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Vegard Olsen

Rock Quarrying

Prediction Models and Blasting Safety

Doctoral thesis

NTNU
Norwegian University of
Science and Technology
Thesis for the degree of
Philosophiae Doctor
Faculty of Engineering Science
and Technology
Department of Civil and Transport Engineering

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PREFACE

My thesis work has been a long term project. For eight years the work has been in progress. I am very grateful to the Construction Engineering Research Group at the Department of Building and Construction Engineering at NTNU, which made it possible for me to carry out time demanding field studies over several years. Also, I have been able to adapt, include, defend and present small, but important research results appearing from the many master theses completed in cooperation with me at NTNU. Alone these results may appear as slender observations; however, together over time continuous research and documentation reveal trends and facts approved for developing new and updating existing prediction models.

The research philosophy, which is strongly rooted at the Construction Engineering Research Group, is based on the ideas and visions of the deceased Professor Odd Johannessen. I am greatly honored to have become a part of his practical engineering research methods.

The aim of the thesis is to provide a toolbox for the planning of quarry design, blast design, drilling and blasting, and loading and transport. My thesis is meant to be a practical tool to be used throughout the whole life of operating a quarry.

Parts of my thesis are based on previous reports published at the research group (e.g. the prediction models for blast design, advance rate and production costs), and should be regarded as updated and improved versions of the previous versions. This confirms my part of the long-term research and development work in the research group, aiming for improving the quarrying industry. Likewise in a few years, some of the models and data presented here will be updated and improved, based on new field studies of new technology, new machinery and improved performances.

The Construction Engineering Research Group started its involvement in rock quarrying, and particularly in rock fill dam constructions, in the early 1970s. I have been involved in quarrying research and development since 1998, when I started to work at NTNU. During these years I will particularly address my thanks to Professor Amund Bruland, whom has guided me through the long term process of this PhD and has been my supervisor par excellence.

I will send my gratitude to students and colleagues whom have contributed in the field work necessary to accomplish the thesis. Also the persons and organizations in the quarrying industry that have contributed in this thesis, and in general to the other research activities at the department. Further acknowledgements are given in Section 1.3 and Appendix A.

Last but not least, I send my warmest thanks to my closest family:
"Ragnhild, Nora, Henrik and Håkon: I love you so much!"
"Mom and Dad: Your support back home, during the last half-year of the thesis work, has been priceless. You are absolutely fantastic!"

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1 Description of the Thesis

This thesis consists of 6 volumes (one summary and five reports) as specified below:

12A-08	ROCK QUARRYING Quarry Design
12B-08	ROCK QUARRYING Blast Design
12C-08	ROCK QUARRYING Bench Drilling
12D-08	ROCK QUARRYING Loading
12E-08	ROCK QUARRYING Transport

Finally, the volume **ROCK QUARRYING Prediction Models and Blasting Safety,** which includes background and results from in-situ rock drilling testing, a paper published in the Journal of Blasting and Fragmentation, along with a summary of the working related to the listed reports.

The toolbox of reports is to some extent based on previous versions of rock quarrying reports. See reference [1] through [10]. Reports 12A-08 and 12E-08 have not been published previously.

All reports in the toolbox are prepared by Vegard Olsen, though colleagues and master students have been helpful in collecting field data and organizing the data in spreadsheets. External assistance has been used to some extent for translation and proof reading. The laboratory rock testing program has been performed at the engineering geological mechanics laboratory at NTNU/Sintef.

1.1 OBJECTIVE AND OVERVIEW OF THE THESIS

The overall goal of the thesis has been to improve the existing rock quarrying toolbox of reports and finalize an extended series within the topic. Extensive field data collection from quarries and construction sites has been carried out to accomplish the task. See Appendix A for all sites and projects visited. All the quarrying processes are studied, however, most emphasis are given to rock drilling safety and drilling performance, and the thesis includes two specific projects within these topics. Hence, the complete thesis work consists of the following three main parts:

1. Improvement of the existing prediction models, concerning design, capacity and cost estimations of rock quarries, developed at NTNU, and the publication of an extended Rock Quarrying report series.
2. Drill bit characteristics and their influence on production capacity, drilling deviation and blasting safety.
3. Rock drilling safety, which includes a model for risk assessment and a comparison of which operation, bench cleaning or rock debris drilling, will have the highest risk of fatal injury concerning detonation of misfires on the bench.

<u>Rock Quarrying Estimation Models</u>

The various reports in the toolbox should describe the quarrying industry from location, placement and design of the quarry, through the blast design planning, drilling and blasting, and finally the loading and transport of the rock mass. Crushing and further treatment or use of the rock mass will not be a part of the thesis. The existing NTNU model reports, and the new ones, will be an effective toolbox for quarry owners, consultants, contractors, manufacturers and others, to improve their work in the rock excavation industry. The toolbox may be used for:

- Optimizing the quarry design and infrastructure regarding esthetics, environment, safety and effective quarry operation
- Blast plan design
- Estimating time and cost of the drilling, loading and transport operations
- Optimizing the operation of the quarry performance by follow-up
- Verifying machine performance and selecting suitable equipment
- Establishing and managing price regulation in contracts

Drill Bit Characteristics

The scope of this part of the thesis is to obtain field test measurements on varying drill bit designs according to drilling capacity and deflection. Particularly in small to medium size quarries with top hammer drill rigs and drillhole diameters ranging from of 76 mm to 102 mm. The purpose of the work is to increase the general knowledge of rock drilling. This includes education of students and updating the report series within the heavy construction engineering field at NTNU.

The thesis will briefly present the basics of rock drilling and rock breaking mechanisms, as well as the drilling methods. Others have described this thoroughly before. The classification of different deviation phenomena will also be presented shortly. The general hole straightness efforts are described more in depth. General capacity and economical issues regarding drill bit designs are given, and safety and economical issues are also presented.

Net penetration rate studies have been carried out at our department since the early 1970s, however not as detailed as in this work. Manufacturers of drilling equipment are carrying out many tests concerning different drilling equipment properties in their in-house laboratories and test sites. However, such data is in general proprietary. Through the literature search carried out in the thesis, no material on testing of different drill bit designs was found.

Deviation measurements are carried out in many different quarries and in various rock conditions, to document and quantify the deviations which appear in the current drilling practice in Norway. The data reveals examples of normal deviation conditions as well as very good and poor examples. Deviation reduction efforts are treated and the economical and safety benefits are discussed.

Regarding deviation, the main goal of the thesis is to clarify the effect of the easy and low cost actions available for drilling straighter holes. A practical approach towards the drilling operator is the main purpose of the test presented. Usually, in the daily operation the operator's resources and time is limited for comprehensive testing. Literature from other deviation studies are found and presented. Very little material on the effect of different drill bit designs are found, except the general experience data published in the drill rig manufacturers' handbooks.

It was not originally planned, but as a result of the testing program, wear characteristics of the different drill bits are presented. Based on these results, grinding principles and basic wear effects are presented in the theory part.

Surface Drilling Safety

One paper, which was published in the Blasting and Fragmentation Journal [32], comprises the last part of the thesis. The first part of the paper presents a risk analysis model to investigate the risk probability of fatal injuries due to misfire detonation, in cases when drilling through the rock debris of the prior sub drilled zone or cleaning the bench before drilling is optional. The second part of the paper presents results from application of the model.

The background for the project was the law text, given in the "Norwegian Explosives Regulations" [33]:

> *The top of the bench must be justifiable cleaned, secured and controlled against misfires. New drillholes must not interfere with previous drillholes with the possibility of remaining explosives.*

Despite this clearly expressed regulation, parts of the industry have used the method of drilling through the rock debris. It has been done in cooperation with their employees, as they find it is less hazardous than to clean the same rock debris by excavators.

The main goal of the project was to identify all possible causes leading to misfires, and to develop a risk analysis model to be able to find the probability of detonation of misfires by drilling or cleaning and the risk for fatal injuries.

The project presents statistical data related to fatal injuries, not only in the quarry and mining industry, but also for other occupational groups. The estimated risks may be compared to the actual risk statistics.

The model has already been used in risk assessments in several Norwegian quarries.

1.2 BACKGROUND OF THE PREDICTION MODEL

The models for estimating time and cost for drilling, loading and transport are empirical, which means that the basic numbers are based on field data and technical information about equipment and machinery. The data are mainly based on Norwegian quarries, but field studies in other countries are also included.

The collection of data is based on established study methods used at NTNU – Department of Civil and Transport Engineering since the early 1970s, with the quarry as a full scale laboratory [3] [5] [7]. The measuring is based on data in a production environment, including a wide range of rock mass and operational parameters. Scientifically this method has disadvantages, as many parameters vary simultaneously. This is compensated for by use of statistical methods in the normalization process. In my opinion, presence on site during the studies is necessary to get as precise data as possible. It is also the best practice for transferring conclusions back to the operation of the quarry later.

The method of collecting data has lead the NTNU report series in a practical rather than a theoretical direction. The close cooperation with external partners, who have asked for practical tools to be used directly in their activities, has also supported the practical development.

In order to get access to data at the quarries, in general an agreement is made to avoid sensitive information being transferred between competitive parties. Hence, our database is not available outside NTNU.

1.3 ACKNOWLEDGEMENTS

Many persons and companies deserve to be mentioned in relation to the work with my thesis. A list of quarries, construction sites and companies that in any way have contributed with information, human resources, equipment or financially are presented in Appendix A. Many persons within these institutions involved in my thesis should get a personal greeting; however there is not room for that here. Still, not mentioned, but not forgotten! I give special acknowledgement to the following parties:

Franzefoss Pukk AS which so openly has let me into their quarries and letting me use their work shop for drill bit grinding and their housing facilities. I will give a special thank to the drill rig operator Birger Skaret, who has adjusted his routines and standard production to fit in my drill bit testing program for many weeks as well as contributed his knowledge and supported my work on the bench. Thank you very much!

Verdalskalk AS and Visnes Kalk AS, and respectively the drill rig operators Terje Minsaas and Torbjørn Johansen for their support during the test program.

Furthermore I want to thank Devico AS, which has put its high-tech deviation measuring system up for free use during the thesis work. Thanks to Victor Tokle and his staff.

I will send my thanks to Atlas Copco AB (Roar Woldseth), DiaTeam AS (Jens Henrik Wold) and Sandvik Norge AS (Lars Landrø), who have supported my work by donating drill bits and grinding tools.

Also I would like to thank the students doing their master thesis in connection to my PhD:

Jarle Andreas Kristoffersen – 2003 [39]
Eirik Spilling – 2004 [41]
Torbjørn Solvoll Bakketun and Roger Gården – 2005 [42]
Magnus Stedenfeldt – 2007 [43]

Under my supervision and guiding they have contributed with important data in my thesis work.

Many thanks to Sindre Log, John Magnus Lello and Andreas Fenheim for helping me in the final phase of the thesis work. Thanks to Karin Bakke Daniels and John Daniels for translation and proof reading.

2 ROCK DRILLING

2.1 INTRODUCTION

The Rock Drilling chapter describes briefly the basics of rock breaking mechanics, rock drilling methods and rock drilling capacities. It represents parts of the external literature found on the subject, as well as knowledge and experience available at the Department of Civil and Transport Engineering at NTNU, which I partly have been involved in as student and employee.

The chapter is also used as a reference for the results presented in Chapter 6.

2.2 ROCK BREAKING MECHANISMS

The mechanisms of rock breaking are well described in the literature, and there is a consensus of the basic principles [15], [26], [27], [28] and [30]. The mechanics of rock breaking section in the handbook of Sandvik Tamrock [22] (which is based on [29]) presents the rock breaking in a quite simple and informative way, and is more or less quoted in the following.

When a tool is loaded onto a rock surface, stress is built up under the contact area. The way the rock responds to this stress depends on the rock type and the type of loading. An example of this would be the drilling method.

Looking at percussive drilling, rock breakage can be divided into four phases:

- Crushed zone
- Crack formation
- Crack propagation
- Chipping

As a tool tip begins to dent the rock surface, stress grows with the increasing load and the material is elastically deformed, see zone III in Figure 1. At the contact surface, irregularities are immediately formed and a zone of crushed rock develops beneath the indenter (the button or insert of a drill bit). The crushed zone comprises numerous micro cracks that pulverize the rock into powder or extremely small particles. Actually, 70 % to 85 % of the indenter's work is consumed by the formation of the crushed zone. The crushed zone transmits the main force component into the rock.

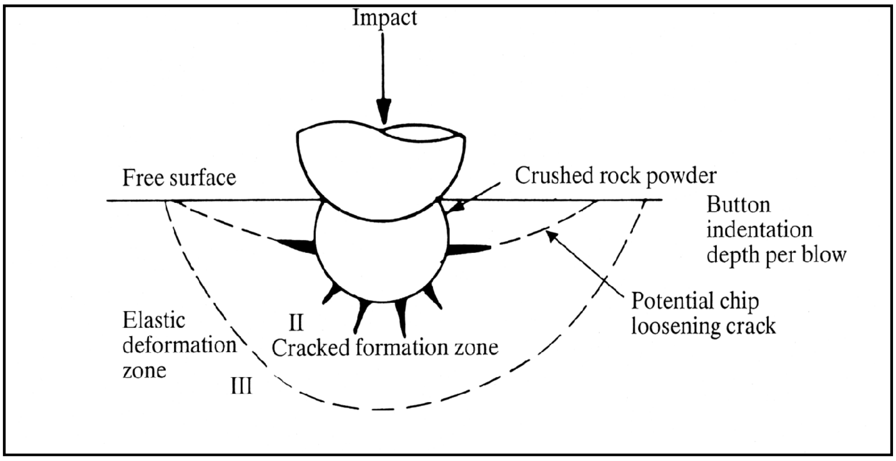


Figure 1 Rock breakage in percussive drilling [19] [29].

As the process continues, dominant cracks begin to form in the rock. This initial stage of restricted growth is described as an energy barrier to full propagation. The placement of major cracks depends on the indenter shape. Generally, the dominant placement of major cracks with blunt indenters, such as a sphere, is located just outside the contact area, pointing down and away from the surface.

After the energy barrier has been overcome, spontaneous and rapid propagation follows, see zone II in Figure 1. At a lower depth than the contact dimension, the tensile driving force falls below what is necessary to maintain growth, thus the crack again becomes stable. The crack is then said to be "well developed".

When the load reaches a sufficient level, the rock breaks and one or more large chips are formed by lateral cracks propagating from beneath the tip of the indenter to the surface. This process is called surface chipping. Each time a chip is formed, the force temporarily drops and must be built up to a new, higher level to achieve chipping. Crushing and chipping creates a crater.

Once the indented compressive stresses are released, as the tool tip leaves the surface, the strain energy in the rock is released and tensile stresses of smaller magnitude are formed in the rock. Since rock is very weak in tension, even smaller tensile stresses are sufficient to extend the cracks near the surface and cause chipping.

In addition to percussive breakage as described above, rock breaking by cutting tools is also used. Here shear forces are used to induce the tensile strength needed to break the rock. This is not common in hard rock and is basically used in soft rock formations, such as salt, coal and soft limestone [14].

2.3 ROCK DRILLING METHODS

General literature on rock drilling methods is well distributed, and the methods are often described in books related to mining and quarrying in general, such as [23] and [25]. Gokhale's Blasthole Drilling Technology book is a composition of basic drilling theory and technical descriptions of drilling equipment and accessories. In addition, the book includes areas of application gathered from most suppliers, manufacturers and educational institutes around the world [28]. As manufactures of rock drills and drill rigs, both Atlas Copco and Sandvik have during the last decades made several compendiums and technical handbooks describing the various drilling methods [14]-[16] and [19]-[22]. In the following, a short presentation is given of the four main drilling methods used in hard rock quarrying and mining, mainly based on the Atlas Copco handbook [14]. The drilling methods are:

- Top hammer drilling
- DTH, Down-the-Hole drilling (also called ITH, In-the-hole drilling)
- Coprod drilling
- Rotary drilling

Common for all percussive methods (top hammer, DTH and Coprod) is that both feed force, percussion power and rotary torque is used to break the rock, whereas the percussion power is the main crushing factor. The hammering impacts from the rock drill piston create stress waves, which are transmitted through the rod to the bit, breaking the rock by its tungsten carbide inserts, as described in Section 2.2. Different piston designs and velocities influence the shape of the stress wave and the efficiency of the drilling. Drillhole deflection is also dependent upon this [34]. Theoretically, the stress wave has a rectangular shape, which length is twice the piston length and height dependent on the piston's impact velocity and the relation between the piston and the drill steel cross section area. Rock drill design and stress wave theory is not described further here.

Top hammer rock drills are usually driven by hydraulic pressure (pneumatic are still used for smaller rigs) and mounted on top of the drill string. The top hammer method transfers both feed, rotation and percussion through one united drill string. See Figure 2. The net penetration rate is high though the energy loss is quite substantial down the hole. Between six and ten percent of the impact energy is lost over every additional standard coupling. Hence, the net penetration rate decreases down the hole. Due to the relatively slim rod, the in-hole deflections can be substantial under unfavourable rock conditions.

In conventional quarrying the top hammer diameter range is 51 mm – 127 mm, hence 76 mm – 102 mm is the most appropriate. Typical net penetration rates in medium to hard and average fractured rock for the latter range is 60 cm/min to 160 cm/min.

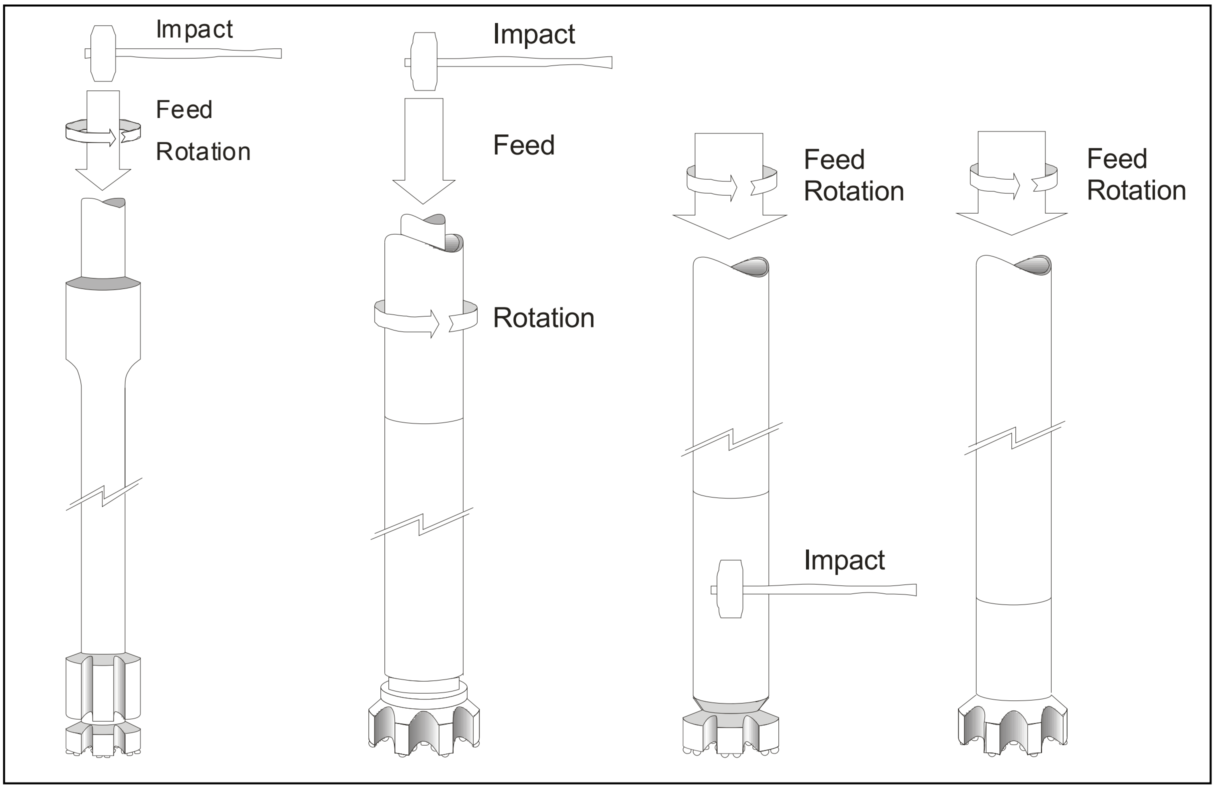


Figure 2 Illustrations of the drilling method principles.
From left: Top hammer, Coprod, DTH, Rotary [3]

The DTH method has a rock drill in front of the drill string (see Figure 2), which is run by compressed air or water transmitted through the drill string. On the whole, air is used in rock quarrying. The DTH system looses very little energy between the piston and the bit, and the net penetration rate stays nearly constant, as long as the flushing capacity is high enough. Two types of DTH rigs exist on the marked; large mining rigs (150 mm – 300 mm diameter range) and smaller quarry rigs (89 mm – 178 mm diameter range).

Using compressed air to run the piston, the net energy utilization is low and the net penetration rate is poor, compared to the top hammer and Coprod drilling ranges. Hole straightness is good.

The Coprod system combines the speed of top hammer drilling with the precision of the DTH method. The percussion is working on top of the drill string, as for top hammer drills. The rotation is transferred separately down the drill string by pipe sections enclosing the impact rods. See Figure 2. The pipes are furnished with stop lugs to hold the rods in place inside the pipe section. The Coprod sections are joined together via the drill pipes. Since the drill pipes transmit rotation force only, stress to the threads is insignificant. The negative effects of the transmission of impact energy through the threads are reduced. The working range of the Coprod rig is 89 mm to 165 mm.

Rotary crushing drilling (rotary cutting drilling will not be described here), breaks the rock by the feed force and rotary torque transmitted through the drill tube to the bit. See Figure 2. The feed force crushes the rock, and the rotary torque serves new hitting points for the buttons on the tricone roller bit as it turns the bit around. Compressed air flushing through the bit makes sure the cuttings are removed during drilling. When the buttons on the tricone bit hit the rock, the breaking principle is the same as for percussion drilling. See Figure 1. (Rotary crushing drilling is also known as rotary percussive drilling [22]).

Rotary crushing drilling is normally used in large mines or quarries with annual production volumes beyond 10 million tons. Common hole diameters are 300 mm to 400 mm. In hard rock, the net penetration rate is low, normally less than 50 cm/min. Nevertheless, the production capacity is very high due to the large diameters. There is hardly any loss of energy in the drill string and the drilling rate stays constant down the hole [3]. Hole deflection is insignificant in quarry and mining applications.

2.4 ROCK DRILLING CAPACITY

The overall goal for a quarry or mine is to get the lowest possible costs through the whole production chain. Drilling costs, including net penetration rate, are only a small part of the complete running of the quarry or mine. Nevertheless, it is important to optimize the drilling by increasing the capacity and still keeping the costs down. The drilling manufacturer industry is continuously developing drill rigs, rock drills and drilling equipment to improve quality and capacity, and to reduce the costs for the customer.

Many terms are used to describe the progress of drilling, such as: Penetration rate, instantaneous penetration rate, net penetration rate, gross penetration rate, drilling rate and drill performance [28]. In this thesis net penetration rate is used, meaning a continuous drill length divided by time. Penetration rate and instantaneous penetration rate describe the same drilling property. The two latter terms involve fixed times and are used in production capacity analysis.

To estimate drilling capacity for different drilling equipment, the net penetration rate and the fixed time in the drilling cycle must be known. Fixed time consists of:

- Moving, alignment and collaring
- Adding drill rods
- Drill rod pullback and decoupling

In addition, changing of bits and control of drillhole must be added.

Depending on hole length and hole diameter, rough estimations show that fixed times varies from approximately 25 % to 50 % of total drilling time, increasing with shallower holes and smaller diameters.

The main goal for a drilling operator is to keep the total drilling costs to a minimum. High net penetration rate is an important part of this goal, however only to a certain point. A too high net penetration rate will reduce the life of the drilling equipment, such as bits, rods, shanks and rock drills, leading to an excessive increase in the total costs. Deflection may also be influenced by the net penetration rate, as a result of unfavourable pressure settings.

At optimal use in any given drilling condition, a decreased bit diameter will give higher net penetration rate and decreased drilling costs; however the drilling costs per blasted volume will increase [3]. This is mainly due to less total drilling time per produced unit. See Figure 3.

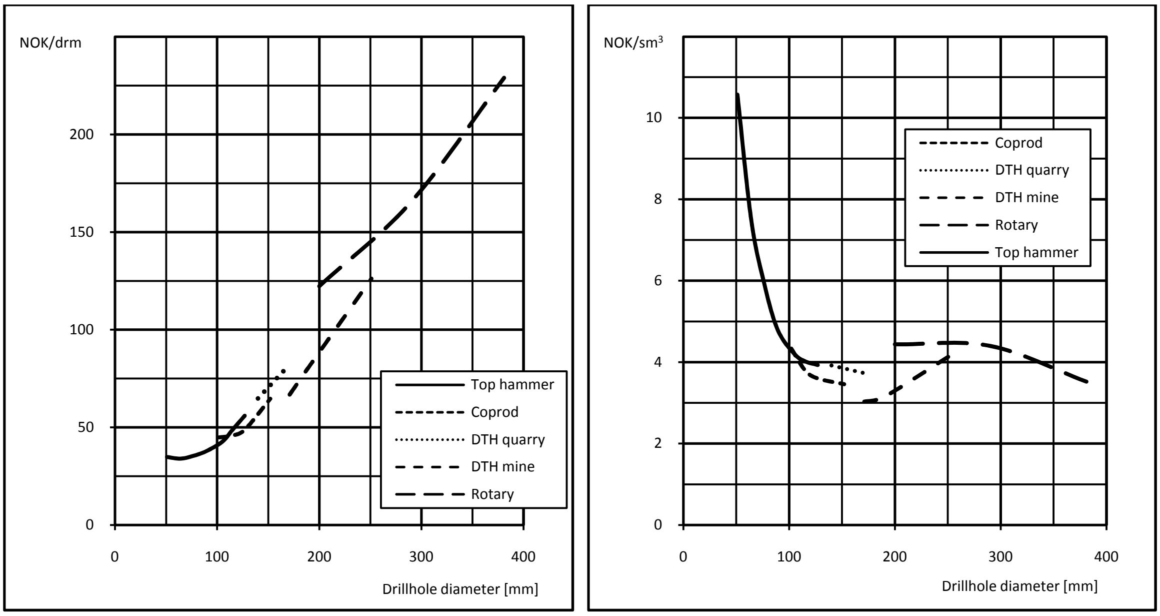


Figure 3 Drilling costs. Left: Costs per drilled meter. Right: Costs per blasted volume. Medium blastability and drillability. Ref. Volume 4.

Drill steel design properties are versatile and complex. It is said that there is one perfect drill bit design for every single drilling job. A total list of design characteristics will not be presented in this thesis, but the most conspicuous net penetration characteristics are discussed. In the following, a description of the factors influencing the net penetration rate is presented, and along with the choices the drilling operator should assess. The net penetration rate is mainly dependent upon the following factors:

A. Drilling Method
- Drilling rig type
- Rock drill type

B. Drill steel
- Drill bit diameter
- Insert type (chisel/buttons)
- Button design (shape and size)
- Drill bit design (front and skirt design)
- Rod or pipe design (couplings and threads)
- Bit-rod diameter ratio
- Drill bit condition (Wear and deformation)

C. Power settings
- Percussion pressure
- Feed force
- Rotation pressure and speed
- Flushing capacity
- Automatic pressure adjustment systems

D. Rock mass properties	Rock brittleness and elasticity
	Rock hardness
	Rock abrasiveness
	Rock mass structure (bedding, jointing, fissuring)
E. Site properties	Hole length (top hammer)
	Water properties

As drill rig, rock mass and site properties normally are fixed, the operators actual controllable parameters lie within points B and C. Nevertheless, the factors in D play an important role in adjusting the power settings.

The drilling method is normally chosen due to the total production demand in a quarry or mine. In a purchasing situation, drill rig models and rock drill types may be an issue. However when the decision is made, the net penetration rate range is fixed within the drilling method and available drilling equipment (B and C in the list above). In the following, the descriptions will be related to percussive drilling properties and particularly top hammer drilling.

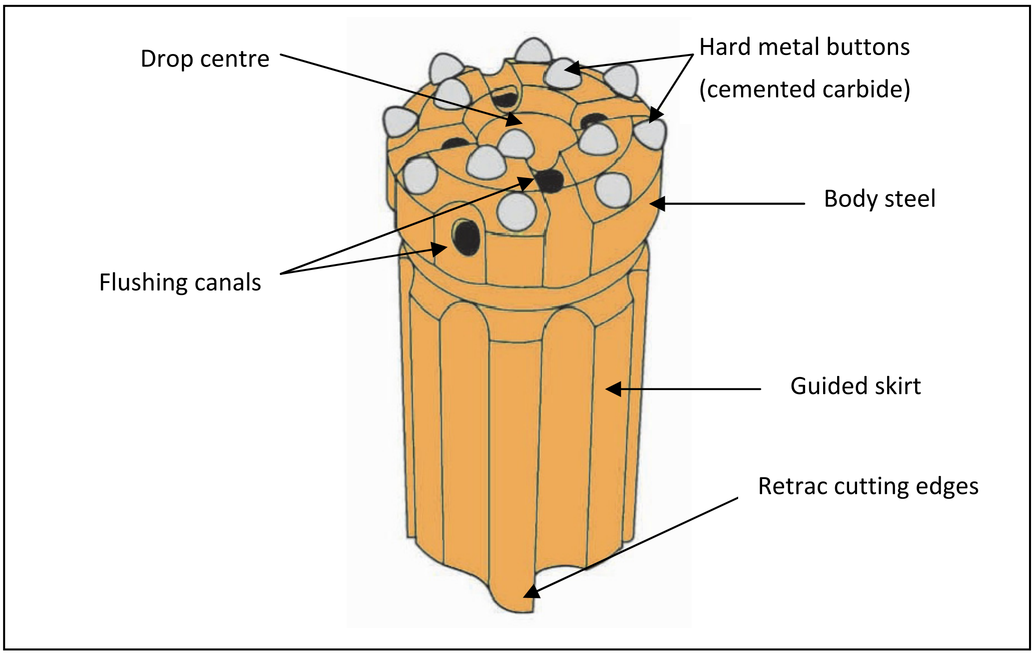


Figure 4 Illustration of top hammer button bit with drop centre and guided retrac skirt [14].

