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# Multivariate evaluation of blast damage from emulsion explosives in tunnels excavated in crystalline rock

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A R T I C L E I N F O

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A B S T R A C T

Blast damage in tunnels is usually regulated in Swedish infrastructure contracts as it can influence the quality and lifecycle cost for tunneling projects. The topic is important for underground constructions with a long operation period such as tunnels for public transport, permanent access tunnels in mines or underground repositories for nuclear waste. This paper aims to evaluate the influence of design and geology variables on the resulting blast fracture length and frequency by means of multivariate data analysis. The analysis was based on data from five field investigations carried out at tunnel sites in Sweden and Finland where emulsion explosives were used. Data was compiled and analyzed using Principal Component Analysis (PCA). Charge concentration was found to be the most influential design variable and hole spacing had limited influence on blast fracturing. Results from the PCA suggest that blast fractures length could be dependent also on geology and natural fractures. Three main groups of fracture patterns were identified, one group with relatively few and short blast fractures, a group with several longer blast fractures and a group with few or a single long blast fracture. The result shows differences in fracture length between the column and bottom charge part of the contour holes, with blast fracture lengths up to approx. 40 cm for the column charge and up to approx. 60 cm for the bottom charge.

## 1. Introduction

Underground facilities that are intended for operation over long time periods, such as infrastructure tunnels, permanent levels in mines and underground repositories for nuclear waste, can benefit from high-quality drilling and charging during their construction. The main quality issues related to tunnel excavation are deviations in geometry (e.g. overbreak) and excavation damage in the remaining rock. These are factors that may influence the costs for maintenance during the operational phase (Drevland Jakobsen et al., 2012). Furthermore, all underground excavations cause redistribution of in-situ stresses that depend on the geometry of the opening and the state of stress relative to the strength of the rock mass. This stress redistribution may lead to development of new fractures and opening or shearing of pre-existing natural fractures around the tunnel. However, in competent crystalline rock, excavation damage can often be simplified to damage only induced by blasting, ranging from micro cracks (grain size) to macro fractures.

In Swedish infrastructure tunnel contracts, blast damage in wall and roof is usually regulated (Svensk byggtjänst, 2017) and for underground repositories for nuclear waste, limitation of blast damage is an important consideration due to the possibility for transport of radionuclides axially along the tunnels in the Excavation Damage Zone (EDZ) (Posiva SKB, 2017). Charging in Scandinavian tunneling projects is today mostly conducted with pumpable emulsion explosives and the charge concentration is lowered for hole types closer to the tunnel perimeter with the purpose of limiting blast damage. However, other parameters such as initiation method, decoupling ratio or hole spacing can also influence fracturing in the remaining rock (Olsson and Ouchterlony, 2003).

Influence of blast design on macro fracturing has been studied by Olsson and Ouchterlony (2003) and Ouchterlony et al. (2010) among others. The influence of excavation damage on tunnel stability has been studied and discussed by Saiang and Nordlund (2009). Everitta and Lajtaib (2004) investigated the effect of the granular structure of the rock type on excavation damage. However, the effect of blast design and geology variables on the development of macro fractures in the remaining rock is still not completely understood.

This paper aims to analyze blast design and fracture data from five tunnel excavation projects in Sweden and Finland where emulsion

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explosives were used. The sites include experimental tunnels, a road tunnel, an underground depot for subway trains and a wastewater tunnel. The analysis was conducted by evaluating the influence of design and geology variables on the resulting blast fracture length and frequency using Principle Component Analysis (PCA).

## 2. Blast-induced fracturing in rock

Several authors have studied the influence of blast design and geology parameters on macro fracturing induced by blasting. In most cases, this has been conducted by applying a dye penetrant near a borehole halfpipe after an investigation slot was opened into the blasted rock surface.

Blast-induced fracture patterns has been studied by Kukolj et al. (2018), who conducted blast and fragmentation tests on mortar and granite cylinders and observed fracture propagation with a high-speed camera. A conclusion from the tests was that the total number of main radial blast-induced fractures remained almost the same along the cylinder axis. The number of blast-induced fractures was observed to increase with increased charge concentration.

Field tests summarized by Olsson (2000) concluded that hole spacing has an influence on blast fracturing. Blast fracture length increased with increased hole spacing and the blast fracturing changed appearance when hole spacing was increased from 0.8 m to 1.2 m. From being composed of isolated radial blast fracturing around individual holes, longer fractures appeared that almost connected the holes. The observed fractures were parallel to the tunnel perimeter and located 10–15 cm into the rock mass. The field tests were conducted with simultaneous initiation by means of electronic detonators. On the other hand, Seccatore et al. (2015) found little or no influence due to increased hole spacing from 0.8 to 1.4 m in a slope excavated in good-quality rock mass. The results were evaluated by observation of remaining visible halfpipes and no blast fractures were mapped. The influence by variations in burden has been found to have only a limited effect on blast fracture length (Drevland Jakobsen et al., 2012; Olsson and Ouchterlony, 2003).

Langefors and Kihlström (1978) suggested that for cautious blasting, the delay time between blast holes should be kept as small as possible. This has been validated in field tests, indicating that simultaneously initiated (delay time < 1 ms) decoupled small contour charges, applied in dry holes, has a damage suppressing effect compared to conventional pyrotechnic initiation (Ouchterlony et al., 2010). Fu et al. (2013) conducted a study where the result from pyrotechnical initiation at the same interval was compared with single initiation (each contour hole detonated at separate intervals) by means of electronic detonators. The findings were that electronic detonators in this case caused less damage (expressed as P-wave velocity) to the remaining rock.

Influence from deviation in drilling is rarely quantified in the literature but often stated to be an important parameter to limit blast damage and overbreak (Cunningham and Goetzsche, 1990; Aijling et al., 2014). Ivanova (2015) conducted field tests to determine the influence from drilling deviation on blast damage and fragmentation. A conclusion was that drill hole deviation, to some extent, influences the length distribution and number of blast-induced fractures.

