Where does the explosive energy in rock blasting rounds go?

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Abstract
The energy partitioning of limestone production blasts has been measured under well controlled conditions; from rock properties determination, structural mapping and monitoring of drilling and charging, to VOD, 3D accelerometer and bench face movement measurements during the blast to postblast fragment size measurements. The efficiency of the transfer of energy from the charges to the surrounding rock has been estimated by introducing a new explosives test, the cylinder expansion test, which is discussed to begin with.

Hereby in-hole losses, seismic energy, kinetic energy of throw and fragmentation energy have been calculated. For the AN doped gassed bulk emulsion used in the quarry, it is estimated that 60-70 % of the explosive energy is transmitted to the rock, that the seismic and kinetic energy terms both amount to roughly 3-12 % and the fragmentation energy to less than 1-2 %. One candidate for the remaining losses, which are larger than the other three terms together, is the crushing of the blast-hole region. A rough estimate for the energy partitioning in bench blasting rounds is finally given.

1. Introduction
The company Nordkalk AB at Storugns on the island of Gotland is Scandinavia’s largest producer of crushed limestone. Historically the amount of fines less than 25 mm in size has entailed a lower sales price or deposited on slag-heaps. There has been a continuous drive to lessen the amount of fines produced.

With the overall goal of decreasing the amount of fines produced by blasting, Nordkalk AB, Dyno Nobel Europe AB and Swedish Rock Engineering Research (SveBeFo) entered the EU-financed project “Less fines production in aggregate and industrial minerals industry”, project no. GRD-2000-25224 in 2001. “Less Fines” has other partners too, Montan-universität Leoben and ballast manufacturer Hengl Bitustein in Austria, the CGES & CGI departments of the Ecole de Mines in Paris, France and Universidad Politecnica de Madrid (UPM) with explosives manufacturer UEE and cement manufacturer Cementos Portland from Spain.

This article presents field tests made at the Klinthagen quarry of Nordkalk. During May 2002, 5 instrumented production rounds were blasted in order to measure where the explosive energy goes. The work is reported in greater detail. These rounds were the basis for continued fieldwork during the spring of 2003 when Swebrec did the work on behalf of SveBeFo.

2. The energy balance of blasting
Detonating explosives form high pressure and temperature gases that can perform much work while they expand down to atmospheric pressure. The theoretically largest amount of work just about equals the detonation energy\(^1\). Many manufacturers report a weight strength based on the related quantity explosion pressure \(E_0\) or a lower value based on the assumption the the gases stop doing useful work at some pressure, e.g. 1000 atm. Langefors' weight strength\(^2\) is hardly used anymore.

Only a fraction \(\eta\) of the explosion energy is transferred to the rock mass. Some of this is converted to useful energy like kinetic energy of throw \(E_k\) and fragmentation \(E_f\). Seismic energy \(E_s\) and other losses are also formed. The energy balance of a blast may thus be written
\[ \eta E_0 = E_r + E_i + E_v + \text{other losses.} \] (1)

Other losses are\(^6\) the residual heat in the blast fumes, heat transferred to the rock mass, air shock waves and shock wave energy that doesn’t result in fragmentation. Of these the residual heat and air shock waves are contained in the factor \( \eta \) of the left member of eq. 1. An example of losses in the rock mass is shock wave and friction losses in the crushed zone that surrounds the blast-hole.

The energy balance has been written on a form that shows what can be measured with reasonable work input. The kinetic, seismic and fragmentation energy components can be measured in the field. This article shows how this was done in the Klinthagen quarry.

### 3. On the work capacity of explosives

A new independent measure of the relative work capacity of explosives has been developed\(^6\). In the cylinder expansion test, copper tubes filled with explosives are shot and the explosive VOD and the expansion velocity of the tube wall are measured. In our tests, final wall velocities of 1100-1500 m/s were measured for civilian explosives in \( \phi \) 100/110 mm tubes. See Fig. 2. Other work\(^7\) was also helpful in developing the test method.

The event is fairly rapid. Within 40-50 \( \mu \)s the wall accelerates and reaches a constant final velocity that is kept until the tube fractures. The event may be followed until the blast fumes have expanded about 8-9 times the original tube volume and more or less have stopped doing work on the copper cylinder.

