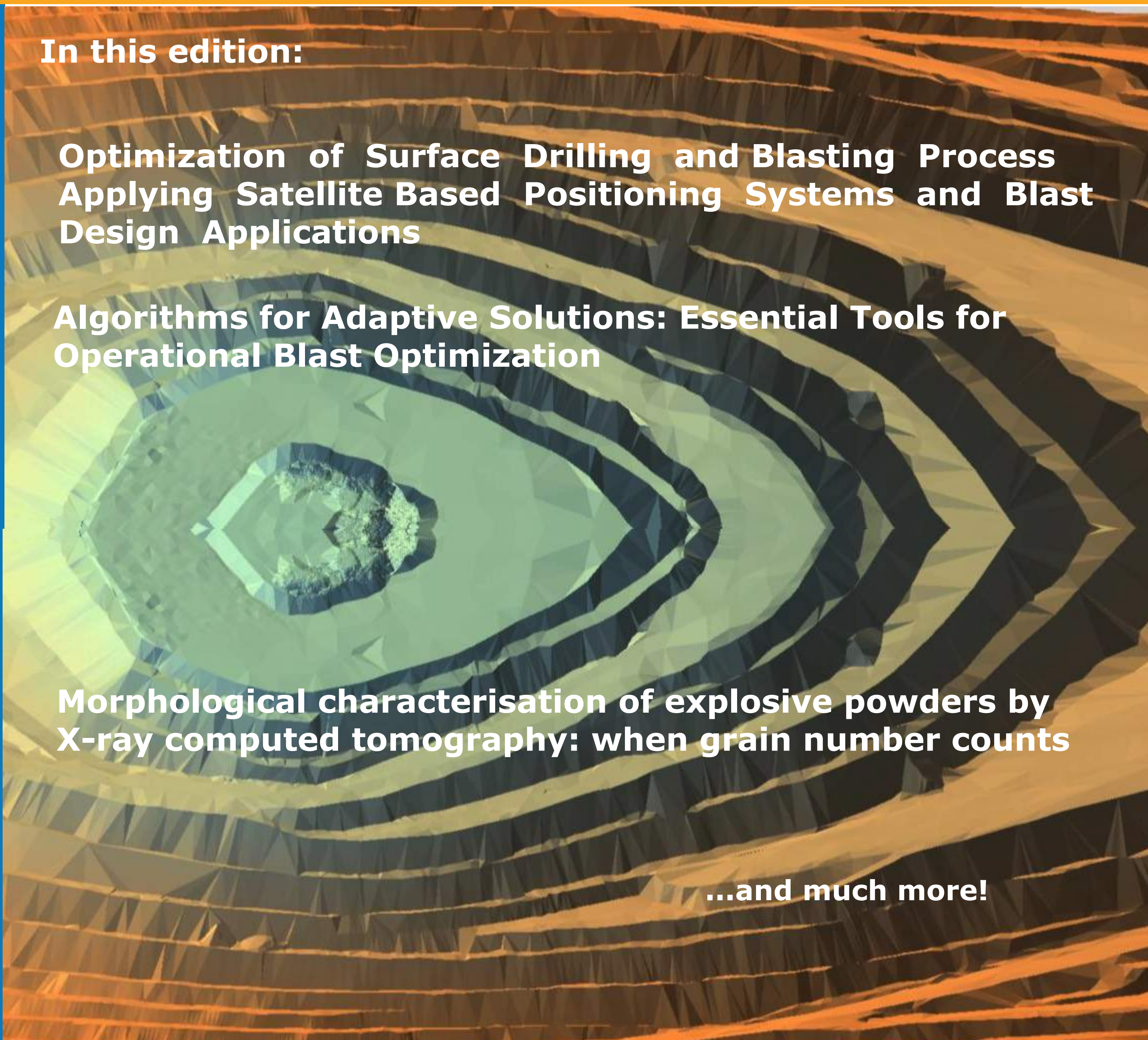
EFEE

NEWSLETTER



In this edition:

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...and much more!

NEWSLETTER

September 2019

We in EFEE hope you will enjoy the present EFEE-Newsletter. The next edition will be published in November 2019. Please feel free to contact the EFEE secretariat or write to newsletter@efee.eu in case:

- You have a story you want to bring in the Newsletter
- You have a future event for the next EFEE Newsletter upcoming events list
- You want to advertise in an upcoming Newsletter edition

or any other matter.

Doru Anghelache, Chairman of the Newsletter Committee and the Vice President of EFEE

and Teele Tuuna, Editor of EFEE Newsletter - newsletter@efee.eu

Dear EFEE members, the President´s voice

I wish to welcome warmly those of you to Helsinki, Finland who are now reading a hard copy of this EFEE newsletter. This is a conference issue for the 10th EFEE World Conference on Explosives and Blasting in Helsinki, Finland. Conference delegates will be handed a hard copy and others will receive the online magazine as usual.

I am writing this foreword mid-August and registration for the conference is well on its way. Numbers indicate that we should expect approximately 10% more delegates compared to previous conference held in Stockholm 2017 – where we had a record audience. Most events will probably be sold out. The exhibition area is also more or less sold out despite the fact that we have reserved more room for exhibitors than ever. This gives me reason to believe that we will have a very successful conference.

I would like to share some info on my home country especially to those of you attending the conference. Finland is a small nation but we have several reasons to be happy and proud of our country. As you may have heard, Finland has been noted to be the happiest country in the world twice in a row by a UN report. Here are some reasons for it.

Finland is one of the largest countries in Europe, approximately same size as Germany, and covers an area of 338 449 km^2 giving our population of 5,5 million an average density of 18 persons / km^2 - approximately 8% of the population density in Germany. Finland is not growded which makes us happy. We do not have major rush hours and we have lots of nature and forests, pure and clean, including thousands of lakes, a vast archipelago by the coast and a beautiful unspoiled and unpopulated tundra area in the north, allowing us to enjoy all aspects of nature and outdoors in all seasons. Nature also provides us with fresh and healthy food which will be served to our delegates also!

Finland has been noted also to be the most free and safe country in the world several times. Finland has a well functioning democracy and there is hardly any corruption here. Education is said to be also top of the world and this is the reason why Finland is one of the most advanced nations in the field of several technologies – including mining and excavation technique and technology. We have to be able to perform our blasting, mining and excavation operations in an effective but still safe and sustainable way. Among others, there are several technical papers presented and technology suppliers from Finland exhibiting in our conference, I hope you will have a chance to hear and meet them and exchange ideas!

Finns are very reliable and punctual people. We take pride in keeping to our word and schedules - always. All this makes living in Finland easy, well organized and fun, I hope you will find our conference and your visit also easy, well organized and fun. In addition I hope that you have a couple of extra days to discover our country outside the capital and the conference centre as well!

I am looking forward to meeting many of you here in Helsinki on 15-18th of September, so that I could welcome you to Finland in person! Those of you who could not make it this time to Helsinki I am hoping to see in Romania, Bucharest in 2021 for the 11th EFEE World Conference on 12-14th of September.

I would also like to take this opportunity to thank all of you who have worked hard for this conference at EFEE, Tyler Events and INFRA and of course especially all our conference sponsors and exhibitors – we couldn't make it without Your contribution, Thank You all!

I warmly welcome You to Finland and the 10th EFEE conference and I hope that you enjoy both!

Jari Honkanen, President of EFEE

"Inspection of conference venues in progress"

Optimization of Surface Drilling and Blasting Process Applying Satellite Based Positioning Systems and Blast Design Applications

ABSTRACT

This paper provides an overview of some economical and safety issues which surface drill&blast managers meet during their rock processing processes. Following topics are under discussion:

-Risk drilling into explosives
-Controlling flyrock
-Lowering vibration levels by advanced drill&blast design
-Improving floor quality and drainage management
-Decreasing surveying and drill pattern set-out costs
-Getting most out of explosives by optimized hole size and drillpattern

New features on drill rigs as well as utilizing satellite based positioning systems, 3D-visualization and blast design applications can substantially improve safety and profitability in rock processing. This article is based on: Jouko Salonen's and Jarmo Kriikka's paper in Malaysian Quarry Congress (QTEG) 2014: Economy and Safety Effects of Satellite Based Positioning Systems and Blast Design Applications in Surface Drill&Blast operations.

Risk of drilling into explosives

Risk of actually drilling into an undetonated explosive charge while drilling is reasonably low. Since the consequences can be fatal, the risk has to be taken very seriously. On terrain benches, the risk occurs basically when drilling close to an already charged drill-hole or a hole that is currently being charged i.e. not a charge that was supposed to have detonated previously. One exception is an undetonated charge in the last row of an earlier blast. The most obvious reason for this is that the detonators were not connected in the first place or they got disconnected somehow. Now we face a case where we do not only have a risk of drilling into an explosive charge but also into an undetonated detonator.

When charging takes place, a shot-firer sometimes encounters a hole that can not be charged. If a drill rig is reasonably near, it can be driven to this hole and clear it or just drill a new hole. Drilling errors, such as incorrect inclination or direction, deviation etc. are dependent on both the drill rig and the operator. There can be a lot of variation as to drill path orientation - but very often there is a tendency for error towards a spesific direction. On one site (railroad construction in Finland) it was clearly observed that rig "number one" suffered from a deviation to the right and rig "number two" forwards. If it is necessary to drive a drill rig close to charged holes - it should be the same rig and operator which were used to drill the original holes. Option number one should be opening (not yet charged) old hole instead of drilling a new one next to it. The location of a rig must also be the same as previously, if the new hole will be drilled. Still, a safety distance must be applied and a rig should have either a remote control or an armoured cabin.

Drilling accuracy is considered to be very good if the maximum error is less than 3 % of a bench height. The problem with this value from a safety and economical aspect is that this value should decrease when a bench gets higher. As a rule of thumb for top hammer drilling we recommend not to drill benches higher than 15 meters. It is normally most economical to only use one bench instead of two - but place strong effort in drilling accuracy, typically by drilling with reduced feed pressure and using guide tubes etc. if the total height of a cut is maximum 20 meters. According to measurements, drilling errors are sometimes more than 3 meters for a typical case with bench height 15 m and hole size 76-89 mm, this equals 20 % of the bench height and "mission impossible" for a shot-firer. The following table illustrates recommendations for the minimum distances from a hole being drilled to a charged hole.

Length of a drill-hole l_d [m]	Minimum distance between drilling and charged explosive [m]
< 6 m	2
6 – 12 m	3
12 – 16 m	4
16 – 20 m	5
> 20 m	safe distance but always > 5

Table 1. Shortest distances between drilling and charged explosives according to Finnish Center for Occupational Health and Safety (Työturvallisuuskeskus TTK) [2]

When comparing these recommendations to normal top hammer drill patterns it is obvious that it is not allowed to drill a new hole behind a charged one. This is due to the fact that in most cases the burden is smaller than the safety limit.

When there is a need for a new hole next to charged holes, a drill&blast supervisor has to consider the following issues:

- Assumed drilling accuracy towards the already drilled holes
- Time required for bringing a rig there and back
- How much is the charging work (and drilling elsewhere) disturbed by re-drilling?
- How long will the blast be delayed and what else will be disturbed?
- What will happen if we blast without one hole? (floor humps, oversize, flyrock)
- Can we move the "back-row" forward? (i.e. not charge all the planned holes – see Fig. 1)

If the bench height is higher than 12 meters the risk of drilling a new hole when neighboring hole is charged is normally considered "higher" than the consequences of blasting without one or two holes even if the missing/uncharged hole is in the middle of the blast. Loading operator must be informed of uncharged holes in a blast.

Hole diameter, ø [mm]	Typical burden, B [m]	Typical spacing, S [m]
38	1.0	1.3
51	1.5	2.0
76	2.3	2.7
89	2.6	3.0
102	3.0	3.5

Table 2. Typical hole size (top hammer) and a drill pattern combinations for blasting granites.

According to safety and economical factor the best – or least bad – solution is chosen. A good shotfirer charges holes in such an order that it is possible to stop charging any time and connect surface delays, put blasting mats on if needed and blast those holes. This is very important and makes it easy to react when anything unexpected happens.

Drill&blast manager follows certain issues, when operating on bench:

- Muck-pile:
  - Is fragmentation uniform or do we see more over-size boulders or less throw in some areas?
  - Is loading more difficult in some areas? Or is the whole blast more difficult to load than normally?
- Floor humps:
  - A floor hump is a fault. Reasons for this can be:
    - Undetonated explosives. This means high risk for undetonated charges and a detonator still being in the hole.
    - Sub-drilling is too short.
    - Rather bad drilling errors. – See Fig. 14.
    - Dead-pressing / deflagration of explosives.
    - Powder factor too low.
    - Hole(s) charged.
- Removing the prior sub-drill zone:
  - Can we drill through the sub-drill zone? Are holes chargeable?
    - Where to place primers?
  - Undetonated explosives
    - Is it safe to drill through the sub-drill zone?

According to Vegard Olsen et al. propability of fatal accidents is ~100 % higher in operations where prior sub-drill zone is removed before drilling the next bench compared to operations where drilling is done through the prior sub-drill zone [3].

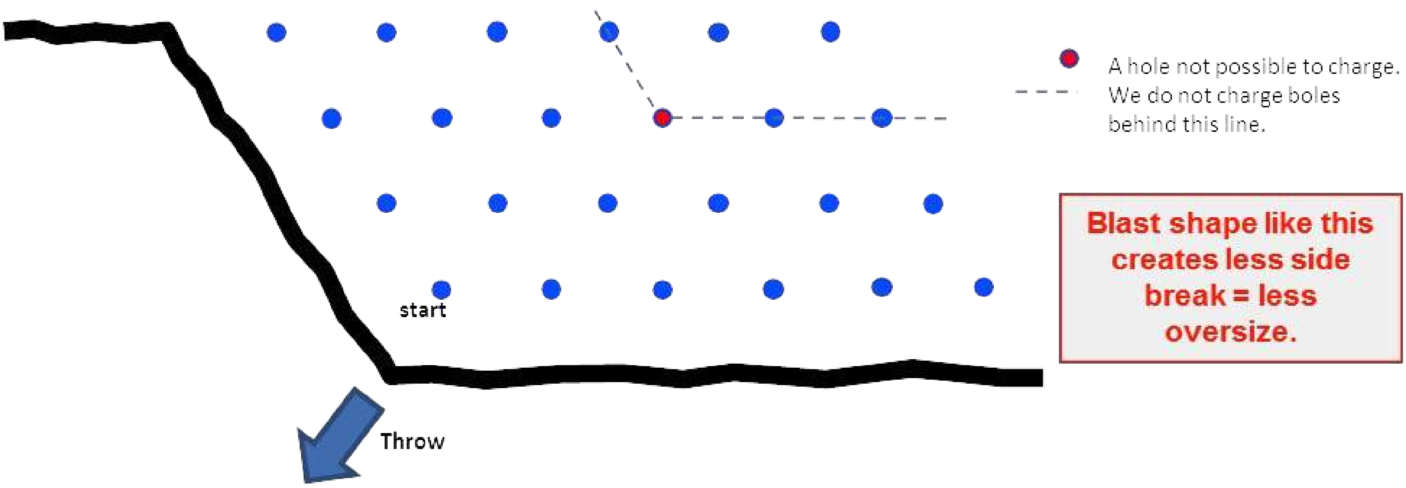

Fig 1. One option is to make a new "back-row".

The primer should be placed just above sub-drill. Risk is minimized by using only bulk explosives in sub-drill area. The drilling operator can be sure of that he/she is never going to drill into neither a detonator nor a primer, if loading reached the right level. It is also a fact that when blasting - having a det and a primer on a floor level - power (released energy / time) will be 100 % higher in a length of 2*sub-drill [5].

It is easy to make a drill plan where starting points of holes are located between the hole bottoms from a bench above, when a digital record of drilling exists. By hole deviation measurements we find out if there is a tendency for certain errors. Combining this information with drilling records helps us to make safe and economical drill plans. Collaring a hole is much easier and faster to do when we do not have to drill where the rock is most damaged. We achieve more drillmeters per shift and use less time for charging. Hole measuring is most beneficial in first and last rows.

Fig 2. Plastic tubes inserted to keep holes open – and thus chargeable.

With a modern blast design software it is possible to design drill pattern so that risk of drilling into previous holes is minimized. Use of this kind of modern tool should be supported by applying accurate GPS positioning and 3D modelling. It is possible to design accurate place and shape for drill plan and optimize burden and spacing from safety and fragmentation point of view, when 3D model has been generated. It is possible to clean previous end points and define critical distance between new and "old" hole in order to avoid risk of drilling into undetonated explosives in previous sub-drill.

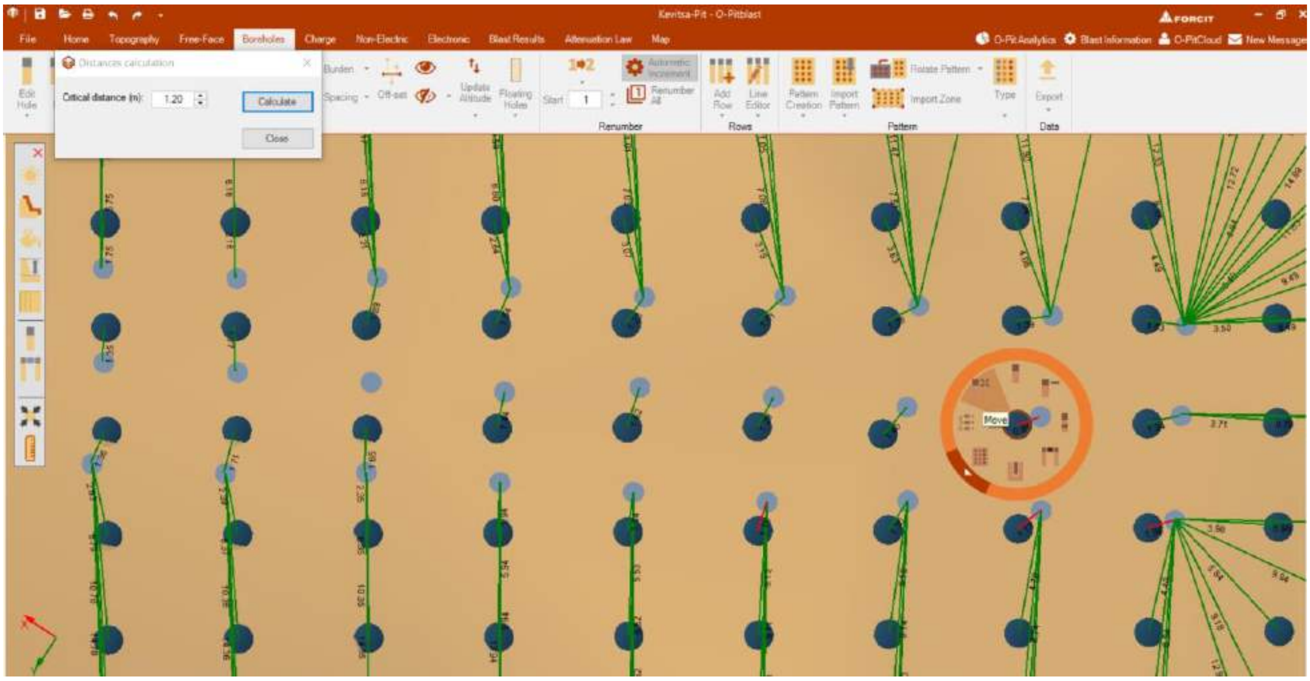

Fig 3 a. Light blue holes are endpoints from previous bench. Blasting engineer can define critical distance between old and new holes and make adjustments when doing the blast design (O-Pitblast Lda & Oy Forcit Ab).

