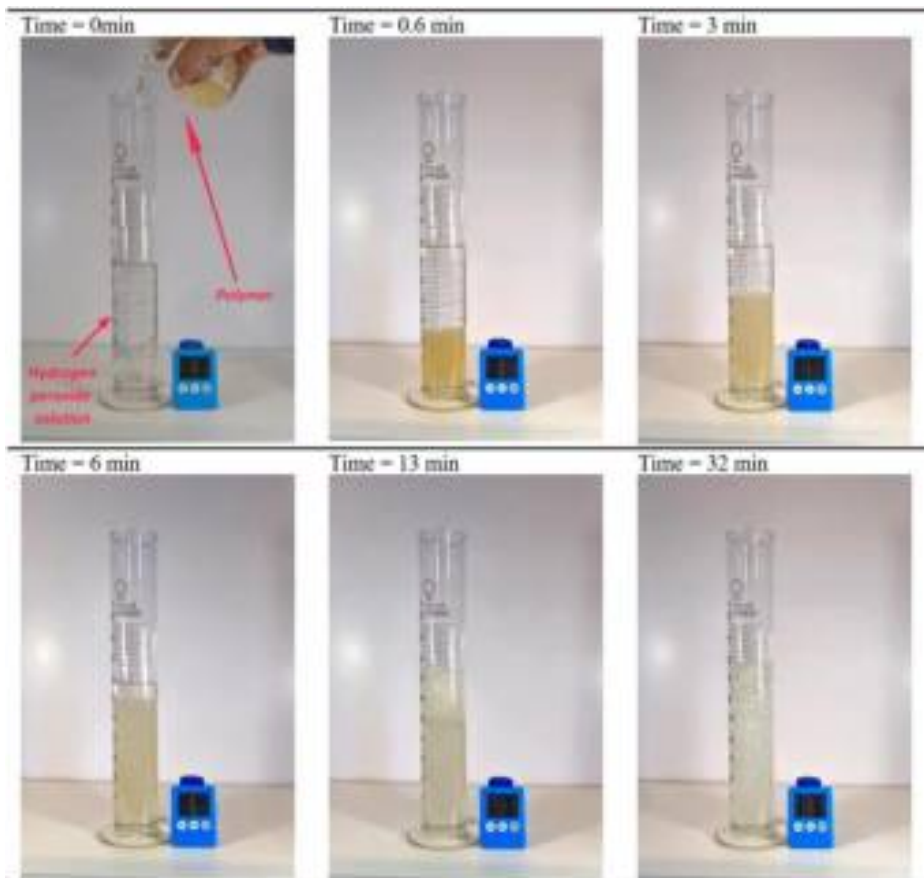


Figure 2. Preparation of prilled HP.

samples with HP 60% and HP 50% w/w solutions were made by diluting HP 70% w/w.

One type of explosive was made using HP and the absorbent material (HP-A), and another set of samples was a mixture of a liquid fuel (glycerine) and the absorbent material (HP-AG). For samples of type HP-AG, the HP was mixed with glycerine first, and then the absorbent material was added.

2.2 Detonation tests - charge preparation

For the detonation tests, the prilled HP was prepared using a 20 litres plastic bucket. The HP (or HP with glycerine) was weighed out in the bucket, and the absorbent material was added. The plastic bucket was rotated to facilitate the HP absorption. After 20 to 30 minutes, the samples were ready for testing.

Explosive weight for the charges varied between 6-8 kg. The samples were loaded into PVC pipes having an inner diameter of 102 mm. A 150-gram booster pentolite was used to initiate each of the explosive charges. The charges were suspended using a rope attached to a steel frame. A picture showing the setup is displayed in Figure 1.

2.3 Instrumentation

Camera: To observe the detonation process, a Chronos 2.1 camera was used (Krontech 2025). The camera was positioned at a distance of 130 m from the charge detonation. The rate of the video capture was 1,000 frames per second. The definition of the movies was 1920x1080.



Figure 3. Preparation of HP-prills.

VOD equipment: The VOD of the samples was continuously measured using a ShotTrack mini-305 (Shottrack 2025), with a coaxial cable externally attached to the pipe to measure the VOD.

3 RESULTS

3.1 Laboratory

Figure 2 shows pictures of the absorption and formation of spheres when adding the absorbent material to HP in a glass cylinder. Upon the addition of the absorbing material, the prilled HP product was ready at around 20-30 minutes. The absorbent material quickly absorbs HP in the first 2-6 minutes and then the absorption becomes slower. No HP solution was left over at the bottom of the glass cylinder; all material was absorbed after 30 minutes.

3.2 Blasting range

Figure 3 displays the preparation of prilled HP for the testing pipes. The prilled HP had a wide particle size distribution.

Figure 4 displays the still frames of the detonation of the different products. It is observed that the sample HP70-AG produces a bright, large flash at 1 millisecond (ms), whereas the flash size decreases for HP60-AG, and then sample HP50-AG did not detonate (see the green oval in the frame at 1 ms, it is observed that half of the pipe misfired).

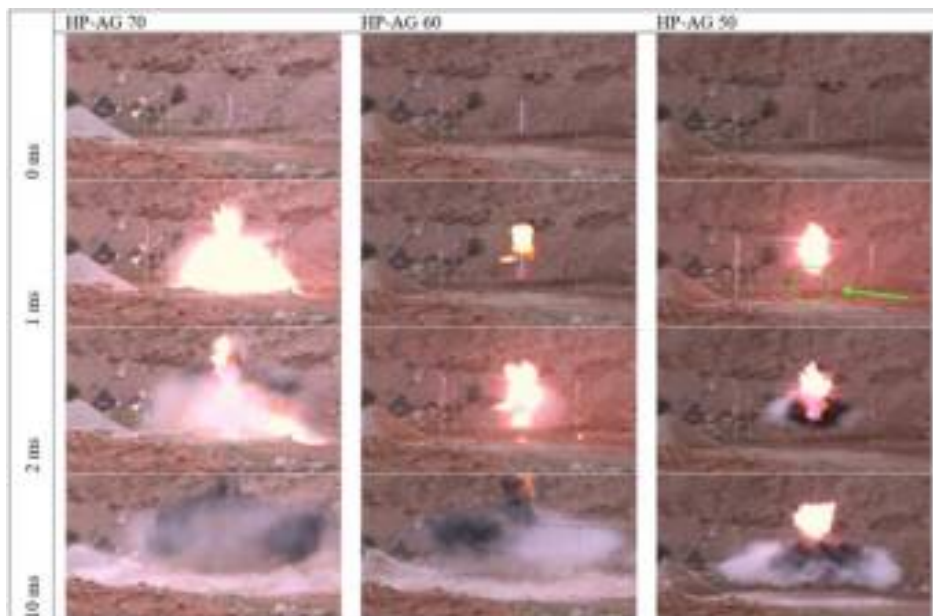
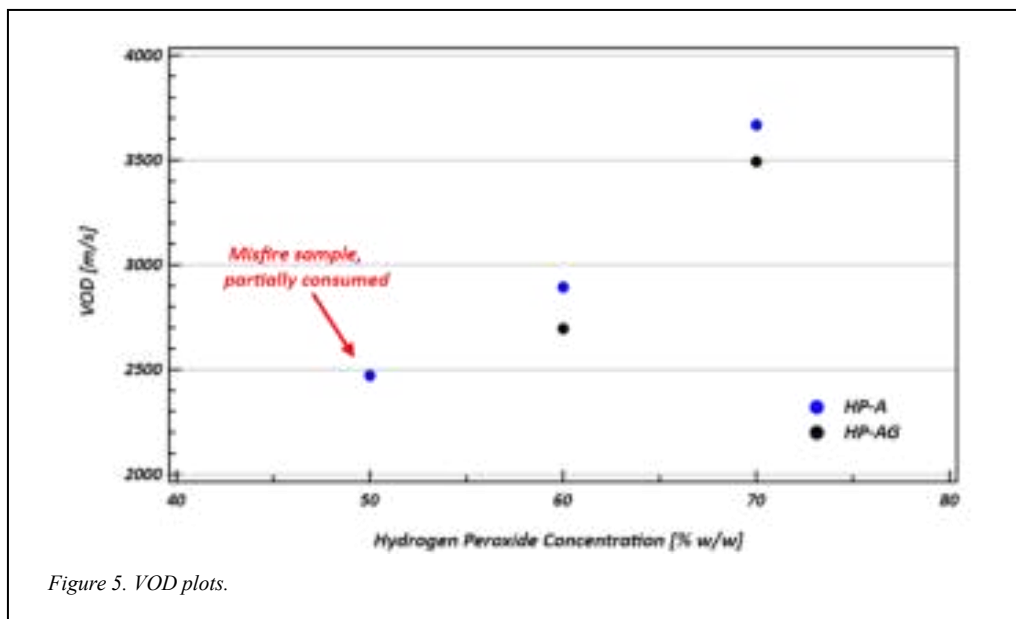


Figure 4. Still frames of the detonation of the prilled HP samples.



4 DISCUSSION

Preparation of the product showed that the prilled product is more versatile than ANFO regarding potential densities. The prilled HP product's density varies between 0.88 g/ml to 1.01 g/ml (refer to Table 1).

Table 2 displays the VOD results for all the samples tested in PVC pipes with a 102 mm inner diameter. Samples HP50-A and HP50 AG misfired. In the case of HP50-A, some part of the pipe detonated (150 mm, with a, approximately VOD of 2450±50 m/s) and that is the VOD displayed in the plots. Sample HP50-AG did not detonate at all.

Table 2. Summary of VOD.

HP (%)	VOD (m/s)	
	HP-A	HP-AG
70	3669	3493
60	2893	2695
50	2471	0

The plots displaying the VOD are shown in Figure 5. A consistent higher VOD is observed for the prilled HP samples made with HP 70% w/w.

It is also observed that samples with only absorbent material (HP-A) display a higher VOD. Because the fuel was only the absorbent material,

the number of spheres/beads present in the mixture was higher than in the case of HP-AG formulations. It is hypothesised that the larger number of spheres contributed to the high VOD observed (more hot spots present).

It is also hypothesised that the prilled HP product could be prepared with other types of fuels, like for example, sugar, isopropyl alcohol, etc. These substances are soluble in water and HP, making them suitable for use as fuel.

Figure 6 displays a picture of an AN prill alongside the prilled HP. The prilled HP exhibit no fissures, porosity, or cracks in their structure, in contrast to AN prills. In fact, the surface is smooth. The fissures, porosity, and cracks in AN prills have long been considered the primary factors contributing to the detonation of ANFO. However, the prilled HP, characterised by a smooth surface and capable of detonation, suggests that the detonation mechanism differs from current understanding.

The igniting mechanism could be attributed not to porosity but to the material's spherical shape, which generates a jet upon impact from the detonation shock (Bowden 1963, Bourne 1989). This mechanism, when applied to void sensitised non-ideal explosives, has been documented in other sources (Araos 2023). Figure 7 displays a schematic of this mechanism. The jetting, which travels at a speed far exceeding that of the detonation shock, impinges on prills, causing friction, temperature increase, and energy release.

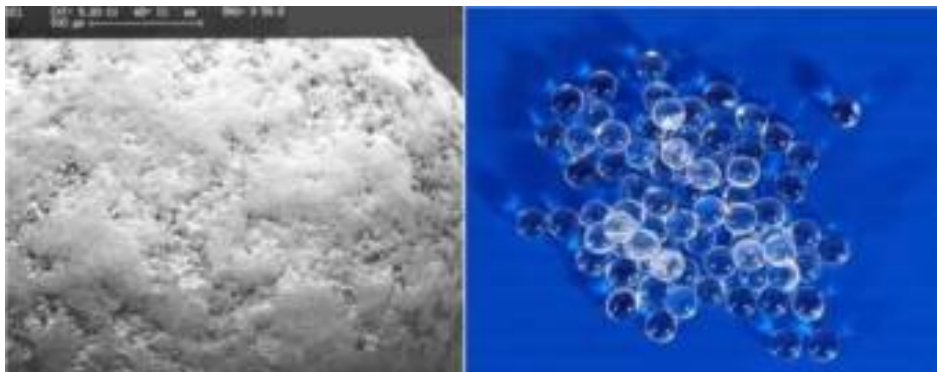


Figure 6. Pictures of AN (left, scanning electron microscopy) and HP prills (right).

Then, when the detonation shock reaches that area, the energy released by the prills sustains the detonation. Therefore, the detonation of HP and AN prills is not related to any porosity in the material, but to the spherical shape of the material, which causes a jet.

Overall, this is a development work, and more tests need to be conducted to confirm the VOD obtained by the prilled HP in this study. Additionally, work will be conducted on the safety side.

5 CONCLUSION

This preliminary study has shown the development of a novel explosive, which is prilled like ANFO, but made from liquid hydrogen peroxide solution and an absorbent material (and other fuels).

The prilled-HP, made with HP 70% and 60% w/w solution and fuel, detonates with similar VOD as ANFO. The prilled HP made with HP 50% w/w solution was unable to detonate, regarding if it had only absorbent material or absorbent material and

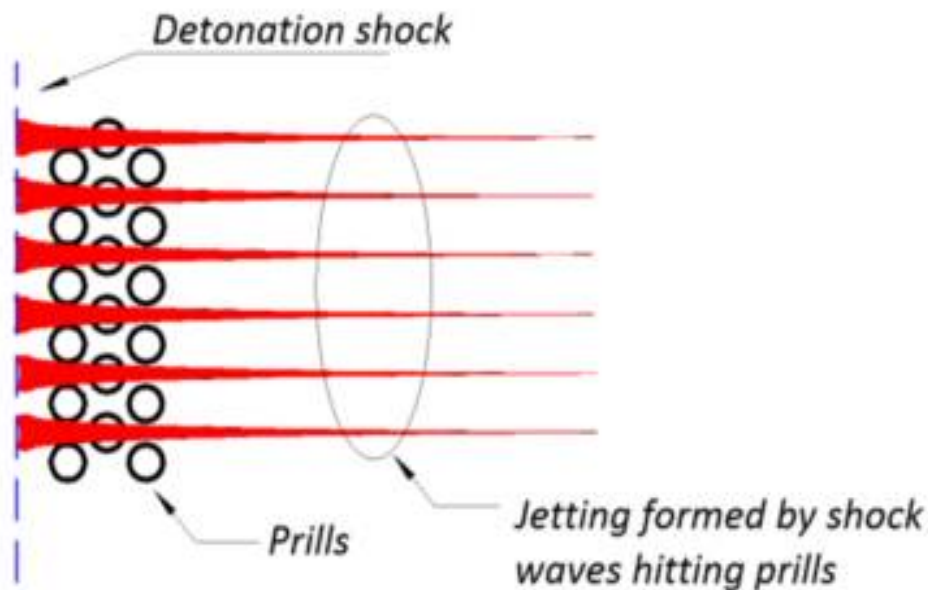


Figure 7. Relationship Jetting formed by the hitting of shock waves on to prills.

glycerine.

The detonation of the prilled HP appears to occur by a jetting formation.

Work will continue to confirm these results and assessing the safety of the product.

6 ACKNOWLEDGEMENTS

The authors would like to thank the Extech for the use of the blasting range located in Queensland, Australia. The data collected has been invaluable to develop this explosive.

REFERENCES

Araos, M. 2013. *WO patent 2013/013272*, Explosive composition.

Araos, M. 2023. Ignition mechanism for non-ideal explosives. In *12th Conference on Explosives and Blasting*. Dublin, Ireland. The European Federation of Explosives Engineers.

Araos, M. & Onederra, I. 2015. Development of a novel mining explosive formulation to eliminate nitrogen oxide fumes. *Mining Technology*, 124(1), pp.16-23.

Araos, M. & Onederra, I. 2017. Detonation characteristics of a NO_x-free mining explosive based on sensitised mixtures of low concentration hydrogen peroxide and fuel. *Cent. Eur. J. Energ. Mater.* 2017, 14 (4): 759-774.

Araos, M. 2023. Ignition mechanism for non-ideal explosives. In *12th Conference on Explosives and Blasting*. Dublin, Ireland. The European Federation of Explosives Engineers.

Bourne, N.K. & Field, J.E. 1989. Cavity collapse in a heterogeneous commercial explosive. In *Proc. Ninth Symposium (International) on Detonation*, Portland, Oregon, pp. 869-878. 1989.

Bowden, F.P. 1963. The initiation and growth of explosion in the condensed phase. In *Symposium (International) on Combustion*, vol. 9, no. 1, Elsevier, (1963):499-516.

Gustavsson, T., Bihlar, A. & Hillman, J. 2025. Sustainable hydrogen peroxide explosives show high performance in tunnel blasting; *Tunnelling into a Sustainable Future* –

Methods and Technologies – Johansson et al (Eds).

Krontech, 2025.

<https://www.krontech.ca/product/chronos-2-1-hd-high-speed-camera/>

Shottrack, 2025.

<https://www.shottrack.com.au/product/shottrack-vod-305-mini/>

RETURN TO CONTENTS PAGE

8. New applications and training

Optimising sublevel cave for sustainable mining

M. Lovitt

Underground Blasting Specialist – ORICA Ltd Australia

ABSTRACT: Sublevel caving (SLC) is a key method for extracting base and precious metals, often bridging open pit and block cave operations. It is highly carbon-intensive due to extensive development and drill-and-blast demands. Despite expectations, coarser fragmentation is preferred in SLC to widen the draw and reduce waste dilution. While burden optimisation is well-studied, ring spacing and initiation system design remain underexplored. SLC's single-ring firings under high stress conditions lead to 10–15% misfire rates, often due to brow breakback, hole dislocation, or poor cave connection. Safety risks and productivity losses drive conservative blasting, adding holes and increasing powder factor, which reduces recovery and increases dilution. Reliable wireless initiation systems could reduce misfires, costs, and dilution while improving recovery. This paper presents global case studies (Africa, Australia, Europe) where wireless blasting improved initiation but sometimes led to overly fine fragmentation, harming recovery. Reducing drill-and-blast intensity, delays, and dilution while improving recovery is key to making SLC more sustainable and profitable.