Rock Mass Properties and Drill Bit Diameter

The net penetration rate of any given drilling equipment combination is very dependent upon the rock characteristics. DRI™ (Drilling Rate Index), uniaxial compressive strength, Schmidt hammer rebound hardness, Taber abrasion hardness, point load index, Brazilian tensile strength and other parameters may be used to classify the rock and the net penetration rate [3] [26]. In this thesis the DRI™ is used. In Figure 5, Sandvik's normalized net penetration rate graphs for their top hammer series are presented. Similar graphs are found in the technical specification brochures from the equipment suppliers for most rock drills. In the report series published at the Department of Civil and Transport Engineering, and as a part of this thesis (Section 7.4), normalized field capacity studies are presented for the most common drilling equipment used in Norway, in order to present independent data to the public.

From Figure 5 we see that both softer rock (higher DRI™ value) and smaller drill bit diameters give higher penetration rates. The figure shows wide curves. This is not only due to the different rock drills presented, but also the uncertainties in the rock mass structure, the variety of drill steel available and variations in the pressures settings.

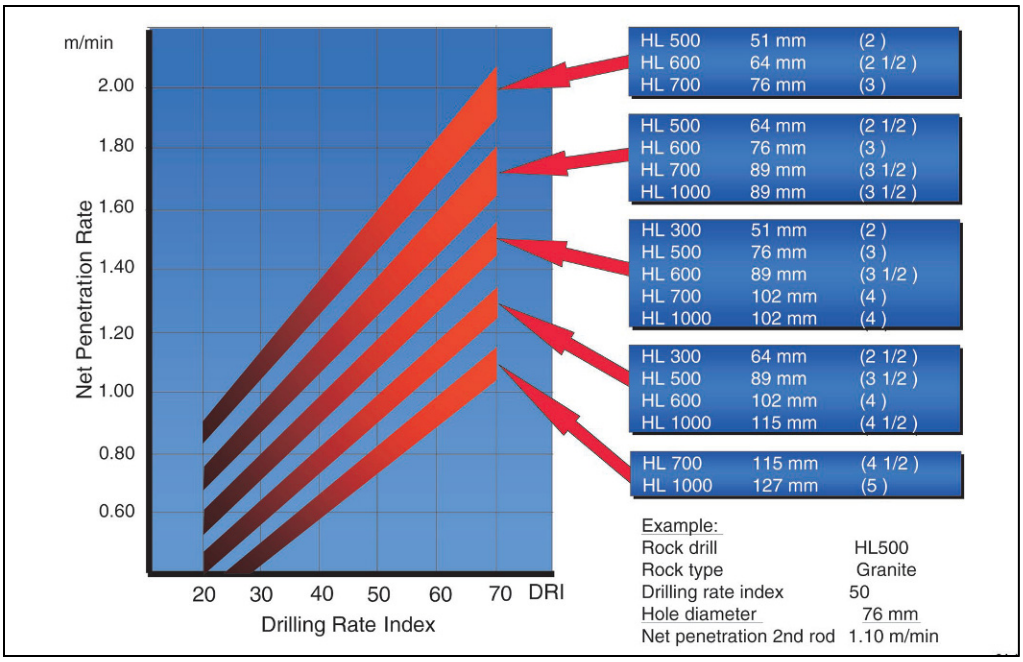


Figure 5 Expected net penetration rates with Sandvik rock drills [22].

Pressure Adjustments

The basic theory of the drilling pressure adjustments is informatively described in the Atlas Copco handbooks: Surface Drilling and Underground Mining Equipment [14] [18], and the Sandvik handbooks: Surface Drilling and Blasting and Rock Excavation Handbook [20] [22]. The text is partly rewritten and summarized below. The theory is thoroughly described in Clarks' Principles of rock fragmentation [26].

As mentioned above, the percussion pressure is the main crushing force transmitted to the rock, and it is obvious that higher percussion pressure gives higher net penetration rate. The explanation of this is higher speed of the piston and consequently higher energy put into the rock. So why do we not give full speed all the time? We can approach the rock drill's maximum pressure capacity if the bit is in good contact with hard and competent rock and the shock wave energy is utilized to its maximum. Conversely, when the bit has poor contact with the rock, the energy cannot leave the drill string, therefore reversing up the drill string as a tensile wave. Reflected energy which is too high will cause drill rod breakage and possible reduced rock drill life. In soft rock, to reduce the reflected energy, the percussion pressure, and thus the energy, will have to be lowered. See the illustration to the left in Figure 6.

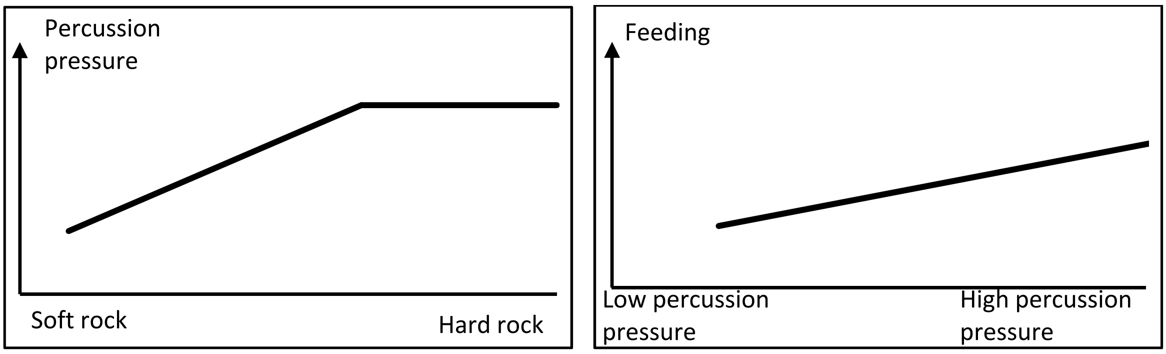


Figure 6 Pressure adjustment relations.
Left: Feed force must be matched to percussion pressure.
Right: Percussion pressure is lowered in softer rock to reduce reflected energy. [14] .

Percussion is the dominant force in rock breaking. However, feed and rotation are also important and need to be adjusted correctly. In practice, when starting to adjust the rock drill pressures, the operator first sets the percussion pressure which the rock can cope with, and then sets the rotation speed according to the percussive frequency and bit diameter. Using button bits, the periphery is turned about 10 mm between each blow. Consequently, the rotation rate is increased using higher impact frequency and reduced bit diameter. See Figure 7.

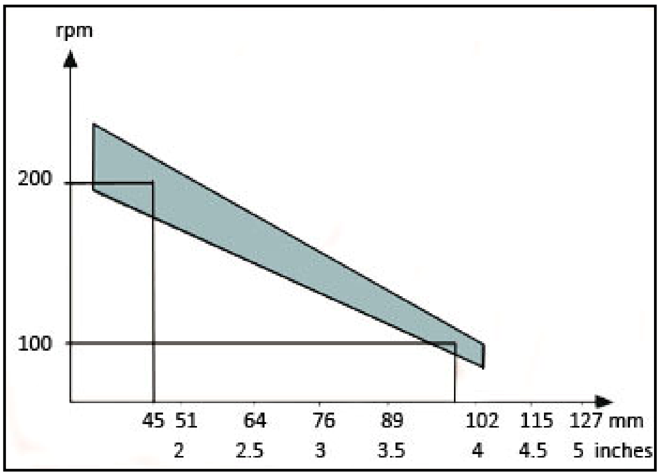


Figure 7 Working relationship between bit diameter (x-axis) and speed of rotation for a given drill [18].

When drilling starts, the feed force is adjusted to get even and smooth rotation. A feed which is too high will give irregular rotation due to high torque, resulting in low shank adapter life. It is important to notice that the feed should be adjusted to the percussion pressure, and not vice versa. There is a linear relationship between the percussion pressure and feed force. See the right illustration in Figure 6.

Automatic pressure adjustment systems are continuously optimizing the pressure combinations according to the response of the rock, improving the average net penetration rate throughout the hole depth.

From Figure 8, it seems the net penetration rate is not proportional to the percussion pressure, but increasing with higher energy input levels [21]. The results from a test, carried out by the candidate in 1999, showed that the net penetration rate decreased in the range of 12 % to 14 % when the percussion pressure was reduced with 7 %, from 140 bars to 130 bars [3].

Flushing capacity becomes more and more important as the power output of the rock drills increase and the net penetration rate gets higher. The flushing medium, air or water, must be high enough to remove the cuttings produced. Air is dominant in surface mining and quarrying. A low flushing capacity may result in packing and increased re-cutting of the cuttings underneath the bit giving reduced net penetration rate.

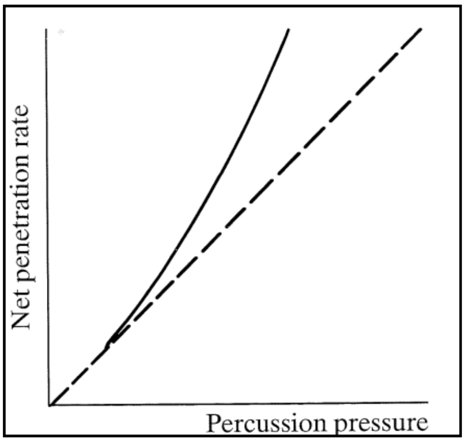


Figure 8 **The curve indicates an exponential increase in net penetration rate when percussion pressure increases [21].**

<u>Drill Steel Design</u>

In 99 % of all rock drilling applications button bits are chosen. Chisel insert bits (cross bits and X-bits) should only be chosen if extremely straight holes are required and nothing else works, and in very abrasive rock [14]. Illustration of a conventional button bit is shown in Figure 4.

The net penetration rate of the cross bits is 15 % to 30 % less than buttons bits [2] [14]. Chisel bits are not a realistic option in the hard rock mining and quarry industry. Typical standard button shapes are shown in Figure 9.

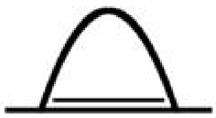
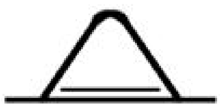
Button shape		Characteristics	Practice
	Spherical	Minimum drilling rates, low bit wear, non aggressive shape.	Rocks with high UCS and abrasivity. E.g. Quartzite and gneiss.
	Ballistic	High drilling rates, moderate bit wear, agressive shape.	Rocks with high UCS and abrasivity. E.g. Sandstone and marl.
	Conical	Maximum drilling rates, high bit wear, very agressive shape.	Rocks with low UCS and abrasivity. E.g. Phylite.

Figure 9 **Button types and their main characteristics [27].**

Due to less contact area between the button and the rock, pointier bits will penetrate more than rounder bits on each blow, and they will induce higher tensile stresses in the rock. Both these phenomenon increase the net penetration rate. The effects are more evident the softer the rock. Naturally, spherical buttoned bits are slower than ballistic bits and Atlas

Copco implies a net penetration rate reduction somewhat above 10 % going from ballistic to spherical buttons [14]. The result from the practical study mentioned above, carried out by the candidate previously, shows reductions of 11 % to 14 % [3].

The drill rod design may influence the net penetration rate. Normally rods with integrated couplings are used today. Compared to standard couplings with 6 % to 10 % energy loss per coupling, the so called speedrods reduce the loss down to 3.5 %. Longer rods and a reduced number of couplings will logically increase the net penetration rate, however registrations indicate the opposite for certain situations. Longer and more slender rods and bits of relatively larger diameters will cause more vibrations in the drill string and more loss of energy [3].

The rod-bit diameter ratio may also influence the net penetration rate due to lower flushing capacity. If the rods are too small in comparison to the bit, this will lead to decreased cutting velocity and reduced flushing capacity, resulting in lower net penetration rate as described above.

Optimal number and placement of the flushing holes are important to obtain enough flushing capacity.

3 WEAR AND GRINDING

In the following, a summary of the practical issues of drill bit wear, found in the literature study, are presented. To some extent, grinding principles will be mentioned. Again the Atlas Copco and Sandvik handbooks are used as references, [14] [20]. For interested readers, the abrasive and breakage mechanics and material alloy theories are described in [26] and [27].

Normally the drill bit buttons are worn flat by the abrasive minerals in the rock drilled. Flattening of the drill bit buttons give larger contact area against the rock, lower induced stresses and less penetrated depth, resulting in lower net penetration rates. The net penetration rate reduction increases as the wear flat increases. See left illustration in Figure 10. Normally the bits should be ground when 1/3 of the button diameter is worn (d/3). Theoretically, the net penetration rate has then decreased by 5 %. Ballistic buttons get worn faster than spherical buttons, thus the net penetration rate difference decreases as they get worn.

Following a d/3 regrinding interval, the bits are ground about 10 times before they are scrapped due to the reduced diameter. The inserts' lengths are also reduced to a minimum, and they normally start to pop out at this level of wear. Button breakage and pop-outs may cause earlier wrecking of the bits.

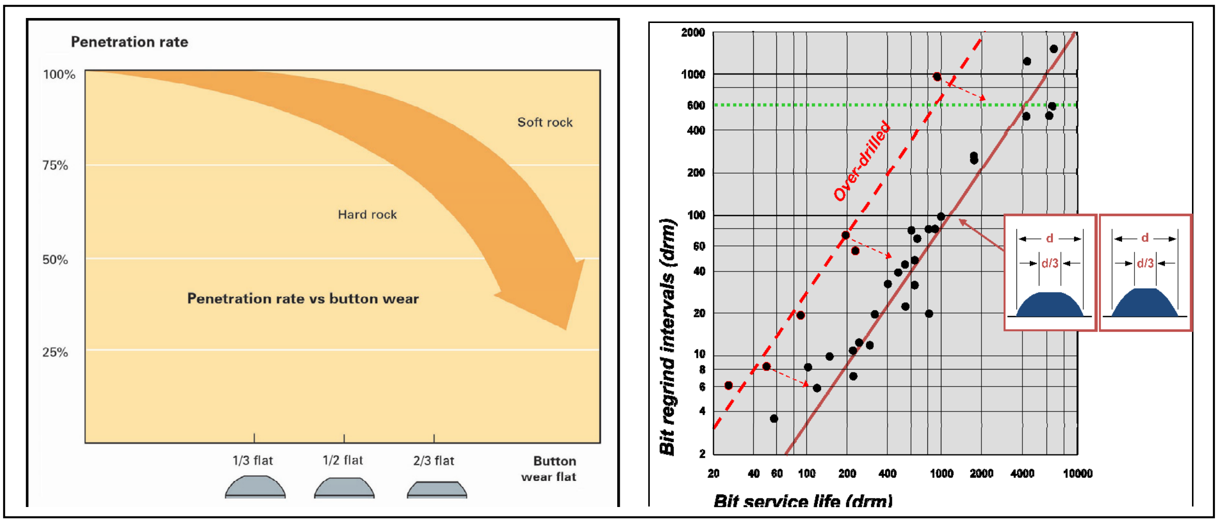


Figure 10 Left: Net penetration rate versus button wear [14]
Right: Increased bit service life due to optimal grinding intervals, found to be about 1/3 of the button diameter for both spherical and ballistic shaped buttons [31].

The d/3 regrinding interval gives more or less the lowest costs regarding bit life, regrinding labour, net penetration rate and gross productivity (i.e. bit changing time) [14] [31]. Looking exclusively at bit economy or bit life, Sandvik [31] notes that one can never grind, or in other words polish, the bit too often. Keeping perfectly shaped buttons at all times results in the lowest drill bit costs. See right illustration in Figure 10.

In practicality, a small wear flat (approx. 0.5 – 1 mm) should be visible after grinding. If grinded until no wear flat is shown, the gain in net penetration rate is less than the financial costs due to reduced bit service life.

The number of buttons, their sizes and positions in the bit face and the gauge button angle are characteristics which influence both the net penetration rate and the life of the bit. The buttons' position is well considered by the bit designer, and optimal crushing geometrics are obtained as the bit rotates between each stroke. The gauge button angle in relation to the drill bit longitudinal axis is important to stabilize the bit and keep the taper distance to the matrix steel. To a certain point, larger angles are positive due to both net penetration rate and bit life. Most gauge bits are placed with 35° to 40° angles. Larger angles reduce button life as the shear forces get too high and excessive button breakage occurs.

If keeping the same number of buttons on the bit, particularly in the gauge, larger buttons will give longer bit life as the total volume of hard metal increases. Keeping the same size while increasing the number of gauge buttons will also increase the total hard metal volume. Increased button diameter at the expense of the number of gauge buttons will not necessarily be positive. Particularly in abrasive rock, skid wear characteristics will appear on the gauge buttons, as the distance between them will be too far, and the bit circularity will be reduced.

In hard rock, to a certain point, small button bits will drill faster than bits with larger buttons. The bit life will be reduced and the choice of the operator is to optimize the total economics due to higher productivity and more service costs. However, the choice is not clear as geology plays an important role as well. In little abrasive and soft rock, larger buttons may be favorable because of their higher protruding depths. The net penetration rate will increase without decreasing the bit service life.

Ideally, the wear of the bit-body steel should follow the button wear. Too little body steel wear may cause too little button protruding depth and too little periphery clearance. Therefore, unfavorable rock breaking conditions occur, along with poorer flushing and increased torque, resulting in reduced net penetration rates. The bit-body steel must then be removed by an angle cutter or similar tool. High bit-body steel wear may lead to exposure of the inserts, causing insert pop-outs and reduced net penetration rates.

If flushing holes are deformed they should be opened by a rotary burr or steel file. Too little flushing air may cause reduced net penetration rate as described earlier.

The life of drill bits is very dependent upon the rock mass abrasiveness. More abrasive minerals give higher wear and lower bit life. Not only do the minerals influence the bit life, but also the grain size and bonding. Some common laboratory methods used for determining the wear capacity of rock are:

- Vickers Hardness Number Rock (VHNR)
- Cutter Life Index (CLI™)
- Bit Wear Index (BWI)
- Cerchar Abrasion Index (CAI).
- Wear Index (F)
- Rosiwal Mineral Abrasivity Rating

In the cost estimation model developed at NTNU [3], the bit life is presented as a function of VHNR.

From Figure 10 we see that the bit service life varies from below 100 drm in the most abrasive rock to around 7000 drm in minimally abrasive rock.

4 ROCK DRILLING DEVIATION

4.1 INTRODUCTION

The following chapter presents knowledge and experience on rock drilling deviation found in external literature and from work carried out at the Department of Civil and Transport Engineering at NTNU, in which I have contributed. This includes supervision of master theses, writing of technical reports and presentations at conferences.

The chapter presents general deviation characteristics. It discusses briefly the economical aspects and efforts to reduce deviation. A summary of measurements found in the literature are presented, as a reference for the results presented in Chapter 6.

4.2 GENERAL HOLE DEVIATION CHARACTERISTICS

Rock drilling deviation is usually divided into four different classes (see Figure 11):

- Collaring deviation
- Alignment deviation (horizontal direction and vertical inclination)
- Drilling deflection
- Drillhole depth deviation

The term "deflection" is used in this thesis as deviation arising during drilling, independent of the initial deviations (collaring and alignment). In the literature, different expressions are used, e.g. bending, trajectory deviation and in-hole deviation. "Designed path" is an expression used for hole deflection, however this relates to intended deviation when following a curved line. This method is often used in oil well drilling and other exploratory long hole drilling projects.

The total deviation is calculated by simple vector mathematics and Pythagoras triangular formula from the coordinates of the intended hole destination point to the actual hole destination point. Incidental deviations will go in any direction, and some of them will counteract each other. Either way, the total deviation and the standard deviation range will increase as the deviation factors are added.

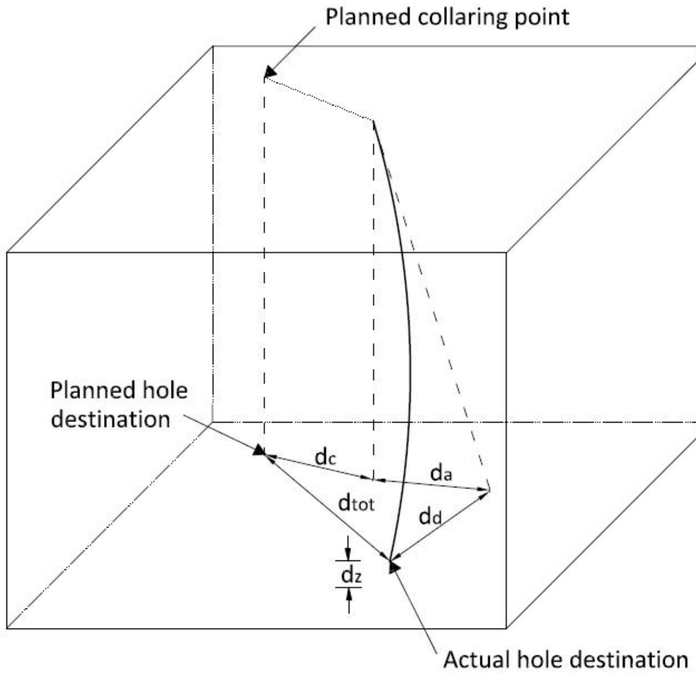


Figure 11 Diagrammatic representation of collaring (d_c), alignment (d_a), deflection (d_d) and vertical depth (d_z) deviations [34].

Collaring Deviation

Collaring deviation is a lateral displacement from a planned location. It is therefore a constant for any hole length. In general it should not exceed one bit diameter [31]. Thus:

$$d_c \le D, \qquad \text{where D = bit/hole diameter}$$

As a comparison, the Norwegian Road Administration accepts a maximum of 100 mm collaring deviation in road cuts, commonly with 64-76 mm drill bit diameter [45].

Alignment Deviation

Alignment deviation arises from inaccuracies in setting the feed boom in a planned direction. The alignment deviation is both horizontal (azimuth) and vertical (inclination). The horizontal deviation becomes less important as the inclination closes up to vertical holes. See Figure 12. The alignment deviation leads to a linear increase of deviation down the hole, normally denoted in percentage of drillhole depth or in cm/m.

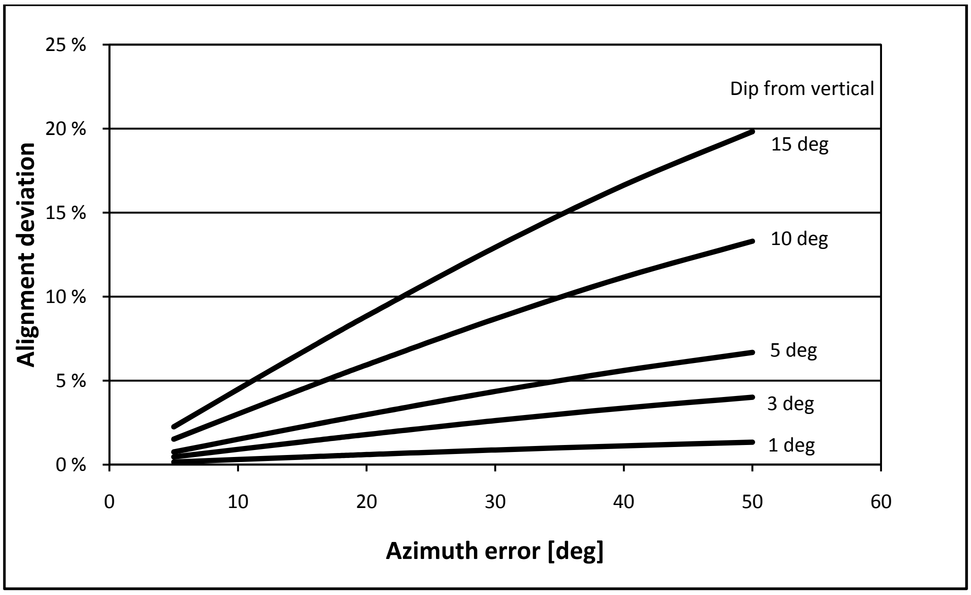


Figure 12 Alignment deviation (horizontally projected) as a function of vertical dip angle and azimuth error. As the dip closes to vertical, any azimuth error contributes less to the total alignment error.

Causes of collaring and alignment deviations include:

- Instability of the drilling rig
- Lack of precision in the surveying and the setting out process (for collaring) and the tools/techniques used to align the feed boom (for alignment)
- The topography at the collaring point (Figure 13)
- The drilling operator's experience and motivation

The Norwegian Road Administration requires a maximum of 2 % alignment deviation in road cuts [45].

Some practical experience data is found in the literature. The Swedish Road Administration's Surface Blasting handbook [46] indicates that drilling without angle control gives an average alignment deviation of 7 %. However with alignment instruments, the deviation can be limited to 1 % without increasing the alignment time.

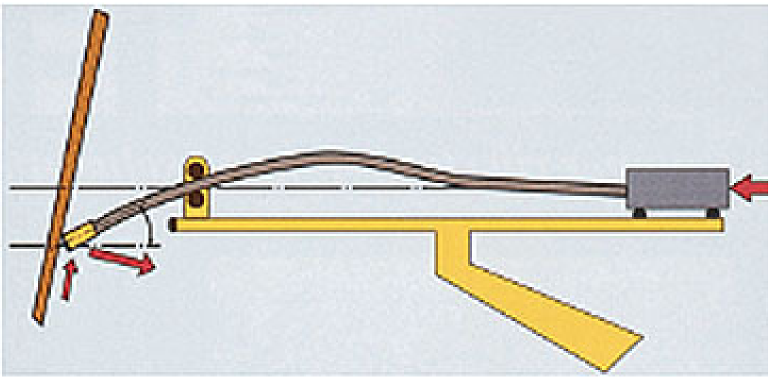


Figure 13 Improper collaring procedure will result in major deviation, measured as alignment deviation. Example from face drilling [17].

Corresponding numbers are given in one of Sandvik Tamrock's Handbooks: Manual alignment will give normal deviations from 1-4 cm/m and aligning with an angle indicator may improve the range from 0.5 –1.0 cm/m [20].

A master thesis from Luleå University of Technology [36] showed that the alignment deviation in average was around 5 %, with maximum values about 10 %, at 6 investigated quarries. No information about angle instrumentation is given.

In the Swedish Rock Engineering Research SveBeFo report [37] and the master thesis from Chalmers University of Technology [35] a study of a particular contour boring case, shows that the alignment deviation was doubled, from 1.3 % to 2.5 %, when the distance between the holes was increased from 40 cm to 80 cm. Further, the study shows that the production holes in the same blast, with less alignment demands, doubled the upper value. Both these results indicate that the operators' motivation is very important in reducing deviation. The SveBeFo report summarizes data from other third party field studies as well. However, they are not presented here, as they were not available to control and do not show particular differences from the already referred papers.

Manual angle calibration of the rig's orientation is today common. The angle instrument is turned manually towards a fixed sighting point. In some quarries the sight point happens to be relatively close to the quarry face and a systematic horizontal direction error may occur. A fan pattern is then revealed, which can be substantial in the worst circumstances (>5 %). See Figure 14.

The alignment deviation examples described above indicate that an accuracy of around 1 cm/m is more or less the optimal level of alignment error regarding the drill rig inaccuracy and cost analyses (i.e. productivity versus blasting effect). The examples indicate that the alignment deviation increases when less accuracy is demanded (e.g. contour holes versus production holes). In production blasting, it is not unusual to see alignment deviations larger than 5 cm/m. With today's equipment the alignment deviation should be closer to 1 % than to 5 %.

GPS assisted drill rigs take away most of the operator errors; however the state of the drill booms will give some deviation. The effect of GPS assisted rigs is not further examined in this thesis.

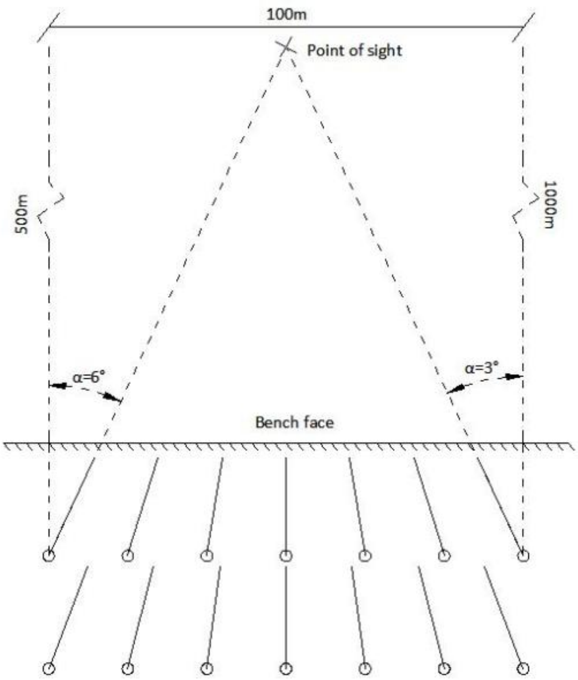


Figure 14 Illustration of systematic alignment deviation due to short distance to sighting point.