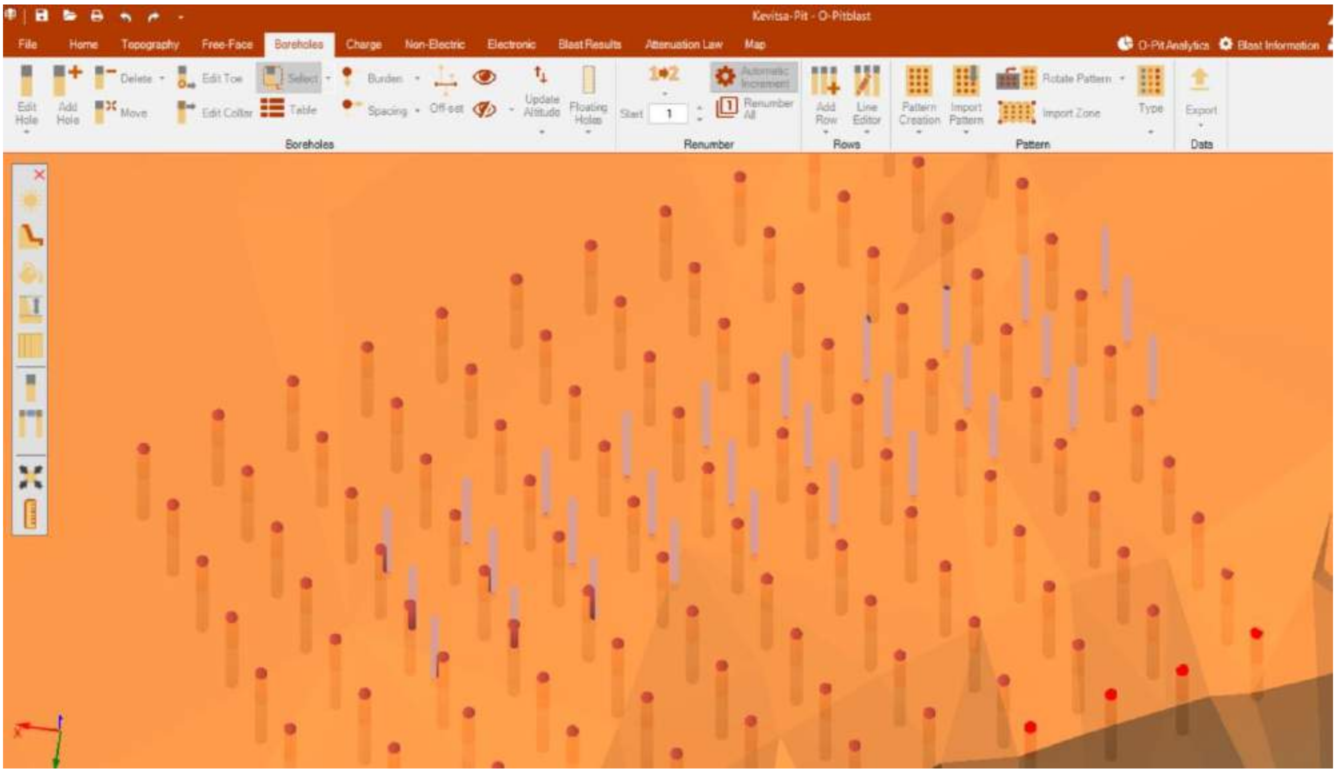

Fig 3 b. 3D modelling picture from Fig 3 a (O-Pitblast Lda & Oy Forcit Ab).

Flyrock

Allan Richards et al. presents a formula for calculating a maximum flyrock distance [8]:

$$L_{max} = \left(\frac{k^2}{9.8}\right) * \left(\frac{\sqrt{Q_1}}{B}\right)^{2.6}$$

where k = *constant*, for granite 27
Q_1 = linear charge [kg/m]
B = burden [m]
L_{max} = maximum flyrock distance [m]

According to the formula, maximum flyrock distance L_{max} is:

Linear charge		Burden B [m]							
Q_1 [kg/m]	k	0.5	0.8	1	1.5	2	2.5	3	4
0.125	27	**30**	**9**	**5**					
0.5	27	**183**	**54**	**30**	**11**				
1	27	**451**	**133**	**74**	**26**	**12**			
2	27	**1110**	**327**	**183**	**64**	**30**	**17**		
3	27	**1881**	**554**	**310**	**108**	**51**	**29**	**18**	
4	27	**2734**	**806**	**451**	**157**	**74**	**42**	**26**	**12**
5	27	**3655**	**1077**	**603**	**210**	**99**	**56**	**35**	**16**
10	27	**8999**	**2651**	**1484**	**517**	**245**	**137**	**85**	**40**

Table 3. Maximum flyrock distance L_{max} in meters.

Flyrock can be divided into four types:

1. Too short burden in the front row (face burst). *Additional flyrock "driven by" floor humps or poor back walls – see Fig. 5b*
2. Too small uncharged height and/or tight row timing (cratering)
3. Unsuitable stemming material or too short stemming (rifling)
4. Tendency to over-charge floor humps and boulders

"Highly jointed rock" is combined with number 1 or 2 as a reason in most accidents.

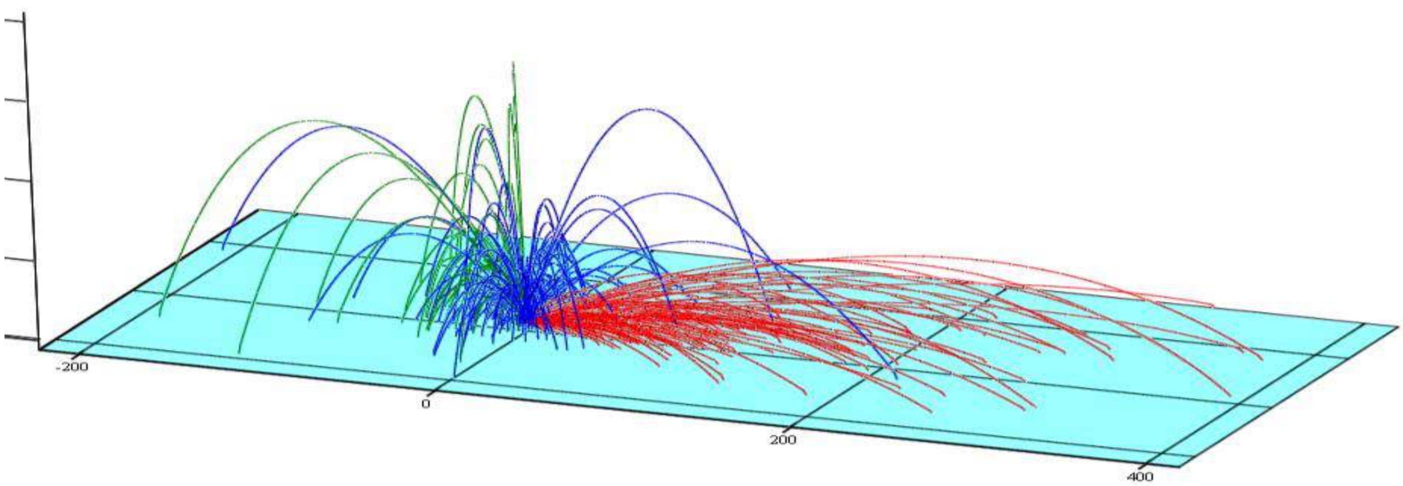

Fig 4. Direction of flyrock. [4]

In rifling, one critical question is how wet is the stemming material? The minimum height for stemming should be increased by ~four times diameter of a hole, if water occurs in un-charged portion of hole.

When there has been a previous blast in front of a new one, drill&blast supervisor must ask himself: "Is powder factor in the first row to be smaller, equal or bigger than powder factor in following rows? See some comments from experienced supervisors:

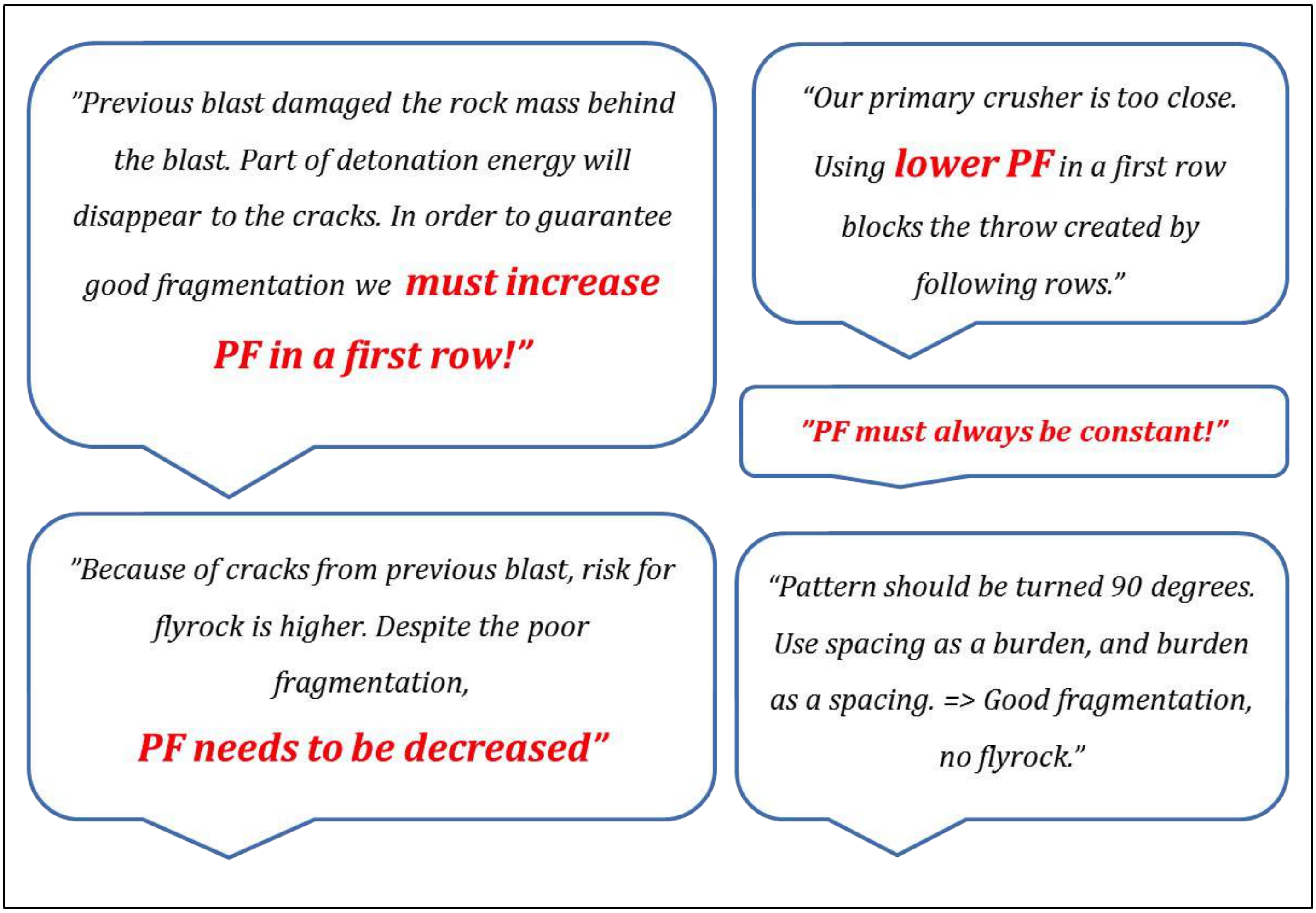

Fig 5. Relationship of burden and linear charge [kg/m] in a first row. One is absolutely not correct!

Higher powder factor might help with shotrock fragmentation but the risk for flyrock will increase. If you want to use a lower powder factor - it is wise to use a standard drill pattern but drilled with worn bits. The linear charge [kg/m] of a hole drilled with worn out bit can be up to 25 % smaller compared to a new bit.

Decreasing inclination is not recommended. Then burden would be longer than normal and sub-drill must be increased in order to avoid floor humps. One option is just to increase burden, especially when drilling takes place before previous blast. Increase of 0.4 m (Ø64 mm) to 1.2 m (Ø152 mm) in first row burden is normally applied. Optimal powder factor is not a constant value based only on rock properties, it is proportional to hole size and objects near the blasting etc. [5].

Using a burden as a spacing and vice versa may cause a floor hump since burden is too long or dead- pressing due to neighboring holes being too close by.

Too short burden in the first row

Explosives detonating too close to a free face is the most common reason for flyrock. Consequences of that are also the most critical ones. Flyrock occurs when powder factor in certain area is too high. Possible reasons for that are:

1. Small burden – especially combined with a toe (see Fig. 6b)
   - Back break from the previous blast
   - Wrong starting point of the hole
   - Wrong inclination of the hole
   - Hole deflection
2. Too big linear charge [kg/m]
   - (Bulk) explosive in a void
   - Too hard tamping of cartridges by a shotfirer
3. Blast not covered properly
   - Some shotrock in front of the blast
   - Blasting mats
   - Gap between shotrock and mats
4. Unwanted detonation because of detonation transmitting from the other hole

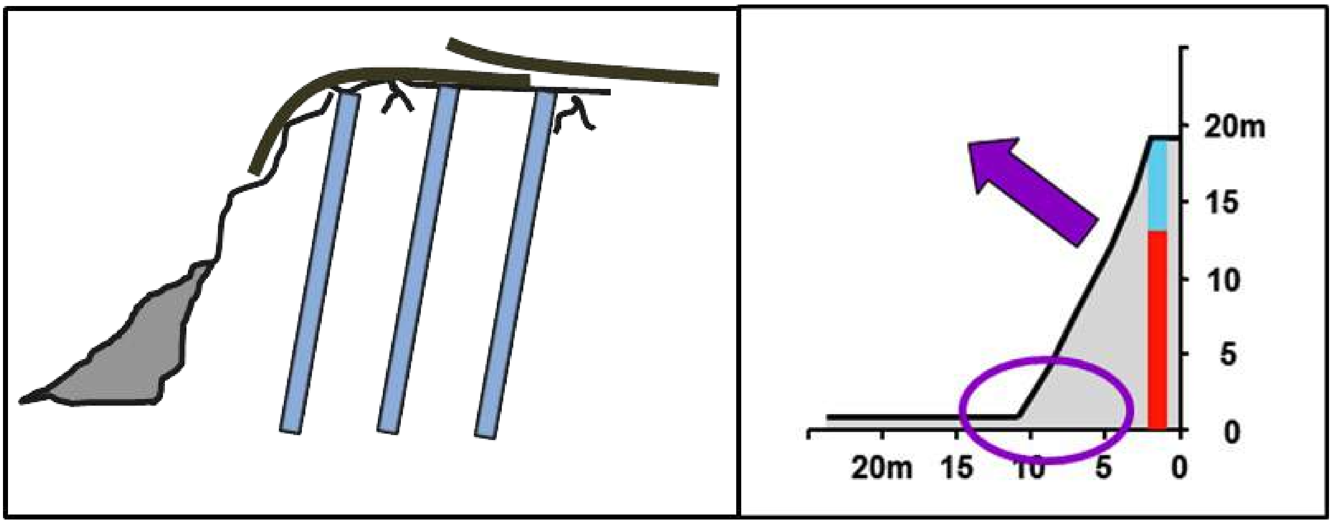

Fig 6. Shotrock and mats to avoid flyrock. A gap between them is dangerous (a). Too long burden on bottom (because of a toe) forces gases to find another way out – this creates a lot of flyrock (b).

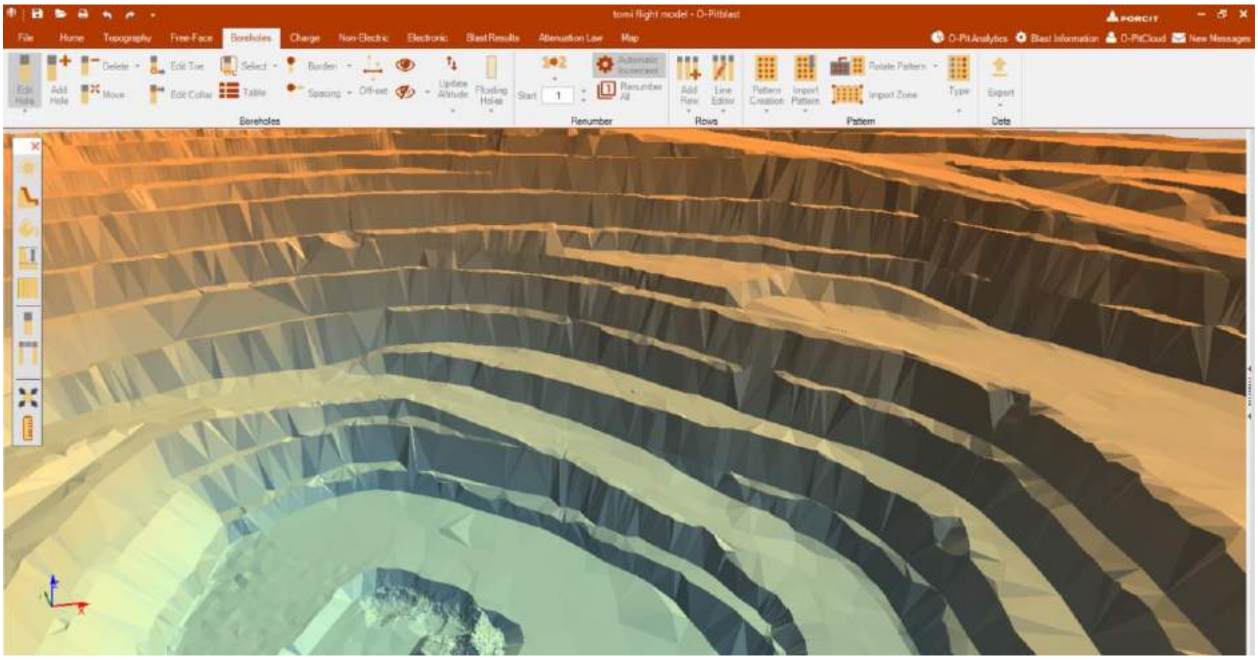

Fig 7a. Scanned face or 3D model gives enough info for economical and safe drill plan designs.

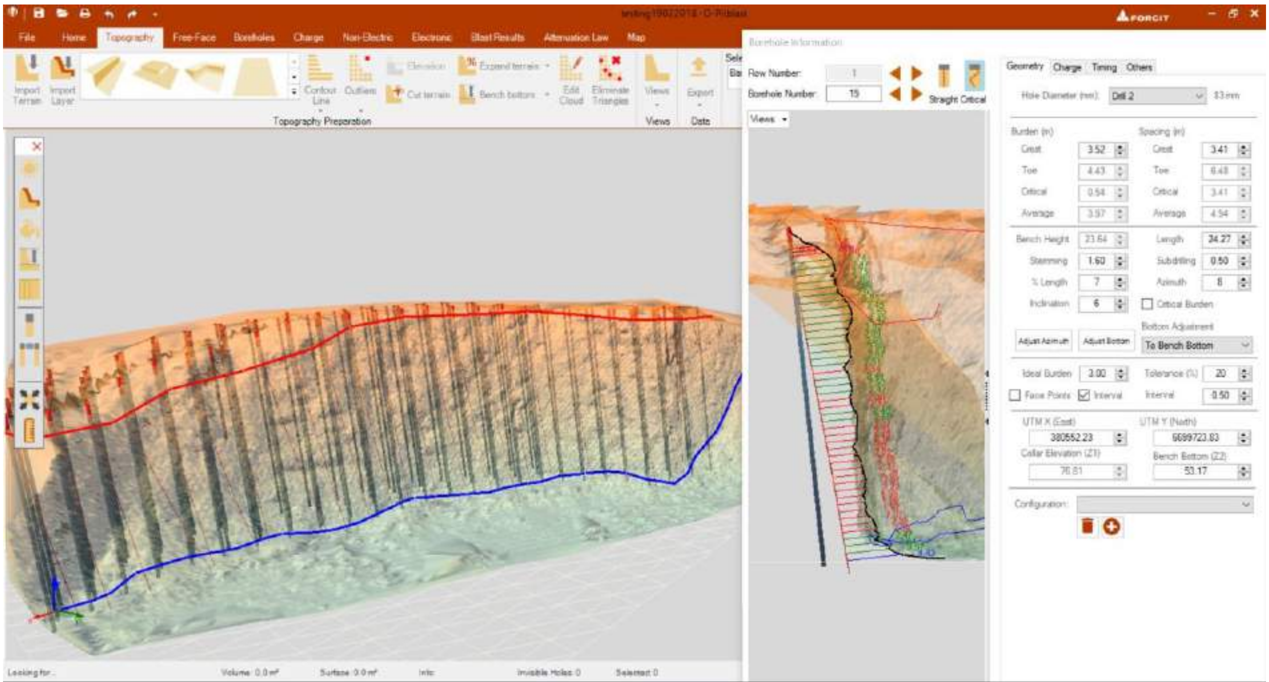

Fig 7b. Real burden and other safety & quality aspects can be evaluated from 3D model after drilling and hole deviation measurements.

BACK TO TOP

Drill rigs have to be reasonably well maintained to avoid having excessive drill hole deviation due to wear and slack. In case of inexperienced operators, rather new and well maintained rigs and GPS devices (like 3D-TIM or Hole Navigation System) are required. More experienced operators are capable of high quality drilling even with older rigs without all electronic devices as long as drill steel is chosen according to bench height and geological rock properties. Quality of drilling should be controlled regularly by deviation measurements.