1 INTRODUCTION

Sublevel cave mining is a mass mining method that is used to mine massive steeply dipping orebodies where the host rock is strong, or not fractured, enough and requires breakage with drill and blast. It is a top-down mining method that does not use artificial support of the ore boundary and usually relies on the surrounding waste to cave into and follow the broken ore down to the drawpoints. It is commonly used as an intermediate method between open pit mining and block caving, as stresses in this intermediate zone are not sufficient to break the rock sufficiently to supply well fragmented rock to drawpoints for high productivity. It is a method that requires significant tunnel development as closely spaced, wide tunnels are developed every 15-30m vertically to minimise the dilution of the ore and maximise recovery. This paper does not offer optimisation strategies for the tunnelling

configuration but looks at the developments in blasting products to deliver the better fragmentation to maximise primary draw. Primary draw, being the draw of broken ore in the ring that has just been blasted. There are 3 distinct areas where this paper differs from many others in this field and that is through the Relative Bulk Strength (RBS) of the explosive, the reliability of detonation and the spacing distance between blast holes. It is through these parameters that costs can be reduced, productivity gained and improvements in recovery and dilution can be gained.

2 DRAW CONTROL

A global review of recovery, dilution and draw control in sublevel caving mines (Campbell 2022) benchmarked more than 20 sublevel cave mines around the world for a Cave Mining 2040 research program. It identified that draw control strategies with respect to fragmentation are crucial for

optimising ore recovery and minimising dilution. Proper blast design and execution to ensure the ore is sufficiently broken up, facilitating efficient draw control. Continuous monitoring of draw points helps in adjusting the draw strategy. This ensures that the ore is extracted efficiently while minimising waste rock inclusion. Implementing grade control measures helps in maintaining the quality of the ore being extracted. By monitoring the grade of the material at the draw points, operators can adjust the draw strategy to maximise ore recovery and minimise dilution. The use of automated systems for draw control allows for real-time adjustments based on fragmentation data. Optimising the draw strategy involves adjusting the draw rate and sequence based on fragmentation and orebody characteristics. This helps in achieving a balance between ore recovery and dilution control. Conclusions identified that while there was significant scatter in the data, modern SLC's 80 - 90% ore recovery at draw percentages of 100-120%, in line with expectations. This shows higher recovery and lower extraction from charts published 30 years earlier pointing to better control on parameters. Dilution was highly scattered with a range between 4 – 43%, outside the general range of 15-30% described in the literature. There is an increasing trend of mines shifting away from drawpoint sampling due to exposure of personnel to active mining areas, slow turnaround time for laboratory assaying, high grade variability and low reliability due to sampling bias.

3 FRAGMENTATION

In sublevel caving, achieving optimal rock fragmentation is crucial for maximising ore recovery and minimising dilution. The key technical aspects of fragmentation size distribution identify the target size should be such that the majority of the rock fragments are small enough to be easily handled by the loading equipment but not so fine that they cause excessive dilution. Typically, a P80 (80% passing size) of around 200-300 mm is considered optimal (Campbell 2022). A uniform fragmentation size distribution helps in maintaining consistent draw control and reduces the risk of hang-ups and blockages in draw points. Greater burdens resulted in better recovery and reduced dilution (Campbell 2018) as well as, for the same geometry, lower energy explosives. Suggesting that, provided the burden was not too great or that the energy was not too little, the ideal fragmentation should be as 'coarse' as possible without jeopardising the complete breakout of each

ring and ensuring the connection with the caving waste. Increase in powder factor, corresponded to over-compaction of previously blasted and cave material, resulting in unfavourable gravity flow behaviour.

Conventional ring design has key problems with achieving ideal fragmentation due to the logistics of underground drill configuration. Geotechnically, tunnels are restricted in width to ensure stability. Narrower tunnels help in reducing the risk of collapse and require less extensive ground support systems. The rock mass can arch over the tunnel, providing natural support and enhancing stability. Wider tunnels can lead to higher stress concentrations at the tunnel walls and roof, increasing the likelihood of rock failure and instability. The presence of geological discontinuities such as faults, joints, and fractures can significantly affect tunnel stability. Narrower tunnels are less likely to intersect multiple discontinuities, reducing the risk of instability. This not only reduces costs but also simplifies the installation and maintenance of support systems. By restricting tunnel widths, mining operations can achieve a balance between operational efficiency and safety, ensuring the stability of the tunnels throughout the duration of mining activities.

The tunnel width restrictions impact the drill design to dictate a radial fan of holes as depicted in Figure 1 to adequately blast the full width of the ring, which results in uneven energy distribution. At the apex zone, where the drill holes are at maximum spacing, the powder factor is lowest. In the core zone, where the drill holes converge, the powder factor is increased by more than 10% which creates more fines and coarse material and so less uniformity to the fragmentation.

4 MISFIRE RATE FOR SLC MINES

Bullock & Hustralid (Bullock 2001) and Kvapil (Kvapil 1998) refer to the ideal spacing as over 1.3 times the burden but this spacing is rarely used. Typically, the drill spacing is around 1 to 1.15 times the burden due to the high level of misfires that sublevel cave mines experience (Campbell 2022). In Campbell's 2022 paper the misfire rate of 10 to 15% was measured based on data collected from multiple sublevel caving operations worldwide. The study involved benchmarking practices across 21 mines operated by 17 different companies on four continents. Further work has been reported from Kemi mine (Vauhkonen 2022) that supports the high misfire rate in SLC mines through nitrate contamination of ground water.

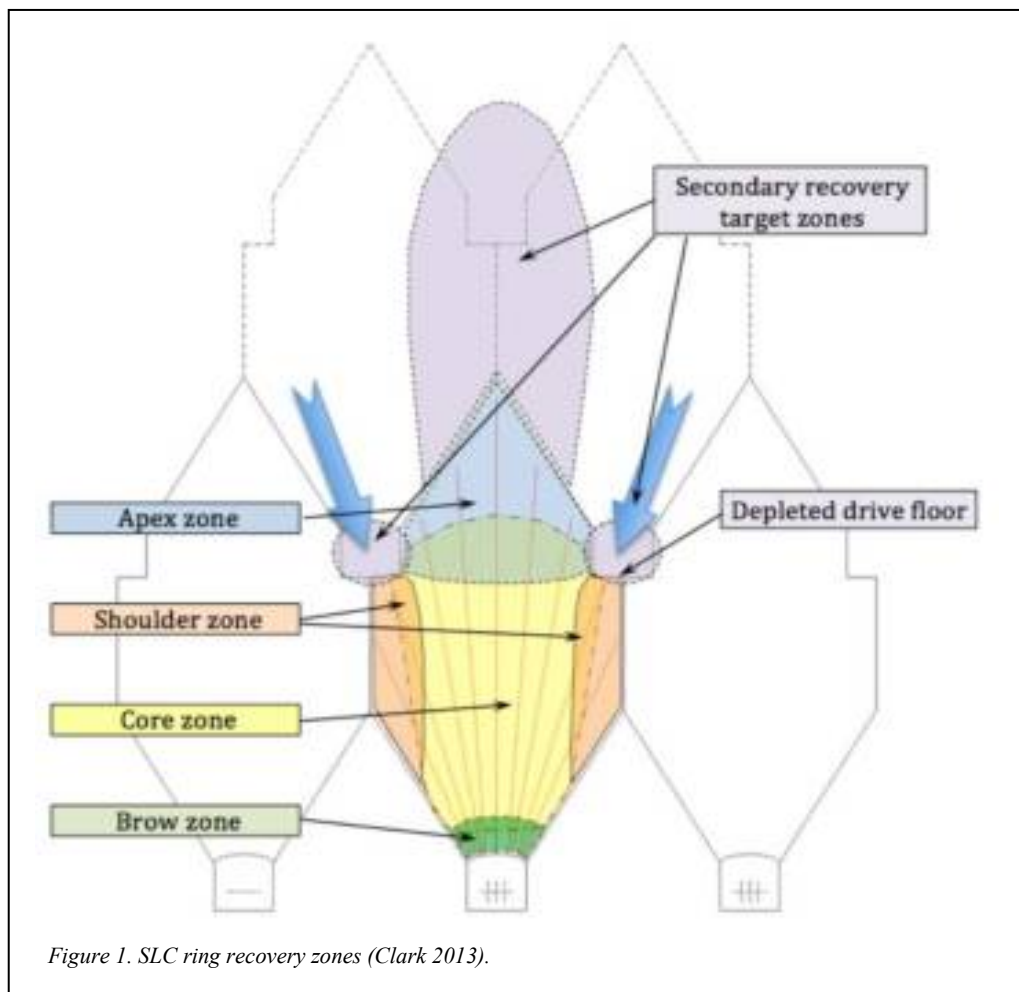


Figure 1. SLC ring recovery zones (Clark 2013).

The misfire rate is high due to blast design incorporating 10-15% extra energy to cope with the misfire rate. Low powder factor exits at the toe and poor blast geometry due to a low spacing to burden ratio.

Misfires can result from product issues from malfunctioning, dynamically desensitised or improperly connected detonators. The presence of water in blast holes can dampen explosives, preventing them from detonating effectively. Rock mass conditions have a major influence with stresses, fractures and faults dislocating the explosive column and isolating explosives from primers leading to misfires. Charging process mistakes or inability to implement the design during the loading and initiation process can also cause misfires.

The major solution to the misfire problem is using wireless booster such as Orica's WebGen™

200 wireless initiation system which uses magnetic inductance (MI) to communicate through the rock, air and water to initiate the booster. The experience of firing ~0.5M units in over 10 thousand blasts, communication is tested before the system is used in the ground giving the highest level to eliminate misfires. Geology where the system has problems in communication can be avoided, so the biggest cause of misfire (physical connections and availability of holes) is eliminated. The very first trial at Ernest Henry SLC Mine in 2016 (C te Kloot 2017) had no misfires and the product has seen much development and refinement since then.

5 DRILLING DESIGN

Analysing the conventional pattern identifies the reason for less-than-ideal fragmentation negatively



Figure 2. Plane section at the toe area of the first 2 blast holes in a SLC ring 2.6mB/3mS, blasting up to the free face.

impacting recovery and dilution. Shorter burdens result in finer fragmentation and, if a blasthole has two directions to blast, it will preferentially fragment finer to the shorter burden and put less energy into the larger burden direction (coarse).

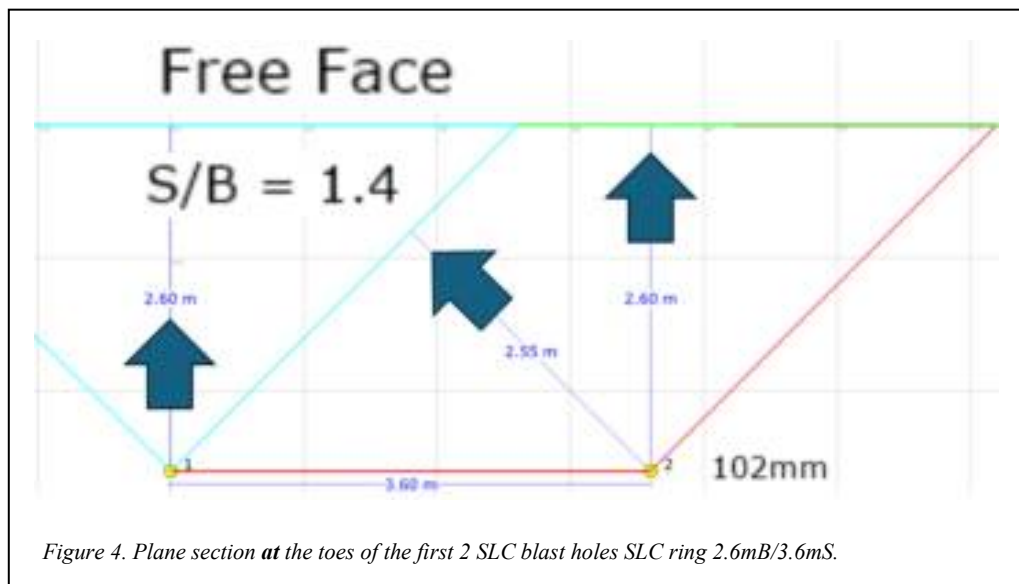
An example pattern from SLC mines of 2.6m burden, 3.0m spacing with a 102mm blast holes the geometry of the first 2 blast holes will be investigated. In a sublevel caving scenario, the horizontal breakout angle to the free face for a blast hole confined by waste typically ranges

from 45 to 60 degrees (Kazakidis 2015). The toe geometry of the first two holes is represented by Figure 2 showing a 45-degree breakout (90 degrees total) for the first blast hole giving the geometry for the second hole. Using a larger crater amplifies problems for subsequent holes.

The second hole has two free faces to blast too, with one being 2.14m and the other 2.6m. These are similar and the energy of the second hole will be distributed evenly. If the crater from the first hole is greater than 45 degrees (depicted in Figure



Figure 3. Plane section at mid length for the first 2 blast holes in a SLC ring, note short burden and region of oversize.



2), the smaller burden will be even less and would give preferential breakage sideways, limiting breakage to the main free face.

Looking mid length down the hole/ring (14m of 28m) the following geometry is present Figure 3. Hole 2 now has a very short distance 1.3m to the crater established by the first hole and the distance to the main free face is greater than double that.

This infers that, in the core of the ring, energy will blast mainly sideways to the crater of the blast hole before it. There are still more of the blast hole closer to the collar that has geometry worse than this. This blasting sideways creates fines, as the short distance sideways vents the energy of the hole preferentially. It also creates oversize, as the preferential venting will put damage through the larger burdened area which will limit the energy of future blast holes to fragment it well Figure 3 region of oversize).

This geometry is repeated for all blast holes in the ring causing a region of fines and oversize in the core zone (centre). Preferential drawing occurs depending on the material available to fill the voids created by the digging of the loader. This could be either waste from above or behind the ring, as well as fines created in the ring from short burdens. It is measured that 10-15% of the holes misfire and this will create oversize and areas of tightness in the misfire areas but overly fine material in the other areas.

Designed with a recommended spacing-to-burden ratio of greater than 1.3 (Bull 2000) such as a ratio of 1.4 (Marklund 1976), gives better geometry for more uniform fragmentation. This

can be seen in the following pattern which has a burden of 2.6 m with a spacing of 3.6 m (Figure 4).

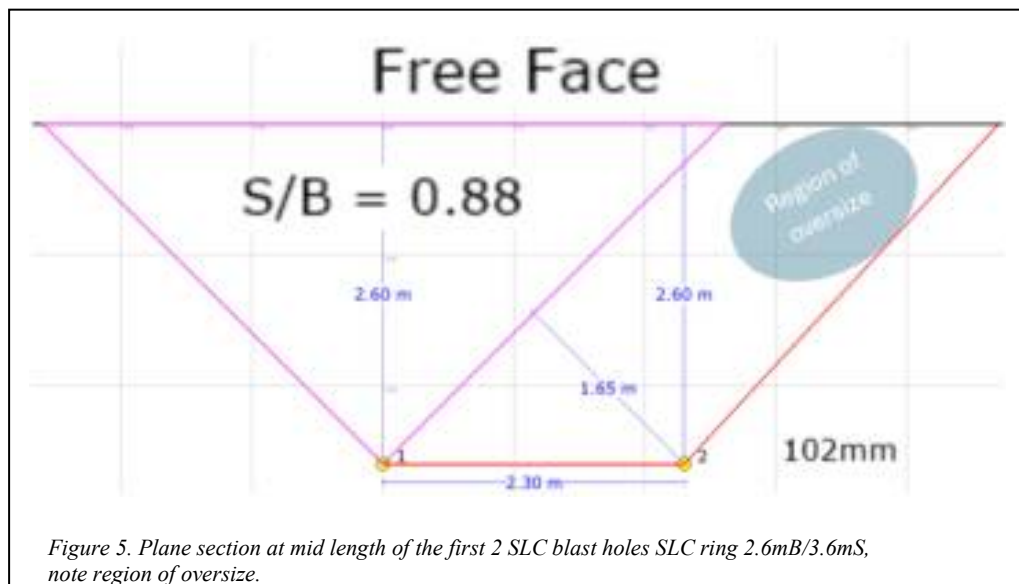
The first hole in Figure 4 shows the same free face geometry after the first blasthole for the second hole (and subsequent holes). The minimum burden to all free faces is almost even at 2.55 and 2.6 m and so more uniform fragmentation should result.

Mid length geometry still narrows, as it is compromised by the 'radial fan' configuration of the blast ring. At the point in study (14 m), the spacing is 2.3 m, but the minimum burden distance is 1.65 m. This is less than the 2.6 m to 'front' of the ring but is better than the original case (2.6 m x 3 m pattern was 1.26 m) and so the fragmentation should be more uniform. The reduced burden will result in fragmentation that will be finer, unless the energy of the explosives is reduced.

Subtek™ 4D™ (Evans, 2025) offers variable RBS within a single hole, enabling energy to be tailored to burden variations. Higher energy can be applied at the toe—where burden and spacing are greatest—while reducing energy in the core and near the collar to minimize fines and brow damage. This helps prevent brow breakback, which can hinder equipment access and lead to more diluted secondary recovery from lower levels.

6 POWDER FACTOR

Sublevel Caving (SLC) is a high-intensity drill-and-blast method where all ore is broken in



single-ring firings in confined blasting. Caving waste acts as artificial burden. Typical powder factors range from 0.5–1.0 kg/t for longitudinal SLC and 0.4–0.8 kg/t for transverse layouts. Lower powder factors improve recovery and reduce dilution (Campbell 2018), with adjustments made via burden and explosive density. A trial replacing ANFO (0.95 g/cc) with emulsion (1.15 g/cc) increased the powder factor by ~20%, resulting in finer fragmentation, narrower draw zones, and reduced recovery. Excess energy also compacted broken ore, hindering gravity flow, highlighting the need to balance energy with burden.

At Ernest Henry Mine, Campbell's work hasn't led to increased burden or reduced explosive density to improve recovery. This may be due to ongoing misfires with conventional initiation systems still in use. Drill deviation and geological variability also affect the reliability of rings breaking to the cave. Lowering the powder factor, whether by increasing burden, reducing explosive density, or due to misfires, can compromise breakage, severely impacting ring recovery and drill drive productivity. Such unplanned issues, including brow breakback and redrilling, significantly reduce both productivity and profitability.

Changing the initiation system, to the more reliable WebGen™ wireless initiation mostly solves the misfire issue and as such the reliability of optimising the pattern can be undertaken without the major impact of 10-15% misfires.