<u>Deflection</u>

While collaring and alignment deviation arise from sources prior to drilling, deflection arises from sources during drilling. Below is presented a list of factors that influence the deflection. The list is based on Sinkala's thesis [34] with some minor specifications:

A. Drilling parameters
- Thrust (feed)
- Percussion pressure
- Torque
- Rotation speed
- Flushing
- Drill string weight
- Anti-jamming system

B. Hole design - Hole inclination/direction
- Hole diameter
- Hole length

C. Equipment components - Piston design
- Chuck/shank clearances
- Couplings and treads
- Stabilizer design
- Drill bit design (Front, skirt and button design, wear)
- Drill rods design (Pipe/ rod)
- Rod/bit ratio

D. Rock and site properties- Structure (bedding, jointing, fissuring or combinations)
- Bedding dip relative to hole incidence
- Interval between joints or bed thickness
- Rock hardness
- Cohesion between beddings/foliations
- Bench floor conditions (rock debris or cleaned bench)

Apart from D, an operator or machine designer directly or indirectly can control deflection through drilling parameters and choice of equipment. Also discussed in Section 4.6.

As collaring, alignment and drillhole depth deviation is normally randomly distributed, the deflection deviation direction is both random and systematic due to the rock mass properties. Sinkala characterizes different rock conditions with different unique deviation phenomena for downward drilled holes. A generalization may be done:

In bedded and foliated rock, the drillholes are observed to either deflect parallel or normal to the bedding or foliation plane. The systematic deviation generally appears to be up-dip when the bedding inclination is less than 40°-50° to the horizontal, and down-dip otherwise. See Figure 15. Severe deflections can be measured under non-optimal conditions.

In homogenous rock the deflection is random and usually small. Increased frequency of the joint fracturing will lead to more systematic deviation, converging toward the bedded rock conditions.

Figure 16 shows the principles of deflection going from soft rock to hard rock, and vice versa. The same occurs in foliated or heavily jointed rocks, hence each structure line will be of harder or softer rock than the adjacent structure lines.

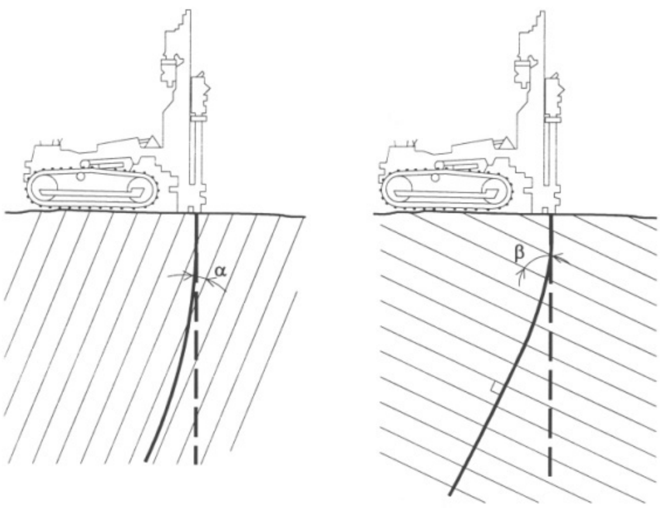


Figure 15 Common borehole deflection tendencies according to, respectively little (α) and large (β) angle with the foliation direction [47].

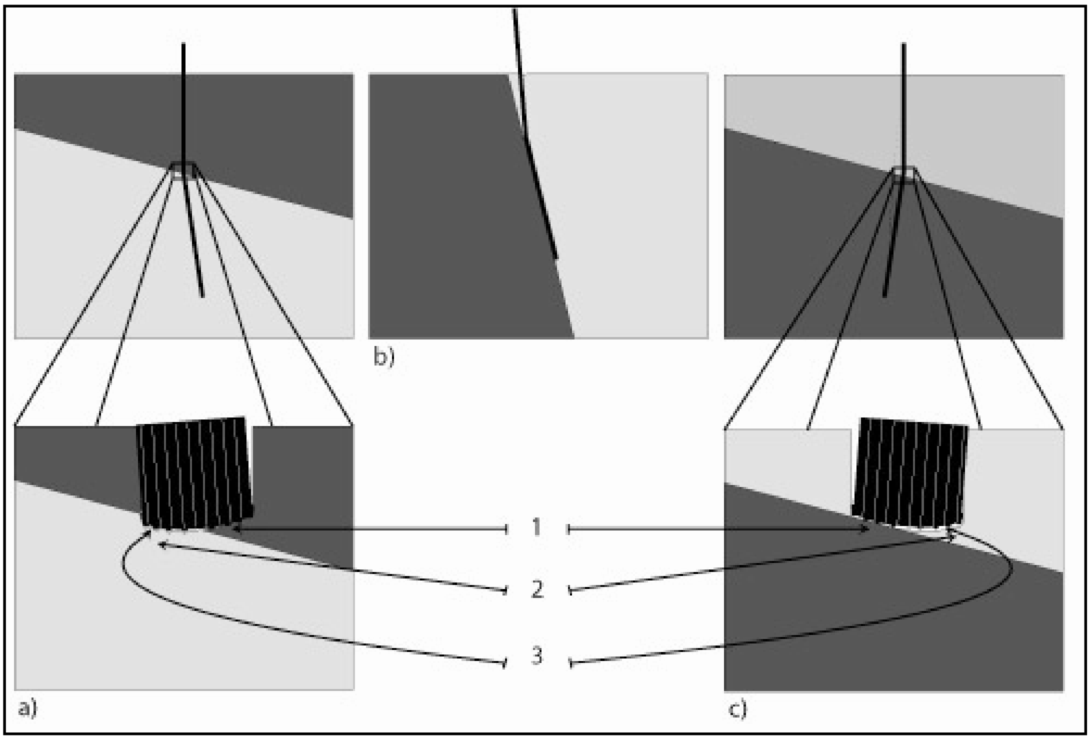


Figure 16 Drill bit deflection due to change in rock hardness. Due to different hardness, the net penetration rate will differ. At 1, the rock is harder than at 2. A moment arises and the bit will dip into 3.
a) From hard to soft rock. b) Skidding along bedding. c) From soft to hard rock. [48].

Sandvik [31] shows similar examples of deflection characteristics. They also point out the importance of button design and the drill bit skirt design. They say that ballistic buttons and their pointy shape will initially break the hard layer faster, making a cut which the drill bit can hook onto, reducing the skidding tendency. A guided skirt stabilizes the bit by reducing the diameter ratio between the bit front and the skirt. See Figure 17. Using pilot tubes the tilting effect will be even more reduced. This effect is the main effect for DTH drilling where the rock drill casing acts like a pilot tube making little room for tilting.

Another important principle illustrated in Figure 17 is the effect of counter forces acting on the bit. The soft part will have a smaller counter force than the harder part. By reducing the energy from the bit (i.e. reduced percussion and feed pressure) the counter forces will be reduced and the hole will be straighter.

When drilling through rock debris, the operator does not know the conditions of the solid rock below. If the bit hits unfavorably onto a rock slope, the bit may skid as described in Figure 16 b) and Figure 17. When the bench is cleaned, the operator will see the slant and take necessary precautions. To reduce the probability of skidding when drilling through rock debris, it is important to drill with reduced pressure until solid rock is reached.

Regarding deflection due to gravity, this is not very common in bench drilling as the deflection normally arises for holes drilled at an angle larger than 15° from the vertical.

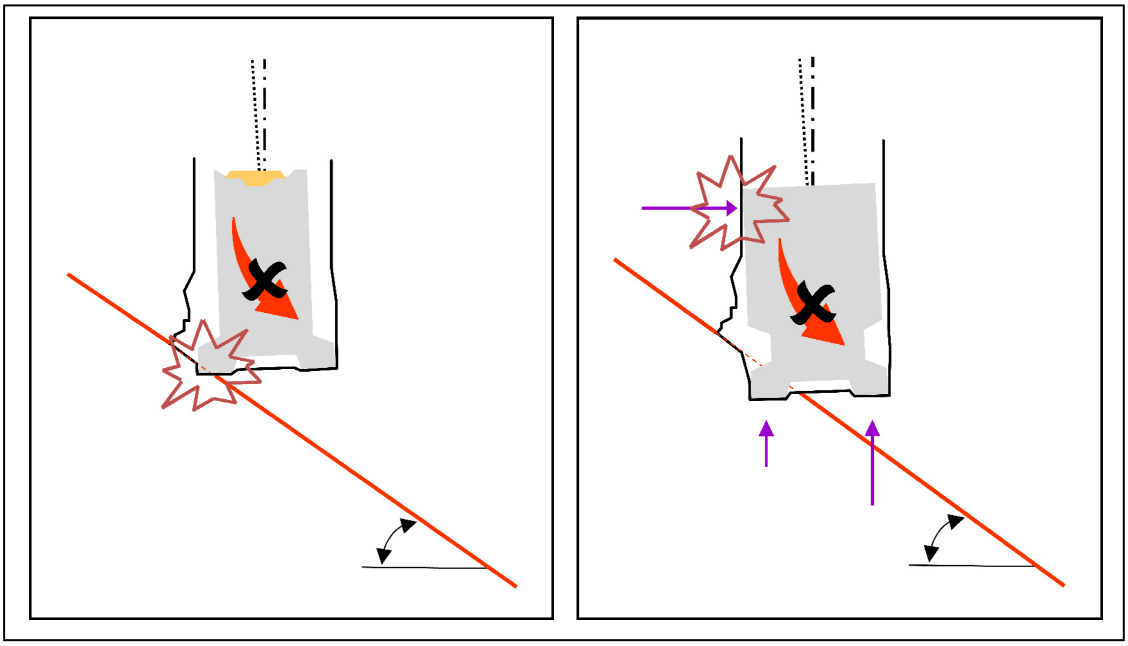


Figure 17 Reduced drilling deflection. [31]. Left: More aggressively shaped gauge buttons will reduce skidding. Right: Guided skirt reduces deflection by counteracting the moment (i.e. the asymmetric counter forces).

Depth Deviation

Drillhole depth deviation in bench blasting is defined as the vertical displacement of the planned hole bottom position, set according to sub drilling and loading level. Alignment and deflection deviations will indirectly contribute to drillhole depth deviation, even though the hole length itself is as planned. Inaccuracy or error in the depth measuring instruments on the rig can give systematic and occasional deviations. Drilling without depth measuring instruments (rare anno 2008) decreases the depth deviation accuracy and makes operator errors more probable.

Faults material, rock debris and bore dust falling into blast holes after drilling can result in larger errors than the drillhole depth deviation itself, if not noticed and removed by the charging operator. Smaller holes are more sensitive to this than larger holes. Often the operator drills the holes a few decimeters longer than optimal due to the negative economical effects of holes not being long enough.

4.3 DEVIATION MEASURING SYSTEMS AND TOOLS

If no digital measuring instruments are available, watchfulness during drilling and visual inspection of half barrels in the bench wall can give information about the hole deviation. Stuck rods, decoupling problems and high torque pressures during drilling, along with frequent drill steel breakages are factors which may indicate considerable hole deflection. On the bench walls, drillhole pipes may be seen and both alignment and deflection properties may be observed. Both rock debris thickness and bench floor underbreak may indicate drillhole depth deviation.

Flashlight measuring down the hole is an easy and practical surveying method to observe single holes and control incidental drilling problems immediately after drilling. Deviation direction can be observed, but the size of the deviation is not satisfactorily controlled.

Besides these simple measuring methods, there are many types of mechanical and digital equipment and instruments for measuring drillhole deviation. Spilling [40] is mainly used to summarize the methods. The main methods are:

- Hydrofluoric acid
- Mechanical compass and angle instrument
- Multi shot and single shot instruments (film recording or electronic)
- Gyros
- Strain gauge measuring
- Optical surveying

One of the oldest systems is the hydrofluoric acid method. The hydrofluoric acid makes an imprint in a test tube which gives the inclination of the hole. No direction is measured. The method is cumbersome and not applicable today.

Measuring by mechanical compass and angle instruments is not commonly used today. The equipment is based on one single measurement at the time, and is very time consuming.

The multi shot and single shot instruments based on a small camera filming or taking pictures of the compass and inclinometer are not competitive with the electronic multi shot and single shot systems available today. The latter systems use magnetometer and accelerometer technology to measure the direction and inclination of a drillhole. In the quarrying and mining industry, the multi shot systems are used. The electronic multi shot (EMS) instruments more or less continuously record the measuring values. The systems are vulnerable to magnetic rocks and local strong magnetic fields, e.g. drill rigs and excavators. Fixed directional rod handling systems may be used in such circumstances. Software is developed to cope with the numerous recordings to make the data easily available.

Some of the most common digital hole deviation measuring systems are listed below. For more detailed information see the manufactures/suppliers internet sites [55] to [61].

- Devico
- Atlas Copco/Transtronic
- MDL
- Pulsar
- Flexit
- Reflex

The Devico system is used in this thesis to measure drillhole deviation. A description of how to use the system is presented in Section 5.

Gyro systems are not applicable for quarries and mines. They are expensive and complicated and the diameter of the tool is too large for typical deviating blast hole diameters (less than 102 mm). The measuring principle of the gyro systems is that the gyro replaces the compass or magnetometer in the EMS systems described above. They are not sensitive to magnetic field variations.

Strain gauge measuring systems measure the bending of the tool inside the hole. Arrangements fixed to the measuring tool make it bend a little and the strain registrations show the direction of the hole. The system is not used for blast holes.

Optical surveying techniques use laser to record the direction of the hole in front of the tool. The laser reflection records the hole periphery in two distances, and the divergence in concentricity show the hole direction. The system is not applicable in the conventional bench blasting industry.

To be able to plot the measured drillholes in the current rock mass and drilling pattern, geodesic instruments must be involved. The drillhole deviation systems usually have a geodesic instrument partner with compatible software and data files. The variety of such systems and equipment is wide and will not be presented in detail here. Nevertheless, a short description of the general systems is given in the following chapter.

4.4 SAFETY IMPROVEMENTS BY DEVIATION MEASURING

Inaccurate drilling of blast holes has negative effects on the safety in rock quarrying, mainly due to uncontrolled throwing of rocks. A minor, but still important effect is related to ground vibrations.

All surface blasting include a risk of damaging nearby structures and objects, and in the worst case, injuring people. The reason for fly rock incidents is basically too high explosive energy in a given volume/area of the bench. If the burden is too small and the current explosive energy is larger than the energy needed to break the rock mass in front of the borehole, the excess energy will throw the rock away uncontrollably. See Figure 18 (A).

On the other hand, too large a burden will result in too little energy for breaking the rock mass in front of the borehole. The explosion energy will then take the path of least resistance, upward along the borehole axis. At the top, the energy will be released and will cause fly rock from the top of the bench. However, this is normally not as dangerous as the frontal fly rock since the direction of the rock will be more vertical than horizontal. See Figure 18 (B). Too large a burden will also cause higher vibration levels.

Measuring drillhole deviation makes it possible to compensate the divergent burden properties by a reduced or a increased specific charge or drilling of new holes. However, the drillhole survey is not sufficient alone and it should be supplemented by scanning of the face. An uneven blast face due to back breaking from the previous blasts is normal.

Visual inspection will reveal large cavities or toes, and adjacent holes can be individually adjusted before drilling, but smaller crucial zones may be ignored. Straight holes can in some cases be just as hazardous as deviated holes. See Figure 18 (C).

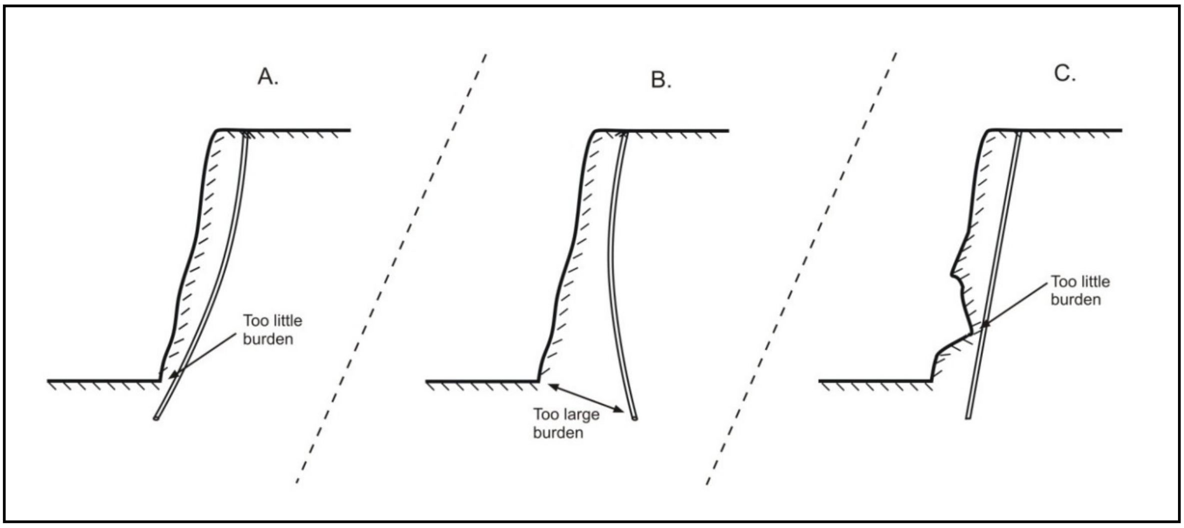


Figure 18 (A) Forward deflection resulting in little burden.
(B) Backward deflection resulting in large burden.
(C) Face cavity result in little burden even though the drillhole is straigth.

Scanning the face should be done before drilling (see Figure 19) as the holes' positions then can be changed. Practically, this may be difficult to manage as the muck pile from the previous blast often lies in front of the face while drilling. Alternatively, scanning after drilling will lead to adjustments in the charging plan. Hand-held scanners are easy applicable instruments, and can be used to control the face and the front holes' positions. The accuracy is lower than that for total station scanning, however, it is a good substitute.

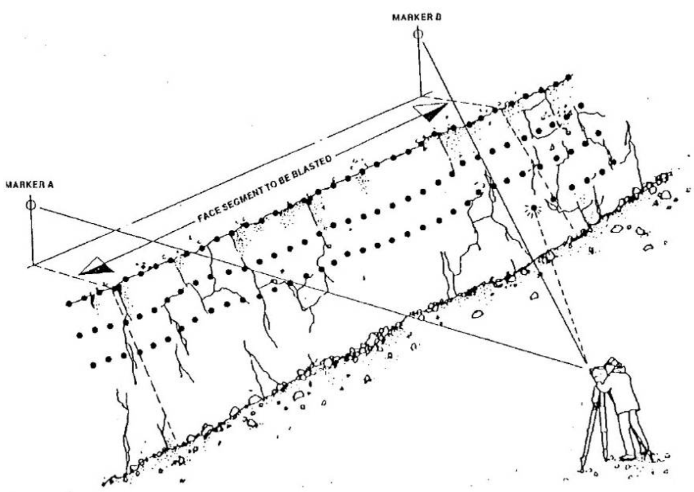


Figure 19 Set-up placement for scanning the blast face [58].

Fly rocks are normally linked to the first and second rows of the blast. On the contrary, vibrations due to drillhole deviation may arise in any part of the blast. Apparently, correct drilling patterns on top of the bench can be skewed significantly in the bottom. See Figure 20. Areas too large will lead to increased vibrations, as well as poor fragmentation and bench floor underbreak. Areas too small will give increased amount of fines, more back break and a less stable bench wall.

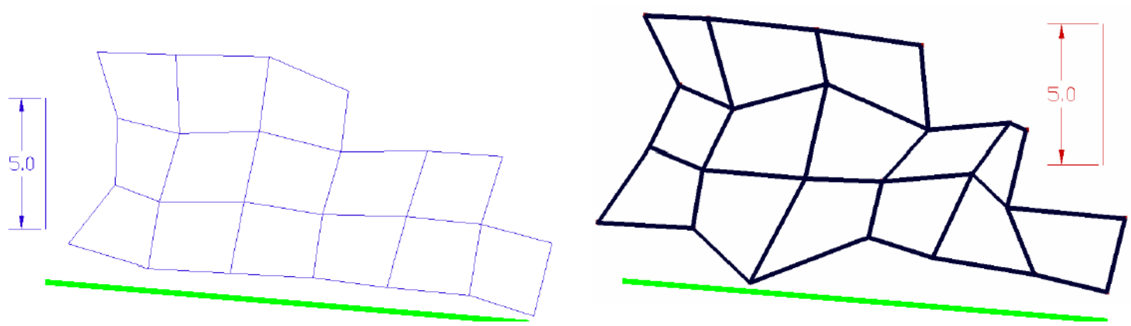


Figure 20 Poor drilling result in bottom of blast [41].
Left/Right: Top/bottom coordinates. Green line indicates the bench face

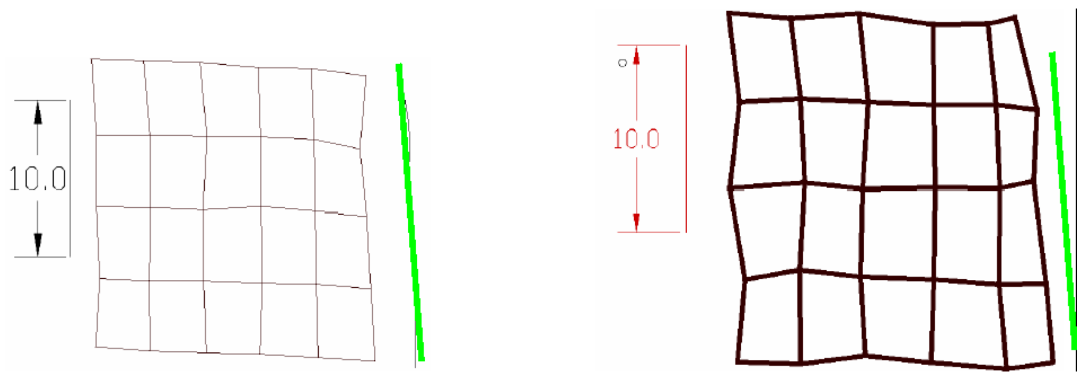


Figure 21 Good drilling result [41].
Left/Right: Top/bottom coordinates. Green line indicates the bench face

Statistics concerning fatal injuries and fly rock incidents in surface drilling are not presented here. The enclosed paper at the end of the present volume provides this information.

4.5 ECONOMICAL BENEFITS BY IMPROVING BLAST HOLE STRAIGHTNESS

As described above, measuring drillhole deviation is used in order to increase safety and make necessary precautions due to replacement or re-drilling of holes or adjusting the specific charge. In a longer perspective, the goal for a quarry must be to drill straight holes. Not only will the safety improve, but also economical benefits will rise. Sinkala [34] and Nielsen [64] have discussed this.

- Less specific drilling
    - Increased drilling pattern due to better toe burden control
    - Less drilled metre due to better distribution of the explosives
    - Subdrilling may be reduced
    - Less relative specific charging

- Improved drilling features
    - Increased drill string life
    - Less breakage stops
    - Reduced rod handling problems
    - Less drill jamming

- Better blasting result
    - Better fragmentation
    - Less fines (waste or low price product)
    - Reduced secondary breaking of blocks
    - Less popping of underbreak
    - Better loading conditions
    - Less dilution of waste rock (particularly in mines)
    - Reduced overbreak (particularly in road cuts)

If increasing the drill bit diameter to drill straighter holes, this will increase the specific charging. Increased specific charging will induce more micro cracks into the blasted rock [38]. In production of blocks and high quality aggregates for ground work, roads, asphalt, concrete etc. where rock endurance is important, the micro cracks will be a life shortening and quality reduction factor and thereby decrease the profit potential. A planned increase in boulder content will to some extent be profitable in the long run quality of the rock products [11]. In a mine to mill process the opposite effect is the main goal.

Nielsen estimates the total cost savings due to larger diameter drillholes and straighter holes to be about 7 NOK/ton (20 %) in an aggregate quarry. The benefits come from better yield of high price products, lower drilling and charging costs, and less secondary breaking [63].

Sandvik estimates the drilling and charging cost reducing potential to be about 10 % to 15 % by introducing alignment instruments and increasing drill rod diameter from 45 mm to 51 mm, in medium deviation conditions [20].

Karlsson estimates the total blasting cost potential at a certain quarry to be 19 %, reducing the current average deviation from 7 % to 0 % [36].

4.6 DRILLING STRAIGHT HOLES

The operational, practical and economical efforts are highly relevant concerning which hole improvement options a drilling operator should choose. Below, possible actions are described with focus on solutions that an operator will find feasible and can carry out alone, or with minor support by service technicians and approval from quarry management.

An improvement management plan must in all circumstances be made, extensive or simple, to be able to record the measurements and to interpret possible effects. To a small extent the theoretical aspects are described below. Thorough analysis of deflection characteristics is derived in Sinkala's doctoral thesis [34].

Collaring and alignment deviation can be substantial. The operator motivation and the drilling rig equipment (standard and state) are the two main factors affecting this deviation.

In the following, a theoretical list of deflection reduction efforts is presented, arranged by the least resource-demanding efforts first and the most demanding last. It is analytical and not scientific.

- Changing drill bit design (buttons, length and skirt)
- Increasing drill steel diameter
- Adjusting drilling parameters
- Increasing drill bit diameter (within same drill rig)
- Purchasing larger drilling rig and equipment or changing drilling method for larger hole diameter ranges.
- Reducing bench height

Changing Drill Bit Design

Looking at the percussion bit assortments, various bit designs are favorable for various rocks. Generally, concerning deviation control, aggressive shaped cutting media has less deflection than smoother designs. Insert/chisel bits, with sharp pointed edges, are considerably better than button bits. However, the net penetration rate is very low, and should only be tried at very special rock conditions [14]. In the same way, ballistic buttons will deflect less than spherical buttons due [31].

Drop centre bits are more favorable than flat front and convex front bits and will give guidance during drilling. The drop centre effect will be reduced during the bit's life as the gauge buttons will be worn more than the front buttons, and a convex bit front will be more and more apparent. Worn bits will deflect more than new ones. Longer bits and guided wings on the bit skirt (retrac design) will improve hole straightness.

Change in drill bit design is an easy and low cost alternative which should be implemented before any other alternatives.

Increasing Drill Bit or Drill Steel Diameter

Drilling larger holes, within the current drilling equipment, can improve hole straightness, as larger equipment has higher stiffness and will generally give straighter holes. However, if the drill rod diameter is not increased along with the drill bit diameter, the opposite can happen. The relative stiffness of the rod (the ratio between the rod diameter and the hole diameter) will be lower and the bit may start wobbling, giving increased deflection. The drill rod will also have more space to bend inside the hole.

Increased borehole diameter should be considered thoroughly before effectuated. Some run–in time is expected and adjustments to the blast design must be done, e.g. increased drill pattern, increased sub drilling and changes in the charging and firing plans. It is a low cost effort, but a bit resource-demanding regarding optimizing the blasting technique.

Increased drill rod diameter will give reduced deflection. The rod will be stiffer and there will be less room for the rod to bend inside the hole. A quantification of stiffness effect is given by Tamrock:

- High stiffness 1-3 cm/m (%)
- Normal stiffness 3-5 cm/m (%)

In very difficult rock formations, guide tubes can replace the first rod. However, experience shows that tubes have a relatively low life when they are exposed to percussion. Larger drill steel gives higher drill cutting velocity, and extended abrasive wear may appear around the rod couplings where the clearance between the steel and the hole wall smallest [20]:

Atlas Copco shows the effect of retrac skirt, drill steel diameter and guide rods in long hole precision drilling in LKAB's mine in Kiruna [18]. A clear effect is shown in Figure 22.