The sum of the kinetic energy of the copper tube and the radial kinetic energy of the gases, the so called Gurney energy \( E_r \), is computed. \( E_r \) is a measure of the work capacity of the explosive. The relative work capacity or the utilisation ratio is then given by

\[ \eta = E_d/E_0. \] (2)

Table 1 shows the measured values of some civilian explosives\(^6\). ANFO prills from three different manufacturers were tested. One manufacturer also supplied an ANFO quality with 7 % of aluminium added. Titan \(^\text{\textregistered} \) 6000 is the pure gassed SME bulk emulsion supplied by Dyno Nobel and used at Klinthagen. Titan 6080 contains 20 % of AN prills.

Em 682 is an AN/SN based emulsion explosive sensitises

### Table 1 Data on the work capacity of civilian explosives.

<table>
<thead>
<tr>
<th>Explosive</th>
<th>Density ( \rho ) kg/m(^3)</th>
<th>Final tube wall speed m/s</th>
<th>Explosion energy ( E_0 ) MJ/kg</th>
<th>Gurney energy ( E_r ) MJ/kg</th>
<th>Rel. work capacity ( \eta )</th>
</tr>
</thead>
<tbody>
<tr>
<td>ANFO</td>
<td>750-900</td>
<td>1100-1250</td>
<td>3.9-4.0</td>
<td>1.6-1.9</td>
<td>0.4-0.5</td>
</tr>
<tr>
<td>ANFO + alu</td>
<td>900</td>
<td>1400</td>
<td>5.1</td>
<td>2.5</td>
<td>0.5</td>
</tr>
<tr>
<td>Titan 6000</td>
<td>900</td>
<td>1200</td>
<td>3.2</td>
<td>1.8</td>
<td>0.5-0.6</td>
</tr>
<tr>
<td>Titan 6000</td>
<td>1150-1250</td>
<td>1300-1500</td>
<td>3.2</td>
<td>2.1-2.3</td>
<td>0.6-0.7</td>
</tr>
<tr>
<td>Titan 6080</td>
<td>800</td>
<td>1100-1200</td>
<td>3.3</td>
<td>1.8-2.0</td>
<td>0.5-0.6</td>
</tr>
<tr>
<td>Titan 6080</td>
<td>1150-1250</td>
<td>1400-1500</td>
<td>3.3</td>
<td>2.0-2.3</td>
<td>0.6-0.7</td>
</tr>
<tr>
<td>Em 682 100%</td>
<td>1100-1200</td>
<td>1300</td>
<td>2.9</td>
<td>1.7-1.8</td>
<td>0.55-0.60</td>
</tr>
<tr>
<td>Em 682 80%</td>
<td>1100-1200</td>
<td>1300</td>
<td>3.0</td>
<td>1.7-1.8</td>
<td>0.55-0.60</td>
</tr>
</tbody>
</table>
by glass micro-spheres. It is similar to many commercial explosives and was developed jointly by SveBeFo, Dyno Nobel AB and Kimit AB. Em 682 contains 20% of ANFO and 80% of the emulsion by weight.

The most important conclusion from the data in Table 1 is that Titan 6080, despite the lower weight strength value given by Dyno, 3.3/4.0 = 84%, performs more work than ANFO because the Eo-value is larger for Titan 6080 at a normal loading density of 1150 kg/m³. The relative work capacity for the Titan series of explosives is 60-70% but only 40-50% for ANFO.

The table also shows that the work capacity of the Titan series explosives increases with an increasing density despite that the VOD starts to drop when the density approaches 1300 kg/m³. The consequence is that the VOD-value may not be a good measure of the work capacity of an explosive.

Adding AN or ANFO seems to have little influence on the work capacities of the explosives in Table 1. Nor are their acceleration phases very different either, except at the lowest densities, 800-900 kg/m³.

Adding aluminium raises the work capacity of ANFO though but not the utilisation ratio $\eta$. The effect of aluminium is to prolong the acceleration phase during which the work is transferred considerably though.