Optimisation can be enhanced using emulsion decks with varying Relative Bulk Strength (RBS), delivered via Subtek™ 4D™ Velcro™ for non-reactive ground and Subtek™ 4D™ Eclipse™ for reactive ground. Unlike traditional methods based on density, this system loads explosives by RBS, enabling rapid, programmable adjustments. It ensures accurate charge columns without gassing tables, top-ups, or 'suck-outs'. With an expanded RBS range of 50–170 %, it offers greater flexibility and energy control, providing a significant advantage in SLC optimisation.

Figure 6 shows a conventional SLC ring with 102 mm blast holes, 21 m drill drive spacing, and 25 m level spacing. Holes are charged with 1.1 g/cc bulk emulsion, assuming a 2.6 m burden and 2.7 t/m³ rock density. Uncharged collars, up to 9 m in some holes, are used to reduce energy concentration in the ring centre. This design raises the core powder factor to 0.58 kg/t, 46% higher than the toe's 0.40 kg/t. The brow region is similarly high at 0.57 kg/t, leading to inefficient breakage due to geometric constraints (see Figure 2 in DRILLING DESIGN).

By increasing spacing to 3.8 m and burden to 2.8 m, and applying tailored energy and reliable initiation, powder factors across all zones are balanced. Further gains are possible by adjusting timing to reduce core fragmentation and enhance breakage at the toes and wings. As shown in Figure 7, core and brow powder factors drop by 34%, toe increases by 4%, and burden rises 8%.

Toe
Powder
factor
0.40 kg/t

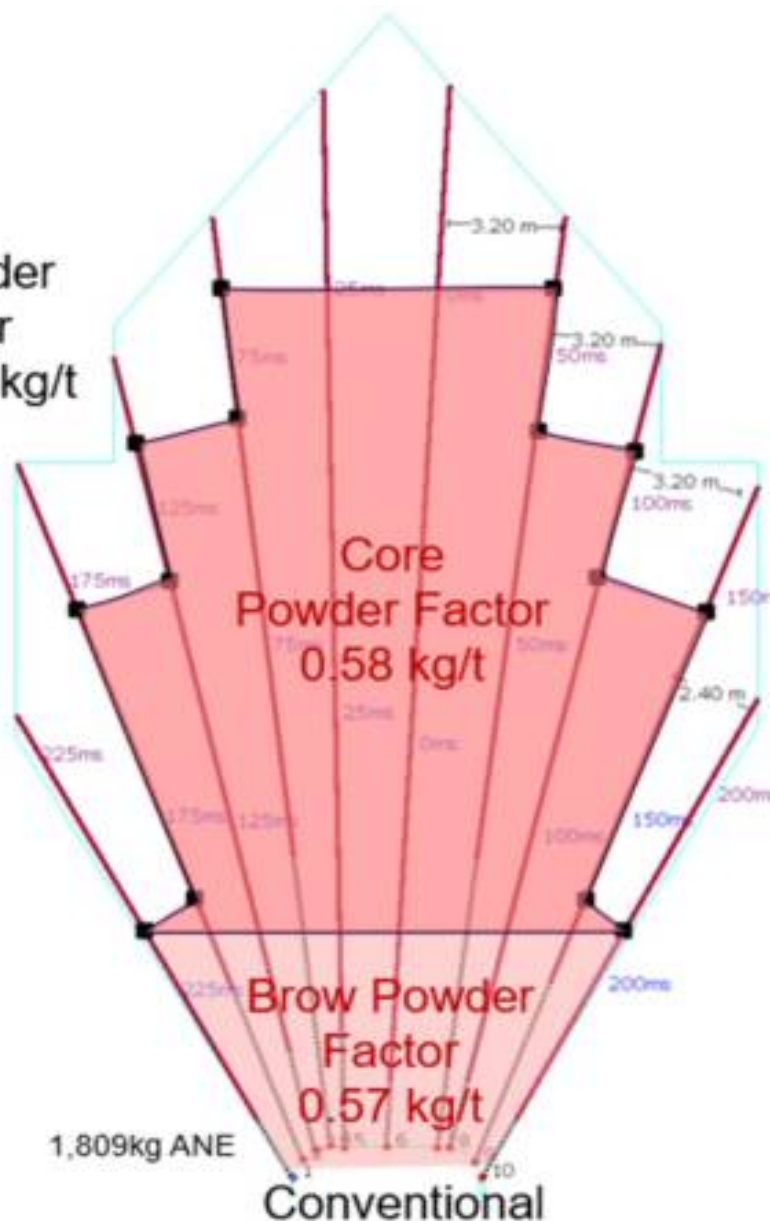


Figure 6. Conventional 102 mm SLC ring (burden 2.6 m) showing powder factors of 3 areas.

Overall, explosive use is reduced by ~20% and drilling by ~17%.

7 CARBON EMISSIONS

Estimating the carbon emissions for a large sublevel cave mining operation involves

considering several factors, including the type of equipment used, energy sources, and specific mining practices. Scope 1 Emissions are direct emissions from sources owned or controlled by the mining operation, such as diesel used in mining equipment. For sublevel caving, significant emissions come from drilling, blasting, and

hauling activities. Reducing drill and blast by 18% will be a significant saving of these scope 1 emissions while increasing primary recovery and reducing dilution.

8 CONCLUSION

Sublevel caving (SLC) remains a critical method for extracting ore in massive deep, steeply dipping orebodies, but its inherent challenges, particularly those related to fragmentation, misfires, and dilution, demand innovative solutions to ensure sustainable and efficient operations. This paper has demonstrated that conventional blasting practices, while functional, often lead to suboptimal

fragmentation, excessive fines, and high misfire rates, all which compromise ore recovery and increase dilution.

The integration of wireless initiation systems, such as Orica’s WebGen™, represents a transformative advancement in SLC operations. By significantly reducing misfire rates, these systems enable more reliable and precise blasting, which in turn allows for optimised ring designs with improved spacing-to-burden ratios. Coupled with the use of variable energy explosives like Subtek™ 4D™, which tailor energy distribution within each blast hole, mines can achieve more uniform fragmentation, reduce over-compaction, and enhance gravity flow behaviour

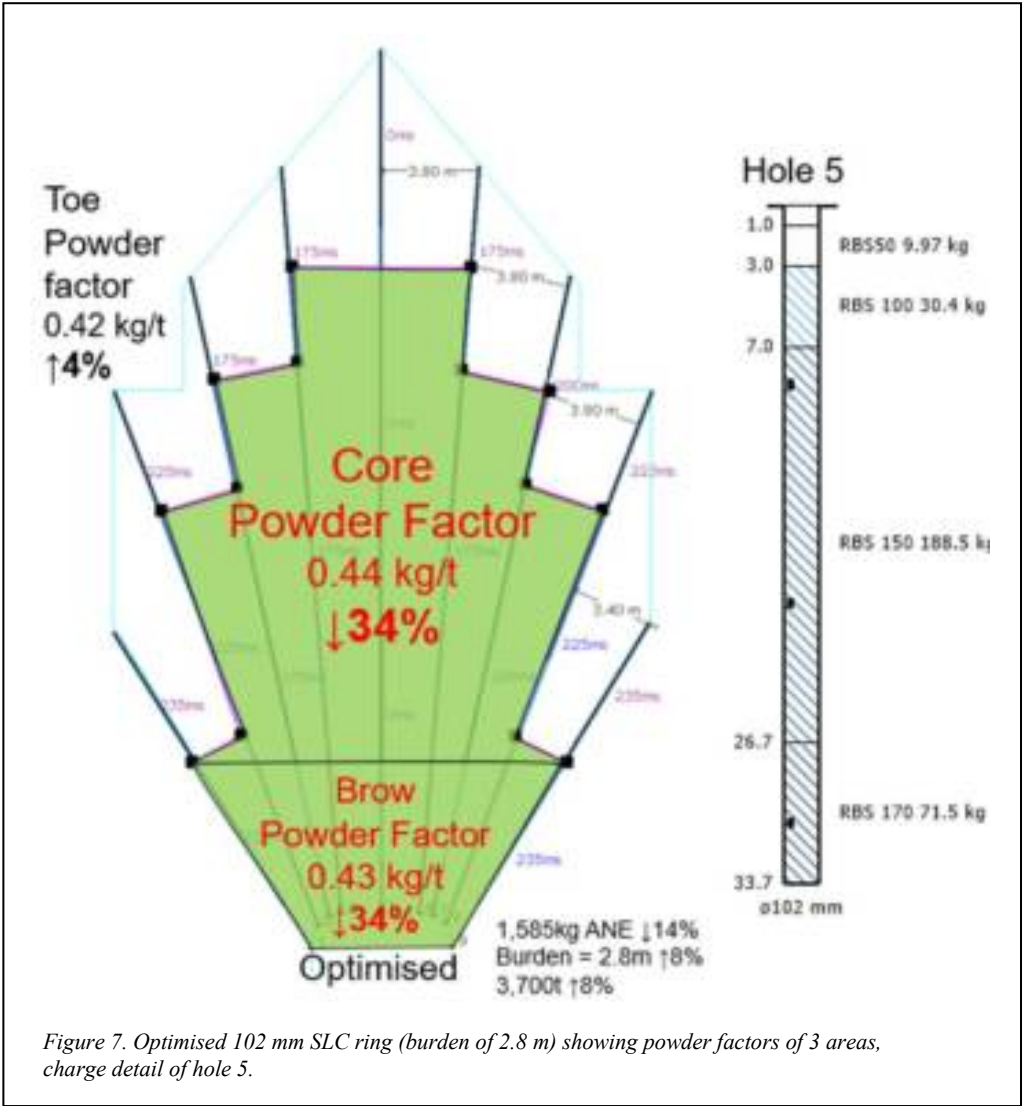


Figure 7. Optimised 102 mm SLC ring (burden of 2.8 m) showing powder factors of 3 areas, charge detail of hole 5.

(patent pending).

These advancements not only improve ore recovery and reduce dilution but also contribute to lower powder factors and reduced carbon intensity, key metrics in the pursuit of more sustainable mining practices. By embracing these technologies, SLC operations can lower their cut-off grades, extend mine life, and reduce environmental impact, positioning themselves for long-term viability in an increasingly sustainability-focused industry.

9 ACKNOWLEDGEMENTS

The author sincerely thanks Amanda Haas Lovitt JP. for her unwavering support, and Orica Ltd for enabling the review of numerous blast outcomes and for supporting the development and presentation of this work. Gratitude is also extended to the engineers whose insights helped shape this research.

REFERENCES

- Bull, G. 2000. Sublevel caving-today's dependable low-cost 'ore factory'. *3rd International Conference and Exhibition on Mass Mining 2000* (pp. 537-556). Brisbane: AusIMM.
- Bullock, R.L. 2001. Underground mining methods: Engineering fundamentals and international case studies. *Littleton, CO: Society for Mining, Metallurgy, and Exploration, Inc.*
- C te Kloot, Z.L. 2017. Improving safety through technology: Orica wireless electronic blasting system trials at Ernest Henry Mine. *13th AusIMM Underground Operators' Conference* (pp. 39-49). Gold Coast: AusIMM.
- Campbell, A. 2018. Effects of blast ring burden and explosive density on fragmentation and ore recovery in sublevel cave mines. *Caving 2018* (pp. 457-470). Vancouver: Australian Centre for Geomechanics.
- Campbell, A. 2022. A global review of recovery, dilution and draw control in sublevel cave mines. *Caving 2022* (pp. 909-925). Perth: Australian Centre for Geomechanics.
- Clark, S.S. 2013. Sub- level caving: engineered to perform. *The Mining Magazine*.
- Evans, S. 2025. Subtek™ 4D™ – Optimised blasting performance through the application of new underground bulk explosive technology. *Underground Operators Conference 2025* (pp. 144-154). Adelaide: AusIMM.
- Player, J. & Perara, V. 2008. A back analysis of dilution and recovery in longitudinal sublevel. *MassMin 2008* (pp. 575-584). Luleå, Sweden: Luleå University of Technology.
- Kazakidis, V. 2015. Design & operating principles in caving methods. *COSMO conference. Ontario: Laurentian University School of Engineering*.
- Kvapil, R. 1998. Gravity flow of broken rock in sublevel caving. *Elsevier Science Publishers*.
- Marklund, I. 1976. The influence of blasting on flow development. *Sublevel Caving Seminar* (pp. L1-L9). Kiruna: Swedish.
- Vauhkonen, V. 2022. Field measurements of brow break and misfires in sublevel caving at Kemi Mine. Oulu: University of Oulu, *Mining Engineering and Mineral Processing*.

[RETURN TO CONTENTS PAGE](#)

*9. Sustainability and
environmental topics in drilling
and blasting*

Benefits of Shell GTL fluids reducing toxic emissions in emulsions explosives

M. Rivera-Torrente

Energy Transition Campus Amsterdam, Shell, Amsterdam, The Netherlands

N. Petropoulos & C. Rodriguez-San Miguel

Division of Mining and Geotechnical Engineering, Luleå University of Technology, Luleå, Sweden

J. Russell, M. Perryman, R. Hunt & T. Wilkinson

Infineum Ltd, Milton Hill Business & Technology Centre, Abingdon, United Kingdom

A. Pienimäki & V. Klippmark

LKAB Kimit AB, Kiruna, Sweden

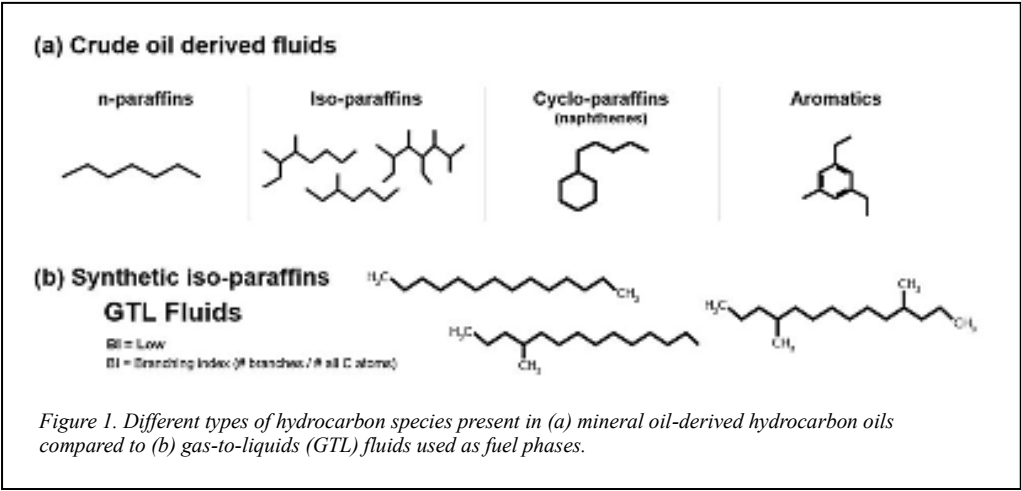
ABSTRACT: Toxic fume formation, particularly CO, is a major concern in ANFO and emulsion explosives, impacting both safety and operational costs. This study demonstrates that using ultra-clean, synthetic gas-to-liquids (GTL) fuel phases formulated with Infineum PIBSA emulsifiers can reduce CO emissions by 20% compared to conventional mineral oils. The reduction is attributed to the chemical composition of GTL fuels—primarily isoparaffins—affecting emulsion properties like surface tension, droplet size, and combustion behaviour. Tests were conducted in a controlled blasting chamber at the Swedish Blasting Research Centre (Swebrec), ensuring consistent and replicable results. Emulsions were manufactured and tested multiple times, showing minimal variation in emissions. The findings highlight the importance of fuel phase selection in optimising explosive performance, extending shelf life, and toxic emissions. Additionally, the study underscores the value of controlled testing environments for accurately assessing post-blast gas emissions.

1 INTRODUCTION

Many different hydrocarbon sources have been used in the literature, ranging from oils like diesel, naphthenic, mineral base oils or even white oils. This selection is commonly related to availability and price of reclaimed fluids directly on-site, rather than performance of the emulsion product. This variability can impact the performance and stability of the emulsion explosives. For instance, the impact of post-blast toxic emissions,

particularly carbon monoxide (CO) and nitrogen oxides (NO_x), is a significant concern in the use of emulsion explosives. Underground (UG) operations are strongly impacted by the formation of CO. Post-blast delay strongly impacts economic performance, and reducing the power needed for an effective air flow in mine ventilation systems (even by only 20%) can have significant impacts in terms of OPEX and CAPEX for the operation.

A consistent methodology has been put forward in the past by the European Union (EU) in the



standard, whereby a chamber with a volume higher than 15 m³ and at least 1.0 kg of explosive in glass tubes are detonated with appropriate probes and sensors to determine CO and NO_x. The Swedish Blasting Research Centre (Swebrec) has developed a test rig for that purpose and has studied the influence of many different aspects in the past, such as the presence of AN prills, aluminium flakes or charge diameter. Another

study has shown the benefit of using GTL fluids in fuel phases for the formulation of bulk UG emulsions. (Rivera-Torrente *et al.* 2025) GTL fluids are manufactured from natural gas by using the Fischer–Tropsch process at the Shell Middle Distillate Synthesis (MDS) plant in Malaysia and the Pearl GTL plant in Qatar. GTL liquids contain only very small amounts of cyclic paraffins (<1-2 %m/m) and aromatics (<300 ppm) and are almost

Table 1. Physical properties of the hydrocarbon fluids used to make the fuel phases studied.

Property	Method	Fluid			
		Unit	G85 ^[a]	G100	Technical Mineral Oil
Kinematic Viscosity at 40 °C	ASTM D445	mm ² /s	2.9	3.5	4.5
Initial Boiling Point	ASTM D86	°C	198	225	265
Final Boiling Point	ASTM D86	°C	343	343	330
Flash Point	ASTM D93	°C	86	104	135
Pour Point	ASTM D97	°C	-27	<-16	-6
Density at 15°C	ASTM D4052	g/mL	0.770	0.789	0.813
Linear paraffins	GC	%m/m	16	14	10
Iso-paraffins	GC	%m/m	82	85	44
Cycloparaffins	GC	%m/m	<2	<1	35
Aromatics	GC	%m/m	<0.03	<0.03	12
Biodegradability	OECD 301F	-	>60%		<60%

[a] Physical properties as reported in the technical data sheet provided by the supplier.

odourless, in contrast to many commercial fuels with high amounts of aromatics or naphthenes (between 5 and 50 wt% of cyclic paraffins).