Scanning a face makes it easy to plan hole positioning in a way that the correct powder factor is achieved – resulting in safe blasting and good shotrock fragmentation. Planning software like O-Pit Blast, Driller's Office and ROC Manager are highly applicable. In some applications like road cuts, scanning cannot always be done. Because of tight schedules and economical pressures we tend to drill even several blasts beforehand. Documented drilling data from previous and present blasts enables high quality blast planning. Experience of a shot-firer plays an important role especially when charging holes in the first row. It is crucial to create a working culture where everyone understands his/her part in the process and where everyone feels responsible not only for one's own work, but for the effort of the whole group.

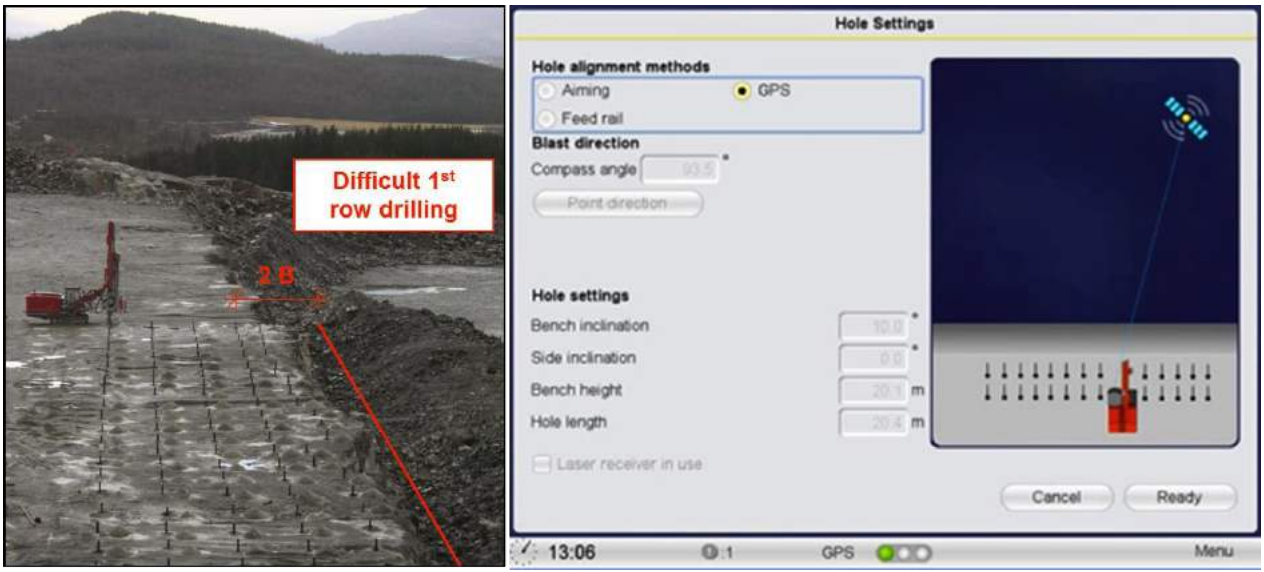

Fig 8. Second row is exactly at the distance of two times burden from the last row of the previous blast. The first row was drilled in order to achieve a safe and effective burden for each hole.

Drilling as a part of vibration control

Decreasing bench height, using smaller hole diameters and going for tighter drill patterns are some of the best tools which enable a shotfirer to reach the allowed level of vibrations.

Good choices in drilling combined with usage of an advanced blast planning software like O-Pit Blast makes vibration management easier.

When drilling accuracy is poor due to limited operator skills, old or poorly maintained rigs, incorrect drill steel or geological reasons, we face serious problems as shown in a graph below:

1. First hole is straight but the operator was afraid of too small burden. He drilled the hole little further from the face and decreased the inclination.
2. Hole started well but because of geological reasons or increasing feed pressure or feed foot slippage, it deflected all the way to the first hole.
3. There is a small error in inclination. This error combined with error in hole number 2, gives us a burden approximately twice as long as planned.
4. Perfect drilling.

Let us assume that the shot-firer used a firing system where instantaneous charge weight equals the hole charge. Drilling error caused the instantaneous charge to be double of the planned one. The vibration will be higher. It was also very possible that the pressure from the detonation in hole number 1 made explosives in hole number 2 dead pressed. That means it will not detonate. The charge would just have deflagrated, if it was almost dead pressed. Some explosives recover from dead pressing, some don't.

Burden for hole number 3 is far too long. Powder factor is too low and rock will not be sufficiently fractured. This results in a floor hump. Even though drilling and charging was perfect after third row, rock will not move properly. The drilling error caused higher vibration and poor liberation.

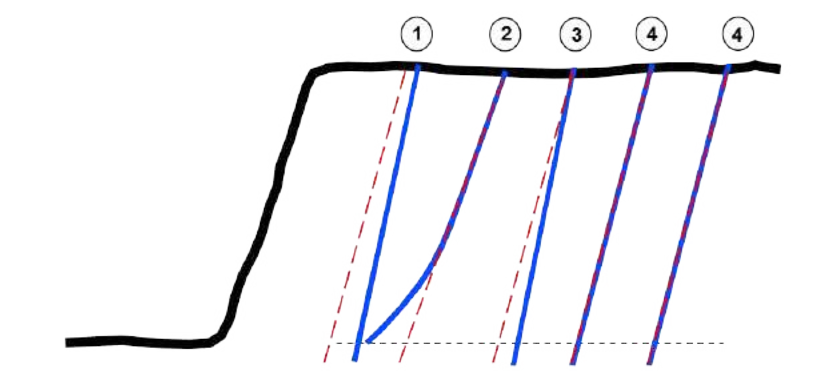

Fig 9. Drilling errors and vibrations.

It is rather common to evaluate PPV by the formula [6]:

$$v_{\max} = k \cdot \sqrt{\left(\frac{Q_{mom}}{R^{1.5}}\right)}$$

First hole detonates as planned but because second hole is touching the first one, hole number two detonates also. If rock was highly jointed, the second hole could have detonated even if it was not so close. Powder factor here is much higher than planned. Because the burden is longer than normal, risk of flyrock is not very high.

where
V_{max} = peak particle velocity (PPV) [mm/s]
k = transmission factor
Q_{mom} = instantaneous charge [kg]
R = distance [m]

Example:

k	R [m]	Q_{mom} [kg]	V_{max} [mm/s]	difference
50	100	20	7.07	
50	100	40	10.00	41 %

Table 4.

Percentual difference in PPV is not affected by transmission factor, distance or hole charge, but values presented are typical ones.

When the quality of drilling reaches the level where drilling depth and pattern are constant, optimal fragmentation can be achieved with reasonably low powder factor. This means smaller hole charge, smaller instantaneous charge and lower vibration levels. Decreasing the total amount of explosives per blast also results in lower vibrations. When going down in PF, importance of delays gets more important.

Some of the modern blast design software provide opportunity to import vibration data from seismographs or automatically from vibration data providers systems like Vipnordic. Integration between blast design software and vibration provider combined with GPS navigation on drill rigs can result in very accurate and detailed blast designs. This opportunity provides possibility to blast bigger volumes using smaller instantaneous charge. The fewer blasts required on construction site, the more cost efficient the project is for everyone working on the site. Critical blasting zones can be analysed in blast design software by calculating attenuation laws based on information from previous blasts in the same area.

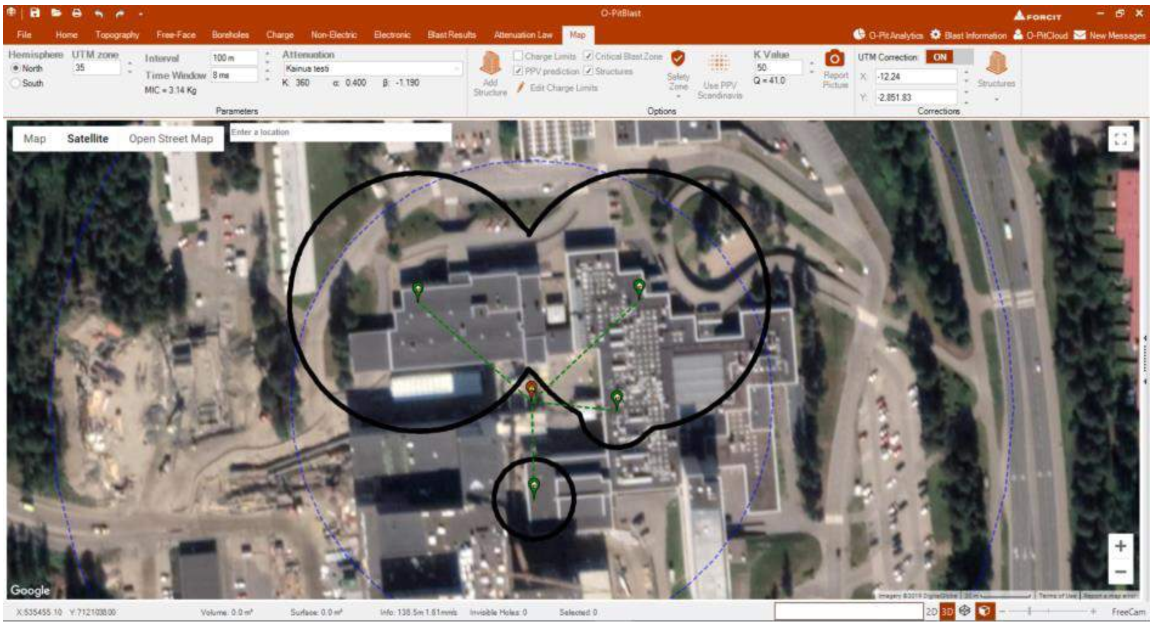

Fig 10. Blast design in a very sensitive hospital environment (O- Pitblast Lda & Oy Forcit Ab).

Drilling data can be transferred to drill rig via IREDES format and export it back to software after drilling. Safe and economical charging of the holes can be performed based on accurate drill hole positioning, hole deviation information and vibration history from the site.

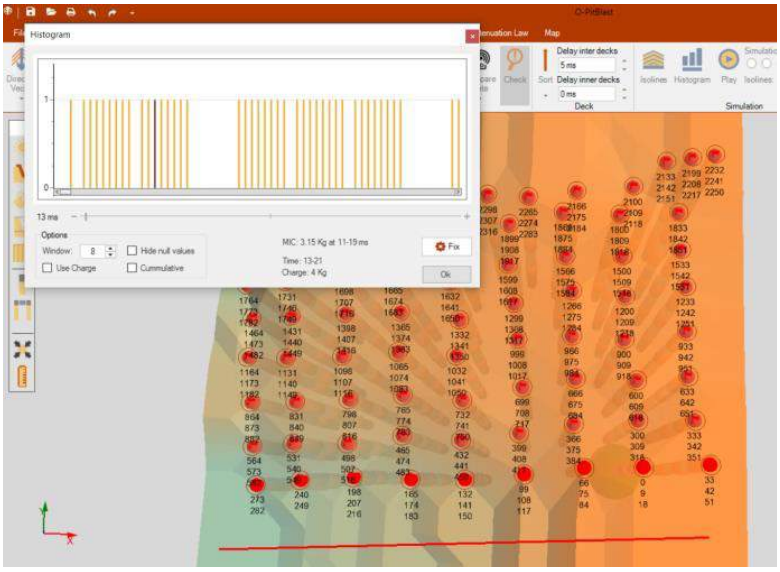

Fig 11. Instantaneous Charge analysis in Blast design software *(O-Pitblast Lda & Oy Forcit Ab).*

Fig 12. Floor hump.

Drilling accuracy and quarry floor

Working site reveals how well the pit process is managed. If the floor and roads are rugged, problem(s) exist most probably in either:

1. Drilling
2. Blasting or
3. Loading

Uneven floors increase costs:

1. Moving and drilling is difficult. There is more drilling for secondary blasting. Capacity is lower.
2. Floor humps require extra blasting or hydraulic breaker work.
3. Charging work is more difficult and slower. Explosive trucks drive slower and cannot reach every point within the blast pattern.
4. Loading of shotrock is difficult which means higher fuel consumption, more maintenance, more wear parts and low loading capacity.
5. Haul roads result in higher fuel consumption and tire costs not to mention lower driving speed/capacity.

The best way to achieve an even quarry floor is high quality drilling. Drilling errors are not acceptable, but the most critical issue is to assure planned depth of holes. Experienced operators can handle this without all electronic devices. Others benefit greatly from laser levels and GPS based positioning systems. Drilling to the right depth is not enough though. An operator has to leave every single hole for a shot-firer in a chargeable condition.

Normally highest economical bench height for top-hammer drilling is between 12 and 16 meters. High benches cause problems with shotrock loading and crushing costs, not to speak of safety issues. Only when geological conditions are very easy and drilling operators are good, can little higher benches be used.

Fig 13. An operator has run out of collar plugs. When raining, cuttings will flow into holes causing floor humps due to too little effective sub-drill.

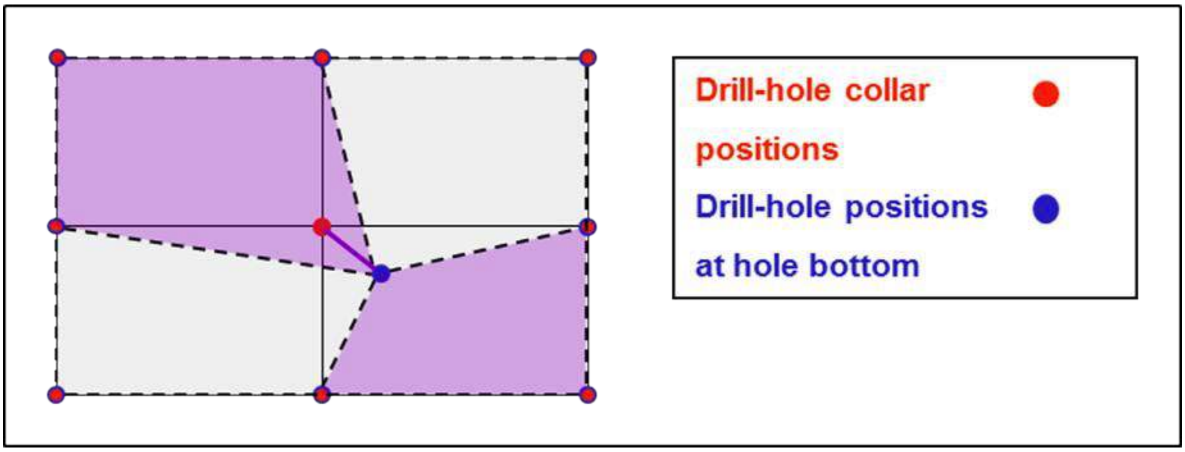

Fig 14. Drilling error only in one hole changes the drill pattern in a bottom. Tighter drill pattern makes finer fragmentation. Wider pattern means coarser fragmentation and a risk for floor humps.

When seeing is limited due to darkness, fog, rain etc. GPS device on a drill rig helps reach the required level in drilling accuracy. Accurate drilling and skillful charging make optimizing the sub-drilling possible. There is always a risk in decreasing the sub-drill. But too long sub-drill causes lot of extra costs due to unnecessary drilling and charging work, consumption of explosives and more difficult drilling and charging on the next level. All this makes optimizing the sub-drill so important economically. [6]

Dewatering is required in many quarries, also because it makes excavation easier. Applying GPS and intelligent drilling together with a drill&blast planning software (O-Pit Blast, Drillers Office etc.) makes it very easy to excavate inclined floor levels. By this, dewatering is handled very economically without any extra work. [5,6]

Surveying

There is lot of surveying work at a drilling and blasting site. It is important because it gives data for decision making. This data helps us to plan, manage and control our processes. Economic issues are always based on measured data. Even though surveying is expensive, it is of paramount importance.

The blasting supervisor goes to see the muckpile after every blast to observe blastinf results visually. Any differences in fragmentation and throw should be noted and evaluated. Be happy for good results – but remember that these must also be evaluated – and not just bad results. Further actions must be made and loading operators informed If there is a suspicion of misfires.

When drill plans are made by software, it should be based on an accurate 3-D model. Surveyor can make a model with a total station (tachymeter) or a scanner. Drones are very often effectively applied in this purpose. After that making a drill&blast plan is easy. Modern drill rigs with a GSM device produce surveying data automatically while drilling. This data is easy to utilize in planning and defining of drilled or blasted volumes. The labor cost for a professional surveyor is rather high. It is a great benefit if even a little part of this work can be done automatically by a drill rig.

Traditionally a drilling operator used to mark the location of each hole on the bench. A surveyor can do the same job for example with a total station. The marking task described above no longer exists when the drill plan is made with software and handled with a drill rig. Overall drilling capacity increases - which means savings in drilling time and costs. A rig "knows" where to drill and guides the operator. The rig even shows which holes can be drilled from its current position. This feature helps an inexperienced operator a lot, especially when using a rig with revolving superstructure capable of reaching for example 3 – 8 holes from one position. If a certain hole cannot be collared from a planned position, an operator can change the starting point a little and the rig helps to align the hole so that the planned hole bottom is reached. This is easy for an operator and good for the process.

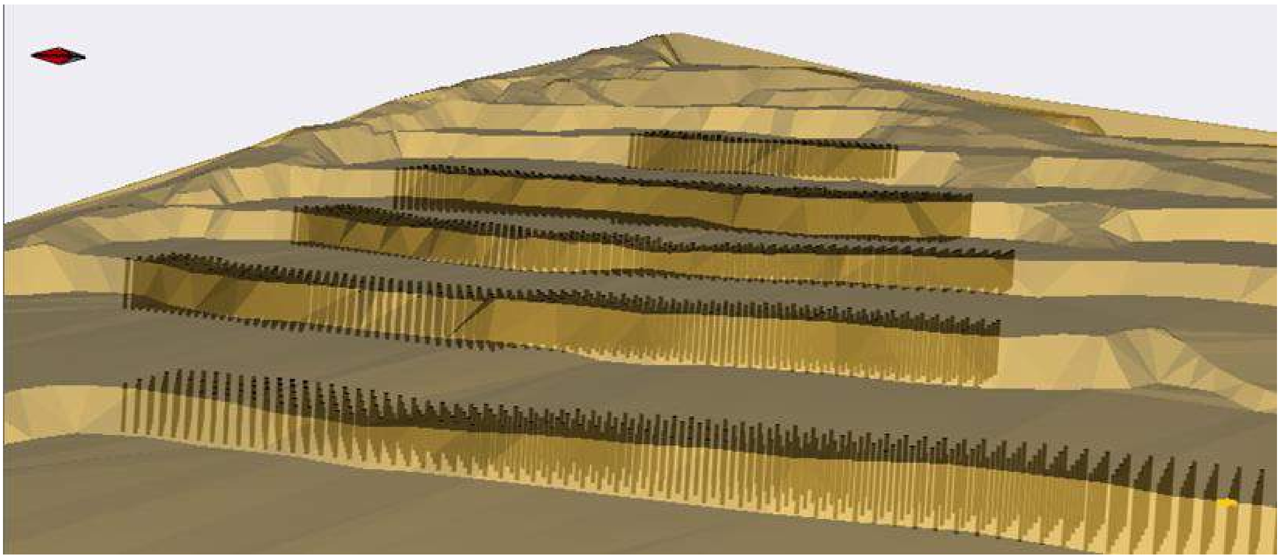

Fig 15. 3D-model mostly based on data from drill rig.

Quality reports concerning drilling include rock volumes. Then we know how many cubic meters or tons were just blasted or how many cubic meters is our drilling worth. This makes it easy to control schedules and invoicing.

Hole size and drill

Selection of correct hole size and drill pattern is an economical optimization process. There is a certain relationship between these two values. See Figure below. All points there are from real cases and correct as such. The line in a picture stands for an average, not for a correct combination. We enlarge the drill pattern when rock is easy to blast, We decrease the drill pattern, if it is difficult to blast. The drill pattern must also be decreased, if we want finer shotrock fragmentation.