The ANFO and Titan tests were all made in ø 100/110 mm tubes, i.e. tubes with an inner diameter of 100 mm and a wall thickness of 5 mm. For explosive E682, tubes with emulsion ø 40/44, 60/66, 80/88 and 100/110 mm were tested, i.e. the explosive to metal mass ratio was in principle constant. The work capacity of the explosive showed no significant influence of charge diameter despite the fact that the measured VOD-values did so, especially for E682 with 20% emulsion. The same conclusions were drawn from tests on ANFO.

The underwater test is another experimental measure of the work capacity of explosive. In comparison the cylinder test has several advantages:

1. The cylinder test has the same detonation geometry as in rock blasting, a grazing, axial stationary detonation in an extended charge.
2. It is easier to study the effect of charge diameter and e.g. decoupling in the cylinder test.
3. From the cylinder charge, about the same relative amount of explosive energy is transferred to the tube as from the blast-hole charge in rock blasting.
4. The explosive stops doing useful work in the cylinder test when the gases or fumes have expanded about 5-10 times, which is close to the limit of 10-20 times that several researchers consider valid in rock blasting. The charge in the underwater test continues doing work down to much greater expansion volumes.
5. The energy in the cylinder test is partitioned much more like in rock blasting.

In the test rounds at Klinthagen, Titan 6080 from a Dyno bulk truck, with a density of 1100-1200 kg/m³ in ø 89-mm holes, was used. Thus there a relative work capacity of about 60-70% is used in the energy balance, corresponding to 30-40% of in-hole losses.

Fig. 3 Muckpile after test round no. 1 in lower bench (bench 2).

4. The field conditions at Nordkalk's Klinthagen quarry.

The blasts at Klinthagen were single row rounds with 20-25, ø 89-mm blast-holes, about 13 m deep, inclined at 16° and 2 m of sub-drilling in 11-m high benches. See Fig. 3. The nominal burden and spacing values were 3.6 and 4.8 m respectively. The charging length was on average slightly less than 10 m and the stemming consisted of about 3 m of gravel. The linear charge concentration was 7.4 kg/m for an average charge of 68 kg and specific charge of about ca 0.4 kg/m³.

The Nonel Unidet® was used for initiation of the rounds, using a U475 cap in a 1.7 kg bottom primer and a U500 cap in a 1.25 kg charge at the top of the explosive column. The surface delays were either Unidet Snapline SL42 in three of the rounds or SL25 in the other two. No differences in blast results due to the different surface delays were found.

The initiation accuracy is determined by the nominal scatter of the bottom caps, ± 6 ms, and becomes in practice about ± 9 ms between two caps. Our function control measurements gave the initiation statistics (mean ± scatter) 23.1 ± 6.0 ms and 43.3 ± 6.0 ms, which was considered fully acceptable.

The VOD measurements showed the characteristic pattern for gassed bulk emulsions, namely that the VOD is highest in the bottom of the charge column and decreases towards the top. We obtained the bottom results 4990 ± 260 m/s and 4410 ± 190 m/s at the top. This was also considered fully acceptable.

The quarrying at Nordkalk is done in two benches, Fig. 3. The limestone in the upper bench (level 1) of the test area consists of horizontally layered stromatoporoid and crinoid facies in the proportions 80/20. The lower bench consists mainly of reef, fragmentary and crinoid facies in the proportions 75/15/10. Differences in arrival times from ground vibration measurements were used to estimate the P-wave speed $c_p$ in the benches. There was a considerable difference in the two benches, $c_p = 4350 ± 550$ m/s in the
upper and 5550 ± 700 m/s in the lower bench.

The test area lay in the South Eastern corner of the quarry. Two rounds behind each other were fired in the upper bench and three in the lower bench, some 300 m to the East in order to estimate the effect of limestone quality and geology. The rock masses had been structurally mapped and the in-situ fracture families had been determined by our French Less Fines colleagues, Moser\textsuperscript{59}.

Our Austrian colleagues had made thorough measurements of rock mechanics parameters, Moser\textsuperscript{60} and also laboratory scale blasting tests\textsuperscript{57}.