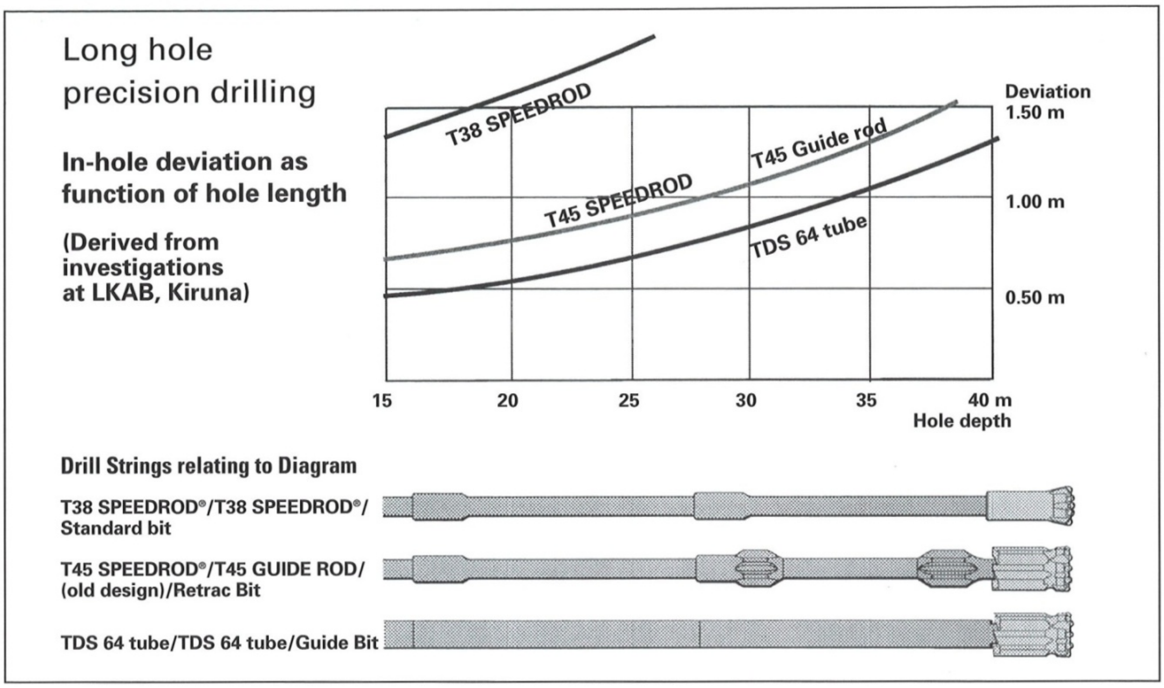


Figure 22 Decreased deflection due to drill steel design.
Results from LKAB Kiruna [18].

As for drill bit design, increasing the rod diameter should be one of the first efforts to implement in a quarry infested by deflection. During testing, changing the rod size requires changing the rod peg device in the work shop, and cannot be altered in the same way as drill bits. Anyway, it is a low-cost testing alternative.

Adjusting Drilling Parameters

To improve hole deflection, also the drilling parameters can be adjusted. The adjustments should be carried out with the guidance of professional expertise, normally available within the supplier organization. Basic principles of drilling pressure settings are shown in Section 2.4.

Reduced percussion pressure and feed force will normally improve hole straightness. Less energy is transmitted from the bit to the rock, and the deflection forces will be reduced, hence the deflection decreases. It is often said that lower net penetration rate gives less deflection; however the actual reason is reduced percussion energy. At any rate, the gain in straightness due to lower energy input and along with lower net penetration rate must be compared to loss of production capacity.

Various adjustments of the rotation speed and the feed force may also influence the deflection. A combination of increased rotation speed and reduced feed force may also give less deflection [16]. Likewise, the bit rotation speed intentionally will be set under the optimum net penetration rate pressure to reduce deflection [22]. No particular circumstances are attached to the statements, and they are probably dependent upon rock conditions and drilling equipment. Nevertheless, the effects are considerably smaller than the percussion/feed adjustment. On the other hand, they hardly affect production capacity.

Together with drill bit and drill steel adjustments, change in performance settings will be a low-cost alternative.

Reducing Bench Height

A factor which in general affects deviation substantially, is the drilled length. As mentioned above, alignment deviation will increase proportionally with hole length. On the contrary, deflection will increase by some power of two or three [34], depending on complex load conditions appearing from the rock and the drill rig at each piston impact.

A rule of thumb presented by Devico [42] gives a practical minimum curve radius of 60 m, when drilling directional core holes with 51 mm rods. Sharper bends are possible, but experience show that the material fatigue rapidly increases with less curve radii. A radius of 60 m gives 5 % deflection at 6 m hole depth, 10.1 % at 12 m and 15.4 % at 18 m.

In any case, the borehole length's influence on deviation will depend very much on the rock conditions. A study shows that the average deflection was only four percent in an easily drilled rock formation with 30 m holes 76 mm drill bit diameter [36]. At another site with poor rock conditions, the same deflections already occurred at six to seven meters hole lengths. Other examples of little and large deflection surely exist, but these are the best and worst examples found in this work.

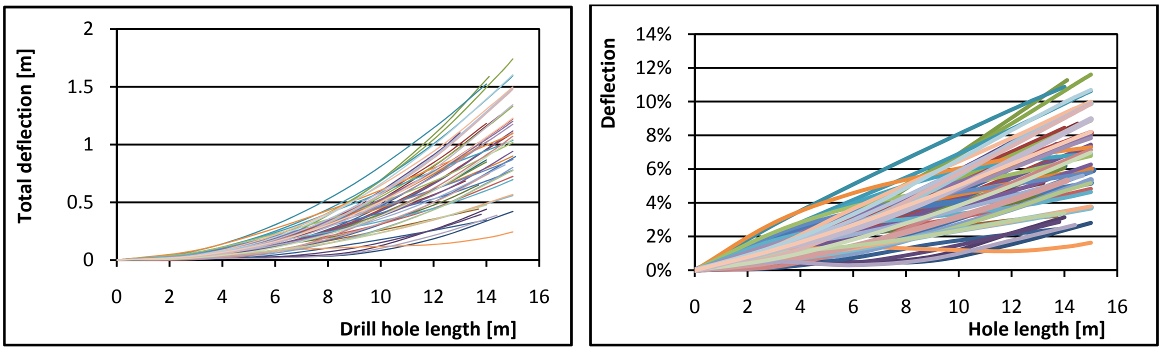


Figure 23 Typical presentation of drillhole deflection. Both the total (left) and the percentage (right) graphs show a relative deviation increase with drillhole length [42].

In most cases a change in bench height will demand large operational adjustments. Quarry design is changed and the blast design must be adjusted. Also other quarry operations will be affected, e.g. in loading where the height of the rock pile will affect which loading equipment is optimal. It will be resource-demanding both in management and practicality.

The low-cost alternatives described above, should be considered first in view of deflection problems.

Purchase of Larger Drill Rig or Changed Drilling Method

The size of the drilling rig and the drilling tools are important factors in deflection control. In general, DTH, Coprod and rotary drilling, with borehole diameters lager than 115 mm (4.5”), hardly give any deflection, in most rocks, mainly due to larger tools and higher stiffness. The troublesome diameter range is primarily less than 115 mm, and primarily for top hammer drilling. DTH and Coprod may also have significant hole deflection in the lower ranges of their hole diameter span, but it is normally less than the top hammer, due to the higher stiffness of pipes compared to rods.

The largest top hammer production rigs will have better premises for drilling straight holes than smaller contractor rigs, since they can handle in particular larger and heavier rods.

Together with changing bench height, purchase of a larger rig or changing drilling method is a relatively huge adjustment in the running of a quarry, both technically and economically, and it should only be considered after a long period of testing the other low-cost measures. It is important to notice that large equipment requires high annual production to be an economic alternative due to high investments.

5 EXTENSIVE DRILL BIT TESTING PROGRAM

5.1 INTRODUCTION

The extensive drill bit testing program presented in this volume of the thesis has primarily been carried out to see the effects different bit designs have on production rate and drilling quality. The drill bit testing program, as described in Chapter 5, and with the belonging results presented in Chapter 6, are carried out by me (see also Section 6.1). Results from the Master theses [40]-[42], and data from external projects, carried out by me, have complemented the general deviation measurement data base. In particular, collaring and alignment deviations. See Appendix J. In addition, deflection results used in analyses of drilling methods, bench floor conditions and drilling parameters are used from the Master theses mentioned above.

5.2 TIME STUDIES

In-situ net penetration rate studies were carried out to see how much the bit design affects the drilling properties. This applies for the net penetration itself and the possible influences on deflection as described in Section 4.1.

When measuring the net penetration rate a digital stopwatch was used. Normally, as used in the NTNU drilling study method (described in the thesis volume 4, Rock Quarrying, bench drilling), the net penetration rate is measured in whole rod intervals. In the drill bit testing program, to achieve more accurate results, the measuring length was decreased to 60 cm. This is approximately 1/6 of the drill rods used, which is 3.66 m. By doing this, it was easier to recognize and reject measurements from different rock conditions and failure zones down the hole, without losing too many drill meters.

Fixed time measurements were not applied as they are independent of drill steel designs.

For all sites, except site 8, the drill rigs had a digital depth meter or a digital data screen. During drilling, I was standing beside the operator cabin looking at the instrument display and recorded the time on the stopwatch for every 60 cm drilled. At site 8, markings on the drill rig boom were used to set the 60 cm intervals. To some extent, the registrations will then have larger standard deviation than in reality; however, it is not visible in the data analysis.

Alternatively, digital measuring by MWD (Measure While Drilling) could have been used to get more instant net penetration rate measurements. However, none of the rigs had such instrumentation.

5.3 WEAR DOCUMENTATION AND GRINDING

Due to wear, the bits had to be ground after approximately 50 meters of drilling in the aggregate quarries. Wear characteristics were recorded to be able to see any net penetration rate or deflection variations during the bit life.

After drilling, all drill bits were cleaned and the bit front and typical wear characteristics were photographed. The diameter of the drill bits was measured by a caliper. The wear was measured at three positions: Gauge buttons, below gauge buttons and rear end of bit. See Figure 24. Length reduction is not measured, as normal length reductions have little influence on both net penetration rate and deflection.

After measuring this data regrinding was carried out. After grinding, a new photograph was taken and the gauge diameter was measured again. The registrations were used to characterize the wear properties of the drill bits and the abrasiveness of the rock.

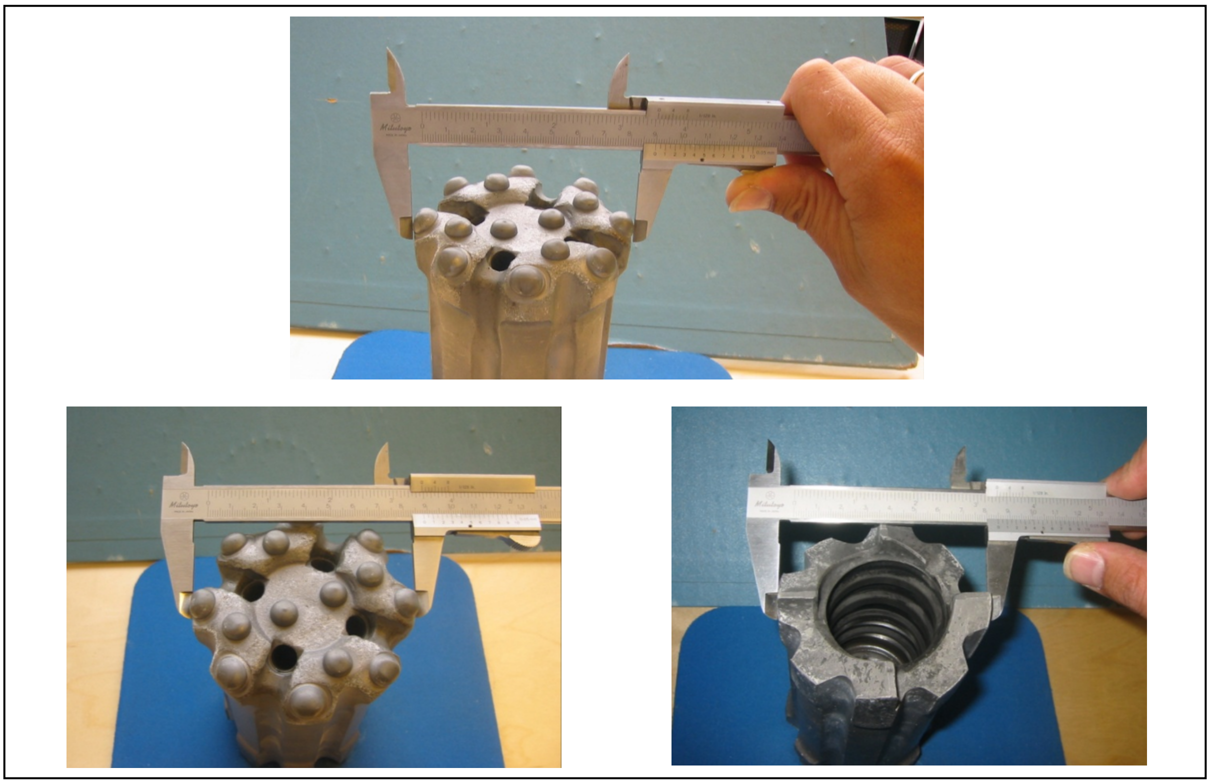


Figure 24 Measuring wear by calliper.

The grinding time of the bits varied according to the button wear characteristics. The grinding time was not measured, but all buttons were ground until no wear surface was visible. Atlas Copco recommends stopping grinding when a small wear area is still visible, due to higher wear ratio when perfectly shaped [14]. However, this was not performed, as it would have been hard to make it equal between all the different bits and buttons.

In some cases, the wear of the buttons was higher than the matrix steel. When the protruding depth became too low, an angle grinder was used to remove the body steel between the buttons.

The grinding of the drill bits was carried out in the workshop of Vassfjellet Pukk AS. The workshop had two grinding machines:

- CME junior; Construction and Mining Equipment
- Sanroc Multi, Sandvik

Fourteen different grinding cups were used to cover the numerous button shapes on the different drill bits. Both machines were used to deal with all varieties from the different manufacturers. Appendix B shows a table and pictures of the grinding machines and grinding cups.

5.4 DEVIATION MEASURING

Deviation Data Acquisition

Looking at the list of actions to reduce deflection, the economical and practical issues of the changes are essential. Drill bit design and diameter, along with rod diameter, are tested because these tests can be carried out with low costs and little planning efforts during the daily operation of a quarry. Changing drilling method, drilling rig, or bench height to improve drilling quality is time consuming, expensive and requires comprehensive planning. Sinkala, mainly looking at the effects of geological conditions and drill rig parameter settings, mentions in his suggestions for further work and studies to examine the effects of different drilling equipment and designs [34].

The test program was started in order to be able to document the results of the simplest deflection reduction actions. These results may be hard to distinguish from the current deflection situation in a quarry, and will often be based on subjective opinions and too few measurements. It was also easy to get access to quarries without disturbing their production queries by introducing this testing program.

Studies related to rock mass parameters include expectations of variations in the measuring data. Both micro and macro structure differences see to this. As the survey conditions are not optimal, it is important that the testing procedure catches possible natural variations within the rock, and randomizes the bits' possibility of hitting the same divergent rock conditions several times.

During the testing program, the various drill bits were picked randomly for every new drillhole. When testing different bit diameters, practical adaption to the drilling pattern was necessary, and in some cases, subsequent holes were drilled with the same bit.

At sites 1 through 7 the rock mass abrasiveness was medium to high, thus, regrind was carried out after 45 to 60 drill meters, or two to four holes depending on hole depth. At each quarry face, as far as it was possible, all drill bits were drilled the same number of meters. In site 8 and 9, the abrasiveness was almost zero and the number of holes registered was limited by the size of the blasts and my time reserved for field studies.

Normally, one drill test sequence lasted for 3 or 4 days of measuring, 2 days documentation and grinding, 1 day of hole deviation measurements and 2 days in the office organizing the data.

Ideally, the drill bits should have been drilled under the same rock conditions. However, in practicality this was not possible as the drill rigs moved to other faces and other quarries after drilling one blast.

Rock drill pressure settings could have been adjusted for every drill bit, but this would have required many drill meters to get the optimal situation, and was difficult to combine with the various drill bits in the test. However, the bits front design; button placement, angle, size and number, were picked within the same bit category in order to avoid this problem.

Drillhole Deviation Measuring

To record drilling deviation, survey instruments and software from Devico were used. The system components are described in section 4.3, and the measuring procedure follows below.

To measure collaring deviation, the drillhole top coordinates had to be measured. Usually, automatic reflecting theodolites or GPS systems with digital site recording software, compatible with the given deviation measuring system, are used. In the drill bit testing program, a total station and a glass prism stick were used and the registrations were manually read from the total station display. The drillhole coordinates were measured in a local coordinate system, and the recorded data was transferred to DeviBlast™.

Figure 25 PeeWee™ tool with shock absorber, DOS based terminal and cable [55].

As shown in section 4.3, each measuring system has unique instruments and software. The measuring procedures are special; however, they are much alike and using more or less the same principles. In this thesis, Devico's drillhole deviation measuring systems DeviTool™ and DeviBlast™, were used. Devico's measuring procedures are shortly described in the following. In the thesis work, the old survey kit containing a DOS-based data terminal was used. Recently it has been replaced by a PDA computer; nevertheless, the basic procedures are the same.

1. Starting up:
   The PeeWee^{TM} instrument and the hand held computer, DOS-based or PDA, are connected, respectively by cable or wirelessly. Battery check is done and wanted measuring parameters are set (time recording and depth intervals).

2. Blasting direction control:
   While connected, the PeeWee^{TM} is used to record the blasting direction. The instrument is oriented in line with the blasting face, or the drilled rows, and the direction is recorded while online with the computer. A handheld compass may also be used.

3. Initializing the PeeWee^{TM}:
   The PeeWee^{TM} and the computer are not connected during logging. Therefore, when initializing the logging, the instrument and the computer are time synchronized before disconnected. The data unit inside the instrument records the direction and inclination values every 5 seconds (the interval can be changed), giving 2 hours and 40 min recording time.

4. Measuring:
   The operator couples a rubber hose to the rear end of the PeeWee^{TM} and the shock absorber is mounted in front. The instrument is placed into the drillhole, rear end at the same level as the collar of the hole, lying on the backside of the hole, and giving the start direction of the hole. The instrument is held still, and the record button on the computer is pressed. A confirmation is given on the computer. The instrument is lowered to the next logging depth. Markings on the rubber hose are set every 2 m (can be changed).

   The same recording procedures are followed until the PeeWee^{TM} reaches the bottom of the hole. The correct hole depth is then visually measured. The final recording for the hole is made after pulling the instrument 10-20 cm above the hole bottom to avoid interferences on the instrument. The computer is then made ready for the next hole. The instrument is pulled back and put into the next hole. The procedure is continued until all wanted holes are measured. Exceeding 20-30 holes, the measuring sequence should be divided in two to reduce the consequences if data is missed.

Figure 26 In situ instrument handling.

5. Uploading data:
   When the measuring sequence is finished, the PeeWee™ is reconnected to the computer for instrument data uploading. The data program selects the instrument data correspondent to the registered data in the computer. A project file is named and saved.

6. Data analysis:
   The hand held computer data file is transferred to a PC and imported to the DeviSoft™ software. Top coordinates are given for the corresponding drillhole logging data and the recorded blasting direction is saved. Screen views of deviation can be seen and exportations to Excel for further analysis may be done. Before leaving the bench the basic data for every hole should be controlled on the hand held computer (magnetic vector and gravity vector). Strange data can be detected and control logging can be made.

Measuring Accuracy

The accuracy and validity of the measurements rely on two main factors: Operational errors and surrounding factors.

The operational errors regard inaccuracy in the first recording, giving the start direction of the actual drilled hole, which is used to estimate the alignment deviation. If not attentive, the operator may hold the hose a little to the side, giving an error in the direction measurement (± 20° is not uncommon). Special care should be taken in low temperatures, as the rubber hose loses its ductility. The hose may lift the instrument away from the back of the hole giving wrong inclination values, particularly for the two first recordings. A 2° error may easily be recorded. Further down, the gravity will turn the instrument in correct position.

Errors of this kind can be seen in the direct deviation plot. See the bend on the solid blue line in Figure 27. However, in an average cumulated deflection curve (cm/m), it will be more prominent (red broken line). The declining part is due to lower slope of the measured deviation values. The error will give a fixed additional deviation for the below measurements. The example shows that the correct hole direction is achieved at 4 m (the third measurement) at the bend on the solid blue line, giving an extra deviation of about 4 cm, shown as the intersection point of the extension of the broken blue line. The absolute alignment measuring error is small, though important to be aware of. In worst cases, the error will be about the size of the collaring error (1 to 1.5 times the drill bit diameter).

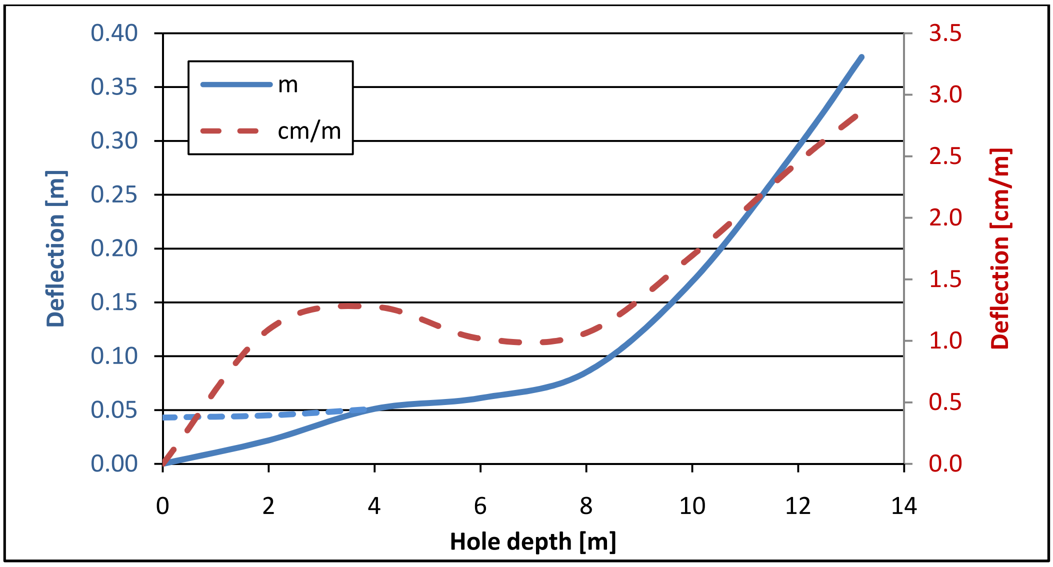


Figure 27 Alignment measuring error seen as the characteristic red dotted curve, secondary axis. Solid blue line shows the measured hole course. Blue dotted line showing actual course, and 4 cm initial deviation error on the primary axis.

Instrument movement while recording will give wrong inclination values. The operator may slip the hose or move. If this happens an extra measurement must be done and the false recording must be deleted afterwards. By controlling the data on the HHT/PDA after download, errors will be discovered as false gravity values.

Surrounding errors will arise from changes in the magnetic field. Iron bearing rock or fault zones may disturb the global magnetic field and give occasional azimuth values and wrong deviation results. By controlling the basic magnetic field values on the HHT/PDA, this error will be discovered. In e.g. iron ores, non magnetic surveying instruments must be used. Heavy machinery may also disturb the magnetic field if operating closer than around 10 meters to the measuring point.

5.5 DRILL BITS

Three drill bit suppliers donated a total of 14 drill bits for the testing. They were:

- Atlas Copco, Secoroc
- Sandvik
- DiaTeam, Boart Longyear

In collaboration with the drill bit manufacturers, relevant and often used drill bits were chosen to fulfill the testing program. A relatively narrow range of drill bit designs was chosen according to diameter, button shape and bit face design. Button and flushing holes placement, gauge button angles and the skirt design vary to some extent as well, but they are not a part of the selection criteria. The technical information of the drill bits used in the testing is shown in Table 1. Detailed specifications may be found in the manufacturers' own product data bases available on the Internet, [56] [62] [63]. Pictures of the bits are presented in Appendix C.

The only link to manufacturer will be presented here. The test focuses on the different geometrical design of the bits, and does not consider technical information such as steel hardening procedures and mineral compounds, which the manufacturers hold. These analyses will be left to the manufacturers themselves to study, and no information beyond the pages of this thesis reports will be published, according to the limitations mentioned in Section 1.2.

Drill bit no	Manufacturer	Diameter [mm]	Bit length [mm]	Face design	Skirt design	Button shape	Gauge [no:mm]	Front [no:mm]	Centre [no:mm]	Thread
No. 1	Sandvik	76	160	Drop centre	Retrac	Semi ballistic	8:11	4:10	2:10	T45
No. 2	Sandvik	89	165	Drop centre	Retrac	Semi ballistic	8:12	4:11	2:11	T51
No. 3	Sandvik	89	165	Drop centre	Retrac	Spherical	8:12	4:11	2:11	T51
No. 4	Sandvik	89	160	Flat front	Retrac	Semi ballistic	9:11	3:10	3:10	T51
No. 5	Secoroc	76	160	Drop centre	Retrac	Semi ballistic	8:11	4:11	1:11	T45
No. 6	Secoroc	89	177	Drop centre	Retrac	Semi ballistic	8:13	4:11	1:11	T51
No.7	Secoroc	89	177	Drop centre	Retrac	Spherical	8:11	4:11	2:9	T51
No. 8	Secoroc	102	181	Drop centre	Smooth	Ballistic	8:15	4:13	1:13	T51
No. 9	Boart Longyear	89	174	Drop centre	Retrac	Ballistic	8:11	4:11	2:11	T51
No. 10	Boart Longyear	89	171	Drop centre	Retrac	Spherical	8:11	4:11	2:11	T45
No. 11	Boart Longyear	76	173	Drop centre	Retrac	Spherical	8:11	4:11	1:11	T45
No. 12	Boart Longyear	102	207	Drop centre	Retrac	Spherical	6:16	3:14	2:12	T51
No. 13	Secoroc	102	180	Flat front	Retrac	Ballistic*	8:16	4:13	2:13	T51
No. 14	Boart Longyear	89	180	Drop centre	Retrac	Ballistic	6:14	3:12	1:12	T51

Table 1 Technical data of drill bits used in test program. *Originally semi ballistic. Reshaped by grinding.

5.6 ROCK MASS CHARACTERISTICS

The drill bit testing program was carried out in Mid-Norway in 5 different quarries and in 8 different rock conditions. The quarries investigated were:

- Vassfjellet aggregate quarry in Trondheim –Franzefoss Pukk AS
- Lia aggregate quarry in Trondheim - Franzefoss Pukk AS
- Fossberga aggregate quarry in Stjørdal - Franzefoss Pukk AS
- Tromsdalen limestone quarry in Verdal - Verdalskalk AS
- Visnes limestone quarry in Eide - Visnes Kalk AS

Documentation of the geological conditions on the bench was retrieved from laboratory testing and visual inspection of the face.

Quarry Face Characterization

A precise survey of the rock mass conditions at the face was not possible to achieve, due to practical obstacles and safety aspects.

Often the muck pile of the previous blast covered the face until hours before blasting, and the loading operation could not be stopped. Surveying simultaneously with the loading is not allowed due to traffic safety and the possibility of flying rock particles from secondary breaking or misfire detonations. If the muck pile was cleared, it was still unsecure to work close to the face due to falling rocks. No resources were available to secure the bench face.

The quality of a geological survey is also questionable, as the blast in front usually induces more joints than the solid rock mass originally contains. Due to this, only the major lines in the rock mass were registered.

QUARRY FACE	ROCK DESCRIPTION
Site 1	Homogenous rock mass. Two joint sets plus occasional joints.
Site 2	Bedded rock mass with frequent joints. Two major fault planes.
Site 3 and 4	Layered and frequently jointed rock mass.
Site 5	Homogenous rock mass. Several joint sets plus occasional joints.
Site 6	Homogenous rock mass. No systematic joint sets.
Site 7	Layered and frequently jointed rock mass.
Site 8	Homogenous bedded rock mass. Occasional joints and fault planes.
Site 9	Homogenous rock mass. Occasional joints.

Table 2 Classification of quarry faces in the extensive drill bit testing program

<u>Laboratory Test Results</u>

One rock sample was taken from each site. A representative 30 kg block was necessary to carry out the described tests (Table 3 and Appendix D). Not all of the laboratory tests are used directly in this thesis. However, with a limited use of extra resources the extra laboratory tests were carried out for any possible future research.

Testing several block samples would have been advantageous to get a more accurate picture of the rock mass. However, the benefits compared to the costs and resources needed were assessed to be small, particularly as the main purpose of the test was to compare drill bits within the same rock mass.

The engineering geology and rock mechanics laboratory at NTNU/Sintef did the testing of the rocks, according to authorized standards [12] and [49]-[54]. Detailed laboratory test reports are not included in the thesis.

QUARRY FACE	ROCK TYPE	DENSITY (g/cm^3)	VHNR	SONIC VELOCITY (m/s)	UCS (MPa)	PLTS (MPa)	DRI	CLI	BWI
Site 1	Greenstone	3.05	800	3877	84	6	56	34	21
Site 2	Granodiorite	2.55	815	5678	163	12	38	10	46
Site 3 and 4	Greenstone	2.87	723	4783	61	10	35	53	22
Site 5	Greenstone	3.05	742	5382	159	10	30	13	37
Site 6	Greenstone	3.08	725	4723	149	11	48	18	25
Site 7	Greenstone	3.03	703	4801	143	13	39	40	34
Site 8	Limestone	2.72	125	6378	81	5	62	98	11
Site 9	Marble	2.71	125	5402	67	4	66	85	9

Table 3 Summary of laboratory tests in the extensive drill bit testing program

5.7 DRILLING EQUIPMENT AND DRILL RIG PARAMETERS

The machinery available in the investigated quarries was used in the testing program. Table 4 shows the drill rig equipment:

SITE NO.	DRILL RIG	DRILL HAMMER	DRILL STEEL
1 : 5 : 6	Atlas Copco Roc D7	Cop 1838	T51/T45 1)
3 : 4 : 7	Atlas Copco Roc D7	Cop 1838	T51
2	Atlas Copco Roc D7	Cop 1838	T51/T45 1)
8	Sandvik Tamrock Pantera 800	HL 700	T45 2)
9	Atlas Copco Roc F7	Cop 2050	T51 3)

Table 4 Technical data on drill rigs used in the extensive drill bit testing program. 3.66 m rods with integrated couplings. **1) Peg device changed to fit T45 rods for the T45 thread bits. 2) T51 adapter used for T51 thread bits. 3) T45 adapter used for T45 thread bits.**

The drill rigs current pressure settings were adapted in the testing program for all drill bits. The different drill bits may have different optimal pressure settings in the actual rock conditions. To be able to optimize the settings for each drill bit at each site, it would have been very resource demanding, and it would have reduced the volume of the testing. A large volume of drillholes tested was assessed to be very important to be able to explore any difference between the relatively similar test bits and the varying nature of the rock mass. Additionally, by experience it would have been difficult to get access to the different quarries if the testing program had too much negative effect on the production.