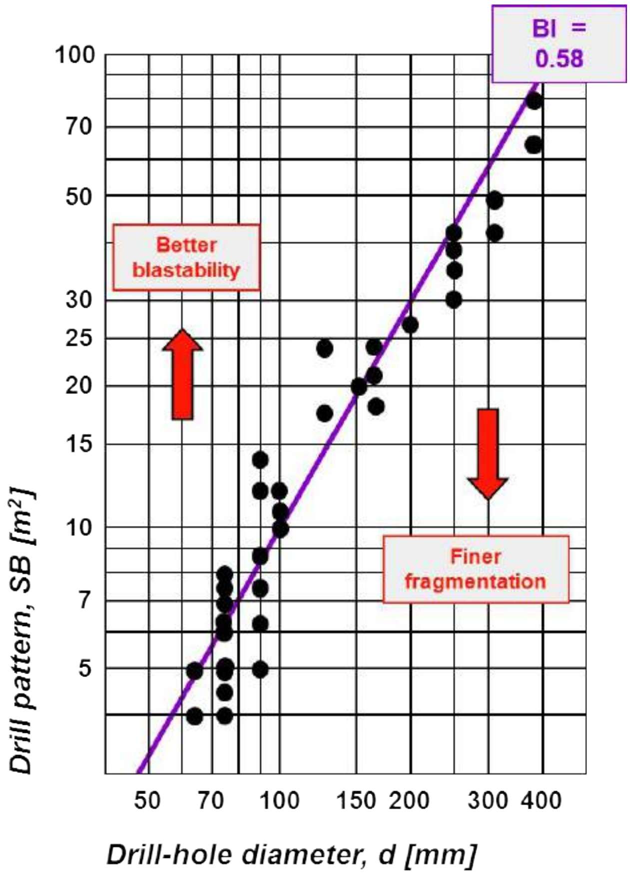

Fig 16. Relationship between hole diameter and drill pattern. [5]

The next graph illustrates that when a staggered pattern is used, we get a more uniform distribution of explosives in the bench than compared to a rectangular pattern. This equals better shotrock fragmentation at no extra cost. Drilling staggered pattern might be too difficult if terrain is rough and operators are not very experienced. It is difficult for a shot-firer to connect holes according to the blast plan, if patterns vary. GPS device does not control the rig but helps the operator to do it and keep the pattern stable.

Energy is often produced by fossil fuels. Explosives energy is relatively cheap energy compared to other energy forms. Investments for drilling and blasting operation can be often saved in later stages such as secondary breaking, loading, hauling and crushing. Even though it is useful to put some extra energy to drilling and blasting process, it does not meant that we could

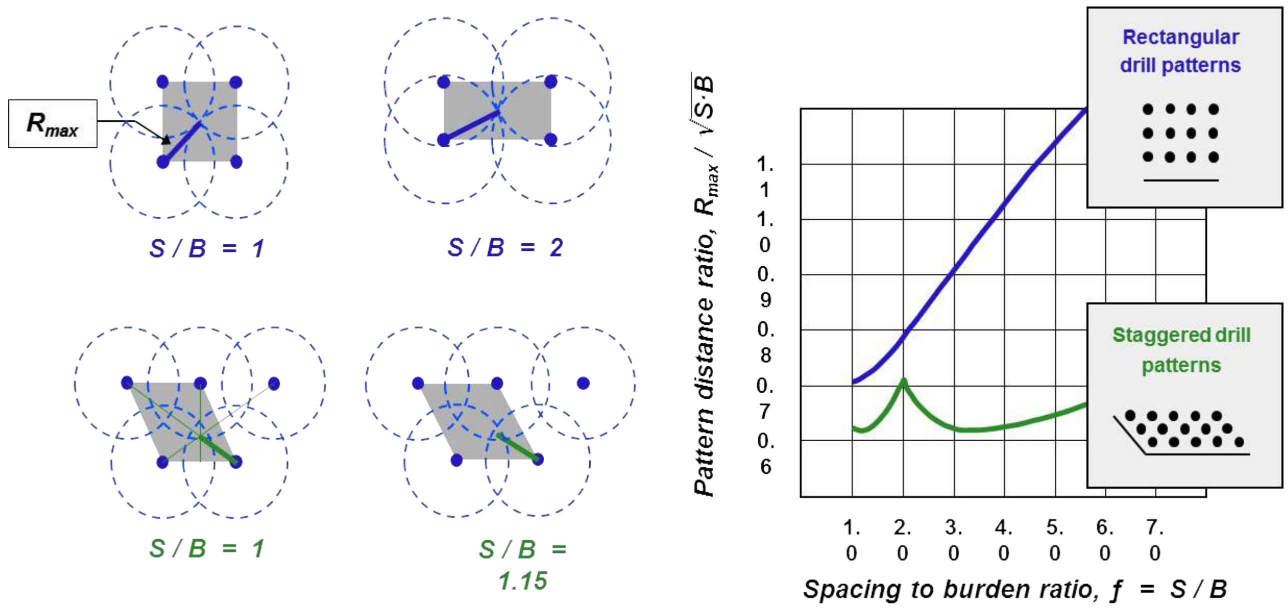

Fig 17. Maximum distance from the middle of the drill pattern to a nearest explosive is smaller when using staggered pattern. [5]

Cost management of drill&blast process

Cost efficiency is as important in drilling and blasting process as in any other business. When analysing drilling and blasting costs it should be remembered that it is always part of a bigger process, such as quarrying, mining or construction operation. Cost analysis should be done from "mine to mil" point of view. Crushing process consumes relatively big amounts of energy.

By using modern tools such as 3D modelling, hole deviation measurements, GPS navigation and blast design software it is possible to save lot of money in rock processing annually. Table below is based on cost level in Nordic countries and several experts opinion about drilling and blasting costs. Table 5 shows how accurate planning and controlling of drill&blast process can result in lot of annual savings. Estimate is based on one drill rig drilling in a quarry in one shift, five days per week.

"old school"	GPS drilling and digital drill planning	GPS Navigation, 3D modelling and **holde deviation control**		70000 drm/a
~1.0	**~0.5**	**~0.3**	m	drilling errors
89	89	89	mm	hole size
15	15	15	m	bench height
6,6	**7,05**	**7,44**	m^2	paCern
99	**105,8**	**111,6**	m^3	volume/hole
1,2	**0,9**	**0,8**	m	subdrill
16,45	**16,15**	**16**	m	hole length
1,8	1,8	1,8	m	uch
6,86	6,86	6,86	kg/m	linear charge
14,65	**14,35**	14,2	m	charged length
100,5	**98,41**	**97,41**	kg	hole charge
1,02	**0,93**	**0,87**	kg/m^3	PF
0,17	**0,15**	**0,14**	drm/m^3	spesific drilling
3	**2,8**	**2,6**	%	oversize percentage, backbreak/first row
2,3	**2**	**1,8**	%	fines percentage (2 €/ton)
3,8	3,8	3,8	€/m	drilling cost per drillmeter
0,631	**0,58**	**0,54**	€/m^3	drilling cost per cubicmeter
0,95	0,95	0,95	€/kg	explosive prize, emulsion
0,964	**0,884**	**0,83**	€/m^3	explosive cost per cubicmeter
0,086	**0,08**	**0,076**	€/m^3	detonators
0,066	**0,062**	**0,06**	€/m^3	charging
0,02	**0,01**	**0,01**	€/m^3	1/0,5/055 floor hump /20 000 m^3
0,135	**0,118**	**0,103**	€/m^3	cost for unmarketable fines (2 €/ton
0,35	**0,327**	**0,32**	€/m^3	secondary breaking
0,016	**0,008**	**0,0053**	€/m^3	scaling 2/1/0,67 h/10000 m^3
0	**0**	**0,075**	€/m^3	Cost of hole deviation measurement (15 holes first row) €/m^3
0,04	**0**	**0**	€/m^3	cost of more difficult crushing (coarser shotrock)
0,012	**0**	**0**	€/m^3	Measuring hole positions 120€/0€/0€ /10000m3
0	**0,005**	**0,005**	€/m^3	Surface modelling ((1h*50€/h)/10000m^3)
0	**0,006**	**0,006**	€/m^3	Drill planning with blast design software
2,32	**2,08**	**2,03**	€/m^3	"Drill&blast related costs"
Savings:	**0,241** €/m^3	**0,29**	**€/m^3**	
	1,587 €/drm	**1,99**	**€/drm**	
	111082 €/year	**139480**	**€/year**	Annual save per 1 rig in 1 shift

Table 5. Cost effect of better control of drilling and blast planning. [9]

Costs and other KPI`s can be followed with good software in real time and managers achieve a good control of the process. KPI`s can be followed in several levels and several relationships can be analysed from individual blast or from the whole project.

Fig 18. Cost follow up in blast design software (O-Pitblast Lda & Oy Forcit Ab).

Jouko Salonen MSc, Blasting Technical Expert, Training Manager - Forcit Consulting BSc

Tomi Kouvonen BSc, CEO - Forcit International

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Global studies of the levels of risk of flyrock

Abstract

After an accident or in the event of a new project, quarries and mines are compelled to estimate the risks induced by their activity in the neighborhood among which flyrock because potentially fatal is surely the most significant.

A model has been developed to estimate risk levels of flyrocks for an entire quarry or public works project. It therefore foresees the variability of geological and geometric blasting parameters within a large volume.

This model is supported by a statistic estimation of the confinement capacity of the rock mass, quantifying the variations of the confinement geometry and of the explosive energy based on audits of the equipment and manpower available to carry out and inspect blasting.

These components converge to determine risk levels, compared to regulation limits. Technical Services of public administration authorities rely on these results to validate the approach presented by the owner and to authorize the resumption of operations.

A blast from a quarry in the South of France was the cause of accidental flyrock to a factory employing several hundred people. As a protective measure, French authorities immediately called a halt to the blasting and thus to quarrying operations. Before activities could be resumed, the authorities required that the quarry owner submit proposals on how to improve blasting operations and mining control processes. Following requests by local residents chiefly concerned, the quarry was especially requested to guarantee a high level of safety for the duration of future operations.

In this article, we shall deal solely with the risk assessment of the overall production. However, the processes relating to blast pattern designs, verification of explosives loading, measurements of the geometry of the blasts, procedures of blasting operations controls, and procedures for processing anomalies or misfires could certainly justify a paper in themselves.

Considering the urgency to provide a rapid response to the authorities in order to resume quarrying operations promptly, there is great temptation to set up flyrock calculation and checking tools for every blast. However, most cases of flyrock that we have had the opportunity to analyze over more than fifteen years result from unusual variations of both the blasting parameters and the rockmass. Risks relating to the overall production of a quarry over a long period can thus not be assessed through calculation methods, however sophisticated they may be, which deal with every single blast on the basis of the nominal characteristics of the geometry, the explosives and the rockmass. Besides, if the extraction organization is not integrated into the assessment process, there is a high risk of accepting blasting situations with abnormally high constraints and thus considerable costs.

Having detailed the hypotheses of the risk assessment model, developed by Egide and which has already been published [1][2], we shall return in more detail to the case of the previous site to examine the implications of the method on the choices of operations.

1. EGIDE FLYROCK MODEL

Flyrock, or 'wild flyrock' if we refer to the terminology used by Little & Blair, corresponds to the propulsion of a rock fragment of varying size over a large distance from the blast, more precisely exceeding the acceptable distance or 'exclusion zone limits' that have been determined or estimated by the blaster. (See reference [3]).

This propulsion depends on the explosive energy used, the geometry of the confining rock mass and the explosive charges as well as the way the rock mass control the explosive detonation. The detonation timing of the different explosive charges used in the blast is also an important factor in the occurrence of flyrock in as far as it is likely to modify the way the explosive charges function and to affect the geometry of the faces developed during the blast dynamics.

Flyrock risk is therefore linked to controlling these different parameters during the entire operation. Explosive energy and geometric blasting parameters seem to be controllable parameters as much as the confinement capacity of the rock mass tends to vary considerably over the term of the project.

The model should determine a risk level to be compared to acceptable thresholds possibly stated in local regulations.

1.1 Blasting plan parameters

Our flyrock investigations inevitably begin by examining the real blasting conditions or prescribed conditions. This includes, not only drilling equipment, the choice of explosives, initiation and geometric parameters, but also methods for inspecting these parameters and the teams' working methods.

The most easily controllable parameters in blasting plans are the explosive energy and the use of delays. On the other hand, even if the height of the benches is generally an easily controlled parameter, it is not the same case for rock thickness around (confining) explosive charges. These varying thicknesses depend on the structure of the massif and on the orientation of the faces within this discontinued volume, on the blasting plan being adapted to these conditions, and also, on the accuracy of the drilling already carried out.

Controlling these variations mainly depends on the level of equipment used to check the burdens for every blast.

Over and above the instruments used to check thicknesses, the human factor remains one of the most important factors in geometry variability, insofar as the operator's care and choice of burden variation which above a certain level a change of explosive charge would need to be envisaged. When carrying out flyrock surveys that lead to an increased awareness of these risks among the companies' employees, whether they are due to a regulatory requirement or the result of an accident, we find ourselves most often working in situations in which relatively high importance is placed on checking rock thickness.

Finally, blasting delays, controlling the blast dynamics, can also influence the quality of confinement.

Initial blasting condition audits make it possible for us to quantify the energy used and the variability of the geometric confinement of the charges.

1.2 Consideration of the rock mass to be blasted

The flyrock estimation model should cover all types of geology and geological structures likely to be encountered at each stage of the operation and the models to determine the different possible configurations of the rock mass are still only in development phase. For this reason, considering the limited knowledge of the rock mass at the survey stage, its behavior can only be taken into account statistically. This approach is not surprising since initial surveys of vibration impact or airblast are generally carried out in the same manner using statistical laws of propagation.

In this study, we are interested in flyrocks relating to infrequent events and which, in most cases, are therefore linked to particular geological situations and high-risk situations, significantly different from the situations commonly encountered in operations. Such high- risk situations are likely to appear in different geological contexts even if they are of a different nature. Thus, karsts or cavities in limestone massifs, areas of weathering in granites, faults or open joints, etc. could be encountered.

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Considering the lack of systematic recording of flyrocks connected with precise blasting parameter measurements, there is insufficient information from past work to differentiate between the geological contexts and even the broad geological formations. At first approximation, each geological situation appears to present the same risk levels: almost the same percentage of accidental flyrock can be found in the different main geological formations.

1.3 -Choice of flyrock model

As a general rule, flyrocks can come from either the upper zone of the blast (flyrocks generated from the head of drilling holes), or the lateral clearance zones (flyrock generated from the face) as is the case for bench blasts.

Flyrock generated from the head of drilling holes follows a bell-shaped trajectory and can travel in any direction; however, its range is comparatively short for blasts carried out in accordance with good practices (that comply with the depth and quality of the stemming material etc.).

Flyrock generated from the face follows a straight trajectory if it is positioned towards the front of the face (a half-space opposite the blast) and travels a relatively long distance for bench blasts carried out in accordance with good practices. Risk linked to this type of flyrock can be completely eliminated by choosing appropriate orientations of the face.

Based on our experience of accident analysis, long-distance flyrock comes from isolated blocks or in a small number of cases which, for this reason, interact with each other very little once ejected from the original rock mass.

The effect of the air on the movement of blocks is complex. Indeed, if the air drag slows down the movement of a block, the air can create phenomena of lift for flat elements. Besides, the wind can favor or hinder the movement. These contradictory effects of the air will be taken into account statistically. The trajectories of the cast blocks can therefore be represented as parabolas and the flyrock will therefore be determined entirely through its speed and initial orientation, at the time of the blast.

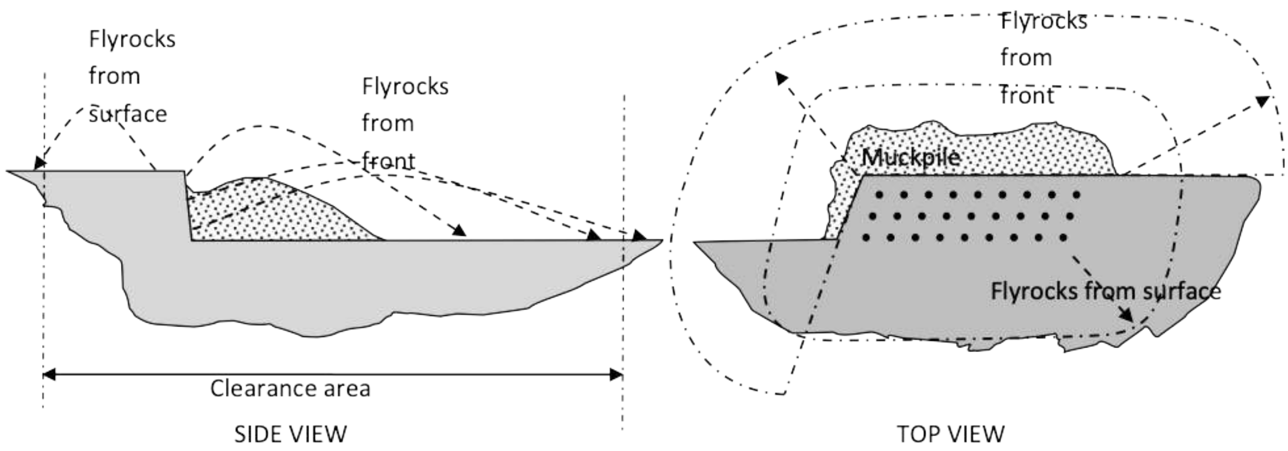

Figure 1 – Areas affected by flyrock generated from blast faces

1.4 Estimation of flyrock range

The variability of rock mass confinement ability, of the thickness of rock confining explosive charges and of blasting situations prompted us to find a model that was both stable and simple to determine flyrock parameters.

The formula put forward by Frank Chiapetta (1983) allows us to obtain a good estimation of the flyrock speed of the blocks coming from the face. (See reference [4]). It can easily be adapted for flyrock produced from the blasting surface. This formula is noted as follows:

$$V = K \cdot \left[\frac{B}{\sqrt[3]{E_l}} \right]^{-1.17}$$

here V is the flyrock speed expressed in m/s, B is the burden or more precisely the thickness of the rock perpendicular to the explosive expressed in m, El is the linear energy of the explosive charge expressed in MJ/m and K is a coefficient expressing the probability of attaining the estimated speed.

This relation is dependent on the explosive energy being implemented, the rock depth and on coefficient K which represents the blasting situation, and particularly the rock mass characteristics, as in the coefficient in the laws of propagation of vibrations and airblast. Our own experiments lead us to slightly modify K factor from original one.

This approach of the rock can therefore be expressed through a statistic variation of the coefficient K. The first estimation of this variation was established assuming that there was a normal distribution of deviations around a mean value based on studies carried out in the United States since the 1980s: the evaluation of the speed of moving fragments from the working face through high speed imaging. (See references [4] and [5]).

The variation of coefficient K varies depending on the level of probability according to a normal distribution. This variation is expressed in the following table:

Probability of speed attainment	**50%**	**5%**	**1%**	**0.1%**	**0.01%**
K	14	25	32	40.7	50.4

Table 2 – Evolution of K with probability

The movement illustrated in each block is regarded as ballistic. The trajectory of a block, subjected to the initial speed of V at an angle of α on horizontal ground and situated at a height of h with relation to the landing surface of the block, is therefore defined by the following parametric relationships according time t:

$$\begin{cases} X = V \cdot \cos\alpha \cdot t \\ Z = V \cdot \sin\alpha \cdot t - \frac{1}{2} g t^2 + h \end{cases}$$

The trajectory of a block, subjected to the initial speed of V, at an angle of α on horizontal ground and situated at a height of h with relation to the landing surface of the block, can also be expressed in the following form:

$$X = \frac{V \cdot \cos\alpha}{g} \cdot \left[V \cdot \sin\alpha + \sqrt{V^2 \cdot \sin^2\alpha + 2gh} \right]$$

Here X represents the maximum range of flyrock and g the acceleration of the weight at an estimated point.

In these estimations, we take angle α as that corresponding to the maximum flyrock distance d. It is an unfavourable hypothesis.

The distances of the flyrock depend on the relative altitude of the explosive charge and on the potential recipient.

1.5 Impact probabilities

In our model based on a normal distribution of flyrock distances around a mean value, there is no maximal flyrock distance. In reality, the explosive energy implemented is a limited, known quantity and the flyrocks range is bounded. But considering the small number of inventories of long-distance flyrock, it is difficult to establish the effect of a maximal distance by substituting the normal distribution by a bell-shaped distribution.

Based on the exploitation hypotheses prescribed for a site, the previous model makes it possible to determine:

- the distance of maximal flyrock for each hole according to the level of probability,
- the probability that a person is impacted by the flyrock for this hole
- the annual probability of impact of a person considering the number of holes per year blasted in the corresponding direction.

The different formulas used in the model are fully detailed in papers previously presented during Fragblast and ISEE conferences [1] [2].