We did thorough follow ups of the staking out, drilling and charging of the blast-holes during the test rounds together with our Nordkalk and Dyno colleagues. On top of the VOD we also measured i) the ground vibrations in order to calculate the seismic energy $E_s$ during the blasts and ii) the bench face movement in order to calculate the kinetic throw energy $E_k$. We calculated the fragmentation energy $E_f$ from the fragment size distribution measured after blasting.

5. Measurement and calculation of the energy components

5.1 Ground vibrations and seismic energy

Our experience tells us that surface mounted geophones can not reproduce the vibrations from blasting rounds accurately enough to e.g. determine blast damage in rock. The frequency contents are filtered and the surface mounting distorts the signal appreciably. We feared that these limitations would apply when measuring the seismic energy at Klinthagen.

Therefore we used dismountable three component accelerometers with steel anchors that were grouted to the bottoms of 4-m deep gauge holes. Three gauge holes were used in each bench. Two were positioned about 20 m behind the first round, i.e. row of holes, symmetrically with respect to and about 10 m from the centre line in order to compare the signals from equivalent hole positions. One gauge hole lay about 30 m behind the first row on the centre line in order to evaluate the geometric damping of the signals better. The measuring range was 10-50 m. The bandwidth 25 KHz of the recording equipment was determined by the DAT recorder SIR-1000 from Sony.

We thus recorded on nine channels during each round without signal losses. A trace is shown in Fig. 4. It may be seen that the individual pulses do not overlap and that they are distinct. The effect spectrum is fairly constant from 20-4000 Hz. Peak vibration velocities (PPV) were obtained after an integration and vector summation of three component signals.

The PPV-values were reasonable and tended to be larger in the upper bench, level 1. See Fig. 5. The PPV-values were normalised against the P-wave velocity $c_p$ and the scaling law could be written

$$PPV = (0.047\pm0.02)\cdot c_p v^2 (Q/68) R^{1.0}.$$  (3)

![Fig. 5 Site scaling law for test area in Klinthagen quarry. Log(PV) as function of Log(R/vQ), where R/vQ is reduce distance. Note the charge normalisation against average charge weight 68 kg per hole in test rounds. Level = bench.](image-url)
Seismic energy $E_a$, MJ

$$\text{Time (s)} = \text{no. of samples/sample rate} \times 4800/s$$

Fig. 6 Seismic energy radiated from hole no. 15 in round no. 1 versus time.

Here $Q$ is the charge size in kg and $R$ is the distance between charge and gauge in m. The value of the damping exponent is high, 2.09, and the scaling law should be checked before it is used outside the 10-50 m range. The calculated prefactor for bench 1 was 0.49 and for bench 2 0.045 but practically speaking 0.047 could be used in both benches as the difference is less than ±5%.

An approximate seismic energy value $E_a$ (J) may be calculated from the time trace of the vibration signal\textsuperscript{19,19},

$$E_a(T) = 4 \pi R^2 \rho v^4 \int_0^T 2 \nu(t) \text{d}t.$$  \hspace{1cm} (4)

$R$ (m) is the distance between charge and gauge and $\rho = 2600$ kg/m$^3$ is the rock density. Eq. 4 doesn't account for a number of factors: an extended charge, wave reflections, a velocity vector that deviates from the normal of the wavefront, outgoing and incoming waves, non-elastic material behaviour etc. The expression is thus seriously simplified but has a distance dependence so that $E_a(T)$ should be reasonably independent of distance $R$ when losses are small.

Luckily the single hole pulses in Fig. 4 do not overlap. The integration time can thus basically be restricted to either 25 or 42 ms. The $E_a(T)$ curves normally display a knee after about 20 ms, see Fig. 6. By then some 80% of the seismic energy has radiated out. Our integrations were normally made considerably longer.

Seismic energy from single holes in blasts $E_a$, MJ

Fig. 7 Calculated values of seismic energy $E_a$ for all holes in lower bench.

The diagram of $E_a$ versus time in Fig. 7 displays a surprisingly large scatter, a factor of 9 roughly between the highest and the lowest values of 5-45 MJ per hole. Table 2 gives statistics for the data lumped with respect to gauge point, bench (level) and round. The average radiated seismic energy is somewhat higher from the holes in bench 1 than from the holes in bench 2, 20 MJ versus 15 MJ. The difference is hardly significant though, considering the large scatter in the data.