QUARRY FACE	PERCUSSION PRESSURE (bar)	FEED PRESSURE (bar)	ROTATION PRESSURE (bar)	ROTATION SPEED (rpm)	FLUSHING PRESSURE (bar)
Site 1	210	50-60	45	89	6
Site 2	200	70-80	50	94	6
Site 3	210	80	40	90	6
Site 4	200	70-80	45-50	90	5.8
Site 5	210	80	40	92	5
Site 6	210	70-80	45	91	5
Site 7	200	70-80	40	90	4
Site 8	160	50	60-70	120	7
Site 9	200	70	40-50	108	7-8

Table 5 Drill rig settings used in the extensive drill bit testing program. Minor adjustments may have been done during drilling.

6 TESTING PROGRAM RESULTS AND ANALYSES

6.1 INTRODUCTION

Before I discuss the results of this thesis, I must remind the reader of the importance of the natural spreading in the measuring results. When working with rock mass the test conditions will vary all the time. Experience from using the rock mass classification models; Q system and RMi system, often shows great divergence in the predicted values over short distances. This has been a challenge when planning and analyzing the results in the test program.

Apparently uniform rock masses have show “strange” results. Instead of scrapping these measurements and make trends obvious, I have included most measurements in the assessments, giving less clear statements and more insignificant results. The fact that the rock mass is inhomogeneous, giving more or less independent results, is also a conclusion worth reporting.

I have chosen to analyse the results in a practical way, more than by advanced statistical analysis, due to the natural variation of the rock. To be able to establish a confident theoretical model on net penetration or deflection, the required studies would have been more detailed and more resource demanding than what were possible within the scope of this thesis. The studies are carried out to find indications of correlations interesting to look closer into at any particular quarry later.

My contribution of field data to the extensive drill bit testing consists of 277 drilled holes. This includes approximately 7000 manually timed net penetration rate measurements, approximately 2500 deviation records (7500 coordinate values) and almost 1000 manually measured wear values. Well over 4000 drilled metres are analyzed and a total of 400 hours are spent in the field, doing time and deviation measurements on the bench. In addition to the extensive drill bit testing program about 750 holes have been measured for deviation in master theses and external projects, whereas 500 by the master students and 250 by me.

To help me organize and analyze the data, I have been using Excel spreadsheets and DeviBlast™ hole deviation software. DeviBlast™ provided the data from the drillhole deviation measurements, and by exporting the data to Excel, further analyses were made.

I have based the main analysis on the measurements of the six 89 mm diameter category bits, which were tested at all sites. These are bit number: 2, 3, 4, 6, 7 and 9. Individually they drilled between 440 and 490 m each and were exposed to the same rock through the testing. Analyses were also carried out on the 76 mm and 102 mm category bits, and two 89 mm bits added later in the testing program. They drilled between 80 and 160 m each and give important supplementary information to the assessments.

There are numerous graphs and tables attached to the analyses of the measurements, and even though most of them are left out of this thesis, I feel there are too many remaining to put them in the main text. Nevertheless, it is necessary to make the results available for the reader, and I have decided to show most of the graphs and tables in the appendix.

6.2 OUTLIERS

As mentioned above, rejection of data, the so called outliers, has been an issue during the analysis of the multiple net penetration rate data sets. A major challenge in the analysis was to find representative data for each drill bit that was comparable to the other bits at the given site. Finding outliers have been a part of this process. In the literature [65] I found a method to estimate an outlier level.

$$Q_{down} = q_{25} - 1.5*(q_{75} - q_{25})$$
$$Q_{up} = q_{75} + 1.5*(q_{75} - q_{25})$$

Q_{down} = Lower outlier level.
Q_{up} = Upper outlier level.
q_{25} = lower quartile of the dataset
q_{75} = upper quartile of the data set.

6.3 NET PENETRATION RATE

Data Preparation

Testing net penetration rate in rock will naturally give a wide range in the measured values. Some rock masses more than others, particularly due to jointing and fault zones. To be able to compare the drill bits' net penetration rates in the same rock conditions, it was found necessary to determine a level in the data sets representing the intact rock conditions. This means the net penetration rate data not affected by joints and fault zones. This level was assumed to be equal to the 10 % percentile level (P_{10}) of the data sets.

To be able to establish this level, data, which seemed to be representing other rock conditions than the dominant conditions, were rejected. The data preparation can be divided into three main steps:

1. Estimation of the theoretical outlier level
2. Rejecting data
3. Estimating the 10 % percentile level

In the process of rejecting data, both the theoretical estimations and visual inspection are used to find the outliers. The visual interpretation means that every hole is analyzed according to how they appear in relation with all the other drill bits at the site. The following presents an example on how I established the P_{10} level.

Figure 31 shows a typical net penetration rate data set for one drill bit at a given site (Site 3 and Site 4 are drilled in the same rock). At first sight it seems that the dominant rock conditions give a net penetration rate slightly less than 100 cm/min for drill bit 6. This impression is strengthened by looking at the other drill bits. By experience, the net penetration rate will not vary more than about 20 % between the assortments of drill bits chosen. See Section 2.4,

Looking at the Hole 8 and Hole 18 at Site 3 in Figure 31, it seems that the drill bit has passed a fault zone, highly jointed rock volumes or an intrusive rock formation. By looking at neighboring holes, these zones often appear systematically through the tested volume. The observations are also checked with pictures of the bench face.

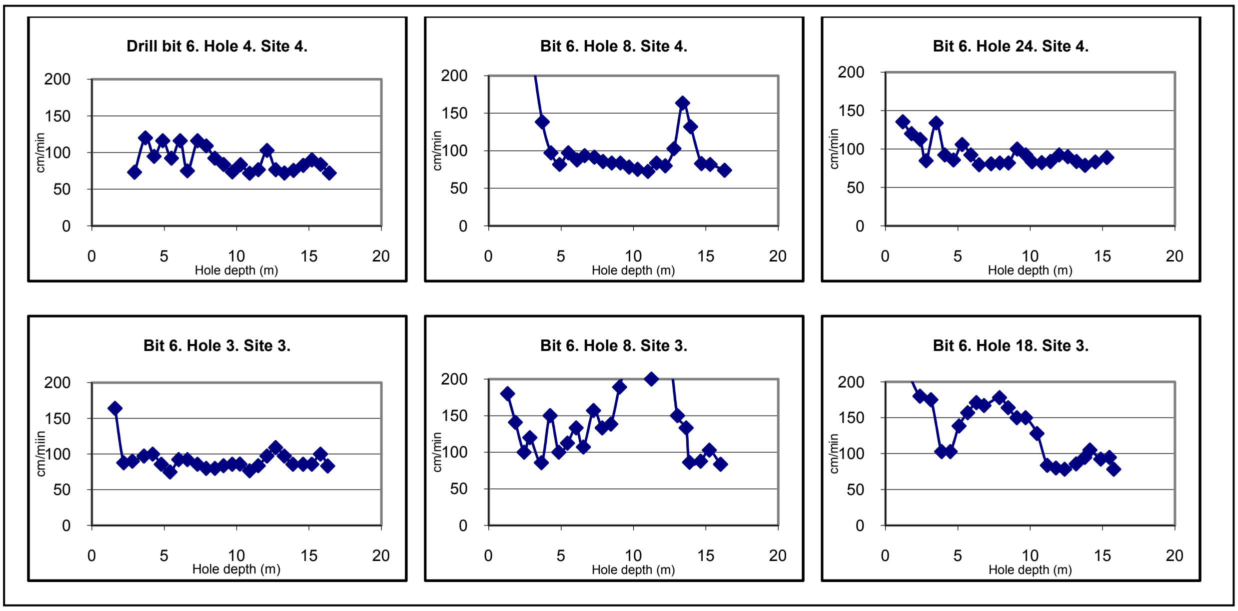


Figure 28 Examples of outliers rejected in the circles.

The data from each drill bit at the same site are put together and analyzed. The upper and lower theoretical outlier levels (Q_{up} and Q_{down}) are estimated. See Figure 29.

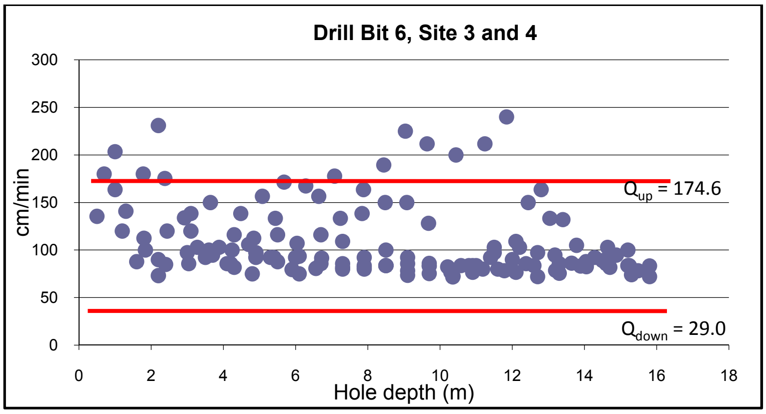


Figure 29 Theoretical outlier levels for Drill Bit 6 at Site 3 and 4.

The outlier method put all data into one dataset, giving a constant outlier level down the hole. This means that a valid low value at the bottom of a hole, by calculation may be characterized as an outlier, even though it should not be. The same may apply for valid high values at the top of a hole.

To handle the slightly decreasing net penetration rate down the hole, a linear regression line of the data sets are made, and the outlier levels are rotated according to this, around the fixed average hole depth value. See Figure 30.

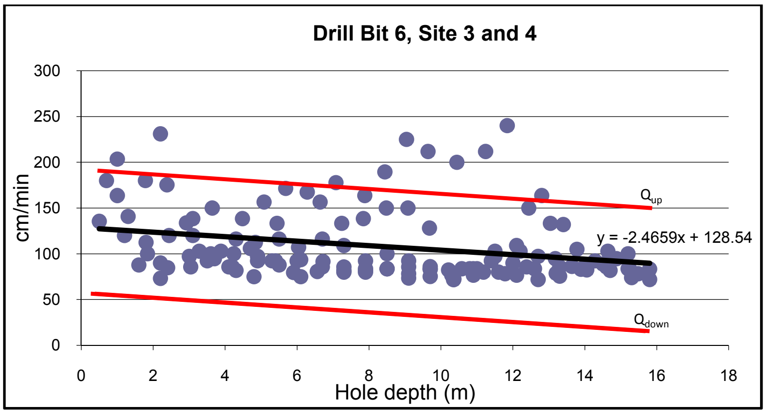


Figure 30 Outlier levels turned to follow the linear regression of the data set.

Based on Figure 30 the theoretically estimated outliers are rejected. Looking back at each hole, the rejections were controlled. In this particular example some manual rejection of values has been done as well. See Figure 31. This applies to Hole 3, Hole 8 and Hole 18 at Site 3. The value in Hole 3 seems to be measured in the prior subdrilled zone from the overlying blast. Based on comparison to neighboring holes it seemed that Hole 8 and 18 were drilled through a fault zone or frequently jointed rock mass, thereby not representing the majority of the results.

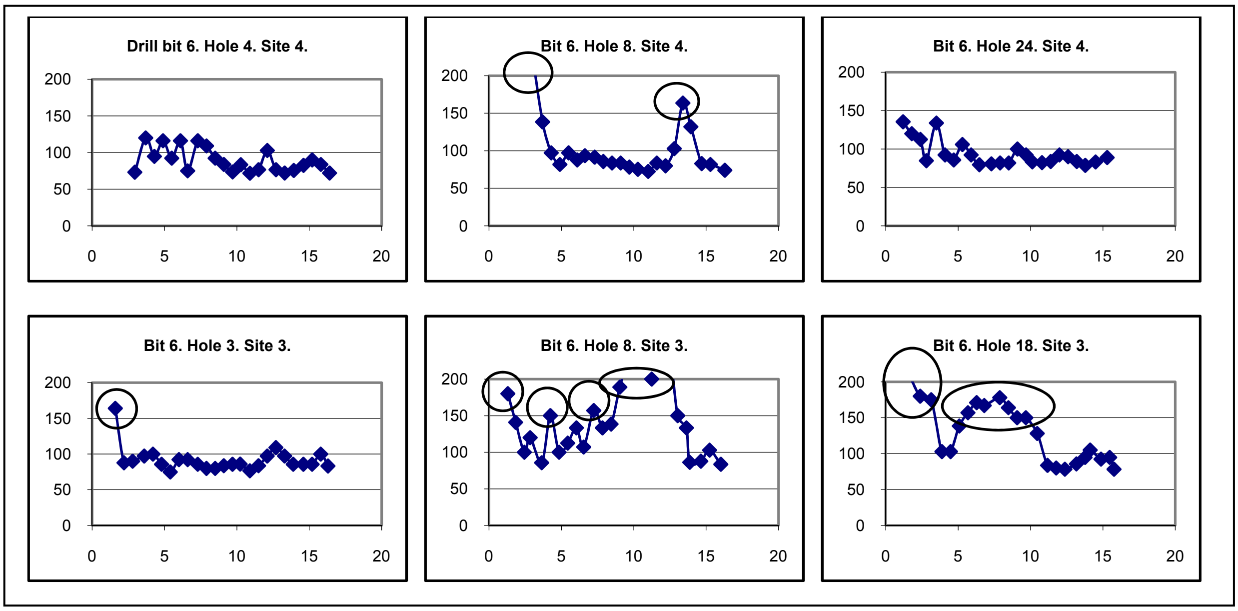


Figure 31 Examples of outliers rejected in the circles.

Another issue to consider in the visual analysis, is the fact that the time was recorded in whole seconds. Generally speaking, this means that: As the penetration rate increases, the one second interval registration represents larger and larger net penetration rate ranges. An example is used to show this:

At a net penetration rate of 100 cm/min, the recorded values would have been 36 seconds over 60 cm. Recording 37 seconds on the same distance would have given 97 cm/min, and 35 seconds giving 103 cm/min. We see that each second interval represents a range of 3 cm/min, giving an accuracy of ± 3 % at 100 cm/min. At 200 cm/min, the same scenario gives a range of 12 cm/min, giving ± 6 % accuracy.

Histograms were used to visualize the distribution of the measurements, with regard to the whole second intervals. The histograms often revealed a distribution that helped the rejection of data. Looking at the present example and Figure 32, the distribution tail above 150 cm/min supports my rejection of the extra values in Figure 31.

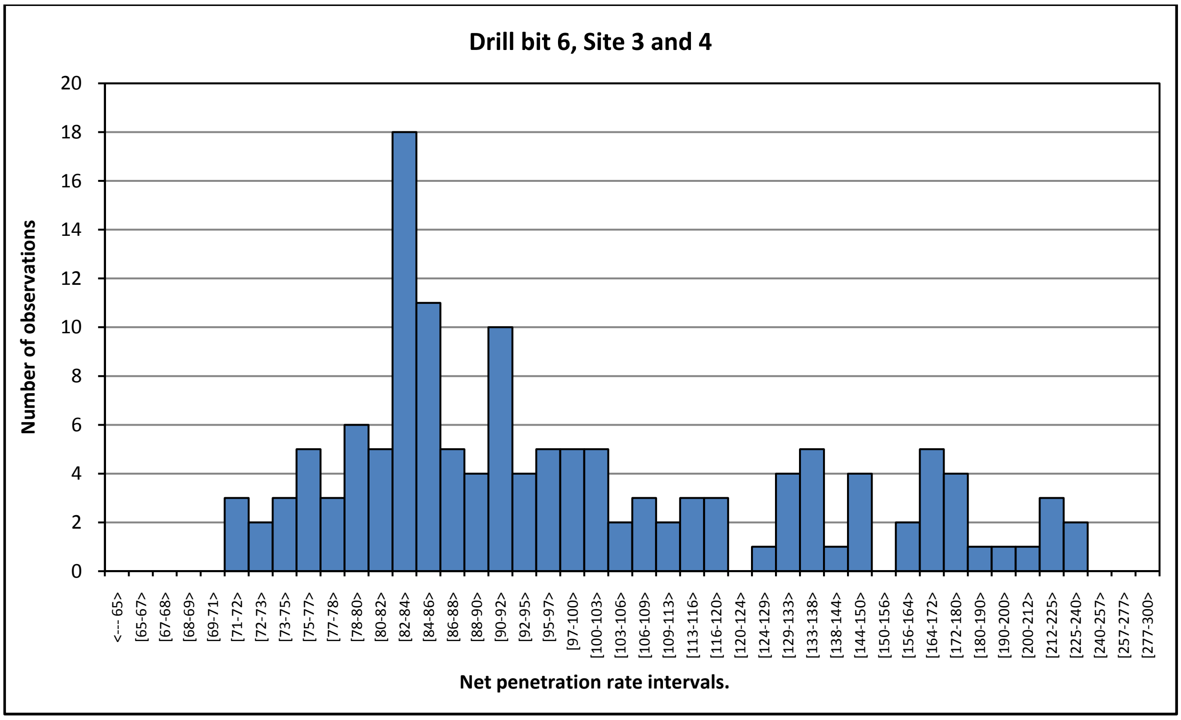


Figure 32 Histogram of measured data of Drill bit 6 at Site 3 and 4.

After all rejections are made, the remaining data set is the one I want to compare with the other drill bits at the same site. See Figure 33. However, comparing the linear regression lines, for all the values, appeared to give too much variation to find any trends across all the different sites. It seemed like distortion of the joint frequency in the rock masses was the reason. To make a comparable level with reduced joint affection, I assumed that the 10 % percentile level of the treated data set would represent intact rock conditions, and that the joint distortion would be more or less removed.

To obtain the decrease in net penetration rate down the hole, I calculated the P_{10} for all values within the same rod number for each drill bit. I then carried out a linear regression according to these new values. See Figure 33. All the estimated regression equations from each drill bit at each site are plotted together in a diagram and compared. See Appendix F.

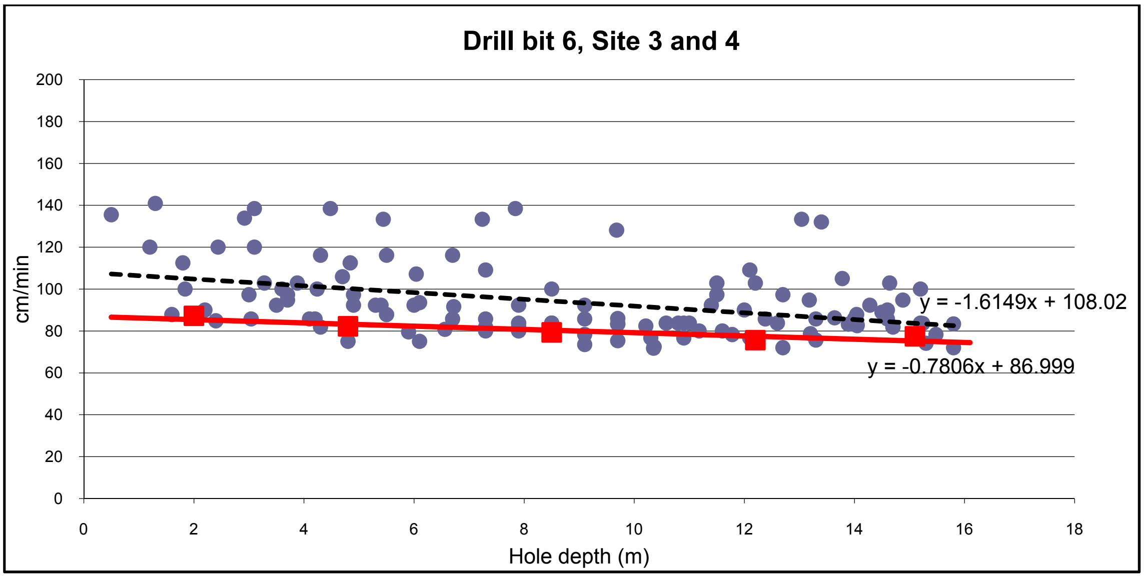


Figure 33 Net penetration data for Drill bit 6 at Site 3 and 4. Red squares indicating the P_{10} value for each rod for all holes. Red dotted line is the linear regression of the P_{10} values.

In some cases, the P_{10} line's slope (Newton quotient) varied substantially to the linear regression, for all the values, due to few measurements over a certain rod. The tendency is typical for the first rod. In these cases, adjustments were made by excluding the values from these rods or making a new point with the average of the values for the next rod.

<u>General Bit Comparison</u>

The analysis of the net penetration rate measurements is complex. Some design parameters seem to give clear, but other features seem to be more complicated than hoped for in the planning process. The net penetration rate analysis was made on the following drill bit design features:

- Drill bit diameter
- Button shape
- Front design
- Wear

When analyzing the net penetration rate results, first a comparison of the 89 mm category bits was made, embracing all the various designs. Based on the net penetration rate value estimated by the regression formula at middle of the second rod (4.8 m), Table 6 shows the ranking of the 89 mm bits from every site and their total sum of rank numbers. Figures are shown in Appendix E and Appendix F. As the first rod only drills about 3.0 m (due to the rod handling system), the middle of second rod is approximately at 4.8 m hole depth.

There seem to be clear differences in the ranking of the bits, and it seems that the recommended drop centre, ballistic buttons and retrac bits (2, 6 and 9) have generally better net penetration rate properties than the three bits with flat front or spherical buttons (3, 4 and 7). The size of the differences is not visible in the table, and this will be discussed more thoroughly in the following, parallel to more detailed analysis of the various bit designs. Also, the data from the 76 mm and 102 mm bits are included when appropriate.

TEST LOCATION	Drill bit 2 Semi-ball. Drop centre	Drill bit 3 Sperical Drop centre	Drill bit 4 Semi-ball. Flat front	Drill bit 6 Semi-ball. Drop Centre	Drill bit 7 Sperical Drop Centre	Drill bit 9 Ballistic Drop Centre
Site 1	4	6	2	1	3	5
Site 2	1	5	6	3	4	2
Site 3 and 4	1	3	6	4	5	2
Site 5	1	6	5	3	6	2
Site 6	1	2	6	4	5	3
Site 7	2	5	6	3	6	1
Site 8	2	6	4	3	5	1
Site 9	1	6	4	2	5	3
Sum of ranks	13	39	39	23	39	19
Rank average	1.6	4.9	4.9	2.9	4.9	2.4
Standard deviation	1.1	1.6	1.5	1.0	1.0	1.3

Table 6 Net penetration rate ranking from all sites. 89 mm bits.

<u>Drill Bit Diameter</u>

Theory and general experience show that the drill bit diameter is a very important parameter regarding net penetration rate. The figures from the test program support this knowledge.

Looking at the figures in Appendix G (Figure G.1) the linear regressions give a slope value between -1.0 and -2.3. The R^2 value for the regressions varies from about 0.6 to 0.9. The presented slope values mean that the net penetration rate reduction varies from 13 cm/min to 28 cm/min per half inch increase (13 mm) in drill bit diameter. The results do not show any difference for soft or hard rock. Comparing the slope value with the correspondent net penetration rate graphs in the project report series at NTNU, see Figure 34, and figures from the manufactures, such as Figure 5, the correlation is high and should strengthen the validity of the measurements. In Figure 5 and Figure 34 the net penetration rate changes with 15 to 20 cm/min and 20 to 25 cm/min per 13 mm change, respectively.

Looking only at bits in the 89 mm category (Figure G.2), 7 out of 8 sites show a decrease in net penetration rate with increased diameter. Both the slope values and the R^2 values vary more than the values in Figure G.1. The R^2 values spread from 0.003 to 0.78. The reason for this is mainly the very small diameter range, and the slope will be highly affected by small variations in the net penetration rate.

To be able to compare drill bit diameter effect on the net penetration rate between the different sites, a relative reference level was estimated. See Figure G.3. In each diagram, the net penetration rate for each bit in every site is set to 100 %. The other bits will then have a relative net penetration rate according to this, within each site. This means that if a certain drill bit in general has high net penetration rate compared to the other bits (e.g. Drill bit 2, Table 6), the values in the diagram for this bit will in general be below 100 %. Vice versa for drill bits with poor net penetration rate (e.g. Drill bit 7).

The average relative slope value for all 89 mm category bits is approximately -0.01. This indicates that the net penetration rate (cm/min) increases by 1 % for every 1 mm the bit diameter decreases. For low net penetration rate values, the absolute difference is smaller than for high values. The same is also visible in the experience data presented in e.g. Figure 5 and Figure 34. In Figure 34, at DRI™ 20, i.e. low net penetration rate, the upper and lower curve varies by approximately 75 cm/min. At DRI™ 70 the absolute difference is about 100 cm/min

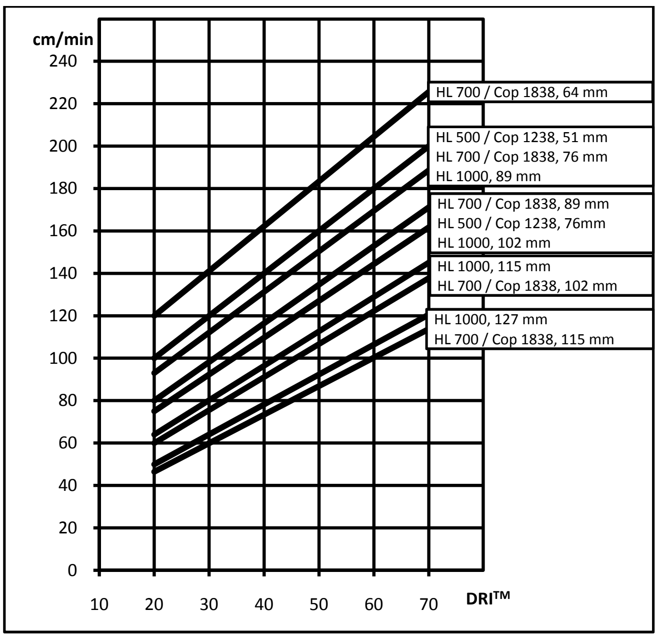


Figure 34 Net penetration rate figures from report 12C-08 Rock Quarrying Bench Blasting. Absolute difference between 20-25 cm/min per 13 mm (half inch).

Under ideal conditions, the net penetration rate should vary according to the change in cross section area of the drill bit, assuming the same power out-put of the rock drill. See Figure 35.

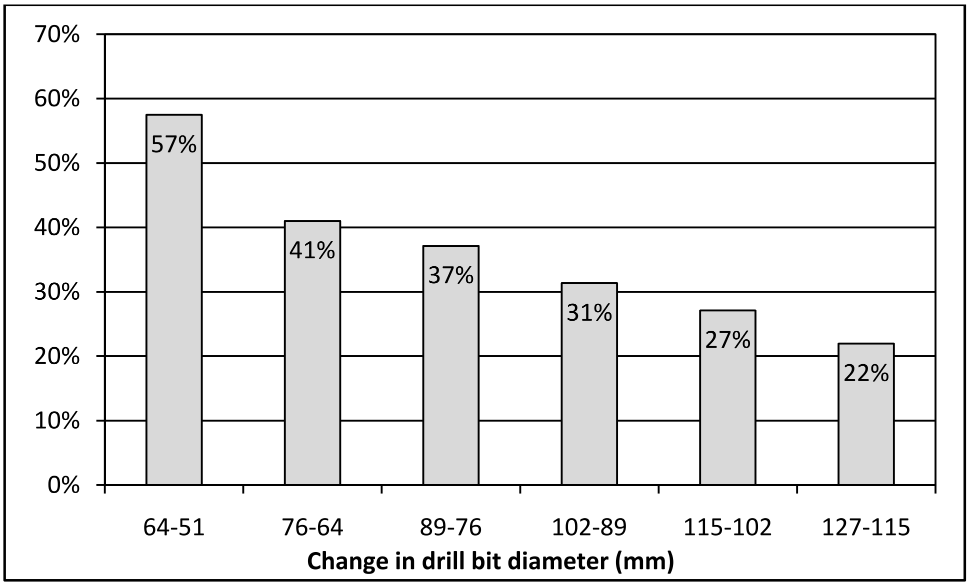


Figure 35 Simple theoretical analysis of the percentage change in net penetration rate according to half-inch change in drill bit diameter.

Going from 102 mm to 89 mm drill bit diameter, a theoretical increase in net penetration rate of 31 % should have been measured. The studies imply an increase of about 20 % (using the average, P_{50} values of the data). Experience data shown in Figure 5 and Figure 34 shows an increase of approximately 15 %.

The recorded values are probably lower due to non ideal conditions. More energy is lost as, for instance, drill string vibrations and heat. The experience data gives a slightly small difference between each bit diameter step than the test program results. A possible reason may be non optimal drilling settings in the testing program for the 102 mm bits. As described in Section 5.7, the drill rigs were optimized according to 89 mm diameter bits.

