Controlled Fragmentation and Contours in Rock Blasting
THEORETICAL AND TECHNICAL APPROACHES

by

AGNE RUSTAN
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AKADEMISK AVHANDLING
som, med vederbörligt tillstånd av Tekniska Fakultetsnämnden vid Tekniska Högskolan i Luleå för avläggande av teknologie doktorsexamen, kommer att offentligen försvaras vid Tekniska Högskolan i Luleå, rum F-341, torsdagen den 8 juni 1995 kl 15.00.

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Dr Ulf Langevors, Nora
Tekn lic Bo Hall, Solocell AB, Danderyd
This volume comprises the following papers:


Preface

The research work reported in this doctoral thesis was partly carried out during 1968-1970 at the Division of Mining at the Royal Institute of Technology in Stockholm and partly through research work at the Division of Mining Engineering at Luleå University of Technology during 1976-1992.

I would like to express my sincere thanks to my supervisor Professor Gunnar Almgren at the Division of Mining Engineering for his help in finding financial support for the research projects and his careful study and relevant suggestions for improvements of my papers and the final doctoral thesis before publication.

The preparation and the laboratory experiments were mainly carried out by Torbjörn Naarttijärvi, Shu Lin Nie, V.S. Vutukuri and Jan Öqvist from the Division of Mining Engineering, LuTH. Ove Alm from the Division of Rock Mechanics at LuTH carried out the testing of the physical and mechanical properties of rock. The field tests in tunnelling at LKAB in Malmberget were performed by Torbjörn Naarttijärvi and Bengt Ludvig. In the full-scale single hole bench blasting tests in the Storugns limestone quarry in Gotland, Shu Lin Nie and Jan Öqvist from LuTH contributed much, as well as Algot Persson and Ingvar Bergqvist from Nitro Nobel. Finally Hans Wirstam and Anders Månsson contributed performing the statistical computer analysis of the relation between burden, spacing and blasthole diameter. It has been a pleasure to have such motivated and skilled people in the research team, and I am very thankful for their contribution to the research.

I am also very thankful for the good co-operation and help from the Swedish industry: Atlas Copco AB, Luossavaara Kirunavaara AB (LKAB), Nitro Nobel AB, Nordkalk AB, and also the Research Mine at Luossavaara.

I would like to express my sincere thanks for the financial support from the Knutberg Foundation, the Swedish Work Environment Fund in Stockholm, the Research Mine in Luossavaara, the Swedish Board for Technical Development (STU), nowadays NUTEK in Stockholm, and Luleå University of Technology.

Further thanks go to the Beijing University of Science and Technology in China and the University of New South Wales, Broken Hill in Australia, which contributed the salaries of S.L Nie and V.S. Vutukuri.

Finally, I want to express my sincere gratitude to my wife Brita and my children Carolina, Sofia, Lovisa, and Mattias for their support throughout my study.

Agne Rustan

May 1995, Luleå, Sweden
Abstract

The Swedish mining and construction industries have a large interest in improving blasting technology, in mining concerning rock fragmentation by blasting (especially underground in large diameter hole blasting, > 100 mm) and in construction and mining concerning controlled and safe contours in blasted tunnels. This doctoral thesis deals with these matters and includes a licentiate thesis (published in 1970) and five papers A-E included in the doctoral thesis and published during 1983 to 1992.

The licentiate thesis (in Swedish) presents the development of a method to determine the ore content in an LHD-bucket after loading iron ore in the mine. The method is based on measuring the bulk density of the ore-waste mixture. In order to investigate the accuracy of the method a theoretical model was established, and verified by model-scale blasting tests and research concerning fragmentation and swelling of blasted ore under varying technical and environmental conditions. Paper A describes a part of the licentiate thesis, the development and choice of a suitable model material for the blasting tests.

Controlled Contour Blasting Technique (Paper B) especially in tunnelling, needs improvement in bad rock. Literature studies and model and full-scale tests were the base for the evaluation of the existing and proposed contour blasting methods.

Controlled Rock Fragmentation (Paper C, D, and E) is an important goal, especially in large hole diameter blasting. New design formulas were therefore developed for mean fragment size, rock fragment size distribution, etc. in connection with technical parameters such as burden or specific charge, spacing and rock impedance. The formulas are based on literature studies and model and full-scale tests. A new relation between the burden and blasthole diameter was derived from statistical analysis of data from one hundred surface and underground mines. The relation is different for surface and underground mines.
Contents

PREFACE

ABSTRACT

1. INTRODUCTION
   1.1 Main objectives of the doctoral thesis
   1.2 Summary and result of licentiate thesis and appended papers A-E
   1.3 Recommendations for future research
   1.4 Contributions from the author and others

Appended papers


Controlled Fragmentation and Contours in Rock Blasting
Theoretical and Technical Approaches

1. Introduction

The Swedish mining and construction industries have a large interest in improving blasting technology, for example in mining concerning rock fragmentation after blasting (especially underground when blasting boreholes larger than 100 mm in diameter), and both in mining and construction concerning controlled and safe contours in blasted tunnels (drifts). This documentation deals with these matters, and includes a part of my licentiate thesis, which was published in 1970, see Paper A, and another four Papers B-E, published during 1983-92.

Controlled Rock Fragmentation means suitable size distribution of the fragmented rock after blasting, for higher efficiency and lower costs in loading, transport and crushing of the blasted rock at mines, quarries and construction works. Oversize (boulders) should be minimised, and sometimes also undersize (fines) to increase the product value from quarries etc. A suitable size distribution can also reduce ore losses and waste rock dilution in mining, preferably in sublevel caving.

The main aim of the thesis is to develop technical means to achieve a more suitable rock fragmentation for bench blasting in mines and quarries, especially in large hole diameter (larger than 100 mm) blasting, see Paper C, D and E.

For sublevel caving the licentiate thesis, part of which is included in Paper A, and which was completed at the Royal Institute of Technology, dealt partly with the rock fragmentation concept, although its main goal was to test a method to quantify ore content in a mixture of ore and waste rock. The developed methodology for model tests was a good base also for the further research in this area.

Controlled contour blasting was not well developed in the middle of the 1970's, especially with regard to the way in which different kinds of reduced linear charge concentrations of explosives and timing between the contour holes influence the overbreak. From the mines and construction works in Sweden there was a general complaint about too much overbreak when blasting in bad rock conditions (poor rock) i.e. in hard intense jointed rock like granite, gneiss and magnetite. It was noted by LKAB that when controlled contour blasting was used in bad rock, a nice contour could be achieved for a short time, but after a while the bad rock was falling down.

The objective of our research, presented in Paper B, was therefore to understand the controlled blasting process better, i.e. to learn what parameters are important for the overbreak and stability of the drift. How does, for example, the timing of the contour holes influence the blast result? Is it worth while performing controlled contour blasting when rock conditions are bad? What techniques should be used? Could better techniques be developed? If the rock stability can be improved, the safety of the workers against injuries and death from falling stones would increase.
1.1 Main Objectives of the Doctoral Thesis

- **Licentiate thesis.** Test a method to quantify the ore content in a mixture of ore and waste, when loading iron ore underground in sublevel caving. The method was based on measuring of the bulk density of the ore and waste rock mixture. The objective of **Paper A** was to develop a model material, representing full-scale conditions concerning the blasted ore fragmentation in the mine and thereby be able to simulate the gravity flow process and waste rock dilution process and test the method in model scale.

- **Paper B.** Develop a controlled contour blasting technique, in tunnelling (drifts) for common Swedish bad rock (poor rock) conditions.

- **Papers C, D and E.** Develop design rules for controlled rock fragmentation especially in bench blasting and when using large diameter holes. In Paper C and D also the angle of breakage was studied and in Paper D backbreak, throw and ground vibrations.

1.2 Summaries and Results of the Licentiate Thesis and Appended Papers A-E.

In the licentiate thesis entitled "The Bulk Density Method to Determine Ore Content in a Mixture of Blasted Ore and Waste Rock - Kinematics, Swelling and Fragmentation when Blasting Against Loose Rock in Bench and Sublevel Caving (In Swedish)", literature studies were carried out concerning methods to quantify the ore content and concerning fragment size distributions, and this was followed up in a field study at the LKAB mines in Sweden. The theoretical base for the bulk density method was investigated through theoretical analysis and model tests for varying mixtures of waste rock and ore, and varying fragment size distributions, especially concerning the influence of swelling. It was found that the bulk density method could be used to determine the ore content in a mixture of ore and waste rock during production loading of iron ore, and the method was found to give an acceptable accuracy. A technical solution was therefore recommended.

Swelling, loosening and fragmentation of the blasted round were investigated by a number of model-scale tests both in sublevel caving and bench blastings layouts. The development of a suitable model material regarding fragmentation for the model blasts was reported in Paper A, see below.

Fragmentation was studied in the model tests using varying specific charges, ignition times and pressures of caved waste rock, including observations of the fragmentation distribution in different parts of the blasted round. Swelling was defined and observed as well, concerning strength of model material (cement content), caved waste rock pressure and ignition times. Movement of single particles, when loading the blasted round, was registered through plastic marks, applied in the model before blasting.

The final model-scaled tests gave, through high-speed photography, data to calculate swelling and velocities of the blasted burden and data to control the accuracy of the bulk density method.
Paper A is entitled "The Importance of Using Joints to Achieve Scaled Fragmentation in Magnetite Concrete used for Sublevel Caving Blast Models", and describes measures to obtain a suitable model material, imitating the blasting process in iron ore mining, specifically sublevel caving in the Malmberget mine. The research also included the development of a suitable blast model technique, i.e. the design of a model blast box, arrangement of drill holes, choice of explosive, charging and initiation technique, and the determination of suitable model-scale, see the licentiate thesis. Paper A describes the work involved in developing a suitable model material, representative of full-scale fragmentation.

It was recommended to use a grained material, in this case magnetite bounded with cement. Weakness planes, for instance joints, are simulated with crushed thin glass plates (used normally for the sample in microscopic analysis) or coarser magnetite grains.

Paper B, entitled "Controlled Blasting in Hard Intense Jointed Rock in Tunnels", summarises four years of research, from 1976 to 1980, initiated by problems in mines and construction works concerning overbreak in bad rock conditions in Swedish hard rock leading to accidents and higher costs. The research was based on extensive literature studies such as "Vibrations and cracking around blastholes" Rustan (1978), "Existing recommendations for controlled contour blasting of drifts and tunnels" Rustan (1979), "Methods to determine blast-induced damage in rock" Finkel (1977), "Model test in controlled contour blasting" including the selection of model material, Naarttijärvi (1978), and finally "Full-scale tests in controlled contour blasting", Naarttijärvi (1979).

The model tests, using a model material determined from the method of dimensional analysis, indicated connections concerning delay time between the contour holes and the radial crack length into the remaining rock. This initiated development of a new contour blasting method, called "cut blasting" which means that the contour holes are blasted in sequence and with a very small time delay (a couple of milliseconds) between the holes. The derivation of the new concept resulted in a formula to calculate the optimal delay time in contour blasting regarding vibration levels.

At the full-scale tests in the LKAB Malmberget mine, different kinds of contour charges and initiation times including cut blasting initiation were tested. A linear shaped charge was also developed and tested. Rock damages from the blastings were recorded with borehole viewers and vibration measurements outside the tunnel contour.

The result showed no significant change in overbreak between the different controlled blasting methods tested. The cut blasting method indicated the smallest blast-induced damage regarding measured vibration levels in the surrounding rock, but the method should be better verified. The linear-shaped charges caused a relatively large blast-induced damage zone and therefore a reduction of the diameter was recommended.

Paper B presents a broad study in the field of controlled contour blasting in tunnelling. The selected methods of testing, both in the laboratory and field, were the most modern at the time of application.

Paper C, entitled "The Influence from Specific Charge, Geometric Scale and Physical Properties of Homogeneous Rock on Fragmentation", aims to contribute to better rock
fragmentation, specifically when using large hole blasting technique underground, a goal at the Research Mine of Luossavaara (1981-1985). Based upon literature studies, model studies were performed concerning mean fragment size of blasted rock and blasting parameters (especially specific charge), blastability connected to critical burden, relationships between burden and mean fragment size and broken volume of rock, and finally some rock strength parameters, e.g. compressive and tensile strengths, strain energy release rates and acoustic impedances connected to critical burden. The influence of acoustic impedance on the fragment size distribution after blasting was shown.

Model materials used were four types of hard rock and two types of artificial materials simulating weak rock properties. Only one hole was blasted at a time in blocks with the dimensions 100x300x300 mm. Normally four single hole blasts could be made in one block. This new test method was given the name "Single hole blasting".

The result is expressed by formulas to calculate, among other values the mean fragment size of rock, rock fragment size distribution, broken volume and angle of breakage, which were based upon parameters such as specific charge, burden, spacing, impedance, etc. Blastability is best expressed through the acoustic impedance of rock and it was verified that both the mean fragment size and critical burden are dependent on the acoustic impedance.

Paper D, entitled "New Method to Test the Rock Breaking Properties of Explosives in Full Scale" is a continuation of Paper C to develop design rules for controlled rock fragmentation in blasting, including a complete blastability test consisting of determination of the fragment size distribution, rock throw and muck pile shape, angle of breakage, backbreak and ground vibrations. The new basic idea was that the performance of the explosive should be investigated in full scale with the actual blasthole diameter, explosive and bench height which are going to be used in the production blasting. The model-scale "single hole blasting" is thereby combined with a full-scale approach.

Model- and full-scale tests were carried out at the Nordkalk AB, Storugns Limestone quarry in Gotland, Sweden in 1986.

The results showed good correlation between model- and full-scale tests regarding how mean fragment size varies with the size of the burden, how the angle of breakage varies with the burden, and how the specific charge affects the fragment size distribution. The single hole blast test procedure was therefore proposed as a standard test of blastability, because blastability cannot only be defined by fragmentation. It includes the determination of critical burden, fragmentation versus burden size or specific charge, angle of breakage versus burden, throw and muckpile shape and overbreak. In the test, it was found that the acoustic rock impedance includes the most important physical and mechanical properties of rock for the mentioned result parameters. The single hole blasting test in the field is to be preferred, because much more information can be gained from that kind of test compared with the single hole model blast test in the laboratory.

Paper E, "Burden, Spacing and Borehole Diameter at Rock Blasting" emphasizes the burden-spacing relation to the borehole diameter, specifically actualized through the introduction of large hole blasting technique underground and on the surface. The old linear design formulas for small hole blasting are not valid any longer. Instead a non-linear function
should be used, owing to statistical analysis of data from more than 100 mines all over the world. The formulas differ for open pits and underground mining because of the greater confinement, longer holes (burden has to be reduced due to borehole deviation) and greater demand on well-fragmented rock for underground mining. A new concept "the practical burden when spacing to burden ratio is equal to 1" was introduced to consider the burden-spacing relation, especially in bench blasting. Future work might include in the formulas the explosive and rock density, eventually also rock structure.

1.3 Recommendations for Future Research

The existing method today to measure bulk density in the bucket of a loading machine (LHD) at LKAB is to register the pressure in the hydraulic lift and tilt cylinders. This method should be complemented by a method to register the volume loaded into the bucket. If this is possible, the accuracy of the bulk density method could be increased. Equipment to store, analyse and interpret the bulk density and volume data should be developed, and it should be possible to link these data to a general database controlling the drilling, charging, blasting and loading in sublevel caving.

The study of the fragmentation after blasting of sublevel caving rounds in the laboratory should be repeated in full scale where the swelling and fragmentation should be studied in different parts of the blasted round.

In controlled contour blasting, algorithms should be developed for determination of the blast-induced damage zone dependent on rock and explosive properties, decoupling of the borehole, impedance ratio between rock and explosive, and the different attenuation of different frequencies generated by a blast. This knowledge is very important for blast designers and for the teaching of blasting at universities.

It should also be determined how the single hole blast data can be used to plan for multiple hole blasting. This must be accomplished by field tests, and such tests are planned in a current application for EU-funding (1995).

The relation between blasthole diameter, burden and spacing presented in Paper E should be improved further by including the explosive and rock properties, the impedance ratio between rock and explosive, the spacing/burden ratio, and the timing between the blastholes.

1.4 Contributions from the Author and Others

The licentiate thesis and Paper A are wholly the work of Agne Rustan.

The research work presented in Paper B was planned and organised by Rustan, who moreover was responsible for two of the literature reviews and the development of the cutblasting theory. Rustan also took part in the analysis and interpretation of the research results, and the presentation of the results in technical and research reports. The international conference presentation in Paper B was composed by Rustan.
In Paper C, the test procedure and the performance of the tests were planned and organised by Rustan, including the hypotheses to be tested. The laboratory work was performed by Naarttijärvi (the multiple hole blasting tests in magnetite concrete) and Vutukuri and Öqvist (single hole blasting in different rock types and rock-like materials). The physical and mechanical properties of rock were determined by Ove Alm, Division of Rock Mechanics at LuTH. Rustan took part in the analysis and interpretation of the laboratory data and composed the final presentation in paper C.

Paper D. The planning and organisation of the full-scale single hole blast tests were accomplished by Rustan, who also participated in the field tests together with S.L. Nie and Jan Öqvist. The procedure to evaluate the fragmentation from the photos was developed by Rustan. Algot Persson and Ingvar Bergqvist from Nitro Nobel AB performed the measurement of the velocity of detonation, the high-speed photography and the measurement of the muck pile shape. Rustan took part in the analysis and interpretation of the data and he prepared the paper for publication.

Paper E. The literature study, the planning of the work, search for relevant data, and the technical derivations and presentations were carried out by Rustan. The statistical evaluation of the data was performed by Hans Wirstam and Anders Månsson with the computer program "Stratgraphics". Rustan prepared the paper for publication.
THE IMPORTANCE OF USING JOINTS TO ACHIEVE SCALED
FRAGMENTATION IN MAGNETITE CONCRETE USED FOR SUBLEVEL
CAVING BLAST MODELS

by

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THE IMPORTANCE OF USING JOINTS TO ACHIEVE SCALED FRAGMENTATION IN MAGNETITE CONCRETE USED FOR SUBLEVEL CAVING BLAST MODELS

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Abstract—Model scale tests have been undertaken to find a model material which can give scaled fragmentation in sublevel caving blast models where the purpose is to study the swelling and fragmentation of the burden when blasted against waste rock and also to study the gravity flow of the blasted burden when being discharged into the drift. To achieve a wished Rosin Rammler distribution of the blasted material in the blasted model, it was shown necessary to introduce weakness plans of different size and stochastic orientation in the model material. Crushed microscopic glass plates or coarse magnetite grains were used for that purpose with success.

1. INTRODUCTION

SUBLEVEL caving is the basic underground mining method used in Sweden. In 1984, 63% of the underground ore in Sweden was mined by this method. The method is mainly used by LKAB (Luossavaara-Kirunavaara AB) in Kiruna and Malmberget mines and by SSAB (Swedish Steel Co.) in Grängesberg. The trend has always been to increase the sublevel caving height. In the beginning, when the method initially was used in Sweden at LKAB in Malmberget, in the Koskullskulle orebody, the sublevel height was only 5 m and today it is being increased from 22.5 to 30 m at LKAB in Malmberget.

Much work has been undertaken through the years in Sweden and also in Canada and Australia to get a better understanding of the gravity flow of broken rock by doing model tests where the blasted burden is simulated by pouring loose magnetite into the models. The gravity flow is thereby simplified too much compared to the reality where magnetite fragments are puzzled together after blasting and magnetite blocks with sharp edges and decreased mobility are created.

This paper describes the development of a model material close to scaled fragmentation when being blasted in a model. By this the swelling and the gravity flow after blasting of the burden in the model can be accurately simulated. It was assumed that scaled fragmentation would be the best method to simulate the gravity flow, during and after blasting. The key task was therefore to find a model material which would give a scaled fragmentation.