Factors that may contribute to the scatter in the calculated values are i) the layered limestone, ii) its anisotropy and iii) the fracturing around a blast-hole that influences the wave transmission from a neighbouring blast-hole. We didn’t observe any effect of the direction of propagation of the waves. The wave shielding effect of fractures is also considered important\textsuperscript{30}.

The uncertainty and the scatter of the seismic energy values made us use the global statistics. For a blast-hole with $Q = 68$ kg of explosive we then obtain

$$E_a = 17 \pm 9 \text{MJ/hole} \times 0.25 \pm 0.13 \text{MJ/kg explosive} = (0.075 \pm 0.040) E_0$$  \hspace{1cm} (5)

when the explosion energy value of $E_0 = 3.31 \text{MJ/kg}$ for Titan 6080 is used. In round numbers the seismic energy thus becomes $E/E_0 = 3-12\%$.

Table 2 Statistics for radiated seismic energy from single blast-holes, $E_a$ in MJ.

<table>
<thead>
<tr>
<th>Gauge point</th>
<th>Lower bench</th>
<th></th>
<th>Upper bench</th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Round 1</td>
<td>Round 3</td>
<td>Round 5</td>
<td>Round 2</td>
<td>Round 4</td>
</tr>
<tr>
<td>1 or 1'</td>
<td>8.5 ± 6.3</td>
<td>17.4 ± 12.0</td>
<td>25.1 ± 9.6</td>
<td>17.2 ± 4.5</td>
<td>15.4 ± 5.8</td>
</tr>
<tr>
<td>2 or 2'</td>
<td>10.8 ± 4.7</td>
<td>10.1 ± 3.7</td>
<td>11.5 ± 3.0</td>
<td>15.2 ± 5.1</td>
<td>15.7 ± 2.8</td>
</tr>
<tr>
<td>3 or 3'</td>
<td>12.9 ± 5.2</td>
<td>19.3 ± 12.0</td>
<td>17.3 ± 4.3</td>
<td>29.9 ± 14.4</td>
<td>23.7 ± 4.6</td>
</tr>
<tr>
<td>Round</td>
<td>10.9 ± 5.4</td>
<td>16.2 ± 10.5</td>
<td>18.1 ± 8.2</td>
<td>20.8 ± 11.0</td>
<td>18.8 ± 5.8</td>
</tr>
<tr>
<td>Bench</td>
<td>15.2 ± 8.5</td>
<td>8.5(64)</td>
<td></td>
<td>20.1 ± 9.6</td>
<td>9.6(31)</td>
</tr>
</tbody>
</table>
5.2 Bench face velocity and kinetic energy

In order to measure the bench face motion we used a high-speed video (HSV), Red Lake MSD Motion Meter 1000 with max 1000 fps. The motion was analysed with the Blaster’s MAS software from MREL. To obtain clear markers, two cardboard boxes were strung on a line from the top of the bench, 4 and 7 m below the crest respectively. See Fig. 8.

On the floor, two boxes were placed 5 m apart to give the length scale. The camera was placed about 300 m from the blast with a near grazing angle to the face, less than 20°, to minimise the angle errors.

The box markers moved with a relatively constant velocity the first 0.25 s and were tracked about twice as long, see Fig. 9. The velocity vectors of the tracked points show a normal bulging face. We measured face velocities in the range 8.5-11.3 m/s. The burden varied somewhat from round to round but there was no systematic correlation with the face velocity.

An assumption about the velocity distribution \( V(x, y) \) in the moving rock mass is needed in order to calculate the kinetic energy. An upper limit is obtained if the velocity is assumed to be constant within the burden. This could be reasonable initially but hardly for any longer period of time for a muckpile that is in contact with the new face as in Fig. 3. This gives

\[
E_k = \int V^2 \, dx \cdot \rho \cdot S \cdot H \cdot \frac{V^2}{2} \cdot \text{max}.
\]  

\( M \) is the rock mass per hole, \( H \) (m) bench height, \( B \) (m) burden and \( S \) (m) the spacing.