There are indications that the bottom charge also could influence blast fracturing in the column charge part of a blast hole. Field tests reported by Bjarnholt et al., (1988) showed increased borehole pressures in blast holes with larger bottom charges.

Persson et al. (1993) stated that in order to explain the extent of damage to the remaining rock it is necessary to study the character, strength, orientation, size and frequency of natural fractures. Natural fractures can both extend and limit the length of blast fractures. Open natural fractures have been observed to extend the length of individual blast fractures up to 100% (Ouchterlony and Olsson, 1998; Ouchterlony et al., 1999). At the same time, natural fractures in the surface rock can act as barriers to blast fracture extension where only about one third of them are able to cross open natural fractures (Olsson and Ouchterlony, 2003).

Properties of the rock material will influence the fracturing process. Important factors are strength properties, degree of brittleness and presence of micro cracks as well as the confinement of the material (Persson et al., 1993). The microstructure can also influence the development of fractures. There are indications that grain size and texture of the rock type influence blast-induced fractures. As grain size increases, the grain boundaries become less isolated and can link up to facilitate fracture extension (Atkinson, 1987). Non-cubic brittle ceramics, similar in behavior to rock materials, become more resistant to fracture as grain size increases up to a critical grain size where further increase in grain size leads to a reversal of this trend (Rice and Freiman, 1981). Field observations have been made in blasting by Nyberg et al. (1999) where few larger blast fractures were observed in magnetite ore relative to other rock types.

Olsson and Ouchterlony (2003) suggested a method for estimating the length of the longest blast fracture by means of an uncorrected fracture length, modified by correction factors for specific geological, geometrical and initiation conditions, $Rc = R_{C0}{\cdot}F_h{\cdot}F_t{\cdot}F_v{\cdot}F_b$, where $R_{co}$ is uncorrelated fracture length, $F_h$ is the correction factor for hole spacing, $F_t$ correction factor for spread in initiation time, $F_v$ is a correction factor for wet holes and $F_b$ is a correction factor for rock type and fracturing. The uncorrected fracture length is calculated based on decoupling ratio, charge concentration and VOD.

Ouchterlony et al. (2010) summarized several previously conducted studies and calculated theoretical fracture length based on field tests, both with cartridge and emulsion explosives. Initiation with both pyrotechnical and electronic detonators used simultaneously was tested with the purpose to create a blast damage table, which can aid designers to meet requirements to limit blast damage in the crushed zone around a blast hole.

## 3. Methodology

Mechanized charging was used at the tunnel sites. After excavation, investigation slots were excavated by diamond sawing in the walls and at some of the five sites also in the floor of the tunnels. Natural fractures and blast fractures from visible halfpipes were then mapped in the slots.

### 3.1. Charging method at the investigated sites

Two component emulsion explosives were used at the sites. The emulsions were sensitized and mixed directly on site when charged in the round holes. Charging was conducted by pumping the two component emulsion matrix into the round holes using a hose. The charged holes were then initiated with a detonator and a primer. Charge concentration was reduced for hole types closer to the tunnel perimeter, by adjusting the thickness of the emulsion string. Fig. 1 shows a sketch of a

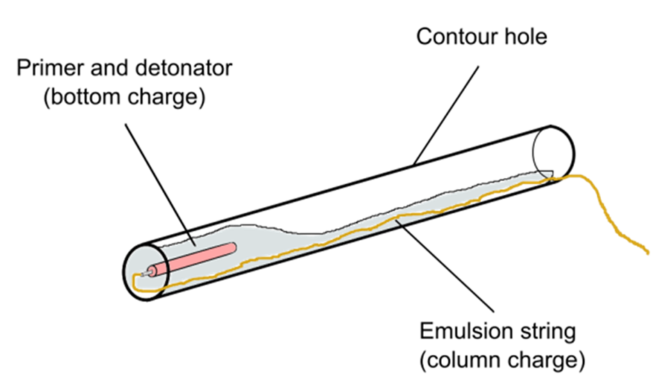


**Fig. 1.** Sketch of string charged contour hole. The bottom charge contains a larger charge concentration, primer and detonator.

string charged contour hole.

### 3.2. Mapping of investigation slots

Fractures directly induced by blasting can be mapped by excavating an investigation slot in the tunnel contour, in sections with visible blasting halfpipes. This method was applied at all sites included in the study. To excavate slots in the tunnel wall or floor, diamond sawing technology was used. A penetration fluid was applied to the slot surface before mapping to improve visibility of the fractures.

Blast fractures do not by definition have filling (mineral, clay material etc.). They extend radially from the borehole and can be observed near the borehole perimeter. Natural fractures can have filling. They can be open, partly open or closed. Some natural may have been partially opened by the change in rock mechanical conditions due to excavation of the tunnel or through the opening of investigation slots. There are also completely open natural fractures. The latter may be of tectonic origin or formed due to the change in stress conditions induced by excavation of the tunnel.

Both natural and blast fractures were mapped and digitalized. From the digitalization, fracture length and frequency could be measured for each blasting halfpipe. Fig. 2 depicts an excavated investigation slot in the tunnel wall at Äspö HRL and Fig. 3 shows part of an investigation slot surface with blast fractures extending radially from a blast hole halfpipe.

### 3.3. Multivariate data analysis

The collected data set include eight variables describing blast design and geology as well as corresponding result variables, such as blast fracture length and frequency. To evaluate the interaction between the variables and the resulting fracturing, a Principle Component Analysis (PCA) was conducted. The aim of PCA is to find directions in the data space that indicates typical features that may not be visible by individual variables. It is used for simplification of data tables, noise reduction, outlier detection, correlation evaluation, classification and prediction (Wold et al., 1987). The method has been used for rock engineering applications by Schunnesson (1990) and Ghosh (2017) among others.