1.6 -Risk and acceptability

In classic pyrotechnic risk analyses, like those defined by NATO regulations utilised at a European level, the probability of a pyrotechnic accident occurring and the effects of this accident on people are analysed separately. These effects, whether those of pressure or thermal effects from accidental explosions, decrease according to the distance from the accident zone.

In the case of accidental flyrock, the triggering event is the blasting. In addition, the effects of flyrock do not decrease with distance: a 200-gram projectile can be fatal at 20 m, as at 1,000 m.

Consequently, the approach to risk is noticeably different from those of other hazards, like for the risk of accidental explosion of explosive storage magazines, in which the effect varies considerably depending on the distance, like for example and airborne shockwave where the pressure decreases with the distance: the effect of flyrock does not change markedly according to the distance; it is only the probability that changes. Indeed, the probability if impact decreases with distance and at the same time the impact zone increases with distance.

In fact, the risk of fatality is the product of the probability of an accident by the fatal probability in a defined danger zone, knowing that an accident has occurred. In our case, this risk corresponds to the probability of impacting a person at a given place, since we have presumed that each impact was fatal.

These risks are compared to the risk of annual 'natural' mortality. In the case of France, the probability of death is given in the following graph. The values are similar to those from a number of other countries.

The lowest annual risk of death (between 5 and 14 years of age according to French statistics) is in the region of 10^{-4}. Added-on risks that increase the probability of death by less than 1% are considered as being unacceptable. Levels of negligible risk can also be defined.

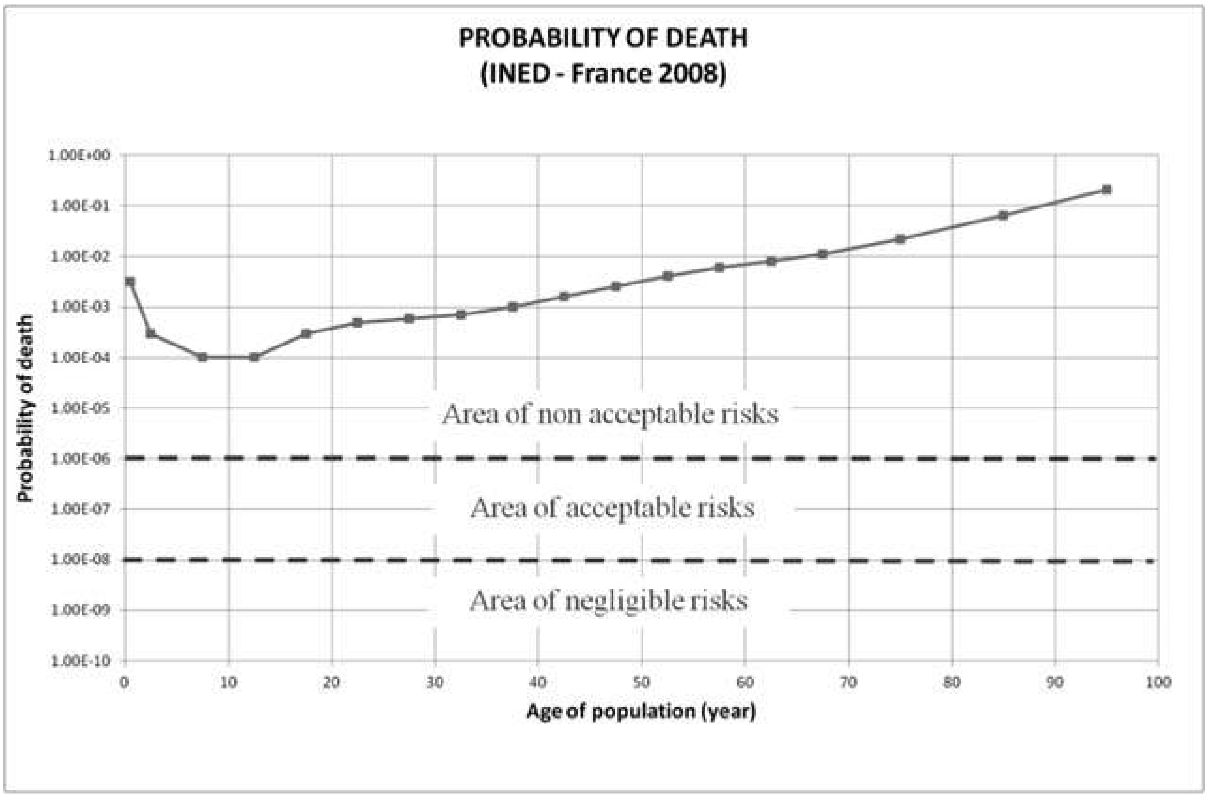

Figure 3 – Probability of death, France – INED 2008

In this way, the NATO rulings integrated in the main into different European regulations accept a maximal risk of 10^{-6} (for a pair of probability event D/P1 and limit of the danger zone Z2) for the external environment. These limits are reinforced for areas with a high-density population for which the maximal risk of 10^{-8} is generally accepted.

The same flyrock leading to significant effects on people only lead to minor damage on infrastructures: The main risks are indeed risks of glazing breakage or damage to roofs or unsturdy partitions.

To translate these results in the same formalism as the French regulatory documents, the risks will be expressed as a pair of the probability of an event E and the boundary of the danger zone Zi (E.Zi) leading to the maximal constraints pertaining to the regulations in force.

2 RESUMPTION OF OPERATIONS IN A LIMESTONE QUARRY

Let us return to the quarry in the South of France. Accidental flyrock from a blast to a nearby engineering factory led to the authorities suspending the blasting rights in the quarry. We carried out a reassessment of the risks connected to the blasting operations at the request of the owner in order to meet the requirements of the regulatory authority and to resume the blasting.

2.1 Neighboring structures and infrastructures

This quarry operates on six benches of a 15 m hillside in a limestone deposit (figure 4). A city of 10,000 inhabitants is situated further down to the south, with the first houses being at a distance of 210 m away. The quarry is also surrounded by a waste collection center at 350 m to the northeast, an engineering factory 125 m to the southeast and a busy railway 45 m away to the west. Furthermore, the service road to the waste disposal center follows the quarry from the south to the northeast.

The first phase of the analysis is to describe the functioning and general organization of the quarry. This includes in particular:

- the description of external neighboring activities,
- the extraction organization based on operating constraints specific to the site,
- phasing and direction of current and intended extraction,
- envisaged blasting patterns on the various levels of the quarry to reach these goals.

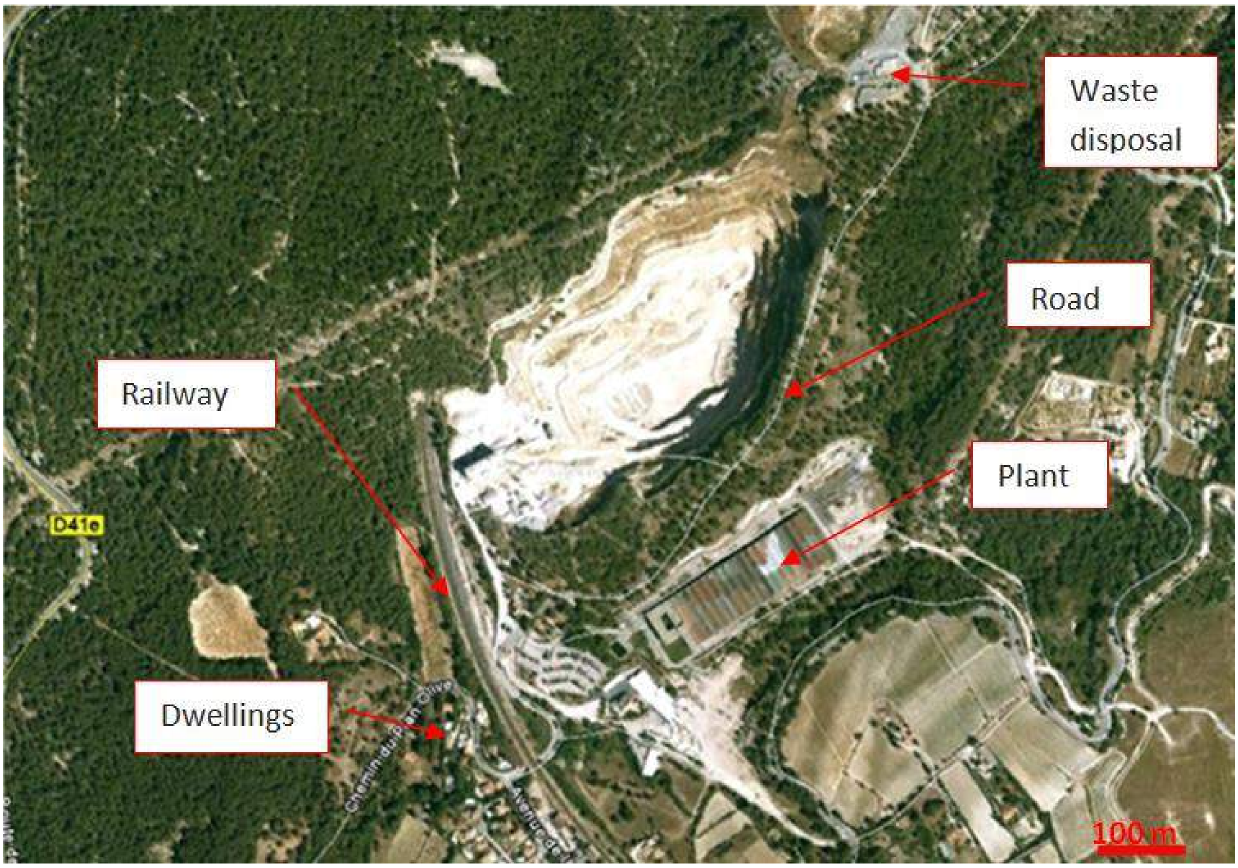

Figure 4 – Map of the quarry A and its neighborhood

Having verified the coherence of the project, concerning the constraints and the production, all relevant information is quantified then introduced into the model, namely:

- the relative positions in terms of altitude and distance from the potential receivers with relation to the various blasting zones planned;
- the blast characteristics and in particular the explosive energy used, the drilling diameter, the pattern, the height of the bench, the overdrilling, the stemmings, the drilling deviations, the state of face, the number of holes for each blast, the number of holes in the first rows, …

This information is completed by other factors making it possible specify the variability of the parameters introduced into the model like, example, the perceived variations of the explosive charges per meter due to the deformation of holes and cartridges, the various methods priming and the various rocky facies of the quarry.

Area	**East part of the quarry**	**West part of the quarry**
Blast patterns	12 m to 15 m high bench blasts + some surface blasts	3 m to 8 m deep surface blasts + 12 m to 15 m high bench blasts for lower level
Benches elevation (m)	50/65, 65/ 80, 80/95, 95/110, 110/125 et 125/140	
Minimum distance to plant - Altitude : 141 m NGF	125 m	138 m
Minimum distance to railway - Altitude : 128/130 m NGF	180 m	44 m
Minimum distance to housing estate - Altitude : 210 m NGF	430 m	210 m
Closest distance to waste collection site - Altitude : 105/136 m NGF	350 m	780 m

Table 5 – Relative position of the blasts and the receivers

2.2 Impact probabilities

The following step is to determine the levels of risk generated by the mining project.

For every blasting situation, that is for every front, every blasting orientation, every blast pattern and every receiver, the risk of impact from flyrock originating from the blasting surface and face is then calculated according to the distance, while taking into account possible screens if required. For each of these blasting situations, safe distances associated with the standard levels of risk as defined by NATO, respected and even expanded by French regulations, are determined.

Table 6 represents as an example, for well-defined blasting situations, the flyrock distances corresponding to standard levels of risks according the NATO rules (and the French regulations) expressed in the form of E.Zi pairs. The situations non-attainable due to the distances between the blasts and the receivers, are shaded in the table, and the unacceptable situations are noted in bold.

Receiver	Receiver altitude m NGF	Bench altitude m NGF	Minimum distance (m)	Levels of risk				
				E.Z1	E.Z2	E.Z3	E.Z4	E.Z5
Plant	141	140	260	65	106	136	170	209
	141	125	160	57	99	128	163	201
	141	110	160	47	90	120	154	193
	141	95	160	32	79	110	145	184
	141	80	160	-	66	99	135	175
	141	65	160	-	48	86	124	164
Housing estate	105	140	220	81	123	153	188	227
	105	125	230	74	116	146	180	219
	105	110	240	68	109	139	173	212
	105	95	250	61	102	132	166	204
	105	80	260	51	93	123	158	196
	105	65	270	39	84	114	149	188
Railway	128	140	45	**71**	**112**	142	177	215
	128	125	55	**64**	**105**	135	169	208
	128	110	65	56	**97**	127	162	200
	128	95	75	45	**88**	119	153	192
	128	80	85	29	78	109	144	183
	128	65	95		64	98	134	173

Table 6 – Limits of the safety zones in the case of surface blasts loaded with Emulstar 6000 in the West part of the quarry and flyrocks from the surface of the blasts.

2.3 Risk and acceptability

The compliance of the situation of every receiver with the current local legislation, in this particular case the French regulations, is then estimated for the various blast patterns according to the number of exposed people and for the various levels of risk. In our example, the compliance check is presented for the engineering factory in table 7.

Whenever the situation does not comply with the regulatory requirements, corrective actions are proposed, such as:

- Changing the orientation of the fronts to bench blasts,
- Changing the pattern to bench blasts,
- Modifying the explosive charges,
- Modifying the top stemming,
- Replacing bench blasting by surface blasting,
- Implementing protection for surface blasts (blasts mats, …),

Situation de tir	Pair « probability of event / effect zone »	Number of persons permanently exposed	Authorised number of persons	Compliance of receiver situation to regulation
Bench blasts Quarry West part	**E. Z2**	**230**	**0**	**NO**
Bench blasts Quarry East part	**E. Z3**	**230**	**< 100**	**NO**
Surface blasts Quarry West part	E. « hors Z5 »	230	No restriction	yes
Surface blasts Quarry East part	E. Z4	230	< 1000	yes

Table 7 – Plant situation before remedies

The incidence of every corrective action on the level of risk is estimated as previously. The owner for his part estimates the interest of every proposed solution and its financial impact in terms of implementation cost, block size and thus costs relating to loss of productivity and production, as well as operating costs. If necessary, the developer can decide to abandon part of the deposit if operations are deemed unprofitable.

Finally, the situation of all receivers and blasts, calculated as the addition of the situation of every individual receiver, is evaluated and presented to the supervisory authority (see table 8).

The authority's technical services rely on these results to validate the approach presented by the owner and to authorize the resumption of operations.

Pair « probability of event / effect zone »	Number of persons permanently exposed	Authorise d number of persons	Compliance of receiver situation to regulation
E.Z3	88.2	<100	yes
E.Z4	640	<1000	yes
E.Z5	687	No restriction	yes

Table 8 – Global situation of receivers after remedies

3 CONCLUSION

Quarry and mines are compelled to estimate the risks induced by their activity in the neighborhood among which flyrocks because potentially fatal is surely the most significant.

In our experience, long-distance flyrock generally corresponds to particular situations of rock confinement in which it is difficult to predict the occurrence with precision but which can possibly be estimated from a statistic model based on cases already recorded.

A flyrock model making it possible to estimate risk levels in the environment has been built using a similar approach to that used in classic pyrotechnic risk studies. It is intended to estimate the risk level for an entire quarry or public works project and therefore foresees the variability of geological and geometric blasting parameters within a large volume. It is therefore significantly different from model based designed to estimated swelling or even flyrocks for a single shot.

This model is supported by:

- a statistic estimation of the confinement capacity of the rock mass,
- quantifying the variations of the confinement geometry and of the explosive energy based on audits of the equipment and manpower available to carry out and inspect blasting.
- determining flyrock parameters with the help of a simple, stable model.

These components converge to determine risk levels, compared to the annual death rate of the population.

This flyrock model, which is as simple to use as are the laws of propagation for vibrations or airblast, could be put in place in numerous quarry or public works sites and make it possible to improve knowledge on the variability of rock mass confinement.

Alain Blanchier

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The memoriam of Alain Blanchier

Alain BLANCHIER was an internationally recognized expert in the field of the implementation of explosives for civil uses. He participated very actively in professional associations (GFEE, FRAGBLAST, FEEM, SYNDUEX, EFEE) operating in the field of mines and quarries, open-cast sites and underground structures. He also worked as an expert with AFNOR and the French Ministry of the Environment. Alain Blanchier is the author of numerous articles and publications that make reference in France and abroad on the topics dealing in particular with sequential blasting, blasting vibrations and overpressures, blasting optimization methods and concepts, mines and control of flyrock risks.

Alain Blanchier était un expert internationalement reconnu dans le domaine de la mise en œuvre des explosifs à usages civils. Il participait très activement aux associations professionnelles (GFEE, FRAGBLAST, FEEM, SYNDUEX, EFEE) œuvrant dans le domaine des mines et carrières, chantiers à ciel ouvert et ouvrages souterrains. Il est également intervenu en tant qu'expert auprès de l'AFNOR et du Ministère Français en charge de l'Environnement. Alain Blanchier est l'auteur de nombreux articles et publications qui font référence en France et à l'étranger sur les sujets traitant notamment du tir séquentiel, des vibrations et surpressions aériennes liées aux tirs de mines, des méthodes et concepts d'optimisation des tirs de mines et de maîtrise des risques de projection.

Algorithms for Adaptive Solutions: Essential Tools for Operational Blast Optimization

Abstract

Blast optimization in open-pit operations is a task that often remains at a theoretical stage. Indeed, when it comes to operational blast design and onsite implementation, time restrictions and field conditions make it difficult to execute a planned pattern accurately and optimization according to local conditions is even more difficult. For example, adjusting a drill pattern to a complex face geometry is not something that can be easily undertaken on site. Blasters and engineers will do their best trying to distribute boreholes over the bench and even then, it can take them hours to achieve an acceptable layout. Similar situations occur with delay sequences, when trying to adjust them to address selectivity or vibration issues. For these reasons, operational blast design will often remain at a very basic level and the required adjustments are laid aside due to the difficulty of implementing them on site.

However, we can recently see important developments in design algorithms that use precise field information to optimize drilling and blasting patterns. The combination of such adaptive algorithms with precise measurement tools and up to date drilling and loading equipment can help engineers to address blasting with an optimization perspective. Moreover, this can be achieved in a much quicker way than traditional “trial and error” methods, which makes it an essential operational tool.

The combination of this technology is leading to a new methodology for operational blast implementation, in which boreholes are set, drilled and charged according to their actual local conditions. This paper shows practical examples on how algorithms and related technologies are operated in mining and quarry blasting.

Introduction

Technological developments in measurement tools (lasers, GPS, drones) or design software have undoubtedly improved open pit drilling and blasting over the last few years. This combination of equipment and software primarily serve a safety purpose by decreasing the risks of flyrock and avoiding major problems with blast geometry [Chavez R. 2007, Jauffret 2016];

Nevertheless, while their contribution is unquestionable, the issue remains as to whether these tools support operational optimization. Do they genuinely help to find the best solutions in the field depending on local geometric and geological variations?

We can safely say that these tools alone are not sufficient to achieve this optimization. Operational optimization involves providing solutions that are tailored to the variable conditions of the ground. This often requires complex thought and adjustments. For example, the design of a drill grid suited to challenging face geometry, or an optimized ignition sequence for complicated mineralization geometries. Though fundamental, these tasks are often approximated due to a lack of time and efficient design tools. In operational conditions, especially during mining operations, a lack of time is the main challenge to optimization.

To alleviate this operational constraint, we therefore need other types of tools which will play an increasingly important role in operational blasting. These tools are algorithms which offer adaptive solutions. This means that they help to adapt the blasting parameters depending on the variable conditions in the blast in order to optimize the desired result.

These results may take various forms such as the level of fragmentation, compliance with the bench geometry, decrease of back-break, displacement during the blast connected with the dilution, vibration levels, etc. They will have diverse degrees of significance depending on the mining operations however they all depend on how the blast energy is distributed in space (geometry) and in time (ignition sequence). The efficient planning of this distribution, with the aforementioned time constraints, requires algorithms which will be able to incorporate the field information and accordingly locally vary the drilling, loading and sequence parameters. They will soon be essential tools for blast engineers enabling them to perform complex blast pattern adjustment tasks in a very short space of time.

The IT sector constantly develops an amazing diversity of algorithms [See for example McCall J. (2005); Cormen T. H (2010)]. They are increasingly invading all fields of modern industry. How can an activity such as blasting – which is primarily based on human experience - take benefit from this trend? Below are a few examples related to the common problems in open pit mining and quarries.