The hypothesis set-up was that the swelling and fragmentation is affected by:

(1) The blasthole pressure and its duration (pressure history curve).
(2) The pressure from the caved rock on the burden which is being blasted.
(3) Gravity forces on the blasted material.

None has been able to measure the swelling of the burden in full scale sublevel caving. Here it is therefore assumed that the swelling is 15–20 volume-%. With the geometry used in the model blast test about 12% of the blasted burden is immediately fallen down into the drift by gravity forces. The swell factor of the blasted rock in the drift, is about 1.7. The fragmentation caused in full scale can be divided into primary fragmentation due to the blasting of the burden and secondary fragmentation when the pieces move downwards by the gravity forces. It is assumed that the best fragmentation is achieved close to the drift, because the specific charge (amount of explosive per unit volume of rock kg/m³) is larger in that part. This paper is a part of the PhD Thesis by Rustan [1].
2. DEMAND ON THE MODEL MATERIAL

The most important demands on the properties of the model material are the following:

1. The model should give a scaled fragmentation at blasting.
2. The secondary breakage caused at the gravity flow should be similar to that in full scale.
3. The model material (simulating magnetite ore) should be easy to separate visual and physically from the caved rock after blasting and after the gravity flow.
4. The burden should if possible be in scale to that in full scale.
5. The density of the model material (ore) should be about \( \frac{1.7}{4500} \) larger than the caved rock material (white limestone) because in full scale the density for magnetite is 4500 kg/m\(^3\) and for limestone 2700 kg/m\(^3\).
6. It should be possible to create long small diameter blastholes in the model material.

By the preparatory test made, the goal was to find a material which suits all these demands as good as possible.

2.1. Model laws

Most engineering sciences use different kind of models to avoid expensive full scale test and sometimes a full scale test on the whole cannot be performed practically.

It is important to point out, already in the beginning, that a complete static, dynamic and geometric model similarity cannot be achieved according to Hetényi [2].

When building a model it is therefore important to concentrate on the parameters which are most important for that process which is going to be simulated and as much as possible strive to scale these parameters correctly.

The model laws are derived by dimensional analysis, see Baker et al. [3], Hetényi [2], Langhaar [4], Mandel [5] and Schuring [6]. One example of static model laws are the scaling of density and Poisson ratio

\[
\frac{\rho_m}{\rho_f} = \frac{E_m l_f}{E_l l_m},
\]

\[
\frac{v_m}{v_f} = \frac{\nu_m}{\nu_f} = \frac{v_m}{v_f} = \frac{\nu_m}{\nu_f},
\]

where \( \rho_m \) = density in model (kg/m\(^3\)), \( \rho_f \) = density in full scale (kg/m\(^3\)), \( E_m \) = Young’s modulus in model (Pa), \( E_f \) = Young’s modulus in full scale (Pa), \( l_m \) = length in model (m), \( l_f \) = length in full scale (m), \( \nu_m \) = Poisson’s ratio in model and \( \nu_f \) = Poisson’s ratio in full scale.

The Young’s modulus for the model material used in the test presented in this paper could only be defined within a very short load range because of the very low compressive strength of the model material. The Poisson ratio for the model material was also impossible to determine because of the very low compressive strength.

The conclusion was therefore that the model laws for solid material could not be used when working in extreme low strength model materials. It was therefore decided to try to fulfil the demands on the model material by trial and error experimental work.

2.2. Model materials available

The model materials can be divided into two main groups, natural and artificial. In rock blasting the natural materials will be the rock in situ. These kind of materials cannot be used here because it is difficult to drill a straight 2-mm diameter borehole to a length of 250 mm in solid rock.

Stimson [7] has made a tabular summary over different kind of artificial model materials (see Fig. 1).

The following are examples of model materials having been used in blasting.

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Non-grained

- **Plexiglas**: Baker and Fourney [8], Gaek et al. [9], Kochanowsky [10], Kutter [11], Langefors [12] and Rustan [13]
- **Ice**: Clark and Saluja [14]
- **Glass**: Gaek [9], Rossmanith [15].
Grained
Mortar Belyaev and Akbukina [16]
Concrete Johnson [17] and Bjarnholt [18].

The artificial model materials are grouped into two main groups namely grained and non-grained. The grained materials are cheap and usually more easily to manufacture in different geometries. The non-grained materials are usually more expensive and some of them can only be ordered in some standard dimensions for example plexiglas. This material has been used by many authors to study fragmentation and crack development. However it is not possible to achieve a scaled fragmentation in plexiglas. The very fine particles are few. It is also difficult to manufacture complex geometries in plexiglas. Plexiglas could therefore not be used for our purpose.

Each model material has its advantages and disadvantages and it is the purpose which determines what material to use.

3. EXPERIMENTATION TO FIND A SUITABLE MODEL MATERIAL

The goals with these tests were to find:

(1) A suitable model material composition according to the demands.
(2) A suitable explosive.
(3) A method to charge the explosive into the 250-mm long holes.
(4) Select a suitable model scale.

Here only the work to find a suitable model material composition will be reported.
Table 1. Size distribution for magnetite from LKAB in Malmberget type MBF and MAC

<table>
<thead>
<tr>
<th>Mesh size (mm)</th>
<th>MBF (Cumulative weight-% passing)</th>
<th>MAC (Cumulative weight-% passing)</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.83</td>
<td>94</td>
<td>—</td>
</tr>
<tr>
<td>0.59</td>
<td>85</td>
<td>—</td>
</tr>
<tr>
<td>0.42</td>
<td>70</td>
<td>—</td>
</tr>
<tr>
<td>0.30</td>
<td>50</td>
<td>100</td>
</tr>
<tr>
<td>0.21</td>
<td>26</td>
<td>96</td>
</tr>
<tr>
<td>0.15</td>
<td>18</td>
<td>89</td>
</tr>
<tr>
<td>0.10</td>
<td>11</td>
<td>74</td>
</tr>
<tr>
<td>0.07</td>
<td>8</td>
<td>60</td>
</tr>
</tbody>
</table>

3.1. Binding medium

First the influence of two kinds of binding medium were studied namely plaster of Paris and cement.

3.1.1. Model materials tested. The model material used in the beginning for the model blast test was a grained material consisting of:

1. Magnetite fines, MAC and MBF (two kind of ore products delivered by LKAB). The size distributions are given in Table 1.
2. Plaster of Paris or rapid cement (the latter manufactured by Gullhögen Co.).
3. Water.

The ratio between MAC and MBF was always kept constant, 1:2 because this ratio will give the maximum density according to the theories of mixing two materials with different grain sizes. The idea is that the MAC-fines shall fill up the pores between the larger grains from the MBF.

After mixing the material it was poured into plexiglas cylinders with the diameter 42 mm and length 42 mm. After 7 days the uniaxial compressive strength of the material was tested.

A test series was undertaken to study the uniaxial compressive strength when the plaster of Paris or cement content was varied.

3.1.2. Result from uniaxial compressive strength tests. The mixture of the test cores and the uniaxial compressive strength values are shown in Table 2.

The uniaxial compressive strength shown in Table 2 is also shown graphically in Fig. 2, where it can be seen that the uniaxial compressive strength is increasing more slowly when plaster of Paris is used, compared to cement. At 5 weight-% cement content the uniaxial compressive strength for cement is about 10 times larger than for plaster of Paris. The strength of the model material when using plaster of Paris was too low and a higher content of plaster of Paris could not be used because this would decrease the density of the model material to much. It was also difficult to achieve an even hardening with the plaster of Paris and it was therefore abandoned as a binding medium.

Table 2. The content of binding medium and its influence on uniaxial compressive strength on cylinders (diameter 42 mm and length 42 mm)

<table>
<thead>
<tr>
<th>Binding medium</th>
<th>Number of cylinders tested</th>
<th>Binding medium</th>
<th>Water (weight-%)</th>
<th>MAC</th>
<th>MBF</th>
<th>Density dry (kg/m³)</th>
<th>Uniaxial compressive strength (MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Plaster of Paris</td>
<td>2</td>
<td>2</td>
<td>10</td>
<td>29</td>
<td>59</td>
<td>3080</td>
<td>0.59</td>
</tr>
<tr>
<td>Plaster of Paris</td>
<td>2</td>
<td>4</td>
<td>10</td>
<td>29</td>
<td>57</td>
<td>3090</td>
<td>0.93</td>
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<tr>
<td>Plaster of Paris</td>
<td>2</td>
<td>6</td>
<td>10</td>
<td>28</td>
<td>56</td>
<td>3080</td>
<td>1.61</td>
</tr>
<tr>
<td>Plaster of Paris</td>
<td>1</td>
<td>8</td>
<td>10</td>
<td>27</td>
<td>55</td>
<td>3060</td>
<td>2.20</td>
</tr>
<tr>
<td>Cement</td>
<td>2</td>
<td>4</td>
<td>11</td>
<td>28</td>
<td>57</td>
<td>3330</td>
<td>4.99</td>
</tr>
<tr>
<td>Cement</td>
<td>2</td>
<td>4</td>
<td>11</td>
<td>28</td>
<td>57</td>
<td>3320</td>
<td>4.89</td>
</tr>
<tr>
<td>Cement</td>
<td>2</td>
<td>4</td>
<td>11</td>
<td>28</td>
<td>57</td>
<td>3200</td>
<td>4.28</td>
</tr>
<tr>
<td>Cement</td>
<td>2</td>
<td>5</td>
<td>11</td>
<td>28</td>
<td>56</td>
<td>3300</td>
<td>7.95</td>
</tr>
</tbody>
</table>
From Fig. 2, we also see that the uniaxial compressive strength is changed much, from 5.0 to 8.0 MPa, when the cement content is varied from 4 to 5 weight-%. The model material with 4 weight-% cement content is so weak that it can be crushed with the fingers.

3.2. Sublevel caving blast models

3.2.1. Hypothesis about fragmentation in model blast test. The mean fragment size \( k_{50} \) of the one is dependent on several parameters where Rustan [1] judged the eleven most important parameters were to be

\[
k_{50} = f(q, Q, c, \rho, \tau, \psi, f, p, B, S, l)
\]

where \( k_{50} \) = average fragment size (m), \( q \) = specific charge (kg/m\(^3\)), \( Q \) = type of explosive, \( c \) = a strength parameter of the rock being blasted, for example, P-wave velocity, dynamic tensile- or compressive strength or specific joint surface area, \( \rho \) = density of the blasted material (kg/m\(^3\)), \( \tau \) = delay time between the boreholes (s), \( \psi \) = borehole geometry inclusive borehole deviations. This determines partly the next parameter, \( f \) = fixation of the holes, \( p \) = waste rock pressure on the blast front (Pa), \( B \) = burden (m), \( S \) = spacing (m) and \( l \) = scale factor.

Parameters studied in the thesis were \( c, \tau, \psi \) and \( p \). All others parameters were kept constant. In this paper only the influence of the strength properties of the model material is presented.

3.2.2. Manufacturing of the models. The casting of the models were done in a mould made by 2-mm thick steel plate (see Fig. 3). The boreholes were marked by straight \( \phi \) 2-mm steel rods which were greased by ordinary machine oil to facilitate its removement after curing of the model. The models were kept in the mould over night and the day after the mould was taken away. The model was swept with wet rags to prevent a too early drying. After 14 days of hardening the model was blasted.

All data for the blast model compared to the full scale blast are given in Table 3.

3.2.3. Blasting equipment and test procedure. Dry fine grained PETN (\( C_5H_8(NO_3)_4 \)) was poured from the top of the model into the boreholes. It was not possible to pack the PETN. The charged quantity for each borehole was measured in order to check that no hang-ups have occurred in the boreholes. If this was the case the PETN was removed and new explosive filled into the borehole.

The ignition of the blastholes were made at the top by ignition beads of type HX 20, sold by Dymatex Co., Sweden. The timing of these kind of beads is very accurate and the precision is within microseconds. They also include more explosives than ignition beads used in conventional blasting caps so they can initiate the PETN. The ignition bead must however be in close contact with the explosive. If not the explosive will fail to detonate.
Table 3. Data for the sublevel caving blast model compared to full scale data at LKAB in Malmberget

<table>
<thead>
<tr>
<th>Geometric parameters</th>
<th>Scale 1:75</th>
<th>Scale 1:1</th>
</tr>
</thead>
<tbody>
<tr>
<td>Burden</td>
<td>24 mm</td>
<td>1.8 m</td>
</tr>
<tr>
<td>Spacing (max)</td>
<td>24 mm</td>
<td>1.8 m</td>
</tr>
<tr>
<td>Front inclination</td>
<td>90°</td>
<td>varies from 70°-85°</td>
</tr>
<tr>
<td>Sublevel height</td>
<td>180 mm</td>
<td>13.5 m</td>
</tr>
<tr>
<td>Pillar width</td>
<td>87 mm</td>
<td>6.5 m</td>
</tr>
<tr>
<td>Width of drift</td>
<td>73 mm</td>
<td>5.5 m</td>
</tr>
<tr>
<td>Height of drift</td>
<td>51 mm</td>
<td>3.8 m</td>
</tr>
<tr>
<td>Area of drift</td>
<td>33.3 cm²</td>
<td>18.7 m²</td>
</tr>
<tr>
<td>Theoretical blast volume (\text{in situ})</td>
<td>613 cm³</td>
<td>258 m³</td>
</tr>
<tr>
<td>No. of boreholes in each round</td>
<td>12</td>
<td>12</td>
</tr>
<tr>
<td>The lowest borehole angle</td>
<td>50°</td>
<td>50°</td>
</tr>
<tr>
<td>Total borehole length</td>
<td>1726 mm</td>
<td>126.8 mm</td>
</tr>
<tr>
<td>Length of borehole No. 1</td>
<td>73 mm</td>
<td>5.48 m</td>
</tr>
<tr>
<td>Length of borehole No. 2</td>
<td>104 mm</td>
<td>7.80 m</td>
</tr>
<tr>
<td>Length of borehole No. 3</td>
<td>144 mm</td>
<td>10.80 m</td>
</tr>
<tr>
<td>Length of borehole No. 4</td>
<td>136 mm</td>
<td>10.20 m</td>
</tr>
<tr>
<td>Length of borehole No. 5</td>
<td>190 mm</td>
<td>14.25 m</td>
</tr>
<tr>
<td>Length of borehole No. 6</td>
<td>216 mm</td>
<td>16.20 m</td>
</tr>
<tr>
<td>Borehole diameter</td>
<td>2.0 mm (not in scale)</td>
<td>32 mm</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Blasting parameters</th>
<th>Scale 1:75</th>
<th>Scale 1:1</th>
</tr>
</thead>
<tbody>
<tr>
<td>Type of explosive</td>
<td>Detonating cord and diesel oil</td>
<td></td>
</tr>
<tr>
<td>Amount of explosive</td>
<td>3.26 g</td>
<td>≈1.200 kg</td>
</tr>
<tr>
<td>Specific charge</td>
<td>5.31 kg/m³</td>
<td>≈0.8 kg/m³</td>
</tr>
<tr>
<td>Charge concentration</td>
<td>1.89 g/m</td>
<td>0.21 kg/m</td>
</tr>
</tbody>
</table>

Fig. 4. Geometry of the sublevel caving blast models and order of initiation of the blastholes. Number within circle shows the delay number.

Table 4. Size distribution for caved rock material surrounding the sublevel caving blast model

<table>
<thead>
<tr>
<th>Size (mm)</th>
<th>Mixing Ratio</th>
</tr>
</thead>
<tbody>
<tr>
<td>3-5</td>
<td>1/3</td>
</tr>
<tr>
<td>5-8</td>
<td>1/3</td>
</tr>
<tr>
<td>8-12</td>
<td>1/3</td>
</tr>
</tbody>
</table>
Fig. 3. Mould for casting sublevel caving models in scale 1:75.

Fig. 5. Ignition prepared for the sublevel caving blast model. Filling of caved rock material (white limestone) has to be completed to the top of the blast box.
3.2.4. *Initiation.* The boreholes are initiated in the same order as in full scale, starting with the two middle holes on the same delay and continuing by always blasting the next side holes at the same time (see Fig. 4).

The delay time which gives the finest fragmentation was studied in bench slabs with parallel boreholes and the same burden and spacing as in the sublevel caving blast models. The short delays were generated by a modular pulse generator type 1395-A made by General Radio. The delay time could be varied stepless from $1 \mu s$ to 1 s. The finest fragmentation was found at a delay time of 0.1–1.0 ms.

The sublevel caving models were blasted in a wooden box reinforced at the outside by iron profiles. The size of the box was, width = 400 mm, length = 450 mm and height = 400 mm. In one of the short sides an opening with the size of a drift was made to facilitate discharge of the blasted ore.

The blast model is placed on the floor of the box and is fastened to the box with a steel coat (1-mm thick iron plate) to the bottom of the box with six 6-mm bolts. The steel coating was covering 130 mm of the length of the sublevel caving model. The coating was necessary because the dynamic forces at blasting were so large so they would move the model even when it was surrounded by waste rock.

In Fig. 5, the sublevel caving model is shown after prepared ignition, but filling of caved rock material (white limestone) to the top of the box remains. The fragmentation of the caved rock material was scaled down to model size except the fines $<3$ mm which were not used because of the creation of a lot of dust. The size distribution of the waste rock is shown in Table 4. Above the waste rock two sandbags each weighing 25 kg were located.

After blasting, the fragmented ore was separated from the waste rock by magnetic separation and sieved manually, because a mechanical sieving would decrease the particle size remarkably, due to the low strength of the material.

3.2.5. *The influence of cement content on fragmentation in sublevel caving blast models.* Rapid hardening cement was used as a binding-medium. Three different percentages were used 3, 5 and 10 weight-%. At 3 weight-% cement content the sieving has to be done very carefully because of the secondary breakage. The largest pieces are taken one by one through the meshes and the fine fragments were hand sieved as carefully as possible. Some secondary breakage still occurred but could not be avoided.

Several burdens could be blasted in one sublevel caving model but here only the first burden was used and compared at a different cement content.

3.2.6. *Result.* The result from the sieve analysis is shown in Fig. 6. The size of the fragments has been multiplied by the scale factor to make it possible to compare with the fragmentation from
a full scale sublevel caving blast. From Fig. 6, we can see that all three fragment distribution curves are too coarse compared to the real fragmentation from sublevel caving blasting. To come as close as possible, it is necessary to use a very low strength magnetite concrete with only 3 weight-% cement. This distribution does not however represent a straight line in the logarithmic size distribution diagram.

This means that the distribution does not fit the Rosin Rammler distribution

\[ y = 100 \left[ 1 - e^{-\left( x/x_c \right)^n} \right] \]  

where \( y = \) cumulative material passing mesh sizes \( x \) (weight-%), \( x_c = \) characteristic mesh size (m), \( x = \) mesh size (m). (Could be any point on the distribution curve if the exponent for \( e \) is corrected by a constant.) In the formula above \( x_c \) is chosen for \( y = 63.2\% \). \( n = \) dimensionless exponent.

To determine the distribution it is necessary to know at least two coordinate pairs in the distribution or one coordinate pair and the exponent \( n \).

The distribution in Fig. 6 at 3 weight-% cement is S-shaped with a horizontal part for fragments between 1.3 and 8.0 mm representing fragments in full scale between 0.1 and 0.6 m. These fragment sizes will not be represented in the models and therefore it is a very bad quantitative representation of the full scale rock. The conclusion was therefore that for soft magnetite concrete the middle size fractions are lacking and something has to be done to create these fragment sizes.

In Sweden magnetite ores, granites and gneisses mostly have three major joint sets. In the model material tested here there were almost no joints, or at least no large ones. One idea was therefore to introduce joints or weakness planes much larger than the grain size of the model material. These weakness planes were supposed to act as joints and therefore the middle size fragments was anticipated to be created.

The crack pattern in the burden after blasting in magnetite concrete with a very high cement content, 5 weight-%, is shown in Fig. 7. Notice that some cracks are oriented just in front of the blast holes. The fragmentation which falls down into the drift is shown in Fig. 8.