PCA was first formulated in statistics by Pearson (1901) who defined the analysis as finding lines and planes of closest fit to systems of

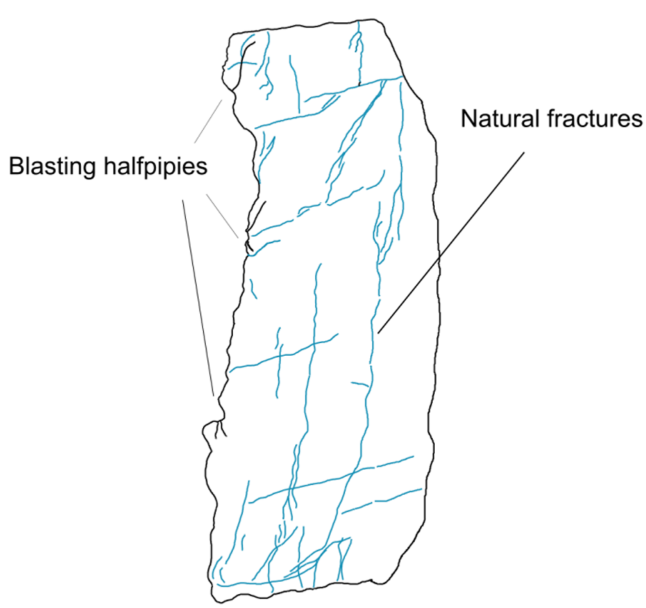


**Fig. 2.** An investigation slot surface in a tunnel wall with marked natural fractures. Hole diameter is 48 mm.

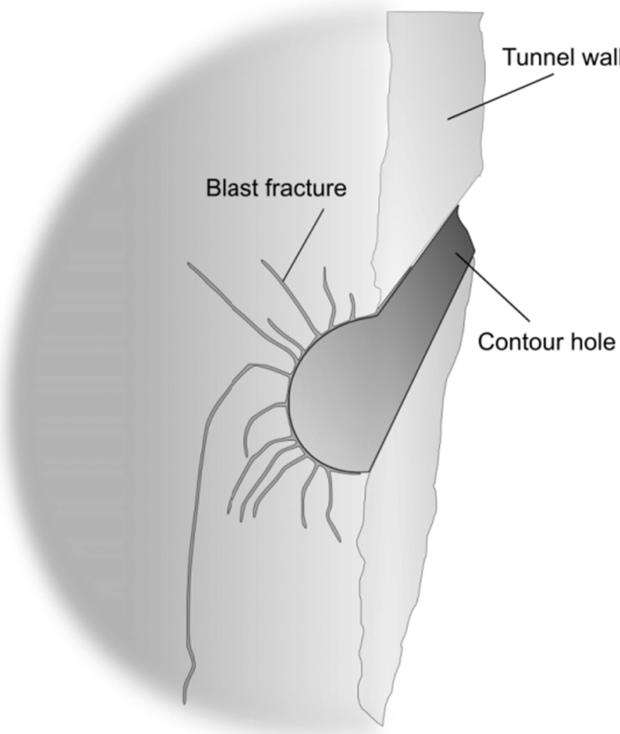


**Fig. 3.** Part of an investigation slot surface showing blast fractures near a blast hole halfpipe. Hole diameter is 48 mm.

points in space. Different methods exist for the calculation of principal components. The NIPALS algorithm (Nonlinear Iterative Partial Least Squares) is commonly used to calculate the principal components (Fisher and MacKenzie, 1923; Wold, 1966). The algorithm calculates the Principal Components (PC) one at a time by means of iteration. Additional principal components are created from the model residuals.

The PC scores of an object is its orthogonal projection on the PC line. The loadings are the projection of variables as direction of the PC. The residuals are the distances between the projection on the PC and the data points. The results from a PC model are best interpreted graphically by plotting the scores and loadings vectors. The scores and loadings plots can be described as windows onto which the data are projected to give a picture of their configuration. The score plot shows the data structure with respect to objects (similarities, outliers). The loadings plot shows the data structure with respect to the variables (correlation, influence). The analysis was done with SIMCA™ Version 14 developed by Umetrics.

## 4. Description of investigated sites

Data was obtained from studies and field investigations at five tunnel sites in Sweden and Finland. The sites include experimental tunnels in Äspö HRL and Sandviks test mine in Tampere, a road tunnel near Tanumshede, a subway depot in Norsborg, Stockholm, and a waste water tunnel in Kista, Fig. 4.

The tunnels were excavated with different requirements, where the experimental tunnels in Äspö HRL and in Sandviks test mine applied an approach with cautious blasting, i.e. simultaneous initiation of the contour holes as well as good precision in drilling and charging. The road tunnel at Tanum, the Subway depot at Norsborg and the Kista tunnel were excavated with conventional requirements. The conventional emulsion explosives Kemitti 810 and Riomex UG were used at the sites. The target density was 1.0 kg/dm$^3$. Table 1 summarizes the site conditions. A summary of the blast designs that were applied at the

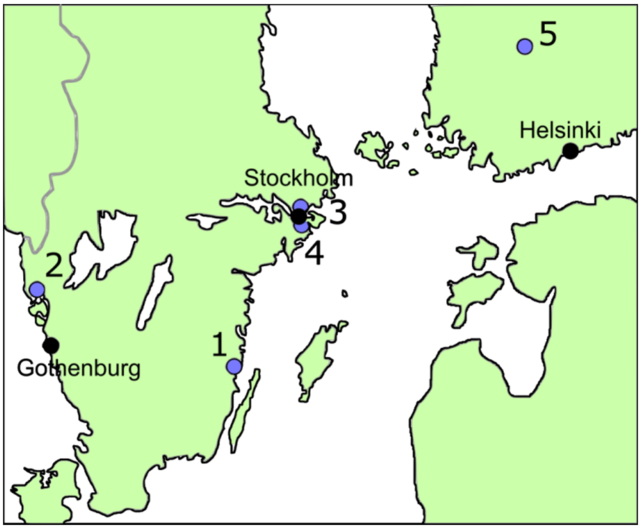


**Fig. 4.** Location of the tunnel sites in Sweden and Finland. (1) Äspö HRL, (2) Gerums tunnel in Tanumshede, (3) Kista tunnel, (4) Subway depot in Norsborg and (5) Sandviks test mine in Tampere.

tunnel sites is presented in Table 2.

