FIRST INTERNATIONAL SYMPOSIUM ON
ROCK FRAGMENTATION
BY BLASTING
LULEÄ, SWEDEN, AUGUST 22 - 26, 1983

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First International Symposium on
ROCK FRAGMENTATION BY BLASTING

August 23 - 26, 1983
Luleå University of Technology

edited by
Roger Holmberg             Agne Rustan

SPONSORS
National Swedish Board for Technical Development (STU)
Swedish Detonic Research Foundation (SveDeFo)
Swedish Mining Research Foundation
Luleå University of Technology

PURPOSE
This symposium is the first international convocation of scientists and engineers committed to open discussion and continuous information exchange on current progress, on-going research and engineering innovation in the field of fragmentation by blasting. By organizing this kind of forum the hope is that the future research can be focused on those areas where most efforts are needed.

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PREFACE

The First International Symposium on Rock Fragmentation by Blasting was held in Luleå, Sweden, at the Luleå University of Technology during August 23-26, 1983. It attracted 157 participants, and 18 accompanying persons from the 23 following countries:

- Australia: 16
- Austria: 2
- Brazil: 1
- Canada: 5
- China: 11
- Costa Rica: 1
- Finland: 5
- France: 2
- Great Britain: 6
- India: 1
- Israel: 1
- Italy: 2
- Nigeria: 1
- Norway: 5
- Spain: 2
- South Africa: 2
- Soviet Union: 1
- Sweden: 67
- Switzerland: 1
- Thailand: 1
- USA: 21
- Zimbabwe: 1
- West-Germany: 2

The symposium was held under economical guarantee from The National Swedish Board for Technical Development and the Luleå University of Technology. The arrangements in Luleå were made under the responsibility of the Swedish Detonic Research Foundation, The Swedish Mining Research Foundation and Luleå University of Technology.

Before the symposium mine tours were arranged to the Research Mine at Luossavaara, to LKAB in Kiruna and Malmberget and to the open pit mine "Aitik" belonging to the Boliden Mineral Co.

The scientific program included 51 papers where 48 papers are published in volume 1 and 2. Three papers are published in this third volume which also includes opening speech, key-notes and discussions.

Preparations for a Second International Symposium on Rock Fragmentation by Blasting are underway.

Dr Per-Anders Persson
Chairman of the Organizing Committee

Roger Holmberg
Head of papers
Agne Rustan
Head of program

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Participants of the First International Symposium on Rock Fragmentation by Blasting in Luleå, Sweden, August, 1983.
Welcome to Luleå University of Technology.  
Vice Chancellor Dr Torbjörn Hedberg  
Opening of the Symposium and Introduction.  
Dr Per-Anders Persson, Chairman of the organizing committee  

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Ulf Langefors, Sweden  

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Agne Rustan, Sweden  

Blasting Experiments in the Luossavaara Research Mine.  
Torbjörn Naartijärvi, Sweden  

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Production Drilling with high Accuracy.  
Lars Hermansson, Sweden  

Energy in Fragmentation.  
Per-Anders Persson, Sweden  

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Ladies and gentlemen,

It is my very pleasant duty to greet you all welcome here to Luleå, to our University and to this part of the country.

I hope that your stay here will be pleasant and we are proud to be able to act as hosts for this first international symposium on Rock Fragmentation by Blasting.

This conference is an important event for this young university. It was founded only 12 years ago, although we sometimes claim this school to be the oldest institution for higher education in engineering in this country. The first such school, a mining college, was founded in Falun in 1822. It was later moved to Stockholm and merged with what later became the Royal Institute of Technology. In 1972 the parliament took the very wise and sensible decision to move the school of mining to Luleå.

The reason behind the transfer was the need felt by some at least, for a better balance between the north and the south of Sweden. The same reason lies of course behind the creation of the other schools here, Mechanical Engineering, Computer Science and Technology and others.

Mining is of crucial importance for this region and will remain so for a long time. The most important mines are located in this area. The present difficulties of the mines can not be overcome without strong efforts in R&D and without good university educated engineers.

What we are doing in the field is therefore a part of our long term efforts to support the economic development of the area. But we do not look upon ourselves as a university only for northern Sweden. This is not true for any field and in particular not in mining where we are the only school in Sweden. And we do not want to restrict ourselves to Sweden. On the contrary, it is our clear ambition to make Luleå into an international center for mining engineering education and research. This conference is a step in that direction. Another step is the establishment of close ties with various other mining universities in the world. In a few days we are going to sign an agreement with the South Central Institute for Mining and Metallurgy in Changsha, China and I extend my particular greeting to representatives from that school present here now. I am sure this will be of mutual benefit.

We have also many students from abroad, both in our doctoral program, as undergraduates or within our special mining technology program. The last one attracts professional engineers from many countries all over the world, mostly from developing countries.

I declare this symposium opened and I wish you all a good conference.
It is a pleasure for me to open this first International Symposium on ROCK FRAGMENTATION BY BLASTING. It is sponsored and arranged jointly by

The Swedish Board for Technical Development (STU)
The Swedish Detonic Research Foundation (SveDeFo)
The Swedish Mining Research Foundation, and
Luleå University of Technology (LuH)

The credit for the initiative to start is due to Dr Agne Rustan, who has put in a great deal of time and devotion in getting it all together, aided and abetted by Roger Holmberg and the staff of the University.

It follows on the First Australian National Symposium on Rock Fragmentation in Adelaide 1973, and national symposia, for example in USA the Society for Experimental Stress Analysis (SESA) fall meeting 1980, where rock fragmentation was discussed.

Many nations arrange national meetings on blasting technique annually. The first meeting in Sweden was held 1954 and it is called "Bergsprängnings-kommitten", Rock Blasting Committee. In Austria an "Informationstag fur Sprengtechnik", Information day for Blasting Technique", began 1969. In 1975 The Society of Explosives Engineers in USA started annually Conferences on Explosive and Blasting Techniques.

By keeping to a limited special area within the science and technology of rock blasting we were hoping to get together a very qualified audience of specialists and practical blasting engineers from the mining and construction world. Looking out over this distinguished group of 157 people from 23 countries of which so many are our good personal friends, I feel we have been greatly rewarded for our efforts.

We also hoped, by this initiative, to entice within the next few years perhaps, some group of people somewhere, perhaps in Australia to organize the second International Symposium on this or another special area within Rock Blasting, so that we may get together gradually a continuous documentation of technical and scientific progress in our field.

On behalf of the sponsors and organizers I wish to express the hope that you all who have travelled so far will go home, at the end of the week, satisfied that you have been part of a useful, warm, informal, and friendly meeting, having learnt something new, having shared with others some of your own knowledge, and, not least important, having found new personal friends among the fraternity of rock blasters.

A special welcome to the few brave ladies who have ventured this far north and into such a man-dominated part of science. I know you will enjoy the thought that this far north you are really on top of the world.
This is indeed far north — we like to call this area the northern cap of the world — and were it not for the friendly warm westerly winds from the Atlantic Ocean and the Gulf stream this part of Sweden would have an arctic climate. In fact, it sometimes has that, even in summer, and you should not be surprised to see some snow on the ground one of these mornings. Still, we call this summer, by definition, because the sun is still up most of the day—time, and far into the night.

So this is the time of year we enjoy the fact that there is no ice on the waters. Therefore we have included on the program a boat trip to one of the islands out in the northern part of the Bay of Bothnia, as an opportunity for you to get together, to get to know each other better, and to talk to each other, unhindered by boring speeches and papers.

This is also the season for some of our peculiar Swedish eating specialities namely the crayfish and the sour herring. Perhaps you may talk one of your Swedish fellow participants into introducing you to either of these truly amazing foods, not really considered to be food at all anywhere else in the world, thus giving you a unique bond together from shared adventures.

This autumn, on October 21, is the 150th anniversary of the birth of Alfred Nobel, the man who invented the detonator and the dynamite, and thereby gave a new direction to the Swedish and international development of fragmentation by blasting.

So, in formally opening this birthday celebration symposium, I would like to repeat, once again, the key words for the symposium:

informal — you don't need to wear a jacket and a tie
warm and friendly — just be your dear natural self
useful — make a point of bringing the new participants into the discussion

I now have the great pleasure to introduce you to our key note speaker Mr Ulf Langefors, author of the first engineering text-book on rock blasting: The Modern Science of Rock Blasting, well-known to most of you because of his many original contributions and publications, former head of the Nitro Nobel Detonics lab and the Swedish Detonic Research Foundation at Vinterviken, and former president of Nitro Nobel.
FRAGMENTATION - A REVIEW

Disintegration of Solids

by Ulf Langefors
Tempera AB
STOCKHOLM

ABSTRACT

In this key-note speech for the First International Symposium on Rock Fragmentation by Blasting, the author gives a broad survey of the fields of structural strength and fracturing of solids. The demands set by practical needs for fragmented rock for different purposes are reviewed briefly. The development of methods for blast design and controlled blasting is outlined, and the importance for easy fragmentation of stressing rock in tension rather than in compression is pointed out. The speech ends with a call for research into unconventional methods of rock fragmentation.

1. MATERIAL AND STRUCTURAL STRENGTH THEORIES

Shell structures

The theory and methods for predicting structural strength developed over the years concern mainly shell structures. In construction and industry, buildings, ships, automobiles, airplanes or space vehicles are all shell structures. Their structural integrity depend on the designer's skill. He must correctly predict the ability of thin walls, membranes and struts to support their own weight and the external forces, and he has to choose materials of appropriate strength.
The finite element methods (FEM) developed by Börje Langefors and others are powerful tools for dealing with this kind of structures. These techniques have reached a level of great accuracy and refinement, thanks to the fact that one- and two-dimensional approximations are easy to make and quite accurately describe the response of thin walls.

Volume design and mass structures

In rock design we face a different set of problems. A rock mass is a truly three-dimensional structure, and often quite a complicated one. The FEM methods can be used wherever the problem has a two-dimensional nature, such as the rock stresses around a long tunnel or a tall shaft. Applications to three dimensions have been made for cases where rotational symmetry makes it possible to transform the mathematical treatment to a two-dimensional version as has been done by Bengt Akesson and others. For calculating the strength of rock caverns of an arbitrary shape a successive application of three-dimensional FEM and other iterative methods is being developed. However, even present-day large capacity computers become very small when we try to introduce the rock structure around a three-dimensional cavern. Our skill is needed to simplify the problem by selecting the critical few fissures for the computer to work with.

In the context of this symposium, the dynamic stress field in the rock mass surrounding a detonating explosive charge in a drillhole is an even more complicated problem where time is a fourth dimension to take into consideration. Even so, we have a surprisingly good grasp of what goes on there. The problem has been attacked from several different angles, by high speed photography where the rock is replaced by transparent plexiglass, by finite difference calculations where the rock is regarded as a compressible and deformable solid but homogeneous medium, and by such approaches as Cundall has made with his block method, where the strength and compliance of the rock fissures are taken into account but the rock blocks themselves are regarded as rigid and incompressible.

2. THE PRACTICE IN EXCAVATING ROCK

Rock excavation is done for one of two major purposes. Either we need the excavated material to use as an ore or a mineral or for material needed in construction. Or else we are making constructions in and by the rock. Water power stations, harbours, water and drainage projects, air fields, road constructions, storage facilities are some examples.

Producing fragmented rock

In excavation for mining or road constructional purposes the need is generally to produce small enough debris that is easy to load and easy to crush. Sometimes, as in dam construction or when building a wave breaker it is necessary to produce quantities of large and regularly shaped boulders.

Even where the main purpose of excavation is to create an empty space, such as a tunnel or an underground machine hall, the fragment size is important because it influences the speed of loading or mucking and the capacity of the transport equipment.

In all these cases the degree of fragmentation influences the economy of the excavation job. Controlling fragmentation is here an important art to learn. The quantity of explosive and its distribution within the rock mass is one. The rock structure is another important factor influencing the fragment size distribution.
Remaining rock design

The use of explosives as a tool to remove rock requires controlled blasting to minimize damage to the remaining rock walls and neighbouring structures. In present-day open pit mining with shot hole diameters in the range 250 - 500 mm (10 to 20") each shot hole may contain tons of explosive and the whole blast may involve the detonation of 200 - 500 tons of explosive. In underground mining large shot hole diameters in the range 150 to 200 mm are increasingly being used. In tunneling larger diameter (50 to 100 mm) and long (3 - 6 m) shot holes are also common. The development in all these areas towards larger blasts gives great savings in the cost of excavation, but also makes greater demands on methods to avoid damage.

In open pit mining, the stability of the pit slopes and the corresponding slope angles influence the economy and safety of the operation. Methods are being developed to produce steeper slopes by controlled blasting that leaves the remaining rock strong enough for the increased stresses that result. Even steepening the slope by one degree may save a tremendous amount of waste rock removal in a deep and large open pit mine or road cutting.

3. THE DEVELOPMENT OF METHODS FOR BLAST DESIGN AND CONTROLLED BLASTING

Background

From Viking times into our days iron and steel have been the mainstay of Swedish economy and industry. The iron nodules on the bottom of our lakes were used up long before the 16th century. The many small iron ore mines in south and central Sweden that gave support to our great Baltic empire and our warrior kings in the 17th and 18th centuries are most of them closed down, but we still have the world's largest underground iron ore mine in operation in Kiruna and Malmberget, and Luleå, where we are convened today, is the site of one of Europe's great modern steel works.

The tradition from so many years of rock and iron ore excavation lives on. Swedish miners and blasting crews are proud and independent men with a positive interest in improvements and new methods, and from Christopher Polhem's great inventions in mining technology in the 17th century onwards Swedish mining and construction engineers have been eager to try out anything new that may bring about speed and economy in the operation.

The methods of underground excavation in mining were taken over and quickly developed further in our century during the great era of hydroelectric power plant building along our northern rivers. I think that Sweden has been among the pioneers in building underground hydroelectric power plants with machine halls completely underground. The first was in Mockfjärd in 1911. It was modernized in the 1950-s and is still in use. During this century also came the great developments of tungsten carbide tipped drill steels, new kinds of nitroglycerine and nitroglycerine-free explosives and waterbased explosives. These have taken the place of Nobel's original dynamites. Electric and non-electric short Interval delay detonation, mechanical equipment for pneumatic or pump loading of explosives and many other tools that are part of our present day rapid excavation techniques have also been developed after the war. All these developments were borne out of this eagerness to try out and experiment with new techniques and to have a free and open exchange of results that has been and still is so characteristic of Swedish industry at its best. Out of this work grew several companies that are now assisting, on a world wide scale, industrial and developing countries to find even better methods for rock excavation, such as Sandvik and Fagersta, Atlas Copco and Nitro Nobel, the Swedish Water Power Board, Skanska and JCC, Boliden and LKAB, and a host of consulting companies.
The Nitro Nobel Detonics lab at Vinterviken, the site of Alfred Nobel’s first factory, has played a certain role in this development of modern technology.

**Charge calculations**

The first step in transforming the art of rock blasting into a branch of engineering, the modern technique of rock blasting, was to find a general formula to describe the interrelationship between charge weight and the size and geometry of the rock mass and the separation of drillholes. In its simple form, the equation

\[ Q = a_2 V^2 + a_3 V^3 + a_4 V^4 \]

has proved a very useful tool in blast design. \( V \) is the thickness of rock to be broken loose in front of the charge of weight \( Q \), and \( a_2, a_3, \) and \( a_4 \) are constants. Equation (1) was coupled with expressions for correction for different charge geometries and charge distributions such as concentrated or extended charges, and different degrees of fixation, by which we meant the greater or lesser accessibility of free surfaces for the broken rock to move into the surrounding free space.

![Specific charge, \( q \)](image)

**Figure 1. Charge in kg/m\(^3\) in conformal scaling with different swell component \( a_4 \).**

**Model scale experiments**

Plexiglass model scale experiments proved to be a useful aid in thinking about the mechanisms of blasting. From these, we began to understand the importance of the ratio between burden and spacing of drillholes in determining the size of boulders and the degree of damage to the remaining rock.

**Smooth blasting and presplitting**

The ideas born when looking at the small plexiglass blocks were tested in real rock, and it was found that virtually undamaged rock walls with neat rows of split-in-half drillholes clearly visible could be produced by very minute extended specially developed Gurit charges placed in holes with a spacing to burden ratio 0.8, using a...
charge weight as low 120 grams per meter of 37 mm drillhole. Similar results were obtained by pre-splitting, where the crack between holes was formed before the rest of the blast was fired.

**Ground vibration measurements and control**

The development of short interval delay detonators and the techniques of smooth blasting made it possible to bring rock blasting right inside downtown Stockholm and other built-up areas. More and more refined instruments to record ground vibrations were developed as the need arose to check the result of decreasing the weight of charge detonated at each interval time. Based on our own and other people's measurements of ground vibrations, the charge level Q/R^{3/2} (which is identical to the inverse of the reduced distance R/Q^{2/3}) was found to be an important damage parameter. Diagrams of the relation between charge weight and distance were produced with the charge level as a parameter, as our experience in avoiding building damage grew.

We realized the possibility of utilizing the random scatter of delay times of detonators of the same delay interval number which forestalled the present-day sophisticated computer methods of delay time control to minimize vibration damage.

**Fragmentation**

Out of the plexiglass model scale blasting experiments grew also the hope of finding a simple relationship between the size of fragment or boulder size and the charge weight or burden. The resulting graph was based on a series of large blasts of medium size hole diameter in rock. It was later verified and adjusted as results of large diameter drillhole blasting became available.

![Figure 2. Relationship between boulder size L and specific charge q for different values of the burden V. Boulder size is the sieve opening size that lets through between 90 and 95% of the rock.](image)
The graph in figure 2 is valid for a spacing-to-burden ratio around 1.25. Later, the prediction from plexiglass model blasts that larger spacing-to-burden ratios gave a smaller average fragment size prompted us to try out large scale blasts using the so-called wide spacing blasting method. By increasing the spacing-to-burden ratio to between 4 and 8 we found that we could decrease the number of boulders in secondary blasting by a factor of nearly 2 and at the same time increase the mucking capacity by 30%. The total cost for drilling, blasting, mucking, transport, and crushing decreased by 15%. By suitably arranging the sequence of detonation times for the holes in a blast drilled with a square drillhole pattern the same effect can be obtained, and this has become common practice in many open pit mines and quarries around the world today.

Later studies by Bernt Larsson and others have indicated that there is a roughly constant ratio of 2.6 between the boulder size $s_{90.95}$, defined as the sieve mesh opening size that lets through between 5 and 10% of the fragmented rock, and the average boulder size $s_{50}$ defined as the sieve size that lets through 50% of the rock.

![Graph showing fragment size distributions from rounds with an average boulder size \( L \) or an average fragment size \( s_{50} \) based on Larssons experimental material.]

4. **GEOMETRY AND GEOLOGY**

The structure of the rock mass with its joints, fissures and open cracks separating more or less homogeneous boulders that are held together by the pressure from surrounding rock has an influence on the strength and stability of the rock mass by far exceeding theoretical calculations or strength tests based on homogenous material.

In the early days we developed simplified rules for correcting the standard charge size to take into account the different leakage of explosion gas into cracks when blasting in layered rock with drillholes parallel or perpendicular to the layers. It is encouraging
to see, in the preprints of this conference, new developments towards a more complete understanding of how the rock mass structure influences the damage and fragmentation due to blasting. This is, I feel, an area where great developments will come. Already, we can say that the geometry is more important than the geology in many applications of rock blasting.

5. ENERGY REQUIRED FOR FRAGMENTATION

The energy required to separate the rock mass into fragments is surprisingly small; the major part of the explosive's energy is lost by leakage and by expansion after the actual fragmentation is completed. This indicates, perhaps, a great potential for improvements in the technology.

When we introduced smooth blasting, for example, it seemed incredible to the experienced rock blasters that such large volumes of rock could be disintegrated with such small and weak charges as were developed for the purpose.

I expect that there are many other areas of rock blasting where an unusual approach may give equally surprising results even more useful than smooth blasting.

6. THE STRENGTH OF ROCK

A basic fact to be applied is the brittle character of the rock strength. Strong as it is - rock in the bible, in literature, in poetry, and in music drama is referred to as the incarnation of strength - it is the rock's weakness that has to be used constructively in developing rock fragmentation methods.

The major weakness that we have to make constructive use of is the low tensile strength of rock. The most effective rock disintegration is done by creating tensile stress, not by compression or crushing.

The stress field tends to direct the tensile stress components towards the free surfaces because that is where the deformations are greatest. By this effect and using tensile rather than compressive loading the remaining rock will get a lower or no tensile stress and therefore stay relatively undamaged, whereas the volume to be excavated will become well fragmented. This may be obtained if charges are initiated in proper time sequence and with drillholes set in proper geometry so that a maximum of tension is created in the disintegrating rock volume.