The blast front after blasting is shown in Fig. 9. Note that in this case hole No. 7, 8 and 9 misfired. Note also the similarity with a blast front in full scale.

3.2.7. Introduction of artificial joints. The importance of joints on the fragmentation has been studied by many authors, see Fourney, Baker and Holloway [19], Rossmanith [20] Singh and Sarma [21], Yang and Rustan [22], and Winzer and Ritter [23]. The interaction of P- and S-waves with joints is very important for the nucleation and propagation of cracks. The local stress field determines in what direction the cracks will propagate.

The joints were simulated by crushed microscopic glass with the dimension 0.10 x 24 x 36 mm. The maximum length of the crushed microscopic glass was 20 mm. The stress waves from the P- and S-waves will interact with the artificial joints and cause fragmentation of different sizes dependent on the size of the weakness planes.

The impedance mismatch between the glass and the magnetite concrete was close to 1 according to the following calculation.

\[ \frac{Z_g}{Z_m} = \frac{\rho_g \cdot c_{g_p}}{\rho_m \cdot c_{p_m}} \]

where \( Z_g = \) impedance of glass (kg/m^2 s), \( Z_m = \) impedance of magnetite (kg/m^2 s), \( \rho_g = \) density of glass (kg/m^3), \( \rho_m = \) density of magnetite (kg/m^3), \( c_{g_p} = \) P-wave velocity in glass (m/s) and \( c_{p_m} = \) P-wave velocity in magnetite (m/s),

\[ \frac{Z_g}{Z_m} = \frac{2500 \cdot 5580}{3460 \cdot 3480} = 1.16. \]

In this model a test series consisting of four burdens one after the other were blasted and examined. The specific joint surface area was varied linearly. It is defined as follows:

\[ \eta = \frac{A_j}{V_j} \]
Fig. 7. Crack pattern in the burden after blasting in a model material where the compressive strength has been too large (7.95 MPa), 5 weight-% cement.

Fig. 8. Fragmentation in the model drift after blasting in the sublevel caving model and before start of ore drawing.
Fig. 9. The blast front after blasting in a sublevel caving model. Three holes misfired. (From left hole No. 7, 8 and 9.)

Fig. 11. The blast front after blasting in a sublevel caving model where microscopic glass has been mixed into the model material to simulate the joints in full scale magnetite.
Scaled fragmentation in magnetite concrete

Table 5. Specific joint surface areas tested in the burdens

<table>
<thead>
<tr>
<th>Burden (No.)</th>
<th>No. of glasses size 24 × 36 mm</th>
<th>Joint specific surface area (1/cm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>2</td>
<td>50</td>
<td>1.17</td>
</tr>
<tr>
<td>3</td>
<td>100</td>
<td>2.35</td>
</tr>
<tr>
<td>4</td>
<td>150</td>
<td>3.52</td>
</tr>
</tbody>
</table>

$\eta =$ specific joint surface area (1/m), $A_j =$ joint area within volume $V_j$ (m$^2$) and $V_j =$ volume occupied by the joint area $A_j$ (m$^3$).

In Table 5 all specific joint surface areas used in the model test are shown. The cement concentration used was 10 weight-% and the delay time between the boreholes, $\tau = 0.1$ ms.

3.2.8. Result. The specific joint surface area and its influence on fragmentation is shown in Fig. 10. The larger the specific joint surface area the more fine fragmentation. Some of the increase of fine fragmentation is due to the fact of increased cracking of the model due to blasting of repeated burdens in the model, but only one burden was blasted at each time. A very important observation is, however, that the fragment size distribution curve is almost straight in the logarithmic size distribution diagram. This means that all sizes of fragments are represented. This was also the goal for the development of the model material. The slope of the fragment size distribution ($n \approx 0.70-0.90$) is also very similar to the fragmentation in full scale sublevel caving blasting ($n \approx 0.75$).

A close examination of the glass plates with a magnifying glass reveals that the glass plates sometimes are cracked and sometimes not. Sometimes the glass plates have disappeared at blasting. The remaining surface behind was flat and undisturbed. No fracturing perpendicular to the joint surface could be observed.

The blast front before blasting of the fourth burden is shown in Fig. 11. Notice the stochastic oriented joint surfaces which simulates a real blasted rock surface even better than that shown in Fig. 9.

3.2.9. Artificial joint surfaces created by coarse magnetite grains. In order to simplify the manufacturing and control of the joint, size, tests have been done later on using coarse magnetite grains as joint surfaces. It was shown that larger grains could act as joint surfaces.

\[ \text{Fig. 10. The fragmentation and its dependence on specific joint surface area (crushed microscopic glass) in the sublevel caving blast model material.} \]
**4. CONCLUSIONS**

To achieve a scaled fragmentation in sublevel caving blasting models it is necessary to use a grained model material, in this case magnetite bounded together with cement has been used. In order not to lose the middle size fragments, crushed microscopic glass or any other weakness plane material for example, coarse magnetite grains have to be mixed into the model material to create weakness planes simulating the joints in full scale.

The advantage of the latter method is that the model material is easier to manufacture. The size distribution of the crushed microscopic glass is difficult to control. One solution however could be to sieve the crushed glass and thereafter mix the different sizes according to a determined recipe. The method of using coarse grains is however to be preferred and is therefore recommended by the author.

This work shows that weakness planes in the natural rock plays an important role for the fragment size distribution and it cannot be omitted when the goal is to create a scaled rock fragmentation in weak model materials.

**REFERENCES**


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CONTROLLED BLASTING IN HARD INTENSE JOINTED ROCK IN TUNNELS

by

Agne Rustan
Torbjörn Naarttijärvi
Bengt Ludvig

CIM Bulletin, December 1985
ROCK MECHANICS

Controlled blasting in hard intense jointed rock in tunnels

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ABSTRACT

Falling rock is one of the major causes of serious injuries and deaths in Swedish mines. The research at the Division of Mining and Underground Construction was therefore directed toward controlled blasting in hard intense jointed rock.

The project was started with an extensive literature review. Model tests with a new equivalent model material have been initiated to find ways of improving blasting techniques. The result showed that long cracks arise at the perimeter holes when there is a long time delay between adjacent holes. This led to the theoretical development of a new initiation method with ultra short delays (1.5 ms) for controlled blasting, called "Cutblasting".

Full-scale tests have been done at LKAB in Malmberget in hard intense jointed magnetite ore. Four different types of perimeter charges have been tested, namely: tube charges, ANFO mixed with plastic beads, detonating cord and linear-shaped charges. Three types of initiation of the perimeter holes have been used, conventional (half-second delay detonators), instantaneous and ultra short "cutblasting initiation" (1.5 ms delay). The change in rock strength before and after blasting has been studied with bore hole periscope, vibration measurement and ocular observation of the rock surfaces.

The full-scale tests showed that "cutblasting" with detonating cord in the perimeter holes gave the smallest damage to the surrounding rock. A combination of cutblasting initiation and tube charges are anticipated to give an even better result. A new method of controlled blasting using linear-shaped charges has also been tested, however, this charge type needs further development.

The full-scale test also showed that the damage which can be seen on the rock surface is about the same for different controlled blasting methods but the not-visual-damage zone varies as much as from 0.1 to 0.7 m.

A new classification system for controlled blasting regarding the damage to the surrounding rock has been devised.

Introduction

Accidents from falling rock in intense jointed rock mass in tunneling is one of the major reasons for serious injuries and deaths in Swedish mines.

A meeting arranged by Prof. I. Janelid and M. Finkel, Royal Institute of Technology regarding the use of controlled blasting in Swedish mines and construction works in 1974, showed that the result from controlled blasting was satisfactory in hard rock with little or medium joint frequency but there was no acceptable method in hard intense jointed rock. The mines engineers

Paper reviewed and approved for publication by the Rock Mechanics and Strata Control Committee of CIM.
agreed that the result was the same even though controlled blasting was not used.

Sponsored by the Swedish Work Environment Fund, the research at the Luleå University of Technology was therefore directed toward controlled blasting in hard intense jointed rock. The Department of Mining and Underground Construction at the Luleå University of Technology has conducted research in controlled blasting for tunneling from 1977 to 1980.

In 1976 the project started with an intense review of the literature of controlled blasting and has been reported in the following reports:

1) Methods to Evaluate the Damage from Blasting in the Rock
2) Typical Structure of “Bad” Swedish Rock
3) Vibrations and Fracturing around a Blasthole
4) Model- and Full-scale Tests in Controlled Blasting
5) Recommendations for Controlled Blasting

**Literature Review**

At the outset, it was realized that it would be necessary to examine the available methods of detecting damage in surrounding rock after blasting in the tunnels or drifts.

This study was done by Menachem Finkel at the Royal Institute of Technology in 1976. Fifteen different methods were evaluated considering both the efficiency of the method and the economy. The following two methods were chosen for the full-scale tests:

1. **Monitoring of the vibrations, 2-20 m from the perimeter holes.** The smallest distance between the perimeter hole and the accelerometer varied between 2 - 6 m. The technique for monitoring vibrations in the bore hole was adopted from the Swedish Detonic Research Foundation (SveDeFo). It is very important that the accelerometers are installed in a correct manner in the bore holes. Several tests with different installations were made which confirmed that the SveDeFo method was the best.

2. **Periscope (bore hole viewer).** The periscope uses a binocular to look into the bore hole and the bore hole wall could be seen via a mirror set at a 45-degree angle to the bore hole axis. The hole is lighted by a battery-powered light bulb.

   The periscope is manufactured by Hagconsult in Stockholm. The maximum depth at which this instrument could be used is only 3 - 4 m when looking on dark rock. The method is very subjective, making it necessary that the same person make the observations in all tests.

   A literature review of all typical structures of “bad” rock in Sweden has been reported by Ludvig. His conclusion was that hard intense jointed rock (three-joint sets) is the most characteristic for “bad” rock in Sweden and therefore the field tests have been concentrated on this type of rock.

   Rustan has studied, in a literature review, the fracture mechanism around a blast hole. A summary of this study gives all the important parameters in controlled blasting (Fig. 1).

   The development and role of the micro-cracks around a blast hole is not very well mapped in field tests and therefore more work must be done.

   A review of performed model- and full-scale tests has been done by Naarttijärvi. This study showed that little effort has been exerted to find equivalent material for model blast tests. Therefore a study of the model scaling law technique was conducted until finally a model material was found suitable for studying the length of the cracks in the model tests.

**Model Test**

The equivalent model material consists of a mixture of epoxy and quartz sand (silversand). Weakness planes to simulate blocky hard rock are introduced in the models. The plate is heated to 250°C in a heat chamber and the plastic sheet more or less dissolves. A maximum of two different joint systems was achieved in the models.

The models are, after hardening, placed into a loading frame where they could be loaded in two directions. The applied load...
corresponds to a depth of the tunnel below ground of 50 m.

Only the perimeter holes were blasted against the opening left after casting. Detonating cord (3 x 1 g/m) was used as tube charge corresponding in strength to 17 mm diameter Gurit charges. Different initiation sequences were used, for example the "conventional" which in Sweden means a simulation of half-second delays for the perimeter holes. The maximum time scatter of Nitro Nobel's half-second detonators is ± 150 ms.

The model tests showed that when the time difference between two perimeter holes is large, long cracks develop in the surrounding material. Ideally the blast holes should detonate simultaneously in the perimeter.

Which is the optimal detonation pattern for the perimeter holes? If the holes are detonated at exactly the same time, the pressure waves will meet exactly half-way between the perimeter holes and the amplitude will be twice as much as if only one hole was blasted. This is, of course, not very good for the surrounding rock stability. The best way would be to let the P- and S-waves from the first hole reflect in the free surface and disappear before the next hole is detonated. This is the main principle for the new "Cutblasting" technique. A theoretical formula has been established which could be used to calculate the optimal delay time between the perimeter holes.

\[
I_0 = \frac{B \cdot \sqrt{B - S^2}}{c_S} \cdot 10^3
\]

where

- \(I_0\) = optimal delay time between two perimeter holes (ms)
- \(B\) = burden for the perimeter holes (m)
- \(S\) = spacing for the perimeter holes (m)
- \(c_S\) = speed of the S-wave (m/s)
- \(k\) = safety factor which is chosen so that the reflected S-wave has passed the neighbouring hole. The safety factor is set at 1.5

For example:

- \(c_S = 2000\) m/s
- \(B = 0.8\) m
- \(S = 0.64\) m
- \(I_0 = 1.4 \pm 0.2\) ms

The scatter in delay time for each blasting cap should not be larger than ± 0.2 ms to avoid cooperation of the waves.

With "conventional" initiating technique it is not possible to achieve the cutblasting effect in the same blast. In field tests the
cutblasting effect was achieved by blasting the perimeter holes in a separate round (Fig. 2).

"Cutblasting" has been proposed as a working term because the rock is cut out the way a knife cuts bread (Fig. 3). This cutblasting should not be confused with blasting the first part of a tunnel round which is called "the cut".

A summary of the different methods for controlled blasting is given in Figure 4.

With a new delay technique for detonators, i.e. electronic delays, the cutblasting technique could be used in underground tunnelling.

**Full-Scale Tests**

The objective of the full-scale tests has been to compare different methods for controlled smooth blasting and to develop more effective methods.

The full-scale tests have been conducted in a blocky hard rock with three different joint systems, which could be found in both granite and magnetite.

The magnetite in the Captain orebody at LKAB in Malmberget, where full-scale tests have been made, shows this typical structure. The problems with rock stability were enormous.

The tests have been performed in a sublevel caving drift on the 540 m level (Fig. 5).

The vibrations from the perimeter holes have been measured in and from the above-lying crosscuts as shown in Figure 6.

All the vibration-monitoring equipment has been installed in the Mercedes laboratory bus owned by the University of Luleå. From drift No. 544, holes have been drilled down to the test drift No. 543 and triaxial accelerometers inserted.

**Geological Description of the Test Area**

The rock in the test drift consists of magnetite of mass type. Three separate joint systems could be found in the orebody; one parallel to the footwall, one perpendicular to the dipping and a third system crossing these two. Figure 7 shows a typical view of the ore structure.

A careful mapping of all the cracks and discontinuities in the test area has been performed by Ludvig and in Figure 8, the results from the mapping of round Nos. 10, 11 and 12 are shown.

The six drill holes for the periscope were 3.5 m long and their directions are shown in Figure 6—two holes in each wall and two holes in the roof. The observation holes were used before and after the blast.

The drill and blast plan for the crosscut is shown in Figure 9. Millisecond delays are used for the cut and half-second delays for the rest of the round. This plan was used for all the full-scale test rounds and is the same as the one used at LKAB in normal drifting. Before the full-scale test started vibration monitoring was done to detect if the inner holes gave larger vibration than the perimeter holes, but as this was not the case, it was not necessary to change the drill plan. During all the full-scale tests the greatest vibration was from the perimeter holes.

To insure that the perimeter holes are not too confined, the ratio between spacing and burden is established at two. Nitro Nobel recommends a S/B ratio of 0.8.

The drilling was performed with a 2-boom Joy jumbo with a drill hole diameter of 45 mm. For accurate drilling of the perimeter holes a special vertical alignment equipment Inclinator Type 1A, Transtronic (Sweden) was used.

Four different types of explosives have been used for the column charge in the perimeter holes (Table 1).

The linear-shaped charge is shown in Figure 10.

Only one metal, V, has been used, because if a notch is created at one side of the bore hole wall, immediately a second crack is developed 180 degrees from the first.

Three different types of initiation of the perimeter holes have been used.

1. Conventional initiation technique with electric half-second detonators.
2. Cutblasting initiation with ultra short delays.
3. Simultaneous initiation, with half-second detonators.

A summary of all technical data from the fifteen blasted rounds are shown in Table 2.

Results

The field tests were conducted during 1979-1980 and the results are very interesting. However, one of our hypotheses could not be verified, i.e. that the best controlled blasting method gives least overbreak.

The different methods for controlled blasting which have been used do not show any significant change in overbreak. In Figure 11 the number of cracks indicated from the periscope per metre is shown before and after the blast. The filled piles show the additional cracks after blasting. It seems to be a correlation in such a way that a round with few natural joints before the blast gets many blast-induced cracks (rounds 10 and 14).

Figure 11 gives a summary of vibration levels from some of the rounds in the field test area. The number of cracks are not comparable between different rounds because the distance between the perimeter and observation holes changes.

The damage around a perimeter hole in the roof is shown in Figure 12.

The bottom charge gives a larger damage zone (higher vibration level) than the column charge. A damage model has been constructed to show the principal damage zones around a tunnel blast (Fig. 13).

Vibration Measurements

The vibrations in the surrounding rock have been monitored at two points; one point with 3 orthogonal accelerometers close to the perimeter — 2 m mounted in a drill hole and another point with 3 orthogonal geophones further away on the surface in the above-lying crosscut — 20 m away. The system chosen so that all occurring frequencies could be measured.

The summary of all vibration monitoring results is shown in Figure 14.

From Figure 14 it can be seen that detonating cord ignited by the cutblasting method gives the lowest vibration level of all compared methods up to a distance of 9 m. Above this distance detonating cord with half-second delays gives the lowest vibration. A combination of cutblasting with tube charges (Gurit) would possibly give an even lower vibration level within 9 m distance.

A comparison of two possible methods, vibration monitoring and periscope viewing to evaluate rock damage, is shown in Table 3.

The combination of ANFO and plastic beads shows an unexpected high-damage zone. Good results have been reported with this method from Norway and Australia. The result might be improved if the content of plastic beads is increased.

The linear-shaped charges also caused a relatively high-damage zone. In this case it is also possible to reduce the damage zone by decreasing the charge concentration. Therefore the work with linear-shaped charges is continued in a separate project sponsored by the National Swedish Board for Technical Development.