## 5. Results

### 5.1. Results from mapping

Table 3 summarizes the number of investigation slots and mapped halfpipes at each site together with references for each study.

A summary of blast fracture length for each test site is presented in Fig. 5. The result from mapping shows differences in fracture length between the experimental and conventional tunnels as well as between the column and bottom charge part of the blast holes. Individual blast fractures in the collected data set have a variation in length from between a few centimeters up to approximately 40 cm and data from the bottom charge extend up to approximately 60 cm.

The longest blast fracture from Äspö HRL for the charge concentration 0.35 kg/m and 0.5 kg/m had a length of 24.5 cm and 24.1 cm respectively. The observed distribution of blast fracture length is shorter for Äspö HRL compared to results from Norsborg, the Kista tunnel and the Sandvik Test Mine. However, the fracture length from the Gerum tunnel is shorter compared to the other sites. One possible explanation is the difference in the number of mapped halfpipes for charge concentration 0.35 kg/m, which is larger for Äspö HRL compared to the other tunnels. Another observation is the long fracture length from simultaneous initiation compared to pyrotechnical initiation in the Tampere Test Mine, which contradicts earlier experience. For the contour, this could be explained by a higher charge concentration for the simultaneously initiated blast holes (0.4 kg/m compared to 0.35 kg/m).

When the distribution of blast fracture length in the Kista tunnel is compared, the result with several long blast fractures caused by the low charge concentration is a bit surprising. The results are, however, comparable with those from the subway depot in Norsborg. Mean values of number of fractures for each mapped halfpipe is presented in Table 4.

The number of blast fractures is larger for the bottom charge halfpipes compared to the column charge. Mean values for the column charge varies between 2 and 4 blast fractures with the exception of the Tampere floor holes with electronic initiation and the blast holes in the Kista tunnel charged with 0.8 kg/m. The charge concentration 1.2 kg/m showed the highest number of fractures in the Kista Tunnel. Around 15 blast-induced fractures were mapped in one of the sections. These fractures were, however, not classified as directly blast-induced.

Density of natural fractures, mapped in each slot is presented in Fig. 6. There is a slightly larger fracture density for Äspö HRL and the Kista tunnel compared to the other sites. Variations in fracture density could be explained by the rock type, for example the larger fracture density value in the Kista tunnel was mapped in a slot excavated in an area with pegmatite dykes.

### 5.2. Data set and variables

Data from mapping of the slots combined have been analyzed together with information on blast design from the sites. The collected data set consists of 58 observations and 8 variables. The variables have been divided into design variables, geology variables and results variables, see Table 5. Examples of correlation coefficients between the results, design and geology variables using linear regression are presented in Table 6.

Based on Table 6 it can be concluded that charge concentration

**Table 1**
Geology and project descriptions for the five sites.

| Tunnel site | Geology | Project description |
|---|---|---|
| 1) Äspö HRL | Diorite, intersected by granitic and pegmatitic dykes | The Äspö HRL expansion project included the excavation of two new access tunnels with several experimental tunnels on the −410 m level. Several of these experimental tunnels were designed with dimensions similar to the deposition tunnels in a KBS-3 repository for spent nuclear fuel. Investigation slots were excavated in two of those tunnels |
| 2) Gerums tunnel, Tanum | The dominating rock type at the site is a porphyric middle grained granite with pegmaitite dykes. The dominating fracture set is steep to vertical. There are also sub-horizontal fractures in the area | The tunnel is part of the E6 high way between Pålen and Tanumshede with two 250 m long tubes. The investigation was conducted in the right most tunnel (Looking north) |
| 3) Kista tunnel | Gneiss-granite with pegmatite dykes | With the construction of new residential areas in Kista, the waste water system was in need for an expansion and new sewer pipes was to be placed in a 1.2 km long tunnel at shallow depth. Investigation slots were excavated in the main tunnel and in a turning niche. Charging in test sections was conducted with larger charge concentrations |
| 4) Norsborg subway depot | Middle grained gneiss with clay or calcite filled fractures with foliation of around 20 cm thickness | The depot is located between 18 and 37 m underground and connected with the Norsborg subway station. Investigation slots were excavated in a smaller tunnel perpendicular to the three main caverns |
| 5) Sandvik test mine | Porphyritic, medium to coarse grained granite. The rock mass is fairly homogenous, massive and mostly sparsely fractured | Four rounds were excavated in the facility with the purpose to study the extension of the EDZ by means of geophysics and fracture mapping. Two of the rounds were excavated with emulsion explosives. Results from mapping of investigation slots in these two rounds were included in this study |

**Table 2**
Summary of blast designs applied at the tunnel sites. Ø48 mm drill bits were used at all sites except in one of the rounds in the Sandvik test mine. In that round Ø41 mm bits was used in the contour.