Button Design

Generally, as described in Section 2.4, ballistic buttons will have a higher net penetration rate than spherical buttons. The results from the extensive drill bit study show the same. See Appendix H.

In Table 7, the relative average net penetration rate for the three button designs (spherical, semi-ballistic and ballistic) across all sites is presented. The relative average means that each net penetration rate value within a site is related to the actual reference bit. In this way the trends across varying rock conditions may be found.

Comparing the relative ratio for the 89 mm category bits in Table 7, the drop centre semi ballistic bits have in average 8.5 % higher net penetration rate than the drop centre spherical bits. There is a slight difference between the semi ballistic bits and the ballistic bit (Bit 2 and Bit 6 vs. Bit 9). The analysis shows that the ballistic bit has in average 0.5 % lower net penetration rate for all sites, even if it is supposed to have higher values. The reasons can be numerous, but there is one clear difference between the bits developed during the test. The wear characteristics show that Bit 9 has more button wear and body steel deformation than Bit 2 and Bit 6. Because of this the buttons' protruding depth became smaller and the flushing holes decreased. Both characteristics may give poorer net penetration rate properties.

REFERENCE BIT (100 %)	SPHERICAL	SEMI BALLISTIC	BALLISTIC	DIFFERENCE (sph.-semi ball)	DIFFERENCE (semi ball – ball)
Bit 2 (semi-ball)	89.96 %	98.02 %	97.49 %	8.1 %	-0.5 %
Bit 3 (spherical)	99.65 %	108.89 %	108.40 %	9.2 %	-0.5 %
Bit 6 (semi-ball)	93.97 %	102.40 %	101.92 %	8.4 %	-0.5 %
Bit 7 (sperical)	100.47 %	109.73 %	109.28 %	9.3 %	-0.4 %
Bit 9 (ballistic)	92.45 %	100.67 %	100.00 %	8.2 %	-0.7 %

Table 7 Relative net penetration rate due to button shape. 89 mm category bits with drop centre.

There is a correlation between the bit diameter wear and the button shape (Figure Q.2), and in relation to the discussion in the section above, the values in Table 7 may be reduced with 1 % point. The ballistic shaped bits are in average 1 mm smaller than the spherical bits.

The results seem to imply that the in-situ effect of ballistic and semi ballistic buttons is less than the data presented in Section 2.4, respectively a 10 % to 14 % increase in net penetration rate.

There seems to be no direct significance between button size and net penetration rate. However, as described in Section 3, smaller buttons seem to be less resistant to wear than larger buttons giving an indirect effect on the net penetration rate.

<u>Bit Front Design</u>

In the test bit assortment, there are only three flat front bits and only one of the six main 89 mm category bits. The foundation of the analyses seems to be a little small, but by looking at Table 6 and Appendix I, it seems that Bit 4, with flat front, generally has relative low net penetration rate compared to the other ballistic shaped button bits. The two supplementary bits (Bit 13 and Bit 14) do not show any trend on this, as the button shape seem to be dominate compared to the front design. It seems that other factors than the front shape cause the relatively low net penetration rate values.

Like Bit 9, Bit 4 has higher button wear and higher body steel deformations than the other bits. The buttons are smaller and has only three flushing channels (the others have four). Additionally, the positioning of the buttons is divergent with one extra gauge button. There are no attempts to analyze this further, but the divergences are too obvious not to be mentioned.

<u>Bit Condition and Wear</u>

All 89 mm category bits have more or less drilled the same amount of meters, and the difference in the wear (measured as diameter reduction from original size) seems to be too little to beat the effect of smaller diameter and higher net penetration rate. The different wear and deformation characteristics described in the two previous sections are not directly correlated to the bit diameter wear. However, at Site 8 and Site 9, completely new bits, equivalent respectively to Bit 6 and Bit 2, were tested to measure any connection between wear and net penetration rate. See Figure 36.

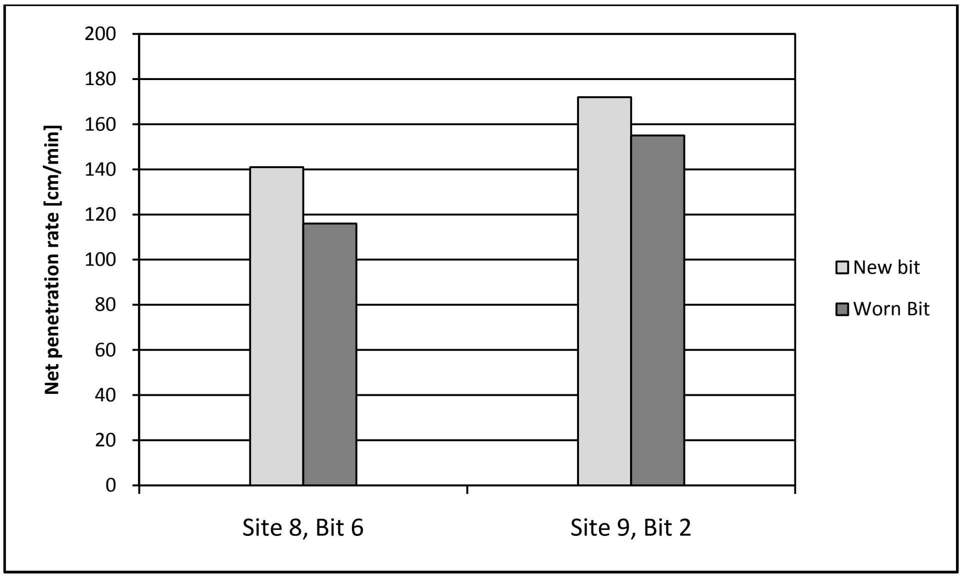


Figure 36 Comparison of net penetration rate between worn and new bits at Site 8 and Site 9.

The figure shows that wear most likely causes a significant reduction in the net penetration rate. The worn bits at the two sites have respectively 17 % and 10 % less net penetration rate than new ones. The data is limited, and only carried out in the two soft rock quarries, which limits the validity of the results. However, it seems that the bits, at the end of their service life, have 10 to 20 % less net penetration rate than new ones, supposing all buttons are still intact. As comparison, experiences from rotary drilling where no grinding is carried out during the life of the drill bit, show a difference of 30 % from the optimal bit shape to the poorest, just before wrecking [3].

As mentioned in Section 5.3, the regrinding interval was about 40 m to 50 m in the aggregate quarries (two to four holes) The measurements show no significant reduction in the net penetration rate for each bit, due to the succession of the holes and the wear of the bit.

Drilled Length

Due to energy loss in the thread couplings, the net penetration rate decreases as every new drill rod is added in the drill string. A stepwise decrease is hardly visible for any single hole, and a general decrease is not necessarily visible either. Nevertheless, the linear regression plot for all registrations for every bit at each site shows decreasing trends. Figure 33 shows a representative example of such plots. The linear regression line for all 60 cm interval measurements for all bits at each site is used to estimate an average slope value of the drilling measurements.

In the NTNU model described in Volume 4, ROCK QUARRYING Bench Drilling, the capacity studies are based on net penetration rates from whole rod intervals. A linear regression based on these values is also made in the analyses to see if there is any significant difference to the P_{50} regression values. The results from each site are presented in Table 8. Regression equations are shown in Appendix E, Figure E.2.

The data referred to in Section 2.4 implies an energy loss of 3.5 % per coupling, or almost 1 % per meter (supposing 3.66 m rods). This implies a varying net penetration rate loss, between e.g. soft and hard rock and between the drill bit diameters. There are slope variations, but no systematic rock or diameter related differences are found.

The results in Table 8 show an average net penetration rate loss per rod of 4.1 %, which corresponds well to the data in Section 2.4. The variations in the results are not studied in detail, but it seems that the results from Site 6 are strangely high compared to the other sites. There might have been some structural variations in the rock mass, too small to be characterized by the outlier rejection procedure. This might have given relatively higher net penetration rates in the top of the bench. Conversations with the drill rig operator at the site, confirms that the actual bench might be in such conditions [66].

TEST LOCATION	SLOPE VALUE 60 cm, P_{50} regression [(cm/min)/m]	SLOPE VALUE Whole rod regression [(cm/min)/m]	AVERAGE NET PENETRATION RATE, 2nd rod [cm/min]	NET PEN. RATE REDUCTION PER ROD (3.66m) 1)
Site 1	-0.86	-0.73	137.9	2.1 %
Site 2	-0.72	-0.78	99.5	2.8 %
Site 3 and 4	-1.10	-1.21	104.5	4.0 %
Site 5	-0.87	-0.83	118.7	2.6 %
Site 6	-2.55	-2.74	118.3	8.2 %
Site 7	-1.01	-0.94	85.8	4.2 %
Site 8	-0.95	-1.08	125.9	3.0 %
Site 9	-0.49	-0.40	149.9	1.1 %

Table 8 Average slope values estimated for each site and percentage decrease from current net penetration rate. 1) Average of both regressions and average net penetration rate at each site (89 mm category bits).

6.4 DEFLECTION DEVIATION

General Deflection View

The deflection analysis is primarily based on the extensive drill bit program. A summary of these deviation measuring results are shown in Appendix K. Results from parallel measurements, both in the same quarries as the test program and other external quarries, are also included to support the test program assessments as well as generally increase the database and knowledge of drillhole deviation statistics. Particularly, the first fact is important as the deflection spreading in the corresponding quarries gives an impression of how large the deflection normally is when the drill steel and the drilling parameters are the same. Tables and “Bull’s Eye” figures of the supplementary measurements are shown in Appendix J.

Below, graphical illustrations of the testing program deflection measurements are shown as “Bull’s Eye” diagrams. The “Bull’s Eye” diagram is well suitable for such presentations, as direction, size and symmetry all appear. The figures can easily be used to control the geological site investigations, and have been used to support the descriptions of the rock mass characteristics in Section 5.6.

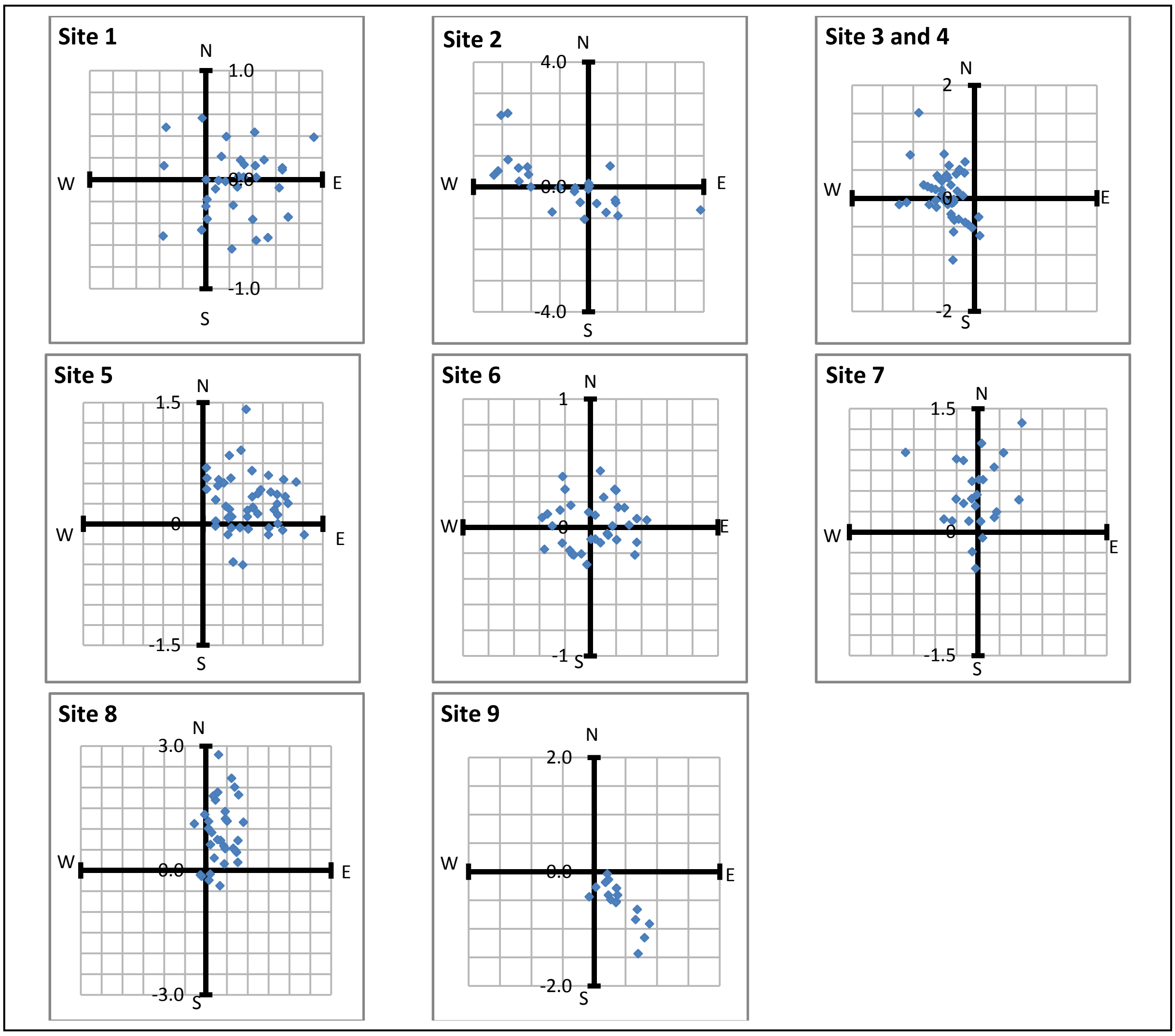


Figure 37 "Bull's eye" plot (North – East plot) of the deflection error in the bottom of the drill bit testing holes. N equals the magnetic north. Absolute deflection measured in metres. The grid scale is equal for both axes.

The drill bit testing program was initiated to examine and quantify the possible effects different drill steel parameters have on deflection. As variation and spreading in the results were expected, and the fact that only 2-4 holes were possible to drill within the same site and blast, due to wear and productivity limitations, it was necessary to compare the bits' properties in different rock masses. In total, there are 9 different sites as described in Section 5.6.

Comparing the diagrams from the standard production drilling (Appendix J) with the test program drilling sites 1 through 7, it seems to be difficult to draw any conclusions, as the deflection distribution is wide in both cases. However, in the following, the results based on detailed studies of each drilled hole in the test program are shown, and there seems to be grounds to conclude over several combinations of deflection versus drill steel design.

General Bit Design

As in the net penetration analysis, a comparison of the 89 mm bits across all sites and different designs are made first. Based on the average deflection, at the bottom of the hole, for each bit, Table 9 shows the ranking of the 89 mm bits from all sites and their total sum of rank numbers. Though the rankings seem to differentiate some of the drill bits, the results are not unique between the sites. It seems that the deflection properties of the bits are less clear than e.g. the net penetration rate properties.

The following drill bit parameters are analyzed, regarding deflection:

- Net penetration rate
- Drill bit diameter
- Drill bit length
- Wear
- Drill bit front design
- Button shape
- Button size
- Skirt-bit ratio
- Rod-bit ratio

The analyses of net penetration rate, bit length, wear and button size are not discussed further in this Chapter, as they showed no clear trends. However the corresponding diagrams are found in Appendix R and Appendix S. A general view of the entire drill bit testing results is given in Appendix E and Appendix K.

TEST LOCATION	Hole depth	Drill bit 2 Semi-ball. Drop centre	Drill bit 3 Sperical Drop centre	Drill bit 4 Semi-ball. Flat front	Drill bit 6 Semi-ball. Drop centre	Drill bit 7 Sperical Drop centre	Drill bit 9 Ballistic Drop centre
Site 1	14 m	5	2	6	3	1	4
Site 2	12 m	3	1	6	4	2	5
Site 3 and 4	14 m	2	5	1	3	4	6
Site 5	14 m	6	5	2	3	4	1
Site 6	12 m	5	3	1	4	5	2
Site 7	12 m	5	4	5	3	1	2
Site 8	17 m	4	5	3	1	6	2
Site 9	14 m	3	5	4	2	6	1
Sum of ranks		33	30	28	23	29	23
Rank average		4.1	3.8	3.5	2.9	3.6	2.9
Standard deviation		1.4	1.6	2.1	1.0	2.1	1.9

Table 9 Deflection rank figures from all sites. 89 mm bits.

Drill Bit and Rod Diameter

When speaking of deflection and drill bit diameter, the rod diameter must be included in the assessments. Larger bit diameter alone will give straighter holes due to (normally) increased length of the bit, however, if the rod diameter is the same, the relative stiffness of the drill rod will decrease and the bit may start wobbling, giving increased deflection as described in Section 4.5. More deflection may also arise due to higher free bending of the rod in the hole. In the test, several drill steel combinations were tried, and the results are discussed below.

In Appendix L the effect of the drill bit diameter and the rod diameter is analyzed. Site 1, 2 and 5 in these appendices show clearly that 76 mm bits with T45 rods deflect more than 89 mm bits with T51 rods. From Site 8, it seems that 76 mm bits deflect more than 89 mm bits with the same rod size (first rod T51 as guiding, the rest T45). At Site 5, Bit 10 (89 mm and T45 rod) has considerably more deflection than the corresponding 89 mm and T51 bits. The deflection values lie in the top of the 76 mm bits with T45 rod. This indicates that the wobbling effect (rod – bit diameter ratio) might have given an additional deflection compared to the 76 - T45 steel combination.

Analysing the 89 mm bits with the 102 mm bits from Site 3, 4, 5, 6, 8 and 9 (all with T51), no unique wobbling effect can be seen, though the majority of the results shows an increased tendency. The relative range is apparently too small to see any clear differences, and further investigations of more extreme ratios, e.g. T51/115, T51/127 and T45/89 mm, could have been interesting to carry out.

The main conclusion of this discussion is that the major deflection reducing effect, concerning the above factors, seems to be the rod diameter. The effect of wobbling seems to be smaller and more random.

Going from T45 to T51, the rod stiffness effect seems to give a reduction in deflection of about 20 – 50 %, at hole depths between 12 m and 14 m and in the actual rock conditions.

Skirt Design

In hard rock quarries, standard bits with standard skirts (see Bit 8 in Appendix C) are rarely used. Due to jamming problems and probable drill string losses if loose rocks or drill cuttings and dust along with water fall, into the hole during drilling, bits with retrac skirts are used instead. Besides the jamming advantage, the retrac design results in straighter holes due to the guiding effect of the wings on the skirt.

All bits in the test were supposed to have retrac skirts and guide wings, however by a mistake Bit 8 was delivered as standard designed skirt. Due to operating problems as described above, the bit was taken out of the test after drilling 8 holes. The results from the 8 holes (Appendix K, Site 1 and 3) show a remarkable increase in deflection for this bit, which is most likely due to the standard skirt design. Bit 8 has a maximum deflection of 2.2 metre and a minimum of 1 metre in 14 m holes, respectively 16 % and 7 %. The comparable bits have maximum 0.35 cm (2.5 %) deflection at the bottom of the hole. The results are removed from the other analysis, as the standard skirt seem to dominate all other bit design parameters.

Looking at Appendix M and the bits with retrac design, a vague skirt end – gauge bit diameter ratio effect seems to appear. Larger skirt diameters seem to give straighter holes. Interpretation of the measurements seems to give a deflection reduction up to 10 % per mm increased skirt diameter.

The length of the skirt (i.e. the bit) will affect the hole straightness, similar to the effect described in Figure 17. The bit assortment tested in the test program show no significant trend according to bit length. See Appendix R.

Button Design

Primarily the button design is used to improve net penetration rate, as ballistic bits drill faster than spherical bits. See Section 2.4. Additionally, a secondary effect is straighter holes for certain rock mass properties, see Section 4.1.

The figures in Appendix N give no unique deflection characteristics between the different button shapes. Comparing the results to rock mass properties, it seems that the deflection decreases dramatically when using ballistic buttons at sites with soft rocks. (Ref. site 8 and 9 with high elasticity and low surface hardness, see Appendix D). There is even a clear difference between semi ballistic and ballistic buttons. The semi ballistic button bits seem to reduce the deflection of the spherical button bits by 30% - 50%. The fully ballistic button bits seem to reduce the deflection by 20 % - 30 % more.

In the hard rock sites (sites 1 through 7), the results show no unique trend, and other factors seem to be more important.

Analyses of button size and deflection have also been done, however this seems to be most important to wear and bit life. Ref. Section 3.

Front Design

In theory, drop centre bits compared to flat front bits, drill straighter holes. The figures in Appendix O show a relatively large spreading in drop centre data, but the decreasing tendency using DC vs. FF seems to be significant if the button shape effect is removed. The effect may be in the magnitude of 10 % to 20 %

Due to wear the front becomes more and more convex and this is supposed to affect the deflection. No significant or unique effect of this is found in the analyses.

Hole Length

Nearly all holes within the diameter range of 76 to 102 mm measured in this thesis have to some extent deflection down the hole.

The deflection propagation may vary in the same rock conditions. Some holes deflect through the whole drilled length, while others deflect in a certain part of the hole until they reach a linear direction. See Figure 38 and Figure 39.

The ranking of deflection between holes may vary down the hole. In the analyses, the measurement at the bottom of the holes are used. Such variations are believed to be due to geological conditions. The average for each bit at each site are used to limit the effect of these variations.

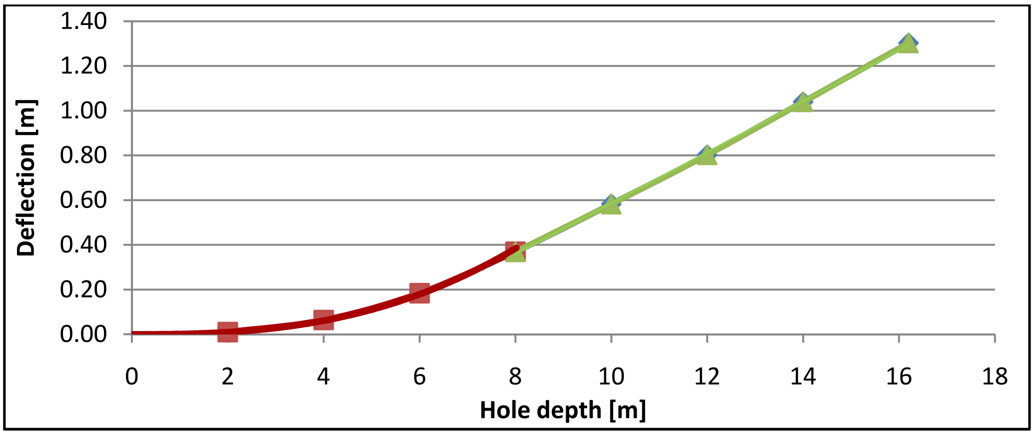


Figure 38 Typical deflection path, red and green line marked line. High deflection in the beginning, and then reaching an apparently linear deflection. Bit 2, Hole 11, Site 4.

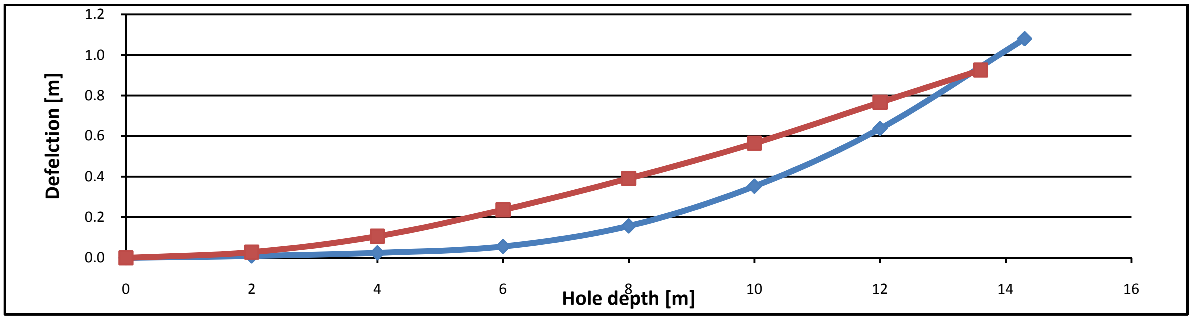


Figure 39 **Different deflection course in the same rock mass (Site 7), giving approximately the same deflection at the bottom of the hole. Red line: Bit 2, hole 13. Blue line: Bit 14, hole 23.**

Comparing the most bending holes of all holes available in the thesis work (almost 500 holes), with the rule of thumb given by Devico (i.e. maximum practical deflection of T51 steel is about 60 m curve radius. See Section 4.6), it seems that this rule more or less gives the upper limit of natural deflecting holes. Only Bit 8 (102 mm, T51, standard skirt) has deflection above the rule of thumb. The second most deflecting hole, from all measures, is just below the 60 m curve radius. See Figure 40.

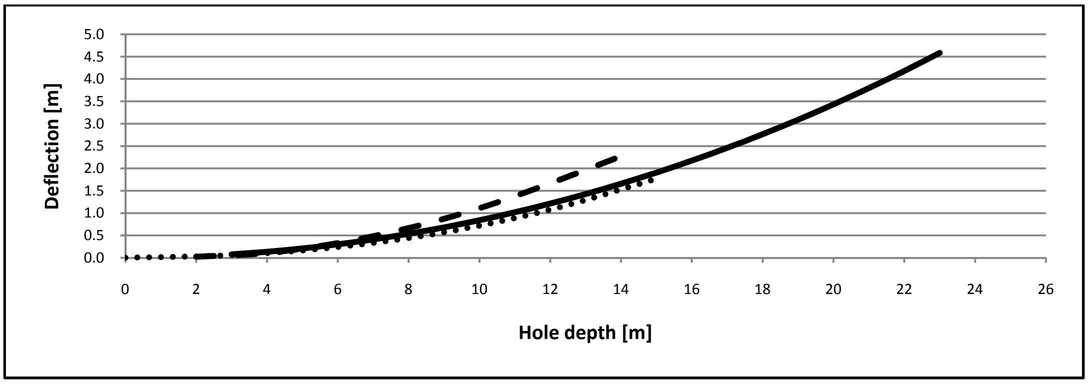


Figure 40 **Natural deflection compared to designed deflection.**
Solid line – theoretical curve with 60 m radius (T51 rod).
Stapled line – max measured deflection for T51 rod and standard skirt 102 mm diameter.
Dotted line – max measured deflection guided skirt T51 rod and 89 mm diameter.

Derivation of the deflection power function was not an issue in the primary stages of this thesis, and little focus has been put on this. Further analysis on whether different rock mass properties or drilling equipment designs give unique deflection power functions is interesting. The data in this thesis may be used for preliminary studies, however, more exact geological registrations must be carried out and the analysis must be planned from the start.

Drilling Parameter Adjustments

There was no room for testing different drilling parameter settings in the drill bit testing program, but results from two of the master theses included in this thesis show some studying on this issue.

In Spilling's master thesis, two tests with changing percussion pressure were carried out in two different quarries [41]. The results of these tests (see Table 10) show decreased deflection with decreased percussion pressure.

TEST 1		TEST 2	
PERCUSSION PRESSURE	DEFLECTION	PERCUSSION PRESSURE	DEFLECTION
140 bar	3.40 %	120 bar	1.3 %
165 bar	3.45 %	140 bar	3.8 %
195 bar	4.34 %	190 bar	4.9 %

Table 10 Comparison of hole deflection with varying percussion pressure.
Test 1: Roc 748 HC, 70 mm diameter. Gneiss, 13 m hole depth
Test 2: Ranger 700, 89 mm diameter. Gneiss, 12 m hole depth.

In the Bakketun and Gården master thesis, two consecutive blasts were measured in the same quarry [42]. When drilling the latter blast, the operator increased the rotation speed from 60 rpm to 90 rpm. The deflection distribution curves of the two blasts are presented in Figure 41, and it shows a decrease in the average deflection from 8.8 % to 6.7 %.

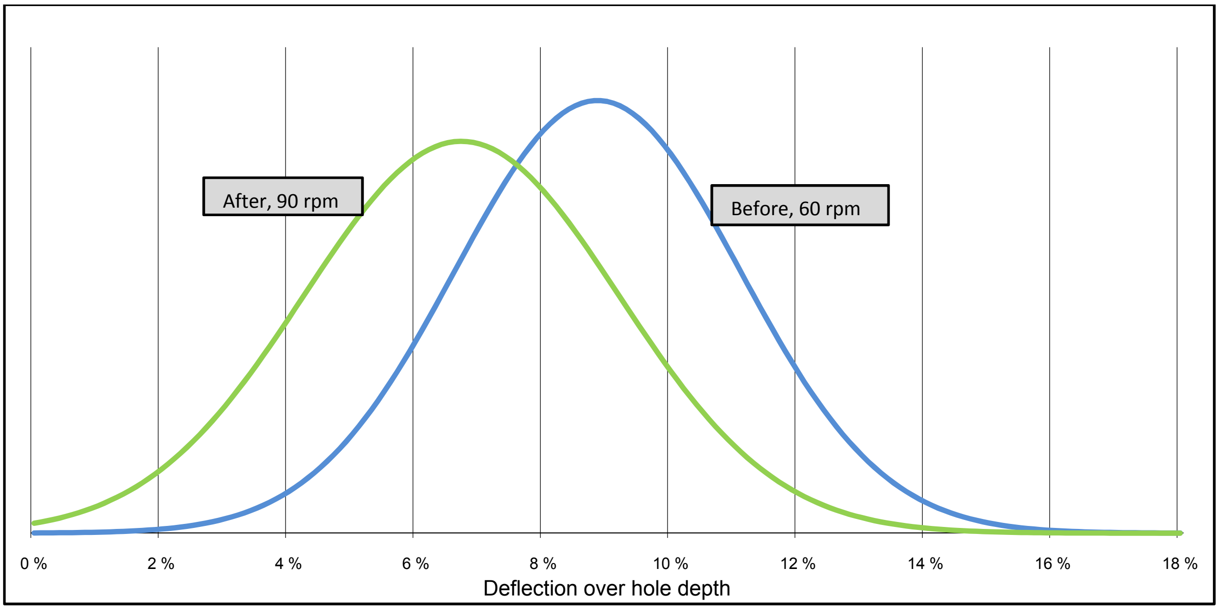


Figure 41 Normal distribution of deflection measurements of two consecutive blasts, in the same quarry, drilled with different rotation speed.
Top hammer drilling, 89 mm diameter holes, 15 m hole length [42].