Although ANFO and emulsion explosives follow fundamentally different phenomena during the chemical reaction, it is worth mentioning that Fischer-Tropsch derived fuels (including GTL), show lower NO_x, CO and particulate matter (PM) emissions in engine combustion processes. In this work, we have studied the impact in toxic CO and NO_x emissions of using different highly isoparaffinic, low branching index fuel phase, G85 and G100. A technical mineral oil with naphthenics (35%/m/m) and aromatics (12%/m/m) was used as a standard market product to compare. Two campaigns were completed, in October 2024 and May 2025, the fuel phase was preblended at the same location, but two different laboratories prepared the matrix mixing it with the oxidizer. For each composition, four to six shots were fired to ensure the robustness of the dataset.

2 EXPERIMENTAL SECTION

2.1 Materials and methods

Shell GTL G85 and GTL G100 are synthetic, paraffinic fluids manufactured at the Pearl GTL and Shell MDS plants, in Qatar and Malaysia,

respectively. PIBSA-amine concentrated emulsifier Infineum E116 (Infineum Ltd, United Kingdom) with viscosity of 2100 cSt at 40 C. The technical mineral oil was Pilot® 140, supplied by Haltermann Carless (Speyer, Germany). The sensitizer consisted of glass spheres Microballoons® K20, supplied by 3M Co (Minnesota, USA).

2.2 Matrix preparation

Emulsion matrices were prepared by pre-blending the fuel phases with the emulsifier according to a standard formulation (Arvanitidis, 2004), known as E682. The E682 emulsion explosive is dual salt composition with Ammonium nitrate (NH₄NO₃) and Sodium nitrate (NaNO₃), and emulsifier used was Infineum E116 (supplied by Infineum UK Ltd, United Kingdom).

The oxidiser was gently poured into the fuel phase while hot at 85 – 90 °C and refined, then transported to the firing site unsensitised. The matrices were sensitised directly on the field by blending the required amount of glass microballoons with the emulsion at room temperature (+5 °C). A Hobart mixer with wire whisk attachment was used for homogenous mixing for 3 minutes. It is worth noting that in the matrix preparation during the second campaign

Table 2. Weight ratios of the components used for the preparation of the emulsions used, as it has been previously published for E682 (Arvanitidis 2004).

Component	Ingredient	Blend A1	Blend A2	Blend B1	Blend B2	Blend B3
Oxidizer	Water	14.5	14.5	14.5	14.5	14.5
	Sodium Nitrate	10.9	10.9	10.9	10.9	10.9
	Ammonium Nitrate A ^[a]	65.3	65.3			
	Ammonium Nitrate B ^[a]			65.3	65.3	65.3
Fuel	Technical Mineral Oil	4.5		4.5		
	GTL G85		4.5		4.5	
	GTL G100					4.5
	Emulsifier (Infineum E116)	1.5	1.5	1.5	1.5	1.5
Sensitizer	Glass Microballoons 3M®K20	3.3	3.3	3.3	3.3	3.3
	Total weight %	100.0	100.0	100.0	100.0	100.0

^[a] Blends A1, A2 & A3 blends were prepared and tested in 2024. Blends B1, B2 & B3 were prepared and tested in 2025. Both ‘Ammonium Nitrate A’ & ‘Ammonium Nitrate B’ were technical grade and sourced close to the time of use.



Figure 2. Photograph of the charge in the blasting chamber ready for detonation.

(Blends B) in 2025, fuel was balanced by volume not by mass, while technical mineral oil has a density of 0.81 g/mL compared to GTL fuels which at ca. 0.78 g/mL. This was done because the mixing rig works on a volume basis.

2.3 Detonation and gas emission monitoring

Blasting chamber: the trials have been carried out 9-13th of June 2024 on FOI test plant at Grindsjön (Sweden) in a 35 m³ blasting BOFORS chamber. The chamber for maximum 5 kg TNT- equivalent explosives was prepared for single shot measurements. Fumes (ppm CO and NO_x) from detonation of oxygen balanced emulsion

explosives sensitised with glass micro spheres in cylindrical glass confiners were measured up to about 60 minutes directly (ca. 1 minutes) after detonation. The glass confiners were used to visually see the explosive homogeneity and to avoid reaction products from the confiner. Carbon dioxides CO₂ % were calculated for O₂ values below 20.5%.

The NSP711 booster based on PETN explosive was a 10 gram paste applied at the end of the detonating chord. Standard No. 8 pyrotechnic detonators (supplied by FORCIT) were used.

Gas detection system: The monitoring instruments for the gases was a flue gas analyser (Ecom EN2), the instrument is equipped with a

Table 3. Summary of the detonations and gas detection data of the first campaign.

Blend (Fuel phase)	Value ^[a]	Amount, g	Density, g/cm ³	VoD, m/s	CO, ppm ^[b]	NO, ppm ^[b]	NO ₂ , ppm ^[b]	NO _x , ppm ^[b]	T _{gas} , °C
Blend A2 (GTL 85)	☞	1162.7	1.2	5318.2	188.5 (235.8)	15.8 (21.0)	0.7 (1.3)	15.8 (32.7)	23.8
	σ	8.57	0.06	139.02	8.44 (10.40)	0.69 (1.00)	0.47 (0.94)	0.69 (1.37)	2.3
Blend A1 (Technical Mineral Oil)	☞	1156.3	1.2	5398.7	243.0 (303.7)	14.2 (19.2)	0.3 (0.7)	14.2 (29.3)	24.7
	σ	19.88	0.01	139.60	5.97 (7.56)	0.37 (0.37)	0.47 (0.94)	0.37 (0.75)	2.1
Booster	-				15 (19)	6 (8)	0 (12)	6 (12)	22

[a] Values represent the mean value $\bar{x} = \frac{1}{n} \sum_{i=1}^n x_i$ and the standard deviation $s = \sqrt{\frac{1}{n-1} \sum_{i=1}^n (x_i - \bar{x})^2}$ of the six shots fired with the emulsions made with each fuel phase. The PETN booster was fired once to record baseline date. [b] Values in brackets are mg/m³.

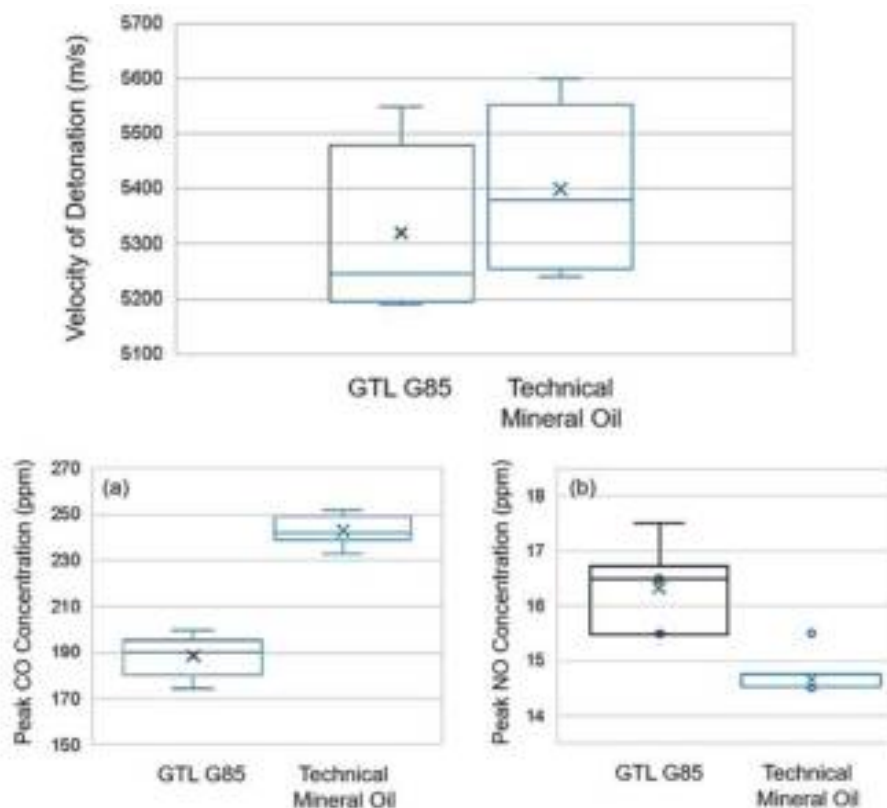


Figure 3. Box plots comparing the (top) Velocity of Detonation (VoD), and (bottom) peak concentration of (a) carbon monoxide, CO and (b) nitrous oxide, NO; of the six shots fired of each emulsion. Boxes indicate 95% confidence interval of the values recorded, while the line indicates the median, and the X mark the average value.

filter and a pump which sucks with 1.6 litre per minute gas flow. The data sampling was set to 1 Hz (1 sample per second). The device is equipped with electrochemical sensors with range $\pm$ accuracy and resolution; O_2 : $0-21 \pm 0.3$ vol. %, and 0.1 %, CO: $0 - 4000 \pm 20$ ppm and 1 ppm, NO: $0 - 5000 \pm 5$ ppm and 1 ppm, NO_2 : $0 - 1000 \pm 5$ ppm and 1 ppm and the IR sensor for CO_2 : $0 - 20$ % with resolution of 0.1 vol. %. The instrument was freshly calibrated and serviced.

The gas probe was inserted after $\sim 1 - 1.5$ minutes after the detonation and measured for 1 hour period. The VoD was calculated based on the slope of the recording from the Microtrap.

3 RESULTS AND DISCUSSION

Figures 3 and Table 3 show the values recorded for the blasts of the first campaign in 2024, including

the calibration with the booster. The data in Table 3 and presented in box plots in Figure 3 show the uncorrected values (*i.e.* the detonation data of the booster has not been subtracted). The factors affecting toxic fume formation have been previously investigated at the Swebrec and others, but the fuel type has been less commonly studied. surface tension or polarity of the hydrocarbons dictate how oil and oxidiser phase interact, hence, droplet size of the emulsion. Mahadevan (2013) explained that smaller droplet size usually shows higher velocity of detonation.

A comparison with the reference non-commercial product (E682) can be made, of which the concentration values after blasting in the same chamber are 185, 23 and 0 ppm, CO, NO and NO_2 , respectively, at 0% oxygen balance, as reported by Nyberg (Nyberg 2015). Figure 4 shows the results of the second campaign carried out in May 2025.

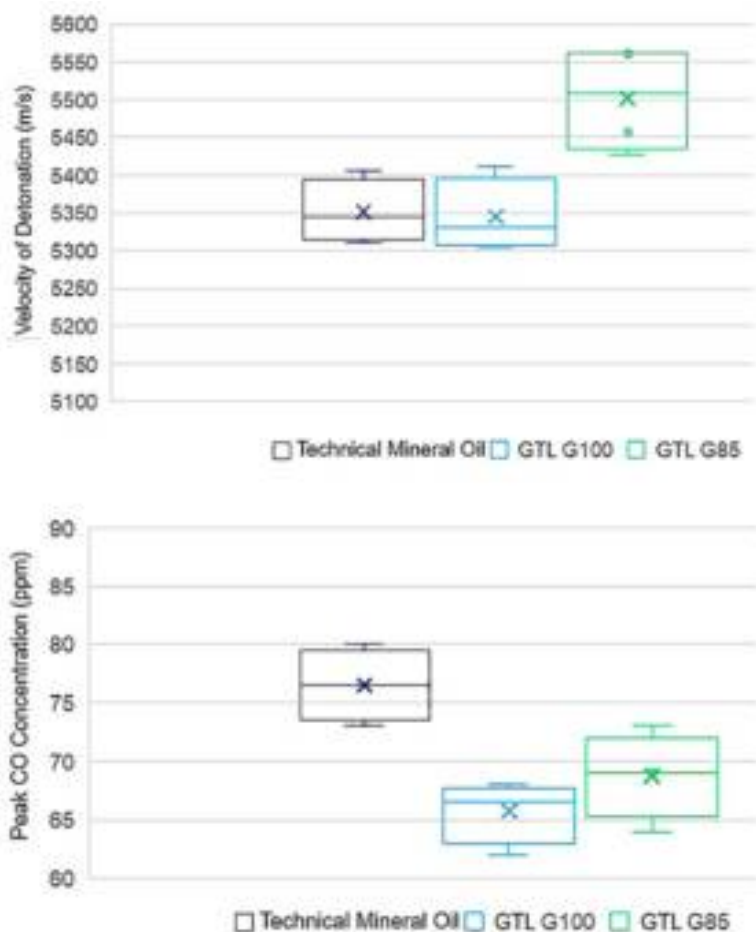


Figure 4. Box plots comparing the (top) Velocity of Detonation (VoD), and (bottom) CO peak concentration of the four shots fired of each emulsion from study 2. Boxes indicate 95% confidence interval of the values recorded, while the line indicates the median, and the X mark the average value.

In agreement with the first round of tests, GTL fuel compositions showed *ca.* 20% lower peak CO concentration, regardless of the oil. A small difference was observed, with G100 producing less CO emissions than Shell GTL G85.

Hence, the oxygen balance has been slightly shifted towards the formation of NO_x (deficient). Thermochemical calculations have confirmed this in the literature, as well as the effects of other variables, like packing, aluminium, dust, or impurities. This explains that the values registered in that case were higher for GTL fuels, in the range of 80 – 90 mg/m^3 , compared to the Technical Mineral Oil, which produced around 45 mg/m^3 . This may also be related to the higher

VOD observed for GTL G85, reaching an average 5500 m/s, compared to the 5350 m/s of both GTL G100 and the Technical Mineral Oil.

Furthermore, the source of ammonium nitrate was different for the second campaign, which could be another cause for this difference. As explained, the matrices prepared in 2025 (Blends B) were prepared with technical AN from another supplier. Further, the matrices were adjusted on a volume basis, meaning that the oxygen balance is slightly deficient (more oxidiser) for the GTL fuel phase-containing matrices. It is worth noting that even though both CO and NO_x are considered toxic, dangerous gases, NO_x may react with soil water and can be identified by odour or visually.

Hence, it can be argued that CO, odourless, tasteless and easily migrating through formation layers may be more lethal. Bahrami *et al.* (2019) at NIOSH showed that the threshold limit value (TLV) for CO (25 ppm) was almost always exceeded over several months, while NO was only sporadically beyond TLV. CO took much longer time to be diluted by ventilation air and decrease to safe operation levels than NO_x. Another aspect to consider is that the Swebrec blasting chamber is such a clean environment, that NO_x formation under field conditions could be higher, as it is typically more affected by variables like the presence of water. This has been shown as less impactful in post-blast CO formation. Nevertheless, safety must always be the top priority and slight negative oxygen balances are often favoured over perfect oxygen balance to limit NO_x emissions, there are strict thresholds for NO_x release and high emissions can be reportable and lead to downtime in many jurisdictions. This set of data shows clearly that the benefit for GTL is that at equal oxygen balance you would expect equivalent NO_x and lower CO.

Stewart reported that toxic gas concentration can be reduced air dilution and distance in UG, also pointing out that CO is usually more difficult to clear (Stewart 2014). Assuming the same volume, air flow, dilution and powder factors, one may compare the time to reach safe CO levels based on Equation 1:

$$G_1 = \frac{G_2}{e^{\frac{t}{\tau}}} \tag{1}$$

For instance, blasts in the work by Bahrami showed an average peak concentration of 40 ppm, and 1.25 h decay time to reach the safety TLV of 25 ppm for CO. Based on the data showed above, *i.e.* Shell GTL can reduce the CO formation in 20%, hence, can decrease the re-entry. Modelling such exponential decay, would result in decreasing the time to reach CO < 25 ppm of *ca.* 5 - 10 min. Further, it is worth bearing in mind the sustainability score of GTL fluids. GTL fluids show an improved biodegradability (>60%) compared to mineral oil products as per OECD 301F standard tests. Another benefit is their limited impact on the aquatic environment, fish early life stage toxicity and algal growth inhibition. In addition, GTL fluids exhibit a very low and eye or skin irritation properties. These features could result in important economic benefits for UG operations in terms of non-productive time, reportable and non-reportable safety incidents, injuries and loss of license to operate.

4 CONCLUSIONS

In summary, we have compared three different fluids in bulk emulsions in terms of post-blast toxic emissions. Shell GTL can effectively reduce the amount of post-blast CO formed by 20% compared to technical mineral oils in the market. This is ascribed to the purely paraffinic nature of the fluid. It is worth noting that the blasts were carried out in a clean environment, keeping most variables unchanged. Further, the lower peak CO concentration with Shell GTL was observed in two different campaigns. This has the potential of helping the industry reduce costs through improved explosive emulsion products, primarily for underground operations.

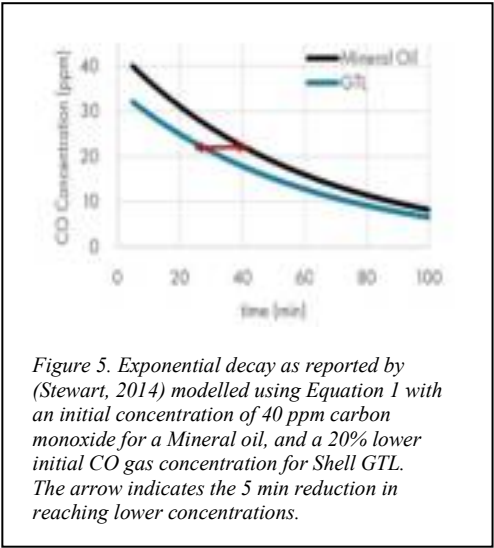


Figure 5. Exponential decay as reported by (Stewart, 2014) modelled using Equation 1 with an initial concentration of 40 ppm carbon monoxide for a Mineral oil, and a 20% lower initial CO gas concentration for Shell GTL. The arrow indicates the 5 min reduction in reaching lower concentrations.

5 ACKNOWLEDGEMENTS

Ulf Stenman (Luleå University of Technology) is gratefully acknowledged for his help and support during the detonation tests. We appreciate the help from Nick Lokum, Martin Lelieveld, Bayan Alhaddar and Ferry de Kruijff (Shell) analysing the fuel properties. We are thankful to Don Van Slyke (Shell) for helping set up this study.