| Tunnel site | Theoretical section | Initiation of contour holes | Hole spacing in contour | Design charge concentration [kg/m] | Explosive |
|---|---|---|---|---|---|
| 1) Äspö HRL | 19 $m^2$ | Electronic detonators | 0.5 m (contour)<br>0.75 m (floor) | Contour 0.35 and floor 0.5 | Kemitti 810 |
| 2) Gerums tunnel, Tanum | 126 $m^2$ | Pyrotechnical detonators | 0.6 m | Contour 0.35 and bottom charge 1.8 | Riomex UG |
| 3) Kista tunnel | 23 $m^2$ | Pyrotechnical detonators | 0.6 m | Contour 0.35,<br>Test with 0.8 and 1.2 | Kemitti 810 |
| 4) Norsborg | 38 $m^2$ | Pyrotechnical detonators | 0.6 m | Contour 0.35 and bottom charge 1.8 | Riomex UG |
| 5) Sandvik test mine | 20 $m^2$ | Different designs, both electronic and pyrotechnical detonators | 0.4 m<br>0.5 m (0.75 m floor) | Contour 0.35, 0.4 and 0.5 | Kemitti 810 |

seems to have some influence on the result variables even if the overall correlation coefficient is low and varies significantly between different result variables ($R^2$ = 0.6–44.2). The variables *Design hole spacing* on the other hand seem to have no correlation to the result variables in the data set. A correlation between *Max blast fracture length* and *Number of blast fractures* can be observed. There is an indicated correlation between both *Max blast fracture length* and *Total blast fracture length* and the variable *Density natural fracture near halfpipe*. When a linear regression is applied, the correlation is, however, limited to a $R^2$ of 6.1% for the variable *Max blast fracture length* and 10.1% for the variable *Total blast fracture length*.

### *5.3. Results from multivariate data analysis*

The choice of a multivariate statistical method for evaluation is motivated by the low $R^2$ of several single variables (Table 6) indicating a possibility that a combination of several input parameters may have higher influence on the result variables.

The collected data set was used to construct a PC model with the variables described in Table 5. The model has a total degree of explained variance of 60.5% for the first two components. A third component explains an additional 14.3%. The score plot for the first two principal components is shown in Fig. 7. The data presented in the score plot can be separated into three different groups, denoted Group a, Group b and Group c. The majority of mapped halfpipes from blast holes with few relatively short fractures and low design charge concentration can be categorized into Group a. Outliers with a single relatively long blast fracture can be categorized into Group b. Blast holes mapped in the bottom charge are represented by Group c. Some score points between the groups (a) and (b) are indicated by halfpipes with relatively few and long fractures. The score points between groups (a) and (c) represent blast holes with larger charge concentrations and more mapped blast fractures, for example the holes charged with larger concentrations from the Kista tunnel and some of the bottom charge data with fewer and shorter blast fractures. Floor holes from Sandvik's Test Mine in Tampere and Äspö HRL are also represented here.

In Fig. 8, the variables *Design charge concentration*, *Number of blast fractures* and *Total blast fracture length* are positively correlated. The variable *Max blast fracture length* is also relatively well correlated to *Design charge concentration*. However, in contrast to the other result variables, there is a larger influence from the second component for this variable.

The geology variables *Density of natural fractures* and *Density of natural fractures near halfpipe* tend to have a negative correlation to the results variables *Number of blast fractures* and *Total blast fracture length* indicating a connection between natural rock mass properties and the induced damage by blasting. The variable *Design hole spacing* seem to have limited influence on the results variables and the variable *Min blast fracture length* is not found to be correlated to any other variable in the data set.

The loadings plot for the first and third principal component is shown in Fig. 9. It can be observed that both the natural fracture density variables are to some extent loading on the third component. In contrast, the variables *Design charge concentration* and *Min blast fracture length* only load on the third component to a limited extent.

As can be expected, a larger charge concentration increases the extent of blast fracturing and a high density of natural fractures decreases blast fracturing to some extent. By observing the loadings plots (Figs. 8 and 9) it can be concluded *that Design charge concentration*, *Max blast fracture length*, *Total blast fracture length* and *Number of blast fractures* mainly load on the first component. The two latter variables load also to a limited extent on the second and third component. The variable *Max blast fracture length* loads on the second and third components to a larger extent compared to the other result variables. The variable *Density natural fractures* loads only on the first and third components. However, the variable *Density natural fractures near halfpipe* loads on all three components.

If a PC model is created excluding the variables that have little influence on the resulting blast fracturing, *Min blast fracture length* and *Design hole spacing*, $R^2$ reaches 77% for the two first components. If an additional component is added, $R^2$ is increased by another 10%. If the variable *Density natural fracture near halfpipe* is excluded in addition to

**Table 3**
Number of slots and mapped halfpipes at each site.

| Tunnel Site | Slots | Number of mapped halfpipes | Reference |
|---|---|---|---|
| 1) Äspö HRL | 3 Wall slots<br>2 Floor slots | 20 | Ittner et al. (2015) |
| 2) Gerums tunnel | 1 Wall slot (wall)<br>1 Wall slot (bottom charge) | 6 | Olsson et al. (2015) |
| 3) Kista Tunnel | 5 Wall slots | 11 | Ittner et al. (2017) |
| 4) Norsborg | 1 Wall slot (wall)<br>1 Wall slot (bottom charge) | 10 | Olsson et al. (2015) |
| 5) Sandvik Test mine | 2 Wall slots<br>2 Floor slots | 11 | Kantia et al. (2016) |