Drilling method

Spilling's master thesis also shows deflection difference results concerning drilling methods [41]. In Appendix J, the measurements are shown as quarry 1 and quarry 4. The results from quarry 1 show a huge difference in deflection when comparing DTH with 140 mm hole diameter and top hammer with 89 mm hole diameter. The measures show that DTH has less than 1 cm/m in average, though the top hammer has 4 cm/m. The holes were drilled within the same blast and under the same rock conditions.

In quarry 4, DTH and top hammer are compared with the same drill bit diameter (100 mm), also on the same blast. The difference is not as clear as above, but a tendency seems to be found: DTH has in average 2.4 cm/m (std.dev 2.0) and top hammer has 3.6 cm/m (std.dev 2.1). The tests are carried out in granitic gneiss, with 15 m hole depth and both drill rigs were adjusted for optimal drilling capacity.

6.5 MARKING AND COLLARING DEVIATION

Ideally, the marking and collaring errors should be measured separately. First, the marking spots coordinates should be measured and controlled against the theoretical coordinates in the blast plan, and the collaring deviation should be measured after drilling. In the investigated quarries, no theoretical blast plan was made and the marking error could not be measured. Additionally, the drillholes were marked during drilling, and only a couple of the forthcoming ones, making it complicated to get the marking coordinates and measure the collaring deviation alone.

However, the lack of the theoretical start point does not necessarily affect the quality of the measuring of drillhole placement error, as long as the drill pattern is known. By measuring actual hole spacing and burdens from the actual coordinates of the neighboring drillhole after drilling, a measure of the drilling accuracy can be made. The measured deviation will be the total placement deviation, including both marking and collaring errors. Systematic marking errors, e.g. all holes marked in a displaced grid, will not be discovered. However, this is of minor importance as the relative current drill pattern still is intact and the blasting will be unaffected.

When marking the holes during drilling, as done in the quarries investigated, the placement error of one hole will not affect the hole spacing or the burden of neighboring holes. A new measurement is made according to the drilled hole, whether or not it is correct. Contrary, if marking a complete blast round at first, or using GPS, each hole will have a fixed place independent of the other holes' placement. The result of these two different procedures is that the first one may give a change in total drilled meter per blasted volume, and the latter will have a fixed amount of drilled meters within a given total blast area.

Table 11 presents a summary of the marking and collaring deviation analyses made. The burden and hole spacing errors are calculated as the absolute length difference between the planned values and the shortest distances (S_{act} and B_{act}) to the neighbouring holes.

The drill pattern error is calculated as the absolute difference between the planned drill pattern area (S_{plan} x B_{plan}) and the actual drill pattern (S_{act} x B_{act}) for all drilled rectangles. The SxB product is not derived directly from the burden and space errors, as the these are absolute values .

LOCATION		QUARRY B	QUARRY C	QUARRY X
DRILLING EQUIPMENT		TOP HAMMER 89 mm	TOP HAMMER 89 mm	ROTARY DRILLING 311 mm
NUMBER OF HOLES		136	91	81
BURDEN	Planned	2.5 m	2.9 m	7.0 m
	Average burden error[1)]	±0.15 m	±0.17 m	±0.11
	Maximum burden	3.13 m	3.42 m	7.29 m
	Minimum burden	1.88 m	2.29 m	6.76 m
	Standard deviation	0.19 m	0.22 m	0.14 m
HOLE SPACING	Planned	3.2 m	3.8 m	9.0 m
	Average hole spacing error[1)]	±0.09 m	±0.07 m	±0.25m
	Maximum hole spacing	3.71 m	3.99 m	9.67 m
	Minimum hole spacing	2.84 m	3.56 m	8.18 m
	Standard deviation	0.13 m	0.09 m	0.32 m
DRILL PATTERN (SxB)	Average error[1)]	5.6 %	5.9 %	4.3 %
	Maximum positive error	17.4 %	12.9 %	9.2 %
	Maximum negative error	-15.8 %	-17.3 %	12.7 %
	Standard deviation	6.8 %	7.0 %	4.3 %

Table 11 Summary of marking and collaring deviation. [1)] Absolute values.

6.6 ALIGNMENT DEVIATION

The results from the analysis of alignment deviation are based on the drilled holes in the drill bit testing program, plus supplementary deviation measurements carried out in this thesis work. The alignment measurements are analyzed as inclination deviation and horizontal angle (azimuth) deviation, and finally, total deviation. The complete set of alignment data is shown in Appendix P. A summary of the data is shown in Table 12. As described in Section 4.1 and Figure 12, the initial inclination angle influences the horizontal angle deviation. Nearly all holes analyzed in this thesis have an initial planned inclination of 10°.

DEVIATION	BEST AVERAGE	WORST AVERAGE	AVERAGE ALL
Inclination	0.9 cm/m	2.9 cm/m	1.8 cm/m
Maximum	2.4 cm/m	7.8 cm/m	-
Horizontal angle	1.1 cm/m	2.3 cm/m	1.7 cm/m
Maximum	4.1 cm/min	9.4 cm/m	-
Total alignment	1.7 cm/min	3.4 cm/m	2.8 cm/m
Maximum	5.2 cm/min	11.9 cm/min	-

Table 12 Summary of the alignment deviation results. The best/worst values represent the lowest and highest values for the investigated sites (12 sites).

The analysis shows an acceptable average alignment deviation level. However, the maximum values may cause both economical and safety problems. An alignment deviation beyond 10 cm/m gives at least 1.5 m absolute horizontal deviation for a 15 m bench, and is over half the burden in the investigated sites (from 2.5 to 3.0 m), surely not favorable for the blasting safety or the fragmentation.

The best case maximum alignment deviation is 5 cm/m. As deviations of these sizes seem to be normal, and happening within nearly every blast, the probability of fatal accidents is significant and an economical saving potential is clear. Additionally, no collaring and deflection deviation are added to the alignment values. Though no deviation measures from GPS assisted rigs are made, it seems that such instrumentation will have a substantial improvement potential. The use of GPS in drilling and the actual benefits of it should be examined closer in future studies.

There are no significant size differences between the inclination and the horizontal direction deviations. At some sites, the inclination average, maximum, and standard deviation values are bigger than the horizontal deviation values. At other sites it is opposite, and it varies more or less fifty-fifty. In Section 4.1, the systematic direction angle error due to a close sighting point is described. The results of the measurements show no significant change in whether the average or the maximum values are due to this error.

For cleaned bench and even rock debris floor, the analysis shows no significant difference either, as long as the overall bench conditions are horizontal. However, two cases with heavily sloped cleaned benches show an increased spreading in the alignment deviation. Both the inclination and horizontal angle average values are almost doubled from the worst average, as well as the worst max values, which are close to 15 %.

Compared with the results from Karlsson and Ouchterlony's work shown in Section 4.1, the average alignment deviation is a little lower for the investigated sites in this thesis. Regarding the road cutting regulations in Norway and Sweden, only the best case average was within the demands of 2 cm/m.

6.7 WEAR

The bits wear properties are measured and documented by diameter wear and photography, as described in Section 5.3. The deformation and wear of the drill bits influence net penetration rate, and to some extent the deflection and this is considered in these assessments. In the following, the measurements are used to discuss the bit life, mainly by comparing rock properties and button design. There is no information about alloys and production methods of the drill bits. The assessments are based on the technical descriptions and visual bit design properties, which are assumed to be equal for all bits. The analysis will mainly be focusing on the 89 mm category bits that have drilled more or less the same length in the same rock conditions. The supplementary bits (76 mm and 102 mm) are drilled too occasionally and they will be mentioned only when appropriate.

Appendix Q presents the figures used in the analysis of service life of the bits.

The 89 mm category bits have all been drilled between 340 m to 390 m in abrasive rock (VHNR 700-800) and they have all been ground 8 times. Additionally, 100 m are drilled in non abrasive rock (limestone and marble), with no need for grinding. None of the bits is totally broken, but due to variable wear, it seems they have different drill meters remaining.

All bits are ground after equal drilled meters. The test shows that some buttons on the bits (particularly semi ballistic or ballistic button bits), in some cases were drilled more than the 1/3 wear flat, and the wear results may be influenced by this. Nevertheless, grinding the bits in different intervals was not carried out, as both the net penetration rate and the deflection measurements were the primary studying issues. Equal number of holes was the most important factor. So, if the bit life absolute values are not completely correct, hence the measurements show differences in grinding intervals, which also imply the different wear resistance capacity of the bits.

Figure 42 Different wear flats after 60 drm at Site 1. From left; Spherical, Semi ballistic and Ballistic.

As described in Chapter 3, the analysis indicates that the spherical button bits have least wear and that the ballistic button bit has most wear. The semi ballistic button bits lie in between. The measurements indicate that the spherical button bits may have doubled bit life compared to the ballistic button bits with the same button size in the current rock conditions and the same grinding intervals. See Figure Q.1 and Figure Q.2.

Analyzing the button size, it seems that larger gauge buttons have longer service life than smaller buttons. The effect seems to give as much as 50 % longer life by increasing the gauge button diameter from 12 mm to 13 mm and keeping the same number of gauge bits. The longer bit life is logic as the total volume of hard metal is increased. The same effect is obtained by increasing the amount of buttons and keeping the same button size. The figures show that Bit 4 with nine, 11 mm, semi ballistic buttons in gauge has lower wear rate than the Bit 2 with eight, 12 mm buttons with the same shape in gauge. Figure Q.3.

Normally when drilled in the current abrasive rocks (Site 1 to Site 7), the drill bits are taken out of production after 550 m to 600 m or at 5 mm to 6 mm diameter wear [66]. If used further, the drilling pattern should be adjusted to obtain a proper blasting result. This gives a wear rate of 1 mm/100 drm, including grinding. The maximum single value measured in the test is 3.16 mm/100 drm, the lowest is 0.11 mm/100 drm. The average wear rate for each bit is shown in Table 13.

Bit 2	Bit 3	Bit 4	Bit 6	Bit 7	Bit 9
1.59	0.80	1.41	1.05	0.81	1.72

Table 13 Average bit diameter wear for the 89 mm category bits.
Values in millimetres per 100 drilled meters.

The average wear rate variations are larger between the bits than between the sites, which indicate that the rock conditions have less impact on the spreading of the results than the bit design. See Table 14. Even though the rock conditions are relatively alike, the measures show higher wear rate for more abrasive rock (higher VHNR), as the theory describes.

	Site 1	Site 2	Site 3 and 4	Site 5	Site 6	Site 7	Site 8	Site 9
Wear rate	1.10	1.90	0.93	1.15	1.39	1.29	0.05	0.00
VHNR	800	815	723	742	725	703	125	125

Table 14 Average bit diameter wear for Site 1 to Site 7. Values in millimeters per 100 drilled meters.

The test results indicate that the absolute bit service life due to abrasive wear will end at approximately 7mm to 9 mm diameter wear. Then, the button length is reduced so much that there is too little steel body steel to hold the buttons and they will start to pop out. Assuming in average 8 mm diameter wear and using the estimated wear rates from the test, the bit service life is predicted. Experiences show an exponential relation between rock abrasiveness and wear [3]. A regression of the test results is shown in Figure 43, compared with the experienced data from NTNU.

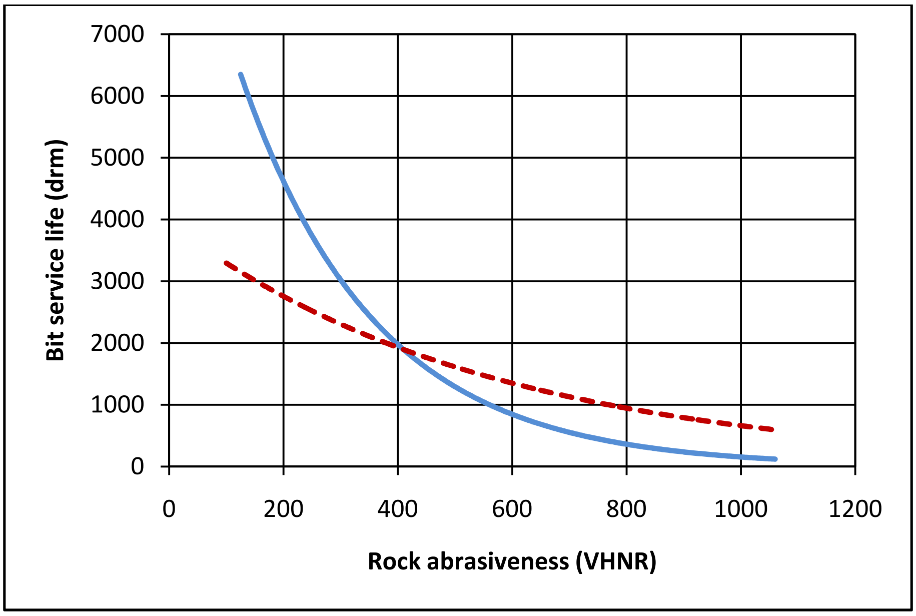


Figure 43 Service bit life as a function of VHNR, 89 mm bits. Red dotted line indicates the existing NTNU model.

For Site 8 and Site 9, no general body steel and button wear is visible, but the bit service life is said to be about 5000 – 7000 dm. Then, general material fatigue appears and the buttons begin to pop out and small body steel bits start to chip [67] [68].

The regression gives lower bit service life in the hard rock area than the NTNU model experiences, and the contrary in the soft rock. Additional information from a quartzite quarry (VHNR 1060) shows bit life less than 100 meters [13]. The regression curve suits this information better than the NTNU model, and it seems that the test results may form a basis for adjusting the model. The hard rock condition in the test program is relatively equal and information from other rock conditions must be added.

VHNR is used as the main abrasiveness parameter as this is also used in the existing NTNU model. Other abrasiveness parameters are also analyzed, and there is a correlation between bit wear for most parameters. See Appendix Q and Figure Q.4

From these figures, it seems that the Sievers' J miniature drill bit test (SJ) [12] and the following Sievers' J Interseption Point test (SJIP) [71] may be an alternative to VHNR as a bit life prediction parameter. Not only the correlation seems to be good, but also the simplicity of the laboratory test method is favorable. The SJ are used as a parameter in the DRI^{TM} test and this allows for the two tests to be performed simultaneously. Further analysis on this matter should be carried out.

6.8 ECONOMICAL ASSESSMENTS

Improving drilling performances, drillhole quality and life of drilling equipment will change the economy of the drilling process. Besides this, the improvement of the drillhole quality will have an economic potential for the entire quarrying process in many fields, as described in Section 0.

In the following, economical assessments of the three drilling characteristics focused on in this paper are made. They are:

- Net penetration rate
- Wear
- Drillhole deviation

<u>Net Penetration Rate</u>

Under equal drilling conditions, the net penetration rate will increase if the drill bit diameter is reduced. However, as described in Section 2.4 and Figure 3, the cost benefits of increased net penetration rate will be overrun by increased specific drilling and charging costs. Therefore, the economic analysis must be related to a fixed drill bit diameter to be able to see the economical effect of changes in the net penetration rate.

Increased net penetration rate will reduce the net drilling time for a hole. Fixed times, such as rod adding, rod pullback and moving between the holes will not be affected. The net drilling time share of the gross drilling time depends on the actual net penetration rate, bench height and the terrain conditions.

The following cost example shows how the costs (NOK/drm) will change by increasing the net penetration rate.

Figure 44 shows how the costs decrease with increased net penetration rate for different net penetration rate levels. At 50 cm/min the costs will be decreased by approximately half of the relative increase in net penetration rate. At 100 cm/min and 200 cm/min the cost changes will be 37.5 % and 25%, respectively. As we see; less effect of increased net penetration rate for faster drilling.

The estimations show that the level of the net penetration rate is the main effect concerning the net drilling time share of gross drilling time. The effect of bench height and changed fixed times (i.e. moving time) is about 1/10 of the net penetration rate effect. The trend is more or less the same for all drill methods, drill bit diameters and specific drilling.

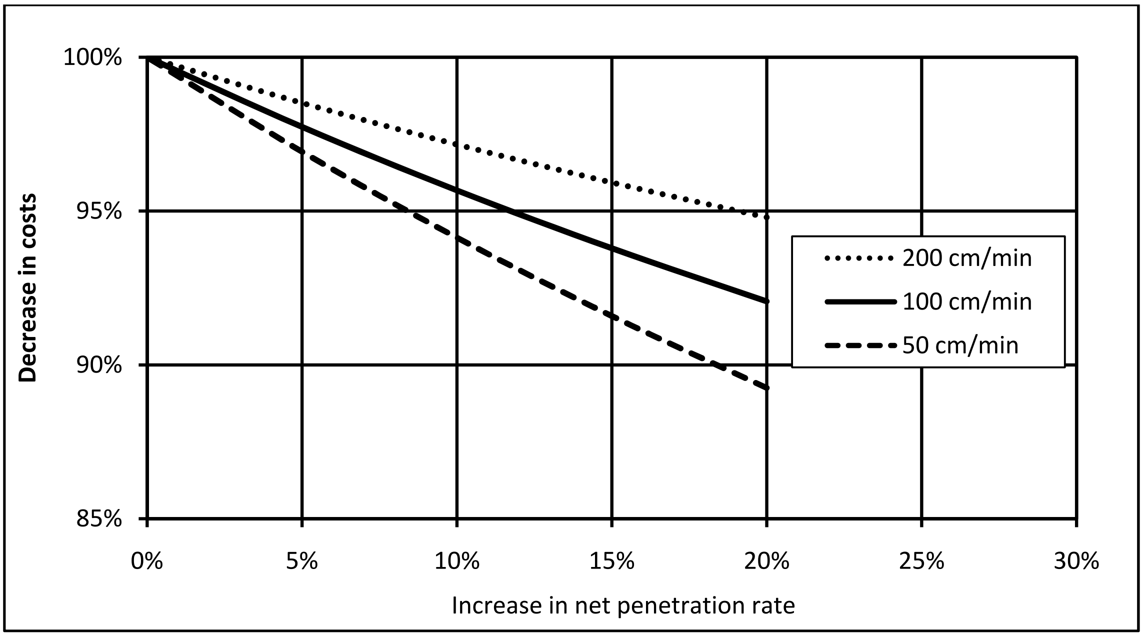


Figure 44 Decrease in drilling costs as a function of increased net penetration rate.

Wear

Drill bit wear costs are dependent on bit life, purchase price and the costs for regrinding (equipment and labour).

Normally, the regrind intervals increase more or less proportionally with bit life, giving the same number of regrinds per bit. The regrinding time will increase by size and number of buttons. This is more or less proportional with bit price. The relative bit wear costs will be independent upon the grinding operation and bit price.

Hence, bit wear costs will increase proportionally with drill bit life.

Drillhole Deviation

Economical benefits by improved hole straightness may be achieved by:

- Less specific drilling
- Improved drilling features
- Better blasting result

The cost potential will be dependent on the original deviation state in the quarry or mine. In the following example estimations a normal aggregate quarry with 76 mm drillhole diameter and 15 m bench height is assumed. It is supposed that the optimal average deviation will be about 2 %, and that the original deviation states have 5 % and 8 % average deflection in the bottom of the hole. Detailed analysis of the basic assumptions will not be presented here. They can be seen in [70].

The estimations are based on normalized costs derived from the Rock Quarrying report series, and the basic costs are not estimated in detail. However, the effect of drilling straight holes is analyzed in detail. The results must be considered as approximate values and may be both higher and lower comparing with real projects.

The comparison includes bench cleaning, drilling, charging, secondary breaking, loading and transport. Crushing and further processing is not included, however it is expected that these costs will also be reduced when blasting with straighter holes. Table 15 summarize the effects gained by improved drillhole deviation.

IMPROVEMENT EFFECT	8 % to 2 %	5 % to 2 %
Less drilling	14.4 %	7.4 %
Less subdrilling	1.3 %	0.6 %
Less charging	15.7 %	8 %
Increased drill rod life	33.3 %	11.1 %
Increased drilling efficiency	12.5 %	6.3 %
Increased loading capacity	18.9 %	10.0 %
Less secondary breaking	40 %	25 %

Table 15 Single effects of improved drillhole deviation.

Costs for bench cleaning and transport are assumed to be independent of the drilling deviation. Reduced waste and low price products because of improved drilling quality are found to be insignificant (about 1 ‰ in this example). Safety precaution costs are not included in the comparison.

Based on the normalized costs from the Rock Quarrying report series, giving a total unit cost of 38.5 NOK/sm^3 delivered in the primary crusher, the total estimated cost reductions because of improved hole straightness will be as follows:

DEVIATION REDUCTION	EFFECT	COST EFFECT
8 % to 2%	12.7 %	5.2 NOK/sm^3
5 % to 2 %	7.0 %	2.7 NOK/sm^3

Table 16 Total effect of improved drillhole deviation. Rock mass delivered crusher.

Drilling improvement efforts will also make some extra costs, i.e. reduced drilling capacity and more expensive drill steel. Estimations indicate that the costs will be about 1.2 NOK/sm^3 and 0.6 NOK/sm^3 in the eight-percent and the five-percent examples, respectively.

Drilling straighter holes is economically profitable. In addition, reduced crushing costs and safety issues must be included to the positive account.

6.9 CONCLUSIONS BIT TESTING PROGRAM

Despite the natural spreading of the rock features within each site and the relative narrow assortment of drill bits, I believe the extensive drill bit testing program has revealed some unique results. The test measurements do not show ground breaking results, however that was not the purpose of the work. The goal has been to increase the general knowledge and data base of practical field study results, and try to verify some of the basic theories which are described in the beginning of the thesis.

Net Penetration Rate

Within the rock masses tested in this thesis, the study shows that drill bits with drop centre and ballistic or semi-ballistic shaped buttons give the best net penetration rate features. This is compared to flat front bits or spherical shaped button bits.

The drill bit diameter plays a very important role in the net penetration analysis. The study shows that the net penetration rate decreases between 1 % and 2 % per mm change in bit diameter.

The study shows that drill bits with semi-ballistic shaped buttons have 8 % to 9 % higher net penetration rate than bits with spherical shaped buttons. The study reveals no clear difference between semi-ballistic buttons and fully ballistic buttons.

No clear difference in the net penetration rate is recorded between drop centre and flat front design alone.

The net penetration rate decreases with wear. Dependent upon button shape, it seems that the effect is between 10% and 20 % at the end of the bit service life. Spherical button bits maintain their speed best.

According to drilled depth the net penetration rate decrease varies slightly between the sites. The average net penetration rate reduction is measured to 4 % per rod (3.66 m).

Deflection

The general deflection level is dependent on the geology. Generally, in aggregate quarries the rock mass has relatively low anisotropy and jointing degree, because of the final product demands on e.g. flakiness. The average deflection values measured in this thesis must be related to that, and deflection may be higher in rock masses with unfavourable conditions.

The average deflection level in the investigated sites lies around 4 % for 15 metre bench heights. The best average is 2.8 % deflection and the worst is 8 %. The deflection increases with increased hole depth. Throughout the entire study, the maximum deflection, for 89 mm bits and T51 rods, seems to equal the bend of a radius of 60 m; respectively 5 % deflection at 6 m hole depth, 10.1 % at 12 m and 15.4 % at 18 m.

The major deflection reduction effect measured is the rod diameter. The reduction seems to be as high as 50 % between the rod diameter steps: 38 mm - 45 mm and 45 mm - 51 mm. A smaller wobbling effect can be seen. If keeping the same rod diameter and increasing the bit diameter by 13 mm (half inch), the deflection seems to increase by 10 % to 15 %. The results are illustrated in Figure 45.

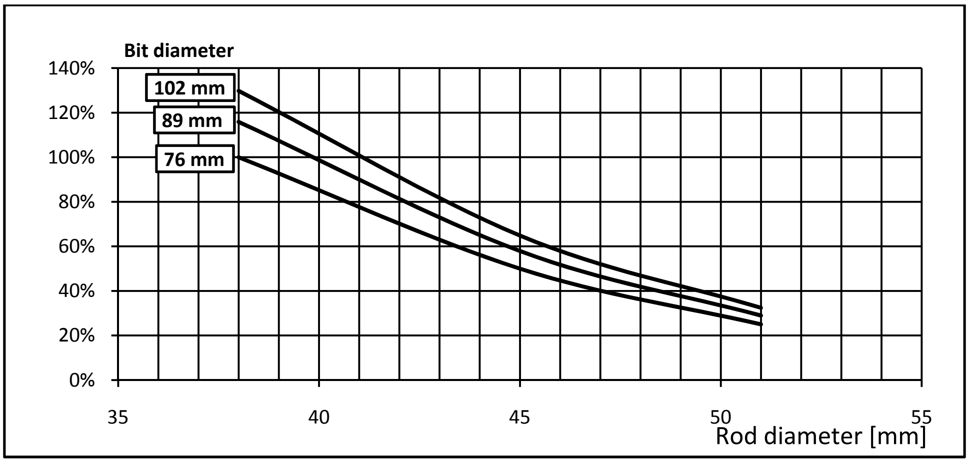


Figure 45 Deflection as a function of drill rod diameter and bit diameter.

Drill bits with guide skirt have a considerable deflection reduction effect, compared to standard skirt bits. Regarding deflection and blasting safety, standard skirt bits should not be used at any circumstance in bench drilling.

For guided skirts, the skirt – gauge bit diameter ratio should be as large as possible without reducing the flushing and increasing the torque substantially. Reducing the skirt diameter by one percent point the studies imply an increased deflection of 10%.

Theoretically, sharp buttons deflect less than blunt buttons. The study shows no unique trend, except for the soft rocks tested. Semi-ballistic buttons seems to reduce the deflection compared to spherical buttons by up to 50 %. Fully ballistic buttons even 25 % points more.

In the study, the drop centre bits, compared to the flat front bits, seem to reduce the deflection by 10 % to 20 %.

Reduced rock drill energy output gives less deflection. Though the data is limited, a supplementary study implies that the deflection is more or less proportional to the energy output.

Adjusting the rotation speed, independently of percussion and feed pressures, may improve hole straightness. In a supplementary study, the average deflection in two consecutive blasts varied by 25 % due to changed rotation speed. Dependent upon the rock mass properties, both increasing and decreasing rotation speed may give optimized hole straightness.

A general impression of all the deflection studies and literature found, is that DTH, Coprod and rotary drilling with drill strings larger than 115 mm have insignificant drillhole deflection (<1 %), within traditional open-cast drilling conditions.

<u>Collaring and Alignment</u>

Collaring deviation is measured to be slightly less than two times the drill bit diameter for the top hammer holes measured (i.e. 76 mm to 102 mm) without GPS instrumentation. This gives an average drilling pattern divergence of ±5 %. Collaring deviation is reduced by using GPS. The results from the rotary drilling equipment with GPS, shows approximately one drill bit diameter deviation.

From the 12 blasts investigated, in traditional quarrying, the best average alignment deviation is 1.7 cm/m. The worst average is 3.4 cm/m. Alignment deviation is strongly dependent upon the motivation of the drill rig operator. Top motivation should keep the alignment deviation below 1 cm/min.

Variation in alignment and collaring deviation between cleaned bench or rock debris drilling seem to be insignificant. Nevertheless, heavily sloped terrain seems to affect the alignment deviation negatively. The studies show a doubling of the worst site average.

<u>Wear</u>

The wear measurements indicate that bits with spherical buttons have 50 % longer service life than semi-ballistic button bits and 100 % longer service life than ballistic bits. Increasing the gauge button diameter by 1 mm seems to increase the bit diameter wear by as much as 50 %.

The wear studies indicates that the Sievers' J miniature drill bit test may be a very good alternative when predicting drill bit service life.

Economical Assessments

The effect of increased net penetration rate affects the drilling costs by half the increase in net penetration rate at 50 cm/min. The effect is lower for higher net penetration rates.

Improved hole straightness reduces the total production costs. Depending on the prevailing conditions the cost reduction potential is about 5 % to 10 % for rock mass delivered to the crusher. Reduced crushing costs and safety issues must be added to the cost reduction account.

7 ROCK QUARRYING REPORT SERIES

7.1 INTRODUCTION

Rock Quarrying and rock mass movement is today carried out in connection with:

- Open-cast mining
- Production of rock mass for road, rail and airport construction
- Production of rock for filling of dams, harbour construction, crushed stone, etc.
- Blasting of road cuts
- Preparation of building sites, industrial areas, etc.