7. FRAGMENTATION - NOW AND IN THE FUTURE

To fragment rock by explosives is an old, but rather crude idea. During the past decades the explosive as a tool has been transformed into something quite reliable and accurate. It is now possible to apply blasting in a precise way and to precalculate the desired technical result with a high degree of accuracy. Even so, it is clear that there are new possibilities to be developed. I would like to see also other methods than blasting further developed for the disintegration of rock such as:

- hydrodynamic
- mechanical equipment for creating tensile stress
- chemical
To conclude, a word on the way to you all who are now facing the task of dealing with the rock and master it:

You have the fascinating possibilities ahead of you to develop new blasting methods, but also the chance to develop the new "NONEX" means that we need to accomplish future building in rock and with rock, and handling rock generally in a professional way in mining and in construction engineering with greater safety and greater accuracy.
GAPS IN THE FIELD OF FRAGMENTATION BY BLASTING
AND SOME THOUGHTS ABOUT FUTURE RESEARCH AND
DEVELOPMENT

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INTRODUCTION

Rock fragmentation by blasting is a field where much research effort has been devoted to both the behaviour of explosives (detonics) and the action of the explosives on the blasted material (blasting). There is, however, still a lot of research to be done and some proposals will be covered in this keynote.

The opinions given are based on my own experience from literature reviews and 10 years experience from model- and full scale tests in blasting. Owing to the rich amount of literature in blasting, some knowledge could easily have escaped me, but now I have the opportunity to have that circumstance corrected by this qualified audience.

Blasting is a wide scientific field where knowledge in chemistry, physics, electronics, mechanics, mining, rock mechanics and, finally, computer technique is needed. There are very few scientists (if indeed any) available with good knowledge in all of these fields. If a person or organization possessing knowledge in all of these fields could initiate good co-operation between experts in blasting, this would be of enormous value for a fast development in the technology.

Great improvements in blasting have been made during the last 10 years, with safer explosives and safer initiation of explosives, less production of toxic blasting fumes, better methods to predict the fragmentation from blasting with the help of empirical formulas or computer programs using radial cracking or wave theory. A better theoretical understanding of the whole blasting process has also been achieved. Despite enormous increase in knowledge, there is still no mining or construction operation without boulders.
One of the main goals in the Luossavaara Research Mine at Kiruna is to develop new technique to minimize the number of boulders and, of course, to eliminate them if possible. This will make it possible to introduce a continuous loading technique of the blasted ore.

An extensive literature review and state of the art report has therefore been undertaken by Luleå University of Technology on contract from the National Swedish Board for Technical Development.

One of the difficulties in making a complete state of the art report in blasting, is that the amount of literature in the field is so vast. In our report it is shown that empirical formulas have been derived by some authors to calculate the whole fragment-size distribution for model blast tests, not only $k_{50}$, and, in one case, a formula has been derived for full scale blasts, the Kuznetsov-Rosin-Rammler model, presented by Claude Cunningham at this symposium.

According to my opinion improvements in blasting would develop faster if more state of the art reports were done and published. It would then be easier to find the problems and define the field for future research.

EXAMPLES OF GAPS IN THE FIELD OF FRAGMENTATION BY BLASTING

In the following, I would like to ask some questions which I have found have not been answered.

Do we really know the breakage process in jointed full scale rock? My answer is no, because we do not have any

* In this keynote state of the art report means a logic review of the knowledge in the field which is examined and, if possible, also an economic evaluation.
3-dimensional methods to measure what happens with cracks during the first stage of the blast. It is necessary to know where the cracks start, at what speed they develop, what happens to the cracks when they meet a weakness plane etc. How does the speed of cracking vary with borehole pressure, borehole pressure history (borehole pressure versus time), confinement of the blast-hole etc? Another interesting thing to study is the number and length of radial cracks and how they vary with borehole pressure and confinement. How much of the breakage is for instance caused by branching of cracks?

In the United States the fragmentation of the already broken pieces when flying through the air has been studied with high speed photography by Stephen Winzer.

In the laboratory many interesting observations have been made in plexiglass about the behaviour of the shock wave and formation of cracks and the interaction between shock waves and joints. Could these results uncritically be regarded as representative for the breakage process in full scale rock?

Another interesting question is how much of the rock is broken in different failure modes, compressive, tensile and shear respectively? That failure criterium which is most common ought to be the most interesting when the rock properties regarding blasting are to be determined.

Another important information in blasting rock is the borehole pressure history. Some methods to measure the borehole pressure are available, but usually only the peak of the borehole pressure could be measured.
Another question is why the angle of breakage in model tests, when breaking one single hole in slab blasting, is between 120-155° for different rocks or rock-like materials, instead of close to 180°, which is the value according to theoretical calculations of stresses around the borehole. An important question to answer is therefore, when and why the two side-cracks are formed.

In the model tests done at the Luleå University of Technology we have shown that only one or two pieces are formed in slab blasting when the burden is close to the critical.

These pieces have rotational forces and therefore, the breakage close to the critical burden is not a perfect tensile breakage.

Another interesting observation, which we made in our model tests, and which has to be explained, is why the optimal burden in slab blasting is about the same as the critical burden instead of less than the critical like in crater blasting.

The theory and understanding of the behaviour of stemming for large holes is not very good. Most of the information given in literature is results from blast-holes less than 35 mm. The optimal size distribution for stemming materials is also not known.

RESULT PARAMETERS IN BLASTING

There are two main result parameters in underground blasting,

. Fragmentation
. Damage to the surrounding rock.
An accurate, cheap and fast method to determine fragmentation is still not available, although one of the projects in the Research Mine is devoted to fragmentation measurement methods. The most accurate and efficient method today is the photographic method, where the fragmentation is digitized manually from photos.

A standardized method to express deviation of measured fragmentation from the real fragmentation has to be worked out. (One idea could be to take the mean value of the deviation at each 10 weight-% of the fragmented mass).

**Rock damage**

Damage can be measured as vibration levels or change in wave transmission, in the surrounding rock. Another way is to measure the backbreak behind the theoretical contour. Sophisticated methods for vibration and wave transmission measurements are available but instrumentation and interpretation costs are high.

Easy, accurate and cheap methods to measure the perimeter after blasting in inclined open stopes from boreholes are not known by us. In the Soviet Union a method has been developed to measure from vertical boreholes the perimeters in vertical rooms (salt) based on the ultra sound principle.

**Blasting of homogeneous rock**

Many scientists have tried to find some parameters to describe the blastability of rock. Langefors introduced early a method to empirically determine the blastability by test blasts in drill holes with 33 mm diameter. This method has given good results in Swedish rocks, but during the last
25-30 years the hole-diameter has been increased in many mines and it is therefore discussable if the rock constant test is representative for large hole-blasting.

At Luleå University of Technology we have developed a method to determine the blastability of homogeneous rock in small scale slab blast tests. This method will be presented in session No. 1. If other researchers are interested in doing the same kind of tests, we will be more than happy to show how to do them. Perhaps our test procedure could be a first proposal for an international standard to test the blastability of homogeneous rock. For the Luleå University of Technology it would be very interesting to get results from testing other rock types all around the world.

**Blastability of the Rock Mass**

A method has to be worked out how to determine the blastability for the whole rock mass.

One possibility could be to increase the burden for single holes in full scale slab blasting until the critical burden is reached. The fragmentation would have to be measured for each burden. The test should if possible be done with that diameter and explosive which is going to be used in production blasting. Of course this kind of test would be very expensive, but in the long run a scientifically and well documented test in the beginning of a new operation will save time and money for the whole period of operation of the mine or construction work.

Another way to attack this problem would be to develop rock mechanics methods to compute a numerical coefficient for blastability (resistans to blasting). In Norway such an index has been developed for drifting by Sassa and Ito.
According to our own experience, the existing rock classification systems are not accurate enough, because closed joints are not taken into account in these systems. We believe that closed joints also have a significant influence on fragmentation.

Initiation Technique

More accurate timing in blasting caps is necessary if sophisticated blasting is going to be done regarding fragmentation, vibration and back break.

International co-operation

All these tasks mentioned cannot be performed by one nation and therefore international co-operation has to be increased. Some co-operation has already begun, for example between Sweden and Australia regarding the use of cross-hole seismic. Before any co-operation starts, it is important that the problems are well defined by state of the art reports.

International Standardization of Rock Blasting

International standards for blasting nomenclature and symbols are lacking and ought to be worked out. Without clear definitions a science will develop more slowly. One example will explain the necessity better. The term "burden", could have many interpretations, depending on the kind of "burden" which is of interest. For example, optimal, maximum, practical, critical, drilled, blasted and effective burden etc.

Work on finding standard methods to determine the properties of explosives has started among the detonics people and in the future hopefully some methods will be standardized.
The working committee is called EXTEST and Dr P-A Persson is the head of that group.

International standards for methods to determine the blastability of homogeneous rock and the whole rockmass should be worked out.

FUTURE RESEARCH AND DEVELOPMENT

The success of future development in fragmentation by blasting will depend on how much the rock-breaking efficiency can be improved. About 20 major parameters control fragmentation and to find the optimal combination is therefore difficult.

- Improvement of the blasting technique can only be done if we have accurate methods to determine the size distribution before and after blasting. Da Gama has developed a computer program to determine the block size distribution of the rock mass before blasting and that will be presented at this symposium. The most accurate method today to determine the size distribution after blasting, is probably to count the number of the boulders and calculate the boulder weight. The rest of the size distribution below the boulder limit is also of interest and therefore cheap and fast methods have to be developed also for that part of the size distribution.

- The accuracy of image processing of fragmentation, which has been studied by Carlsson and Nyberg, could probably be improved in the future, but the resolution has to be increased and be made better than the human eye. If that is possible, I think that automatic image processing will be a good tool to determine size distribution fast and with high precision.
The contour of the rock after blasting is very important for the following holes regarding the size of the burden and therefore a state of the art report should be done about all fast, cheap and accurate methods available to measure contours underground.

Methods to measure the borehole pressure in full scale blasts should be developed.

New methods have to be developed to measure the 3-dimensional cracking process in full scale rock and to study the dynamic fragmentation mechanism more in detail.

To understand the breaking process better a detailed mapping of the form and the original position of each fragment should be done.

Regarding the behaviour of stemming, formulas should be established empirically to determine the resistance force of stemming versus borehole pressure, borehole diameter, size distribution of stemming material, packing, density, cohesion and finally friction against the borehole wall.

More accurate formulas which consider rock structure should be established to predict fragmentation in full scale. Not only $k_{50}$ but the whole size distribution.

Computer calculation methods should be established to predict the vibration level in any point in the rock regarding several extended charges (the whole round). With the computer the influence of different timing on vibration and fragmentation could then be studied.

International standard for blasting nomenclature, symbols and formulas in blasting.
An international standard should be worked out to determine the blastability of homogeneous and inhomogeneous rock.

A classification system for the rock mass regarding blasting should be established for production blasts.

For a better coordination of the international work, an organisation has to be established with a presidium at the top and working groups dealing with each of the above-mentioned problems. The working groups should collect as much knowledge as possible and publish the information in state of the art reports. After this work has been done, some ways to find a solution to the problems would be discussed in the working groups. Then each research institution interested in doing some work should apply for contracts in their own country. A recommendation from the international working group would thereby be of good help in getting contracts. If the research is organized in this way, I do not consider it as any obstacle that certain problems could be penetrated in several countries at the same time. I think this would be a fruitful and stimulating way for the researchers doing the work and the final result would be of a higher quality than if it were achieved without competition.

Personally I am of the opinion that an international blasting symposium should be held each four or five years, and at the same time the working groups should meet and present their results.

Finally, I would like to suggest that my proposals can be discussed during the symposium and at the panel discussion at the end. I am convinced that this Blasting Symposium with high quality contributions from all around the world will be of great importance for the future blasting research.
BLASTING EXPERIMENTS
IN THE LUOSSAVAARA RESEARCH MINE

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INTRODUCTION

The "Research Mine" is the first large, intensive effort in mining research financed by the Swedish Government. Over a 5-year period beginning in 1981, the Swedish Mining Research Foundation will work in the "Research Mine" on the development of mining systems and equipment. The principal purpose of this research is to increase the competitive strength of the Swedish mining industry.

The Foundation was formed to promote the introduction of large diameter blasthole open stoping, and the development of applied rock mechanics, in Sweden.

The previously abandoned Luossavaara Iron Ore Mine in Kiruna has been reopened for the project. The total production during the planned 5-year project period is approximately 2 million tonnes. The income from the sales of ore which will be mined during the course of the project will cover a large portion of the costs for the research.

The Foundation's organisational structure can be seen in Fig 1. The work in the Research Mine consists of a number of projects within the fields of Rock Mechanics, Mining Engineering and Mining Equipment Design.
Totally about 70 people are employed at the mine, including contractors. The Swedish Mining Research Foundation employs 15 people.

The decision for the layout of the Research Mine has been made with consideration for both the research program as well as production requirements. See Fig 2.

![Fig 2: Layout for the Luossavaara Research Mine.](image)

The 200 m long and 10-30 m thick orebody dips at approximately 60°. The ore has been divided into four primary stopes, A-D, each with a length of 35 m. Between the stopes, similar sized pillars have been left.

The drilling of the production holes is performed using the 'in-the-hole' technique, the diameter of the holes being 165 mm.

From cross-cuts on the 265 m level, the holes are drilled down to an undercut on the 355 m level. The hole lengths are approximately 100 m, except in stope D.
where the drilling is conducted from the 320 m level using 40 m long down holes and 60 m long up holes. The D stope blastholes are also 165 mm in diameter.

BLASTING EXPERIMENTS AT THE RESEARCH MINE.

The aim of the blasting research that is being conducted at the Research Mine is to obtain:

- controlled fragmentation
- controlled vibration levels and rock damage, and
- high efficiency and reliability of blasting

In an effort to achieve these goals some experiments have already been performed, and fullscale tests in the mine are about to commence.

Blasting research at the Luossavaara mine consists of three major components;

1. Theoretical work
2. Development of equipment essential for the blasting experiments, and
3. Fullscale tests in the Research Mine

Work has commenced on the development of a mathematical, computer based model to predict the size distribution of fragments produced by a given blast. Scale model tests examining the influence of certain blasting parameters on the resulting fragmentation, have also been performed.

The fragmentation model in use at the Research mine was developed by Hans Hjelmberg at Nitro Nobel, and is based upon fragmentation formulas that have been in use, in Sweden, for a long time. A significant advantage which the Hjelmberg model has over other models, which are based on the same mathematical algorithm, is that the Hjelmberg model uses the true hole pattern. This feature of the model is possible
due to the fact that all production blastholes at the Research Mine are surveyed for hole deviation. Briefly, a calculation using this model begins with the assignment of an area of influence for each hole in a blast. This is, at present, facilitated by using a specified breakout angle. Depending on the shape of the free face, and also on the delay time, the breakage area can have different sizes and shapes. Fig 3.

Next, the fragmentation size distribution for each hole in the blast is calculated and the individual hole distributions are then combined to give a total size distribution. The theories and formulas used in this model are presented in a paper, at this symposium, by Hans Hjelmberg from Nitro Nobel.

The aforementioned scale model blasting experiments have been conducted at the division of Mining and Rock Excavation at the University of Luleå, and financed by the Research Mine. The aim of these scale model tests was to describe how parameters like the specific charge, geometric scale and rock structure, influences the degree of fragmentation resulting from a given blast. From the results of these tests it was hoped that the blastability of rock could be described as a function of readily attainable physical rock parameters.

The results of these scale model tests are presented in two papers, at this symposium, by A. Rustan and Zu Guang Yang, respectively.

It is hoped that the results of these scale model tests can be used to improve the fragmentation model. At this stage it is planned to include into the fragmentation model terms related to the blastibility and in-situ structure of rock.
Fig 3  Fragmentation Model by H. Hjelmberg.
DEVELOPMENT OF EQUIPMENT

In order to perform the blasting experiments which have been planned, it has been necessary to develop the equipment and techniques required to:

- measure bore hole deviation
- measure precise initiation times
- provide accurate delays
- measure fragmentation size distribution, and
- monitor seismic signals

In order to minimize the delay scatter which can sometimes cause problems, especially in large blasts with many delays, an electronic blasting machine has been built. Up to 50 different delay times can be provided, and the accuracy is better than 1 ms. The Research Mine staff is currently exploring the possibility of using the electronic blasting machine in conjunction with surface delays. By using the shortest delays available, it is suspected that this technique will be the most accurate and reliable. Of prime importance in this experimentation is to be certain of when and if the experimental charges have detonated. At the Research Mine, there are currently two methods of detecting the exact time of detonation of each charge. Firstly, this information can be obtained from the analysis of the blast vibration records and secondly, the information can be obtained by using an electronic timer which has been developed for this purpose. The timer operates by placing a pair of wires in each charge, the timer registers the time at which the circuit was broken and the result is written on a printer.

Most of the experimentation at the Research mine concerns fragmentation, consequently methods facilitating the measurement of the size distribution of a blast, is of prime importance. To screen the rock from a blast is in many cases impossible, and also expensive, consequently it has been necessary to support, and actively participate in the development of methods which will indirectly
estimate the size distribution of fragments resulting from a blast.

At the University of Luleå, mainly at the Division of Industrial Electronics, a method for estimation of fragment size distribution using automatic image processing has been developed. This work has been financed by the Research Mine.

Pictures, taken by a T.V. camera installed in a stope's drawpoint, are fed to a microcomputer, where they are digitized and the size distribution of the rock particles shown in that picture is evaluated. The method is presented in a paper at this symposium by Olle Carlsson. The equipment is now installed in the mine.

The pictures which are used by the microcomputer for fragmentation assessment calculations are also recorded on videotape, thus they can be manually digitized and the results used to test the accuracy of the automatic system.

To calibrate the optical methods, a screen which will provide 3 different size fractions, has been installed before the crushers on the surface. It is planned that at least 10% of the rock removed from the mine will go through this screening process. Fig 4.

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**Fig 4** The production flowsheet with methods of fragmentation evaluation.
Boulders, which in this case are rocks larger than 75 cm, are separated on a grizzly underground, and measured.

Consequently, there is every reason to expect accurate estimates of the true size distributions resulting from the blasts, and these distributions will be compared with the results of the optical methods.

FRAGMENTATION TESTS AT LUOSSAVAARA MINE

During the design stages, it was decided that the blasthole patterns which were finally adopted should be such that the resulting fragmentation would be suitable for a continuous loading system, such as a hydraulic feeder and belt conveyor. The maximum sized rock particle that can be handled by the continuous (loading) system is 75 cm. Simple calculations showed that if the drilling and blasting patterns were designed to return a $k_{50}$ of 12-14 cm, the resulting broken rock should be suitable for continuous loading. Similar calculations showed that no individual blast hole should have a deviation, such that it produced muck with a courser $k_{50}$ than 28 cm, thus avoiding too many boulders and secondary blasting.

To meet these requirements, a staggered hole pattern using a 3.3 m burden and a 4 m spacing in stopes B and C, for the upholes in stope D, as well as in all pillars, was decided upon. It was also decided that ANFO would be used in the majority of the blasts, due to its low cost.

To ensure against the production of an unacceptably large number of boulders it is essential that the blasthole deviation should not exceed 1 m, which represents an accuracy of about 1% for the 100 m long production holes used in the Research Mine. Results on drilling accuracy are to be presented in a paper, at this symposium, by Lars Hermansson.

The first in a series of blasting experiments planned for the Research Mine involves the use of different
powder factors. These experiments will include the use of two different explosives in stope B, ANFO and a high density TNT-slurry called Reolit. In stope A, the blasthole pattern will be increased to a 4 m burden and a 5 m spacing, using two different explosive types.

These tests will be conducted in Areas 1 and 2 in stopes A and B. See Fig 5. It is planned to use an inverted benching mining technique to extract Stopes A and B, using a 20 m bench height. In this way, it will be possible to monitor at least 3 different blasts under similar conditions of explosive type and hole pattern. Each blast has been designed to break approximately 15,000 to 20,000 tonnes. The planned variation in powder factor is from 0.9 kg/m$^3$ up to 2.3 kg/m$^3$.

Fig 5 Fragmentation test areas at the Luossavaara Research Mine.

Problems which can occur with respect to blast damage to the hanging-wall, due to high vibration levels, mean that it is necessary to include in the research program several experiments related to limiting vibration levels. There are at least two ways to limit the vibration levels from an explosive charge. Firstly, decreasing the length of the charge which, if the length isn't made too short, should have no effect on the peak level of
vibrations in the immediate vicinity of the blast, but should reduce vibration levels at some distance from the blast. Secondly, reducing the diameter of the charge by using a decoupled charge should reduce the vibration level not only in the immediate vicinity of the charge, but also at points remote from the blast, due to the lower total amount of explosive used, as compared to a fully loaded hole.