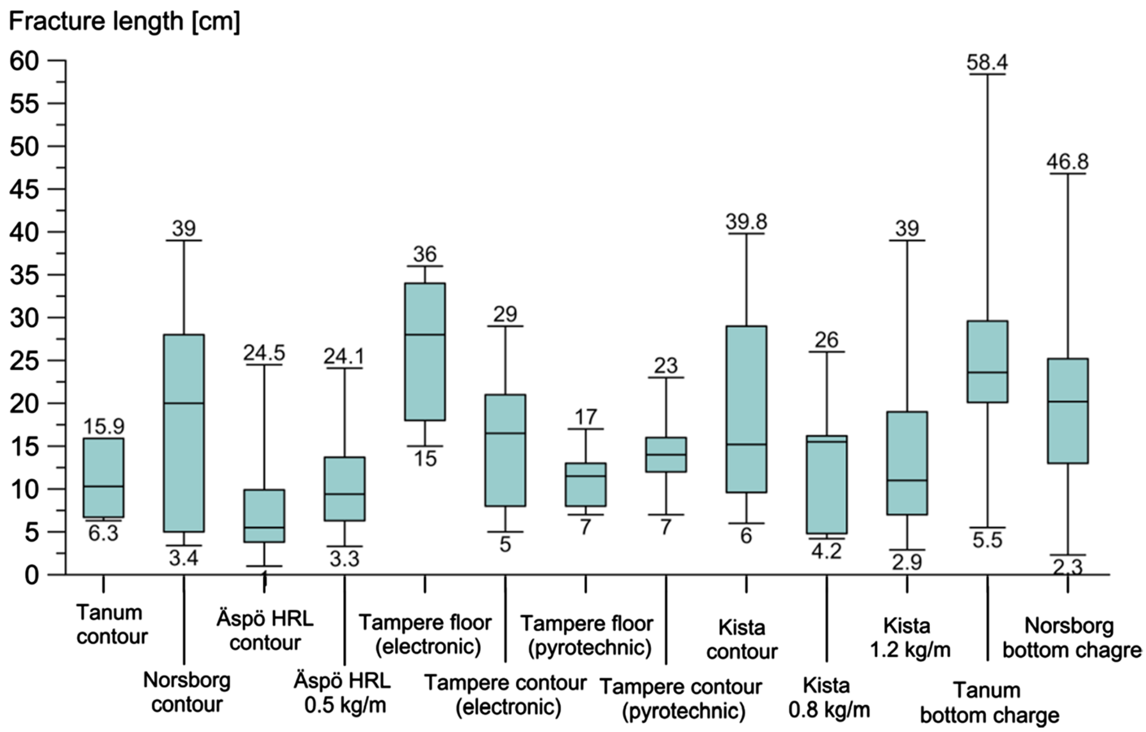


**Fig. 5.** Fracture length for each mapped blast fracture. See Table 1 and 2 for conditions and blast design at the sites.

**Table 4**
Number of blast fractures for each mapped halfpipe displayed as mean and standard deviation.

| Tunnel site and hole type | Mean/standard dev. [cm] |
|---|---|
| Tanum contour | 3.50/2.12 |
| Norsborg contour | 2.00/1.26 |
| Äspö HRL contour | 3.00/2.09 |
| Äspö HRL floor and other 0.5 kg/m | 5.25/5.19 |
| Tampere floor (electronic) | 9.00/– |
| Tampere contour (electronic) | 2.50/1.00 |
| Tampere floor (pyrotechnic) | 2.33/1.53 |
| Tampere contour (pyrotechnic) | 4.67/2.52 |
| Kista contour | 3.25/1.71 |
| Kista 0.8 kg/m | 1.33/0.58 |
| Kista 1.2 kg/m | 4.50/3.11 |
| Tanum bottom charge | 12.50/6.24 |
| Norsborg bottom charge | 11.50/7.5 |

the previous changes, the two first components reach an $R^2$ of 82.7%.

## 6. Discussion

Based on observations by Kukolj et al. (2018), it is feasible to assume that the fracture patterns mapped in the excavated slots represents the fracturing along the column or bottom charge of the blast hole and that variation in fracture length and number of fractures along the hole is limited. The length of individual blast fractures in the collected data set varies from a few centimeters up to approx. 40 cm. One exception is the bottom charge part where the length of the mapped blast fractures extends up to approx. 60 cm. These findings are in line with an estimated longest fracture length of approx. 40 cm for single and 25 cm for simultaneous initiation of contour holes based on the blast damage table by Ouchterlony et al. (2010) for the charge concentration 0.35 kg/m. However, the bottom charge would generate a longest blast fracture of approx. 175 cm according to the blast damage table. This indicates that there is a need for an update of the blast damage table for the larger charge concentrations in the bottom charge. Increased number of blast fractures, observed with larger charge concentration, is in line with findings by Kukolj et al. (2018).

The result from the PCA gives indications on how blast design and geology variables interact with different aspects of blast fracturing. The result suggests that the second and third component generally describes variations in the geology while the first component describes influence from charge concentration and natural fractures on the resulting blast fracturing. The difference in correlation between the variables *Total blast fracture length* and *Max blast fracture length* could for example be explained with a larger influence from geological variations on longer blast fractures.

The findings indicate that natural fractures can both extend and limit blast fracture length, depending on the situation. Results from the PCA show that the number of blast fractures and total fracture length are more correlated with charge concentration than the length of the longest fracture. An explanation to this could be that natural weakness in the rock mass could ease the fracturing, leading to randomness in the extension of longer blast fractures. This suggests that the longer blast fractures could be more dependent on geology compared to shorter blast fractures. It could also explain the fracture pattern of the group of outliers with just a few or a single long blast fracture.

The density measure ($m/m^2$) of natural fractures seems to describe a system of natural fractures to some extent blocking growth of blast fractures. As described by Ouchterlony and Olsson (1998) and Ouchterlony et al. (1999) natural fractures can both extend and limit blast fracture length depending on the situation. The findings from the PCA confirm this. A possible influence on fracture length due to natural fractures could be observed as outliers in Group b (Fig. 7) as well as in the difference in correlation between the variables *Max blast fracture length* and *Total blast fracture length* to *Design charge concentration*. In those cases, the fracture length could have been extended by natural fractures or weaknesses in the rock mass.