DISCLAIMER

Copilot® AI was used in the study for summarising text, improving readability and conciseness of this text. No scientific nor technical conclusions or calculations are reported or derived from AI or automated algorithm processes.

REFERENCES

- Ali, F. *et al.* 2024. Investigation and mapping of toxic fumes produced by detonation of ANFO explosives in underground space. *Propellants, Explosives, Pyrotechnics.*, 2024., p. 49: e202400002.
- Arneson-Westbrook, L. *et al.* 2020. Terrestrial toxicity of synthetic gas-to-liquid versus rude oil-derived drilling fluids in soil. *Environmental Toxicology and Chemistry.*, 2020., <https://doi.org/10.1002/etc.4658> : Vol. 39., pp. 721–730.
- Arvanitidis, I., Nyberg, U. & Ouchterlony, F. 2004. The diameter effect on detonation properties of cylinder test experiments with emulsion E682. - *Stockholm, Sweden: SveBeFo Report 66*; ISSN 1104-1773, 2004.
- Bachman, H.E. & Totman R.S. 1986. Dry ammonium nitrate blasting agents. (*ExxonMobil Chemical Patents Inc*): US4736683A. – Priority Date: 05 08 1986.
- Bahrami, D., Yuan, L. & Rowland J.H. 2019. Impact of post-blast toxic emissions of CO and NO_x in surface mining. *Mining, Metallurgy & Exploration.*, 2019., <https://doi.org/10.1007/s42461-019-0058-6> : Vol. 36., pp. 513–521.
- Boogaard, P.J. *et al.* 2017. Toxicological and ecotoxicological properties of gas-to-liquid (GTL) products. 1. *Mammalian toxicology. Critical Reviews in Toxicology.*, 2017., <https://doi.org/10.1080/10408444.2016.1214676> : Vol. 47., pp. 121–144.
- Brown, D.M. *et al.* 2020. Biodegradability assessment of complex, hydrophobic substances: Insights from gas-to-liquid (GTL) fuel and solvent testing. *Science of The Total Environment.*, 2020., <https://doi.org/10.1016/j.scitotenv.2020> : Vol. 727., p. 138528.
- Carrillo, J-C. *et al.* 2022. GTL synthetic paraffin oil shows low liver and tissue retention compared to mineral oil. *Food and Chemical Toxicology.*, 2022., Vol. 159., p. 112701.
- Cranney, D.H. & Hansen, J.R. 1998. Gassed emulsion explosive (*Dyno Nobel Inc*): US6022428A. Priority Date: 10 02 1998.
- Foudazi, R. *et al.* 2015. Physical chemistry of highly concentrated emulsions. *Advances in Colloid and Interface Science.*, 2015., Vol. 220., pp. 78–91.
- Harris, M.L. & Mainiero, R.J. 2008. Monitoring and removal of CO in blasting operations. *Safety Science.*, 2008., 10: Vol. 46., pp. 1393–1405.
- Harris, M.L., Rowland, J.H. & Mainiero, R.J. 2003. Toxic fume comparison of a few explosives used in trench blasting. *International Society of Explosives Engineers.*, 2003., Vol. 2., pp. 319–336.
- Hoek, A. 2005. The Shell GTL Process: Towards a World Scale Project in Qatar. *Chemie Ingenieur Technik*, 2005., <https://doi.org/10.1002/cite.200580047>: Vol. 77., pp. 1172–1172.
- Leahy, J.G. & Colwell, R.R. 1990. Microbial degradation of hydrocarbons in the environment. *Microbiological Reviews.*, 1990., <https://doi.org/10.1128/mr.54.3.305-315.1990>: Vol. 54., pp. 305–315.
- Mahadevan, E.G. 2013. Ammonium nitrate explosives for civil applications, slurries, emulsions and ammonium nitrate fuel oils [Book]. - *New York: WILEY-VCH Verlag GmbH & Co. KGaA*, 2013.
- Mainiero, R.J., Harris, M.L. & Rowland, J.H. 2007. Dangers of toxic fumes from blasting. *Proceedings of the 33rd Annual Conference on Explosives and Blasting Technique, International Society of Explosives Engineers.*, 2007., pp. 1–6.
- Malkin, A.Y. *et al.* 2004. Effect of droplet size on the rheological properties of highly concentrated w/o emulsions. *Rheologica Acta.*, 2004., Vol. 43., pp. 584–591.
- Marlow, J.M. & Bush, J.H. 1998. Thickened emulsion compositions for use as propellants and explosives (*Lubrizol Corp*): US5936194A. – Priority date: 18 02 1998.

- Martel, R. *et al.* 2011. Carbon monoxide poisoning associated with blasting operations close to underground enclosed spaces. Part 1. CO production and migration mechanisms. *Canadian Geotechnical Journal.*, 2011., Vol. 41., pp. 371-382.
- McMillian, M.H. & Gautam, M. 2001. Combustion and emission characteristics of Fischer-Tropsch and standard diesel fuel in a single-cylinder diesel engine. SAE Transactions: *Journal of Fuels and Lubricants.*, 2001., 4: Vol. 110., pp. 1659-1674.
- Nyberg, U. *et al.* 2015. Short time measurements of toxic fumes from detonation of emulsion explosive, *ISSN 1653-5006, 2015:1 - Stockholm, Sweden: SweBrec*, <https://ltu.diva-pot.al.org/smash/get/diva2:997234/FULLTEXT01.pdf>, 2004.
- Onederra, I. *et al.* 2012. Understanding main causes of nitrogen oxide fumes in surface blasting. *Mining Technology.* 2012;121(3):151-159. doi:10.1179/1743286312Y.0000000019, 2012.
- Overtom, R., Fabricius, N. & Leenhouts, W. 2009. Shell GTL, from bench scale to world scale. In: Alfadala HE, Rex Reklaitis GV, El-Halwagi MM (eds) *Proceedings of the 1st Annual Gas Processing Symposium*. Elsevier, Amsterdam., 2009., Vol. <https://doi.org/10.1016.>, pp. pp 378-386.
- Perry, J.J. 1979. Microbial cooxidations involving hydrocarbons. *Microbiological Reviews.*, 1979., Vol. 43., pp. 59-72.
- Pollack, R.A. 2001. Thickened water in oil emulsion composition (*Lubrizol Corp*): *US6939420B2*. Priority Date: 01 11 2001.
- Rivera-Torrente, M., Jong, Y-T. & Araos, M. 2025. Reducing the fuel phase content with synthetic gas-to-liquids (GTL) *Fluids in Emulsion Explosives. Propellants, Explosives, Pyrotechnics*, accepted, 2025.
- Sapko, M. *et al.* 2002. Chemical and physical factors that influence NOx - production during blasting - exploratory study. *Proc. 28th Conf. on 'Explosives and blasting technique'*, ISEE., 2002., pp. 317-330.
- Stewart, C.M. 2014. Practical prediction of blast fume clearance and workplace re-entry times. *10th International Mine Ventilation Congress, IMVC2014.*, Sun City, Johannesburg, South Africa: 2014.
- Suceska, M. *et al.* 2022. Prediction of concentration of toxic gases produced by detonation of commercial explosives by thermochemical equilibrium calculations. *Defence Technology.*, 2022., 12: Vol. 18., pp. 2181-2189.
- Union European *EN 13631-16:2004 - Explosives for civil uses - High explosives - Part 16: Detection and measurement of toxic gases: 2014/28/EU Explosives for civil uses*, 2004.
- Wang, Y. & Wang, X-G. 2017. Crystallization of interphase droplets in emulsion explosive matrices. *Journal of Dispersion Science and Technology.*, 2017., 10: Vol. 38., pp. 1483-1488.
- Whale, G.F. *et al.* 2018. Toxicological and ecotoxicological properties of gas-to-liquid (GTL) products. 2. Ecotoxicology. *Critical Reviews in Toxicology.*, <https://doi.org/10.1080/10408444.2017.1408567> : Vol. 48., pp. 273-296.
- Zhang, K. *et al.* 2022. Peculiarity for the stability of three different emulsion explosives. *Journal of Dispersion Science and Technology.*, 2022., 11: Vol. 43., pp. 1725-1734.
- Zhang, K. *et al.* 2017. Storage modulus' variation in aging and its application in assessing the stability of emulsion explosive matrix. *Journal of Dispersion Science and Technology.*, 2017., 11: Vol. 38., pp. 1517-1522.

Improving airblast predictability with composite wave modelling

A.S. Romphf

Austin Powder, Canada

C.K. McKenzie

Blastechnology, Australia

ABSTRACT: Blasting-induced air overpressure is a common source of complaints from external stakeholders around quarries and surface mines. Urban encroachment around blasting operations is becoming prevalent, which creates a need to reliably predict airblast overpressure levels to ensure compliance with government regulations. Traditionally, the relationship between peak overpressure level, and cube-root scaled distance has been used to predict peak overpressure levels. This statistical approach is often fraught with significant scatter and does not consider the impact that various other blast design parameters and environmental factors may have on peak overpressure levels measured around a blasting operation. In an attempt to examine air overpressure predictability, signature holes were fired in numerous locations throughout North America and the collected data was reviewed considering possible contributing factors such as front-row burden, stemming length, hole diameter, mass of charge, and direction between each charge and the monitoring location.

A composite waveform of a full production blast was then created using linearly superimposed modified Heelan-Blair wavelets which closely resemble the pulses generated from single blastholes. Amplitudes for each wave are scattered using a Monte Carlo technique to simulate observed randomness between upper and lower limits of observed trends. Simulated full-production waveforms compare favourably in terms of dominant features, shape, and amplitude with measured waveforms. Wind velocity and wind direction were also recorded and their apparent effects were evaluated against published recommendations.

1 INTRODUCTION

Blasting-induced overpressure occurs every time a blast is fired. Stakeholders located far-field around a blasting operation often will remark about two distinct events which vibrate structures. Ground vibration, which travels at a higher velocity than sound, arrives first, and overpressure which follows. Each of these environmental effects of

blasting is perceived similarly to a stakeholder and as such, regulatory limits exist for both peak ground vibration and air overpressure levels. These limits are sometimes related to damage potential with a factor of safety applied. Traditionally, a regression of cube root scaled distance is plotted against peak overpressure levels to predict the expected amplitude. However, there are numerous factors which can add uncertainty to

overpressure prediction. To better understand overpressure prediction, experiments including 38 single hole blasts monitored at over 700 monitoring points were conducted throughout North America.

2 BACKGROUND

Regression analysis is often used to examine the relationship between a dependent and independent variable. For overpressure prediction, peak overpressure vs. scaled distance is typically plotted. The generalised equation for prediction of peak overpressure level is shown in equation 1 (ISEE 2011).

$$OP = K_a \times SD_{op}^{-n} \quad (1)$$

where OP is the air overpressure level in Pascals, SD_{op} is the overpressure scaled distance generally defined as linear distance in metres divided by the cube root of the mass of charge in kilograms, while K_a and n are site-specific values determined by the regression trend.

As overpressure levels from blasting are often regulated in decibels, a logarithmic unit, the previous equation can be expressed in logarithmic form as:

$$dBL = dBL_0 - 20n \times \log(SD_{op}) \quad (2)$$

where $dBL_0 = 20 \times \log\left(\frac{K_a}{20 \times 10^{-6}}\right)$, and 20×10^{-6}

Pa is the reference pressure from which air pressure decibel levels are determined.

Richards & Moore (2002), Blair (2004, 2011), King-Siem & Moore (2004), and Segarra *et al.* (2011) suggest that the main sources of overpressure from a blast include:

- Air Pressure Pulse (APP), generated by the displacement of the blast face.
- Gas Release Pulse (GRP), caused by the venting of gases from the face.
- Stemming Release Pulse (SRP), caused by venting and ejection of stemming material from the blasthole collar.
- Rock Pressure Pulse (RPP) Ground motion measured by the microphone which is usually insignificant at typical compliance monitoring distances.
- Initiation System Pulse (ISP) Overpressure produced by the initiation system which is typically insignificant in the case of most modern initiation systems.

Richards & Moore (2002), King-Siem & Moore (2004), Blair (2004, 2011) and Segarra *et al.* (2010, 2011) all suggest that overpressure from production blasting can be highly directional so that the K_a and dBL_0 , will be influenced by factors such as the direction between the blast and the monitoring point and the delay timing of the blastholes.

Blair (2011) used a piston model to distinguish between the APP pulses from ground movement at the bench face (APP_{face}), and those from the largely-independent ground movement at each blasthole collar (APP_{collar}). He then goes further to illustrate that the factors of bench height and blasthole spacing may also introduce directionality to the propagation of the APP_{face} overpressure waves.

The degree to which a charge is confined is a key variable to consider when thinking about overpressure levels, as a blast with a low relative burden and or stemming is more likely to have major venting occur and a higher velocity of motion than a blast where all charges are heavily confined. Linehan & Wiss (1980) reported on the effect of stemming length on the peak amplitude of the SRP, and Richards & Moore (2002), proposed equations to predict the effects on peak overpressure levels of changing both stemming length and front-row burden.

Some of the factors contributing to overpressure such as the GRP and the SRP can be controlled with sound blast design. Modern blast design software can help a blaster determine where to position charges ensuring all charges are adequately confined. Buffer blasting into muck can also limit the APP_{face} , while APP_{collar} , can be controlled with stemming length and by using a suitable material. Adding confinement to a blast will certainly reduce overpressure, however it will also have a consequence of decreasing powder factor which may be detrimental to digging and downstream production.

Blast delay timing introduces a directional effect on overpressure as demonstrated by Richards & Moore (2002). Siskind *et al.* (1984) suggested that limiting inter-hole delay time to about half the speed of sound can limit amplification in the direction of inter-hole sequence. However, a preferential sequence for overpressure reduction may have detrimental effects on ground vibration or fragmentation.

McKenzie & Romphf (2025) suggested that the shape of a scaled Heelan waveform as proposed by Blair and Minchinton (2006), referred to in this paper as the Heelan-Blair wavelet, for near-field

vibration modelling may be used to describe the shape of the elemental overpressure wavelet at any distance to facilitate dynamic modelling using linear superposition. Overpressure pulses measured in front of the face tended to have greater amplitude and frequency than pulses measured lateral to or behind a free face.

Factors such as wind velocity and barrier effects add additional variability to overpressure and should also be considered. Temperature inversions may elevate overpressure levels far-field as suggested by Siskind *et al.* (1984) and Richards & Moore (2002), though such effects seem to apply at distances in excess of 2000 metres.

3 SINGLE HOLE TESTS

In an effort to better understand the degree to which various factors affect overpressure levels, regression plots were prepared comparing the results of 38 single hole tests monitored at over 700 locations. The test charges were fired in limestone, dolomite, taconite and granite. Charge diameters were 89, 102, 115, 127, 140, 171 and 406 mm. Minimum burden varied from 1.75 to 7.0 m, while stemming lengths were between 1.5 and 16.75 m. Mass of charge varied from 40 to 2449 kg. The regression in Figure 1. shows that although overpressure decreases with an increase in scaled distance, the amount of scatter even

when comparing single holes is so great as to render prediction efforts futile.

4 CONTRIBUTING FACTORS

The factors observed to influence air overpressure include mass of charge, distance from blast hole to receiver, angle between the blasthole and receiver, delay timing, wind velocity & direction, material quality factors such as rock mass rating, burden ratio, R_B , and stemming ratio, R_{st} , where:

$$R_B = B/d \tag{4}$$

Where B is the burden in metres, and d is the hole diameter in mm.

$$R_{st} = St/d \tag{5}$$

Where St is the stemming length in metres.

As only two parameters, mass of charge and distance from blast to receiver, are considered in a typical regression analysis, it is no surprise that a great degree of scatter would be present.

Directionality was examined by plotting a traditional regression enhanced by heat-mapping data points using a greyscale value associated with the angle between a line normal to the face and the receiver. Burden ratio, stemming ratio and wind were examined by regression, focusing on a single test site to minimise variation in rock properties.

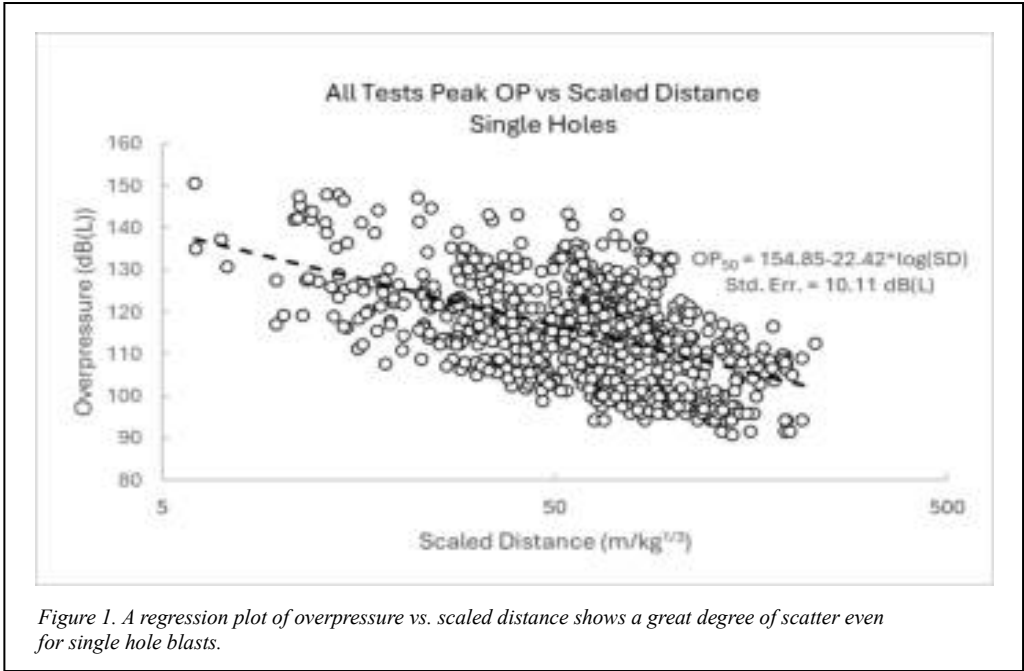


Figure 1. A regression plot of overpressure vs. scaled distance shows a great degree of scatter even for single hole blasts.