The form of the muckpile probably corresponds better to a velocity distribution that is zero at the blastholes and increases linearly towards the face in horizontal direction. With no vertical variation the kinetic energy is reduced to a 1/3 of the maximum value in eq. 6. This is an approximate lower limit for the kinetic energy. Retardation effects caused by block collisions in space could e.g. explain the

Fig. 9 High-speed image from round no. 4, about 100 ms after face starts to move.

| Table 3 | The calculated values for the kinetic energy, \( E_k \) per blast-hole. |
|---------|------------------|-----------------|---------------|---------------|---------------|
|          | Lower bench      |                | Upper bench   |               |               |
|          | Round 1 | Round 3 | Round 5 | Round 2 | Round 4 |
| Initial velocity, m/s | 10.1     | 10.7     | 8.6     | -         | 11.1         |
| - bench height, m      | 10.5     | 11.0     | 10.1    | -         | 8.4          |
| - volume, m³/hole     | 182.6    | 191.3    | 175.6   | -         | 146.1        |
| Maximum \( E_k \), MJ | 24.2     | 28.5     | 16.9    | -         | 23.4         |
| Minimal \( E_k \), MJ  | 8.1      | 9.5      | 5.6     | -         | 7.8          |
| \( E_k \) variation, MJ| 5.6 – 28.5| 7.8 – 23.4| -       | -         | -            |
| \( Q \) charge, kg/bol | 68       | 74       | 71      | -         | -            |
| \( E_{\text{r}}/Q \), MJ/kg | 0.08 – 0.39 | 0.13 – 0.40 | - | - | - |
| \( E_{\text{r}}/E_k \), %   | 2.4 – 11.6 | 4.0 – 12.0 | -       | -         | -            |
span in values.
The calculated values for the kinetic energy are shown in Table 3.
Thus we may estimate the kinetic energy of the test rounds at Klinthagen to about as large as the seismic energy, i.e. 3-12 % of the explosion energy.

A closer study of the bench face movement in the two other quarries that participate in the Less Fines project has been made, the amphibolite quarry of Hengs Bitstein and the limestone pit of Cements Portland (CP). The estimates of the kinetic energy have the same uncertainties and are 5-16 % for Hengl and 2.5-7 % for CP.

5.3 Fragment size distribution and fragmentation energy

The fragmentation energy consists of the energy supplied when new surfaces are created. Starting from the fragment size distribution (cumulative mass passing versus screen size) and an assumption on fragment shape, the total area of the muck pile may be roughly calculated.

Nordkalk has installed digital equipment to measure the fragments size distribution at the primary crusher. Every truckload is photographed by a video camera on a frame with lighting before it is dumped into the crusher. The system keeps track of whether the truck comes from the upper or the lower bench. One round may consist of several hundred truck loads, see Fig. 10.

Normally the image analysis will not resolve pieces smaller than 1/20 of the largest rock in the image well. At Nordkalk the resolution limit is 50 mm, which is higher than the fines limit 25 mm. Therefore a second video camera, with a resolution of 10 mm is installed over the belt after the primary crusher. It can thus measure the combined amount of fines that blasting and crushing creates before the material is fed into the processing plant. These data are however not directly suitable for estimating the blasted size distribution.

We have used the cumulative Rosin-Rammler distribution to describe the fragmentation

\[ P(x) = 100 \cdot \left[1 - 2^{-\left(x/x_{50}\right)^{0.5}}\right]. \] (7)

Here \( x_{50} \) is the screen size through which 50 % of the fragments pass and the uniformity index \( n \) is a measure of the slope of the curve. Assuming that there are \( N_{h} \) holes in the row, its volume becomes \( V_{0} = N_{h} \cdot SBH \). For reasonably cubic fragments within a size interval \( \Delta x \) around the side length \( x \), their number \( \Delta N_{h} \) and the corresponding surface area \( \Delta A \) are related by \( \Delta A = \Delta V_{h} \cdot 6x = 6V_{h} \cdot P(x) / x \cdot \Delta x \). An integration over all fragment sizes yields

\[ A_{\text{round}} = 6V_{0} / (x_{50})(\ln 2)^{1/2} \cdot \Gamma(1-1/n). \] (8)

The gamma function \( \Gamma(1-1/n) \) has a finite value as long as \( n > 1 \). The basic estimate of the surface area of the

Fig. 10 Fragment size distributions from truckloads from round no. 4, level 1.