Out of concern for stability, it has been decided to use reduced strength charges as mining advances toward the hanging-wall. This decision has provided the opportunity to study the effect on the fragmentation of decoupled charges in areas III and IV in stope B, as well as using short charges separated by stemming material in stope C. It is also planned to study the effect of airspacers. See Fig 5.

The lower part of stope D will be used to evaluate VCR as a primary stoping method. See Fig 6.

Fig 6 VCR test in Stope D at the Luossavaara Research Mine.
Three different hole patterns will be used, as well as three different explosive types. The VCR-blasting in the D-stope is a joint project with SveDeFo. Researchers from SveDeFo have performed a number of crater tests at the Research Mine before the plans for D stope were finalized.

A primary goal of the Luossavaara project is to achieve high efficiency and reliability in blasting. Toward this end it has been planned to use large blasts. Among other things, the use of large blasts will reduce problems associated with backbreak, and minimize the time required for charging the blastholes.

In order to implement these large blasts, techniques for decking the holes must be introduced since the amount of explosive on each delay must be kept below a certain maximum. Consequently experiments have been planned for C stope which will study deckblasting, the purpose of which is to examine different stemming materials, different stemming lengths, and various initiation and charging systems in order to achieve a reliable blasting method.

CAREFUL BLASTING

As previously mentioned, rock damage due to high vibration levels can cause serious problems not only in the hanging-wall, but also in pillars and in surrounding production blastholes. Another goal of the blasting research in this project is to use careful blasting techniques in an effort to limit blast induced damage to hanging-walls and pillars.

This component of the research program will involve measuring the vibrations from a number of different blasts under similar rock conditions. A variety of methods will be used to assess the level of damage caused by each of these blasts and it is hoped that this type of data can be used to develop a prediction model for blast induced damage.
VIBRATION MEASUREMENTS

Vibration measurements are performed using triaxially mounted accelerometers in special monitoring holes in the hanging wall and also in the pillars. See Fig 7.

It has been planned to use 8 measurement points at any one time, 4 in the hanging-wall, and 4 in the pillar. This configuration should provide the opportunity to study the attenuation of the seismic signals with distance from the blast. The reason for measuring the vibrations both in the hanging-wall, and in the pillar, is that differences are expected in the seismic signals. These differences are expected due to the difference in rock properties between the ore and waste rock, and also the weak hanging-wall contact zone is expected to heavily influence both the amplitude and frequency content of the seismic signals generated by each blast.
To record and analyse these vibration signals, the Research Mine, with financial support from the National Board for technical development, has decided to use a digital measurement and analysis system.

The system, which has been developed by Geodynamik in Stockholm, consists of 24 measurement channels and a central processing/storage unit. Each measurement channel consists of analog input circuitry with variable sensitivity, an A/D-converter with 12 bit resolution, and has a maximum sampling rate of 200 kHz. Each channel also has a RAM-memory of 128 k samples.

The central processing/storage unit consists of a Digital 11/23 computer with a graphics terminal and external storage capability on either floppy or Winchester discs. The computer will be delivered with Software for advanced signal processing. Analog signals arriving at the input of this system are converted and stored in the RAM-memory. The electronics will continue sampling as long as there is signal arriving, or there is space left in the memory. After completion of the input cycle, data in the input buffers is dumped to the central processing/storage unit and the analysis can begin.

The analysis that has been planned will consist mainly of studying the attenuation of the amplitude and frequency of the seismic signals generated by the different types of charges and blasting methods. The vibration monitoring system will also be used to determine the time of detonation for each of the charges and thereby detect faults such as misfires.

Special analysis of this data is also planned in joint projects with other research institutes, and consultants.
ROCK DAMAGE

Three different methods will be used in the Research Mine for the measurement and recording of rock damage. The first method is the Rock Mechanic System, consisting of extensometers and shear strips, installed in the hanging-wall and in the crown. See Fig 7.

The second method incorporates a bore-hole TV-camera, for visually mapping the cracks and other flaws which occur, both in the hanging-wall and in the stopes. See Fig 7.

The third method employs a cross-hole seismic surveying technique to measure the extent of the rock damage.

The Research Mine is involved in a joint project with the Australian Mineral Industries Research Association Limited (AMIRA) in using cross hole seismic surveying to detect rock damage. This joint project will involve two researchers from the JULIUS KRUTTSCHNITT MINERAL RESEARCH CENTRE (JKMRC) coming to work at the Research Mine for at least half a year.

The method as it will be used in the Research Mine utilises an ultrasonic source and an array of detectors, in this case hydrophones in water filled boreholes. Cross hole scanning at regular intervals down the holes is conducted before and after a blast and various acoustic properties are calculated including p wave velocity, p wave risetime, signal amplitude and attenuation coefficient. The acoustic properties are changed due to induced damage and these changes can be used to define the extent and degree of damage. In the research mine, the crosshole technique will be used both in the orebody, in stopes B, C and D, using the production blastholes, and in the hanging-wall, using specially drilled 6 1/2" diameter holes. See Fig 7.

From the experimentation which is planned for the Research Mine at Luossavaara, by the Swedish Mining Research Foundation and other companies and research organizations,
it is believed that the primary goals of:

- controlled fragmentation
- controlled vibration levels and rock damage and
- high efficiency and reliability in Blasting,

can be achieved.
In your presentation you asked about the availability of (1) crack velocity, and (2) borehole pressure as a function of time during a blast. If these numbers are useful they are available on a code which should be described in a paper Thursday morning. There are available in the following way. The crack velocity is an average statistically calculated value for propagation of the cracks, which Langefors called the semistatic part of the blast. They come out as a function of the rock properties, the type of explosive you put in, decoupling and the geometry (burden). The pressure as a function of time does come out if you need it, it is not usually asked for. People are usually more interested in the semistatic, what we call the dynamic stress field, in America.

My question is: Are you aware in Sweden that these things are available? Would you be interested in getting access to them?

Agne Rustan:

I am very glad to hear that you have this information and of course I am very interested to take part of that information. I have a comment back to Roger Favreau. Have you done any experimental test to verify your computer model?

Roger Favreau:

The answer to this is of course yes and the very short paper on Thursday will just briefly glance some of this. The written paper which is in the book, in the late arrived paper section, does give more information on experimental verification as well as list some references where more information is available.
A STUDY OF BENCH BLASTING IN
RHYOPORPHYRY AT A REDUCED SCALE AND
THE STATISTICAL ANALYSIS OF THE REGULARITY
FOR FRAGMENTATION DISTRIBUTION

Senior Engineer Ma Bailing
Engineers Zeng Shiqi, Zou Dingxiang
and Guo Chuji
Maanshan Research Institute of Mining
China

ABSTRACT

Half scale tests have been undertaken to study the influence of delay time and spacing-to-burden ratio on fragmentation and particle size distribution in rhyoporphry. Statistical analysis, of eight blasted rounds with three holes in each round, shows that the mentioned parameters have significant influence on the fragmentation.

INTRODUCTION

The optimization of open pit blasting is a new field in the scientific research work of mines. With the help of mathematical means and achievements of computer techniques, it can calculate the complicated process of blasting fragmentation and obtain the correlation between the parameters of blasting and the results of blasting in order to get an optimum solution for making the lowest mining cost.

A three-dimensional mathematical model of the rock fragmentation process in bench blasting and a mathematical model of calculating various sizes of fragments from bench blasting in an open pit have been studied and set up by ourselves.

There is presently no perfect and practical method both fast and accurately for measuring the composition of the fragments on the spot, therefore, it is quite practicable to adopt a bench blasting experiment at a reduced scale to verify the calculated results from mathematical model. The feature of the experiment at a reduced scale is much close to that of commercial blasting in practice. Meanwhile, the composition of various fragments can quantitatively be measured.

In May to August 1982 we carried out an experiment with eight blasts in a quarry of Guan Shan Copper Mine, Jiang Shu Province, using the bench blasting method at a reduced scale. About 100 kg of explosives were employed in the experiment, including preparing the experiment site and cutting benches. Near 250 cubic metres of rock were blasted and cleaned. During the experiment, the geometric parameters and the result of blasting were measured. Besides that, the
physical-mechanical properties of the rock blasted and the parameter of the explosive used in the experiment were also measured, and a large number of data was obtained. Thus, all the raw data might be provided for verifying the result from the mathematical model.

Based on the data obtained from the experiment, through the statistical analysis, some laws of the size distribution have been gained in case of changing parameters of blasting.

A BRIEF DISCRIPTION OF THE EXPERIMENT

The main objective to do this experiment was to get the raw data required for examining the result of the mathematical model used in homogeneous rocks, that is to say, to get some physical-mechanical properties of the rock, some parameters of the explosives to be adopted, geometric parameter of the blasting and composition of fragments. The rock mass for the experiment requires good completion and unweathering, without visible joints and fractures. The quarry of the Guan Shan Copper Mine generally accords with the demands above. The rock type of the quarry is rhyoporphyr. Its physical-mechanical properties measured in place and from the samples are as follows see Table 1.

The other parameters of the experiments are diameter of the cartridge \(d = 32\, \text{mm}\), diameter of the borehole \(d = 42\, \text{mm}\), number of holes in each blast \(n = 3\), depth of each hole \(h = 1,20\, \text{m}\), Charged hole length \(h_C = 0,70\, \text{m}\), length of the stemming \(h_S = 0,50\, \text{m}\); Charge of each hole \(a = 0,525\, \text{kg}\); Spacing to burden ratio \(m = S/B\).

The non-electricity detonating system with the combination of blasting cap and shock tube was adopted. When instantaneous initiation was used, shock tube connected to each primer in various holes should have the same length. And when delayed initiation was adopted, the delayed time could be controlled by the length of shock tube. No 2 rock ammonia dynamite explosive was loaded in each borehole. The detonating velocity of a commercial cartridge (200 mm in length and 150 g in weight) was 2931,2 m/s on the surface and 3088,8 m/s in the borehole.

To begin with, the stripping of a weathering bedding on the primary bench of the quarry was done. An inclined slope of the bench was then reformed into two vertical benches (1,4 m high for each) with smooth blasting techniques. Single row of vertical holes was drilled along the bench. Before next blast, the bench face should be repaired by smooth blasting in order to eliminate the damage to the bench caused by the last blast, and to ensure a standard slope of the bench before the experiment. The parameters of each blast are given in Table 2.

The fragments of each blast was collected and dimension of each piece measured. Meanwhile, they were manually classified into 11 grades according to its biggest linear dimension of each fragment, see Table 3.

The amount (number of particles and the weight of each grade were also counted.

Eight blasts which were devided into five groups according to the spacing-to-burden ratios and the timing of initiation were fired during the experiment (besides that, four pre-test blasts were also fired). More than 15 000 pieces of fragments were measured. The data obtained could be seen in Table 4.
A. The distribution function of the size-weight

The distribution laws were studied based on the data of the size-weight in Table 4. We tried R-R distribution (i.e., Rosin-Rammler Distribution Function) and G-G-S distribution function (i.e., Getes-Godin-Schummann Distribution Function) into a linear equation by way of taking the logarithm of these equations, and then conducted regressive analysis of the test data.

(1) R-R distribution

\[ Y = 100 \left( 1 - \exp \left( -\left( \frac{X}{X_0} \right)^n \right) \right) \]

where:

- \( Y \) = cumulative weight per cent of the under size \( X \) (%)
- \( X \) = size of fragments (mm)
- \( n \) = distribution parameter
- \( n = \frac{\ln \ln 2}{\ln \left( \frac{\bar{X}}{X_0} \right)} \);
- \( \bar{X} \) = mean value of size distribution
- \( X_0 \) = distribution parameter of fragments, or the size value (mm) when the cumulative weight percent of the under size being \( 1 - \frac{1}{e} \% \) (approximate 63.21%)

There are two examples in Figure 1.

(2) G-G-S distribution

\[ Y = 100 \left( \frac{X}{X_m} \right)^B \]

where:

- \( Y \) = cumulative weight per cent of the under size \( X \), %
- \( X \) = size of fragments, mm
- \( B \) = parameter of size distribution
- \( X_m \) = the largest size of fragments, mm

There are two examples in Figure 2.
Figure 1. Size-weight distribution of fragmentation with Rosin-Rammler equation.
First International Symposium on ROCK FRAGMENTATION BY BLASTING
Luleå, Sweden, August, 1983

Figure 2. Size-weight Distribution of Fragmentation in G-G-S Coordinate

Table 5 relates the regression calculated results and the distribution parameters with the help of two kinds of the distribution functions.

The test of correlation coefficients showed that their correlation was very good on the significance level $X = 0.01$ when we state the distribution laws of every blasts as the two distribution functions.

B. The effect of spacing-to-burden ratio on the size-weight distribution

The regression analysis mentioned above present that the size-weight distribution of every blast just follow R-R or G-G-S distribution functions. Through linearization, they present a linear relationship in the relevant coordinate system, that is to say, the regression equation was a straight line. The intercept and slope of the straight line, generally in the view of the size-weight distribution, showed the varying extent of fragmentation.

A significance test (i.e., analysis of variance) was performed for the regression calculated results of the two distribution functions in order to study whether there was any significant effect upon the test results by changing S/B ratios. The analytic result showed that it significantly affected the size-weight distribution to change S/B ratios, see table 6.

Figure 3 presents the comparative results of the size-weight distribution based on different S/B values (m) when using instantaneous initiation.
As the value $m$ increased, the cumulative weight per cent of the under-size also increased in the case of the same fragment size, therefore, the result of fragmentation was obviously improved. For example, the size of fragment is taken as 500 mm, the mean per cent of cumulated weight of the under-size from two blast tests would be 31.75% when $m = 1$; it would be 51.46% when $m = 2$; and 67.49% when $m = 3$.

C. Effects of S/B ratios on the percentage of boulders.

An analysis of variance was performed to determine the significance of the effect of S/B's ratio on the percentage of boulders, where we took the sizes of 500 mm, 800 mm and 1 000 mm as the standards of boulders respectively, see Table 7.
We could draw an inference from the analyzed results of the significance tests, that the change of the percentage of boulders would be significant if we change S/B ratios. And the percentage of boulders would apparently decrease as the S/B ratio is increased.

D. Effects of delay time to size-weight distribution

The variance analysis was also carried out in order to test any significant effects to the size-weight distribution on two test levels, i.e. instantaneous and delay initiations, see Table 8.

The test results showed that the delay time did significantly affect the size-weight distribution of fragments.

Figure 4 shows the comparative results of the size-weight distribution between delay and instantaneous initiations.

In the case of the same S/B ratio, the fragmentation of delay initiation was better. The regressive line of delay initiation lay in the upper site of the regressive line of instantaneous initiation, that is, the percent of undersize increases in the same rock size.

Figure 4. Effect of Delay Time $\Delta t$ on Size-Weight Distribution (R-R Coordinate).
E. Distribution function of the amount (number) of fragments.

According to the data in Table 4, the distribution of the amount of every classified grade from various blasts was studied. There are two histograms of the amount of fragments in Figure 5.

We tried to make regression analysis of the distribution of fragmentation with the negative exponential function.

\[ Y = A \exp(-BX) \]

where:

- \( Y \) = amounts of fragments
- \( X \) = size of fragment, m
- \( A \) and \( B \) = parameters of distribution

This equation presents a straight line on the half-logarithmic paper. The negative exponential distribution curves in Figure 5, were drawn based on the result of regression analysis. There, their ordinate present the amounts of fragments \( Y \), and their abscissa present the classified size of fragments (i.e. the largest linear dimension of fragments in m.) Table 9 shows the results of regression analysis of fragments' amount (number) distribution in each blast.

Figure 5. Histograms and Distribution Curves of Fragments Amounts.
When the significance level was selected = 0.01, the test of correlation coefficient showed their correlation was fairly good.

In Figure 6 and Figure 7 charted on the base of the regressively calculated results, we may separately find the effects of both changeable spacing-to-burden ratios (m) and timing of initiation (Δt) (while other conditions were constant) on the distribution of the amounts of classified fragments.

Distribution Functions: $Y = A e^{-Bx}$

Regression Equations: $\ln Y = \ln A - Bx$

Figure 6. Effect of Different Values of m on Distribution of Fragments Amounts.

That is, (1) along with the increasement of value m, both the intercept and the slope of the regression line increase. Meanwhile, the amounts of small fragments increase and the amounts of large fragments decrease, therefore, the quality of fragmentation is improved. (2) The fragmentation of delay initiation is better than that of instantaneous initiation. This law has been showed in the both cases of $m = 1$ and $m = 2$. 

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Distribution Function: $y = Ae^{-Bx}$
Regression Equation: $\ln y = \ln A - BX$

$m = 1$

Figure 7. Effect of Delay Time $\Delta t$ on Distribution of Fragments Amounts.
CONCLUSIONS

1) In comparison with model experiment, the bench blasting at a reduced scale is much close to the full-scale bench blast in reality. Because of its reduced scale, less manpower, material resources and time may be needed and fragmentation can be measured quantitatively. Thus the experiment is much easy to be carried out. As viewed from verifying the mathematic model of rock blasting in homogeneous medium, the data obtained from various blasts might accord with the requirements.

2) With given parameters in bench blasts at a reduced scale, size-weight distribution may be presented by R-R distribution function

\[ Y = 100 \left\{ 1 - \exp \left( -\frac{X}{X_0}\right)^N \right\} , \]

or by G-G-S distribution function

\[ Y = 100 \left( \frac{X}{X_m} \right)^B \]

3) The distribution of fragments' amount (number of particles) follows the negative exponential function: \( Y = A \exp \left( -BX \right). \)

4) The distribution of fragmentation (both the weight and the amounts) may significantly be affected by S/B ratio and timing of initiation in the bench blasting at a reduced scale. Along with the spacing-to-burden ratios \( m = 1, 2 \) and 3, both slopes and intercepts of distribution curves successively increase. That is, at the same fragment size, the cumulative weight per cent and the amounts of fragments also successively increase, the percentage of boulders decrease and the average dimension of fragments reduces, thus fragmentation is improved. In the same way, it may be reflected from distribution functions that the blasting fragmentation of delay initiation is much better than that of instantaneous initiation in same conditions.
Table 1. Physical mechanical Properties of the Rhyoporphyr

<table>
<thead>
<tr>
<th>Unit weight</th>
<th>Compressive strength</th>
<th>Tensile strength</th>
<th>Shearing strength</th>
<th>Poisson's ratio</th>
<th>Modulus of elasticity</th>
<th>Longitudinal velocity</th>
<th>Transversal velocity</th>
<th>Attenuation coefficient of longitudinal wave</th>
</tr>
</thead>
<tbody>
<tr>
<td>t/m³ 3</td>
<td>kg/cm²</td>
<td>kg/cm²</td>
<td>kg/cm²</td>
<td></td>
<td>kg/cm² x 10⁹</td>
<td>in-place samples</td>
<td>in-place samples</td>
<td></td>
</tr>
<tr>
<td>2.56</td>
<td>1144</td>
<td>45</td>
<td>120</td>
<td>0.18</td>
<td>27.5</td>
<td>4100</td>
<td>3691</td>
<td>2357</td>
</tr>
</tbody>
</table>

Table 2. Blasting Parameters in the Experiment. Parameters for each blast with 3 holes.

<table>
<thead>
<tr>
<th>Test series (No)</th>
<th>Burden b (m)</th>
<th>Spacing s (m)</th>
<th>Timing of initiation t (ms)</th>
<th>S/B ratio</th>
<th>Number of tests</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>0.75</td>
<td>0.75</td>
<td>0</td>
<td>1</td>
<td>2</td>
</tr>
<tr>
<td>2</td>
<td>0.53</td>
<td>1.06</td>
<td>0</td>
<td>2</td>
<td>2</td>
</tr>
<tr>
<td>3</td>
<td>0.43</td>
<td>1.30</td>
<td>0</td>
<td>3</td>
<td>2</td>
</tr>
<tr>
<td>4</td>
<td>0.75</td>
<td>0.75</td>
<td>1.9</td>
<td>1</td>
<td>1</td>
</tr>
<tr>
<td>5</td>
<td>0.53</td>
<td>1.06</td>
<td>1.9</td>
<td>2</td>
<td>1</td>
</tr>
</tbody>
</table>

Table 3. Sieved classes.