Two of the variables, *Min blast fracture length* and *Design hole spacing*, have little influence on the resulting blast fracturing in the data set. In the case of *Design hole spacing* the limited influence could be due to a limited variation between the studied tunnels. Hole spacing at the studied sites is also short relative to other studies. Olsson (2000)

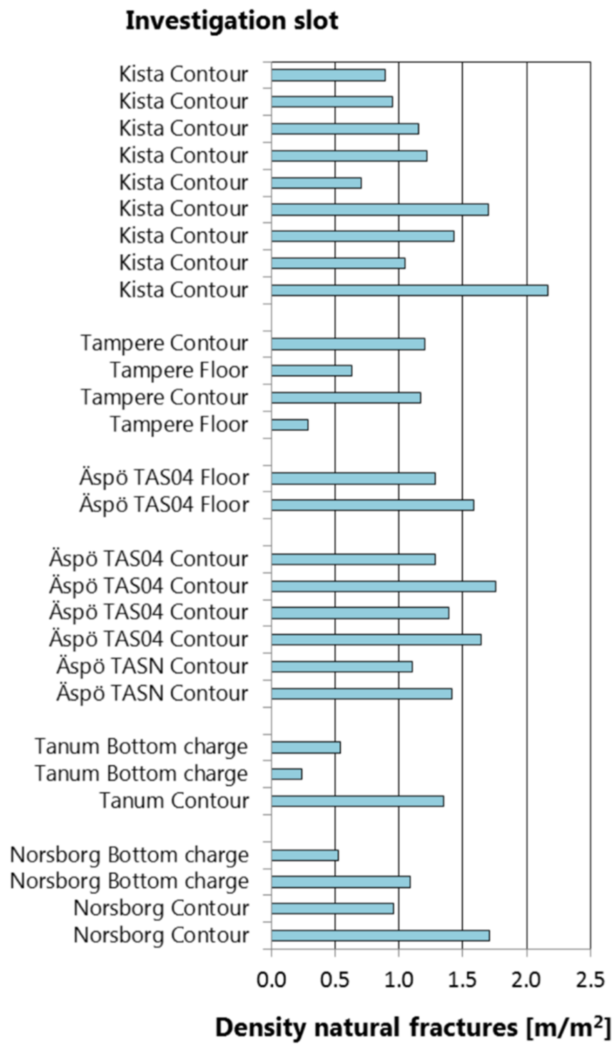


**Fig. 6.** Density of natural fractures for each mapped slot side.

reported a threshold for hole spacing near 0.8 m where an increase up to 1.2 m resulted in a change in blast fracture patterns with longer blast fractures, parallel to the tunnel perimeter, as well as a small increase in blast fracture length due to increased hole spacing from 0.5 to 0.8 m. The upper limit for design hole spacing in this study was 0.75 m for the floor holes in the experimental tunnel TAS04 in Äspö HRL and the limited influence is comparable to the results reported by Olsson (2000) for an increased hole spacing up to 0.8 m.

**Table 6**
Overview of the correlation between the results, design and geology variables using linear regression.

| Variables evaluated with linear regression | | $R^2$ [%] |
|---|---|---|
| Total blast fracture length | Design charge concentration | 44.2 |
| Number of blast fractures | Design charge concentration | 39.1 |
| Max blast fracture length | Design charge concentration | 21.9 |
| Min blast fracture length | Design charge concentration | 0.6 |
| Total blast fracture length | Design hole spacing | 1.0 |
| Total blast fracture length | Density natural fractures near halfpipe | 10.1 |
| Max blast fracture length | Density natural fractures near halfpipe | 6.1 |
| Max blast fracture length | Number of fractures | 34.4 |

Due to the fact that bottom charge design was similar at the sites, the influence from increased borehole pressures in blast holes with larger bottom charges that were reported by Bjarnholt et al. (1988) should give a uniform influence on the results.

There are several aspects of the blasting process that are not quantified in the PC model and may explain some of the noise in the data set. Such aspects are drilling precision (Ivanova, 2015), texture or grain size of the rock types (Atkinson, 1987; Nyberg et al., 1999), orientation of natural fractures in relation to the tunnel (Olsson and Ouchterlony, 2003), initiation method (Langefors and Kihlström, 1978; Ouchterlony et al., 2010) and the possibility of water filled blast holes (Ouchterlony and Olsson, 1998). An additional source of noise is variations between design data and actual charge concentration. Further research should focus on these aspects.

## 7. Conclusions

The following conclusions have been drawn based on the result from mapping and the statistical evaluation:

- This study has identified three main groups of fracture patterns; blast holes generating relatively few and short fractures, blast holes generating several longer fractures and blast holes generating few or a single long blast fracture.
- Blast fractures length varies from a few centimeters up to approx. 40 cm in the collected data set. One exception is the bottom charge part where the length of the mapped blast fractures extends up to approx. 60 cm.
- Charge concentration was found to be the single most influential design variable and is more correlated to the total blast fracture length and the number of blast fractures but less correlated to the longest blast fracture.
- The designed hole spacing, that varied between 0.4 and 0.75 m, had limited influence on blast fracturing as well as on other variables. One explanation for this may be that the spacing is short compared to other studies, for example Seccatore et al. (2015) and Olsson

**Table 5**
Description of design, geology and results variables.

| Variable | Description |
|---|---|
| *Design variables* | |
| Design hole spacing [m] | Hole spacing according to the drill plan |
| Design charge concentration [kg/m] | Planned charge concentration. Bottom charges are regarded as fully charged holes |
| *Geology variables* | |
| Density natural fractures [m/m$^2$] | Length of natural fractures per area in the slots |
| Density natural fractures near halfpipe [m/m$^2$] | Length of natural fractures within 4 drill bit diameters (20x20 cm) from the halfpipe |
| *Blast fracture results variables* | |
| Number of blast fractures [-] | The number of blast fractures mapped from each individual half pipe |
| Total blast fracture length [cm] | Total length of mapped blast fractures from each individual halfpipe |
| Min blast fracture length [cm] | Minimum length of mapped blast fractures from each individual halfpipe, the shortest fracture. |
| Max blast fracture length [cm] | Maximum length of mapped blast fractures from each individual halfpipe, the longest fracture |