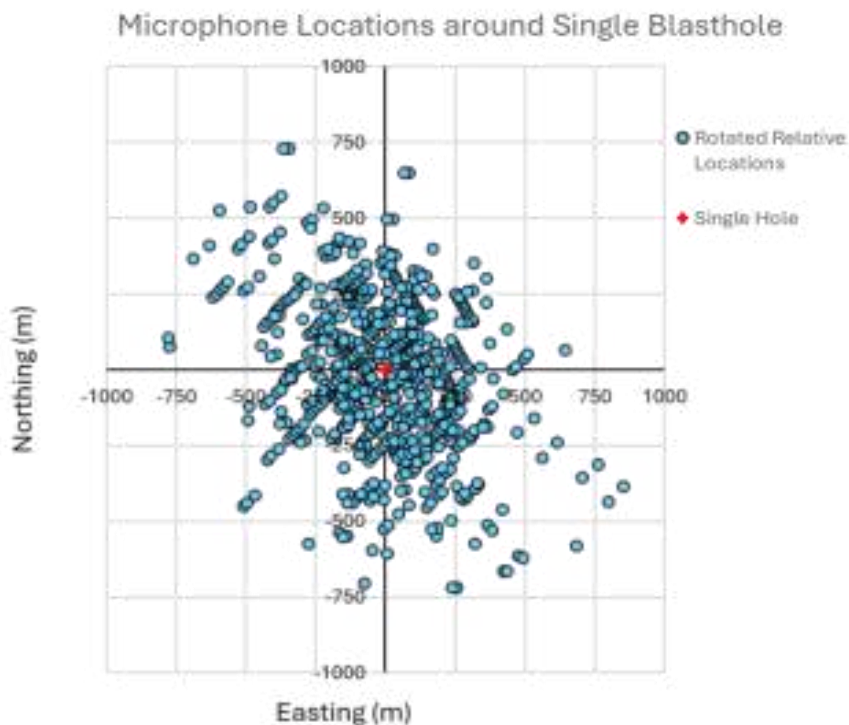


Figure 2. The position of blast monitors relative the blasthole. In the diagram a blasthole is at the origin while relative monitor positions for all tests are oriented around an E-W face line. 38 blastholes with varying properties were fired.

4.1 Direction

To examine the influence that relative direction may have on overpressure, monitors were setup in

arrays at varying distances and angles around single holes as illustrated in Figure 2. A monitor directly in front of the blasthole was assigned an angle of zero, while 180 was directly behind

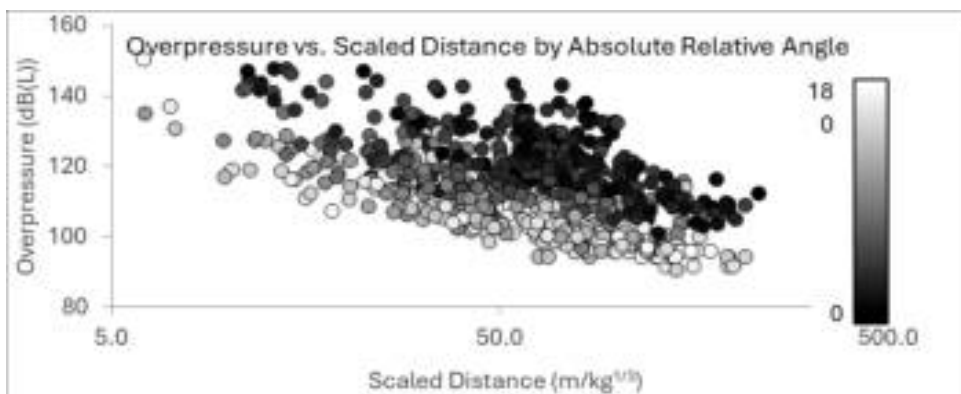
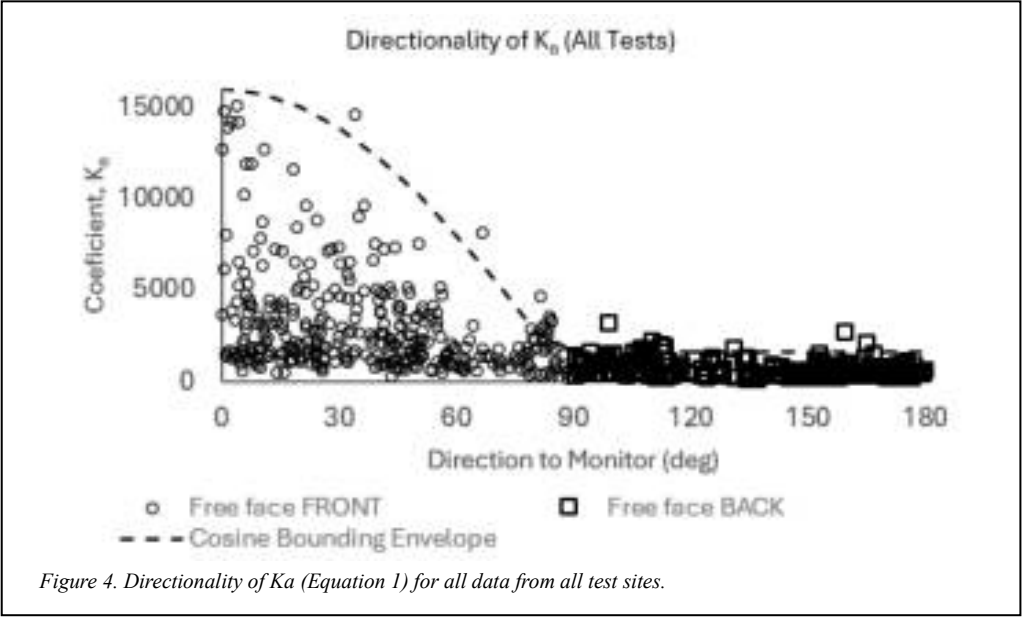


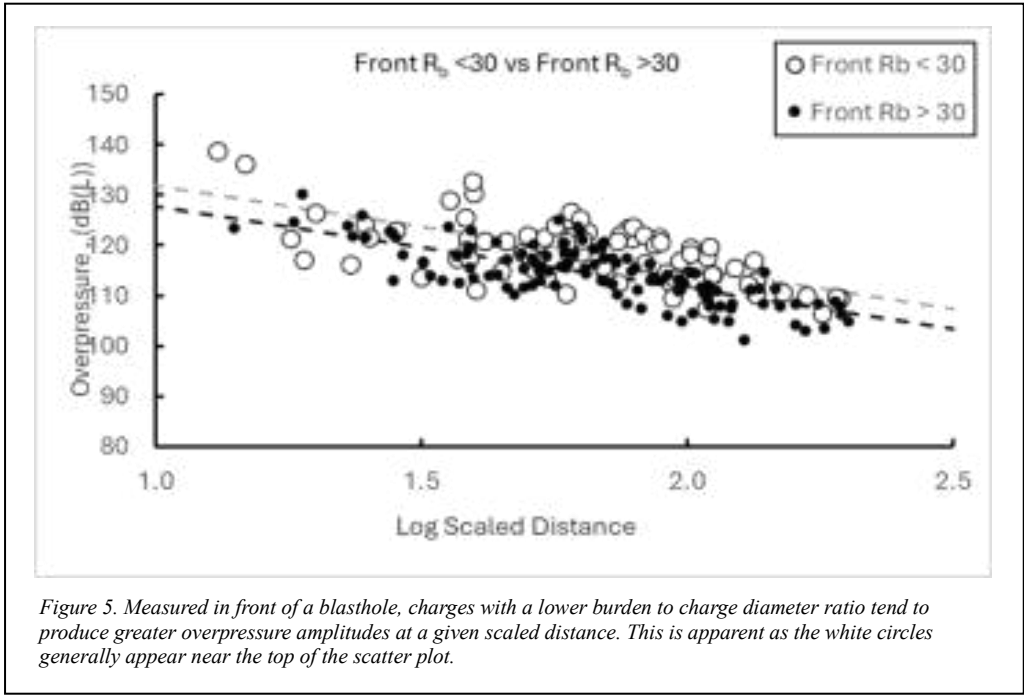
Figure 3. Shading the scatter plot points by absolute angle from the blasthole suggests that directionality has a significant influence over overpressure amplitude and scatter. Subdividing overpressure data into directional sets is likely to improve predictability.



the blasthole.

The entire single hole data set was greyscaled by absolute angle relative to a line normal to the free face. Figure 3 demonstrates that at a given scaled distance, the highest overpressure amplitudes tend to occur in front of the face tapering off at 90 degrees.

Scatter appears to be reduced for the lighter-shaded points (angles greater than 90 degrees) than for the darker-shaded points (angles less than 90 degrees) implying that receivers will experience greater amplitudes and variance of overpressure levels in front of the face varying by direction, while receivers behind the face experience



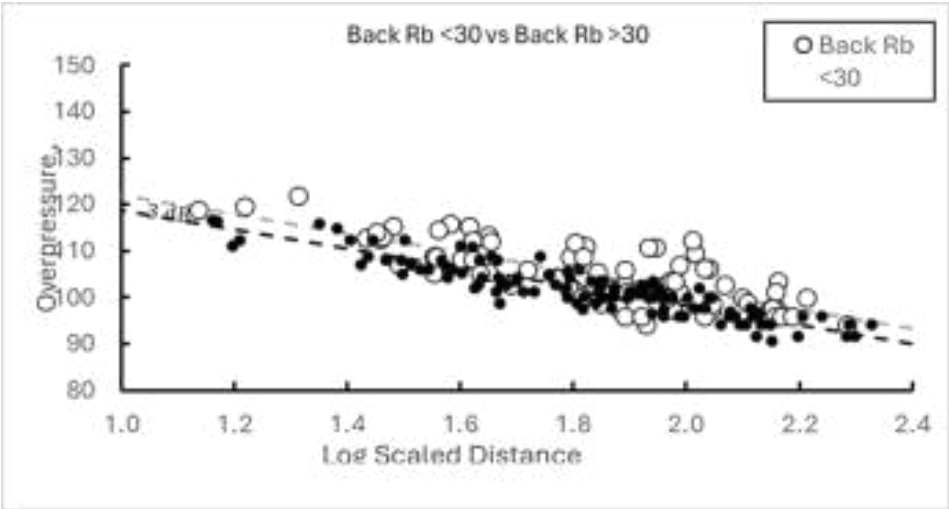


Figure 6. Measured behind a blasthole, charges with a lower burden to charge diameter ratio tend to produce greater overpressure amplitudes at a given scaled distance. This is apparent as the white circles generally appear near the top of the scatter plot.

a lower overpressure amplitude with seemingly less variance by angle. It is apparent that direction has a strong influence on overpressure levels even from single holes. As such, overpressure predictability can be improved simply by dividing blast data into subsets of monitors behind and in front of the blast.

The directionality of the K_a term of equation 1, for receivers at angles less than 90 degrees, is proposed to be a cosine taper applied to overpressure amplitudes in excess of those measured behind the free face according to McKenzie & Romphf (2025) as shown in Figure 4.

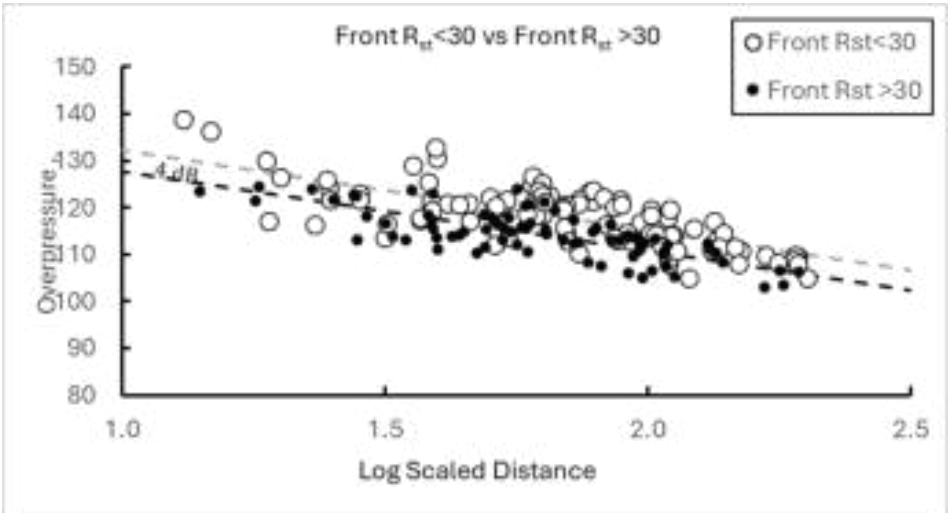


Figure 7. Measured in front of a blasthole, charges with a lower stemming to charge diameter ratio tended to produce greater overpressure amplitudes at a given scaled distance. This is apparent as the white circles generally appear near the top of the scatter plot.

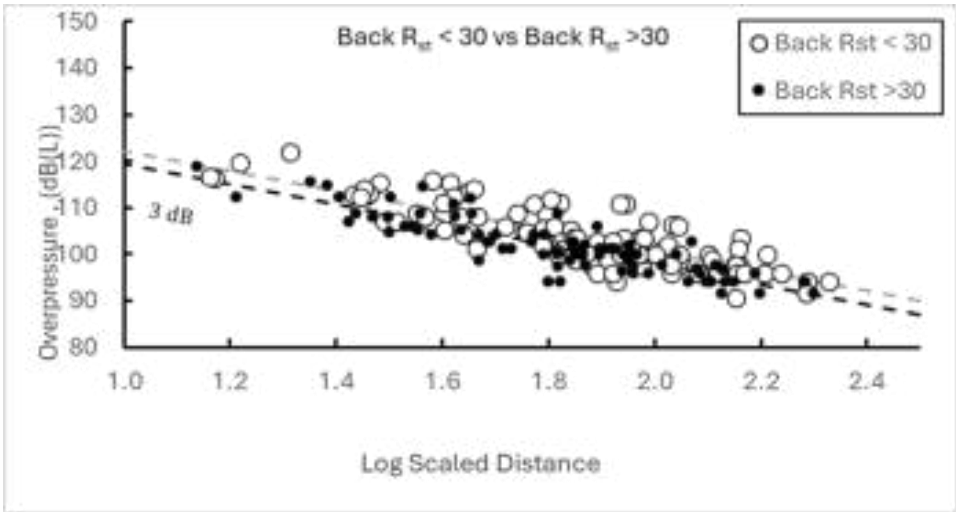


Figure 8. Measured behind a blasthole, charges with a lower stemming to charge diameter ratio tended to produce greater overpressure amplitudes at a given scaled distance. This is apparent as the white circles generally appear near the top of the scatter plot.

4.2 Confinement

Overpressure amplitudes from the test site where the most data points were collected in front of and behind the free face were divided into subsets and compared by burden ratio and stemming ratio.

It is intuitive that charges which are less confined should produce greater overpressure levels at a given scaled distance. Figure 5 shows a regression plot of overpressure levels vs. scaled distance where points white circles have a burden ratio of less than 30 charge diameters while black dots have a burden ratio of greater than 30 diameters, without a significant difference in slope. It is apparent that burden ratio has a significant influence on overpressure measurements forward of a blast face.

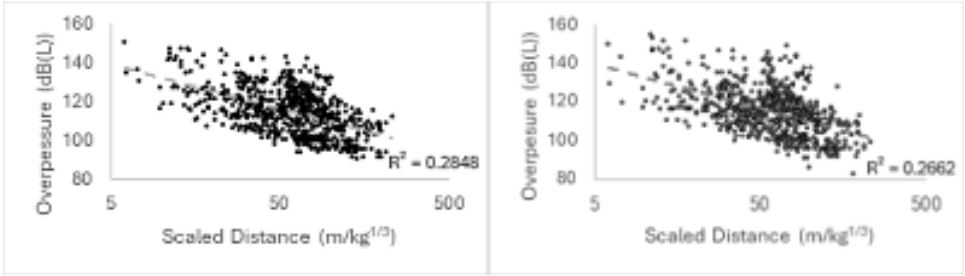
Similarly, behind the free face, using the same technique as the previous figure, Figure 6 shows that the burden ratio appears to have significant influence on overpressure levels albeit less than in front.

Substituting stemming for burden ratio in Figures 7 and 8, we see an effect similar to that observed with burden ratio. Stemming ratio also has an influence on peak amplitudes in front and behind the face, neither of the confinement conditions seem to induce as much scatter or amplitude difference as direction.

4.3 Wind, velocity, direction

The effects that wind velocity and direction have on overpressure amplitude are worthy of consideration. During data collection, for the single hole dataset, wind velocity at the time of each test were determined with a combination of low flying drones, ground-based anemometers and remote weather stations. Discrepancies between data collection methods occurred at times, with contradictory wind velocities and sometimes directions at height versus those measured at ground level. Due to the fluid nature of wind, a blaster may have difficulty deciding which velocity value to apply when considering the potential the effects of wind. Wind direction was recorded with drones and confirmed by mylar balloons affixed to some blasthole collars, values were cross-examined with the direction and velocity of dust movement immediately post-blast via video analysis.

The effects of wind velocity and direction were examined in the same manner as Linehan & Wiss (1980), whereby a wind blowing directly at a receiver from a blasthole was assigned an angle of zero, while a wind opposite, blowing from the receiver to the blasthole was assigned an angle of 180 degrees. The vector velocity of the wind was then applied as follows, according to the convention described.



Figures 9a (left) shows the original regression plot with less scatter than Figure 9b (right) which is the same data with the Linehan and Wiss (1980) wind modification applied.

$$V_e = V_w \times \cos \Phi \quad (6)$$

Where V_e is the effective wind velocity, V_w is the velocity of the wind as measured in the direction of wind, and Φ is the difference in angle between the hole to monitor azimuth and wind azimuth.

Linehan & Wiss (1980) applied a wind correction to the attenuation exponent n of equation 1 in their paper:

$$n_m = n_0 - 0.0165 \times V_e \quad (7)$$

Where n_m is the modified n value, n_0 is the original best fit slope value. The value of -0.0165 is used in this paper as wind velocities were analysed in km/h.

Burgher (1987) suggested that at distances less than a mile, wind may not be a significant source of dampening or re-enforcement from his experiment. While Blair (2004), noted that the Linehan & Wiss (1980) modification would be representative of the climatic conditions at the time and location of their test. Figures 9a and 9b show that for the single hole data set, when the Linehan & Wiss modification is applied, more scatter is introduced to the data. Scatter is reduced slightly, when the wind velocity coefficient is increased to -0.004.