<table>
<thead>
<tr>
<th>Grade</th>
<th>1</th>
<th>2</th>
<th>3</th>
<th>4</th>
<th>5</th>
<th>6</th>
<th>7</th>
<th>8</th>
<th>9</th>
<th>10</th>
<th>11</th>
</tr>
</thead>
<tbody>
<tr>
<td>Size Range (mm)</td>
<td>&gt;1000</td>
<td>&gt;200</td>
<td>&gt;300</td>
<td>&gt;400</td>
<td>&gt;500</td>
<td>&gt;600</td>
<td>&gt;700</td>
<td>&gt;800</td>
<td>&gt;900</td>
<td>&gt;1000</td>
<td></td>
</tr>
</tbody>
</table>

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Table 4. Number (amounts) and weight of fragments from each sieve class.

<table>
<thead>
<tr>
<th>Classified size</th>
<th>Test serie No</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>1</td>
</tr>
<tr>
<td>Less than 100 mm</td>
<td>amounts</td>
</tr>
<tr>
<td></td>
<td>weight</td>
</tr>
<tr>
<td>100-200 mm</td>
<td>amounts</td>
</tr>
<tr>
<td></td>
<td>weight</td>
</tr>
<tr>
<td>200-300 mm</td>
<td>amounts</td>
</tr>
<tr>
<td></td>
<td>weight</td>
</tr>
<tr>
<td>300-400 mm</td>
<td>amounts</td>
</tr>
<tr>
<td></td>
<td>weight</td>
</tr>
<tr>
<td>400-500 mm</td>
<td>amounts</td>
</tr>
<tr>
<td></td>
<td>weight</td>
</tr>
<tr>
<td>500-600 mm</td>
<td>amounts</td>
</tr>
<tr>
<td></td>
<td>weight</td>
</tr>
<tr>
<td>600-700 mm</td>
<td>amounts</td>
</tr>
<tr>
<td></td>
<td>weight</td>
</tr>
<tr>
<td>700-800 mm</td>
<td>amounts</td>
</tr>
<tr>
<td></td>
<td>weight</td>
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<tr>
<td>800-900 mm</td>
<td>amounts</td>
</tr>
<tr>
<td></td>
<td>weight</td>
</tr>
<tr>
<td>900-1000 mm</td>
<td>amounts</td>
</tr>
<tr>
<td></td>
<td>weight</td>
</tr>
<tr>
<td>Larger than 1000 mm</td>
<td>amounts</td>
</tr>
<tr>
<td></td>
<td>weight</td>
</tr>
</tbody>
</table>

Amounts = number of particles
Table 5. Table of the results of regressive calculation and of the parameters of size distribution

<table>
<thead>
<tr>
<th>s/B = m ratio</th>
<th>timing of initiation (ms)</th>
<th>Results of regression calculation</th>
<th>parameter of size distribution</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Slope</td>
<td>Intercept</td>
</tr>
<tr>
<td>R-R Distribution Function; Regressive equation;</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>$Y = 100 \left( 1 - \exp \left( -\frac{X}{X_0} \right)^2 \right)$</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>$\ln \ln \left( \frac{100}{100-Y} \right) = \ln x + A$</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>1</td>
<td>0</td>
<td>1.1998</td>
<td>-8.4186</td>
</tr>
<tr>
<td>2</td>
<td>0</td>
<td>1.4869</td>
<td>-9.5653</td>
</tr>
<tr>
<td>3</td>
<td>0</td>
<td>1.6555</td>
<td>-10.1798</td>
</tr>
<tr>
<td>1</td>
<td>1.9</td>
<td>1.2804</td>
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<tr>
<td>2</td>
<td>1.9</td>
<td>1.4871</td>
<td>-8.9571</td>
</tr>
<tr>
<td>G-G-S Distributive Function; Regressive equation;</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>$Y = 100 \left( \frac{X}{X_m} \right)^B$</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>$\ln y = \ln x + A$</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>1</td>
<td>0</td>
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</tr>
<tr>
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<tr>
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<td>0</td>
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<tr>
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<td>0.9377</td>
<td>-1.9119</td>
</tr>
<tr>
<td>2</td>
<td>1.9</td>
<td>0.9606</td>
<td>-1.7555</td>
</tr>
</tbody>
</table>
Table 6. Variance analysis for the parameters of the size-weight distribution on three different test levels separately with m = 1, 2 and 3.

<table>
<thead>
<tr>
<th>Distributive functions</th>
<th>Parameters tested</th>
<th>Origin of variance</th>
<th>Sum of squares</th>
<th>Degree of freedom</th>
<th>F TEST</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Intercept</td>
<td>Among groups</td>
<td>393.2291</td>
<td>2</td>
<td>P=12.16&gt;5.02</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Within groups</td>
<td>874.37582</td>
<td>54</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Slope</td>
<td>Among groups</td>
<td>160.6687</td>
<td>2</td>
<td>P=14.20&gt;5.02</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Within groups</td>
<td>305.5822</td>
<td>54</td>
<td></td>
</tr>
</tbody>
</table>

Table 7. The variance analysis of the percentage of boulders on three different test levels m = 1, m = 2 and m = 3

<table>
<thead>
<tr>
<th>Size of boulders (mm)</th>
<th>Test serie (No)</th>
<th>Percentage of boulders on each test level</th>
<th>Origin of variance</th>
<th>Sum of squares</th>
<th>Degree of freedom</th>
<th>F test</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>1</td>
<td>2</td>
<td>3</td>
<td></td>
<td></td>
</tr>
<tr>
<td>500</td>
<td>1</td>
<td>73.73</td>
<td>50.04</td>
<td>37.63</td>
<td>among groups</td>
<td>1442.5586</td>
</tr>
<tr>
<td></td>
<td>2</td>
<td>65.37</td>
<td>42.31</td>
<td>26.24</td>
<td>among groups</td>
<td>129.8012</td>
</tr>
<tr>
<td>800</td>
<td>3</td>
<td>55.42</td>
<td>29.61</td>
<td>6.12</td>
<td>among groups</td>
<td>1804.0974</td>
</tr>
<tr>
<td></td>
<td>4</td>
<td>42.30</td>
<td>17.70</td>
<td>7.18</td>
<td>among groups</td>
<td>157.5531</td>
</tr>
<tr>
<td>1000</td>
<td>5</td>
<td>43.85</td>
<td>20.33</td>
<td>2.20</td>
<td>among groups</td>
<td>1402.6621</td>
</tr>
<tr>
<td></td>
<td>6</td>
<td>37.67</td>
<td>7.66</td>
<td>7.18</td>
<td>among groups</td>
<td>111.7609</td>
</tr>
</tbody>
</table>
Table 8. Variance analysis of the size-weight distribution on two test levels, t = 1.9 and t = 0.

<table>
<thead>
<tr>
<th>S/B = m</th>
<th>Parameter of tests</th>
<th>Origin of variance</th>
<th>Sum of squares</th>
<th>Degree of freedom</th>
<th>Mean square</th>
<th>F test</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Intercept</td>
<td>between groups</td>
<td>0.4188</td>
<td>1</td>
<td>0.4188</td>
<td>( F = 11.50 &gt; 7.70 )</td>
</tr>
<tr>
<td></td>
<td></td>
<td>within groups</td>
<td>1.020</td>
<td>28</td>
<td>0.0364</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Slope</td>
<td>between groups</td>
<td>0.04105</td>
<td>1</td>
<td>0.04105</td>
<td>( F = 287.35 &gt; 7.70 )</td>
</tr>
<tr>
<td></td>
<td></td>
<td>within groups</td>
<td>0.0040</td>
<td>28</td>
<td>0.000143</td>
<td></td>
</tr>
<tr>
<td>2</td>
<td>Intercept</td>
<td>between groups</td>
<td>3.6217</td>
<td>1</td>
<td>3.6217</td>
<td>( F = 103.46 &gt; 7.70 )</td>
</tr>
<tr>
<td></td>
<td></td>
<td>within groups</td>
<td>0.9101</td>
<td>26</td>
<td>0.0350</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Slope</td>
<td>between groups</td>
<td>0.0037</td>
<td>1</td>
<td>0.0037</td>
<td>( F = 29.15 &gt; 7.70 )</td>
</tr>
<tr>
<td></td>
<td></td>
<td>within groups</td>
<td>0.0033</td>
<td>26</td>
<td>0.000127</td>
<td></td>
</tr>
</tbody>
</table>

Table 9. The regressively calculated results of the distribution of fragments' amount in each blast

<table>
<thead>
<tr>
<th>S/B = m</th>
<th>Timing of initiation</th>
<th>Results of regressive calculation</th>
<th>Standard residual error</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>( \Delta t ) (ms)</td>
<td>Intercept: ( B )</td>
<td>Slope: ( A )</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Distribution function: ( Y = A e^{Bx} )</td>
<td></td>
</tr>
<tr>
<td>1</td>
<td>0</td>
<td>-5.6118</td>
<td>6.4968</td>
</tr>
<tr>
<td>2</td>
<td>0</td>
<td>-6.7786</td>
<td>7.1655</td>
</tr>
<tr>
<td>3</td>
<td>0</td>
<td>-9.1532</td>
<td>7.9249</td>
</tr>
<tr>
<td>1</td>
<td>1.9</td>
<td>-6.5223</td>
<td>6.9774</td>
</tr>
<tr>
<td>2</td>
<td>1.9</td>
<td>-7.3588</td>
<td>7.2115</td>
</tr>
</tbody>
</table>
Questioner: Tim Hagan. Answerer: Lars Hermansson

Absolute deviation as you have used it is meaningful when the diameter and length of the blasthole have been selected. But when the general range of blasthole diameters is being considered, it is preferable to define the allowable deviation in terms of the designed burden distance. This is because the explosive charge is concerned primarily with the percent error in its designed burden distance rather than in the absolute deviation. For the general range of blasthole diameters, therefore, I suggest that blasthole deviation be expressed as absolute deviation divided by the designed burden distance. I would suggest that we call this the LIMITING RELEVANT BLASTHOLE DEVIATION.

Lars Hermansson:

Absolute deviation is a value related to designed hole pattern at a certain hole length. In our case 1% hole deviation at 100 m hole length is equal to 25% deviation in burden and spacing. Our goal for accuracy in production drilling was based on that deviation.

To put a value on deviation for a lot of holes in a bench round is not a good expression for a blasting situation because every hole in the round has its own volume to work on, and that volume is dependent on the firing sequence.

Questioner: Giovanni Rossi. Answerer: Lars Hermansson

I agree with Mr Hermansson about the usefulness of a concrete platform for conferring stability to drill rigs, like the one shown in his paper. This is one of the ways to achieve a better precision of blastholes. Can the author tell us which is the variation in drilling costs (for instance percentage wise) connected with the building of this platform?

Lars Hermansson:

In the research mine the preparation works (surveying, concrete floor etc) are done for 20-30 holes (= one cross cut) at the same time, before the drilling starts. Totally the cost for preparation works is about two percent of total drilling costs for these holes.

Questioner: Tim Hagan. Answerer: Per Anders Persson

As you know my previous papers on breakage mechanisms, I have been concerned for some time, that too much explosion energy is being needlessly wasted immediately adjacent to the charge. In your Figure 1 on page 119, you indicate that even with a charge of very small diameter in a strong rock (i.e., a granite) 30% of the explosion energy is lost in local crushing and/or plastic deformation. This is a very high potential loss. In view of the fact that many rocks are weaker than granite and that fully coupled charges are in common usage, do you consider that my concern about close-range energy losses is justified?
Per Anders Persson:

1. For optimum use of a given charge weight, certainly the charge should be decoupled, or rather the peak blasthole pressure decreased to decrease crushing loss, perhaps by making the drillhole diameter as much as double the charge diameter. But,

2. In practical rock blasting in hard rock the drilling cost per unit drillhole volume may approach the cost per unit volume of explosive. It may then be more economical to take the 30% loss in order to save 75% of the cost of drilling. In soft rock, on the other hand, drilling cost is often low, and then decoupling may be an improvement as pointed out by Dr Hagan.

Questioner: Len Margolin. Answerer: Per—Anders Persson

In terms of the energy that is being used up, I like to suggest that what you call category two probably is a dominant source of energy dissipation. In our calculations in oil shale we find out that surface energy is perhaps an order of magnitude smaller than the plastic dissipated energy measured. As you correctly has pointed out that scales differently with crack size and burden, than the surface energy does, so you must distinguish.

The other thing is that it occurs to me that energy dissipated in friction, that is due to "shear cracking" might be very considerable. I would guess in our calculation perhaps half the energy is dissipated as friction loss. These numbers seems to be consistent with the difference between experimental and theoretical calculation we have.

Per Anders Persson:

It is interesting that you point out this. It would really be equivalent to saying that the fracture toughness figures, the R in my equation, should be taken from a test with more of a compressive and also shearing action. The values we had available were a factor of 50 or 250 higher than that what I used here. It was a little bit of shock for us. We have always been used to considering radial cracks extending under the influence of splitting open forces. It may of course be that when you have jointed rock structure most of the fracturing is determined by the joint directions rather than by the radial directions from the borehole. In that case you would as Margolin says have much more of shearing deformation in cracks than splitting open. That may be the explanation for our funny results.

Questioner: Bibhu Mohanty. Answerer: Per—Anders Persson

The difference between empirical observations and the theoretical value in Per—Anders Persson's paper, to quantify energy relationships, could be dependant on if static or dynamic strength values have been used in the calculations. Did you determine the dynamic strength?

Per Anders Persson:

No, we did not. You would expect to get another factor of two, perhaps higher strength and therefore better agreement. I think that would be a good thing to look into.
Back some years ago, we were concerned about loss of energy in borehole dilatation when blasting in oil shale. In an ideal moment, viewing the borehole dilatation as a purely hydraulic expansion, we calculated that some 17% or so of the total energy could be partitioned purely into that. I discussed that with Dr Julius Ross, who confirmed the calculation but not liking it assuming the original data he went to look on old USBM springing data. He found from that looking on gelatine dynamite in granite in the same way, that slightly over 100% of the total energy could be measured. He concluded that some of the cavity has to be dust blown out prior to that they are measuring it, but it did seem that a lot of energy was located in local dilatation. A second observation from the Geokinetic site, volumetric expansion caused by Atlas Aqua-flow Explosive was measured in cylindrical springing tests both stemmed and unstemmed as been significantly less than from ANFO, despite the fact the Aqua-flow had probably between 3-3.6 times the detonation pressure of the ANFO.

Per-Anders Persson:

Rock blasting is indeed a complicated thing. The leakage of reaction products into cracks from a borehole, if it takes place to a considerable extent, would eat up a tremendous amount of available energy, because the volume that opens up is fairly large and also the possibility of thermal loss of heat to the crack surfaces is also very large. One of the first thing that Finn Ouchterlony did, when he started working at the Swedish Detonics Research Foundation, was to look on leakage of gas into a crack and we thought initially that this would be quite an easy problem to calculate the flow of the gas and the decreasing temperature due to the loss of heat by convection. It turns out to be very complicated. The only thing we could do was to state that because of the cooling, the crack would serve as a very much larger opening than it really was, because as the gas cooled its volume would decrease and therefore the crack could eat up much more energy than we thought initially. Finn started to look on the actual mechanism whereby gas could go into the cracks. We then began to feel that, at least when we have fully charged holes with high energy explosives, the plastic deformation and crushing of the material immediately surrounding the borehole would prevent leakage of gas into the cracks. Also the fracture mechanics calculation that Finn Ouchterlony did later, showed that the experimental crack length that we found in real life blasting in homogeneous rock was smaller than the theoretical. From this two observations we began to feel that gas does not normally leak in and fill the full length of cracks.

Discussion 2

Did the equation of state that you use, include the effect of solid products of explosion and if it did, did you do any tests on explosives with a very high amount of solid products say for example ANFO with 50 % aluminium?

Francis Otuonye:

One of the calculation actually was made with "Kinestik", which is a "binary" explosive and it has got some ammoniumnitrate. When we did the calculation on PETN we did not have any solid component. The binary explosive and some of them do have solid components.
Questioner: Bibhu Mohanty. Answerer: Kenneth Mäki

Is the Brazilian tensile strength always higher than the regular uniaxial "pull" test, even in pre-blasted material? If this is consistently so for post-blasted rock, it would appear that the Brazilian test is less sensitive to variations in sample size, sample distribution and even chance occurrence of a microcrack in the specific sample. Therefore, the Brazilian test should reflect the intrinsic tensile strength of rock more accurately than the pull test. Furthermore, in stress-wave induced cracks the stress-wave "sees" only a very narrow zone in rock at any given time and not the whole rock (or sample), and therefore, the pull test would yield unrealistically low tensile strength.

Since all such tests, compressive or tensile, employ load rates orders of magnitude lower than encountered in blasting, these results may not be relevant to fragmentation by blasting. We may have to design entirely new experiments to measure strength parameters relevant to blasting.

Kenneth Mäki:

The Brazilian tests reported in the paper gave approximately equal or higher average strength values than the uniaxial tensile strength tests, both before and after blasting. Yet not published results obtained at SveDeFo on a sedimentary limestone indicate similar relations. In the paper, reference is given to a French investigation where similar relations between Brazilian and uniaxial tensile strength test results were obtained. As you point out this indicates that the Brazilian test is less sensitive to specimen size and the contents of fissures and gives the strength of the unfissured rock matrix or in other words the intrinsic tensile strength. According to this line of reasoning, first suggested by the French scientists in question, the relation between the Brazilian strength and the uniaxial tensile strength may be regarded as reflecting a strength vs stressed volume relation given by the structuring of the rock material.

It may well be so that the Brazilian strength is perferrable in connection with blasting. For the future I also have a wish for a dynamic strength test. At this stage of development however we should not neglect any of the static tests due to the fact that they may be used for characterization of the small scale structuring of the rock and due to the fact that the characterization is performed in tensile stress fields.


Delays have proven to be of questionable benefit in pre-splitting in the field. While delaying the blastholes has been extensively tested successfully in post-splitting operations. What are your comments?

K.R.Y Simha:

We are aware that traditional pre-splitting rounds rely on simultaneous detonation of the charges and the few instances where delays were used no significant improvement was observed. We believe that delaying the detonations holds promise in improving the performance of any controlled blasting operation and in particular pre-splitting. It is still too early to scale our model.
studies to the field situations to suggest any quantitative recommendations but in the light of theoretical developments as well as the experimental results presented in the paper, we believe that delaying in presplitting, if designed properly will improve the operation of pre-splitting.

Tim Hagan: A comment to Simha's paper

It has been a lot of experience to show over the years that just about simultaneous detonation of adjacent presplit holes gives the best result. This is usually achieved in practice with a detonating cord trunk line where the detonation is about 7 000 m/s. If we even are in a very hard strong rock and might have a p-wave velocity of as high as 5 800 m/s, that really means that when each presplit charge detonates the stress wave from an adjacent charge has not arrived at that hole when it shots. This means that the stress waves do collide between the holes not necessary in the midway point but certainly between the holes. I think this is beneficial. People strive to get this simultaneity. If they cannot use a detonation cord trunk line, because it is an environmentally sensitive area, they tend to use electric instantaneous detonators and pass a high firing current through them so they get the simultaneous detonation that way. I really think that a very important part of Dr Simha's paper is looking at the delay between postsplit holes. Now in tunneling we cannot do presplitting in the perimeter of that tunnel, we have to put them on the final delay and those final delay detonators they usually have got a very appreciable scatter like ± 150 ms. When the day is arrived when we are using electronic detonators and we have got all this five sec delay detonators in the perimeter of the tunnel and they are really electronic timed it is going to tell what sort of timing accuracy we are going to need to get a good postsplitting result.

Roger Holmberg:

Of course we will get a better result if we can do the timing correct in tunneling, but still we very often have sensitive structure of whatever in the neighbourhood and perhaps it is not possible to use electronic detonators, because we actually will then fire quite a lot of holes in the same delay which will not be benificial for the vibration part sent into the rock mass.

Tim Hagan:

As many of you probably know Prof Fournery and his team at the University of Maryland are doing some big excellant work on diametrically notching blastholes to control the fracture direction. If such notching type blastholes could be provided in the perimeter of a tunnel, I think that we could employ with us notched blastholes and very accurately electronically delayed detonators to give us very smooth round faces, backs and walls of tunnels and overcome the potential overbreak problem.

Agne Rustan:

Controlled blasting tests in tunneling have been made also by the Swedish Detonic Research Foundation and Luleå University of Technology. We have for example found that a ultra short time delay between the perimeter holes gives less damage to the remaining rock. For a normal tunnel area of about 25 m², it will be 1.5 ms between the perimeter holes. The theory is that when you start to initiate the second hole the waves from the first hole should have escaped. Therefore it will be no co-operation of waves between the holes like in pre-splitting.
We have also studied the use of linear shaped charges to make notches in the perimeter holes. There is a research report outside in the exhibition area for those who are interested. Briefly I could say, after the field tests being done, that the principle of linear shaped charges works very well. We can get well defined notches and we can also proof that these notches starts the cracking. We can proof that the gas is penetrating into these cracks at an early stage, because the explosive we use is extrudable PETN and we get a black soot from this kind of explosive. After the blast we can see where the gas has penetrated. We have found that the gas penetrates into the artificial made cracks very rapidly and it will be more difficult for the gas to escape into the joints.