Fig. 11 Fractures in a Simbloc simulation for fragmentory limestone.
muckpile thus becomes 6V₀/x₀ and it is modified somewhat by the value of n.

Some of the surfaces in the muckpile are not new, they existed already in-situ. To distinguish between the fresh and in-situ surfaces in the muckpile is hardly practicable. Therefore we utilised the structural mapping made and an ensuing simulation, see Fig. 11, to estimate the existing area in-situ\(^1\). Equation 8 with in-situ data has been used to compute A_{in-situ}.

If we further know the specific surface energy \(\gamma\) (J/m\(^2\)), required to create fracture surfaces, then we may calculate the fragmentation energy as

\[
E_t = A_{in-situ} \gamma = (A_{round} - A_{in-situ}) \gamma = E_t,round - E_t, in-situ
\]

(9)

The surface energy may be measured either in the form of the Rittinger coefficient R (m²/J) or of the fracture toughness \(K_b\) (Pa·m\(^{1/2}\)). Both quantities were measured for the rocks of the Less Fines project\(^1\) and they yield quite similar \(\gamma\)-values. Here we have used \(\gamma = 1/R\), which roughly is about 100-150 J/m\(^2\).

Table 4 shows that about 15-20 m\(^2\) of surface per m\(^3\) of intact rock was created. The in-situ fracturing is larger in the upper bench (level 1), about 6.0 m\(^2\)/m\(^3\), than in the lower one, 2.8 m\(^2\)/m\(^3\). The blasting seems to create less new surface in the upper bench though so the total surface in the upper bench muckpiles becomes 22.0 m\(^2\)/m\(^3\) and 21.2 m\(^2\)/m\(^3\) in the lower bench muckpiles, i.e. in practice the same!

Converting to fragmentation energy the result becomes 3.3 kJ/kg in the upper bench and 6.5 kJ/kg in the lower. This corresponds to the negligible amounts 0.1-0.2 % of the explosion energy 3.3 J/kg!\(^1\)

6. The measured components of the energy balance

The energy components that have been computed from the measurements in the Klinthagen quarry may now be entered into the energy balance of eqn. 1. The results are shown in Table 5 together with data from the other quarries in the Less Fines project\(^1\) and literature data\(^2\)\(^3\). It is striking that the loss term makes up about half or more of the work transferred from the explosive to the rock mass at the first three sites. As already mentioned, the residual heat of the blast fumes and the air shock waves are already contained in the utilisation ratio \(\eta\), as is the axial kinetic energy of the fumes.

Our colleagues\(^1\) give a slightly different explanation. The show that by assuming an incomplete chemical reaction \(\eta < 1\), the can with reasonable accuracy calculate the expansion curves of the cylinder tests. Both explanations are probably needed in order to give an accurate picture of the process in the blast-hole.

The values for the seismic and kinetic energies were given with broad intervals. The correctness of the fragmentation energy hinges on how well P(x) describes the fragment size distribution in the fines range.

There are unfortunately many indications that the Rosin-Rammler distribution gravely underestimates the number of fragments smaller than say 5 mm\(^1\). There are further few cases of screened full size blasts.

One such case is the Bärarp rounds\(^2\)\(^3\). There are several screened distributions that cover the range 0.074-1000 mm. An estimate\(^1\) is that 70-80 % of the fracture surface is made up of fragments smaller than 0.1 mm and 90 % of fragments smaller than 1 mm. Therefore the fines part of the distribution is essential when calculating the fragment-