A new classification system for rock damage when controlled

**TABLE 2. Summary of the charge and initiation technique for all rounds**

<table>
<thead>
<tr>
<th>Round No.</th>
<th>Bottom Charge</th>
<th>Column Charge</th>
<th>Total Charge</th>
<th>Bore Hole Pressure</th>
<th>Method of Initiation</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Explosive</td>
<td>Charge Conc.</td>
<td>Explosive</td>
<td>Quantity</td>
<td></td>
</tr>
<tr>
<td></td>
<td>(kg/m DXB)</td>
<td>(kg/m DXB)</td>
<td>(kg DXB)</td>
<td>(MPa)</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Length (m)</td>
<td>Total Charge</td>
<td>Bore Hole</td>
<td>Method of</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Charge Conc.</td>
<td>Pressure</td>
<td>Initiation</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>(kg/m DXB)</td>
<td>(MPa)</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>3</td>
<td>ANFO*</td>
<td>1.51 Gurit</td>
<td>1.24</td>
<td>49</td>
<td>Conventional</td>
</tr>
<tr>
<td>4</td>
<td>ANFO</td>
<td>1.51 Gurit</td>
<td>1.24</td>
<td>49</td>
<td>Conventional</td>
</tr>
<tr>
<td>5</td>
<td>ANFO</td>
<td>1.51 Gurit</td>
<td>1.24</td>
<td>49</td>
<td>Conventional</td>
</tr>
<tr>
<td>6</td>
<td>ANFO-</td>
<td>1.51 ANFO-</td>
<td>1.16</td>
<td>18</td>
<td>Conventional</td>
</tr>
<tr>
<td>7</td>
<td>ANFO-</td>
<td>1.51 ANFO-</td>
<td>1.16</td>
<td>18</td>
<td>Conventional</td>
</tr>
<tr>
<td>8</td>
<td>ANFO-</td>
<td>1.51 ANFO-</td>
<td>1.16</td>
<td>18</td>
<td>Conventional</td>
</tr>
<tr>
<td>9</td>
<td>DXB**</td>
<td>1.12 Gurit</td>
<td>0.82</td>
<td>49</td>
<td>Simultaneous</td>
</tr>
<tr>
<td>10</td>
<td>DXB</td>
<td>1.12 Detonating cord</td>
<td>0.98</td>
<td>22</td>
<td>Conventional</td>
</tr>
<tr>
<td>11</td>
<td>ANFO</td>
<td>1.51 Gurit</td>
<td>0.94</td>
<td>49</td>
<td>Conventional</td>
</tr>
<tr>
<td>12</td>
<td>ANFO-</td>
<td>1.51 Detonating cord</td>
<td>0.98</td>
<td>22</td>
<td>Cutblasting</td>
</tr>
<tr>
<td>13</td>
<td>ANFO</td>
<td>1.51 ANFO- and plastic beads</td>
<td>1.16</td>
<td>18</td>
<td>Conventional</td>
</tr>
<tr>
<td>14</td>
<td>ANFO</td>
<td>1.51 ANFO- and plastic beads</td>
<td>1.16</td>
<td>18</td>
<td>Conventional</td>
</tr>
<tr>
<td>15</td>
<td>ANFO</td>
<td>1.51 ANFO- and plastic beads</td>
<td>1.16</td>
<td>18</td>
<td>Conventional</td>
</tr>
</tbody>
</table>

*ANFO = Ammonium nitrate and fuel oil.
**DXB = Dynamex B.
*** = In this round the perimeter holes were blasted one by one, the scatter of initiation could therefore be regarded as very large (infinite).
TABLE 3. Comparison of damage zone around a tunnel perimeter determined with vibration monitoring and bore hole viewing

<table>
<thead>
<tr>
<th>Round No.</th>
<th>Column Charge</th>
<th>Initiation Method</th>
<th>Damage limit* (m)</th>
<th>Vibration Monitoring</th>
<th>Bore Hole Viewing</th>
</tr>
</thead>
<tbody>
<tr>
<td>12</td>
<td>Detonating cord</td>
<td>Cutblasting</td>
<td>0.10</td>
<td>0.4</td>
<td></td>
</tr>
<tr>
<td>9</td>
<td>Tube charge</td>
<td>Instantaneous</td>
<td>0.25</td>
<td>0.3</td>
<td></td>
</tr>
<tr>
<td>5, 6 and 11</td>
<td>Tube charge (Gurit)</td>
<td>Half-second detonators</td>
<td>0.35</td>
<td>0.5</td>
<td></td>
</tr>
<tr>
<td>8, 13 and 15</td>
<td>ANFO + plastic beads</td>
<td>Half-second detonators</td>
<td>0.55</td>
<td>0.7</td>
<td></td>
</tr>
<tr>
<td>10</td>
<td>Detonating cord</td>
<td>Half-second detonators</td>
<td>0.65</td>
<td>0.75</td>
<td></td>
</tr>
<tr>
<td>14</td>
<td>Linear-shaped charge</td>
<td>Half-second detonators</td>
<td>&gt;0.6</td>
<td>—</td>
<td></td>
</tr>
</tbody>
</table>

* Vibration damage limit is set at 700 mm/s. Damage limit observed by periscope is set to that distance where more than three new cracks develop per metre. If the number of cracks is lower, there is a considerable uncertainty in determining if the rock has been damaged because of measurement problems.

Damage is based on particle velocities larger than 700 mm/s or alternatively more than three new cracks per metre observed hole length. With this system, the proprietor of a building can prescribe a certain damage class to the contractor for a given underground construction work.

Acknowledgment

Thanks is given to the Swedish Work Environment Fund LKAB in Malmberget and Luleå University of Technology for their help in sponsoring the project and for permitting the publication of this paper. We are also grateful to LKAB in Malmberget for their help in preparing the test rounds.
THE INFLUENCE FROM SPECIFIC CHARGE, GEOMETRIC SCALE, AND PHYSICAL PROPERTIES OF HOMOGENEOUS ROCK ON FRAGMENTATION

by

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THE INFLUENCE FROM SPECIFIC CHARGE, GEOMETRIC SCALE AND PHYSICAL PROPERTIES OF HOMOGENOUS ROCK ON FRAGMENTATION

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ABSTRACT

One of the major goals at the Swedish Research Mine at Luossavaara, 1981—1985, is to achieve a fragmentation with a minimal number of boulders, when using underground large hole blasting technique (Ø 165 mm holes). This will make it possible to introduce continuous loading and transportation of the blasted ore even in hard rock mines.

The Department of Mining and Underground Construction at the Luleå University of Technology has on contract for the Research Mine studied how specific charge, geometric scale and physical properties of rock effects the fragmentation. Model blast tests have been done during 1981-1982 in six different homogenous materials, 4 types of hard rock and 2 types of artificial made weak rocks with the aim to establish a more accurate formula for blast fragmentation predictions. The blastings have been done in slabs, 100 mm high and 1200x1200 mm, as multiple hole blasts (only one hole blasted each time) in magnetite concrete to determine the influence from specific charge and geometric scale. Critical burden tests have been made in blocks 100x300x300 mm as single hole blasts in the six materials mentioned.

The result shows that the angle of breakage decreases with increasing burden and the maximal volume of broken rock is achieved just before the critical burden is reached. The impedance was the most important physical property when calculating the blastability. It was also found that critical burden was not sufficient to characterize the whole fragmentation distribution, but it was one necessary part of the blastability concept. Other factors which have to be given are the whole size distribution curve at different burdens. The critical burden did not vary much between rocks with different uniaxial compressive strength (175-330 MPa), not more than 30%. Formulas for calculating angle of breakage, broken volume and mass, critical burden, k_C, and slope of fragmentation in homogeneous materials in model scale are given in the report.
1 BACKGROUND WITH LITERATURE REVIEW

One of the major goals for the Research Mine at Luossavaara (1981-1985) at Kiruna, Sweden is to achieve a fragmentation with a minimum number of boulders when using large holes Ø 165 mm in a narrow (~20 m) magnetite ore-body. If the number of boulders could be reduced this will for the first time open the way to automatic loading and transportation of hard rock ore from the open stopes to the ore chute.

In spite extensive research has been done to study fragmentation by blasting there are still much fundamental information missing. This was discovered during an extensive literature review of fragmentation made by Rustan 1981. The study shows that there are several empirical formulas for blast calculations, but none of the equations gives the whole fragment size distribution in full scale blasting. For example a very common used formula for rough hand calculations used in Sweden is

\[ B = 46 \cdot d \]  

where

- \( B \) = burden (m)
- \( d \) = hole diameter (m)

This kind of formula doesn't describe the size distribution. Other authors have therefore developed methods to calculate the midpoint of the fragmentation curve \( k_{50} \) or \( k_{80} \) from the specific charge used. The following formulas have been found in the literature.

\[ k_{80} = k_1 \cdot \frac{1}{q^2} \]  

BOND (2) Lowrison, 1974

\[ k_{80} = k_2 \cdot \frac{1}{q} \]  

RITTINGER (3) Lowrison, 1974

\[ k_{80} = k_3 \cdot \frac{1}{e^{q}} \]  

KICK (4) Lowrison, 1974

\[ k_{50} = k_4 \cdot \frac{1}{q^{0.56}} \]  

OLSSON (5) Olsson, 1952

\[ k_{50} = k_5 \cdot \frac{Q^{1/6}}{q^{4/5}} \]  

KUZNETSOV (6) Kuznetsov, 1973

\[ k_{50} = k_6 \cdot \frac{(B \cdot B_i)^{0.29}}{q^{1.18}} \]  

SveDeFo (7) Lundborg, 1971

where

- \( k_{80} \) and \( k_{50} \) = the mesh size where 80 respectively 50 weight-% of the fragmented material passes
- \( k_1 \) - \( k_6 \) = constants depending on rock and blasting parameters
- \( q \) = specific charge (powder factor) kg/m\(^3\)
Q = amount of explosive per hole (kg)
B = real used burden (m)
B₁ = burden used if burden and spacing are equal (m)

The exponent for q is varying between 0.56 and 2. We got therefore interested to see what rock parameters determine the exponent.

The Kuznetsov formula takes into account the total charge per hole. For a constant bench height this could be a way to tell how the fragmentation is affected by the hole diameter. The Kuznetsov formula is also the only one including a rock strength parameter, in this case the compressive strength of rock. Our model tests have unfortunately shown that the compressive strength cannot be correlated with the fragmentation \(k_{50}\) for model blasts in homogeneous rock.

The formula developed by the Swedish Detonic Research Foundation takes into account also the relation between hole distance and burden \(S/B\) and the geometric scale because the blasted area per hole and the burden are included in the formula. Still the rock parameters are lacking in the formula. They are included in the constant.

The goal for our model tests was therefore to determine

- Why the exponent of q is not constant.
- Check how the geometric scale and specific charge effect the fragmentation.
- Include a parameter which describes the blastability of homogeneous rock in a mathematical formula.
- Include a factor which describes the influence of joints with different frequency, direction and strength. This work was done by Yang Zu Guang 1983, a guest researcher at the div of Mining and Rock Excavation at Luleå University of Technology. The result from this work is reported under session 6 at this symposium.

Besides the mathematical formulas mentioned here, there are still two methods to determine fragmentation. One is a descriptive method where the radial crack distribution is used to calculate the fragmentation. This method was developed by Harries (ICI) 1978 in Australia. The second method is to use finite element or finite difference codes to calculate the fragmentation caused by the pressure waves propagating into the rock. Fracture mechanic formulas have been used to describe when the rock breaks. The latter method was developed by a group of scientists at department of Shock Physics at Stanford International Research Institute, McHugh, Curran and Seaman 1980 and has afterwards been adopted with modifications by Sandia Corporation in Albuquerque, New Mexico by Grady, Kipp and Smith 1980 and Los Alamos lab in New Mexico by Johnson, 1979.

2 FRAGMENTATION VERSUS SPECIFIC CHARGE AND GEOMETRIC SCALE - MODEL TESTS

To determine the influence of specific charge and geometric scale on fragmentation, single hole model tests have been performed in magnetite concrete, a mixture consisting of 74 weight %-% magnetite, 13 weight –% cement and 13 weight –% water. The size of the model is shown in Fig 1.
Fig. 1 Magnetite concrete model used for multiple hole blasts, only one hole blasted each time, to study the influence of specific charge and geometric scale on fragmentation.

The following model data were used.

Model dimensions: 100×1200×1200 mm. Height 100 mm  
Borehole diameter 3 mm (1 g/m), 4 mm (3 g/m) and 5 mm (5 g/m)  
(Charge concentration)  
Burden: 25, 35, 45, 55 and 65 mm  
Spacing: 50, 70, 90, 110 and 130 mm  
Spacing/burden: S/B = 2  
Explosive: PETN (detonating cord with the charge concentrations 1, 3 and 5 g/m)  
Ignition: Nitro Nobel HX20 microsecond ignition pearls with 27 mg ± 5 mg PETN and 7 mg ± 2 mg silverazide  
Number of holes repeated with the same parameters: 10-25  

Plastic foam was used to prevent secondary breakage. After blasting all fragments were collected and sieved.

The angle of breakage was measured on the top of the model and the mass broken is given by the sieve analyse. The orientation, number and length of the radial cracks are measured in the remaining model material and the penetration depth is defined after Fig. 2.
The strength data for the magnetite concrete compared to other rocks or rocklike material are shown in Table 1. The influence of specific charge on angle of breakage and crack penetration at constant burden, spacing and decoupling is shown in Table 2.

Table 1. Strength data and critical burdens for materials blasted.

<table>
<thead>
<tr>
<th></th>
<th>Primary data</th>
<th>Secondary data</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Uniaxial compressive strength MPa</td>
<td>Static elastic modulus GPa</td>
</tr>
<tr>
<td>Magnetite concrete</td>
<td>12.1</td>
<td>6</td>
</tr>
<tr>
<td>13 weight -% cement</td>
<td>14.4</td>
<td>7.7</td>
</tr>
<tr>
<td>Magnetite concrete</td>
<td>160</td>
<td>53</td>
</tr>
<tr>
<td>6 weight -% cement</td>
<td>188</td>
<td>52</td>
</tr>
</tbody>
</table>
Table 2. The influence from specific charge on angle of breakage and crack penetration at constant burden, spacing and decoupling.

<table>
<thead>
<tr>
<th>Charge concentration, PETN (g/m)</th>
<th>Bore-hole diameter (mm)</th>
<th>Burden (mm)</th>
<th>Spacing (mm)</th>
<th>Number of holes tested</th>
<th>Angle of breakage, top side (°)</th>
<th>Crack penetration depth (mm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>3</td>
<td>25</td>
<td>50</td>
<td>10</td>
<td>112-168°</td>
<td>143°</td>
</tr>
<tr>
<td>3</td>
<td>4</td>
<td>&quot;</td>
<td>&quot;</td>
<td>21</td>
<td>125-180°</td>
<td>152°</td>
</tr>
<tr>
<td>5</td>
<td>5</td>
<td>&quot;</td>
<td>&quot;</td>
<td>21</td>
<td>103-180°</td>
<td>148°</td>
</tr>
</tbody>
</table>

There is a small increase in breakage angles at higher charge concentrations from 143 to 150°. A greater crack penetration depth was expected with increasing charge concentration, but the opposite was achieved.

The influence from charge concentration on $k_{50}$ at a constant burden of 25 mm and hole distance 50 mm is shown in a double logarithmic diagram, see Fig. 3.
The influence from charge concentration on \( k_{50} \) is linear at 25 mm burden. When the burden is increased at 5 g/m, \( k_{50} \) increases. The relationship between \( k_{50} \) and charge concentration at higher burdens is anticipated to follow the dashed lines. The following formula has been established

\[
k_{50} = 1.1 \cdot B^{3.22} \cdot Q_p^{-1.18}
\]

(8)

\( k_{50} \) = mesh size where 50 weight -% of the blasted material passes (mm)

\( B \) = burden (mm)

\( Q_p \) = charge concentration (g/m)

\( k_{50} \) versus specific charge is shown in Fig. 4.

Fig. 4 \( k_{50} \) versus specific charge

After regression analysis the following formula has been established

\[
k_{50} = 0.44 (B \cdot S)^{0.46} q^{-1.18}
\]

Only valid for magnetite concrete 25 < \( B \) < 65 mm
3 < \( Q_p \) < 5 g/m
0.8 < \( q \) < 4 kg/m

(9)

S/B = 2
The exponent for $q$ is the same as that used by the Swedish Detonic Research Foundation (SveDeFo) in their formula. The SveDeFo formula is based on sieve analysis from full scale blasts in Swedish hard rock, granite and gneiss. The same exponent was not expected because of big differences in scale, different materials, explosives and decoupling. The first step of establishing a new mathematical formula now has been done, but the formula is not complete until the rock properties have been included in the formula. The completion can be divided into two steps.

- The properties of homogeneous rock
- The properties of jointed rock

In the following the influence from the properties of the homogeneous rock is going to be investigated.

The properties of jointed rock and its influence on fragmentation is reported separately by Yang Zu Guang 1983.

3 THE INFLUENCE OF PROPERTIES OF ROCK OR ROCKLIKE MATERIAL ON ANGLE OF BREAKAGE, CRITICAL BURDEN AND FRAGMENTATION

Part 1. Pilot study

The goal of this part was to establish the relation between critical burden and potential strain energy release rate for the material blasted. Critical burden is defined as the smallest burden without breakage.

Dimensional analysis was used to determine suitable $\pi$-factors and one interesting $\pi$-factor which was studied more in detail is

$$\pi = \frac{Q_p \cdot Q_v}{L \cdot G_m}$$

where

- $Q_p$ = charge concentration (g/m)
- $Q_v$ = specific energy in the explosive (Joule/g)
- $L$ = a geometric quantity (for example burden)
- $G_m$ = approximate potential strain energy release rate (J/m$^2$)

The critical burden was determined in blocks, consisting of Öjeby granite and Kiruna magnetite (size 100x300x400 mm) and modelmaterial (size 100x300x300 mm). The hole diameter was 6 mm and charge concentration 10 g/m (detonating cord with PETN). Four different materials were tested, magnetite concrete with 6 or 13 weight-% cement, Kiruna magnetite and Öjeby granite.

Test result

The strength data and critical burdens are shown in table 1.

The calculated critical burden, with help of the $\pi$-factor, is compared with the actual measured versus the approximate potential strain energy release rate $G_m$ in Fig. 5.
It is too big difference between theoretical and actual measured values. One reason could be that the \( m \)-factor doesn't include the density of the material. In part 2 it will be shown that density plays an important role for the critical burden. The difference between the theoretical curve and measured critical burden is too large and therefore the use of this \( m \)-factor to calculate critical burden has to be abandoned.

Part 2. Extensive study

The goal of this part is to more generally see what rock properties are important and can be correlated with the angle of breakage, critical burden and fragmentation. The following strength parameters were determined:

- Uniaxial compressive strength \((\sigma_c)\)
- Tensile strength (Brazilian Test) \((\sigma_t)\)
- Approximate fracture toughness \((K_{IC})\)
- Approximate potential strain energy release rate \((G_m)\) was calculated from \(K_{IC}\)
- Shear strength
  - Cohesion (C)
  - Angle of internal friction
- P-wave velocity \((c_p)\)
  - In the model block
  - In \(\varnothing\ 46\) mm diamond cores
Density
- in the model block tested
- in Ø 46 mm diamond cores

Dynamic elastic modulus
Static elastic modulus
Static Poisson's ratio

A similar extensive study but in crater blasting has been made by USBM, Johnson and Fischer 1963 and correlations have been found between optimal burden and tensile strength as well as compressive strength. No significant correlation has been found between critical depth (burden) and material properties.

At the time our project was planned the following correlations were expected between $k_{50}$ or volume of broken material versus burden see Fig. 6.

![Hypothetical correlations between $k_{50}$ or volume of broken rock versus burden.](image)

Test procedure

To give a broad variation in rock strength 2 artificial materials were included, light concrete and magnetite concrete. The rest of the materials were granite from Öjebyn at Piteå, gabbro from Kallax, 8 km south west of Luleå, Henry quartzite from an open pit belonging to LKAB, 1 km northwest of Luossavaara and finally magnetite from the Research Mine at Luossavaara. These rocks or rocklike materials represent a big variation of rock properties. Representative blocks were collected at the locations and cut into small square blocks with the height ~100 mm, and side length ~300 mm, see Fig. 7.

Boreholes were drilled parallel to the free surface at the midpoint of each side and at increasing burdens. Only one hole was blasted to each side. In Fig. 8 is shown a blast very close to the critical burden in magnetite.
Fig. 7 Geometry of the blocks.

Concrete. Due to corner effects all sides couldn't be used in all materials. Sometimes only two holes in one block could be blasted.

The critical burden tests mentioned in Part 1 have been performed with 10 g/m PETN detonating cord and this charge concentration was too much, compared to the size of the blocks. In the beginning it was therefore planned to use as small charge concentration as possible, that means 1 g/m. One disadvantage with this charge concentration after test blastings was that it gave very small critical burdens for the high strength rocks. The optimal charge concentration was therefore chosen to 3 g/m.

The detonating cord was initiated at the top outside the hole and in the same manner as the earlier tests. The result measurements were increased in comparison to the earlier tests, with sieve analysis for each burden blasted, measurement of angle of breakage, broken volume etc.

Fig. 8 View of a placement of 2 large pieces after blasting. The right side piece has rotated 180 degrees.
Test results

New results were found, but not all in agreement with the hypothesis. The strength tests data are shown in Table 3.

Table 3. Physical properties of rock or rocklike materials. (The strength data are mean values of five repeated tests from each material).