Questioner: Gert Bjarnholt. Answerer: Agne Rustan

In the paper given by Agne Rustan, I can see that detonating cord has been used. There are two implications. The PETN is surrounded by textile and plastic material and this prevents the gases to penetrate into the cracks. Secondly, looking on the charge concentration (g/m) it seems you are working in the low pressure range and almost close to the smooth blasting range. Is this something you have considered in interpreting your results? The charge is 5 g/m maximum and in a 5 mm borehole this gives about 5 kbar borehole pressure which is fairly low compared to production blasthole pressures (25 kbar) normally used.

Agne Rustan:

The charge concentration used in the model test where the influence of physical properties of homogeneous on fragmentation has been examined was 3 g/m. We had a big difficulty to find the correct concentration, because if we use a higher concentration we have to make bigger blocks and if we use a lower concentration the burdens will be too small. The charge concentration was therefore optimized as good as possible in this size of the blocks. Your comments could be correct. The blasts simulates perhaps more a smooth blasting case.

David Holloway:

About the mid crack coming from the free surface mentioned by Agne Rustan our observation has shown, that this crack is formed at the arrival of the p-wave or perhaps a certain time after that you have got the spall fracture.

Questioner: Göran Lande. Answerer: Agne Rustan

Since you have been using detonating cord, you introduce by the plastic coating in the borehole a filter. It may be the reason that the filtering of high frequency energy occurs. Maybe that is the reason for lack of smaller fragments in the models. Having a filter in such models you have obviously an impedance relationship. Do you think that having different interaction on the borehole wall you could achieve another relationship because you have shown a good relation with critical burden and impedance?

Agne Rustan:

Of course it could be a difference in fragmentation if you fill up the borehole with explosive instead of having it decoupled. I cannot tell how much that difference that will make on fragmentation.

Agne Rustan: I have a question to Göran Lande

How did you get the impression that we did not have any fine fragments?

Electronic publication by Daniel Johansson 2015-06-25 after permission by Editors-in-chief Agne Rustan and Roger Holmberg
Göran Lande:

I looked at the photos in the paper.

Agne Rustan:

The photos with big boulders shows the result achieved with a burden close to the critical burden and at this burden you should not achieve any fine fragments. When the burden is decreased, smaller and smaller fragments are achieved.

Tim Hagan: A comment to Agne Rustan.

I think, that in Australia and North America as well, about half of the explosive used annually is used in surface coal mining and sedimentary strata. These are often weak rocks and sandstone would usually be the strongest of these rocks. So the strongest rocks for half of the explosives used, is weaker than any of the rocks that Rustan has examined. I therefore respectfully would suggest and I think this would be an advance to the international value of the very good work you are doing, if you could get rock samples with lower tensile strength than that one you have considered, it would have had much wider universal appeal.

Mick Lownds: Written comment to the paper by Agne Rustan.

The relationship between mean fragment size from blasting and powder factor (specific charge) is an important one; it is central to any attempt to predict the dependence of total rockbreaking costs on blasting parameters.

It has been shown in the paper by Dr Rustan and co-workers that this function has taken many forms, and has displayed different sensitivities of mean fragment size to powder factor, in their and previous studies.

For the purpose of this comment, the function in question is generalized to

\[ x = q^{-b} f(v) \]  

where:

- \( x \) is some size characteristic of the fragmentation
- \( q \) is the powder factor (kg/m³)
- \( b \) is the exponent in the generalized power law.
- \( f(v) \) is a function of other blast variables which need not be specified here.

What aspects of model scale and real blasting influence the exponent \( b \)?

The authors of the last paper have shown that in competent rock \( b \) is different for different rock types, but they were unable to find a correlation between measured rock properties and \( b \). In addition, the range of \( b \)'s determined in model blasts (1.6 - 2.9) differs from the 1.18 recommended for actual blasting. This difference is believed to be important, and to reflect a general property blasting. Real blasting is done in material with pre-existing fractures, some of them induced by previous nearby blasts. The only fragmentation law which specifically recognizes pre-existing fracture is that originating from the Bond equation:

\[ W = W_i \left( \frac{P}{F} \right)^{\frac{4}{3}} \sqrt{\frac{1}{100}} \]  

where:

- \( W \) = energy needed to decrease particle size from \( k_{80} \) (before blasting) to \( k_{80} \) (after blasting)
- \( W_i \) = proportional constant = Bond's work index (kwh/t)
- \( P \) = \( k_{80} \) (after blasting) (mm)
- \( F \) = \( k_{80} \) (before blasting) (mm)
Rewriting equation 2 with the symbols of equation 1, with $x_0 = \text{pre-existing fragment size}$, gives

$$q = (x^{-\frac{1}{2}} - x_0^{-\frac{1}{2}}) \cdot f(v)$$  \hspace{1cm} (3)

$$b = \frac{3 \ln x}{\ln q}$$

$$= 2(1 - \frac{x}{x_0})$$  \hspace{1cm} (4)

This equation, plotted on Figure 1, shows that if all the work of comminution is to be done by the explosive ie $x/x_0 = 0$, which is the case for single shot model blasts, then $b$ will have its maximum value. Alternatively, when no comminution is required of the explosive ($x = x_0$), $b = 0$ which is to be expected. While the use of the Bond equation in this context is hypothetical, it does illustrate that the more faulted the material to be blasted is the smaller will the dependence of fragmentation achieved be on explosive used. This rule will apply when comparing single shot with multishot blasting in competent rock models as well: In multishot blasting the first row of holes conditions the medium via back-cracking so that for later rows $x/x_0 = 0$, and $b = .2$. Single model blasting should therefore not be expected to produce a useful relationship between fragmentation and specific charge.

![Figure 1](image)

**FIGURE 1**
Dependance of exponent $b$ on $X/X_0$. Bond Law of Comminution assumed.

I appreciate the comment from Mick Lownds about our model test and I agree to most of your comment but I want to clarify a little better the purpose of our model tests. The objective was to study what parameters are important when blasting in a homogeneous material. We have not expected to use the result for full scale blast, where the influence from joints is very large as shown by many authors.

Every little change in a model test could have influence on the exponent b for example if the decoupling or the delay between the holes or the explosive is changed etc. Of course one hole tests therefore is not a very good representation for full scale blast but on the other hand a standardized method to check the blastability, or better the relative blastability of rock, should not be too complicated to perform. Delayed model blasts probably better simulates the full scale but that would increase the cost for model tests enormously and the test would not have been undertaken under such circumstances. We have done the tests with geometric scale and specific charge in magnetite concrete as multiple hole blasts because it was cheaper to cast big blocks compared to diamond saw natural blocks.

The different value for b when blasting single or multiple holes could also be due to different blasting geometry of the holes.

Lowrison 1974 (1) has shown in a diagram that the exponent used in Rick's, Bond's and Rittinger's laws are dependent on particle size. In the blasting range Kicks law is therefore thought to be the best and for grinding Rittinger and Bond's laws, see also Figure 2.

\[ dE = -K \cdot \frac{dx}{x^n} \]

Figure 2. Energy consumption versus particle size according to Lowrison 1974.


When you calculated your explosive charge used in your experiments were you governed by some calculations on the magnitude of the pressure pulse in the borehole that was suitable for your model material?

K.R.Y Simha:

Charge calculations were primarily based on our experimental experience gathered over the years at the University of Maryland Photo Rock Mechanics Laboratory. In this particular experimental investigation the amount of explosive used was designed to initiate two cracks at the notches and provide sufficient energy to drive these two cracks to the model boundaries. At the same time the pressures generated are several orders of magnitude smaller than field conditions but we believe that the basic phenomenon remains the same, once the radial cracks have extended beyond the plastically deformed crushed zone surrounding the borehole.
Questioner: Roger Favreau. Answerer: Stephen Winzer

Stephen Winzer's paper & Bergmann's paper both presented fragmentation versus delay results of manifest high quality field work. Yet your paper concludes that delay can be too long while Bergmann does not. Can you comment?

Stephen Winzer:

First we must compare apples with apples. Bergmann's data is from blocks, rather than full scale tests. His data for blocks and my block test data compare quite well. I see a sharp change between zero and 1 ms/ft, with little change thereafter. The full-scale data are different, showing an inflection at 2.0 or 3.6 or 4.0 ms/ft depending on rock type or fracture density. This could be due to a number of factors, including problems with scaling from block tests, where burden velocities are 2-3 times full scale velocities; time available to develop and open fractures, (if fractures from the first hole are opened too much before the next hole fires, that portion of the bench does not benefit from stress waves from the next hole; and the relationship between pattern size and wave velocity in the rock).

Questioner: Dinis Da Gama. Answerer: Stephen Winzer

Please comment on the methods of evaluating jointing frequency in the rock face, and for the size analysis of fragments that you mentioned in your presentation?

Stephen Winzer:

Joints are mapped by traditional geological means (dip and strike, spacing, and comments about whether they are open or filled). Joints, bedding planes and other non-systematic fractures are mapped (traced) from the film or a 35 mm slide projected on the screen of a photo-optical digitizer. A rock quality index is then developed, using either number of fractures per unit of area, or length of fractures per unit of area.

Fragments are mapped in the same manner, by tracing the perimeter by hand with the digitizing courser, along with maximum and minimum length. These data are transmitted to the computer, which is programmed to calculate size distribution, etc.

Questioner: Gert Bjarnholt. Answerer: Stephen Winzer

What direction of strain are you measuring?

Stephen Winzer:

With two component gauges, we measure two mutually perpendicular directions in the horizontal plane. Six component gauges consist of two gauges on each of three faces of a tetrahedron (see Figure 1 in the text of the paper). The two
component gauge is mounted in the hole with one component perpendicular to the plane of the two or five boreholes (roughly the plane of the face). The six component gauge has one face of the tetrahedron directed at the plane defined by the boreholes. The six component gauge allows the full-field stress to be determined.

Questioner: Thomas Adams. Answerer: Stephen Winzer

Edwards at Los Alamos has reported a linear relation between mean fragment size and average joint spacing. Do you see such a correlation?

Stephen Winzer:

We have data from only two benches in two different rock types, therefore the data are insufficient to establish a trend. The average fragment size does differ between massive and fractured parts of the granite face, being smaller for the more heavily fractured face. Fragment size is also smaller in the limestone, which has a higher fracture density.

Questioner: Roger Holmberg. Answerer: Stephen Winzer

Regarding the slide you showed, we saw a drop in fragment size when the delay of 20 ms was used. Is it a real delay time effect or can it be dependent on the structure geology? The real fracture frequencies are usually hard to observe from the film. You also showed the slide from blasts in limestone when the optimal delay time was 30 ms. If the blasting geometry is the same for the two rock types how do you explain such a long extra time, 10 ms, is needed in limestone?

Stephen Winzer:

1) Compare the fracture densities for the 20 and 40 ms delay tests. They are almost the same, indicating that the large difference in size at 80% passing is due primarily to delay effect. There will also be some effect due to structure, of course, but in this case, where explosive energy and detonation velocity are constant the effect may be less noticeable than where the explosive varies, as the stress available for fracture initiation and growth is constant.

2) The limestone peak, at 36 ms, should be compared to the peak (inflection point) for the fractured portion of the granite face, as the fracture densities are more nearly similar. Differences in delay for optimum fragmentation are likely due to differences in wave speed, attenuation, fracture toughness and crack propagation velocity between different rock types. The effect of these variables on fragmentation at different delays needs to be worked out and experimentally verified.

Questioner: Mohinder Saund. Answerer: Stephen Winzer

You have shown two cost curves for loading and crushing. The decrease in costs is obviously due to improved fragmentation. Did you correlate the boulder size, the size of equipment and the costs? If so, could you please explain the results?
Stephen Winzer:

The equipment size was constant for a given bench in most cases. Any changes in equipment size were recorded, as were different operators during the course of the program. The primary crusher was a 42 inch. gyratory. Oversize were recorded by the shovel or front-end loader operator, but these data were often unreliable. Loading rate changes were principally due to improved muck distribution, and to a lesser extent, resulted from fewer oversize. Crushing time was heavily dependent on fragment size (larger fragments, longer crushing time for a truckload). Costs decreased for both operations, however cost of loading is dependant on other factors as well as loading rate, therefore the per cent change is not the same.

Questioner: W.H. Wilson. Answerer: Paul Worsey

Have you designed and built an experimental or prototype model of electronic detonator? If so, have you tested it in environment of shock, high pressure, and electromagnetic radiation? What "family" of electronics was used (TTL, ECL, etc)?

Paul Worsey:

I would like to answer the first part of this question, but present commitments would both compromise myself and possibly some explosive manufacturers.

However the micro electronics will be CMOS based most probably of Super-micron technology with either EPROM or EEPROM utilized.

We are assured by silicon foundress that microchips are resistant to high shock levels and any problems that may arise would be confered to external connections. The unit can be easily made to the same resistance as existing electric detonators for quasistatic loading and EMR is not thought to be of any serious concern.

Questioner: Peter B. Nahan. Answerer: Paul Worsey

When do you foresee integrated detonators available at a reasonable price?

Paul Worsey:

With the present economic climate I do not foresee any on the market in the next two years perhaps longer due to investment decisions. They will be marketed at a competitive reasonable price. If this was not possible, certain Explosive Companies would not be presently engaged in development programs.

Tim Hagan:

As Paul Worsey has stated, currently available delay detonators occasionally fire out of sequence. This leads to sub-optimum blasting results. But sub-optimum blasting results are also obtained where a late-firing front row charge is situated in front of an early firing charge in the second row. If cumulative scatter prevents good progressive relief of burden, fragmentation and muckpile looseness will be reduced and overbreak, ground vibration, instability potential and misfire rate will increase.

Questioner: Gunnar Almgren. Answerer: Paul Worsey

What is the response from the mining and construction industry concerning your detonator?
The response over the previous six months has been excellent from explosive users with impressive interest accompanied by a great flow of questions.

The majority of explosive manufacturers have now started their own research in this area, but they remain somewhat cautious due to the high capital investment involved.

Questioner: Terry Liddy. Answerer: Paul Worsey

In Broken Hill we have been very interested in electronic high accuracy detonators for several years and have had a lot of discussion with ICI in Australia about the development and eventual trial of these detonators. One question that has been asked is how accurate does a high accuracy detonator have to be. In your paper you mentioned ± 10 μs and ± 1 ms for present detonators (which I think is rather generous). In fact, if we could get detonators that accurate, I think the need for the development of an electronic detonator could be much better evaluated. There is a lot of expectation about the benefit of electronic detonators, all of which are not justifiable. For example, the example used in your presentation of two holes close together causing desensitization. It is interesting to note that the stress wave released by one charge in massive u g rock such as Broken Hill. With a stress wave velocity of 6000 m/s, the time for movement from one hole to next takes 10 μs for holes a few inches apart. Thus you still may not get any change in your problems. It is very important to realize the application and limitations of the system and not expect it to solve all the problems. Perhaps you would care to comment on that and also the accuracy improvement as related to the cost of development of that accuracy which is of course very important if the detonator is to be an economic proposition for both mines and explosive manufacturers.

Paul Worsey:

Required accuracy obviously depends on the type of blast that is being undertaken. Obviously one can never be too accurate, certainly for seismic considerations and research to name but a few.

The ± 10 μs refers to the accuracy that can be achieved by the electronic package without any special considerations in design and the ± 1 ms to seismic caps (zero delay). An ordinary zero delay blasting cap for instance, could be expected to take up to 8 ms to fire with a spread of up to as much as 3 ms. High number delay caps have been recorded to fire out of sequence and thus ± 10% in delay inaccuracy can be looked on as a very conservative figure. The misfire example shown in the presentation was most probably due to a reduction of the charge size from 1 1/4" to approximately 3/4", the original hole diameter reduced by 50%. However one cannot be totally certain of this and this is a line of research that we will be engaged in presently.

For high accuracy the electronics of the integrated electronic detonator present no problem, however design modification will have to be made with respect to the bridgewire match-heads compared with currently used static insensitive blasting cap match-heads due to their relatively high firing times and scatter.

This development will only represent a minute proportion of the capital outlay of development, testing and construction of a sophisticated highly automated production line.
First International Symposium on ROCK FRAGMENTATION BY BLASTING
Luleå, Sweden, August, 1983

Questioner: Roger Holmberg. Answerer: Paul Worsey

With the electronic caps you sketched, can you see any difficulties to let the last detonated cap send a message to its neighbour (next delay) that it can start the "count down"? If the previous cap failed can "the count down" be interrupted?

Paul Worsey:

It would be easy to incorporate this into micro-chip design. However as the IED can survive on its own capacitive unit it would not know to shut down if the leadwires were to be cut by ground movement or fly from previous holes. In effect this could mean that because a shut down signal was given not all of the units could respond, thus perhaps giving a worse result. In addition I would like to reference to Frank Chiappettas paper in that to avoid cut offs, all blasting caps in a round should be initiated simultaneously.

Questioner: Roger Favreau. Answerer: Richard Dick

Could your gauge be used to measure horizontal velocity and has this been done?

Richard Dick:

There is no physical reason not to use the gauge in the horizontal direction as long as there is some component of velocity in that direction. We have used the gauge in angled holes in tests that had vertical charges. The signal records were clean but complicated.

Questioner: Dane Blair. Answerer: Richard Dick

What was the frequency response of the velocity gauge as this is highly relevant to any integration performed on the record?

Richard Dick:

The frequency response was probably 100-200 kHz. In our experiments the frequency response of the gauge was much lower than the recording electronics. The gauge needs more study to determine the response in the frequency domain to accurately characterize the gauge. This will probably be done in the future.

Discussion 6

Questioner: Stephen Winzer. Answerer: Olle Carlsson

Question 1) Did you try various lighting angles and intensities to test the effect on distribution?

Question 2) Did you calibrate against samples with different, but known, size distribution?

Question 3) Did you investigate the effects of shape of particles on apparent distribution?
Olle Carlsson:

Answer 1) The lighting angle is of great importance. The boundaries of the fragments have to be sharp.

Answer 2) The equipment has been tested on different and known size distributions in model scale, but we have not yet found a general way to calibrate it.

Answer 3) We assume that the shape of the fragments are uniform.

Questioner: Zeev Jaeger. Answerer: Olle Carlsson

We tried to compare the fragment distribution (of a crater test) achieved by image processing method with the results of sieving analysis, and found the fit is not so good. One of the difficulties seems to be the passage from fragment area to fragment volume which is strongly dependent on the particles' shape. What is the ratio volume/area in your case? Does it vary with rock type?

Olle Carlsson:

We assume that the length fraction is equal to the volumetric fraction. This is true if the fragments are uniform and the length is taken at the same place on each fragment.

Questioner: John Grant. Answerer: Olle Carlsson

Using photographic techniques to represent the fragmented rock, at Mt ISA MINES, we found the following problems in attempting to automate the technique.

Problem 1) An insufficient grey range was available to sufficiently discriminate discrete particles and correct resolution could not be consistently achieved.

Problem 2) The drawpoint material observed visually at a surface was not representative to the fragment distribution due to differential rilling for underground stopes.

Problem 3) Dust setting time between L.H.D cycles was insufficient to permit continuous photography. Loading actually had to stop for five minutes.

Olle Carlsson:

Answer 1) The light angle must be arranged so the fragment boundaries are sharp. We mounted the light close to the TV-camera.

Answer 2) We calculate the mean value of a number of pictures when the loader is working, and therefore we get a result from the whole round.

Answer 3) This problem was not so big when we tested the equipment. We mounted the TV-camera in the back near the muckpile.
To what do you attribute the course fragment size. Could it be boundary effects.

Gert Bjarnholt:

I attribute the fairly course fragment size mainly to a large burden.

Equation (1) pertains to the front of a plane wave. How can you utilize this equation for an expanding cone-shaped wave and especially to the peak of a detonation wave?

Gert Bjarnholt:

Considering that the measuring station is located at a distance of 70 borehole radius from the charge the plan wave assumption should be rather good. Using equation (1) relating radial peak strain and peak particle velocity I have only used data from the very front of the stress wave. This wave has a rise time of about 20 μs and the stress is in the linear elastic range. Detonation waves in condensed materials are normally characterized by reactive shocks with rise times less than 1 μs and peak stress levels an order of magnitude above the linear elastic range for most materials.