Adjustment of the drilling to complex faces

The drilling geometry is the fundamental basis for any successful blast. Given the growing similarity of industrial explosives in the mining and quarry sector, drilling is the principal energy distribution operation for a blast. However, a standard drill grid is often applied, whereas this grid should be adjusted to the local geometric conditions.

In quarry blasting, as well as in open pit mining, one major problem involves adjusting the drilling pattern to the geometric variations of the quarry face. As a matter of fact, due to the over-break of the previous blast, the walls are sometimes very uneven and do not represent a well-defined pattern (Figure 1). In open pit mining, this problem is often ignored and the standard drill grid is set by visually arranging the holes close to the crest. These are rarely sloped which may lead to very significant toe thicknesses (refer to Figure 2). The consequences are often oversize, remaining toes and uneven floors (Figure 3).

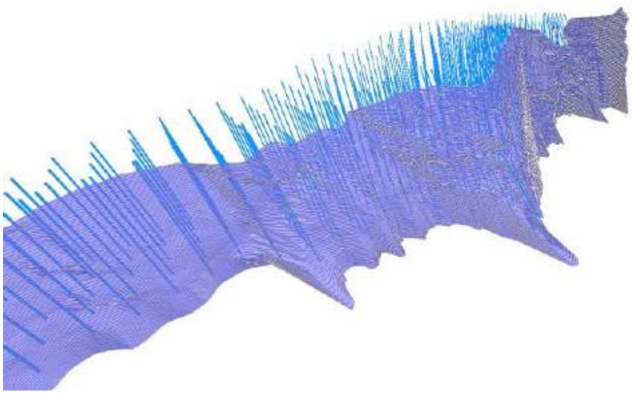

Figure 1: Uneven face

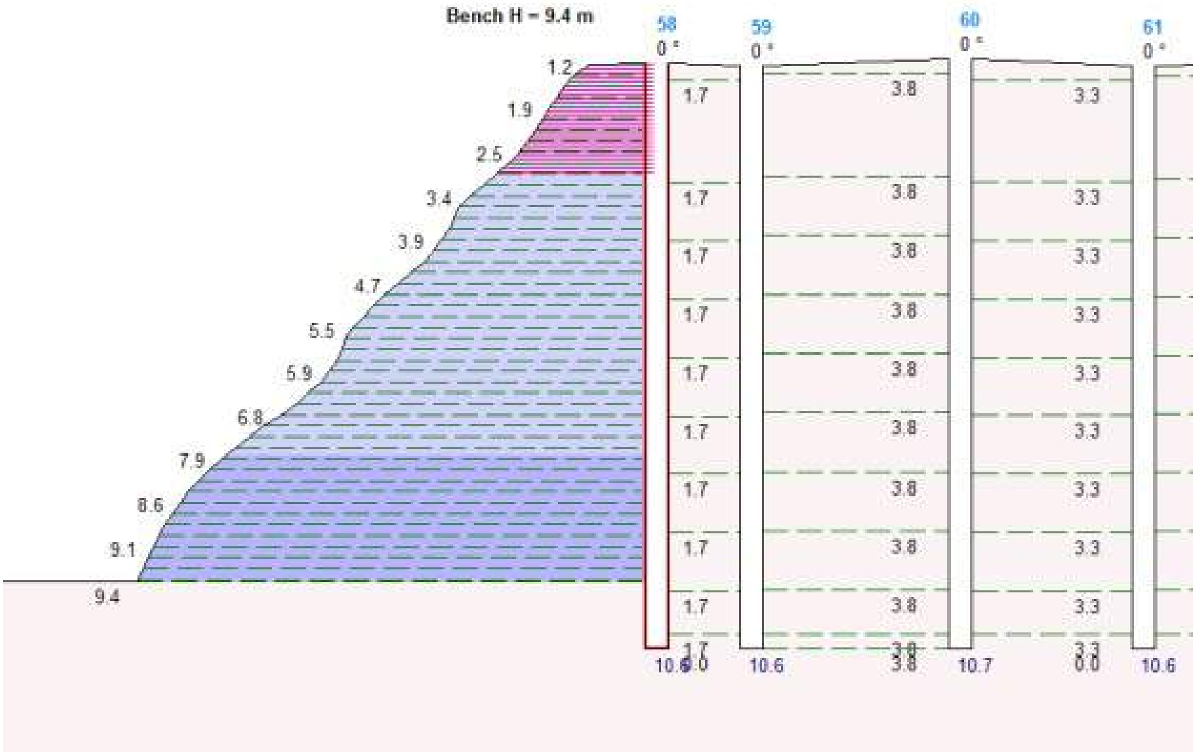

Figure 2: Unsuited slope close to a free face

Figure 3: Remaining toes, consequence of very significant toe burdens

Based on sound operational practice, the holes of the first rows should be sloped in order to comply with the toe burden and thereby obtain proper detonation of the round. Figure 4 shows two drilling profiles adapted to the local slope of the face. However, designing a blast pattern of this type for dozens or even hundreds of holes would require time and effort that the engineers do not have in the field. Hence the very approximate approach of this type of situation.

With adapted calculation algorithms, this can now be avoided. The algorithms try to find an optimum solution by adapting the positions of the drilled holes to the actual face geometry. This will be done while ensuring a regular pattern for the rear of the blast. Once the geometric model of the face has been imported, the user will simply determine the area where the holes must be set up. In a few seconds, the algorithm will provide a setup proposal which complies with the toe burden instructions from the free face. The algorithm will ensure that the holes in the back rows will quickly converge towards the nominal grid as shown in Figure 4. In the input parameters, the tolerances for the minimum crest burden will be specified. We could also include the desired slope intervals (for example, every 5 degrees).

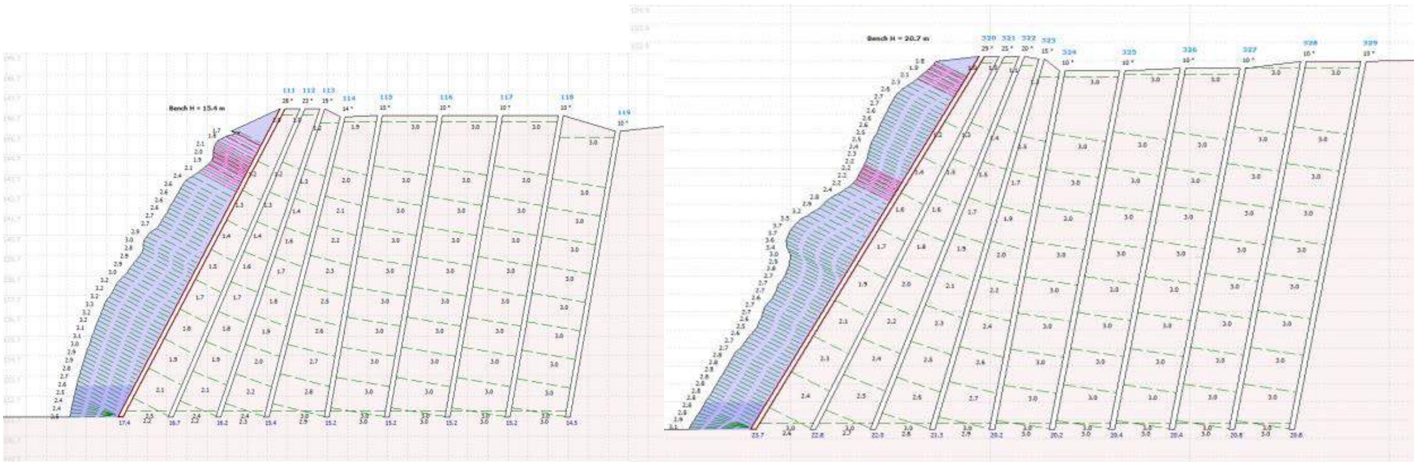

Figure 4: Drilled holes adjusted to the slope of the face

Figure 5 shows the top view of a drill grid automatically designed by the EXPERTIR software for the complex face of Figure 1. The difficult work related to setting up the holes is thereby made almost instantaneous for the grids comprising several dozens or even hundreds of holes. The engineer will therefore be able to focus on checking the profiles and making any adjustments that are deemed necessary.

This algorithm recently proved its worth in our drilling-blasting operations. The technique is even more efficient when it is combined with the Hole Navigation Systems (HNS) offered by drilling machine manufacturers. After the software has designed the set-up, the toe and crest hole coordinates are remotely transmitted to the drilling machine. The apparent complexity of the variable angles and depths supplied by the algorithm is resolved by the accurate GNSS guiding in terms of positioning, azimuth, slope and depth.

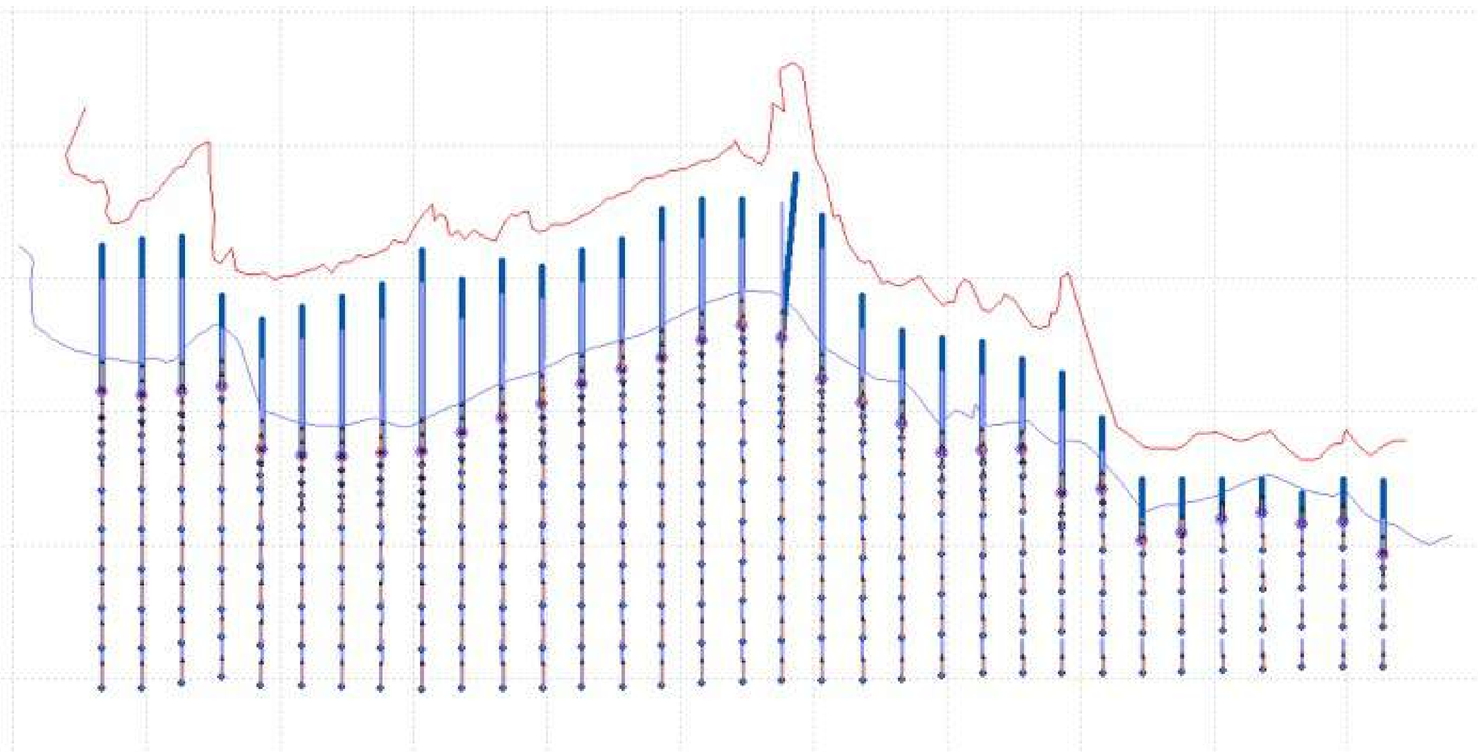

Figure 5: Drill grid automatically obtained for the adjustment of a face

This combination between setting up algorithms and automatic drilling systems allowed us to achieve substantial operational improvements such as time saving, improved control over fragmentation, proper break-out of the toes and flat floors. For example, it is in use in the frame of blast engineering services that Swedish EPC teams provide to quarries. The blast shown in Figures 6 and 7 show a typical design at Quarry ALUNDA. In this case the face was very uneven. The target was to obtain a well distributed drill pattern while ensuring a more even face at the back of the blast. The blast planning procedure consists in surveying the face with drone. Based on the survey, the drill pattern is designed with the EXPERTIR software and sent to the drill rig via an IREDES file for automatic navigation and hole positioning.

The adaptive algorithm delivered an even distribution of borehole toe positions, as shown in Figure 7 (green square dots). According to the engineer in charge of design, the adaptive algorithm saved him at least 1 hour in drill pattern design in this case. Moreover, a "manual" setting of boreholes with the same software would not have given him such an even distributed drill grid at the toe, which is the basis for a good blast result. However, he is aware that the automatic drill pattern design must be a proposal that needs to be revised and adjusted by human intervention. But the more time he saves for the cumbersome and complicated drill pattern layout, the more time is left for his adjustments to ensure all safety requirements related to flyrock, for example.

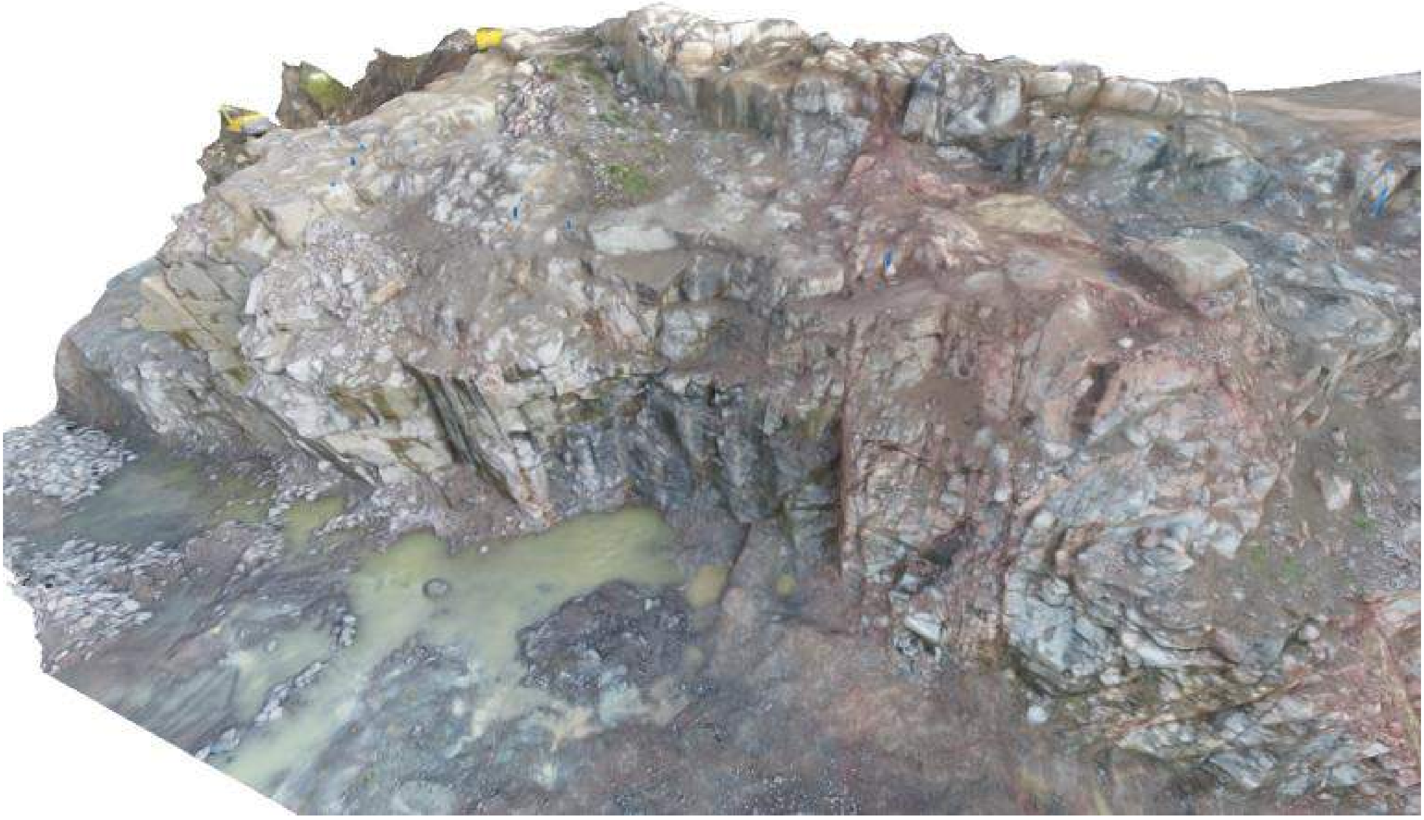

Figure 6: Bench to blast at Alunda quarry (Sweden)

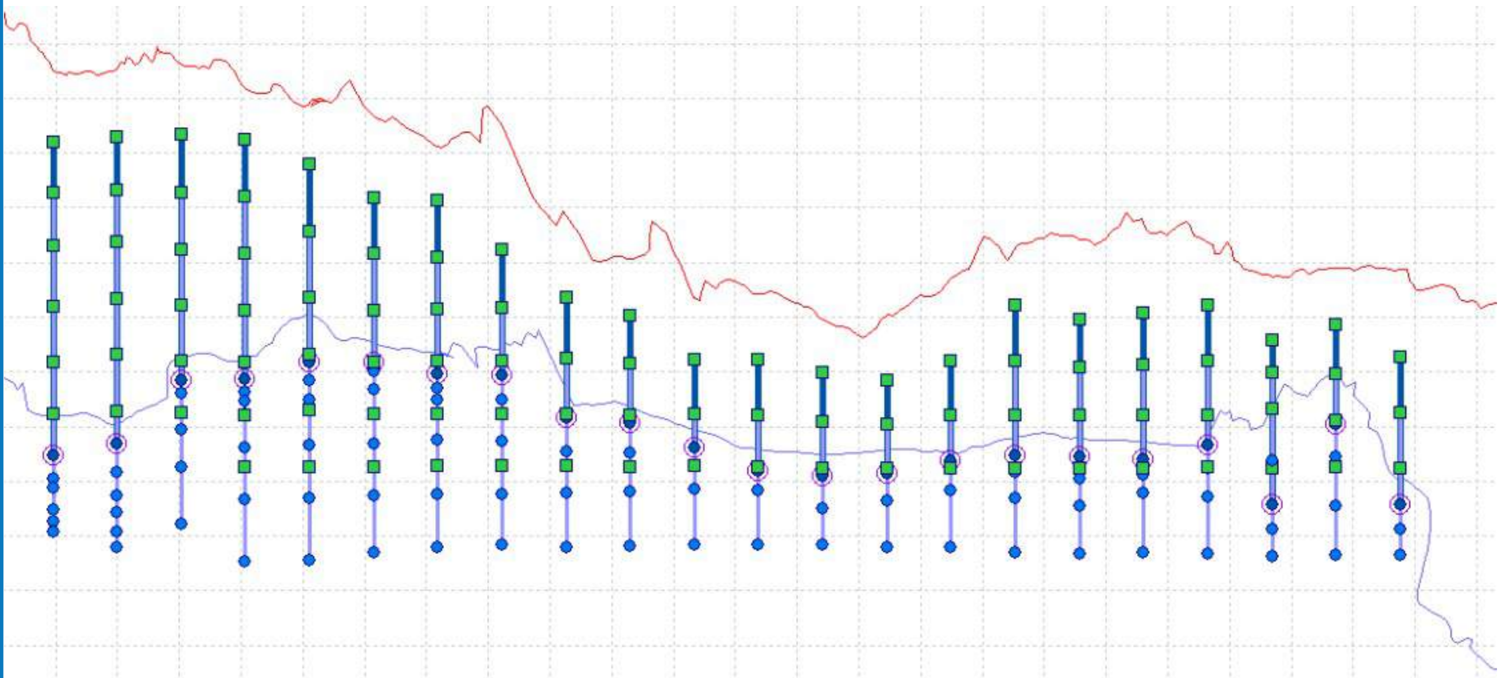

Figure 7: Drill pattern with automatic adaptive algorithm

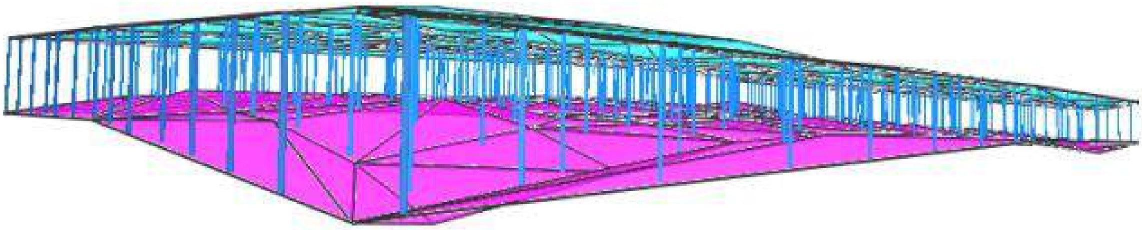

Figure 8: Top and bottom surfaces of a bauxite layer.