Further dividing the single site regression data into wind velocity subsets of ‘with the wind’ and ‘against the wind’, data was plotted as a regression at one of the test sites. Figures 10 and 11, illustrate a non-existent, wind effect with little difference in

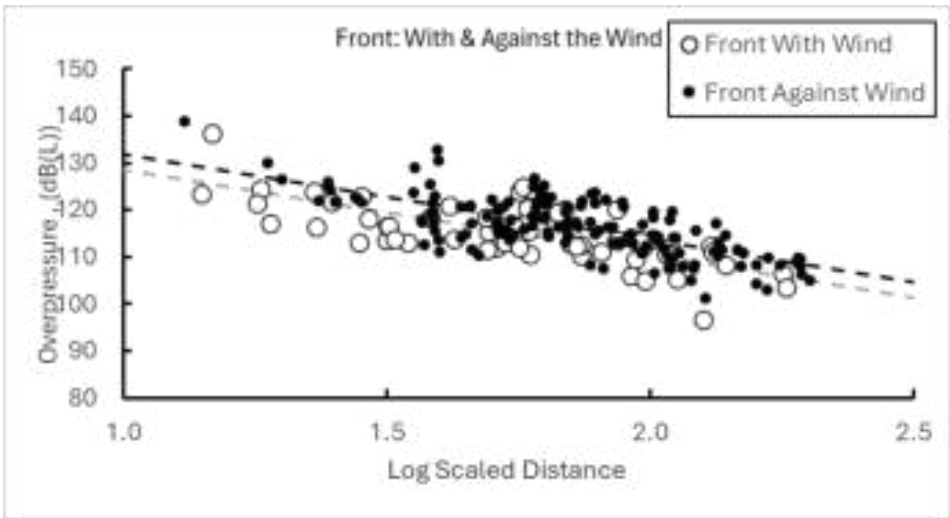


Figure 10. Wind had no apparent influence over overpressure level at a given scaled distance in front of the face. White circles are with the wind, while black dots are against the wind.

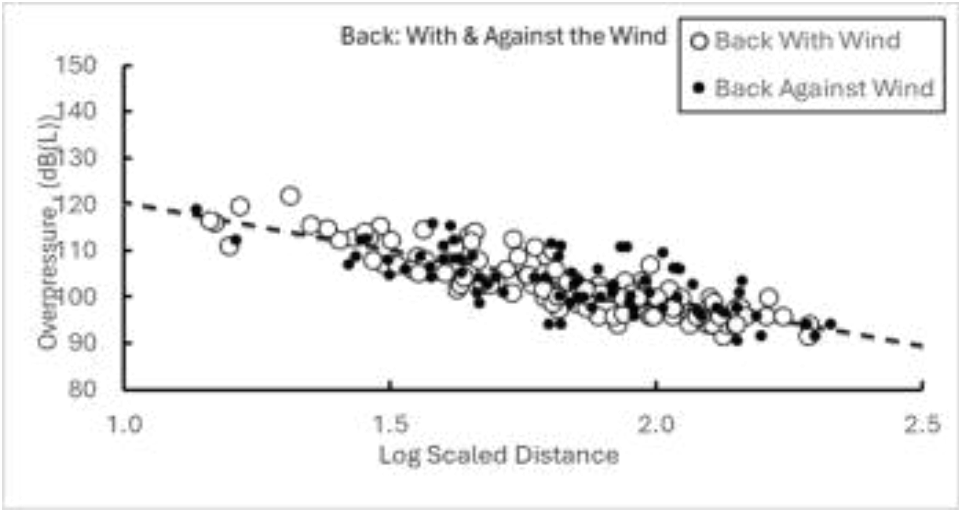


Figure 11. A regression plot behind blastholes with and against the wind direction suggests no influence of wind on overpressure level for the conditions tested. White circles are with the wind, while black dots are against the wind.

scatter, slope, or intercept. Measured wind velocities were between 5 and 30 km/h for the data included.

5 FULL PRODUCTION BLAST OVERPRESSURE MODELLING

Richards & Moore (2002) and McKenzie (2004) demonstrated a synthetic overpressure wavelet superposition, where a dampened sinusoid was used to simulate a full-scale blast incorporating timing.

McKenzie & Romphf (2025) suggested that a scaled Heelan wavelet, proposed by Blair & Minchinton (2006) for the purpose of near-field vibration modelling, could be made to closely resemble overpressure waves collected during single hole tests, with the advantage over a truncated damped sinusoid of having a zero average value, which is important for linear superposition models.

The Heelan-Blair equation is given by:

$$S(\tau) = \gamma_{\eta}(\tau^{\eta} - 2\eta\tau^{\eta-1} + \eta(\eta-1)\tau^{\eta-2})e^{-\tau} \quad (8)$$

Where $S(\tau)$ is the wave amplitude, with a maximum value of 1, γ is a normalising constant, τ is dimensionless time equal to $b \times t$ where t is time in seconds and b is a frequency term with units of s^{-1} .

Richards & Moore (2002) reported the peak levels of overpressure for ‘burden controlled’ and

‘stemming controlled’ blasts to vary according to the respective equations:

$$D_{120} = \left(\frac{k_b \times d}{B}\right)^{2.5} \times \sqrt{m}, \text{ \& } D_{120} = \left(\frac{k_{st} \times d}{SH}\right)^{2.5} \times \sqrt{m}, \quad (9)$$

Where D_{120} is the distance in metres at which the peak overpressure level is expected to be 120 dBL, k_b and k_{st} were presented as constants, d is the blasthole diameter in millimetres, B is the front row burden, SH is the minimum stem height in metres, and m is the explosive mass per delay. The ratios d/B and d/SH are the inverse of the burden ratio and the stemming ratio respectively.

The Richards & Moore (2002) equations can be re-written as the following, more general and extended versions of equation (1) for propagation forward of the free face:

$$dB(x) = 120 + 20m \times \text{Log}(K_F \times R_B^{-n}) - 20m \times \text{Log}\left(\frac{x}{D_{120}}\right) \quad (10)$$

Where: $dB(x)$ is the peak decibel pressure at distance x metres from the charge, n is the amplitude-distance attenuation exponent in scaled distance equation (1), K_F is a fitted constant, R_B is the burden ratio, n is a fitted exponent effectively describing the strength of the front-row burden influence for which Richards & Moore (2002) proposed a value of 2.5, and m is the kilogram mass of explosive per delay.

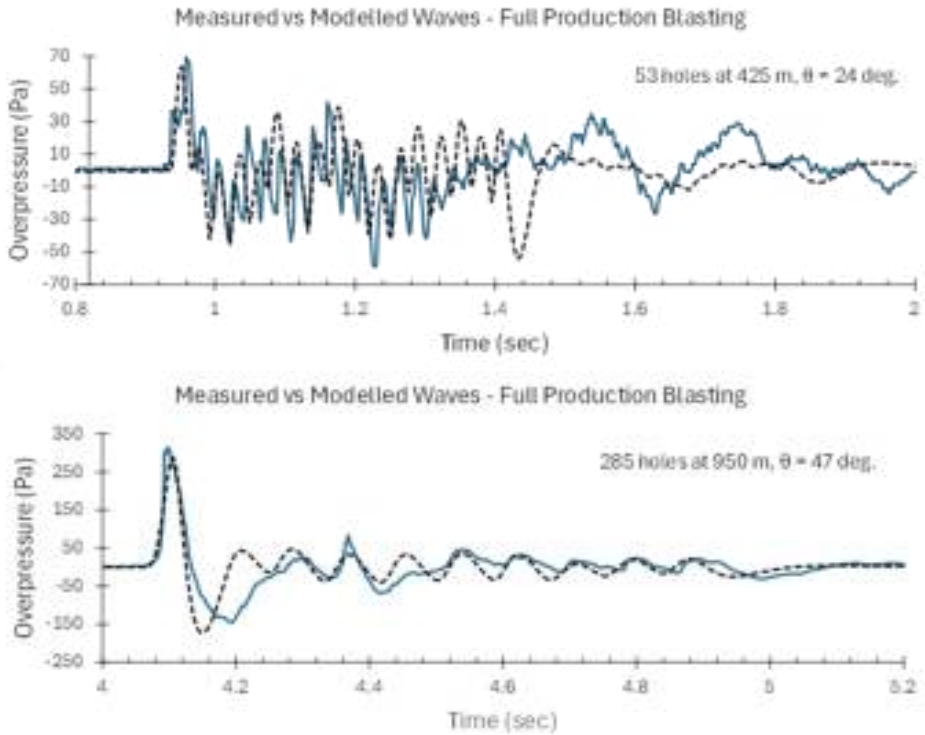


Figure 12. Measured vs. modelled waves at receivers in front of a full production quarry (upper) and mine (lower) blast. Dominant features are successfully reproduced using the Heelan-Blair wavelet.

Similarly, for propagation behind the free face:

$$dB(x) = 120 + 20n \times \log(K_B \times R_{st}^{-\omega_2}) - 20n \times \log\left(\frac{x}{\sqrt{m}}\right)$$

Where: K_B is a fitted constant, R_{st} is the stemming ratio, ω_2 is a fitted exponent effectively describing the strength of the stemming length influence, for which Richards & Moore (2002) proposed a value of 2.5.

The exponent of 2.5 in each of the equations by Richards & Moore (2002) indicates a similar, and rather strong, influence of burden for receivers in front of a free-face blast, and of stemming length for receivers behind a free-face blast. The data from this study does not support either an equal, or such a strong influence of those two variables, though modelling has adopted the same general form for their respective influences.

Additionally, a cosine term, $\cos(\theta)$, has been added to the forward propagation equation to reflect the observed, strong variability in the coefficient, K_F , with direction as demonstrated in

Figure 4. It is this directionality that is responsible for a significantly greater scatter (Standard Error of Prediction) in the scaled distance graphs for measurements made forward of the free face compared with those made behind the free face. The peak amplitude for face-hole wavelets is then assigned by:

$$dB(x, \theta) = 120 + 20n \times \log(K_B + K_F \cos(\theta) \times R_{st}^{-\omega_2}) - 20n \times \log\left(\frac{x}{\sqrt{m}}\right) \quad \text{for } 0 \leq |\theta| \leq \pi/2 \quad (11)$$

And for confined holes, or holes behind the free-face, as:

$$dB(x) = 120 + 20n \times \log(K_B \times R_{st}^{-\omega_2}) - 20n \times \log\left(\frac{x}{\sqrt{m}}\right) \quad \text{for } \pi/2 \leq |\theta| \leq \pi \quad (12)$$

Where, K_B and K_F are regression constants.

A simplified overpressure model would include a summation of time-delayed Heelan-Blair

elemental waves generated by each independently delayed charge in the blast:

$$C(t) = \sum_{i=1}^N A_i \times S(t - d_i) \quad (13)$$

Where t is time in seconds, $C(t)$ is the composite, or summed, resultant time history waveform representing the sum of separate elemental waves, $S(t-d_i)$, and A_i represents the amplitudes of the individual waves according to charge weight, burden ratio, stemming ratio, direction to/from receiver as per equations 11 and 12, and N is the total number of independently-delayed charges in the blast.

A composite waveform of a full production blast was created using linearly super-positioned Heelan-Blair wavelets which closely resemble the pulses generated from single blastholes. A Monte Carlo routine randomises frequency and +/- duration within limits which may be dependent on site-specific factors such as rock type or rock condition. The shape characteristics of the wavelets for each charge are modified based upon observed trends in the collected data from the single hole tests. Simulated composite overpressure waveforms were compared to those of measured pulses with a reasonable agreement between predicted and measured waveshape in terms of dominant features, frequency, duration, and peak amplitude.

6 CONCLUSION

Airblast overpressure modelling is a complex topic due to the fluidity of air and several other contributing factors which influence outcomes. For receivers in front of a blasthole, angle between source and receiver has a significant influence, while behind the blasthole, angle does not seem to have an influence on peak overpressure level. Burden and stemming ratios appear to have an impact on peak levels, though potentially weaker than the effects proposed by Richards & Moore (2002) though it should be noted that the single holes in the data set which were analysed for burden and stemming ratio didn't exhibit violent stemming ejection or face venting. The apparent influence of wind velocity and direction on air overpressure levels were minimal for the dataset, suggesting that the Linehan & Wiss modification should not be applied at typical compliance distances. Linear superposition using a scaled Heelan-Blair wavelet seems to reasonably reproduce airblast overpressure waveforms for full-scale blasts.

7 ACKNOWLEDGMENTS

There was a significant amount of work involved in collecting data for this paper. The authors wish to thank Adam Nichols, Dirk Carolan, Curtis Hoskins, David Klingspor, Evan Smart, Nick Adams, Nicolas Dewood, Ricardo Medina Garcia, Isaac Erickson, Roger Goodweiler and Chuck Palmcook of Austin Powder for their extensive attention to detail and effort. Thanks are also due to the Austin Powder management team for the support of this project. The assistance of Vibratex personnel in various US states in providing, setting up monitors, and retrieving data from the vast arrays of seismographs was critical in the program and is also gratefully acknowledged.

REFERENCES

- Blair, D.P. 2004. Analysis and modelling of airblast and ground vibration. *AusIMM Explo 2004*, Perth, July, pp 1-9.
- Blair, D. & Minchinton, A. 2006. Near field blast vibration models. *Proceedings of 8th International Symposium on Rock Fragmentation by Blasting – Fragblast 8*, pp152-159, Santiago, May.
- Blair, D.P. 2013. Piston models for airblast due to the bulk movement of ground. *Jnl Rock Fragmentation by Blasting*, Taylor & Francis, pp 301- 308.
- Blasters Handbook*, 18th Edition, 2011, Air Vibrations, *International Society of Explosives Engineers*, Chapter 26 pp 581-596, Jon Stiehr editor.
- Burgher, K.E. 1987. Determination of open pit blast sound levels using a proxy charge and multiple linear regression. *Society of Mining Engineers Annual Meeting*, Denver, CO, Feb 24-27, 7p.
- King-Siem, R. & Moore, A.J. 2004. Calibration of an airblast prediction model. *AusIMM Explo 2004*, Perth, July, pp 21-25.
- Linehan P. & Wiss, J.F. 1980. Vibration and air blast noise from surface coal mine blasting. *SME-AIME Mining Engineering Transactions*, April, pp 391-395.
- McKenzie, C.K. 2004. Elemental wavelet overpressure modelling. *AusIMM Explo 2004*, Perth, July, pp 27-35.

- McKenzie, C.K. & Romphf, A.S. 2025. Exploring the complexities of airblast overpressure. *Proceedings of the 14th International Symposium on Rock Fragmentation by Blasting – Fragblast 14*, pp29-38, Antalya, Türkiye, April.
- Richards A.B. & Moore, A.J. 2002. Airblast design concepts in open pit mines. *Proceedings of 7th International Symposium on Rock Fragmentation by Blasting – Fragblast 7*, pp553-561, Beijing, August.
- Segarra, P., Domingo, J.F., López, L.M., Sanchidrián, J.A. & Ortega, M.F. 2010. Prediction of near-field overpressure from quarry blasting. *Applied Acoustics*, 71, Elsevier, pp1169-1176.
- Segarra, P., López, L.M., Sanchidrián, J.A. & Domingo, J.F. 2011. Asymmetric propagation of airblast from bench blasting. *6th EFEE World Conference on Explosives and Blasting*, Lisbon, 18-20 September.
- Siskind, D.E., Stachura, V.J., Stagg, M.S. & Kopp, J.W. 1984. Structure response and damage produced by airblast from surface mining. *Report of Investigations, RI 8485*, 85 pages.

Bioremediation technology combined with explosive soil treatment (bioexplosion)

A. Szalay & I. Zádor

Hungarian Society for Blasting Technology, Budapest, Hungary

J. Földesi

Detonet Ltd, Miskolc, Hungary

I. Diószegi

CerTrust Ltd, Budapest, Hungary

B. Fehér

Division for Biotechnology, Bay Zoltán Nonprofit Ltd for Applied Research, Szeged, Hungary

P. Habi

Elgoscar Ltd, Budapest, Hungary

ABSTRACT: As a result of previous and current civil and military activities, soils are still contaminated with hydrocarbon derivatives in many places. The aim of the technological development presented in the publication is to develop an innovative remediation technology (bioexplosion) for the treatment of soils contaminated with hydrocarbon derivatives. The bioremediation processes currently applied in environmental remediation use special microorganisms (bacteria) to break down hydrocarbon pollution. Combining these bioremediation processes with explosive soil loosening technology is beneficial from several points of view. On one hand, by using industrial explosives, the structure of the soil can be modified in a way that is favourable to the living conditions of the bacteria, and on the other hand, the bacteria and the additional chemical agents can be dispersed homogeneously in the contaminated soil.

1 INTRODUCTION

1.1 Justification of the development work

As a result of previous and current civil and military actions, the soils are contaminated with a great many hydrocarbon derivatives to this day. Oil pollution can make the soil infertile and, by seeping into deeper layers, can also endanger the water resources found there. As an example,

currently, nearly two million m³ of soil is contaminated with oil derivatives in Hungary.

The most commonly used soil remediation methods in practice are soil aeration, soil washing, phytoremediation, thermal processes and bioremediation. Bioremediation is a branch of biotechnology. It is a process that uses living organisms (bacteria, microorganisms) to break down pollutants and convert them into less toxic or non-toxic substances. Soil treatment with

bacteria having known and well controlled properties is considered environmentally friendly process, primarily because many hydrocarbon compounds can actually break down and be converted into harmless substances (CO_2 and H_2O) under optimal conditions. The disadvantage of bioremediation processes and their application limitation is that bacteria are usually only able to spread and carry out degradation processes in contaminated soil to a limited extent, depending on the leakage factor (-k) of the medium. This limitation can be overcome by using bioexplosion processes.

1.2 Bioexplosion

The effectiveness of bioremediation can be increased by creating optimal living conditions for hydrocarbon-degrading bacteria by creating a favourable soil structure. To this end, the aeration, water conductivity, and thus biological activity of the soil must be improved. This can be achieved by explosive loosening of the contaminated soil. The combination of bioremediation with explosive soil loosening is called bioexplosion.