For pressure pulse rise times that are very small the full dynamic problem of wave interaction with a fluid filled borehole must be solved using Fourier-Bessel techniques. We have recently solved such problems and do not end up with the equations you have shown. How did you derive such equations?

Gert Bjarnholt:

The formula was derived using 2-dimensional finite difference code modelling of a sharp shock entering a fluid filled borehole. The details are published in a SveDeFo report DS 1981:2 "A numerical investigation of a method for stress wave measurement in rock". (1981)
First International Symposium on ROCK FRAGMENTATION BY BLASTING
Luleå, Sweden, August, 1983
SESSION 4
FEM and Finite Difference-Codes
Chairman: Gwynn Harries, vice chairman: Per-Anders Persson
Discussion 7

Questioner: Stephen Winzer. Answerer: Len Margolin

1) In view of Fourney's joint initiated fracture mechanism confirmed to occur in rock macrocracks or foci of impedance mismatch (bedding planes) must be taken, into account in fracture/fragmentation models. Can your code do this? How is it done?

2) Could your code simulate effects of errors in firing time, or variation in energy or detonation velocity of explosives?

Len Margolin:

1) The effects can be put in two different ways. Macroscopic joints can be modelled as discontinuities (slide lines). In the 2-dimensional code some geometries are not available. Particularly a vertical joint is not possible to simulate in a cylindric symmetric code. The other way it can be done is of course statistically as part of the crack distribution.

2) The code is capable of modelling explosives of arbitrary detonation velocity and energy output. (Only reasonable values can give reasonable answers and even this is not guaranteed).

Questioner: Kenneth Maki. Answerer: Len Margolin

The approach to regard the rock mass as a precracked medium is very interesting. The question is however: How well can you model a rock mass consisting not only of microcracks and fissures but also of larger size joints, shear fractures etc? If such structures could have been included following a detailed geological investigation at the blasting site maybe the agreement between the predicted fractured volume and the real crater volume would have been better?

Len Margolin:

I think we got very excellent agreement with the measured craters. If you look on the top of the crater it varies 20–30% in its own dimension even it should be symmetric in theory and this is due to inhomogenities in the rock. We can include the macrostructure if someone want to map it. In oil shale it is very difficult to map almost anything. Oil shale is just a terrible material.

Questioner: Zeev Jaeger. Answerer: Len Margolin

Question 1) For convergence sake the code should work with small stops of time and space. So, the distribution of cracks is cut off (the cell size). How do you include the effects of large cracks (joints, faults etc)? How is it assured that the BCM E.O.S is not cellsize dependent?
Question 2) Experiments with small strain rate result with a few cracks (6—12) around the borehole. Is the code capable of reproducing such results?

Question 3) If one is trying to predict the stress wave in a new type of rock using your code, what are the rock material data he has to supply? Is it needed to measure again the distribution of cracks in this rock?

Len Margolin:

Answer 1) Most solid dynamic codes exhibit mesh dependence. Special work was done with shale to identify and overcome this. The code does not now exhibit such a dependence.

Answer 2) Yes. The crack statues of the competent material is a material property as important to simulations as the wave speeds and density.

Answer 3) To characterize the material it is necessary to know what the physical properties are. If it is a new material you must go and measure the crack distribution.

Questioner: S.M Hochberger. Answerer: Len Margolin

Few others in the last few days have stressed the importance of shear in their calculation of rock fragmentation. How important is it and how can it affect spacing—burden etc?

Len Margolin:

Experiments of blocks broken by explosives where the cavity is seen by X-ray, show the need for shear cracking. More details can be given in private communication.

Questioner: Alexander Spatthis. Answerer: Len Margolin

Is shale code available for use by other researchers?

Len Margolin:

Yes, the code is available to others. In addition we would like to encourage interaction and collaboration with anyone who has data or experimental diagrams. I would warn you however that the code is not to run on small machines. It is very much an artifact of our technology on computer technology. (SHALE runs on a CDC 7600 or a CRAY).

Questioner: Stephen Winzer. Answerer: Tom Adams

Question 1) Can you simulate a bench type configuration (2 free faces, the bench face and the top) with more than one columnar charge?

Question 2) How does the predicted stress field, and predicted fracture differ for explosives with \( V_{det} > V_p \)-wave and \( V_{det} < V_p \)-wave?

Question 3) Does your code predict size distribution? How does it compare with measured size distribution?
Tom Adams:

Answer 1) The bench type configuration is inherently 3-dimensional, so we cannot do this rigorously until the 3-dimensional code is ready. In 2-dimensional, we can still look at basic phenomenology by making runs with a cylindrical charge in a cylinder of rock or runs in plane strain.

Answer 2) For \( V_{\text{det}} < V_p \), we can run a variety of explosives and see. The ratio \( V_{\text{det}} / V_p \) changes the angle of the conical shock wave from a long cylindrical charge, so it changes the results qualitatively. For \( V_{\text{det}} < V_p \), stress-wave propagation strongly affect the performance of the explosive?

Answer 3) The code will be able to calculate fragment size statistics from the crack distribution, but this has not been implemented yet. It will be done soon.

Questioner: Albert Funk. Answerer: T.F Adams

If the energy of the explosive were maintained constant and the gas volume were cut in half, how would the fragmentation be affected?

Tom Adams:

In the code at present, this would simply change the inertia of the explosive gas and would probably not strongly affect the results. In reality, and in the code when it takes the penetration of gas into the fracture network into account, the amount of gas available to extend cracks and tumble the rubble should be significantly reduced.

Questioner: Zeev Jaeger. Answerer: Tom Adams

Question 1) Did you compare fracture densities or fragment distributions calculated by the code with quantitative experimental results?

Question 2) The direction of the pressure has an ordering effect on cracks (For example, under tension crack direct themself perpendicular to the tension stress). Is the angular behavior of the code predicted crack density similar to the experimental? For example, are there more vertical cracks (parallel to borehole) then horizontal ones?

Tom Adams:

Answer 1) Comparisons with fracture densities have not yet been made, in part because there is no suitable experimental method to measure cracks or flaws inside apparently solid rock. We have tried to use cores to look for in situ flaws, but the flaws are very difficult to identify. Cross-hole ultrasonic techniques, as are employed at the Loussavaara Research Mine look very promising. Regarding fragment size distributions, the shale code has such information in the crack densities. The actual calculation of fragment size distribution has not yet been put in the code, but will be done soon. (See also answer to S. Winzer's question No 3).
Answer 2) The bedded crack model (BCM), follows the cracks in a statistical sense, so individual flaws are not described. However, "BCM" does calculate crack distributions in various orientations. In the crater calculations, I do see enhanced radial and circumferential cracking directly outside the explosive and enhanced horizontal cracking in the spall layer near the surface.

Questioner: S.M Hochberger. Answerer: Tom Adams

You stated that stemming is important in the blast planning—how important in the calculation of burden-spacing explosives quantity etc?

Tom Adams:

The stemming will be very important in situations where the gas must be held long enough for it to penetrate into the shock-induced fractures.

Questioner: Keith Britton. Answerer: Tom Adams

Question 1) Your slides and film do not show crack coalescence to form macro-fractures. What is needed to add this and hence predict loosening planes and fragment geometries?

Question 2) The literature and field experience show that under most circumstances, presence or absence of stemming has but minor influence on effects, about 10%. Very little difference is seen in oil shale, in borehole dilation—some in fragmentation—slightly more in bulk motion. Is this field finding consistent with your modelling results? If not please comment.

Tom Adams:

Answer 1) The bedded crack model (BCM) takes fracture into account by reducing the effective elastic moduli (including the effect of the rate of change of the moduli). Crack coalescence and separation of layers occur when the effective modulus for that orientation goes to zero. By analogy to self-consistent field solutions, the effective elastic modulus vanishes as the third moment of the crack size distribution reaches a certain critical value. Some calculations of spall have been done by B.W Smith at Los Alamos taking crack coalescence into account this way. The spall separation and ballistic motion of the spalled layer was quite satisfactory.

Answer 2) Recent tests in oil shale with single boreholes also show little influence due to stemming performance. This conclusion is different from that inferred by us from the multiple borehole test.

I would prefer to appeal to mining engineers and blasters to tell us modellers how important stemming performance is for fragmentation. We need to tap the large body of empirical information that field blasters have accumulated.

Questioner: Bibhu Mohanty. Answerer: Tom Adams

Question 1) Could you not resolve the differences between field results and those from SHALE (for single-hole blast) through additional tests, such as changes in P, S velocities, attenuation and porosity or permeability?
Question 2) How well does SHALE reproduce the fracture patterns in the gross scale, (e.g. radial cracks and onion ring, cracks observed on surface from a crater shot below optimum DOB)?

Tom Adams:

Answer 1) Additional laboratory measurements of the initial flaw distribution with optical microscopy or with the use of a scanning electron microscope (SEM), would be useful. However, the differences between the calculations and the field results are more likely due to the high pressure explosive product gases as they penetrate into the shock-induced fracture network.

Answer 2) The shale code does predict successfully the distribution of cracks as a function of crack orientation. The description is statistical, so individual cracks are not followed in detail.

Questioner: Zeev Jaeger. Answerer: Stuart McHugh

Your code needs several unusual coefficients. These parameters are not regular material properties. Trying to use SRI-code for a new rock, how can one measure the required constants?

Stuart McHugh:

It is true of course that the fracture parameters, coefficients in the NAG (Nucleation And Growth) relations, are not regular material parameters. However most fracture models require a set of material dependent parameters to describe fracturing. The NAG model parameters are material dependent and are treated as the subset of material properties that control fracturing.

The SRI NAG (or NAG-FRAG) code has been used for a wide variety of materials. Ashfall Tuff and Devonian shale are described in this paper and the fracture parameters for these two materials are given in this paper. In addition, work has been done on oil shale, Arkansas novaculite, basalt as well as many different plastics, metals, ceramics and other materials. This work and the relevant NAG parameters are documented in SRI reports and papers.

Measuring the NAG parameters can be done in different ways. The first is an iterative procedure in which a set of NAG parameters is chosen, computational simulations of fracturing are performed and compared to experiments, and then the NAG parameters are changed and the simulations rerun until one has a set of parameters that allow the fracture pattern to be reproduced. This procedure is most effective when only a couple of NAG parameters are altered. (Parameters T8 through T14 are fixed by considerations of fragment geometry and are not usually altered. Parameters T3 and T7 are fixed by measurements of the initial flaw distribution. Parameters T2 and T5 are simply thresholds for nucleation and growth and can usually be fixed by laboratory measurements. Usually only T1, T4 and T6 need to be evaluated in an iterative procedure). The second method is to do carefully controlled plate impact tests in which a tensile stress pulse is propagated through the material. The amplitude and duration of the pulse are carefully controlled, and if the number and lengths of the resultant cracks are counted, this fracture data can be related to the stress data to determine the NAG parameters. Although the second method is preferable it is also much more time consuming, expensive and requires laboratory equipment and instrumentation often not readily available. A third method of determining the parameters is to use a combination of laboratory measurements with semi-theoretical arguments to estimate some of the parameters, and then the iterative method is used to evaluate the remaining parameters. Each of these methods has been used and is documented in various SRI reports.
The nucleation rate coefficient $T_4$ varies from $3 \times 10^{-7}$ for Tuff through $3 \times 10^4$ for Devonian shale to $1.6 \times 10^{10}$ for Oil shale (ref 1). The difference in order of magnitude appears surprisingly large, and indeed, unphysical. Could you comment please? Can this parameter be measured in a model independent way?


Stuart Mc Hugh:

There are two reasons for this variation in the magnitude of the parameters. The first is that the rock itself varies substantially from region to region in the geologic formation. For example, the MultiFrac experiments were conducted in a thick layer of Ashfall Tuff that, within the layer, was supposed to be uniform, isotropic and homogeneous. However one MultiFrac test produced only compaction of the borehole region and no observable fracturing (which would give a nucleation coefficient, $T_4$ of zero) and another test in the same formation only a few tens of feet away produced substantial fracturing (and hence a nonzero nucleation coefficient). Obviously not all rocks (or even parts of the same geologic formation) were created equal.

The second reason is that there are different versions of the NAG model that use different assumptions about the distribution of inherent flaws (depending on the level of detail that one wishes to use in the material characterization). The use of the different models usually leads to only slight variations in the NAG parameters.

The major differences in the parameters usually are due to variations in the rock properties.

The parameters can be measured independently but the measurements require laboratory equipment not often readily available.
Discussion 8


Question 1) What is the dynamic disintegration criterion which is used by BLASPA?

Answer 1) I have tried a variety and more or less settled on the tension stress impulse.

Question 2) What are the physical assumptions you made in the code? Please compare them to the ones used by S.R.I or Los Alamos model.

Answer 2) The answer to this would fill a book: essentially they are the same as the other codes, namely conservation of momentum, energy etc. Put in another way, all these codes are solutions of the 3-dimensional wave equation plus boundary conditions plus change in state of the material. The main differences with Los Alamos is that I use a tension criterion, not Griffith's theory.

Question 3) Is BLASPA code free for use by the public like S.R.I and Los Alamos codes, or one has to buy it?

Answer 3) Normally access to BLASPA is leased for a given number of simulations.

Question 4) How many experiments in the mine one needs in order to fit all the parameters needed for Blaspa?

Answer 4) None. The study starts from the production blast procedure.

Questioner: Algot Persson. Answerer: Roger Favreau

Can your program be used to simulate the effect of crack planes in the rock mass?

Answer: Yes, this is done when the mine feels that crack planes are a key concept. When planes are included their direction must be known, and the computer costs do go up somewhat.

Questioner: Peter B Nahan. Answerer: Roger Favreau

How does BLASPA handle the 3-dimensional blast situation with respect to time?

Answer: BLASPA is fully 3-dimensional for bench blasts or crater blasts and time is continuously a variable. For example one could imagine a gauge embedded in the blast; BLASPA can, within its precision, output as a function of time of the...
various stresses and movements. In practice and usually for the sake of consistence, answers are outputted only at certain times and places. As computer films becomes easier to achieve, however their use will probably make more evident the time aspect intrinsically built into BLASPA.

Questioner: Lorenzo Zago. Answerer: Roger Favreau

Unless one really believes that BLASPA is the ultimate answer to bench blasting design problems, what are the present major limitations to the actual reliability of the program? And in which directions do you envisage the future program developments and/or improvements?

Roger Favreau:

BLASPA is not the ultimate answer, it is improving all the time. Its chief advantages is that, already at this time, it has been found capable of lowering blasting costs.

The main direction of development at the present are: 1) An improved subroutine to blast coal seams; 2) An improved buffer blast subroutine; and 3) Further development of the underground BLASPA.

Regarding the present limitations they are largely computer problems: costs, accessibility at the mine, a desire of the part of users for more graphic outputs.

Questioner: Agne Rustan. Answerer: Roger Favreau

In your presentation you mentioned that most of the gaps I have mentioned in my keynote address could be filled with the BLASPA model.

As an example: Could you show me the theoretical formula to calculate the resistance of stemming versus the size distribution, cohesion—, and density— of stemming and finally the friction between the stemming and the borehole wall?

Roger Favreau:

The stemming effect went into the computer programme BLASPA many years ago so I do not remember the formulas used to account for the variables, though I could dig them up. But this misses the point; the effect of stemming is in BLASPA as a secondary mechanism, and results correlate with field results. The point of view of a practical blasting engineer is a desire to know how to achieve his job at production blasting; so he uses BLASPA to help him decide how much stemming to put in. He does not care so much about the equations used, as he cares about help to make better decisions.

Questioner: Claude Cunningham. Answerer: Roger Favreau

In your fragmentation contours which work on tensile strain, a complete absence of breaking around the sub drill zone is shown. On the other hand we know there is complete shattering and wide fragmentation around the hole, even if the toe itself does not pull. Please comment!
Roger Favreau:

If you look at the written paper you will see that there usually is breakage below the hole. In fact there always is, as shown by the "crushing" in Figure 1 of the written paper; I just have not shown it in the figure for ANFO, because it is not very important for practical blast design.

Questioner: Giovanni Rossi. Answerer: Claude Cunningham

In the Rosin-Rammler representation size distributions of blasted rock are in some cases S-shaped or exhibit a variable curvature, thus originating problems as far as the choice of the right value of fragmentation index \( n \) is concerned. How can this drawback be overcome?

Claude Cunningham:

Your question is related to the finding that fragmented rock does not always conform to the Rosin-Rammler curve. I agree that, especially under certain conditions, the curve may not apply. The Kuz-Ram model operates however on the assumption that Fragmentation does conform to the Rosin-Rammler curve, and I emphasize that this is a "first aid" approach which must not be looked to for accurate analysis.

Questioner: Peter Lilly. Answerer: Claude Cunningham

In our open cut iron ore operations in the Pilbara I have found that the Kuz-Ram model gives good results using ANFO. Using higher density, lower relative weight strength (RWS) explosives is not as successful. How can \( k_{50} \)-equation be altered to take this energy yield problem into account?

Claude Cunningham:

The introduction of emulsion types of explosives has raised problems in terms of relating actual field performance to the Relative Weight Strength as calculated. I believe that in hard rock types these explosives can be assumed to have an effective RWS of 120% of the calculated strength. We do not yet have sufficient data to confirm this or suggest the trend in soft rocks, but the rating of modern explosives in different rock types and hole diameters is a major problem which must be addressed.

Questioner: Giovanni Rossi. Answerer: Mick Lownds

In the plots relating standard deviation of drilling to means of oversize or to R.R.S. exponent \( "n" \), the standard deviations are given in meters. Can the author specify the blastholes lengths?

Mick Lownds:

The modelling is 2-dimensional. In considering long, deviating blastholes, simulations would be at several planes across the holes and the overall fragmentation would be synthetized from the component slices.
Model studies both at Maryland and by Swift etc at Lawrence Livermore have revealed that there is a dramatic enhancement in fragmentation for delay times comparable with the transit time of the P-wave between the boreholes. Did you also observe this trend and if so, did you utilize this result in your computer code?

Mick Lownds:

The picture in the paper of three holes fired one at a time shows crack extension of the first crack pattern due to the stress wave from the second hole. This effect, however, is smaller than the destructive interference upon simultaneous firing; it was not included in the code.

Questioner: Zeev Jaeger. Answerer: Mick Lownds

Question 1) Can you explain the criterion you used in your model to decide how many radial cracks originate from each borehole? Is this number related to the pressure pulse or explosive parameters?

Question 2) Is spall taken into account in your model?

Mick Lownds:

Answer 1) There are several methods for calculating the crack number as a length function. In the "parent" of this model, Harries does calculate this crack intensity. I have found none of their approximate methods more convincing than the others, nor do they fit the observed crack patterns well. The function used is guessed to make the pattern represent qualitatively the sort of crack pattern I was seeing in the models; both marble, shale and plexiglass.

Answer 2) Spall is recognized as a mechanism, but the intention here was to model only the dominant mechanism in fairly competent material. Spall was not included.

Questioner: Giovanni Rossi. Answerer: Per-Anders Persson
(concerning Hans Hjelmbergs paper.)

Grinding and blasting are physically different processes, the former being characterised by an abrasion grinding component which is negligible in blasting. Can the author explain on which grounds he was able to establish a mathematically workable relationship between grinding and blasting?

Per-Anders Persson:

Maybe the shear deformation and shear crack propagation mode in blasting, as pointed out by Dr Margolin, introduces into the process of fragmentation by blasting a mechanism similar that active in grinding. In any case, the proof of the pudding is in the eating, as pointed out by the late professor W. Weibull, and the two processes seem to be describable by the same type of expression.
Discussion 9


Question 1) You presented data on fragmentation versus intershot delay which were not in the preprints. When is the minimum in this curve in terms of ms/m of spacing?

Answer 1) The minimum fragment size occurred at about 360 μs delay for a 5 inch burden. If scaled directly this would mean 0.864 ms/ft or 2.84 ms/m of burden. This is a bit smaller than those results obtained in larger rock testing by Bergmann, Winzer and Chiapetta. Please bear in mind however that direct scaling from the plastic tests to large rock blasting is incorrect. Scaling is at present not possible and we only use the plastic tests to suggest mechanism of fragmentation.

Answer 2) We feel that the absence of a tensile tail in the stress pulses due to inadequate bottling up the gases? Have enough experiments been done with realistically confined gas in the hole to form an opinion about the relative importance of radial versus non-radial fracture in fragmentation.

William Fourney:

The data on the delayed layered model testing can be found in Rock Fracture Mechanics edited by H P Rossmanith CISM Lecture Series No 275 Springer Verlag, Vienna - New York (1983).

Question 2) Is the absence of the tensile tail in the observed stress pulses due to inadequate bottling up the gases? Have enough experiments been done with realistically confined gas in the hole to form an opinion about the relative importance of radial versus non-radial fracture in fragmentation.