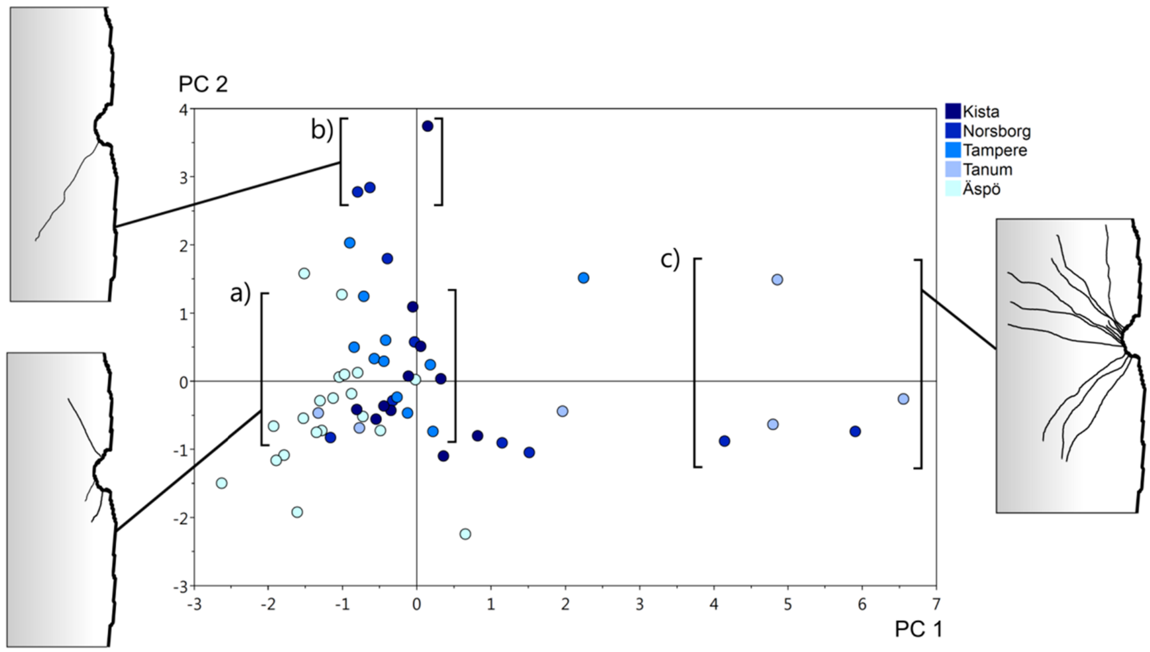


**Fig. 7.** Scores plot from PCA of the mapped halfpipes from the five sites. Holes from the same site are marked with the same color in the figure. *Group a* represents the majority of the mapped halfpipes from contour holes with few relatively short fractures and low design charge concentration. *Group b* represents outliers with a single relatively long fracture. *Group c* consists of data from the from the bottom charge.

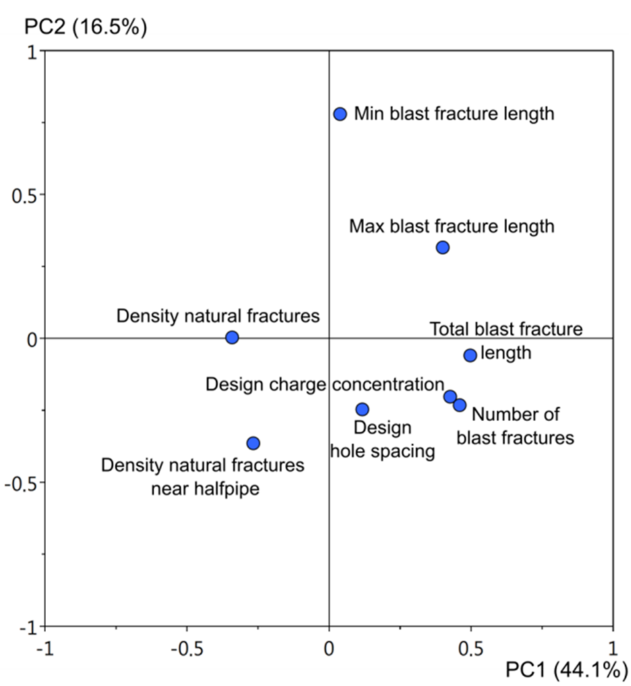


**Fig. 8.** Loadings plot for the first and second component.

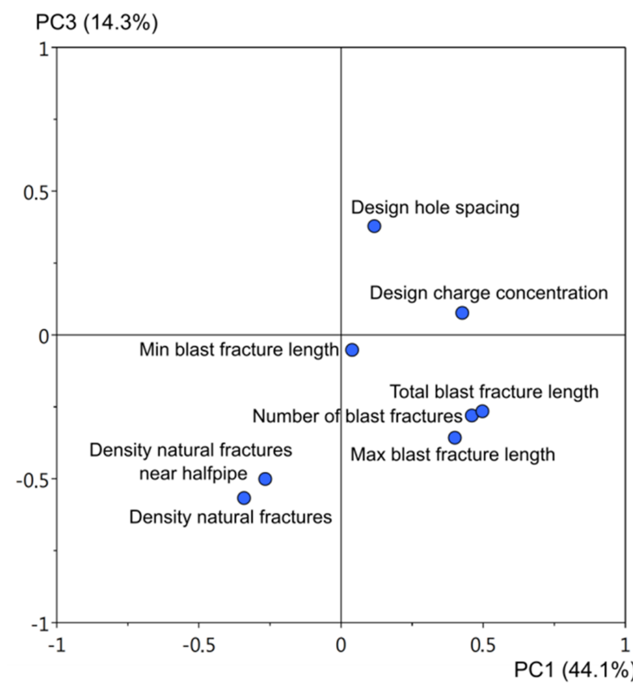


**Fig. 9.** Loadings plot for the first and third component.

(2000), were spacing was increased up to 1.2 and 1.4 m respectively. The findings corresponds reasonably well with the small increase in blast fracture length due to increased hole spacing from 0.5 to 0.8 m reported by Olsson (2000) for simultaneous initiation and the limited influence observed by Seccatore et al. (2015). Another explanation is the possible influence from differences in drilling precision between the sites.

- The suggested density measure ($m/m^2$) seems to represent the influence from a system of natural fractures blocking blast fracture growth to some extent. This indicates a connection between natural rock mass properties and the induced damage by blasting.

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