<table>
<thead>
<tr>
<th>Rock or rocklike material</th>
<th>Uniaxial compressive strength</th>
<th>Tensile strength (Brazilian test)</th>
<th>Fracture toughness approximate $K_{IC}$</th>
<th>Pot strain energy release rate, appr $G_m$</th>
<th>Shear strength cohesion</th>
<th>Angle of friction, degrees (coefficient of friction)</th>
<th>P-wave velocity</th>
<th>Density</th>
<th>Dynamic elastic modulus $E_p$ (secant) $E_p$ (tanemial)</th>
<th>$E_p$(initial) $\nu_p$</th>
<th>Static elastic modulus $E_s$ (secant) $E_s$(tanemial)</th>
<th>Static Poisson’s ratio $\nu$</th>
<th>$\nu$ (tanemial)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Light concrete Siporex</td>
<td>3.43</td>
<td>0.604</td>
<td>0.08</td>
<td>3.38</td>
<td>0.85</td>
<td>11.8 (0.21)</td>
<td>2302</td>
<td>490.4</td>
<td>2.54</td>
<td>7.29</td>
<td>1.15</td>
<td>7.05</td>
<td>0.10</td>
</tr>
<tr>
<td>Magnetite concrete</td>
<td>14.4</td>
<td>5.44</td>
<td>0.23</td>
<td>7.78</td>
<td>11.44</td>
<td>8.0 (0.14)</td>
<td>3476</td>
<td>3457</td>
<td>38.2</td>
<td>38.5</td>
<td>6.55</td>
<td>6.83</td>
<td>0.19</td>
</tr>
<tr>
<td>Basaltic magnetite</td>
<td>174</td>
<td>8.73</td>
<td>-</td>
<td>0.72</td>
<td>0.36</td>
<td>14.3 (0.25)</td>
<td>2402</td>
<td>4934</td>
<td>22.8</td>
<td>27.2</td>
<td>77.9</td>
<td>77.3</td>
<td>0.27</td>
</tr>
<tr>
<td>Kieby granite</td>
<td>188</td>
<td>10.5</td>
<td>1.41</td>
<td>36.8</td>
<td>93.09</td>
<td>23.0 (0.25)</td>
<td>4908</td>
<td>2645</td>
<td>59.3</td>
<td>62.0</td>
<td>61.7</td>
<td>61.4</td>
<td>0.14</td>
</tr>
<tr>
<td>Vallax gabbro</td>
<td>279</td>
<td>18.8</td>
<td>-</td>
<td>0.72</td>
<td>0.36</td>
<td>24.6 (0.46)</td>
<td>6688</td>
<td>3052</td>
<td>106.2</td>
<td>110.8</td>
<td>98.9</td>
<td>98.7</td>
<td>0.26</td>
</tr>
<tr>
<td>Hornfels Quartzite</td>
<td>331</td>
<td>21.47</td>
<td>-</td>
<td>2.26</td>
<td>0.33</td>
<td>21.8 (0.40)</td>
<td>5585</td>
<td>2444</td>
<td>74.6</td>
<td>67.9</td>
<td>77.0</td>
<td>76.7</td>
<td>0.26</td>
</tr>
</tbody>
</table>

The middle crack to the free face is caused by bending which can be proofed by several photos of which one is shown in Fig. 8. One of the two pieces has rotated 180°.

Angle of breakage

The angle of breakage is interesting because it determines the volume broken and the real specific charge, which means actual charge used to the real broken volume. The angle of breakage decreases with increasing burden, see Fig. 9. The maximum angle is 156° and the minimum 119°.

If P-wave velocity is known, the angle of breakage can be calculated from the following equation

$$ \theta = 131 + 6.98 \times 10^{-3} c_p + (0.580 - 3 \times 10^{-4} c_p) \times B $$

(11)

$$ \theta = \text{angle of breakage (degrees)} $$
$$ c_p = \text{p-wave velocity (m/s)} $$
$$ B = \text{burden (mm)} $$
$$ c_p = 2300-6700 \text{ m/s.} $$

The broken volume versus burden is shown in Fig. 10 and the equation for calculating volume of broken rock is of the form

$$ V = k_1 B^{k_2} $$

(12)

126
Fig. 9 Angle of breakage versus burden.

where

\[ V = \text{volume (cm}^3\text{)} \]
\[ k_1 = \text{coefficient which varies for different materials 0.45-9.09} \]
\[ k_2 = \text{coefficient which varies for different materials 0.96-2.13} \]
\[ B = \text{burden (mm)} \]

If the angle of breakage is constant for all rocks the volume of broken rock can theoretically be calculated by the equation

\[ V = k \cdot B^2 \quad (13) \]

\[ V = \text{volume} \]
\[ k = \text{constant} \]
\[ B = \text{burden} \]

Because of the fact that the angle of breakage decreases with increasing burdens the increase in volume will be less as compared to equation (13) for all materials tested.

When the blasted mass is studied versus the burden the differences between different rocks is large, because of the big difference in density, see Fig. 11.

The Luossavaara magnetite with density 4934 kg/m\(^3\) gives the largest mass broken at a certain burden and leight concrete with density 490 kg/m\(^3\) gives the lowest mass broken.
Fig. 10 Volume of fragmented material versus burden.

Critical burden versus impedance.

A very good correlation has been found between critical burden and the impedance of the rock. Impedance means the density of the rock multiplied by the P-wave velocity of the rock. The correlation is shown in Fig. 12.

The equation is

\[ B_c = 69.5 - 1.79 \times 10^{-6} \times I \]  

\( B_c \) = critical burden (mm)  
\( I \) = impedance (kg/m²s)  
\( R \) = correlation coefficient = 0.96

The equation is valid for I's between (1-20) \( \times 10^6 \) kg/m²s.

A high impedance means that the blast waves are transmitted very well through the rock and a low impedance the opposite. The impedance has been used very much by USBM, Atchison et al 1964, to describe the efficiency of the explosive in a certain rock. The theory says that the best energy transmission is achieved when the impedance ratio of explosive to rock is equal to 1. The impedance of the explosive is defined as density of explosive multiplied with the detonation velocity.
Fig. 11 Mass of fragmented material versus burden.
Fig. 12 Critical burden versus impedance.

**Critical burden versus tensile strength ($\sigma_t$)**

There is also a non linear correlation between critical burden and static tensile strength, see Fig. 13.

Through regression analysis the following equation has been established

$$B_c = 64.07 - 8.43 \times 10^{-6} \ln \sigma_t$$  \hspace{1cm} (15)

$B_c$ = critical burden (mm)  
$\sigma_t$ = static tensile strength (Brazilian) (MPa)  
$R^2$ = correlation coefficient = -0.95

The equation is valid for $\sigma_t$ between 0.6-21.4 MPa.

**Critical burden versus approximate potential strain energy release rate ($G_m$)**

Critical burden can also be correlated with $G_m$, see Fig. 14.

Through regression analysis the following equation has been established

$$B_c = 73.29 - 8.737 \ln G_m$$  \hspace{1cm} (16)
Fig. 13 Critical burden versus tensile strength.

Fig. 14 Critical burden versus $G_m$, J/m$^2$
The best rock properties to correlate with critical burden is therefore impedance.

For the hard rocks tested the difference in critical burden is not very large \( B_c = 37 \text{ mm} \) for gabbro and \( B_c = 48 \text{ mm} \) for granite. The difference is only 30%.

A good correlation was expected between the critical burden and \( k_{50} \) for different rocks at different burdens because the hypothesis was that critical burden should be an index of blastability. The test result shows that this correlation is fairly good except for Kallax gabbro and Öjeby granite, Fig. 15.

The critical burden is therefore not enough to describe the blastability of rock. The fragmentation distribution has also to be given.

![Fig. 15: \( k_{50} \) versus critical burden for six different rocks or rocklike materials.](image)

---

### Mathematical Formulas

- \( B_c \) = critical burden (mm)
- \( G_m \) = approximate potential strain energy release rate (J/m²)
- \( R \) = correlation coefficient = 0.84
Fragmentation distribution

In Fig. 16, a, b, c, d, e, and f are shown how the fragment size distribution is changed, in a double logarithmic diagram, when the burden is increased in the materials tested. The fragment distribution is getting more and more coarse when the burden is increased but the slope of the curve does not change much. For larger burdens the fragmentation distribution cannot be represented by one regression line. This indicates that there are only a few big boulders and very little fines. The variation in slope of the fragmentation curves in the double logarithmic sieve analyse diagram has been calculated by regression analyze and are shown in Table 4 and in Fig. 17.

Table 4. Fragmentation gradients

<table>
<thead>
<tr>
<th>Rock or rock-like material</th>
<th>Fragmentation gradient</th>
<th>Range</th>
<th>Average</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>n degrees</td>
<td></td>
</tr>
<tr>
<td>Light concrete</td>
<td>0.36-0.54</td>
<td>20-28</td>
<td>0.46</td>
</tr>
<tr>
<td>Magnetite concrete</td>
<td>0.63-0.74</td>
<td>32-37</td>
<td>0.68</td>
</tr>
<tr>
<td>Luossavaara magnetite</td>
<td>0.58-0.75</td>
<td>30-37</td>
<td>0.68</td>
</tr>
<tr>
<td>Ojebry granite</td>
<td>0.52-0.70</td>
<td>27-35</td>
<td>0.61</td>
</tr>
<tr>
<td>Kallax gabbro</td>
<td>0.82-0.96</td>
<td>39-44</td>
<td>0.87</td>
</tr>
<tr>
<td>Henry quartzite</td>
<td>0.71-0.91</td>
<td>35-42</td>
<td>0.83</td>
</tr>
</tbody>
</table>

Fig. 17 Fragmentation gradient versus impedance.
a) Light concrete

b) Magnetite concrete

c) Luossavaara magnetite

d) Öjeby granite

e) Kallax gabbro

f) Henry quartzite

Fig. 16: Size distribution curves of the fragmented material at different burdens.
Through regression analysis the following equation has been established

\[ n = 0.430 e^{3.64 \times 10^{-8} I} \]  

(17)

\[ n = \text{slope of fragmentation curve in a double logarithmic diagram} \]
\[ I = \text{impedance = density of the material times P-wave velocity (kg/m}^2\text{s)} \]
\[ R = \text{correlation coefficient} = 0.96 \]

A high \( n \)-value means that the size range of fragments is smaller. Three groups or families of rock or rocklike materials can be identified and a scale with four different families has been suggested, see Table 5.

Table 5. Grouping of rocks into families because of the different slope of fragment size distribution after blasting.

<table>
<thead>
<tr>
<th>Material</th>
<th>( n )</th>
<th>Impedance ( x 10^6 ) kg/m(^2)s</th>
<th>Family</th>
</tr>
</thead>
<tbody>
<tr>
<td>Light concrete</td>
<td>0.46</td>
<td>0-5</td>
<td>Very low impedance</td>
</tr>
<tr>
<td>No material tested</td>
<td></td>
<td>5-10</td>
<td>Low impedance</td>
</tr>
<tr>
<td>Magnetite concrete</td>
<td>0.68</td>
<td>10-15</td>
<td>High impedance</td>
</tr>
<tr>
<td>Luossavaara magnetite</td>
<td>0.68</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Öjeby granite</td>
<td>0.81</td>
<td>15-20</td>
<td>Very high impedance</td>
</tr>
<tr>
<td>Henry quartzite</td>
<td>0.83</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Kallax gabbro</td>
<td>0.87</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

**\( k_{50} \) versus burden**

The \( k_{50} \) versus burden is shown in Fig. 18.

Only for one rock the curve is separate, namely the light concrete. The other curves crosses each other one or several times. Because of this fact it is difficult to find any good correlation between \( k_{50} \) versus burden and different rock properties. Another astonishing thing is that gabbro and quartzite, with very high compressive strengths, were expected to give the highest \( k_{50} \) but instead granite gives the highest.

**\( k_{50} \) versus specific charge (q)**

\( k_{50} \) versus specific charge for different rocks is shown in Fig. 19.

All curves are separate except for granite, which crosses the curves for magnetite concrete, Kallax gabbro, Henry quartzite and Luossavaara magnetite. Luossavaara magnetite and quartzite gives the highest \( k_{50} \) for large specific charges. The reason for this could be that quartzite has the highest compressive strength of all materials tested and that the density of the luossavaara magnetite is very high. High specific charges are probably the most interesting area for actual field applications.
Fig. 18 $k_{50}$ versus burden

Fig. 19 $k_{50}$ versus specific charge
The following equation has been derived to calculate \( k_{50} \) for a given specific charge in a given material:

\[
k_{50} = \frac{5.18 \times 10^{-3} \times 0.588}{q^{2.14}}
\]  

\( k_{50} \) = the mesh size where 50 weight-% of the broken material passes (mm)  
\( I \) = impedance (kg/m²·s)  
\( q \) = specific charge (kg/m³)

The exponent for \( q \) varies with rock type in the following manner, see Table 6.

Table 6. Exponents and prop constants to be used in equation (18)

<table>
<thead>
<tr>
<th>Material</th>
<th>Exponent for ( q )</th>
<th>Prop constant</th>
<th>Correlation coefficient ( R )</th>
</tr>
</thead>
<tbody>
<tr>
<td>Light concrete</td>
<td>2.31</td>
<td>18.4</td>
<td>-0.97</td>
</tr>
<tr>
<td>Magnetite concrete</td>
<td>1.99</td>
<td>66.1</td>
<td>-0.95</td>
</tr>
<tr>
<td>Luossavaara magnetite</td>
<td>1.65</td>
<td>81.2</td>
<td>-0.90</td>
</tr>
<tr>
<td>Öjeby granite</td>
<td>2.89</td>
<td>106.4</td>
<td>-0.99</td>
</tr>
<tr>
<td>Kallax gabbro</td>
<td>2.30</td>
<td>91.2</td>
<td>-0.97</td>
</tr>
<tr>
<td>Henry quartzite</td>
<td>1.70</td>
<td>76.9</td>
<td>-0.78</td>
</tr>
</tbody>
</table>

\( k_{50} \) versus density of the blasted material

Some correlation has been found between \( k_{50} \) and density at different specific charges, see Fig. 20.

Fig. 20 \( k_{50} \) versus density of blasted material.
Above the density 2500 kg/m$^3$ the $k_{50}$ doesn't increase at low specific charges 1.5 kg/m$^2$.

$k_{50}$ versus cohesion

In Fig. 21 $k_{50}$ versus cohesion is shown at different burdens.

![Graph showing $k_{50}$ versus cohesion for different burdens.](image)

Fig. 21 $k_{50}$ for different burdens versus cohesion.

Rocks with larger cohesion give higher $k_{50}$ at different burdens. The difference in $k_{50}$ for hard rocks is not very large. The larger the burden (low specific charge), the larger is the scatter in $k_{50}$.

Acceptable burdens

Although the total mass of fragmented material increased with increasing burdens, the fragmentation was not satisfactory for higher burdens. From the sieve analysis results, maximum acceptable burdens were determined and have been given in Table 7. Not acceptable burdens are those where the size distribution curve is far from linear in the double logarithmic diagram. The ratios between acceptable and critical burden are also shown in Table 7. This value gives an idea in which range the real used burden could be.
Table 7. Maximum acceptable burdens and ratios between max acceptable burden and critical burden.

<table>
<thead>
<tr>
<th>Rock or rocklike material</th>
<th>Maximum acceptable burden ($B_A$) mm</th>
<th>Critical burden ($B_C$) mm</th>
<th>Ratio $B_A/B_C$ %</th>
</tr>
</thead>
<tbody>
<tr>
<td>Light concrete</td>
<td>45</td>
<td>69</td>
<td>65</td>
</tr>
<tr>
<td>Magnetite concrete</td>
<td>25</td>
<td>50</td>
<td>50</td>
</tr>
<tr>
<td>Luossavaara magnetite</td>
<td>25</td>
<td>41</td>
<td>61</td>
</tr>
<tr>
<td>Öjeby granite</td>
<td>22</td>
<td>48</td>
<td>45</td>
</tr>
<tr>
<td>Kallax gabbro</td>
<td>33</td>
<td>37</td>
<td>89</td>
</tr>
<tr>
<td>Henry quartzite</td>
<td>30</td>
<td>42</td>
<td>71</td>
</tr>
</tbody>
</table>

Weight of boulders as a criteria for blastability

In practical mining the coarse part of the fragmentation distribution is causing most problems and therefore some calculations have been made to determine the ranking of the materials regarding boulder weight, see Table 8.

Table 8. Weight-% boulders at different boulder size limits 10, 20, 30 and 40 mm. The burden is between 20.4-21.8 mm.

<table>
<thead>
<tr>
<th>No</th>
<th>Material</th>
<th>Boulder mesh size</th>
<th>Blastability ranking</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>10 mm 20 mm 30 mm 40 mm</td>
<td>10 20 30 40</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Weight -% boulders</td>
<td>mm mm mm mm mm</td>
</tr>
<tr>
<td>----</td>
<td>-------------------</td>
<td>--------------------</td>
<td>---------------------</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Weight -% boulders</td>
<td>10 20 30 40</td>
</tr>
<tr>
<td></td>
<td></td>
<td>10 mm 20 mm 30 mm 40 mm</td>
<td>10 20 30 40</td>
</tr>
<tr>
<td>1</td>
<td>Light concrete</td>
<td>32 18 5 0</td>
<td>1 2 2 2 1.75</td>
</tr>
<tr>
<td>2</td>
<td>Magnetite concrete</td>
<td>45 10 0 0</td>
<td>2 1 1 1 1.25</td>
</tr>
<tr>
<td>3</td>
<td>Luossavaara magnetite</td>
<td>62 59 47 35</td>
<td>3 5 5 5 4.50</td>
</tr>
<tr>
<td>4</td>
<td>Öjeby granite</td>
<td>78 55 50 42</td>
<td>5 4 6 6 5.25</td>
</tr>
<tr>
<td>5</td>
<td>Kallax gabbro</td>
<td>67 38 5 0</td>
<td>4 3 3 3 3.25</td>
</tr>
<tr>
<td>6</td>
<td>Henry quartzite</td>
<td>80 62 45 30</td>
<td>6 6 4 4 5.00</td>
</tr>
</tbody>
</table>

In Table 8, the mean value of the ranking order have been calculated for all four boulder size limits. A low number means it is more easy to fragment. The ranking list will be as follows.
1. Magnetite concrete  1.25  
2. Light concrete  1.75  
3. Kallax gabbro  3.25  
4. Luossavaara magnetite  4.50  
5. Öjeby granite  5.25  
6. Henry quartzite  5.50  

Increasing boulder weight

This result is a little different from that achieved by looking only on $k_{50}$. The conclusion is therefore that $k_{50}$ alone is not sufficient to express the blastability of rock. If the ranking between rocks is studied for another burden the ranking could of course change.

Despite light concrete has the lowest strength it doesn't give the finest fragmentation. The reason for that is that there is a big crushing zone when blasting in light concrete $\sim 25$ mm. Lot of energy is therefore lost close to the charge and there will be less energy to fragment the outer regions.

CONCLUSIONS

Critical burden

- The angle of breakage decreases with increasing burden for all materials except Luossavaara magnetite. The angle varies between $119^\circ$ to $156^\circ$ and is dependent also on P-wave velocity ($c_p$) of the material.
- A good linear correlation exists between critical burden and impedance (Density of the blasted material multiplied with P-wave velocity). Correlation coefficient $R = 0.96$.
- A non linear correlation exist between critical burden and static tensile strength (Brazilian) $R = 0.95$ or critical burden and approximate potential strain energy release rate. $R = 0.84$.
- The critical burden is an important parameter when describing blastability, because it tells the minimum burden where breakage doesn't occur. For hard rocks the critical burden doesn't vary very much, not more than 30%.
- Maximum acceptable burdens from the point of view of satisfactory fragmentation vary between 45-90% of the critical burden.

Fragmentation (size distribution)

- Fragmentation ($k_{50}$) decreases with the increase in specific charge. For large specific charges $k_{50}$ is more or less constant. In full scale blasts it is therefore very important to know where on the curve the actual blast in the mine is situated.
- Fragmentation ($k_{50}$) versus burden doesn't give clearly separate curves as expected for different materials. The most coarse fragmentation was expected for Henry quartzite with the highest uniaxial compressive strength but instead granite gave the most coarse fragmentation.
- Fragmentation ($k_{50}$) versus critical burden gives a good correlation for all materials tested except Kallax gabbro and Öjeby granite.
- Fragmentation ($k_{50}$) increases with the density of the material.