Answer 2) We feel that the absence of a tensile tail in the stress is due to radial crack initiation at the borehole. That is, the tensile stresses are consumed as cracks are initiated. In all of our testing we feel we bottle up the gases for at least as long as it takes for the stress waves to initiate fractures. The fragmentation produced by our models is an error, therefore, on the conservative side. That is, the full gas effect can only increase the fragmentation achieved. In full scale tests, an examination of the muck pile after the shot reveals very little pie shape segments. We therefore conclude circumferential or non radial cracking is of extreme importance. We have however not to date prepared model experiments in which we contained the gases long enough to prove this hypothesis.

Questioner: Thomas Adams. Answerer: William Fourney

In blasting in the jointed model, you pointed out that joint induced fractures do not occur in the layer containing the borehole. Does the presence of the joints actually protect this layer or does ordinary fracturing occur as the "S-wave" passes by?

William Fourney:

The presence of the joints does not protect the borehole layer. One still gets radial fractures and in fact the layer tends to act as a wave guide as the radial cracks change direction to near parallel to the board lines. Joint induced fracture does not occur in the borehole layer as the stress waves pass out from the borehole. When the waves reach the boundary and reflect as tensile wave the borehole layer is fragmented very nicely.
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Questioner: Calvin Konya. Answerer: William Fourney

In a real life situation such as in blasting deep trenches, we often find that a horizontal mud seam located in the loaded portion of the borehole will cause problems, (almost no breakage will occur). Please explain why this occurs and how joint initiated fracture are changed due to the mud seam.

William Fourney:

Our experience would indicate that the stress waves would not be transmitted across the mud joint and hence breakage below the joint would be minimal. On the other hand the stress waves would be transmitted back into the rock above the mud seam and breakage there should be enhanced.

Calvin Konya:

If there is an open joint, will these joint initiated cracks not occur at other joints?

William Fourney:

They will occur at other joints on the side of the borehole, in other words towards the borehole but nothing cross the open joints. My own opinion without having done tests with that nature is that a mud filled joint will tend to concentrate more of the stress wave energy within that isolated layer. I would expect to see not only more joint initiated fracturing, but also more fracturing caused from the reflected stress waves coming back quicker before it had time to attenuate. We found for example when we went away two layers from the borehole that we have had enough attenuation that the additional fracturing at the second joint was not as great as at the first and by the time we got three joints it was not existent. I must point out that we did a very limited amount of studies. We did not vary the layer thicknesses etc.

Questioner: Paul Worsey. Answerer: William Fourney

Have you progressed to simulating any multiple hole blasting such as pre-splitting intersected with parallel discontinuities at varying angels to the hole line? If so could you elaborate?

William Fourney:

We have not investigated multiple hole fracture control blasting in the presence of large discontinuities with photo elastic models. We have investigated with photoelastic 2-dimensional models the ability of a propagating fracture to cross large discontinuities with different interface properties under a study funded by the US Department of Energy (DOE). These models used bonded interfaces and interfaces pressed together with varying compressive stresses. Straight as well as curved interface were studied, but angle of crack approach was not a parameter investigated. Our preliminary findings, published as quarterly progress report to DOE, pointed out that the most important requirement for successful crack penetration was the ability to transmit frictional forces or displacement across the joint.
Questioner: Giovanni Rossi. Answerer: Dinis Da Gama

Question 1) Has the author had the opportunity of determining the work indexes of the jointed basalt of table 5 of his paper with Bond’s test?

Question 2) Has the author had the possibility of estimating some quality index of the basalt in situ, like for instance Deere’s RQD for the areas where the seven blasts were carried out?

Dinis Da Gama:

Answer 1) The determination of work indexes for jointed basalt subjected to blasting was possible by means of calculations after the blasts happened. Use of the Bond’s test could be done only for intact basalt blocks, therefore not including joints and so those tests have not been conducted during the research.

Answer 2) Yes, there was intensive geotechnical survey of the basalt, because the work described was done in basalt quarries for various dams under construction in Brazil. Classification of basalt types depended on the fracture density and certainly this density is proportional to the work.

Questioner: Gert Bjarnholt. Answerer: Yang Zu Guang

In table 1 the P-wave velocity of magnetite is given as 2402 m/s. Is this a typical value and how was it measured?

Yang Zu Guang:

P-wave velocity has been measured both in blocks 300 x 300 x 100 mm and 42 mm drillcores with a Pundit Ultrasonic tester. Mean value for the readings in the block was 2402 m/s. Five drillcores were examined with two readings in each and the mean value for each core was calculated. The scatter for P-wave velocities in the cores were from 2201 to 2605 m/s. A much higher P-wave velocity was expected about 4500 m/s. We do not know why the velocity was so low.

Göran Lande:

Additional comment to my paper "Influence of Structural Geology on Controlled Blasting in Sedimentary Rocks–Case history".

In connection with my paper dealing with blasting in difficult geological formations a new idea is presented for explosives customising and dosing in the borehole with respect to rock physical properties. The objective of the proposed idea is to optimize explosives and its performance. It means that it is necessary to adapt the dynamic input from explosives in such way that it will match rock properties and give designed results at the same time.

The goal can be achieved by modification of presently used explosives, but in order to do so, we must know in advance the parameters which govern present performance. Reasonable way of getting required parameter definition is to investigate explosives function by means of source spectrum analysis and its signature characteristics. Methods for rock properties investigation by means of logging are today well developed and computerized borehole logging is a step toward computerized charging with respect to transfer function of geological formation.
Questioner: Roger Favreau. Answerer: Cameron Mc Kenzie
(concerning Williamson's paper)

I fully agree with you strong message to the effect that movement and swell
can be just as important as actual fragmentation. My questions are;

Question 1) Would you feel that swell measured from surveying of the muck pile
is a fair evaluation of the quality of movement?

Question 2) Have you got data to compare your shovel efficiency measurements
with swell measured by surveying of the muck pile?

Cameron Mc Kenzie:

Answer 1) Shovel monitoring is a means of assessing the combined effects of muck
pile packing, size distribution and shovel/muck pile geometry. Although surveying is considered as an adequate indication of muck
pile movement, it conveys no information concerning the effects of
size distribution.

Answer 2) The possibility of using profile surveying alone as an indicator of
blasting effectiveness is currently being investigated for a dragline
operation in Central Queensland. In this instance, where bucket size
is approximately 100 m³, the effect of size distribution is expected
to be minimal.

Questioner: Jeff Bailey. Answerer: Cameron Mc Kenzie
(concerning Williamson's paper)

Could you provide a comment on the shovel monitoring technique as far as the
influence of shovel operator attentiveness might effect its usefulness as
referred to by Kai Nielsen in his presentation?

Cameron Mc Kenzie:

One of the major objectness in defining an index of digability was to obtain
a parameter which could be readily defined and evaluated and which would be
insensitive to operator characteristics and attentiveness. The index can there-
fore be used to investigate more closely the observed productivity variations.
The index is an attempt to isolate muck pile characteristics from operator and
machine characteristics. Several test were made to investigate the degree of
influence by operators and the index indicated only slight variations for very
large variations in production rate produced by different operators (with
deliberate variation in operating procedure) on the same shovel in the same muck
pile.
Questioner: Roger Favreau. Answerer: Kai Nielsen

Are the results of your Figure 3 representative of most of your studies; in particular, is the "loading" curve usually the one that shows the most variation?

Kai Nielsen:

I did not show you the formula I have used. One of the key factors are the exponent in this formula and that would be different for different operations.

Tim Hagan: A comment on the very important subject raised in the paper by Angelo Medda.

Drilling of course, is chapter one in blasting and it is very obvious that if we get drilling deviation and the toe burden becomes excessive, we are going to get poor fragmentation. If it slopes the other way and the burden gets to small we will get to fine fragmentation. There is a relatively small volume of rock broken out by that charge and lot of that energy wastefully dissipated in form of noise and airblast and vibrations in general, a lot of kinetic energy is put into rock movement, which perhaps is not desirable at that stage. But perhaps less of this is the fact that when we get cumulative deviation, a front row blastholes bending backwards and the holes directly behind in the second row moving forwards, we could get that earlier firing charge damaging the second charge. Perhaps it could undergo sympatetic detonation. This would of course bypass the delays in the blast and that would be bad as we know. Perhaps we could get a dynamic desensitization that is one charge lateraly compressing by one means or another, the charge waiting to shoot on a later delay and as we move out of the area of nitroglycerine based explosives more and more forwards ANFO, watergel blasting agents and emulsion blasting agents, this dynamic pressure desensitization becomes an increasingly potential problem. Perhaps these two charges getting much closer together than they should be, the earlier fired charge could get a radial crack going through into that charge which is waiting to shoot. The gas pressure will jet through that crack and actually jet the charge from the blasthole. This perhaps could explain some of the lack of detonation observed in recent times, particular in under ground mines where blastholes are long and deviation is the greatest potential problem.

Giovanni Rossi:

Tim Hagan makes things more interesting just in that way he tells them. The problems you have mentioned has been observed in our rounds. The deviation of the boreholes have been examined by a photo inclinometer. Sometimes the deviation was very small at the hole bottom but large in between and that is the reason for big boulders.
Laboratory and field tests have been undertaken to study how an increase of specific charge in blasting can increase the efficiency and lower the cost of mineral dressing of different ores.

The investigation showed the possibility of changing the mechanical characteristic of broken ore by means of blasting effect and the possibility of better mineral grains opening and exposing at the expense of microcracking at grain boundaries, assisting in grade of ore extraction by dressing.

In certain cases it is better for some mines to increase explosive use for breaking in order to decrease further ore crushing and dressing costs.

The main aims which must be attained during ore crushing to minerals opening may be formulated as follows:

First - change of the mechanical properties of ore to decrease its strength for lowering power consumption of the following breakage;

Second - loss of strength must be at the grain boundaries, to provide the necessary selectivity of minerals opening;

Third - not only a surface layer must be exposed to the process of loss of strength, but also the whole piece of ore.

Currently, the development of processes of ore crushing is aimed at improving crushing and grinding. In some cases crushing cost makes up to 60% of production cost and a share of capital cost for construction and equipment of crushing and grinding plants makes up to 50% of total capital costs of mining processes.
Under these conditions the problem of decreasing mechanical crushing costs at the expense of raising the part of blasting during the processes of ore crushing is quite solved.

To show the effect of blasting on mineral opening the problem of static load of medium, containing grains, with various elasticity characteristics, has been considered. It was shown that depending on the ratio of Young's modulus and Poisson's coefficients of containing medium and grains stresses may be concentrated on their boundaries, exceeding strength limit and hence cracks may be formed.

A similar problem was solved for dynamic loading. It was found out, that in some cases depending on the ratio of length of stress wave and size of grains, microcracks may be at the boundaries when grain strength is lower than the strength of containing rocks.

Thus, the possibility of breaking the containing rock and preserving the integrity of ore grains by producing a suitable stress field in a rock was shown. The stress field in the medium may be regulated by the following factors: Type of explosives, construction of charges, their geometry and their initiation in space and time.

To determine the change of rock strength under the influence of dynamic loading we have carried out investigations both in the laboratory (using ore samples exposed to blasting shot) and in the training ground by constructing charges of various size.

The change of mechanical characteristics of copper-molybdenum ore samples while throwing on them various weights from various heights was tested using the method of impact testing machine. The blasting shot was effected in such a way, that visible cracks did not appear. After this the crushing strength of samples was fixed. Quite simple relationships allowing to assess the strength change depending on the energy, transmitted to a sample by dynamic loading, were determined.

In the impact testing machine the change of rock strength was tested by firing the blast-hole charges. After firing a vertical blast-hole charge ore samples at various distances from the charge were selected and their strength was estimated by intensive crushing by impact testing machine (Prof Baron's method). Repeated measurements carried out showed the change of ore strength depending on the blast intensity.

Experiments of repeated blasting shot of ferruginous quartzite samples have been also carried out. Experiments have been carried out in such a way, that in all cases the total energy transferred to the sample by explosion was the same, however in one case it was a result of a single explosion, while in the other case it was a result of three explosions. It was found out, that a share of fine fractions in granulometric composition of broken ore increased by the repeated action. For example, in the test with ferruginous quartzite (diameter - 40 mm, height - 40 mm), a share of fractions with the diameter less than 5 mm increased twice by the same energy.

To investigate the possibility of mineral grains opening direct experiments have been carried out to determine the change of contact grain strength with containing medium and the direction of microcracks propagation. Contact strength of ferruginous quartzite in the zone of intergrowth of grains in quartz were investigated by a special technique. It was determined that contact strength of ore minerals was 11 times that of the magnetite-quartz pair and 1.3 time that of the hematite quartz pair. It follows, that by using blasts contact strength of intergrowth zones of ore minerals with quartz, is the first to decrease.
Then investigations of microcracks propagation in samples have been carried out. For this purpose polished samples previously subjected or not subjected to blasting were photographed with great magnification and the cracks were fixed. Let us take as an example an experiment on magnetite grain opening when the crack passed the distance of 1796 μm. Out of 58 opening magnetite grains 38 (65.6%) were detached from quartz by the crack on the contact boundary. In five pairs of magnetite grains (17%) the crack passed between the grains and only in ten grains (17%) the crack propagated across them.

Thus, the above experiments showed the possibility to increase mineral opening for some types of ore at the expense of increasing the intensity of blasting.

For the purpose of studying this process in industrial conditions experiments have been carried out in one of the mines of our country, where ferruginous quartzite is mined. 60,000 t of ore with increased explosive consumption of 1.74 kg/m³ instead of the earlier 0.93 kg/m³ were mined. Samples of 50 t were crushed to less than 300 mm in the crusher on the pit edge. Then the energy used to crush the ore further to less then 25 mm was estimated using jaw and cone crushers of industrial size.

Tests of dry magnetic separation have been carried out with the crushed material. The greater the use of the explosive, the more intensive is the tailing in dry separation. The results of wet magnetic separation have shown that the greater is the use of the explosive, the higher is the fraction of iron extracted.

Thus, with increased explosive use by 30 to 40% the diameter of an average piece of broken ore is decreased by 30% and the class over 750 mm is decreased by 5%. Further ore crushing requires a lower amount of energy. The results of magnetic separation showed the increased grinding capacity (6-7%). It is important to note, that the increase in the use of the explosive does not result in the decrease of coercive force. This resulted in improving magnetic parameters of separation.

Similar experiments have been carried out with apatite-nephelite ore. The experimental technique provided the selection of ore samples from 80 to 100 kg broken with various explosive use and their further processing at a dressing mill. Properties of ore were controlled at all stages of processing. Crushing and grinding power use was measured.

Investigations have shown, that there exists a linear relationship between the increased use of the explosive and decreased energy necessary for further grinding. This law is valid in range from 0.5 to 4 Mcal/m³. It was also determined, that the greatest grade of ore was observed in the range of fraction 0.063 - 2.25 mm; it makes possible to separate broken fines and to dress them without grinding.

The whole complex of carried out investigations showed the possibility of changing mechanical characteristic of broken ore by means of blasting effects and the possibility of better mineral grains opening and exposing at the expense of microcracking at grains boundaries, assisting in grade of ore extraction by dressing.

In certain cases it is better for some mines to increase explosive use for breaking in order to decrease further ore crushing and dressing costs.
Questioner: Roger Favreau. Answerer: Giovanni Rossi

Your analysis deals with the final volume or surface specific energies. Such analysis is called for; yet the breakage process in blasting is dynamic, so that the ultimate explanation must address the mechanisms whereby dynamic stresses and their associated stress energies break rock. Have you tried to link your work to different fracture criteria, such as peak stress, stress impulse, Griffith's theory etc?

Giovanni Rossi:

The available data did not allow such an analysis, since they only concerned the overall energy balance of the rock breaking operations reported. Furthermore, Griffith's theory for instance (with the modifications brought to this theory by latter authors) seems, in our mind, more relevant for rock cutting processes than for blasting.

Questioner: Claude Cunningham. Answerer: Per-Erik Lindvall

Your paper illustrates very clearly the problems caused in fan drilling by the long holes and the variation in angle and depth. Even at 16 m sub-level interval the hole positions can overlap. Now you are planning to increase the interval to 20 m, which must accentuate the problem. Would it not be better to consider converting to blasthole stoping with long parallel holes?

Per-Erik Lindvall:

In my paper I am presenting some of the steps that we have taken to prevent large hole deviation in our new sublevel caving layout. For example:

- a new type of drill rig
- ITH (In The Hole) machines

To convert a mine from one mining method to another, takes at least in our case, many years. We are of course always trying to look ahead if there is any possibility to mine out the ore in any cheaper way. So, blasthole stoping with long parallel holes has among other mining methods of course been considered, but at the present time we have no intention to introduce it in Malmberget. At our LKAB mine in Kiruna, the stoping with long parallel holes, is going to be tested in a near future.

Questioner: Giovanni Rossi. Answerer: Per-Erik Lindvall

Question 1) Can the author illustrate the criteria on the grounds of which he considers as normal Gaussian theoretical distribution of hole deviation?

Question 2) Did the author detect higher deviations for the less sloping holes of the fans?

Question 3) What instrument was used to measure the deviation?
Per-Erik Lindvall:

Answer 1) I think that there is no question about that the variation in burden and hole inclination errors are true "normal Gaussian distributions".

I am not sure which distribution curve that fits best for the hole curvature. The one that describes it in the best way, is probably not a true normal Gaussian curve. In our case, where we drilled from one level up through the floor on the level above, there were nothing that indicated that we as a first assumption could not look at the hole curvature as a normal Gaussian distributed deviation. So, for us and for the purpose that we are using these results, it is an assumption that describes and also explain some of the reasons why we have such a low chargeability in the fan closest to the caving front.

Answer 2) No!

Answer 3) The instruments that has been used is for

a) Variation in burden : Ordinary mine surveying equipment

b) Inclination of holes: We put a laser about 2 m up in the hole and measured the laser spot on the floor and the collar of the hole with a theodolite.

c) Hole curvature: For the 105 mm holes we have used a gyro—compass and for the 57 mm holes we drilled the holes up through the floor above to be able to measure the hole curvature of the holes.
First International Symposium on ROCK FRAGMENTATION BY BLASTING
Luleå, Sweden, August 1983
PANEL DISCUSSION

PANEL:

WILLIAM HUSTRULID, Professor at Colorado School of Mines, USA
MICK LOWNDS, Blasting physicist at AECI, South Africa
INGEMAR MARKLUND, Min Research Leader at LKAB, Sweden
LARS MONTAN, Msc Mining, Atlas Copco MCT, Sweden
LEIGH NEINDORF, Drilling and Blasting Superintendent, Mount Isa Mines Ltd, Australia

MODERATORS:

Roger Holmberg
Agne Rustan

OPENING OF THE PANEL DISCUSSION

Roger Holmberg

The panel discussion will consist of two parts namely

Part 1. The organization of international blasting research and standardization.

Part 2. New developments in technique to control fragmentation and their impact on modern mining and construction engineering.

Before we start the discussions I would like to say some words about the First International Symposium on Rock Fragmentation by Blasting.

The principal objective with this symposium was to create a forum for discussions on various topics on fragmentation.

We have during four interesting days been listening to a large amount of good quality papers. The papers have covered a range from basic and applied research to final results achieved in the practical mining operations.

Researchers presenting their papers have shown that their theoretical studies and modelling have agreed most precisely with the conducted experiments. Presented semi-empirical fragmentation models based on field investigations have been found to serve as reliable tools for predicting the rock fragmentation when any important parameter is varied - if they are used with intelligent judging.

Each of the speakers has through their work added additional pieces to the puzzle of understanding the fragmentation process. No paper presented gives the ultimate solution of how to optimize the fragmentation with regard to the total excavation operation for a general site.

Here skilled people like you have to pick up the piece that will fit into each specific operation in order to improve the results. We need people that understand and can transfer the achieved information to practical blasting situations. That is why I believe a type of symposium like this one is fruitful for future research and practical applications.
Those of you who visited the tours in the beginning of this week had the opportunity to see the mining operations in Luossavaara, Kiruna, Malmberget and Aitik. I believe you then saw that there is a significant gap between the precision of tools the mines can use and the precision of tools the scientist use for controlling, measuring and predicting the fragmentation.

In the field a number of uncontrollable parameters influences the result occurred. Human mistakes, lack of proper initiation systems for massblasts, lack of diagnostic field tools, dilution problems due to caving, unexpected misfires etc.

You all know that very good improvement can be achieved if the precision can be higher. Those of you who visited the Research Mine and saw the developed high precision drilling equipment easily understands what this means in higher productivity and less development costs.