2 EXPERIMENTAL WORK

The three basic tasks of the R&D work plan aiming at elaboration of the bioexplosion technology are:

- Test the viability of bacteria (i.e. how they are affected by the heat and pressure of blasting).
- Determine the proper blasting layout and optimal blasting parameters to achieve the desired soil loosening. This work was carried out using a series of experiments on uncontaminated soils. During the work, great attention was paid to the analysis of the seismic effects of the blasting.
- The experiences and information gained in uncontaminated areas were adapted to the remediation of areas contaminated with hydrocarbon derivatives.

2.1 Testing the viability of bacteria

There is no information available on the magnitude and duration of the pulse-like pressure and the temperature of the blasting that the bacteria can withstand. As a first step in the R&D work, we performed small-scale, easily repeatable tests on the survival of bacteria placed into the soil, using the UCM 006 type blast chamber (Figure 1). The bacteria to be tested were enclosed

in liquid-filled tubular plastic containers (Figure 2.) placed in the sand inside the blast chamber.



Figure 1. Explosive chamber prepared for test.

After blasting the reduction of the total mass of bacteria in the soil inside the explosive chamber was between 2.5 - 5.2%. This result establishes the practical applicability of the bioexplosion remediation process: the shock wave of the blasting can be used to 'spread' the bacteria that break down hydrocarbons in the soil.

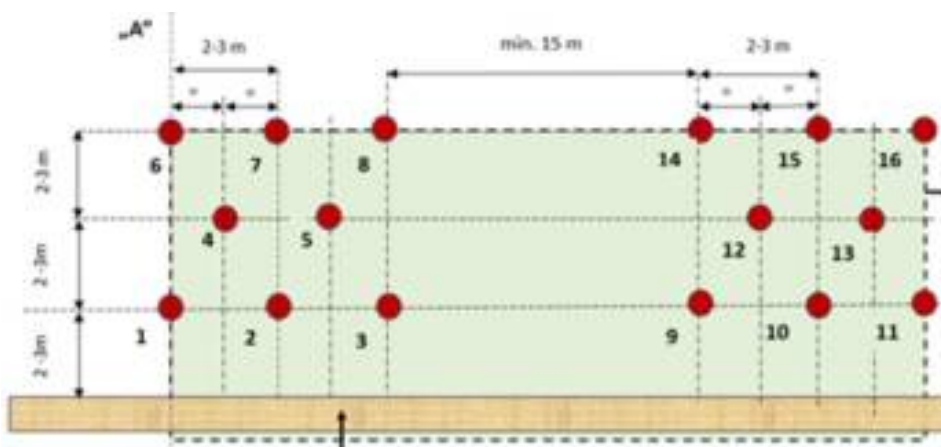


Figure 2. Hydrocarbon-degrading bacteria in plastic containers.

2.2 Soil loosening experiments in uncontaminated areas

The work was carried out within the frame of a cooperation agreement with the Hungarian Ministry of Defence, in the area of a former army base (barrack) which is currently out of use.

Both the bacteria and explosives were placed in a purposefully designed borehole system. The aim of the field blasting technology is to investigate the optimal arrangement of the boreholes, explosive charge arrangements and charge timings. As an example, we present one of the experimental blasting arrangements carried out in the barrack area. (Figures 3, 4 & 5.).



Explosive used: Emulex1. The weights of the charges are: Q_A (top) = Q_B (middle) = 0.6 kg and Q_C (bottom) = 0.8 kg.

Charge initiation system: NONEL. Fuses: 8 pcs 450 ms NONEL Indetshock fuses, 8 pcs 475 ms NONEL Indetshock fuses, 8 pcs 500 ms, 9 m, Indetshock fuses.

The experiences and results of the soil loosening experiments carried out on the sandy-clay soil of the barracks in 2023 can be summarised below:

- To optimally modify (loosen) the soil structure, multiple explosive charges must be placed inside a borehole and these charges must be initiated with properly

calculated timing.

- Between the explosive charges crushed stone should be used as choke.
- The hydrocarbon-degrading bacteria mixed with water can be poured into the boreholes directly on the choke materials.
- A necessary condition for successful soil loosening is the creation of a trench that allows the blasted soil structure to move and thus crumble.
- Trenching is also important in terms of reducing seismic effects.
- The above experiences and results were utilised in the remediation experiments of petroleum-contaminated soils.

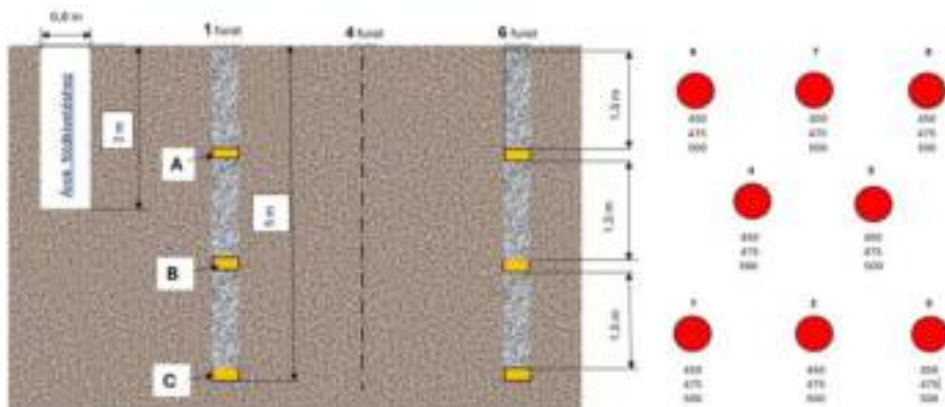


Figure 4. Arrangement of the charges in the boreholes.



Figure 5. Charges packed into biodegradable PLA housings.

2.3 Remediation of a petroleum-contaminated area

Location: Approximately 12*10 m area in the vicinity of a damaged DN300 mm petrol pipeline. In the designated area, our cooperation partner (Elgoscar Ltd) mapped the soil contamination up to a depth of 7 m from the soil surface, as shown

in Figure 6.

For analysing the effectiveness of the bioexplosion three sub-areas (A, B & C) were established in the test site and six measurement points (FCP1-FCP6) were appointed, for controlling the results of the bioexplosion experiments (Figure 7.).

- Sub-area A: only the bacteria (inoculum)

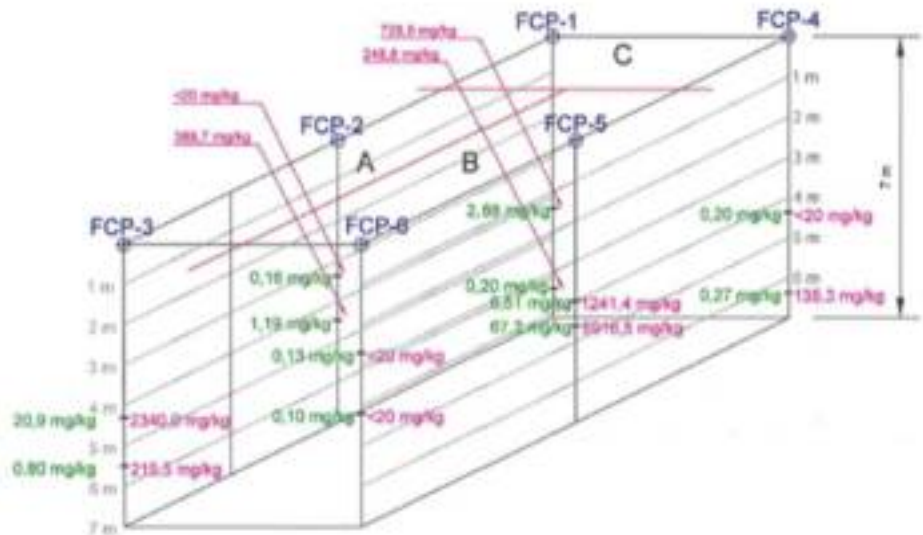


Figure 6. Soil contamination values in the area designated for the remediation experiment - magenta colour: TPH=Total Petroleum Hydrocarbons - concentration of total mineral hydrocarbons (mg/kg soil); green colour: PAH= Polycyclic Aromatic Hydrocarbon - concentration of polycyclic aromatic hydrocarbons (mg/kg soil).

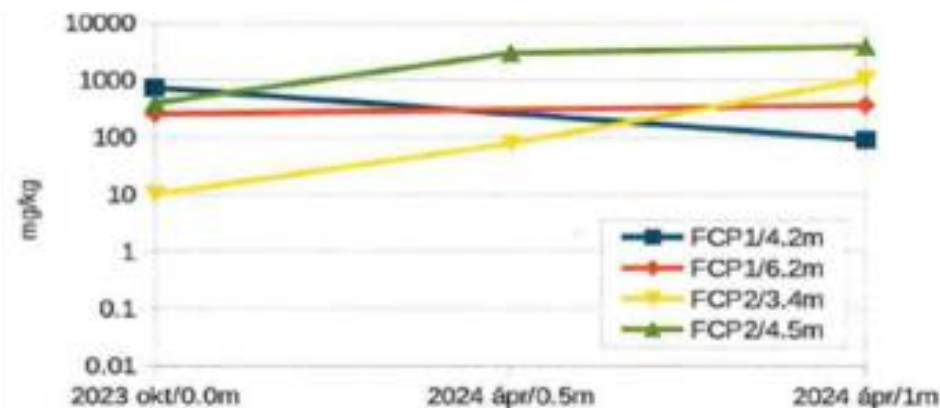


Figure 8. TPH measurements on the untreated sub-area C (control area, not treated with bioexplosion nor bacteria).

3.1 Measurements in the untreated sub-area

Measurements taken around the boreholes FCP1 and FCP2 in the untreated sub-area showed the following results: the petroleum contamination in the soil in the untreated sub-area continued to spread significantly between October 2023 and April 2024 (red, yellow and green lines), and in one case it decreased slightly (blue line).

3.2 Measurements in the sub-area treated with bacteria (without blasting)

Measurements taken around borehole FCP3 in the sub-area treated with bacteria without blasting showed the following results: petroleum pollution increased slightly between October 2023 and April

2024 in the area treated with bacteria (brown and blue lines). Measurements were not carried out around borehole FCP4.

3.3 Measurements in the sub-area treated with bioexplosion

In measurements taken around the borehole FCP5 in the sub-area treated with bioexplosion we observed a significant decrease in TPH contamination at a distance of 0.5 m from the borehole, while the level of contamination has practically not changed at a distance of 1 m (Figure 10).

(Remark: during the mechanical works carried out on the sub-area B, the soil near the borehole FCP6 was mixed, therefore we cannot correctly evaluate the measurement results.)

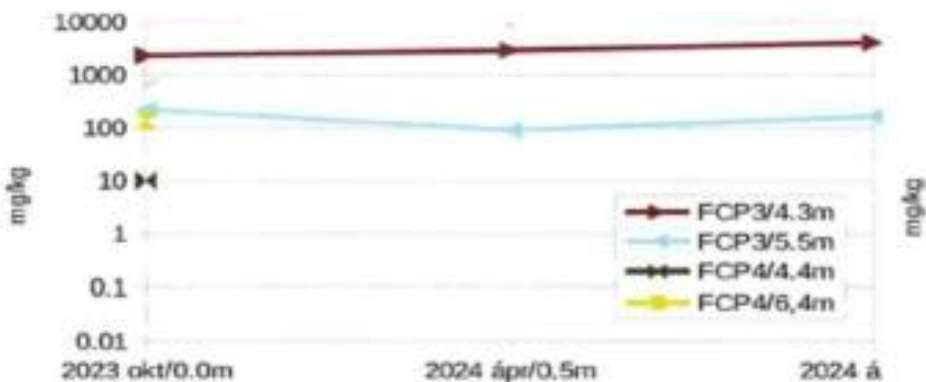


Figure 9. TPH measurements on the sub-area A treated with bacteria (without blasting).

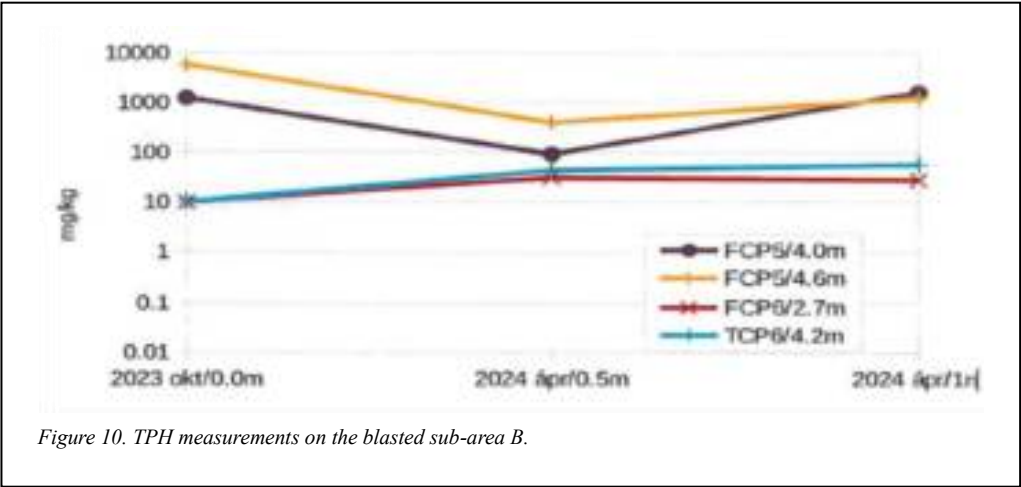


Figure 10. TPH measurements on the blasted sub-area B.

4 CONCLUSIONS

Changing of the TPH contamination on the sub-areas A and C is not significant (Figure 8. and Figure 9.). This shows that without explosive soil loosening, the bacteria that break down hydrocarbon contaminations are unable to spread, ‘move’ in the soil.

On the sub-area B around the borehole FCP5 we observed a significant decrease in TPH contamination at a distance of 0.5 m from the measuring point, while the level of contamination has practically not changed at a distance of 1 m (Figure 10). This means that the scope of soil loosening of the currently used charge structure does not reach 1 metre. As the work continues, we will change the charge design and/or the borehole layout.

Taking into account the experience of the work carried out and the results of the measurements, we can state that combining the well-known bioremediation processes with explosive soil loosening technology is beneficial from several points of view. On one hand, by using industrial explosives, the structure of the soil can be modified in a way that is favourable to the living conditions of the bacteria, and on the other hand, the bacteria and the additional chemical agents can be dispersed homogeneously in the contaminated soil.

We consider the bioexplosion technology to be a useful complement to currently used environmental remediation technologies.

RETURN TO CONTENTS PAGE

Author index

- Aarnio M. 121
 Ağan Y. 315
 Aimone-Martin C. 13, 23, 151
 Alvarez J. 237
 Andersen C.A. 219
 Araos M. 453
 Arbi M. 435
 Arriagada P. 175
 Astorga C. 261
- Berg A. 175
 Blair D.P. 3
 Blázquez J. 261
 Brzozowski A. 63
 Burnham R.E. 151
- Cardu M. 271
 Cavanough G. 325
 Choi B. 373
 Choi J. 373
 Coarita E. 249
 Cortés H. 175
 Cortés R. 175
 Couceiro P. 261
- Davidsavor T.A. 151
 Desrochers J. 33
 Diallo B. 291
 Diószegi I. 497
 Domotor R. 131
 Dowding C. 13
 Dufresne K.M. 409
- Etulle L. 33
- Fagerlund M. 53
 Fehér B. 497
 Földesi J. 497
- Gagnon A. 97
 Galleguillos M. 261
 Gerat A. 203
 Gerlich M. 397
 Ghiciei E. 301
 Gilbert R. 209
 Gil Greene R. 183
 Gjødvad J. 13, 75
 Gjødvad J. F. 23
 Gómez R. 75
- Habi P. 497
 Halme T. 343, 447
 Hänninen T. 121
- Hara M. 397
 Heide G. 423
 Henríquez J. 175
 Hoffman J. 351
 Hüdaverdi T. 315
 Huerta Valer G.S. 291
 Hunt R. 475
 Hwang N. 191
- Jamett N. 175
 Järvinen A. 343
 Jern M. 13, 23, 53
 Jitea C. 301
 Johansson D. 229
- Kara S. 191
 Keller K. 423
 Kilo M. 423
 Kim H. 373
 Kim S. 373
 Klippmark V. 475
 Konya A. 209
 Kreivi J.A. 387
 Kroke E. 423
 Kuutti H. 343
- Laing M. 191
 Laszlo R. 301
 Lee S.J. 191
 Lee Y. 373
 Little T.N. 163
 Lokou G. 249
 Loukides C. 13
 Lovitt M. 463
- Madrid J. 261
 Manríquez L. 183
 Marijnissen J.S.C. 387
 Martínez L. 261
 McKenzie C.K. 485
 Moles C. 209
 Morales D. 175
 Moser T. 23, 43
- Navarro J. 261
 Nel R.P.G. 417
 Newman S. 13
 Nicola A. 301
 Núñez Recabal E. 183
- Oh S. 373
- Park Y. 191
- Parkinson M. 131
 Perryman M. 475
 Petropoulos N. 107, 229, 475
 Pienimäki A. 475
 Pina A. 97
 Prasetya H. 435
 Proulx J.P.R. 33
- Rapps C. 203
 Ratliff J.K. 151
 Ravanal P. 261
 Riahi O. 249
 Rivera-Torrente M. 475
 Rodrigues San Miguel C. 229, 475
 Romano J. 351
 Romphf A.S. 485
 Rönkkö T. 343
 Rudenko D. 13
 Rumjantsev M. 343
 Russell J. 475
- Sahu R. 281
 San Miguel C.R. 107
 Schlothauer T. 423
 Seccatore J. 271
 Silvonen V. 343
 Simon B. 301
 Stenzel G. 325
 Stenzel G.J. 335, 417
 Szalay A. 497
- Taoum S. 249
 Tapia F. 261
 Tatom J. 351
 Thibault E. 291
 Tobar E. 183
 Todaro C. 271
 Torrance A. 325
 Trzeciński W. 397
- Vergara P. 261
 Vikum Sri Mendis H.C. 107
- Wallace J. 87
 Wilkinson T. 475
- Yang R. 33
 Yi C. 229
- Zádor I. 497