Adjustment of the drill grids to variable depths

Another problem involved in the design of blast patterns is the adjustment of the drill grids to variable depths. All blasting engineers have already faced this problem, for example for ramp blasts or when it involves blasting an ore layer of variable thickness. For example, this last case can be seen in the CBG mine in Sangaredi, Guinea. The mining of the bauxite layer requires drilling variable depth holes to limit the dilution with the waste rock. These variations can be seen in Figure 8 which shows the top and bottom surfaces of the bauxite layer.

Whenever this type of blasting is carried out with a constant drill grid, the specific energies as well as the blast fragmentation conditions vary depending on the depth. For instance, a pattern which was designed for a depth of 8 meters (26 ft) will have an entirely different effect on 4 m (13 ft) deep holes. The final stemming length, which remains constant, will impact the specific energy which will decrease for lower depth holes. As a result, when the depth decreases, the pattern should shrink in order to avoid the oversize and toes remaining between holes.

Figure 8 demonstrates that it would be unrealistic to attempt to manually design a constant energy drill grid even if a lot of trial and error is applied. Accordingly, we have developed an algorithm that adaptively readjusts the grid. Based on the decrease in the power of the layer, the grid will shrink and on the contrary, will increase if the power increases. This makes it possible to guarantee the most constant energy distribution possible.

On the left hand-side of Figure 9, we can see an example of a standard grid compared with the adaptive grid on the right. The color represents the specific energy (in MJ/m3) on the zones of influence of the holes. The standard grid shows even more significant energy disparities than the set-up with the adaptive grid (the green color represents the target energy). This more homogenous distribution of the energy must lead to a more consistent fragmentation and increase in mine productivity.

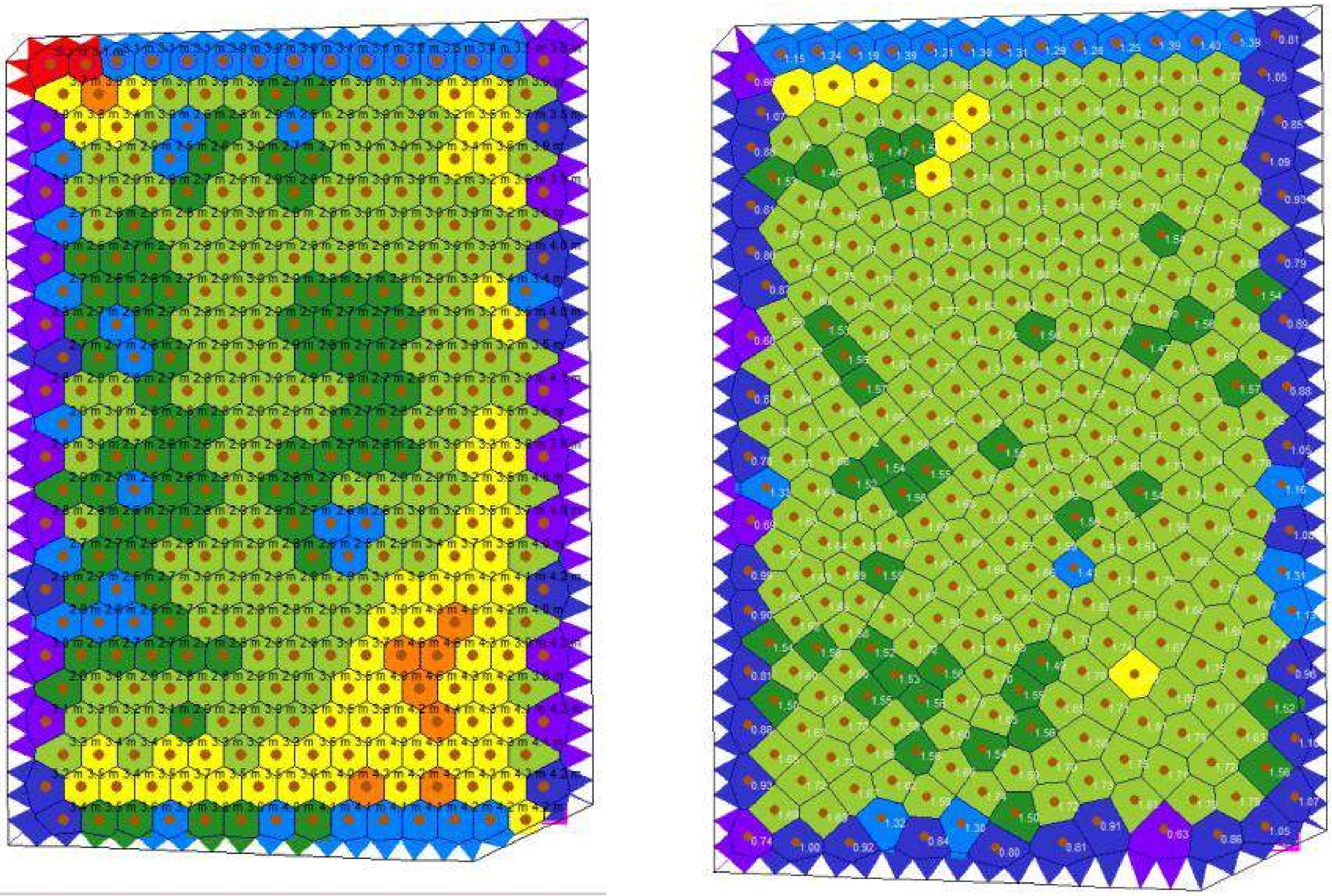

Figure 9: Comparison of energy distributions between a standard grid and an adaptive grid (on the right).

Test are in preparation at CBG mine to assess the actual benefits the method provides in terms of fragmentation. One of the points that must be addressed is the type of ignition. Indeed, this adaptive grid complicates the design of the non-electric ignition and the rows of the holes are less straight.

This problem does not occur with electronic ignition which is used to design sequences at constant ignition speed while adapting to the variations in the position of the holes.

Following is another example of the algorithm for the operational design of adaptive ignition sequences with electronic detonators.

Design of ignition sequences for selectivity in gold mining

Electronic detonator systems provide complete flexibility, in that any time can be programmed to the closest millisecond. One advantage is being able to adapt the time delay to the position of the holes. However, the appropriate algorithms are also needed to design optimized sequences in acceptable times for production. This is often a difficult task if no computer tool is used. In some circumstances, it is even unsolvable when an adaptive sequence is required to satisfy variable blast parameters. One example is a sequence that must adapt to the shape of ore zones in a gold mine to minimize dilution.

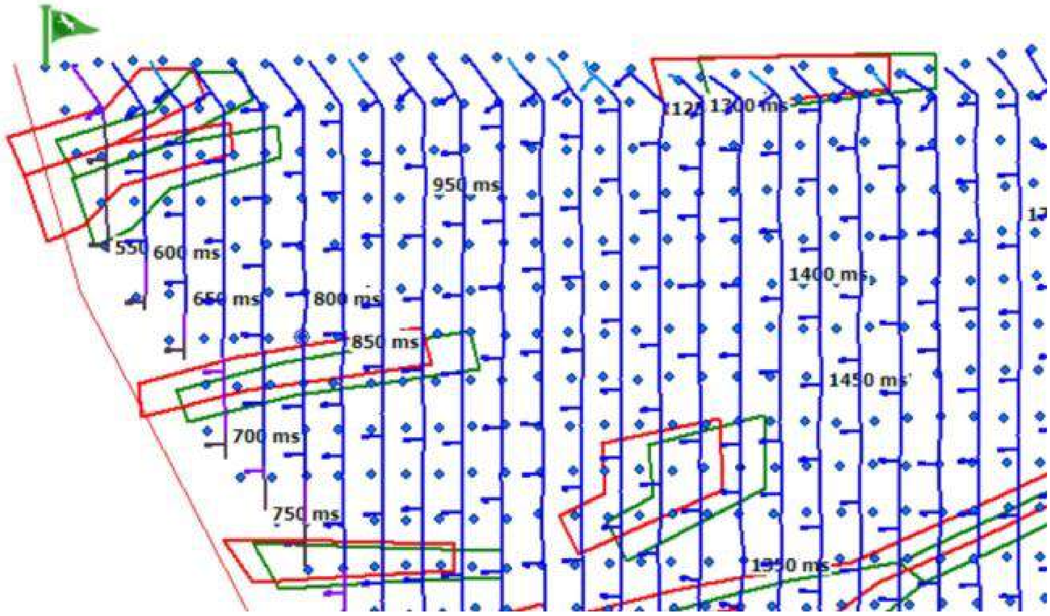

Figure 10: Isochronal lines of the sequence and movement of the ore polygons (measured with the BMM system)

In gold deposits with delineated and extended ore zones, it is well known that lateral movements must be avoided during the blast in order to limit the dilution [Thornton D. (2009)]. As a matter of fact, the more perpendicular the movements are to the axis of the ore during the blast, the more surface area there will be between the waste rock and the ore, thereby increasing the dilution. Therefore, a movement must be obtained that is as much as possible in the direction of the main axis of the ore polygon. To this end, the isochronal lines of the sequence which determine the movements of the solid rock during the blast, must be fully perpendicular to the axis of the ore.

This problem is key for mining operations which try to adapt the sequence to the geometry and the direction of the ore veins. Here again we come up against an operational problem: detonator systems which are not electronic do not have adequate flexibility to vary the sequence and properly direct the isochrones. Even when electronic detonators are used, it is still a complex process to create a sequence that adapts to the shape of the ore using conventional software.

With regard to ore polygons with simple shapes, such as a rectangle, the design may be carried out without excessive difficulties. However, for the more complex polygons, which change their direction, the sequence will have to be adapted to the change in direction. This is further complicated when there are several polygons with different directions in a single blast.

Here again, a dedicated algorithm will help us to locally adapt the sequence and save time. As regards the holes close to the ore polygon, the algorithm will automatically identify its shape and generate times leading to perpendicular isochronal lines (Figure 11). This will be carried out depending on the selected ignition speed level. The movement of the solid rock during the blast will take place in the extension of the ore polygons, limiting lateral movements as much as possible.

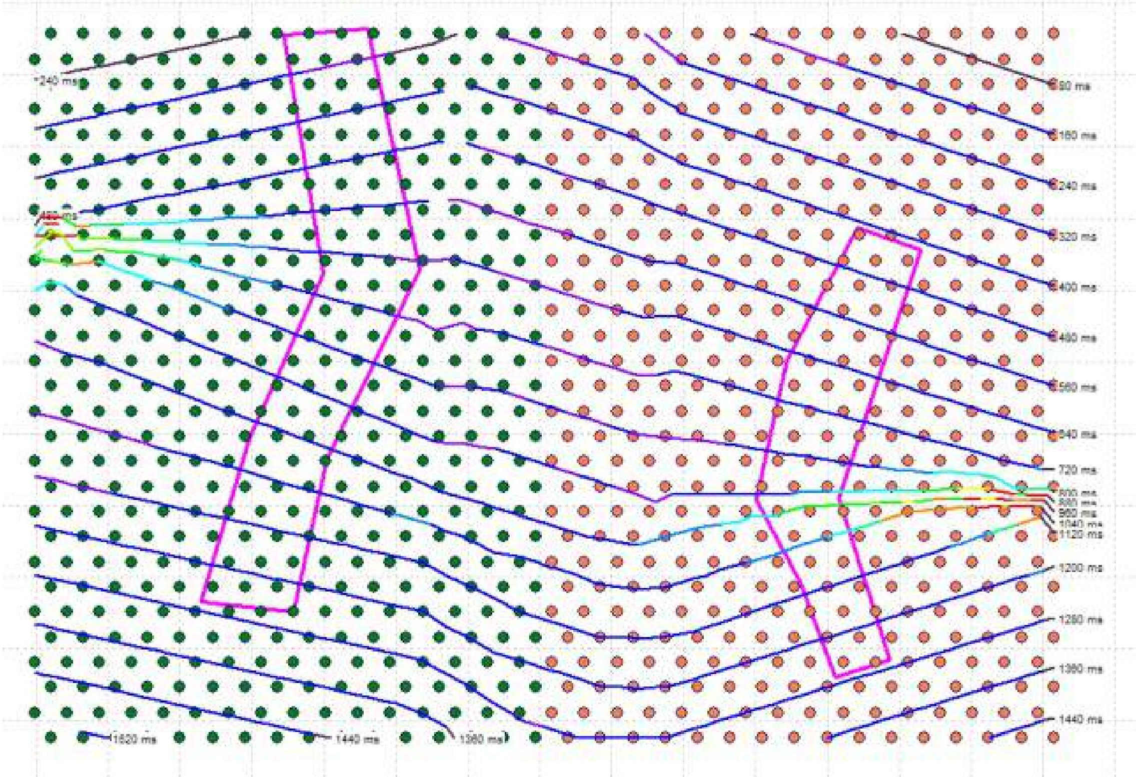

Figure 11: Example of an adaptive sequence based on the variable direction of the ore zones (view of the isochrones, same delay lines).

Tests carried out on blasting operations in Sweden and Côte d'Ivoire revealed that this technique significantly lessens the movement of the blasted ore. This has a high economic consequence due to the better selectivity obtained. For instance, since June 2018 the Agbaou gold mine in Côte d'Ivoire has adopted electronic detonators combined with the adaptive design algorithm for delay sequences. A study was carried out to compare results obtained with electronic detonators versus non-electric initiation, which was conventionally in use in the mine. The set for comparison comprised 24 non-electric blasts and 13 electronic blasts. The comparison was focused on the displacement of ore polygons, paying special attention to the displacement across the strike. As mentioned above, this lateral movement produces most of the ore dilution and must be avoided as much as possible. Movements were measured with Blast Movement Monitoring® technology, which tracks the position of ore polygons before and after the blast [Watson M.E. (2017)].

We observed a significant reduction of the cross-strike movement for electronic blasts. They ranged from 0.3 m (1 ft) to 2.58 m (8.5 ft) compared to 0.5 m (1.6 ft) to 4 m (13 ft) for non-electric (Figure 12).

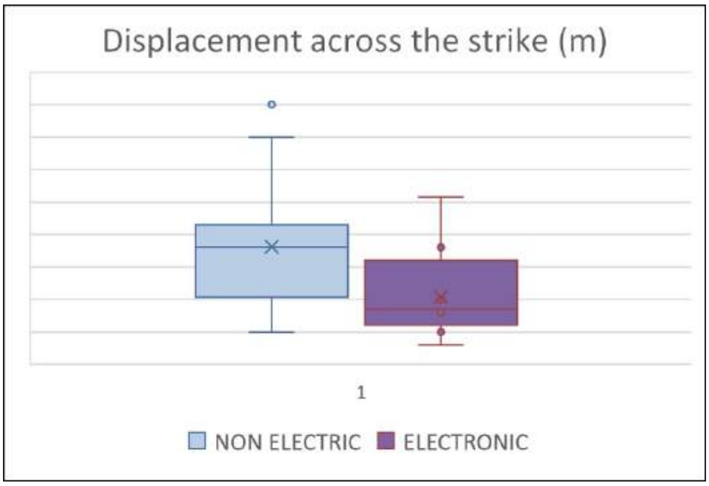

Figure 12: Movement values across the strike for non-electric and electronic blasts

Typical difference in displacement is shown on figure 13 where one can see that the movement across the strike is less for electronic blasts.

The percentage of blasts with across the strike displacement less than 1 m (3.3 ft) raised from 25 % to 69 % (see Figure 14). The vertical displacement was also reduced, whilst movement along the strike increased slightly. Latter is understandable, as all effort is put into orientating the movement in that direction.

It is worth mentioning that these improvements in displacement were accompanied by a significant improvement in fragmentation due to the reduction of inter-hole and inter-row delays. This allowed them to increase the drill pattern in most areas, which led to additional savings. Electronic initiation, along with the adaptive sequence design, replaced non-electric initiation in Agbaou mine and is currently used for all blasts.

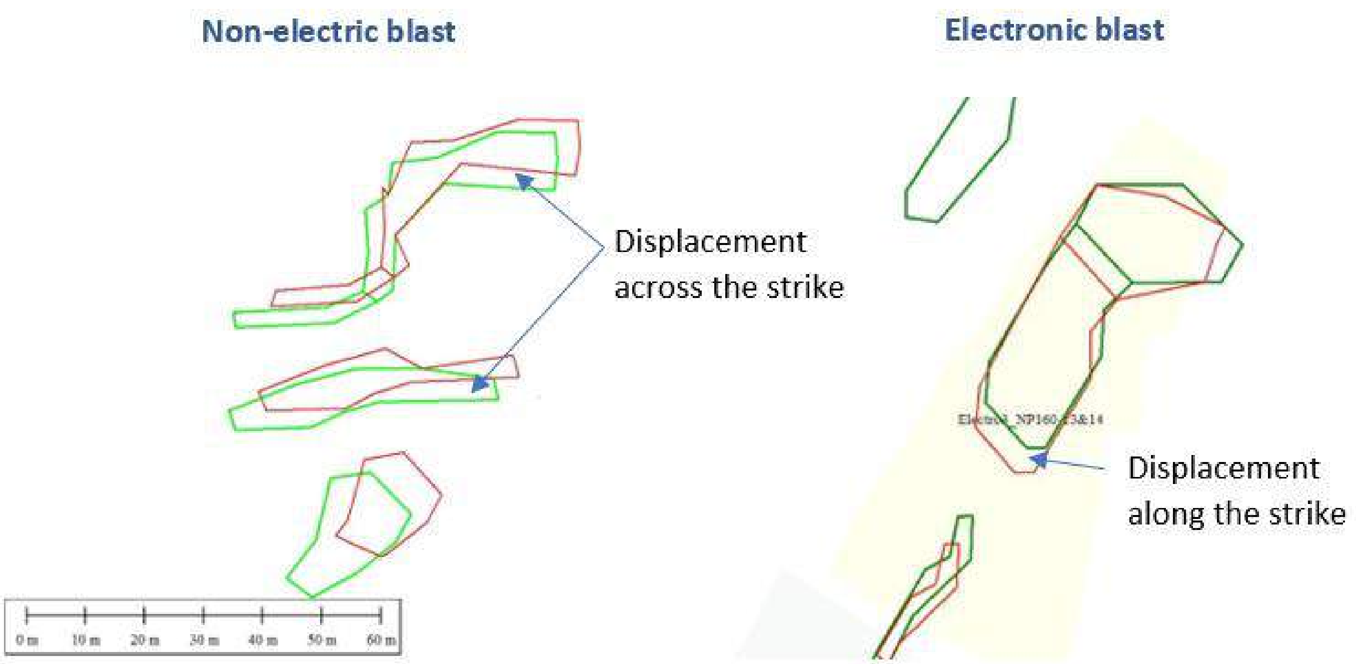

Figure 13: Example for position of ore polygons before (green) and after the blast (red). Measured with BMM system.