### Table 5 Energy balances of blast rounds at different sites.

<table>
<thead>
<tr>
<th>Site</th>
<th>No. of rounds</th>
<th>(\eta\cdot E_0) %</th>
<th>(E_s) %</th>
<th>(E_k) %</th>
<th>(E_f) %</th>
<th>Losses in rock, %</th>
</tr>
</thead>
<tbody>
<tr>
<td>Klinthagen</td>
<td>5</td>
<td>60-70</td>
<td>3-12</td>
<td>3-12</td>
<td>0.1-0.2</td>
<td>38-53</td>
</tr>
<tr>
<td>La Concha</td>
<td>2</td>
<td>40-50</td>
<td>2-6</td>
<td>16-25</td>
<td>0.1-3.2</td>
<td>16-22</td>
</tr>
<tr>
<td>Eibenstein</td>
<td>2</td>
<td>65</td>
<td>0.4-2.5</td>
<td>6-27</td>
<td>-</td>
<td>35-58</td>
</tr>
<tr>
<td>Spathis(^2)</td>
<td>1</td>
<td>58</td>
<td>5</td>
<td>27</td>
<td>0.3</td>
<td>26</td>
</tr>
<tr>
<td>Hinzen(^1)</td>
<td>5</td>
<td>-</td>
<td>0.1-5</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
</tbody>
</table>

\(^1\) Spathis, 1952
\(^2\) Hinzen, 1953
tation energy.

This makes it possible that the amount fresh fractures
surface created by the blast could have been underestimated
by say up to a factor of 10. Even if that were the case, it
would only amount to 1-2 % of the explosive energy.
Simply stated, it seems that fragmentation energy in any
case is less than the margin of error of the other energies.

Heat deposited in the rock is another possible energy
loss. It has several parts, e.g. shock wave heating, friction
work during crushing of the rock around the blast-hole and
heat transfer from the blast fumes that flow in the widening
fractures.

There is evidence that the crushing work may be consid-
erable. An analogy with projectile penetration gives the
number 1 kJ/cm² = 1 GJ/m³ for hole expansion. If the
blast-hole is expanded to twice its original size during
charging, a Ø 89 mm blasthole with the volume 6.2 l/m
consumes about 6.2 MJ. The hole itself contains 7.4 kg of
explosive. The crushing work would then be 0.84 MJ/kg
or about 25 % of the explosion energy. This term is of the
right order compared to the loss term 38-53 % at
Klinthagen in Table 5.

An estimate based on Bond’s equation with a work
index Wi of about 10 kWh/ton, crushing the rock down to
an x of 100 µm, says that this would require about 10
kWh/ton or 10 MJ/m³. This is two orders of magnitude
smaller than the penetration number above. The difference
is that Bond’s equation describes crushing or grinding
under atmospheric conditions, not under a high blast pres-
sure.

Another indirect piece of evidence is half-scale blasting
tests with Ø 20 mm charges in granite to determine the
maximum burden. Putting the charge in a hole of double
the charge volume permitted the charge to break out a
larger burden than when the charge was in direct contact
with the blast-hole wall. Accepting crushing around the
blast-hole as the major energy loss in blasting requires
more direct evidence though.

7. Conclusion

The measurements made at Nordkalk’s Klinthagen quar-
ry and at the other sites of Table 5 show that a rough
description of the energy partitioning of a blast with bulk
emulsion explosives would be
- Crushing and other losses in the rock mass 20 – 40 %
- Kinetic energy of throw 10 – 20 %
- Seismic energy 5 – 10 %
- Fragmentation energy 0.1 – 2 %

The calculated kinetic energy varies by at least a factor of
three, the measured seismic energy for a whole round by at
least a factor of two. The fragmentation energy, even with
the blasting fines included, lies well within the uncertain-
ties of the other two components! There is some evidence
to say that crushing around the blasthole is the dominating
energy loss. This is hard to measure in the field though.

What has been gained by this energy balance approach,
by showing that the energy required to create the fresh
fracture surfaces in a muckpile makes up for an insigni-
ficant part of the explosive energy?

- Firstly, within the Less Fines project, we conclude that
the energy balance hardly is a viable approach to min-
imise the amount fine material created by blasting.

- Secondly we believe that a deeper understanding of the
energy transfer from explosive to rock through the zone
surrounding the blasthole carries the possibility of
improving the blasting process. Swebrec ha initiated
work where expanded cylinder tests hopefully will shed
light on this.

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