In the following panel discussion we are first going to discuss the organization of international blasting research and standardization. After that new techniques to control fragmentation and their impact on modern mining and construction engineering is going to be discussed.
The Organisation of International Blasting Research and Standardization

Moderator: Agne Rustan

On the first question, if, when and where a Second International Symposium on Fragmentation by Blasting is going to be held the discussion can be summarized as follows.

The interest for a Second Symposium is large and the opinion was that it should be held within three years. Both USA and Australia are interested in organizing the Second Symposium. In USA two potential locations were mentioned, at the University of Maryland and at the Colorado School of Mines.

The second question dealt about the need for an International Society of Rock Fragmentation by Blasting. The tasks for this organization could be to organize:

1. Publication of a scientific journal in blasting (without advertisements)
2. Discussion of technical problems in working groups.
3. Nomenclature. (Add hoc group)
4. Standardization

The discussion about an international organization concentrated on the problem to find the economical resources to establish such an organization. This problem has to be solved first.

The panel suggested that a steering committee should be established to work on both question. Agne Rustan was proposed as chairman and calling together.

The audience was invited to propose ideas on the two questions within six month.
Moderator: Roger Holmberg

Before the discussion started Peter Reich and Jack Colle made a short presentation of the use of water jet to slot blast holes both axial and radial. This technique could perhaps in the future be combined with ordinary blasting to achieve a more controlled fragmentation in certain situations. The documentation for this presentation is given at the end of the panel discussion.

Roger Holmberg had prepared three questions for the panel and the audience. The first question was:

1) What results from the given papers at the symposium are to the point today to be transferred to practical applications?

Comments were given about better possibilities to control blasting totally, especially fragmentation, but also blasting damage. Much experience and knowledge can thus be transformed to blasting operating personnel but also to manufacturers in order to design more suitable equipment. Better awareness of varying conditions has been underlined but also the value of exact intervals and timing in blasting and hole accuracy.

Equipments to be used more in practical applications were especially commented to be high-speed photography, vibration recorders and the marking out of hole patterns: Electronic detonators were considered to be close to practical application.

More research is however needed concerning the large hole blasting process, relationships burden/spacing/timing and how to get better fragmentation. The tensile stresses should be used more in the blasting process instead of compressive stresses which are used today. Tensile stresses are used to break the rock in high pressure water jet cutting as presented by Flowex.

The second question was:

2) What is the best way to transfer presented developments to practical applications?

Comments were given about the big value of seminar participants to make summaries of the seminar to be presented in mining journals, congresses but also in local meetings and company activities. Oral presentations and courses are better than sending out thick scientific documents, which could be hard for practical miners to read. Descriptive models could also be of value to present theories for operating staff.
The third question was

3) Could the theorists gain information for future research by meeting practical people in this type of symposium?

Comments were given of better awareness concerning the practical application of theories, especially that practical people emphasizes more ways to solve their problem of today rather than theories. Detailed comments were also made concerning the need of more theory behind oil practices including co-operation with metal mining. Of value could also be to study the effect of charges in different geometries with the computer models.

The symposium was closed by the chairman of the Organizing Committee Dr Per-Anders Persson.
1. Introduction

Small scale hydraulic fracturing is a promising new method for relatively safe, efficient and vibration free hard rock excavation. Fractures are easily induced in rock using only moderate pressures. Fractures alone, however, are fairly useless unless they can be made to link up so as to separate rock fragments for excavation or provide some other useful function such as providing a well defined tunnel perimeter for the control of overbreakage. The control of small scale hydraulic fracture orientation has been made possible by the development of an ultra-high pressure water jet capable of cutting deep slots in hard rock. When a deep slot is introduced into the wall of a borehole which is then pressurized, a hydraulic fracture initiates at the slot tip and propagates parallel to the slot for some distance. This observation forms the basis for the development of a small scale hydraulic fracturing technique with useful applications in mining, tunneling and excavation. Initial tests with this technique have been successful in producing controlled small-scale hydraulic fractures.

2. Hydraulic Fracture Initiation

Hydraulic fracture initiation occurs from a borehole when hoop stresses around the hole become greater than the rock tensile strength plus any lithologic stresses due to overburden or tectonics. The magnitude of the hoop stress in a pressurized borehole is equal to the applied pressure. The tensile strength of rock varies considerably and depends somewhat on the technique used for measurement. Typical useful values range from 3.5 MPa for a relatively weak sandstone to 35 MPa for some basaltic rocks (Jaeger and Cook, 1976).

The magnitude of overburden stresses depends on depth and the rock density. A reasonable rock density for sedimentary overburden is 2,700 kg/m³ which gives a rate of increase of lithostatic stress with depth of 26 MPa/km. Overburden stresses thus require a significant increase in hydraulic
fracturing pressure at depths of a few hundred meters and eventually come to dominate hydraulic fracture initiation. In general, the state of stress underground is triaxial and may include a large non-lithostatic component. A summary of measurements of the state of stress in the crust of the earth is given by Gay (1980).

The magnitude of the difference between vertical and horizontal stress components can be as high as the calculated lithostatic stress. In an underground excavation a differential stress arises because of the presence of a free surface. Under conditions of triaxial loading a hydraulic fracture will propagate perpendicular to the direction of least principle stress. In the case of an oil or gas well one of the principle stress directions is usually vertical. Haimson and Fairhurst (1970) describe a technique for inferring the magnitudes and orientations of the principle stresses in wells from observations of hydraulic fracture initiation, propagation and flow-back pressures and the orientation of the initial fracture. In laboratory tests of hydraulic fracturing under conditions of triaxial stress they note that fracture always occurs parallel to the borehole unless a stress concentration is introduced in the wall such as a sharp corner at the bottom of the hole. These results show that it is possible to initiate a hydraulic fracture in a plane other than the one perpendicular to the least principle stress direction under conditions of high differential stress.

A stress riser may be introduced into the wall of a borehole by cutting a slot with a high pressure water jet. Fine water jets driven at pressures of up to 400 MPa have proven to be an effective means of producing narrow slots in hard rock. An arbitrary slot geometry can be cut in a borehole. Both lateral and disked slot geometries, Figure 1, will be considered here.

The stresses required to initiate fracture from a slotted borehole may be estimated from linear fracture mechanics theory and the fracture toughness of the rock. The stress field near the tip of a pressurized slot, Figure 2, is described by the opening mode stress intensity factor (mode I SIF) defined by

\[ K_I = \lim_{r \to 0} \sigma_{yy} \cdot \sqrt{2\pi r} \]

where \( \sigma_{yy} \) is the applied stress in the y direction. When the SIF exceeds a critical value, known as fracture toughness, a fracture will initiate. The SIF...
for a penny shaped fracture with a radially varying pressure distribution,
pressure is given by

\[ K_I = 2\sqrt{\pi} \int_0^a \frac{r P(r)}{\sqrt{a^2 - r^2}} dr \]  

(2)

where a is the slot radius. This equation may be integrated for a constant
pressure from R to a to give the SIF for a disked slot.

\[ K_I = 2p\sqrt{\alpha(a^2 - R^2)} \]

(3)

Numerically calculated values for the SIF of a lateral slot emerging from a
circular hole are given in Sih (1973). Normalized values of the stress
intensity factor for a disked slot and for a pair of opposed lateral slots
emanating from a pressurized borehole are shown in Figure 3. When no slot is
present there is no stress magnification and the stress intensity factor is
zero. The value of the SIF grows rapidly with slot depth so only a small slot
is sufficient to give rise to a significant stress magnification. As the slot
depth becomes greater than the borehole radius the borehole no longer affects
the SIF as much. The asymptotic values represent SIFs for a penny shaped slot
and a long flat slot. It is interesting to note that the SIF introduced by a
disked slot is about twice that of a lateral slot of the same depth.

The tip of a slot cut with a water jet is not perfectly sharp as assumed
in the determination of SIF. The state of stress very near the tip of the
slot will not be singular in any case since deformation will occur to
accommodate the high stresses. The size of the deformation zone in rock has
been estimated by Schmidt (1982). He considers that inelastic deformation
will occur when the maximum normal stress, calculated from the linear elastic
theory of fracture, exceeds the ultimate tensile strength of the rock. Within
this region the elastic theory breaks down as micro-cracking occurs. The size
of the deformation zone is given approximately by

\[ r_0 = 0.2 \left( \frac{K_{IC}}{\sigma_u} \right)^2 \]

(4)

where \( \sigma_u \) is the ultimate tensile strength. As long as the slot tip radius lies
within this radius fracture mechanics theory will provide a reasonable
estimate of stress intensity and fracture initiation may be predicted from
measurements of fracture toughness. The appropriate value to use for ultimate tensile strength is not clear. A useful lower limit is the ultimate tensile strength measured on a bench scale sample. For example; the tensile strength of St. Cloud granite is about 20 MPa while its fracture toughness has been measured to be 1.5 MPa\(\sqrt{m}\). This gives a deformation zone size of about 2 mm. This value is expected to vary somewhat for different rocks. If the slot tip radius is greater than a millimeter or so the pressures required to induce fracturing may be greater than predicted by the fracture toughness since the actual stress intensity will not be as high. High pressure water jets are capable of cutting slots with a radius of 0.1 mm.

Figure 4 shows the variation of fracture initiation pressure with slot depth for both slot geometries for St. Cloud granite. A slot of only a few millimeters depth has a large effect on the fracture initiation pressure. Beyond about 10 mm the effect becomes small for practical slotting capabilities. Ingraffea and Schmidt (1978) have shown an effect of crack length on fracture toughness measurements. The observed fracture toughness of Indiana Limestone decreases with crack length below about 30 mm. The fracture initiation pressure might thus be somewhat lower than indicated in Figure 4 were it not for the effect of slot tip bluntness which has the opposite effect.

All of these calculations have been made with the assumption of zero overburden stress. If a uniform, lithostatic, overburden stress is present the fracture initiation pressure will be increased by the magnitude of the stress. If a deviatoric stress is present the component normal to the slot should be used to obtain the additional pressure required for fracture initiation. The effective fracturing pressure will be the difference between the applied pressure and the non-deviatoric stress component.

3. Fracture Propagation and Arrest

The SIF for a uniformly pressurized, penny shaped fracture is given by equation 3. The stress intensity increases as the root of fracture radius so fracture extension will continue and accelerate as long as the pressure is maintained. Alternatively the pressure required to maintain fracture propagation may be reduced. If an overburden is present the minimum pressure becomes dominated by the lithologic stress.
The volume of fluid required to fill a penny shaped fracture is obtained from the expression for crack opening displacement given by Sneddon (1946),

\[
\text{COD} = \frac{4(1-\nu^2)P_{\text{eff}}}{E}(a^2-\nu^2)^{\frac{1}{2}}
\]

where \(P_{\text{eff}}\) is the effective pressure, \(E\) is Young's modulus and \(\nu\) is Poisson's ratio. The penny shaped crack takes the shape of a flattened ellipsoid with volume

\[
\text{COV} = \frac{16(1-\nu^2)P_{\text{eff}}}{3E}a^3
\]

A reasonable value for the Young's modulus of granite is about 20 GPa while Poisson's ratio is generally less than 0.20. The effective pressure required to extend a fracture with a radius of 1 meter is only 0.4 MPa which gives a total volume of 0.1 liter or less. If the fracture is over-pressurized to 20 MPa the fracture volume is 5 liters. Small scale fracturing thus requires only a limited volume of hydraulic fluid as long as none of it is lost.

Fluid losses due to rock and fracture permeability can cause large pressure drops during hydraulic fracturing. Rock masses are permeable to most fluids to some degree. Hydraulic fluid loss can occur both due to the intrinsic permeability of the rock or into larger scale flaws. Rock permeability results in a decrease in the effective stresses at the slot tip. If hydraulic fracturing is used for excavation the fracture will eventually reach a free surface resulting in a pressure drop due to the high permeability of the fracture itself. The effects of permeability may be overcome by dynamic fracture pressurization.

If the rock is highly permeable, such as in a porous sandstone, the fracturing fluid will leak off into the rock pore space. When this occurs the pore pressure surrounding the crack will equalize the fracture pressure and no net stress will be imposed upon the rock matrix. The rate at which the pore pressure, \(p\), increases will be a function of the pressure gradient and time. The one dimensional flow is given by Darcy's law

\[
Q = K \left( \frac{dp}{dx} \right)
\]
where $Q$ is the volume flow rate, $dP/dx$ is the pressure gradient and $k$ is the permeability. For continuity,

$$\frac{dQ}{dx} = 0 \quad (8)$$

and

$$k \frac{d^2P}{dx^2} = 0 \quad (9)$$

which can be solved to give

$$P = \left( \frac{Q}{k} \right) x + C \quad (10)$$

where $C$ is a constant. Permeability effects become important if the pore pressure becomes comparable to the fracture pressure over a length scale given by the fracture size. If the porosity is $\mu$, the fluid will flow to a distance $X$ given by

$$X = \frac{Qt}{\mu} \quad (11)$$

These equations can be solved to give the penetration of a fracturing fluid into the surface of a permeable material as a function of time

$$X = \left( \frac{P_0 \kappa t}{\mu} \right)^{\frac{1}{2}} \quad (12)$$

where $P_0$ is the surface pressure. A plot of the penetration of water into Berea sandstone as a function of time is shown in Figure 5. The permeability is assumed to be 0.217 Darcy and porosity is 0.182. Both values are very high for rock and thus provide a good estimate of the limits of effective pressure in permeable rocks. Fracturing pressures for this material should occur at pressures between 1 and 10 MPa. The water penetration after 10 milliseconds will be about 1 centimeter. If the pressure rise time for water entering a slot with a radius of a few centimeters is much greater than 10 milliseconds the pore pressure near the slot will increase significantly and fracturing may not occur. A means of rapid pressurization of slots is thus important for hydraulic fracturing on a small scale in permeable rocks.
Although most rocks are much less permeable than Berea sandstone, joint systems and other naturally occurring flaws will cause a high bulk permeability which will lead to a loss of pressure in the fracture. The induced hydraulic fracture provides the escape route for the fracturing fluid when it intersects a free surface or joint. The pressure gradient from the fluid inlet to the free surface will be a function of flow rate and the crack opening displacement. A significant pressure gradient must be maintained if a fracture which has extended to a free surface is to be completed so as to remove a rock fragment for excavation.

Fluid flow between two flat plates depends upon the cube of the plate separation. Tsang and Witherspoon have derived the expression for radial flow from an inlet of radius, \( r_e \), through a fracture of radius \( r_w \).

\[
Q = \frac{2\pi}{1n\left(\frac{r_e}{r_w}\right)} \frac{6^3}{12n} P
\]

(13)

where the inlet pressure is \( P \) and the fluid viscosity is \( \eta \). The plate separation may be estimated from our observations of residual fracture opening after a hydraulic fracture test of a large block of Sunset Pink granite. A residual fracture opening of about 2\,\text{mm} was observed. Since a pressure gradient exists on the opened fracture the SIF at the remaining rock ligament will be lower than if the pressure were uniform. Equation 3 may be integrated to give the SIF for a penny shaped fracture with a radially decaying pressure distribution

\[
K = \pi \sqrt{\pi a} \left(1 - \frac{\pi}{4}\right)
\]

(14)

The pressure required to extend the fracture is 4 MPa. The required flow rate from a 2.5 cm diameter inlet hole to maintain this pressure is 4.5 m\(^3\)/s.

4. Effects of Deviatoric Stresses

Fracture curvature occurs in a non-uniform stress field in response to shear loading on the crack tip. Cotterel and Rice (1979) have shown that a fracture will attempt to propagate under the condition that the shearing mode SIF (mode II) is zero. The mode II SIF is defined in the same way as \( K_e \) by

\[
K_{II} = \lim_{r \to 0} (\tau_{xy}) \cdot \sqrt{2\pi r}
\]

\[
^* \text{SIF = Stress Intensity Factor } K_{II}
\]
When $K_\pi$ is non-zero the fracture will curve at an angle $\theta$ given by

$$\theta = \frac{2K_{II}}{K_{I}}$$

(16)

This equation is only accurate for curvature angles of less than 15 degrees. Fracture curvature occurs in the direction in which $K_\pi$ is decreasing. The mode II SIF due to a uniform shear stress on a penny shaped fracture has a maximum value of

$$\text{max } K_{II} = \frac{4\tau}{(1-\nu)\pi} \sqrt{a}$$

(17)

where $\tau$ is the applied shear stress. Suppose the fracture is oriented at an angle $\beta$ to a uniaxial compressive stress, $\sigma_1$. The fracture curvature angle will be given by

$$\theta = \frac{4\sigma_1 \sin \beta \cos \beta}{(1-\nu)\pi}$$

(18)

Curvature of the fracture may thus be controlled by orienting the original slot perpendicular to one of the principle stress directions or by overpressurizing the slot using dynamic pressurization.

5. Effect of a Free Surface

The presence of a free surface near a pressurized fracture will alter the state of stress at the crack tip by introducing a shear load. Pollard and Holzhausen have investigated the effect of a free surface on the mode I and mode II stress intensity factors at the tip of a hydraulic fracture parallel to a free surface. They show that $K_\pi$ is significant for a fracture whose size is more than one-third its distance from the surface. Figure 6 shows their calculations of $K_L$ and $K_\pi$ for a buried horizontal slit subject to uniform pressure and to a linearly decaying pressure gradient. The presence of the free surface leads to a shear stress on the fracture which decreases as the fracture inclines towards the surface. When the fracture half-length equals the distance to the free surface,

$$K_{II} = 0.13 K_I$$

(19)
which implies a fracture curvature angle of 15 degrees. When the fracture half-length is twice the distance from the free surface the curvature angle is 30 degrees. Recognizing that the accuracy of the curvature equation breaks down at higher angles it is still possible to make a qualitative estimate of the fracture path. Under these conditions a hydraulic fracture initiated from a disked slot will be expected to intersect the free surface at a radius of three or four times its initial depth. Figure also shows that pressurizing a fracture by a linear gradient, highest in the center, increases the ratio of \( K_f \) to \( K_r \) thereby increasing the rate of curvature and decreasing the radius at which the fracture will intersect the free surface. This might be expected in the case of high rates of pressurization where a significant pressure drop occurs along the fracture radius.

6. **Summary and Conclusions**

Several aspects of small scale hydraulic fracturing have been considered in this paper. The results indicate that a system for producing controlled hydraulic fractures on a small scale is feasible. The pressures required to induce hydraulic fracturing are well within the range of conventional hydraulic equipment. Permeability in porous rocks requires that a dynamic pressurization be applied to prevent pore pressure buildup in the rock which can prevent fracturing. High flow rates in the fracture also give rise to pressure gradients which are useful in completing fractures. Hydraulic fractures induced from a disked slot parallel to a free surface should curve towards and intersect the surface within a reasonable distance allowing excavation of the separated rock fragment to take place.

Under many conditions fracturing is necessary in rock masses subject to a substantial differential stress. Under these conditions fracture curvature is expected. Initial fracture orientations may be controlled by the slotting orientations provided the slots are deep enough. Proper slot geometry and pressurization may also prove helpful in producing large flat fractures.

The results of this work have led to the design and testing of a prototype small scale hydraulic fracturing tool. Tests with this tool indicate that under conditions of zero confining stress a disked or lateral slot will control the orientation of hydraulic fractures. In addition fractures initiated near the free surface will approach the free surface as predicted.
Successful fracturing tests have been carried out to date in Sunset Pink and St Cloud (charcoal) granites, in a highly permeable sandstone and in andesite.

7. Acknowledgements

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8. References

Cotterel, B. and J.R. Rice (1979), Slightly curved or kinked cracks, Int. J. Fracture, 16,155-169.


Figure 1. Slot Tip Geometry Showing Deformation Zone
Figure 3. Stress Intensity Factors for Lateral and Disked Slots in a Pressurized Borehole.
Figure 4. Fracturing Pressure for St Cloud Granite

\[ K_{IC} = 1.5 \text{ MPa}\sqrt{\text{m}} \]
Figure 5. Water Penetration into a Permeable Sandstone

Berea Sandstone

\( k = 0.217 \text{ Darcy} \)

\( \mu = 18.2\% \)
Figure 6. Mode I and Mode II SIF, for a Pressurized Surface Horizontal Slit. From Pollard and Holzhausen (1979)
ERRATA SHEET

- Text on page 14 should be moved to page 16 and vice versa.

- Formula (3) on page 809 should be changed to

\[
\frac{p_t}{p_i} = \left[ 1 + \frac{\rho_i c_{si}^2}{\rho_t c_{pt}^2} \right]^{-1}
\]
# First International Symposium on ROCK FRAGMENTATION BY BLASTING
Luleå, Sweden, August, 1983

## 1983-08-13

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Luleå, Sweden, August 23–26, 1983

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