Blastability

Blastability of rock is a very complex function of many parameters. If the blastability shall be compared between different rocks the following parameters must be equal; burden, specific charge, explosive charging and initiation and finally mesh size for boulders. The blastability of homogene-
ous rock or rocklike material in model blast slabbing can be calculated by the following formulas

\[ k_{50} = k_1 \frac{I^{0.59} \cdot (B \cdot S)^{0.46}}{q^i} \]

Valid only for one hole blasts and if multiple holes are used and single hole blasted a S/B-ratio of 2.

\[ n = 0.43 \cdot e^{3.64 \cdot 10^{-8} \cdot I} \]

Valid of one hole blast.

\[ B_c = 69.5 - 1.79 \cdot 10^{-6} \cdot I \]

\[ \theta = 131 + 6.98 \cdot 10^{-3} c_p + (0.580 - 3 \cdot 10^{-4} c_p) \cdot B \]

where

- \( k_{50} \) = the mesh size where 50 weight %- of the blasted mass is passing (mm)
- \( k_1 \) = constant
- \( I \) = impedance = density of the rock times P-wave velocity (kg/m²'s)
- \( B \) = burden (mm)
- \( S \) = spacing (mm)
- \( q \) = specific charge or powder factor (kg/m³)
- \( i \) = exponent depending on type of rock. Varies between 1.65-2.89. No correlation has been found with rock properties for single hole blasts, when critical burden is tested in different materials. For magnetite concrete the value is 1.99. For multiple hole blasts (one hole blasted each time) in magnetite concrete the exponent is 1.18, the same as the exponent used in the SveDeFo-formula for calculation of fragmentation in full scale blasting.
- \( B_c \) = critical burden (mm)
- \( \theta \) = angle of breakage (degrees)
- \( c_p \) = P-wave velocity (m/s)

The results pronounce the importance of using impedance when calculating fragmentation or blastability.

ACKNOWLEDGEMENTS

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NEW METHOD TO TEST THE ROCK BREAKING PROPERTIES OF EXPLOSIVES

by

Agne Rustan
Shu Lin Nie

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New method to test the rock breaking properties of explosives in full scale

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ABSTRACT: During our research it was found that, still today, the rock breaking properties of an explosive can not be accurately determined by laboratory test or theoretical calculations. The new test method proposed here for bench blasting is based on single hole blasts in model- and full scale with different burdens up to and greater than the critical burden. The tests in full scale should be done with that hole diameter, length of hole, explosive and stemming etc., desired to be used in the real operation. The new method includes determination of the fragment size distribution, the backbreak, the throw, the centre of gravity change and the damage to the remaining rock for each burden tested. This will yield the maximum information possible for understanding how the explosive interacts with the rock. The field tests to examine the single hole blasts were done at Storungs limestone quarry on Gotland, and the model blast tests at Luleå University of Technology. They showed that there exist a good correlation between the model- and full scale tests regarding the angle of breakage versus burden and the relation of kg versus burden, where the function has the same shape, and finally the power for the specific charge is the same both in the model- and full scale. Critical burden and fragmentation gradient are well correlated with the impedance of the rock in the model tests. It seems that the model test can reveal many of the rock breaking properties of an explosive in full scale. The test procedure used for the model and full scale test were therefore proposed as a first idea to an international standard. Based on the systematic investigation carried out at Storungs limestone quarry it was theoretically possible to increase the profit up to 8 million SEK/year by changing burden and spacing and thereby reducing the amount of fines (<25 mm), which is treated as waste in the quarry.

INTRODUCTION

For several years the Division of Mining at Luleå University of Technology, has been involved in research to improve existing empirical formulas to calculate fragmentation prior to blasting. Some of this research was reported by Rustan and Vutukuri (1983). In simple model tests involving single holes the influence of specific charge, burden, geometrical scale and physical properties of rock or rocklike material on fragmentation, volume broken and angle of breakage was studied.

The results were so interesting that we decided to study the same thing in full scale tests. The objective was thereby to see if the same tendencies could be observed. The model test could then be used to make predictions of some of the blast results in full scale.

In the full scale test we also wanted to test the validity of two existing empirical fragmentation formulas. These are the Kuznetsov (1973) formula and the formula used by the Swedish Detonic Research Foundation and Nitro Nobel described by Hjelmberg (1983).

A third aim of the full scale test was to study the breakage mechanism of rock.

The fourth aim was to find a better method to test the blastability of the rock than that used by Langefors or Sassa and Ito (1970). Langefors uses in his formula a rock constant c. According to Fraenkel (1954) it is determined in the following way.

"For practical use the blastability of rock, c (kg/m²), can be determined by test blasting with one single vertical hole with 33 mm bottom diameter, hole depth 1.33 m and with that charge which is needed to give a 1 m high vertical bench and 1 m burden a breakage and throw of maximum 1 m."

The burden used in the test is therefore very close to the critical burden when using the Langefors test method.

The rock constant is normally 0.4 kg/m² and could vary by ±25% according to Larson (1974).

A second method to examine the blastability of rock was introduced by Sassa and Ito (1970). Based upon blastability studies in tunneling conducted in more than 50 different rock types in Norway they proposed a "Resistance to Blasting Field Index" RBFI. They also developed a "Resistance to Blasting Laboratory Index" RBLI by regression analysis of mechanical properties of rock measured in the laboratory and crack frequency studies at the blast site in the field.

A comparison of these two existing methods to determine the blastability of rock with the method proposed in this paper will be done at the end of this paper.

MODEL BASS TEST IN STORUNGS LIMESTONE

Model blast tests were carried out in blocks using the same procedure described by Rustan and Vutukuri in (1985). A 1 m² block from the 3 m burden in the field after blasting was chosen for model tests. The block was cut into several blocks of size 100 x 300 x 300 mm. Single hole blasts with 6 mm blastholes were made in the blocks with one hole at each side of the blocks. Detonating cord with charge concentration 3 g/m and length 100 mm was used as explosive. The burden was increased from very small up to and over the critical burden. The fragmentation from each burden, was sieved and the angle of breakage measured.

FULL SCALE TESTS

The full scale tests were performed at KOPARK AB's limestone quarry Storungs on the Island of Gotland in Sweden during May-June 1986, see fig 1. The age of the bedrock is from Silurian time. It is horizontally layered. The Brazilian tensile strength in the most competent layer ranged between 6.59-9.42 MPa with a mean value 8.25 MPa measured perpendicular to the bedding planes and 5.81-9.84 MPa with mean value 7.88 MPa parallel to the bedding planes.
The boreholes were drilled with an Atlas Copco Rotamec 130 C precision blast hole drill rig, see Figure 2.

The boreholes were drilled with an Atlas Copco Rotamec 130 C precision blast hole drill rig, see Figure 2.

The density was 2650 kg/m³ with a variation between 2650-2880 kg/m³. P-wave velocity parallel to the layers was 6100 m/s (Variation 5800-6400 m/s). Perpendicular to the layers, it was 6200 m/s with a variation of 5800-6600 m/s.

The Impedance of limestone (density x P-wave velocity) was 16.2 x 10⁻⁶ kg/m/s which is very high and the limestone has therefore to be regarded as a hard rock and not a soft rock as we anticipated before starting the test.

The following technical data were used in the full scale test, see Table 1.

Table 1. Technical data for the full scale test at Storugns.

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Bench height:</td>
<td>16.9-17.6 m</td>
</tr>
<tr>
<td>Borehole diameter:</td>
<td>95 mm</td>
</tr>
<tr>
<td>Burdens tested:</td>
<td>1, 2, 3, 4, 5, 6, and 7 m</td>
</tr>
<tr>
<td>Inclination of bench:</td>
<td>90°</td>
</tr>
<tr>
<td>Subdrill:</td>
<td>0 m</td>
</tr>
<tr>
<td>Bottom charge, Dynex:</td>
<td>33 kg</td>
</tr>
<tr>
<td>Cartridges slipped down by gravity: Dynex M</td>
<td></td>
</tr>
<tr>
<td>Column charge ANFO:</td>
<td>75 kg</td>
</tr>
<tr>
<td>Initiation:</td>
<td>Electrical blasting cap in the Dynex at the bottom of the charge.</td>
</tr>
</tbody>
</table>

The boreholes were positioned so the angles of breakage would not overlap one another. In Figure 3, view of the test area is shown.

The holes were completely straight as evidenced by the fact that the hole bottom could be seen from the top of the hole.

All holes were drilled vertical. Water was removed from wet boreholes using a piece of detonating Dynex.

If the full scale test layout should equal the model test layout, only one kind of explosive should have been used because the models did not have a fixed bottom which requires stronger explosive. In layered rocks usually the bottom of the bench uses an existing bedding plane so no bottom charge should be necessary. The field test procedure which is going to be recommended as an International Standard has to be used in other types of rock as well where it is necessary to use a strong bottom charge. Therefore both bottom and column charges were used in the field test. In fact we are testing two explosives in combination but the main interest is directed to the main charge, ANFO.

The detonation velocity of the ANFO was determined by Nitro Nobel, with sensors distributed along a thin rope.

The dynamic breakage of rock, directly in front of the borehole was recorded by a Sankeo Super 8 camera (18 frames/s) and from the side by a Paillard-Bolex H 16 camera with 64 frames/s.

The maximum fly rock range was observed visually. After blasting the muck pile was formed into a rectangle by an LHD and photographed from a skylift (Maximum height 17 m). Each photo covers an area of about 4x5 m. Also sections of the rock pile was photographed and compared with the fragmentation of the top surface. Some of the rock material was loaded and moved into a new rock pile and photographed at the top surface, to examine if that would change the fragmentation distribution. More details about the fragmentation measurement are given in the paper presented at this symposium by Nie Shui Lin and Rustan (1987).

Maximum throw of the main part of the fragments was measured together with maximum height and length of the rock pile. The diameter of the crater at the top of the hole and crater depth has also been measured parallel and perpendicular to the front and also the depth of the crater. All new cracks at the top of the bench were measured regarding length and direction. This was easy to do because the bench had been completely cleaned.

Damage to the remaining rock

Damage to the remaining rock was determined by geophones located at distances of about 10 m, 30 m and 105 m behind the blastholes, see Figure 3. The maximum frequency that could be measured accurately with the geophones at 10 m distance was 1,600 Hz and at 30 and 105 m distance, 500 p/s. Expected fre-
Figure 3. General view of the test site with the blastholes and geophone positions. Typical joint directions before blast are shown. Full scale tests at Nordkalk AB’s limestone quarry at Storugns, Gotland 1986.

Figure 4. The measured angle of breakage in model- and full scale blast tests in Storugns limestone, Gotland, 1986.

The formulas for the model and full scale test are follows:

For model blast

\[ \theta = 156.2 - 0.333 \times B \quad R = -0.64 \]  

For full scale blast

\[ \theta = 165.7 - 6.88 \times B \quad R = -0.93 \]

This formula is only valid for full scale blasts from 0 up to critical burden

Critical burden versus impedance

In Figure 5 the judged critical burden for Storugns limestone is compared with the critical burden for other materials.

Figure 5. Critical burden at model blasting with 6 mm diameter blastholes in different rock types and rocklike materials according to Rustan and Vutukuri (1983) and Rustan and Nie (1987).
The correlation between critical burden and impedance can be written as

$$B_c = 68.5 - 1.69 \times 10^6 \times I, \quad R = 0.94 \quad (3)$$

$$B_p = \text{critical burden (mm)}$$

$$I = \text{impedance (kg/m s)}$$

$$R = \text{correlation coefficient}$$

This formula is valid only for model blasts

**Size distribution versus burden**

The size distributions after blasting with different burdens in Gotland Limestone are shown in Figure 6.

The fragment distributions for each burden are represented by regression lines. Burdens of 15.0, 19.7 and 25.2 mm show a change of gradient in the weight-% versus mesh size distribution curve at the 90 weight-% value.

The fragmentation gradient $n$ (Rosin-Rammler) is increasing with the burden. Values of $n$ have been found between 0.93-1.18 with a mean value of about 1.10, see Table 2.

The $n$-values achieved in Storugns limestone are compared in Table 3 with other rock and rocklike material which have been tested by Rustan-Vutukuri (1983).

The correlation between the fragmentation gradient and the impedance is shown in Figure 7.

![Figure 6. Size distribution curves from model blast test of single holes at different burdens in Storugns limestone.](image)

**Table 2. Fragmentation gradient $n$ in the Rosin-Rammler size distribution formula determined by the model blast tests in Storugns limestone.**

<table>
<thead>
<tr>
<th>Burden (mm)</th>
<th>Fragmentation gradient $n$</th>
</tr>
</thead>
<tbody>
<tr>
<td>15.0</td>
<td>0.93</td>
</tr>
<tr>
<td>19.7</td>
<td>1.10</td>
</tr>
<tr>
<td>25.2</td>
<td>1.14</td>
</tr>
<tr>
<td>30.2</td>
<td>1.12</td>
</tr>
<tr>
<td>34.8</td>
<td>Almost constant</td>
</tr>
<tr>
<td>Mean</td>
<td>1.10</td>
</tr>
</tbody>
</table>

A regression analysis gives the following formula to calculate fragmentation gradient for different rock and rocklike materials at model blasting

$$n = 0.54e^{33 \times 10^{-9} \times I} \quad R = 0.91 \quad (4)$$

This formula is only valid for model blast tests and gives the mean value for a certain material. The range of variation can be seen for each material in Table 3.

**Table 3. Fragmentation gradients for burdens with acceptable fragmentation (the distribution is almost linear in the double logarithmic sieve analysis diagram). Rustan and Vutukuri (1983) and Rustan and Nie (1987).**

<table>
<thead>
<tr>
<th>Rock or rocklike material</th>
<th>Impedance $x 10^9$</th>
<th>Fragmentation gradient $n$</th>
<th>Range</th>
<th>Mean value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Light concrete</td>
<td>1.13</td>
<td>0.52-0.56</td>
<td>0.34</td>
<td></td>
</tr>
<tr>
<td>Luossavaara magnetite</td>
<td>11.9</td>
<td>0.76-0.87</td>
<td>0.80</td>
<td></td>
</tr>
<tr>
<td>Magnetite concrete</td>
<td>12.0</td>
<td>0.81-0.92</td>
<td>0.87</td>
<td></td>
</tr>
<tr>
<td>Ojeby granite</td>
<td>13.0</td>
<td>0.76-0.84</td>
<td>0.80</td>
<td></td>
</tr>
<tr>
<td>Henry quartzite</td>
<td>14.8</td>
<td>0.81-0.93</td>
<td>0.88</td>
<td></td>
</tr>
<tr>
<td>Storugns limestone</td>
<td>16.2</td>
<td>0.83-1.18</td>
<td>1.10</td>
<td></td>
</tr>
<tr>
<td>Kallax gabbro</td>
<td>20.4</td>
<td>5.00-1.05</td>
<td>0.94</td>
<td></td>
</tr>
</tbody>
</table>

According to Figure 7 the Storugns limestone has a much higher $n$-value than expected. The reason is judged to be that the material is very heterogeneous. It was also difficult to find the critical burden in the model test, because long cracks always developed to the back side of the model. The size of the model seems to be too little for model blast tests.

![Figure 7. The slope of the fragment size distribution (fragmentation gradient) versus the impedance for different rocks and rocklike materials in model blasts. Rustan and Vutukuri (1983) and Rustan and Nie (1987).](image)

The scales of X and Y-axes have been chosen so that the inflection point for the two $k_p$-curves coincide. This means that a different scale factor for length has to be used to describe burden ($1:83$) and $k_p$ ($1:83$). The agreement between the two curves in Figure 8 are quite good.

The $k_p$-curve from the model blast test is however always lower than the $k_p$ curve from the full scale blast. The form of the curve is the same for the model and full scale and this means that such information regarding the blastability of rock might be possible to achieve by model blast tests.

The following formulas have been derived for model and full scale blasts.

The variation of $k_p$ values for the model blast tests with the burden are shown in Figure 8 and compared with the $k_p$ values from the full scale test at Storugns.
According to Chiapetta 1983, two parameters which affect the throw of fragments are charge concentration in the borehole and the burden. In our case the charge concentration was constant, but the burden (specific charge) was varied from 1 to 4.2 m. The maximum length of throw was of course achieved at 1 m burden. No breakage was observed for the 5 m and 7 m burdens. With the 6 m burden one side moved 1 m, but the main prism in front was only cracked by a vertical crack in the middle.

Lowering of gravity point

The central portion of the rock piles after blasting are shown in Figure 9.

In the same figure the centre of gravity point in the middle section of the rock pile has been marked by a black dot. Larger burdens produce a greater height of the gravity point.

A calculation was done to see how much energy per kg rock broken is generated due to the lowering of the gravity point during blasting. This is compared with the explosive energy per kg broken rock in Table 5.

At 4.2 m burden the gravity energy is about 25% of the explosive energy. This is a significant and it contributes probably to a considerable amount of the breakage of rock, despite this energy is generated during a much longer time than the explosive energy, 3.6 sec respectively 4 ms.

Fragmentation versus burden

The fragmentation from the 1 m burden was difficult to collect because it spread over a large area of the quarry. The size distributions from the four

---

**Figure 8.** Comparison of $k_{50}$ versus burden for model- and full scale blasts in Storugns limestone.

Model $k_{50} = 4.1 \times 10^{-6} \times B^{2.7}$ $R = 0.99 \ (5)$
Full scale $k_{50} = 0.047 \times B^{1.6}$ $R = 0.92 \ (6)$
Model $k_{50} = 0.025 \times q^{1.1}$ $R = 0.97 \ (7)$
Full scale $k_{50} = 0.052 \times q^{1.1}$ $R = 0.95 \ (8)$

$k_{50}$ = mesh size where 50 weight-% of the material is passing

$B$ = burden (m)
$q$ = specific charge (kg/m$^3$)

The exponent for specific charge is the same in both model and full scale. It is therefore interesting to see if this will be valid for all rock types by full scale tests in other rock types.

The exponent for burden is not the same in model and full scale blasts. One reason for this is that the angle of breakage is smaller in full scale compared to model blasts.

**RESULTS FROM THE FIELD TESTS**

The results from the field test are very encouraging. We have got a much better understanding of the breakage mechanism and some previous knowledge from other field tests has been confirmed.

**Table 4.** Throw from measurement of throw are shown in Table 4.

**Table 4.** Throw from full scale single hole blasts in Storugns limestone quarry on Gotland, 1986.

<table>
<thead>
<tr>
<th>Burden (m)</th>
<th>Throw (m)</th>
<th>Height of rock pile (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1.0</td>
<td>55</td>
<td>0.8</td>
</tr>
<tr>
<td>2.1</td>
<td>32</td>
<td>2.0</td>
</tr>
<tr>
<td>3.0</td>
<td>30</td>
<td>3.5</td>
</tr>
<tr>
<td>4.2</td>
<td>18</td>
<td>10.0</td>
</tr>
</tbody>
</table>

**Figure 9.** The centre profile of the muck pile after blasting showing the centre of gravity point for each burden. 95 mm single hole blasts at Storugns limestone quarry on Gotland, 1986.

<table>
<thead>
<tr>
<th>Burden (m)</th>
<th>Explosive energy per kg rock broken (kcal/kg)</th>
<th>Energy generated by centre of gravity change (kcal/kg)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1.0</td>
<td>0.432</td>
<td>0.019</td>
</tr>
<tr>
<td>2.1</td>
<td>0.123</td>
<td>0.019</td>
</tr>
<tr>
<td>3.0</td>
<td>0.095</td>
<td>0.018</td>
</tr>
<tr>
<td>4.2</td>
<td>0.049</td>
<td>0.013</td>
</tr>
</tbody>
</table>
196

burdens where the burden broke (1.0–4.2 m) are shown in Figure 10.