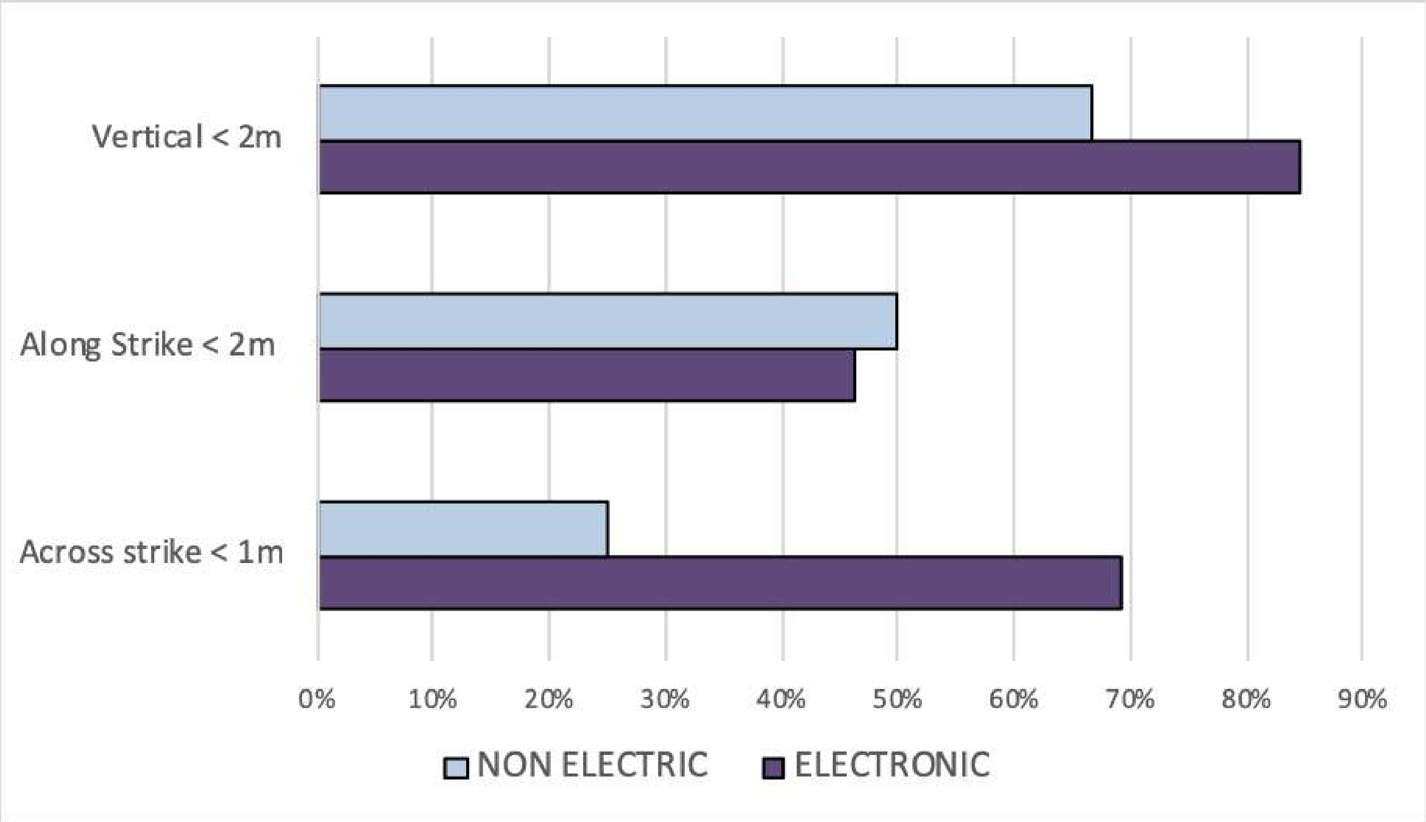

Figure 14: Percentage of blasts with efficient displacement (non-electric and electronic).

Conclusion

The genuine optimization of blasts in operational conditions involves systematically adapting the blast parameters to the variable conditions of the field. Given the rate of production operations, this optimization can only be practically obtained with specialized algorithms providing adaptive solutions: they must include information from the field and then locally vary the drilling, loading and sequence parameter accordingly.

The examples shown in this article are aimed at solving daily design problems in open pit mining. They relate to the optimization of the drilling positions and blasting sequences depending on geological and geometrical variations of the ground. The solutions provided by the algorithms described herein are all intended for better explosive energy distribution. The effective control of this distribution in daily operations provides better management of the desired blast result and therefore a better productivity of the downstream production process.

Until recently, these tasks were impossible to achieve in production conditions however they are now carried out in a few seconds with dedicated algorithms. In this approach, the measurement techniques play an even bigger role as they must provide these algorithms with accurate information to gain even more benefit from their adaptive nature. For example, the increased use of 'measure while drilling' technology along with variable energy bulk emulsion adds yet another possibility for adaptive blast design with the explosive energy in the hole being varied against rock quality or strength as well as adapted for blast geometry.

This development of algorithms for real-time adaptive solutions is only in its early stages. It is changing the way operational drilling-blasting operations are designed and performed.

Dr. Ricardo Chavez, EPC-Groupe
Dr. Rob Farnfield, EPC-UK
Mathieu Lachamp, EPC-Groupe

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Pan-European Competency Certificate for Shotfirers

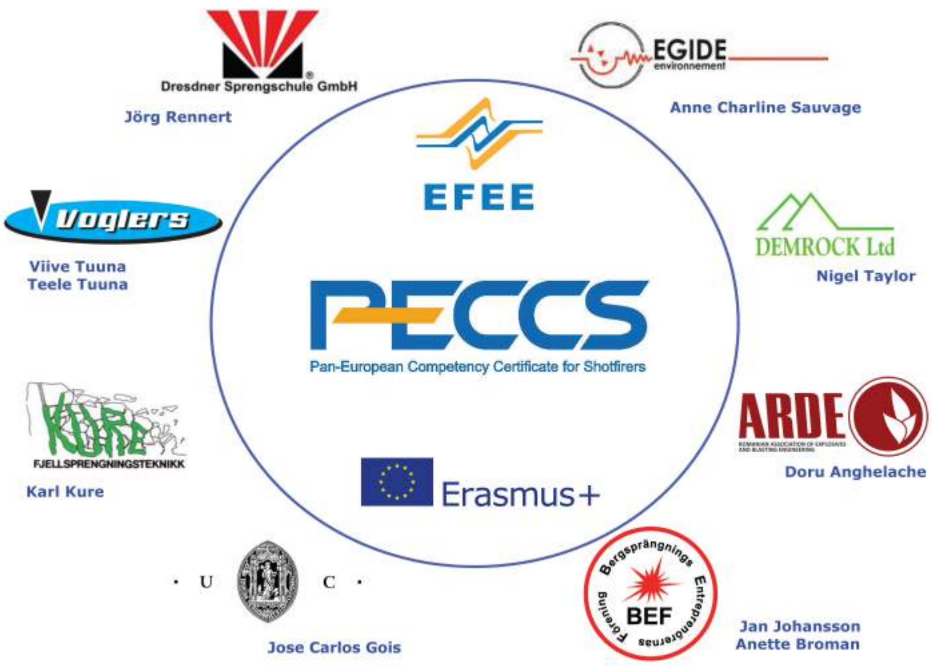
Erasmus+

The PECCS project for the shot-firer certificate started in 2016 and has ended in August 2019.

The certificate will be issued by the European Federation of Explosive Engineers - EFEE, which has 24 members in Europe and elsewhere (Please visit www.efee.eu for more information). This technical competence certificate will come with a learning program - a course for shot-firers which comes with a Guidebook for teachers and extra Online Learning Environment. The course consists of 8 chapters and has been created by experienced specialists from different parts of Europe.

With this strategic partnership for enhancing the shot-firers vocational education and training, we wish to harmonize overall standards for shot-firers in EU and improve the possibilities of mobility in this industry.

We are now looking for contact with different educational entities and training organisations all over Europe to cooperate on issuing the Pan-European Competence Certificate for Shot-firers and blast designers. In case of interest, contact us.



More information: www.shotfirer.eu. or info@shotfirer

Morphological characterisation of explosive powders by X-ray computed tomography: when grain number counts

Abstract

Ammonium nitrate (AN) prills are commonly used as an ingredient in industrial explosives and in fertilisers. Conventional techniques (such as BET or mercury intrusion) can measure the open porosity and specific surface area of AN prill, but the closed porosity is not obtainable. This work was focused on evaluating X-ray computed tomography (CT) as a non-destructive technique for the assessment of porosity in AN prills. An advanced data processing workflow was developed so that the segmentation and quantification of the CT data could be performed on the entire 3D volume, yet allowing the measurements (*e.g.*; volume, area, shape factor...) to be extracted for each individual phase (prill, open porosity, closed porosity) of each individual prill, in order to obtain statistically relevant data. Clear morphological and structural differences were seen and quantified between fertiliser and explosive products. Overall, CT can provide a very wide range of parameters that are not accessible to other techniques, destructive or non-destructive, and thus offers new insights and complementary information.

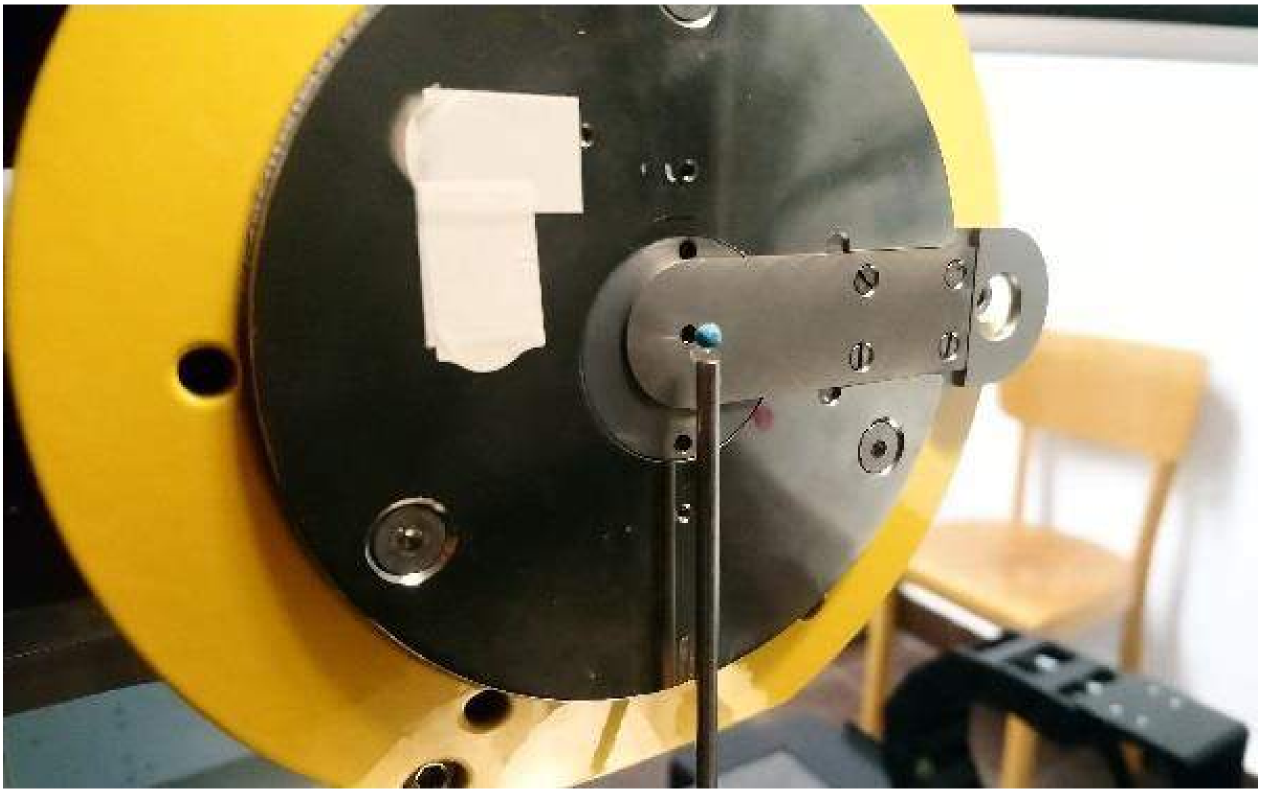
Keywords: ammonium nitrate, prill, non-destructive characterisation, porosity, specific surface

Introduction

In the mining industry, the term ANFO for **a**mmonium **n**itrate / **f**uel **o**il specifically describes a mixture of solid ammonium nitrate prills (see Figure 1a) and diesel fuel (commonly AN 94.5 % / FO 5.5 % in weight [1]), widely used as a bulk industrial explosive. While the worldwide production of AN for fertilizer is around 40,000 tonnes per day [2], the global ammonium nitrate market is expected to reach USD 6.18 billion by 2025 [3]. One of the major performance predictors of the ANFO prills is the fuel oil retention, which is itself governed by the porosity of the AN prills. Presently, the oil retention capacity of ammonium nitrate in prilled and granulated forms is determined by means of a standardised test in the EU [4]. However, this method faces technical difficulties, mainly because the porosity from different types of ammonium nitrate prills varies significantly. The porosity connected to the prill surface, *open porosity*, is available for oil retention. The pores not connected to the prill surface, *closed porosity*, are not available for oil retention but are however important for the explosive sensitivity. The current test methods cannot account for the closed porosity, which explains some of the technical limitations of such test.

One way to investigate and characterise the porosity of AN prills in a more accurate and differentiating way is to use X-ray computed tomography (CT). We believe that CT could be an invaluable tool to the explosives community, by providing qualitative and quantitative measurements of both the open and closed porosity, and the total surface area of AN prills.

a) AN fertilizer prills



b) Single AN fertilizer prill glued onto a C fibre rod

Figure 1. Example AN prills (a) and single prill mounted on carbon fibre rod (b).

In this paper, a data processing workflow was developed to extract these measurements from the high resolution scan of a single AN prill. However, to obtain more representative data, the workflow was amended to extract the same data on a lower resolution scan covering around 20 AN prills, *i.e.* to extract the measurements for each individual prill grain whilst performing the segmentation on the entire 3D volume only once.

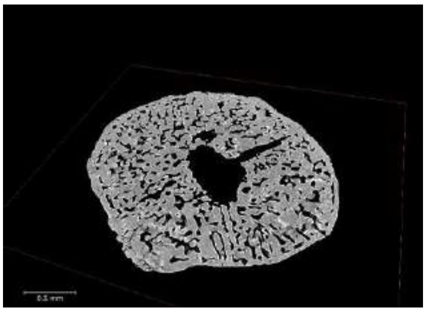
Material and method

Two types of AN prills were CT sanned, the type labelled hereafter *type E* used in the mining industry as a constituent in ANFO mixtures, and the type labelled *type F*, used as a fertilizer in farming. A first scan was performed on a single prill glued onto a carbon fibre rod (see Figure 1 b), to obtain the best voxel size possible (around 2.5 μm) and assess the dimensions of the porosity. A second scan was performed on several prills contained in a polyimide tube of 4.2 mm diameter, so that a good compromise between voxel size (around 5 μm) and field of view (number of grains scanned) was attained.

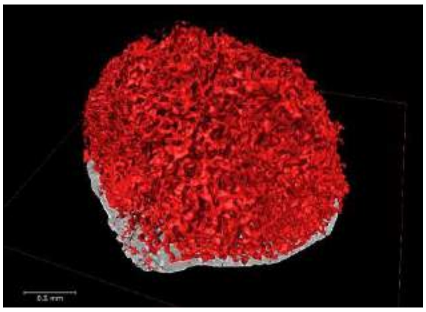
Results & discussion

The examples of 2D slices from the 2 prill types (Figure 2a and f, respectively) qualitatively show the differences in porosity and overall structures. The data processing workflow developed here was aimed at extracting the most relevant structural parameters of the AN prills, both on a global and a local scale, and in a quantitative fashion. Accordingly, the volume fractions of each phase (prill, open porosity in Figure 2b, and closed porosity in Figure 2c) were recorded, as well as the associated equivalent diameter, volume, surface and shape factor. Type E prills, have an open porosity around 30 % and a closed porosity content around 0.3 %. In addition, by using distance transforms, the average radial volume fractions for a set of prills (Figure 2d) can be plotted (Figure 2e) and it reveals that the closed porosity in mainly present in the first 100 μm from the outer prill surface.

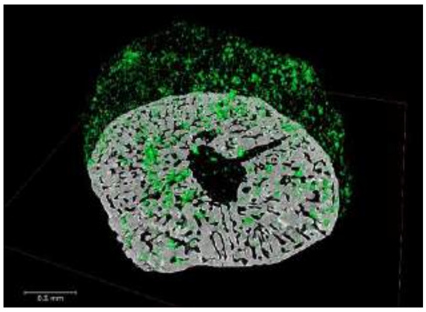
Overall, the results presented here demonstrate that CT can be successfully applied to the morphological characterisation of AN prills in a non-destructive manner, as a wide range of morphological parametars can be extracted, in addition to overall volume fraction values. The future work will focus on comparing the present CT results to those of conventional techniques such as BET and mercury porosimetry, as assess which morphological parameter are most relevant to the mining industry.



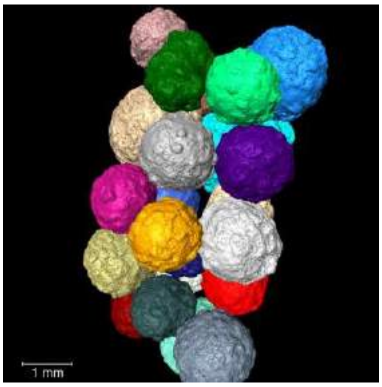
a) 2D slice from type E prill



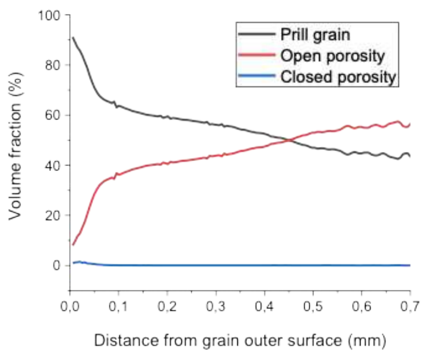
b) 3D rendering of open porosity for type E prill



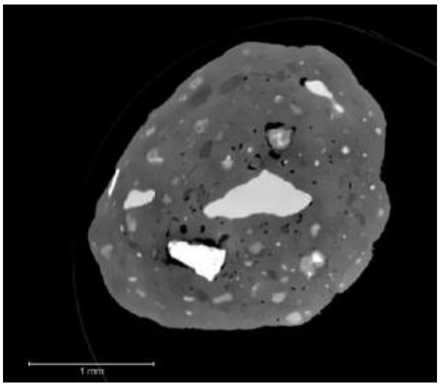
c) 3D rendering of closed porosity for type E prill



d) 3D rendering of type E prills



e) Radial volume fraction for set of prills shown in d)



f) 2D slice from type F prill

Figure 2. Overview of CT results from AN prills.

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New EFEE Members

EFEE likes to welcome the following members who recently have joined EFEE.

National associations

ASSOCIATION OF EXPLOSIVES MANUFACTURERS IN BULGARIA, Bulgaria

Individual Members

Cherie Little, Blasting Geomechanics Pty Ltd,

Dane Blair, Blasting Geomechanics Pty Ltd, Australia

Sun Weibo, XI'AN UNIVERSITY OF SCIENCE AND TECHNOLOGY. China

Shlomi Kanias, EMI - Explosives Manufacturing Industries, Israel

Corporate members

VOGLERS EESTI OÜ, Estonia

Upcoming National Events

Informationstagung für Bohr-, Spreng- und Ankertechnik
Place: CAMPUS SURSEE Bildungszentrum Bau, CH-6210 Sursee LU, Switzerland

Date: 13. / 14. September 2019
Official language: German
Website/Contact info regarding the conference: www.sprengverband.ch

Upcoming
International Events

International Congress on Rock Mechanics and Rock Engineering
September, 13-18, 2019
Foz do Iguassu, Brazil
http://www.isrm2019.com

ISEE 46th Annual Conference on Explosives and Blasting Technique
January 26-29, 2020
Denver Colorado, USA
https://www.isee.org/conferences/2020-conference

SME Annual Conference
February 23-26, 2020
Phoenix, Arizona USA
www.smeannualconference.com

SME Annual Conference
February 28-March 3, 2021
Denver, CO, USA
www.smeannualconference.com

WORLD TUNNEL CONGRESS 2020
May, 15-21, 2020
Kuala Lumpur, Malaysia
www.seacetus2017.com/4/443/welcome-to-malaysia/

SAFEX International Congress #20
May 27-29, 2020
Salzburg, Austria
https://iexpe.org/safex-congress-bulletin-call-papers/

EUROCK 2020
June, 15-19, 2020
Trondheim, Norway
http://www.eurock2020.com/hjem.cfm

World Mining Congress
July 20-22, 2021
Brisbane, Australia
www.wmc2021.org

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- **marketing of EFEE memberships**
- **finding additional advertisers and members**

The applicant should be self-motivated and have adequate written and verbal English and an enthusiasm for sales work. Knowledge of the explosives engineering industry is an advantage. The position is also suitable for a student.

This position is for part time work with estimated working time of 10-20 hrs / month with potential to increase.

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