![Figure 10. The influence from size of burden on fragmentation at bench blasting with single holes (95 mm) at Storugns limestone quarry on Gotland, 1986.]

By the naked eye it was possible to see that the fragmentation size increased considerably with the burden, because the change of specific charge is large from 0.13 kg/m³ to 1.22 kg/m³.

The distributions in Figure 10 are not linear in the double logarithmic size distribution diagram. All distributions show a change of inclination at 90 weight-% material passing. This is in good agreement with the result from the model blasts shown in Figure 6. The gradient of the fragmentation distribution (n) is increased when the burden is increased in full scale, see Table 6.

Table 6. Rosin-Rammler fragmentation gradient at different burdens in full scale tests at Storugns limestone quarry on Gotland, 1986.

<table>
<thead>
<tr>
<th>Burden (m)</th>
<th>Fragmentation gradient (n)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1.0</td>
<td>0.65</td>
</tr>
<tr>
<td>2.1</td>
<td>0.30</td>
</tr>
<tr>
<td>3.0</td>
<td>0.49 (Almost constant)</td>
</tr>
<tr>
<td>4.2</td>
<td>1.11</td>
</tr>
</tbody>
</table>

For the burdens most used in practical rock blasting, the fragmentation gradient seems to be almost constant, n = 0.90.

The mean value for the model blast test with burdens between 19.7–34.8 mm was 1.15. The relation between n values in the full scale (nf) and the model (nm) is therefore:

\[
\frac{n_f}{n_m} = \frac{0.90}{1.15} = 0.78
\]

For the mean value of 0.90 there would be a safety margin so that the anticipated fines could be reduced by 4 weight-%, making a net income of 4 million SEK/year. Now we realize that the systematic well controlled field tests could be of large economic value for the companies.

The formula to calculate the Rosin-Rammler fragmentation gradient from model blast tests applies also very good to the full scale tests, therefore the formula above can also be written as:

\[
I_f = \frac{I_m}{n_f} = \frac{4907 \cdot 2660}{6160 \cdot 2660} = 0.80
\]

A large gradient is an advantage in the Storugns quarry because there is little demand for limestone with particle size less than 25 mm. The amount of fines (less than 25 mm) was determined through weighing with continuous measuring conveyors after the primary crusher. These values were compared with the extrapolation in Figure 10. The result is shown in Figure 11.

The two curves are nicely parallel to each other and of course the amount of fines is larger after the primary crusher, because loading, transporting on trucks and crushing has introduced more fines than can be measured after blasting. The normal amount of fines after crushing in Storugns production blasting is 32 weight-% and by lowering the specific charge we now know that the amount of fines can be reduced to 24 weight-%. Each decrease of 1 weight-% would result in increased amount of income for the quarry of 1 million SEK/year. A maximum of 8 million SEK could therefore be saved if the specific charge could be reduced close to that value which gives the critical burden.

For practical use there must be a safety margin, so the anticipated fines could be reduced by 4 weight-%, making a net income of 4 million SEK/year. Now we realize that the systematic well controlled field tests could be of large economic value for the companies.

A comparison has been made of the forecast of the k50 values determined from photos and the calculated k50 values according to Kuznetsova and Swedish Detrain Research Foundation (SveDeFo) formulas.

\[
k_{50} = \frac{0.45}{0.78} = 0.58
\]

\[
k_{50} = \left(\frac{1}{1.15}\right)^{-0.6} = 0.82
\]

\[
k_{50} = \left(\frac{1}{1.15}\right)^{-0.5} = 0.78
\]

\[
k_{50} = \left(\frac{1}{1.15}\right)^{-0.6} = 0.82
\]
A = Rock factor = 7 for medium strength rock
    = 10 for hard highly fractured rock
    = 13 for hard slightly fractured rock

q = specific charge (kg/m³)
Q = total charge weight per blasthole expressed in kg troltyl (TNT). If the weight strength of ANFO is denoted 1, TNT will have the strength 1.15. Usually the amount of explosive in the subdrilled section in bench blasting is not included, because this part affects the fragmentation in the column area very little

s = structure factor
s = 0.60 Highly fractured rock
    0.55 Fractured rock
    0.50 Lightly fractured rock
    0.45 Almost homogeneous rock
    0.40 Homogeneous rock

h₀ = uncharged part of the blasthole (m)

H = depth of blasthole (m)

B = burden (m)

S = hole distance (m)

c = rock constant according to Langesfors definition. Amount of Dynamex M (kg/m³),

The result of the comparison of the k₅₀ values are shown in Table 7.

Table 7. Forecast of k₅₀ at full scale blasts of single holes at Storungs limestone quarry on Gotland, 1986.

<table>
<thead>
<tr>
<th>Burden (m)</th>
<th>K₅₀ (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1.0</td>
<td>440</td>
</tr>
<tr>
<td>1.1</td>
<td>770</td>
</tr>
<tr>
<td>3.0</td>
<td>1000</td>
</tr>
<tr>
<td>4.2</td>
<td>1310</td>
</tr>
</tbody>
</table>

From Table 7 we observe that the forecast of k₅₀ is not very good, mainly dependent on the fact that the formulas are established for multiple hole blasts and not single hole blasts. Another fact was, that the angle of breakage was not known when the forecast was undertaken. The calculation afterwards has been done with the actual angle of breakage observed after blasting.

The conclusion is that the two empirical formulas can not be used to predict k₅₀ in single hole blasting.

Angle of breakage versus burden

The angle of breakage was determined after the full scale blast. The result has already been shown in Figure 4. The angle is decreasing with increased burden. This result is the same as found in six other materials by Rustan and Vutukuri 1983 and with the model tests made in Storungs limestone. The only difference is that the angle of breakage is about 9° less in full scale. If more full scale tests are done in other rock types this decrease could perhaps also be verified.

Crater shape, radial cracks and fragmentation at the top of the bench

The top of the bench is broken like a crater blast, especially at large burdens. More of the explosive energy is directed vertically than horizontally. The fragmentation was difficult to collect at small burdens (1-4 m) and therefore no size distribution has been determined for the crater rock. Visually it is easy to see that the fragments are small compared to the fragments at the bench floor. The crater radius and crater depth and its dependence on burden are shown in Figure 12.

![Figure 12. Crater dimensions at the top of the bench when blasting single holes (95 mm) in Storungs limestone quarry on Gotland, 1986.](image)

With one exception, the 4.2 m burden, where the crater was very small all these measures are increasing for burden up to 5 m. This burden was very close to the critical burden for the limestone. At large burdens (6 and 7 m) the crater is so large that the broken rock does not have enough velocity or energy to leave the crater. For these burdens the crack system consisting of radial and circular bending cracks can be studied, see Figure 13.

![Figure 13. Schematic view of radial and circular bending cracks at the top of the bench at single hole blasting (95 mm) in Storungs limestone quarry on Gotland, 1986.](image)

The circular and radial bending cracks created at the top surface are very important for the fragmentation. In the existing numerical models this breakage mechanism is not included, because these cracks are first developed when the rock surface starts to deform. Therefore the existing numerical models have to be extended with this type of breakage mechanism.

Bending cracks at the bench front

The front of the bench is cracked mainly be vertical and horizontal bending cracks. The vertical crack just in front of the borehole is a bending crack.
starting at the surface and moving inwards towards the borehole. At both sides of the central crack two thin vertical cracks have been observed on each side. These two cracks are also bending cracks, see Figure 14.

Figure 14. Schematic view of the horizontal and vertical bending cracks found in the front of the bench after single hole blasting in Storugns limestone (95 mm hole diameter) on Gotland, 1986.

The middle crack was observed (first by Rustan and Vutukuri 1983) in model tests. Ash (1973) has pointed out generally the importance of the bending cracks in his doctoral thesis. The vertical bending cracks were easy to observe for those burdens which did not break. These cracks for 5, 6 and 7 m burden can also be seen on the high speed film. Photos from burden 5 and 7 m after blasting are shown in Figure 15.

The width of the middle crack was observed to decrease with increased burden.

The reason for the vertical middle crack being the widest is that if the quasi-static pressure in the borehole is regarded as a pressure vessel with the smallest wall thickness equal to the burden the material movement will start just in front of the borehole. The movements and the velocities further away to the sides will be less because the burden is larger.

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Backbreak and vibrations behind the blastholes

Backbreak is difficult to measure and quantify and therefore only visual observation was done after blasting. The result is shown in Figure 16.

Backbreak is defined here as the maximum outflow depth of rock behind the two side cracks. According to Figure 16 a linear relationship exists between backbreak and burden.

Vertical particle velocity

The vibrations measured with geophones showed that the initial vertical particle velocity amplitude for the P-wave was the same, at the same distance from the borehole, independent of the size of the burden. However, the largest particle velocity for the whole vibration packet increased with the burden, see Figure 17.

If a log-log diagram with maximum vibration velocity (mm/s) versus distance is used to describe the attenuation of blast waves, the measure points should be on almost a straight line. This is not the case for the tests at Storugns. The measured vibration velocities at 10 m distance from the blasthole are too high. The reason for this is probably a vertical heave at the top surface influencing the measured vibration. Holmberg (1981) for example, measured a heave of 230 mm of the top surface, 8 m behind a 191 mm blasthole.

Figure 15. The middle bending crack in the bench front after blasting of single holes (95 mm) in Storugns limestone on Gotland, 1986. Both burdens are larger than the critical burden.

Figure 16. Estimated max amplitude of backbreak behind the two dominating side cracks to the bench front. Single hole blasts (95 mm) in Storugns limestone quarry on Gotland, 1986.

The explanation why a blasthole with larger burden gives a larger vibration amplitude, 10 m behind the blasthole could be the following.
When the burden does not break, the quasi-static pressure will work during a longer time in the blasthole than if the burden is released. This would give more time for gas to penetrate into surface cracks and cause a heave of the top surface.

For smaller burden much of the explosive energy is used for cracking and movement of the rock. In the case of burdens greater than the critical, little energy is used for rock movement and cracking and more energy can be used in form of vibrations and quasi-static borehole pressure and relaxation of rock when the cracking starts.

In Figure 18 the maximal vertical particle vibration velocity (mm/s) is shown for the second and third measurement points.

At 30 m distance from the blasthole the vibration is almost the same for all burdens except burden 2.1 m, which has an amplitude of 30 mm/s instead of 20 mm/s. At 105 m distance the vibrations are very small 3 to 6.5 mm/s and it is difficult to see any correlation with the burden. Large burdens do not necessarily give larger vibrations at large distance.

Crack directions in the remaining rock

The typical crack directions in the remaining rock after blasting were found in the same directions as observed joint direction before blasting. This is a proof for that the weakest parts of the rock breaks first and not the homogenous parts. We were astonished over the very long and wide cracks at the top surface, 20-30 m long and 20-30 cm wide, even at the 7 m burden.

Comparison of the new method to determine the blastability of explosives in rock with old methods

As mentioned in the background, Langefors and Ito-Sassa have developed methods to determine the blastability of rock. We will now compare the new method with these previous ones, see Table 8.

From Table 6 we can see that the maximum information about the blastability of rock is given by Rustans method, because fragmentation, angle of breakage, throw and damage to the remaining rock is studied at the same time.

Proposal for a standard method to test the blastability of explosives in rock

For a complete description of the blastability of rock, the following information is necessary.

1) Critical burden. This is a very important measure for those who are constructing drill- and blast-plans. If the critical burden is exceeded there will be a misfire. The blast must work even if one hole misfires.

2) Fragmentation versus burden or specific charge. This information is important for how size distribution changes with the burden. To describe a size distribution, at least three points on the curve have to be known, if the distribution is not linear in the double logarithmic size distribution diagram. For International Standard two of the following values are proposed $k_1, k_2$, weight-% less than 50 mm if the distribution is linear. If the distribution is not linear like in our model and full scale tests in Storugns limestone quarry, one point must be selected.

Table 8. Comparison of methods to test the blastability of rock.

<table>
<thead>
<tr>
<th>No.</th>
<th>Method to test</th>
<th>Rock type</th>
<th>Specific charge</th>
<th>Simulation</th>
<th>Result parameter</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Langefors rock constant (m-elastic)</td>
<td>Rock mass</td>
<td>Low</td>
<td>No</td>
<td>Specific charge</td>
</tr>
<tr>
<td>2</td>
<td>Sassa and Ito (RBI)/index</td>
<td>Rock mass</td>
<td>High</td>
<td>Yes</td>
<td>Advance per round</td>
</tr>
<tr>
<td>3</td>
<td>Rustan and Møll</td>
<td>Rock type (model blast)</td>
<td>Rock mass (full scale)</td>
<td>Yes</td>
<td>Size distribution of fragments and vibrations when the burden is exceeded.</td>
</tr>
</tbody>
</table>
where there is a change of gradient in the size distribution.

3) Angle of breakage versus burden. This information is important if the total fragmentation from a multiple hole blast is going to be calculated as the sum of what each hole can break and fragment.

4) The impedance in the model and full scale test is determined. Some parameters of the rock like fragmentation gradient and critical burden are dependent on the impedance. The static tensile strength is not necessary to determine but is good to have if the material is going to be compared with other materials.

5) The structure of the rock is determined by line-mapping of the top surface or if this is not possible by diamond drilling. Detailed information must be given in the form of:

- Number of weakness systems
- Distance between weaknesses in their individual systems
- Filling materials in the weakness zones and their strengths
- Amount of water in the weakness planes
- Shear strength for different kinds of weaknesses

6) Maximum throw for single pieces and maximum throw for the main mass of the blasted rock is determined.

7) Damage to the remaining rock is determined by measurement of backbreak and vibration behind the blasthole.

With these seven steps complete information is given about the rock blasting properties.

Conclusions

A comparison between full scale and model blast tests shows that a good correlation exists between angle of breakage versus burden and $k_\text{exp}$ versus burden.

The fragmentation distribution is not linear for different burdens. Both in the model and full scale blast there is a change of gradient at about 90 weight-% passing. This can be interpreted in our case that there are too many large blocks.

With the photographic method (each photo covering 24 m$^2$) sizes less than 50 mm cannot be determined and therefore 50 mm is proposed as the lowest point to give, when the size distribution is defined by three points. If the amount of fines less than 50 mm has to be determined after photographic interactive image analysis, the only way is to extrapolate the curve.

The scale ratio for critical burden between model and full scale blast tests was about 1:100, roughly the same as the burden ratio 1:83.

The results are very encouraging for the development of accurate empirical formulas to forecast fragmentation. Now full scale tests have to be done with the same procedure in different rock types all over the world.

Acknowledgement

Our best thanks are directed to Nordstal AB to let us use their quarry for these tests and let us publish the research result. The financing of the project was made by the Swedish Board for Technical Development, the Chinese Government, Luleå University of Technology, Nitro Nobel AB and Atlas Copco.

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BURDEN, SPACING AND BOREHOLE DIAMETER AT ROCK BLASTING

by

Agne Rustan

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ABSTRACT: Most text books on rock blasting claim that the relationship between burden and borehole diameter is linear. The statistical calculation presented here with real values from one hundred open pit and underground mines indicates that the relationship follows a power function. For underground mines the burdens are consistently lower than those for open pit mines because of higher ore densities, greater confinement in blasting, and finally greater demand for well fragmented rock. The results shown in this paper are recommended to be considered in basic teaching of rock blasting technique. The formulas can be used for a first rough estimation of practical burden and spacing. The formulas do not however give any information about the fragmentation. For that purpose it is necessary to use other formulas not described in this paper.

1 INTRODUCTION

The research at Luleå University of Technology has been directed towards methods for controlled contour blasting and algorithms to calculate fragmentation in production blasting.

Some of the empirical formulas *) which may be used to calculate the maximum burden were reported by Rustan (1987) but it was found that these formulas can not be used for a large range of blasthole diameters, 50-400 mm. The reason for this is that the formulas were not checked over the whole borehole diameter range mentioned above.

This paper describes formulas to calculate the charge quantity and the maximal or practical burden according to a literature review.

Later in the paper a statistical analysis is presented where the objective was to find the relation between practical burden (with an anticipated burden to spacing ratio of 1) and borehole diameter using data from one hundred underground and open pit mines.

The result showed that the relation follows a power function and it is not linear as most formulas showed according to the literature review. The relationship between burden and borehole diameter is also different for underground and open pit mines.

2 LITERATURE REVIEW INCLUDING CRITICAL ANALYSIS

The literature was searched for formulas where practical burden and borehole diameter are included. Also formulas including total charge and burden were studied because these formulas can be rewritten into formulas which include burden and borehole diameter.

In the beginning of the literature review it was found that there are no formulas available to determine the practical burden directly. Existing formulas calculates the maximum burden and the practical burden can thereafter be calculated after correction for the borehole deviations.

Usually borehole deviations are not well known and they have therefore to be anticipated. The influence of borehole deviations on fragmentation is badly known, and to perform the corrections for borehole deviations during the calculation of the practical burden introduces many errors.

Fragmentation formulas will not be reported because they can first be used when the burden is known.

The formulas presented here will only be useful for bench blasting geometries. The terms burden and spacing used in this paper refers to the start position of the boreholes. For mines with sun feather drilling the spacing refers to the spacing at the borehole bottom.

2.1 Charge calculations formulas

The selection of the proper burden in blasting depends on about 30 different parameters, see Table 1.

*) Formula stands throughout this paper for an empirical equality which does not mean a perfect equality like a mathematical equation.

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This paper was originally presented at the Third International Symposium on Rock Fragmentation by Blasting, held Aug 26-31, 1990 in Brisbane, Queensland Australia. Permission to publish it in this journal was given by the Australian Institute of Mining and Metallurgy, Parkville, Victoria, Australia. Some editorial correction to the original paper has been undertaken.
Tabel 1 Parameters influencing the selection of the optimal burden at blasting.

**GIVEN PARAMETERS**

- **Rock sample**
  - Dynamic compressive strength
  - Dynamic tensile strength
  - Youngs modulus
  - Fracture toughness
- **Rock mass**
  - Impedance
  - Number of joint systems
  - Direction of joint systems
  - Joint frequency
  - Joint filling material
  - Faults or crushing zones. direction and weathering.

**SELECTED PARAMETERS**

- **Blast geometry**
  - Confinement
  - Number and orientation of free surfaces
  - Orientation of the blasthole axis to the free surfaces
  - Blasted burden
  - Blasted spacing
  - Bench height
  - Number of rows per blast
  - Charged length of blasthole

- **Explosive**
  - Charge diameter
  - Density of explosive
  - Detonation heat
  - Detonation velocity (Depends partly on borehole diameter)
  - Decoupling ratio
  - Specific charge
  - Impedance ratio of explosive to rock

- **Initiation**
  - Delay time
  - Scatter of delay in the detonators
  - Number of delays used, and the use of deck charges initiated on different delays
  - Sequence of initiation

**RESULT PARAMETERS**

- **Fragmentation**
- **Throw**
- **Efficacy**

The most important parameter is the borehole diameter. (We anticipate that the decoupling ratio of the charge is 1 throughout this paper).

The next most important parameter is the texture and structure of the rock mass.

When the borehole diameter is known the charge can be calculated according to the following equation.

\[
Q = \pi \left( \frac{d}{2} \right)^2 \left( l_b \rho_{eb} + l_c \rho_{ec} \right) \quad \text{or} \\
Q = f(d^2)
\]

(1)

\[
Q = k_2 l^2 + k_3 l^3
\]

(2)

\[ L = \text{a linear dimension at blasting. For example the burden } B \]
\[ k_2 \text{ and } k_3 = \text{proportional constants} \]

Belidors formula can be seen as the first step to find a relation between total charge and burden according to the principles of Taylor series expansion. This implies that the unknown parameter, the burden, is written as a series of burdens with increasing integer exponents. According to Belidor only two terms in the series should be used.

If formulas (1) and (2) are combined we get the following formula

\[
d^2 = f(k_2 B^2 + k_3 B^3)
\]

(3)

If \( k_3 = 0 \) the relation between borehole diameter and burden is linear.

Langefors and Kihlström (1978) made a similar Taylor series development like Belidor,

\[
Q = k_0 + k_1 B + k_2 B^2 + k_3 B^3 + k_4 B^4
\]

(4)

Three more terms were used. The value of the constants for Swedish hard rock are

\[
k_0 = 0 \quad k_1 = 0.010 \quad k_2 = 0.400 \quad k_3 = 0.004 \quad \text{and finally } k_4 = 0 \]

for \( i \geq 4 \).

The values for the different terms at \( B = 1 \) respectively 10 m will be the following

\[
B = 1 \text{ m} \quad B = 10 \text{ m}
\]

\[
k_1 = 0.01 \quad 0.1 \quad k_2 = 0.40 \quad 4.0 \quad k_3 = 0.004 \quad 0.04
\]

In both cases \( k_2 \) is the dominating term and the error is small if \( k_1 \) and \( k_3 \) are set to zero. If equation (1) and formula (4) are combined and all \( k \)'s are zero except \( k_2 = 0.4 \) the relationship between borehole diameter and burden is linear

\[
B^2 = f(d^2) \quad \text{or} \quad B = f(d) \quad \text{for} \quad B = 1-10 \text{ m}
\]

(5)

The powder factor is defined as charge quantity per volume unit or mass unit. The variation of powder factor with burden can be illustrated if formula (4) is divided by the volume \( V \). The relation is shown in Figure 1.

In Fig 1 it is shown that the powder factor is almost constant for practical burdens (\( B_p \)) usually used in blasting, namely \( B_p = 1-10 \text{ m} \). For small scale blasting and very large scale blasting the powder factor increases. The
reason for this, (Langefors-Kihlström, 1978) is that the specific surface area is increased at small burdens. At very large burdens the powder factor increases because the volume increases rapidly and more and more energy is needed for the throw of the fragmented material.

3 EMPIRICAL FORMULAS TO CALCULATE THE MAXIMUM BURDEN RELATED TO THE BOREHOLE DIAMETER

First the formulas presented in basic literature on rock blasting will be presented. All these formulas illustrate a linear relation between burden and borehole diameter. Later on a new formula indicating a non linear relationship will be shown.

3.1 Linear formulas

Langefors and Kihlström (1978) have developed an empirical formula to calculate the maximum burden. The constants $k_0$, $k_1$, $k_3$, $k_4$ were set to 0 in formula (4) and a correction factor for explosive properties, blastability and geometric blast parameters were added.

$$B_m = 0.958 \, d \sqrt{\frac{p_e \, s}{c_0 \, f \, (S_d/B_d)}} \tag{6}$$

The formula is only valid for borehole diameters in the range of 0.03 - 0.089 m according to personal communications with Kihlström (1989).

- $B_m$ = maximum burden for good breakage (m)
- $d$ = borehole diameter (m)
- $p_e$ = density of explosive in the borehole (kg/m$^3$)
- $s$ = weight strength of the explosive
- $c_0$ = corrected blastability factor (kg/m$^3$)
- $c_0 = c + 0.07/B$ for $B \leq 1.4$ m
- $c$ = blastability factor (kg/m$^3$) (Necessary powder factor to break but not throw the rock).
- $f$ = confinement of the borehole
- $S_d$ = drilled spacing (m)
- $B_d$ = drilled burden (m)

The relation between burden and borehole diameter in formula (6) is linear.

The maximum possible variation range of the maximum burden can be calculated if the minimum and maximum values for the parameters within the square root symbol are known.

Let us anticipate the following maximum values: $p_e = 1600$ kg/m$^3$, $s = 1.0$, $c_0 = 0.42$ kg/m$^3$, $f = 1.0$ and $S_d/B_d = 8$ and the following minimum values: $p_e = 800$ kg/m$^3$, $s = 0.85$, $c_0 = 0.34$ kg/m$^3$, $f = 0.75$ and $S_d/B_d = 0.1$.

Maximum values in the numerator are combined with minimum values in the denominator and this gave the maximum value of 76 for the proportional constant in formula (6). Minimum values in the nominator area were then combined with maximum values in the denominator. This gave the smallest value 14 for the proportional constant in formula (6).

---

![Fig 1. Relation between burden and powder factor. Langefors and Kihlström (1978).](image1)

![Fig 2. Relation between maximum burden ($B_m$) and borehole diameter ($d$). Langefors and Kihlström (1978), Ash (1963), and Konya (1968).](image2)
The Langefors-Kihlström formula could therefore be written, see also Fig 2.

\[ B_m = (14 \text{ to } 76) \, d \]  

(7)

An even more simplified version of the Langefors and Kihlström formula can be achieved if we anticipate that the density of the explosive in the borehole \( \rho_e = 1.0 \), the weight strength \( s = 1.0 \), the corrected blastibility factor \( c_0 = 0.44 \) and finally \( S_d/B_d = 1.0 \).

\[ B_m = 46 \, d \]  

(8)

This formula or rule of thumb is commonly used for simple charge calculations in Sweden according to Brännfors (1973). The formula is shown together with other linear formulas in Fig 2.

Formula (6) does not give any information about the practical burden which can only be calculated if the borehole deviations are known. When using formula (6) for borehole diameters larger than 89 mm it can be shown that the calculated value deviates more and more from the actually burdens used in practice. The reason for this will be explained further on, but before that, other empirical formulas for \( B_m \) versus \( d \) shall be illustrated.

Pearse (1955) developed an empirical linear formula for \( B_m \) and \( d \).

\[ B_m = 8.5 \, d \sqrt{\frac{P_d}{\sigma_t}} \]  

(9)

\[ B_m = \text{maximum burden (m)} \]
\[ d = \text{borehole diameter (m)} \]
\[ P_d = \text{maximum detonation pressure (MPa)} \]
\[ \sigma_t = \text{maximum tensile strength of rock (MPa)} \]

This formula is not easy to use when the maximum tensile strength is not known. Experimentally it is also difficult to determine the maximum detonation pressure.

Ash and Pearse (1962) and Ash (1963) modified Pearse formula. The constant 8.5 and the factor \( \sqrt{P_d/\sigma_t} \) were replaced by one single constant \( K_b \). From field tests the following formula was derived.

\[ B_m = K_b \, d \]  

(10)

\[ B_m = \text{maximum burden (m)} \]
\[ K_b = \text{proportional factor varying between 20-40 depending on rock- and explosive parameters.} \]
\[ d = \text{borehole diameter (m)} \]

The formula is also illustrated in Fig 2.

Ash (1968) presented an empirical formula by Konya where the maximum burden is proportional to the diameter of the borehole.

\[ B_m = 38 \, d \sqrt{\frac{\rho_e}{\rho_r}} \]  

(11)

\[ B_m = \text{maximum burden (m)} \]
\[ d = \text{borehole diameter (m)} \]
\[ \rho_e = \text{density of the explosive (kg/m}^3\text{)} \]
\[ \rho_r = \text{density of the rock (kg/m}^3\text{)} \]

It is important to notice from this formula that the density of the rock is included as a parameter. In the Langefors-Kihlström formula (6) the effect of the density of the rock is not included directly but indirectly because the rock density influences the blastability factor \( c \).

The importance of the density of the rock in blasting has been verified by Atchison (1964), Atchison (1968) and Leinz and Thum (1970) and also by Rustan and Vutukuri (1983) the latter especially for fragmentation. Research at Luleå University of Technology showed that the product of rock density and P-wave velocity (the impedance) is very important for fragmentation and rock blastability.

The inclusion of explosive density and rock density in the burden diameter blast formula by Konya can be derived mathematically according to Rustan as follows,

\[ q = \frac{Q}{m} = \frac{\pi}{2} \left( \frac{d}{2} \right)^2 I_c \frac{\rho_e}{B_{pl}^2 H_b \rho_r} \]  

(12)

\[ q = \text{powder factor (kg/t)} \]
\[ Q = \text{total charge per borehole (kg)} \]
\[ m = \text{blasted mass of rock (kg)} \]
\[ I_c = \text{length of charge in the borehole (m)} \]
\[ B_{pl} = \text{practical burden = the burden when the ratio between spacing and burden equals one (m)} \]
\[ H_b = \text{bench height (m)} \]

Other symbols have been explained earlier in the paper.

Equation (12) can be rewritten to

\[ B_{pl} = f(d \sqrt{\frac{\rho_e}{\rho_r}}) \]  

(13)

Mathematically it is therefore a linear relationship between practical burden (\( B_{pl} \)) and borehole diameter (\( d \)) and the square root of explosive density divided by rock density.

If it in practise can be shown that the burden is directly proportional to the borehole diameter and all other parameters are constant when blasting in a certain rock type, this implies that the powder factor should not change when the burden is changed according to equation (13).
3.2 Maximum influence of explosive density and rock density on the burden

According to Lama and Vutukuri (1978) the density of rock varies from 1670 kg/m³ for chalk (Nareth in Israel) to 5070 kg/m³ for hematite (Soudan in Michigan, USA). The density of explosive in the borehole can vary from 800 - 1600 kg/m³.

As shown earlier maximum values in the nominator are combined with minimum values in the denominator and vice versa. The values are put into formula (11).

\[ B_m = (15 \text{ to } 37) d \]  \hspace{1cm} (14)

This formula has been introduced into Fig 2 for comparison with other linear relationships between burden and borehole diameter.

If we anticipate common values for explosive density and rock density, 1000 respectively 2800 kg/m³, formula (11) can be simplified to

\[ B_m = 23 d \]  \hspace{1cm} (15)

3.3 Power formula developed for open pit mines

A diagram given in a brochure, from Atlas Copco showed that the relation between burden and borehole diameter was not linear in open pit blasting. The following formula was derived by curve fitting.

\[ B_p = 19.7 d^{0.790} \]  \hspace{1cm} (16)

\[ B_p = \text{practical burden (m)} \]
\[ d = \text{borehole diameter (m)} \]

The formula is shown in Fig 3 and it is valid for borehole diameters from 50-300 mm in open pit blasting.

In formula (16) the ratio of spacing to burden has not been taken into account.

Usually the S/B ratio is equal to 1 for large borehole diameters in open pit mines, but at borehole diameters less than 100 mm, the S/B-ratio could vary between 1.5-2.0 and in extreme cases it could be as high as 8.

4 POWER FORMULAS DERIVED BY STATISTICAL ANALYSIS

4.1 Underground magnetite mines in Sweden

To eliminate the influence of different rock densities, values for borehole diameters and their respectively burdens were collected from magnetite orebodies at Luossavaara-Kirunavaara AB (LKAB) and the Research Mine at Luossavaara, see Table 2.

The following formula was derived for the values given in Table 2.

\[ B_p = 8.50 d^{0.525} \]  \hspace{1cm} (17)

\[ B_p = \text{practical burden (m)} \]
\[ d = \text{borehole diameter (m)} \]

Observe that the relation between burden and borehole diameter follows a power function. The LKAB formula is compared with the Langefors and Kihlström formula and the Atlas Copco formula in Fig 3.

One possible reason why the burdens used at 102 and 165 mm borehole diameter are small could be that the

**Table 2** Relation between practical burden and borehole diameter when blasting in Swedish magnetite ore. Density ~ 4800 kg/m³

<table>
<thead>
<tr>
<th>Location</th>
<th>Practical burden (m)</th>
<th>Borehole diameter (m)</th>
<th>Borehole length (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>LKAB/Kimna</td>
<td>1.8</td>
<td>0.052</td>
<td>17</td>
</tr>
<tr>
<td>LKAB/Malmberget</td>
<td>2.5</td>
<td>0.102</td>
<td>25</td>
</tr>
<tr>
<td>Loussavaara</td>
<td>3.3</td>
<td>0.165</td>
<td>90</td>
</tr>
<tr>
<td>Research Mine</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Fig 3. Burden versus borehole diameter. Atlas Copco practical burden values refer to open pit mines and LKAB practical burden values to magnetite underground mines in Sweden.
fragmentation should be fine in underground blasting. Another reason could be that the borehole length is comparatively long compared to surface mines and hole deviations have to be compensated by decreased practical burden.

Almgren and Klippmark (1981) showed for example that drillhole deviations could cause additional costs for ore losses and waste rock dilution when mining narrow orebodies. These costs might be as large as the operating cost for sublevel stoping in large orebodies.

Another reason is, that the boreholes can not be drilled parallel underground, because the drilling drifts are too narrow and the pillars between the drilling drifts too wide. Only at the bottom of the borehole sufficient burden and spacings are achieved.

4.2 Statistical derivation of formula for practical burden ($B_{p1}$)

The values used to derive formula (17) were very few, only three values and the zero point, and the basic values to derive formula (16) were also lacking. Therefore about 100 values for practical burdens and borehole diameters were collected from different references. For example an extensive list from Canadian open pit mines by Dubnie (1972) was used. Other sources were Almgren (1988), Nielsen (1986) and Bauer (1978). Together this represents borehole diameters from 48 to 381 mm. Primary and calculated data are shown in Table 3, see Appendix 1.

It is not sufficient to record the relation between burden and borehole diameter because the powder factor is determined by both the drilled burden ($B_d$) and drilled spacing ($S_d$). In Appendix 1 the theoretical area per hole has been determined, that means the product of $B_dS_d$. If the inclination of the bench face is 90$^\circ$ this product is the same as $B_pS_p$ (practical burden x practical spacing).

The practical burden ($B_{p1}$) when spacing and burden are equal was calculated according to the following definition.

$$B_{p1} = \sqrt{B_p \times S_p}$$

(18)

$B_{p1}$ = practical burden when $S/B = 1$, (m)

$B_p$ = practical burden (m)

$S_p$ = practical spacing (m)

A Taylor series fit was done for the relation between $B_{p1}$ and $d$ according to the following equation

$$B_{p1} = k_0 + k_1d + k_2d^2 + k_3d^3 + k_4d^4$$

(19)

With the help of the computer program "Statgraphics" it was shown that $k_0 = 0$, and the constants $k_2$ and larger (as shown in equation (19)), were so small that they could be neglected.

This means that the best approximation of a Taylor series fit was

$$B_{p1} = k_1 d$$

(20)

However a power curve fit, see Fig 4, gave an even higher correlation coefficient and therefore the Taylor approximation formula 20 was not used.

$$B_{p1} = 23.4 d^{0.855}$$

(21)

Correlation coefficient $R = 0.90$

In Fig 4 each single value is evenly distributed above and below the power curve fit. Comparison is made to the Langefors and Kihlström formula $B_m = 46d$.

Almost all values are below the Langefors and Kihlström curve for average Swedish rock. This means that the Langefors and Kihlström curve is not valid for large borehole diameters.

From Fig 4 it can also be seen, that the values for borehole diameters 165 mm underground are all below the regression curve. This could imply that the burden and spacing used underground are too small. One explanation can be that this borehole diameter needs wider drifts and
thinner pillars between the drifts to be able to give the boreholes a sufficient burden and spacing. Normally however it is not possible to increase the width of the drift and decrease the width of the pillar due to the stability of the drifts.

There are however two underground mines, Mount Isa and Kidd Creek, using 90 m long and 200 mm diameter blastholes underground where the mean value for the practical burden is very close to the regression line. This might imply a rational mining or the fragmentation may be too large.

Fig. 4 also shows that almost all underground values are below the regression line. It was therefore necessary to separate data from underground mines from open pit data and make a regression analysis for each of them.

4.3 Statistical formula for open pit mines (d = 89-381 mm)

The following formula was derived for open pit mines.

\[ B_{p1} = 18.1 \cdot d^{0.689} \pm 52\% \text{ Expected maximum } \]
\[ -37\% \text{ and minimum value} \] (22)

Valid for borehole diameters 89-381 mm

Correlation coefficient \( R = 0.78 \)

The formula is shown in Fig. 5.

4.4 Statistical formula for underground mines (d = 48-165 mm)

The data from the 200 mm borehole diameter was neglected because they deviated very much from the general trend. It is also known that these large borehole diameters can cause damage underground, and vibration restrictions on the surface might make it difficult to use these borehole diameters in the future.

The 200 mm boreholes were drilled with the rotary crushing drilling technique. This method gives smaller borehole deviations. That could be one of the explanations why these borehole diameters can break a comparatively large area compared to the 165 mm boreholes. The latter borehole diameters are usually drilled with the in the hole hammer technique.

In Fig. 5 the relation between practical burden and the borehole diameter for underground mines is,

\[ B_{p1} = 11.8 \cdot d^{0.630} +40\% \text{ Expected maximum } \] (23)
\[ -25\% \text{ and minimum value} \]

Correlation coefficient \( R = 0.94 \)

The formula is valid for borehole diameters from 48 to 165 mm.

4.5 Practical advise for calculation of the practical burden and spacing

Formula (22) should be used to calculate the practical burden \( B_{p1} \) for open pit mines and formula (23) for underground mines. When this has been done, it is easy to calculate burden and spacing for any S/B-ratio according to formula (18). These formulas can be used as a first step in roughly determining the burden and spacing when drillhole deviations, explosive, and rock properties are not well known. The calculated values can be adjusted according to the following.

* If the rock density is large, reduce the calculated burden value and vice versa.
* If the borehole deviations are large, reduce the calculated burden value and vice versa.
* If the strength of explosive is large in hard rock, increase the value and vice versa.
* If the strength of explosive is small in soft rock, the burden value should be increased and vice versa.

5 CONCLUSIONS

This paper shows that the linear relation between burden and borehole diameter could only be used in a small diameter range. To be able to cover all borehole diameters used in practice today it is necessary to use a power
formula, one for open pit mines (22), and one for underground mines (23).

The basic formulas given in rock blasting text books are recommended to be revised according to this information.

The reason why the underground formula is different from the open pit formula might be

* Larger confinement of the boreholes underground.
* Cautious blasting has to be undertaken close to the hanging- and footwall underground.
* The borehole length is larger underground and the borehole deviations have to be compensated for a smaller burden and spacing. In open pit mines all boreholes can be drilled parallel to each other and the borehole deviation is small at large diameters. The bench height is generally only 15-20 m.

One reason that the relationship for open pit mines is not linear is that when the borehole diameter is increased the bench height is not scaled up, and for large drill hole diameters only bottom charge is therefore used to break the rock. It is however a well known fact that the rock will fragment better if the bench height is increased.

When the boreholes are getting further apart it is also necessary to use a higher powder factor to keep the fragmentation below acceptable values.

The Langefors and Kihlström formula presented in Fig 1, is not correct in the burden range 1-10 m, because according to the diagram the powder factor should be almost constant, but the statistical analysis shows that the powder factor (kg/m³) increases with burden according to Fig 5. Only at a linear relation between the burden and borehole diameter the powder factor (kg/m³) will be constant.

Probably the burden and spacings are too small when using 165 mm boreholes underground. If the boreholes can be arranged parallel and drilled with smaller borehole deviations it might be possible to increase the burden and spacing from 3.8 x 3.8 m to 4.8 x 4.8 m.

6 FUTURE WORK

This is the first step to create a more scientific rock blasting formula for calculation of the burden related to the borehole diameter. It was found mathematically that the power formulas presented in the paper might be improved if the explosive density and rock density are introduced into the formulas like in formula (11). New proportional constants have therefore to be derived. The influence of rock density and explosive density must however be examined by field tests to get the correct values of the constants k₁ and k₂ in formula (24) and (25).

\[ B_{pl} = k_1 \cdot d^{0.689} \sqrt{\frac{D_c}{P_r}} \quad \text{open pit mining} \quad (24) \]

\[ B_{pl} = k_2 \cdot d^{0.630} \sqrt{\frac{D_c}{P_r}} \quad \text{underground mines} \quad (25) \]

Other parameters like rock texture and structure are more difficult to include in the formula. To include these parameters it is necessary to undertake detailed structure and strength analysis of the rock mass in many operations.

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Appendix 1.

Table 3. Drilling data used for determination of the relation between burden, spacing and borehole diameter