As mentioned in the description of the thesis, the main emphasis has been on rock quarry operations. The topic is divided into six main areas of interest:

1. Rock quarry – placement and design
2. Blast design
3. Bench drilling
4. Loading
5. Transport

The individual work tasks are closely connected. The operational requirements and equipment must be chosen based on a complete evaluation of the quarry operation. The

choice of explosives, drillhole diameter and ignition system should be adapted to the quarry system, geological conditions and demands on the product.
The ROCK QUARRYING report series published as a part of this thesis, embrace the areas mentioned above. New data and operational methods are continuously adapted into the series, however, the historical data and theories still remains as the solid foundation of the reports.

The following sections summarize the work carried out concerning republishing of the report series. The developing and updating work on the existing reports and the work made on the new ones are described.

7.2 REPORT 12A-08 ROCK QUARRYING QUARRY DESIGN

The Quarry Design report has not been published as a part of the Rock Quarrying series before.

The work has been consisting of translation of Norwegian texts and editing the different parts into one united report. The text is based on parts of the Construction Engineering compendium and projects in carried out in cooperation with the architect office Plan Arkitekter.

7.3 REPORT 12B-08 ROCK QUARRYING BLAST DESIGN

The Blast Design report is an update and renewal of the previous report published in 1998 [5].

A complete text, figure and example editing process has been carried out. New explosive and firing products are implemented. The previous report had several weak points, particularly as a textbook. Though the estimation model has been correct, the definitions and explanations of the blast design model were incomplete and at times missing. The estimation examples were confusing and hard to follow. This is, as I see it, well prepared in the new report.

The methodology of analyzing the blasting result for improvement purposes was a bit disorderly in the previous report. The Blasting Result chapter in the report is now expanded and more concrete.

A considerable job is made to adjust the parameter symbols in the report according to the descriptions set by the ISRM Commission on Rock Fragmentation [69].

Besides the editorial work mentioned above, the report has been expanded to cover information and experiences about two special blasting applications: Block stone blasting and road cuts. Particularly the first is well described.

7.4 REPORT 12C-08 ROCK QUARRYING BENCH DRILLING

The Bench Drilling report is based on previous reports published in 1978, 1990 and 2000 respectively [1], [2] and [3].

The main work of the report has been to implement drilling studies carried out at the Construction Engineering Research Group at NTNU since the last report in 2000. A total of 30 different drill rigs and sites are studied and included in the report. Mainly, the studies are carried out by master students under my supervision. These studies come in addition to the extensive drilling program described in this thesis report (volume 1).

The extent of each study varies, yet the least resource demanding studies will include one week of work (one day planning, one day travelling, one day studying, one day of data treatment and one day in the laboratory). A rough estimate implies that in total one man-year of capacity studies are included in the latest Bench Drilling report.

A summary of the sites visited are shown in Appendix A. In the Bench Drilling report's appendix (volume 4), a detailed list of the visited sites including the equipment are listed.

The drilling studies have been used to update or confirm the net penetration rate figures in the report. The top hammer drilling studies found a basis to adjust the previous graphs. Some old rock drill models were omitted and now the hydraulic top hammer graphs are put into one figure (Figure 3.2). Previously, there were two.

Since the previous report there has been an extensive development within the DTH drilling, particularly within the drill rig design. The previous reports were based on large mining rigs with masts. The recent years, DTH rigs similar to the large top hammer production rigs are developed. An implementation of the smaller DTH rigs is made in the report (Figure 3.6).

One of the major efforts made in the 2000 report were the study of Coprod rigs. Only one study of this is made in this edition, and no basis for changing the performance features of the Coprod method is made.

A couple rotary drilling studies are made. They revealed no ground for changing the existing performances.

Besides the standard net penetration rate studies, the extensive testing on different drilling equipment, carried out as a part of the thesis, is implemented in the Bench Drilling report. The main results from these tests are described in the Study Result chapter.

The new report has an expanded section concerning the basics of the bench drilling. In this part the deviation and deflection theory is new. Some of the deviation test results from the main thesis are shown in the Study Result chapter.

The NTNU cost prediction model has been completed in the new report. The report may now be used for detailed estimations of costs, without the need for auxiliary literature. It has the latest prices on drill rigs and drilling equipment (December 2008).

Besides the performance and cost updates, and the expansion of the Bench Drilling Basics, Drilling Costs and Study Result chapters, a complete editorial and proof reading process have been performed.

7.5 REPORT 12C-08 ROCK QUARRYING LOADING

The Loading report has previously been published in 1978 and 1992, respectively [6] and [7].

The main work of this report has been to implement hydraulic backhoe excavators in the prediction model. These machines were not included in the previous reports. Further, updates of the existing performance and cost figures for wheel loaders and hydraulic front shovels.

In total, about 40 individual loading machines have been studied. Compared to the drilling studies, the loading studies are usually a bit smaller due to the numerous repetitions and no need for laboratory testing. In contrast, the amount of numbers analysed are higher. In average four days are used per study, in total roughly three quarters of a man-labour year of capacity studies is used on field studies in the new report. An extensive part of this has been performed in one Master thesis [44]. See the report for detailed information about sites and machines studied.

In addition to the general studies, an extensive study of block stone loading performance is carried out. The main results of this study are included in the Study Results chapter.

The extent of the editorial work of the Loading report has been somewhat less than the three preceding reports. Nevertheless, all text is proof read, and adjustments are made concerning the succession of the report. Supplementary information is included in some parts as well.

As for the Bench Drilling report the NTNU cost prediction model is completed in the new edition. The latest prices of the loading machinery are included (December 2008).

7.6 REPORT 12C-08 ROCK QUARRYING TRANSPORT

The Transport report has not been published as a part of the Rock Quarrying report series before.

The main text has been translated from Norwegian. Most of the text is parts of the NTNU cost model report [8], which is previously only published in Norwegian. General information about the transport machines is included. Like the two preceding reports, the NTNU cost prediction model is included as well. The Transport report lay out is similar to the other reports.

A complete update of the performance and cost figures and examples are carried out.

8 SUMMARY AND RECOMMONDATIONS FOR FURTHER WORK

8.1 SUMMARY

The thesis work is divided in three major parts:

- The main thesis report, presenting an extensive drill bit program test.
- A report series about rock quarrying, consisting of five reports.
- One published paper regarding rock drilling safety

The major part of the thesis report volume 1 treats the drill bit testing; theory, research program and results. The overall purpose of the thesis work has been to establish a complete report series concerning rock quarrying. Parts of the report series has been partly published by the Construction Engineering Research Group at NTNU as project reports since the 1970s, however the complete series is finalized through the work of this thesis.

An extensive search in literature and in the internet for books, papers, conference proceedings and methods describing the scope of my work is done. To a large extent the literature found are focusing on basic principles, theoretical modeling or laboratory experiments, but to a little extent related to practical work. However, I would like to emphasize the work of Thomson Sinkala [34] and Finn Ouchterlony [37] of the Luleå University of Technology, which complements the work in my thesis, particularly on drillhole deviation.

The report series is meant to be a toolbox for the quarry industry, contractors and consultants, along with educational and research institutions, dealing with rock quarrying, traditional blasting and earth moving projects. The prediction models and the experience data recorded in the report series are widely demanded within the target group. Each report in the series has the same main title: ROCK QUARRYING, with the following subtitles:

- Quarry Design
- Blast Design
- Bench Drilling
- Loading
- Transport

Each report can be used within its field separately. Still, the consistency of the quarry processes is well obtained between the reports.

The Quarry Design report describes the main issues concerning practical and efficient design of quarries, as well as current esthetical norms. The combination of cost efficient and esthetical guidelines makes the report unique, and to my knowledge it is the first of its kind. The text is kept at a minimum as illustrations are widely used.

The Blast Design report presents the prediction model for estimating specific drilling and charging, developed at the Construction Engineering Research Group. The model is based on blasting studies made in the 1980s, as an alternative to Langefors' estimation model which is applicable for small blast hole diameters only. Further, the report is meant to be a tool and guide for optimizing the blasting process. The report shows how the various blasting parameters affect the blasting result and explains how to analyze the blasting result.

The NTNU Blast Design model may be used to make a complete blast design. Description of charging methods and explosive types are shown, along with the use of firing systems and the design of a firing plan. Principles for special blasting features, such as block stone production and road cut blasting are presented. Basic geological characteristics are treated.

The Bench Drilling report presents empirical drilling performance data from long term and continuous research work. At the Construction Engineering Research Group at NTNU, bench drilling has been a focus of attention since the 1970s. The report is built on, as far as I know, the world's largest and widest field performance database, excluding the drill rig and rock drill manufacturers' own databases. Adding the studies made through the work of this thesis, the report presents state-of-the-art performance data for the standard rock drilling methods:

- Top hammer
- Coprod
- Down-the-Hole
- Rotary drilling

In the report, a prediction model for estimation of capacities and costs for bench drilling is shown. The model provides rock mass characteristics, along with performance, service life and costs of drilling equipment. Rock breaking mechanisms and the principles of the drilling methods are shortly described. In addition to the model data, the report presents study results for more detailed drilling analysis and optimization of the drilling process. This includes drillhole deflection aspects which are new in this version of the report. To ensure the quality of future field studies, a detailed drilling study guide is shown.

Compared to the other reports in the series, the Bench Drilling report is the most extensive one concerning presentation of background studies.

The Loading report presents empirical data from studies of the rock pile excavation process in quarries and mines. The report has been updated from the edition published in 1992. The major development since the previous report is the introduction of hydraulic backhoe excavators. At present, the report contains performance data for the following loading equipment:

- Wheel loaders
- Hydraulic backhoe excavators
- Hydraulic front shovels

Besides the empirical capacity and cost prediction model, the report emphasizes the key role the loading operation plays in optimizing the quarry process. For instance, loading performance studies are well suited to characterize the blasting result. The report contains a well defined loading study guide, and presents typical study results needed to normalize the recorded data due to bucket fill factor, turning angle and truck size. The working principles of the different loaders are explained and illustrated.

The Transport report, in contrast to the other reports in the series, contains less field performance data. Regular studies show that the manufacturers' performance diagrams are well suited to decide velocity of the vehicles. The strength of the report as a quarry and mining tool, lies in studies of fixed times and the experiences on how e.g. poor road surface and geometry, along with narrow maneuvering areas affects the total transport cycle times. The report is well adapted for other transport works and the parameters for other types of bulk material are included. The basic theory about road grip and traction power is also presented in the report.

The complete Rock Quarrying report series presents an extensive compilation of field performance and experience data from the last four decades in Norwegian blasting and earth moving projects. To some extent data from other parts of the world are included as well. The prediction models are mainly based on field studies, collected in close cooperation with site management and the operators on site. The importance of onsite presence during testing has a very high priority in the data collection. In my opinion, the presence on site during the research is one of the strengths of the databases and the prediction models. Good and precise data is obtained and it seems to be the best practice for transferring recorded results into knowledge and into the continuous improvement process of quarrying operations.

It is important to emphasis the extent of resources needed to continuously keep a database up-to-date and to be able to use the report series in teaching. At the Construction Engineering Research Group at NTNU the report series are used in the teaching of master students. The reports are used in the basic courses as well as in the advanced courses. At the Construction Engineering Research Group, it has been a tradition to include master students in the process of updating the models, closely followed-up by a supervisor to maintain quality of the field measurements. I have also pursued this policy in my thesis work. Field studies of state of the art equipment and methods, along with correct research practice result in good candidates for the quarry and mining industry. By continuously implementing the master thesis results into the data base, the general data base increases as well as keeping the knowledge up-to-date.

Concerning the fact that the prediction models are based on empirical data from many individual sites with site specific parameters and conditions, the models may not always exactly correspond to the measured field performance data. However, analyzing field data against the model, acceptable correlations are found in most cases.

As I see it, the report series appears to be a complete and unique textbook collection and toolbox for practical teaching, planning and operation of quarries.

Along with the general updating of the report series it has been necessary to collect and generate new knowledge. I wanted to focus on the quality issue of the drilling process, embracing safety, performance and costs. The safety aspect has included both flyrock issues and the working safety of the drilling operator, more precisely the unexplored hazard of misfires remaining in the rock debris from overlying blasts.

Through the extensive drill bit testing program I have collected and systematized the factors affecting the drillhole deviation. The safety and cost benefits are described and discussed, along with resource assessments concerning implementation of deviation improvement efforts. According to the literature I have found during my work, no published work contains such information. The thesis work has increased the general data base on drillhole deviation, which may be used to quantify and classify the accuracy and quality of drilling. In my opinion, the increased knowledge about drilling deviation features, put together and presented in this thesis, has revealed and clarified a neglected economic potential in the quarrying industry, in addition to the safety concerns which are already well documented.

The work with the hazard of misfires in the rock debris clarified the causes of misfires, and which actions can be made to reduce the problem. Further, the published paper on misfire hazard presents a risk analysis model, which was developed to compare the total risk for fatal injuries by drilling through the rock debris as an alternative to cleaning the bench before drilling. The risk assessment shows that the rock debris drilling is less risky than cleaning the bench. The results of the analysis have been communicated to the responsible authorities, so that the current regulation text, which demands cleaned bench before drilling, may be reconsidered.

Concerning the overall safety of the quarry and mining work, the probability of fatal injuries as a result of misfire detonation on the bench is a minor risk. Some may question the research resources used and be of the opinion that the risk assessment focus should have been on other more prominent fatal injury causes, such as the flyrock issue. To a certain degree I share that opinion. Nevertheless, I defend the use of resources, as a clarification of the misfire problem has been demanded from the industry and the operators, who have been defying the regulations by drilling through the rock debris. Also the fact that finding and incidental detonation of misfires happens more or less regularly at quarries, construction blasting sites and underground blasting, defend my choice.

8.2 RECOMMENDATIONS FOR FURTHER WORK

In my opinion, the continuous work updating the Rock Quarrying report series should endure. Looking back at the preceding reports, my experience is that there will always be smaller and larger improvements within equipment performances and the quality of materials, along with new techniques and method developments. New drill rigs and rock drills are available in the market, GPS assisted drill rigs have made their entrance in the industry, along with intelligent machine systems on loaders and drill rigs, to mention some things. Further field studies must be carried out to maintain the legitimacy of the models.

In my opinion, digitalized logging systems in the machinery will improve the studying process, however exclusively as a supplement to the manual field studies and presence on the bench. Concerning this, I will emphasis that the use of students in the research work still is essential to maintain continuity and a quantity of field studies in a long term aspect. The research costs are kept low, and the field studies are less dependent upon the staff available at the university. My experience is that the students normally have a desire for in-situ testing and practical tasks in their final stages of their studies. Also I think it is positive that they in most cases are free of preconceived ideas and experiences.

In drilling specifically, automatically rod adding systems should be studied. When a sufficient amount of data is available the model may be adjusted according to net capacity and gross capacity measures. The effect of GPS concerning deviation control is also a development that should be studied. Deviation measurements should be carried out to extend the general data base, and further studies of a wider range of drill steel parameters should be performed. The effect of reduced rock drill power output is not completely investigated either. At the end, the major goal must be to get more accurate data on the cost effects with increased hole straightness.

Within drill steel service life, I think more work should be carried out to examine the relation between drill bit life and the Sievers' J laboratory test. The SJ test is easy to perform, and accurate rock data can be obtained cheaply. The SJ value is also a parameter in the DRI™ value, which is used to characterize the drillability of the rock, and in many cases two birds can be killed by one stone. A good correlation between SJ and drill steel life will improve cost prediction and the production planning.

In more general terms I will recommend quarries and mines to exploit their opportunities for documenting their operational characteristics, develop their own data bases and improve their premises for increased efficiency, lower costs and make larger profit. This includes capacity studies, along with close follow up of machines and equipment.

Concerning blasting work safety, the general focus on the matter from the authorities and trade organizations must continue. Nevertheless, I would recommend a tightening of the blasting regulations due to the excellent deviation measuring systems, which are available in the market. There should be no excuses for flyrock incidents anymore. In principle it is a good practice that all holes in the first row of a blast should be measured, along with scanning of the bench face. In Sweden [72] and Great Britain similar regulation descriptions already exist. Additionally, it should be considered to impose regular extended deviation measurements (entire blasts), to document the general deflection properties in the rock and to improve the premises for planning further works safely. Flyrock is by far the most dangerous aspect in blasting and must never be underestimated!

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PUBLISHED PAPER

ROCK DRILLING SAFETY - BENCH TOP CLEANING VERSUS BLASTED ROCK DEBRIS DRILLING

Due to document conversion error in the published version (i.e. some figures and equations) the following paper corresponds to the version submitted to the Blasting and Fragmentation committee. Hence, page breaks varies from the published version.

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ROCK DRILLING SAFETY - BENCH TOP CLEANING VERSUS BLASTED ROCK DEBRIS DRILLING

Vegard Olsen & Amund Bruland[1)]

ABSTRACT:
This paper presents analytical assessments around unintended misfire detonation, within rock debris drilling and bench top cleaning, primarily in the quarrying and mining industry. The main purpose of the work has been to see which operation has the lowest fatal accident probability. To be able to do this a risk analysis model is developed and the basics of the model are described first in the paper. Further a practical approach of the model use is presented. This includes a summary of general fatal injury statistical data, which is used to establish input parameters in the model. The result of the work shows that the rock debris drilling operation has at least half the probability of fatal injury compared to bench top cleaning. The result of the work should bring along rewriting of the Norwegian explosives regulation. The statistical data also shows that the blasting industry generally has a high fatal injury rate compared to other industries and basic risk reference levels. Risk reduction efforts concerning the examined operations are described.

Keywords: Rock drilling, overburden, misfire, fatal injury rate, explosives safety, risk analysis.

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1 INTRODUCTION

In bench blasting operations there has been an unexplored safety hazard concerning misfires from the overlying blast. In the literature misfire is defined as "the failure of an explosive charge to fire or explode properly when action has been taken to initiate it", ref. [6]. In this paper the remaining explosive parts are referred to as misfires. This hazard is mainly related to cartridge explosives and primers, and not to bulk explosives. Misfires can be located in the remaining rock debris from the prior sub drilled zone and can endanger the bench cleaning operation as well as the drilling operation.

The blasted rock debris, as used in this paper, is defined as the remaining loose material from the sub-drilled zone from the overlaying blast, after the rock pile has been excavated. See Figure 1. Drilling through this part is mainly relevant in relatively large quarries and open-pit mines.

The bench top cleaning operation is defined as the excavation of the blasted rock debris down to solid rock before the drilling operation starts. See Figure 5. Normal cleaning procedure with optimal cleaning capacity is assumed and ordinary watchful operators. This means that no precautions are taken due to possible misfires. Cautious cleaning, by slowly and carefully removing of the blasted rock debris, will be the safety procedure if a misfire most likely is present in a limited area. Anyway, this will be too expensive to perform continuously.

In the Norwegian quarrying industry, mainly based on top hammer drilling and 76 – 102 mm blasthole diameter, cleaned bench conditions has been the most common situation when drilling. The recent years it has been more common to drill through the rock debris, down to 89 mm blasthole diameter, in order to improve the business economics. Mainly by:

- Taking away the bench cleaning costs.
- Achieving higher total gross drilling capacity due to an even bench floor and easier maneuvering of the drillrig.
- Getting clear charging and coupling conditions, with faster charging time.

These advantages are meant to exceed the negative effects:

- Reduced net drilling capacity due to instability of the borehole and drill string jamming.
- Increased drill steel costs.
- Debris fallouts and blocking of the borehole before charging

The safety issues are described later in the article.

From experience, rock debris drilling with borehole diameters from 76 mm and smaller are ineffective and uneconomical.

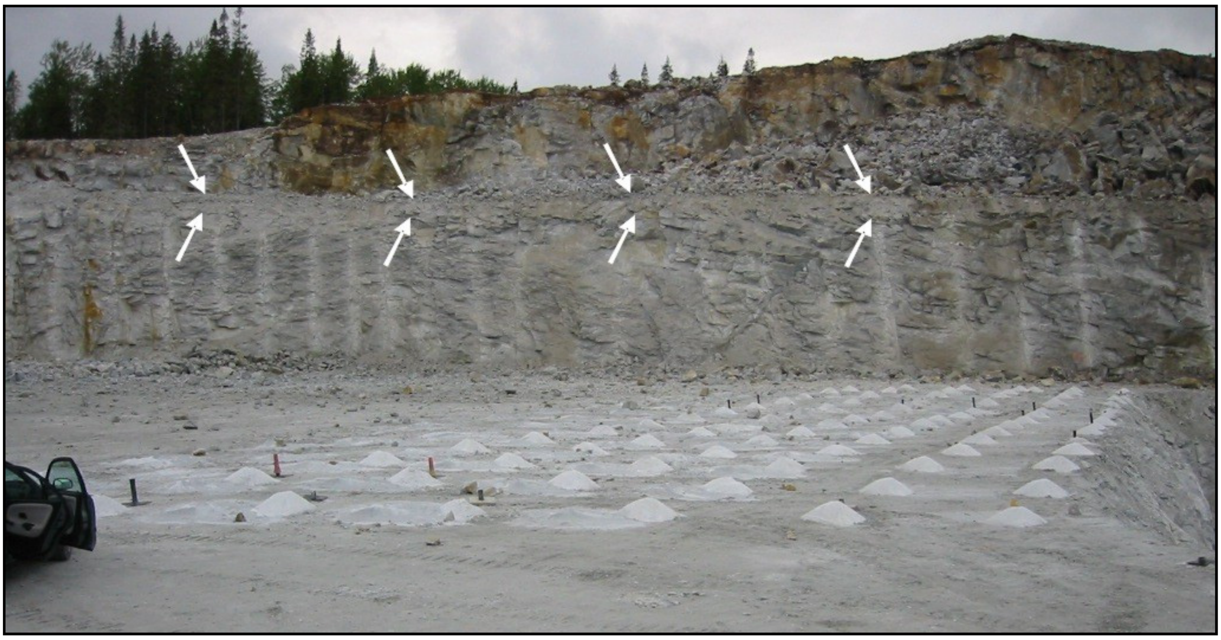

Figure 1 Blast ready for charging at Brønnøy Limestone Quarry, Norway. Holes drilled through the rock debris. Even bench floor conditions. The arrows point out the rock debris for the overlaying benching level. Photo: Vegard Olsen.

In 2004 the Norwegian University of Science and Technology (NTNU), the Norwegian Tunnelling Society (NFF) and parties of the construction, mining and quarrying industry initiated a project to describe and quantify the risk concerning the two operations and make it possible to estimate the probability of unintended detonation of undiscovered misfires in the rock debris and finally the risk for fatal injuries.

The background for the project was the law text, given in the "Norwegian Explosives Regulations" [1]: *The top of the bench must be justifiable cleaned, secured and controlled against misfires. New drillholes must not interfere with previous drillholes with the possibility of remaining explosives.*

Despite this clearly expressed regulation text, parts of the industry has openly and deliberately ignored it, in cooperation with their employees, as they think it is less hazardous than excavate the same rock debris.

The main goal of the project was to identify all possible causes leading to misfires, and to develop a risk analysis model to be able to find the probability of detonation of misfires by drilling or cleaning and the risk for fatal injuries. The paper is divided in two. First the basics of the statistical model and the possible causes and effects related to detonation of misfires are described. Secondly the model is used as a tool to estimate risk probabilities for rock debris drilling and bench top cleaning, and make an assessment of which is least risky.

2 RISK ANALYSIS MODEL

2.1 Definitions and Basic Statistics

General risk definitions are commonly used in risk assessment literature, e.g. as described in [3]. *Risk* is defined as the combination of the likelihood and the consequence, of a specified *hazard* being realized; see Equation (1). It is a measure of harm or loss associated with an activity.

$$\boldsymbol{Risk = Likelihood \times Consequence} \tag{1}$$

Likelihood is expressed as either a frequency or a probability. Frequency is a measure of the rate at which events occur over time. Probability is a measure of the rate of a possible event expressed as a fraction of the total number of events. *Consequence* is the direct effect of an event, incident or accident. It is expressed as a health effect, property loss, environmental effect, evacuation, or quantity spilled. *Hazard* is the inherent characteristic of a material, condition, or activity that has the potential to cause harm to people, property, or the environment. In the paper the likelihood of the current hazard is unintended detonation of a misfire, and the consequence measure is fatal injury.

The calculations in the risk analysis model are based on basic probability statistics [2]. The used statistic definitions are described below and illustrated in Figure 2.

An *event* is a subset of a sample space. The *union* of the two elements A and B, denoted by the symbol A ∪ B, is the event containing all the elements that belong to A or B or both. If A and B are any two events, then

$$\mathbf{P(A \cup B) = P(A) + P(B) - P(A \cap B)} \tag{2}$$

The *intersection* of two events A and B, denoted by the symbol A ∩ B, is the event containing all elements that are common to A and B. Two events are *disjoint* if A ∩ B = Ø. That is, if A and B have no elements in common. From Equation (2) this gives:

$$\mathbf{P(A \cup B) = P(A) + P(B)} \tag{3}$$

The *conditional probability* of an event B, given A, denoted by P(B | A), is the event when B occurs and the event A is known to have occurred. Mathematically this means:

$$\mathbf{P(B \mid A) = \frac{P(A \cap B)}{P(A)} \qquad if \qquad P(A) > 0} \tag{4}$$

Two events are *independent* if the event A does not affect the probability of event B, which gives:

$$\mathbf{P(A \cap B) = P(A) \times P(B)} \tag{5}$$

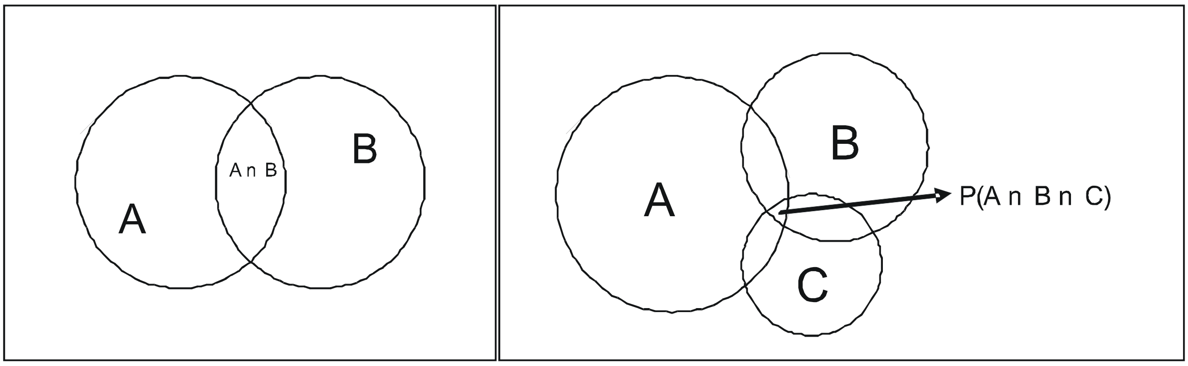

Figure 2 Venn diagrams illustrating probability spaces of two and three events.

2.2 Simulation Tool

The model uses different symbols to characterize the calculations carried out. With an eye to Figure 3 we see that four symbols are used.

- The quadrangle is the event of any probability calculation.
- The circle represents the input values/nodes given by the user of the model.
- The half circle (AND operator) means that the event E happens, only if the events e_1, e_2 and e_3 happen coincidentally.
- The crescent (OR operator) means that the event E happens if at least one of the events e_1, e_2 and e_3 happen, including two or more of the events happening coincidentally.

The AND operator has the following mathematical function, presuming e_1, e_2 and e_3 are independent events:

$$\mathbf{P(E) = P(e_1 \cap e_2 \cap e_3) = P(e_1) \times P(e_2) \times P(e_3)} \tag{6}$$

The OR operator has the following mathematical function, presuming e_1, e_2 and e_3 are independent:

$$\mathbf{P(E) = P(e_1 \cup e_2 \cup e_3) = P(e_1) + P(e_2) + P(e_3) - P(e_1) \times P(e_2) - P(e_1) \times P(e_3) - P(e_2) \times P(e_3) + 2 \times P(e_1) \times P(e_2) \times P(e_3)} \tag{7}$$

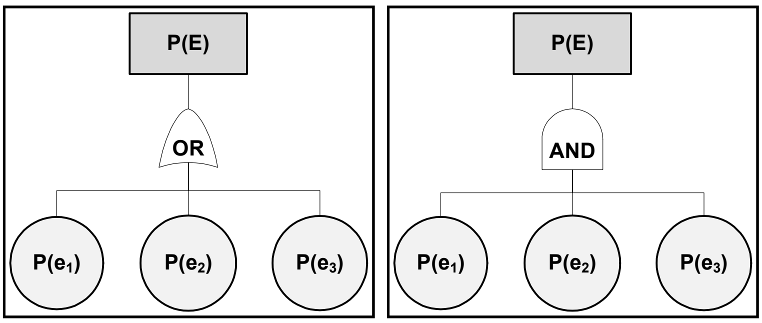

Figure 3 Symbols of the simulation tool.

2.3 Cause and Effect Analysis

In order to construct the risk analysis model, a cause and effect diagram (also called Ishikawa diagram [4]) was established to clarify all possible factors that can influence the occurrence of misfires and finally lead to an unintended detonation on the bench.

The diagram was primarily divided into three main events:

1. A misfire must be present
2. The misfire must be hit
3. The misfire must detonate when hit

Additionally, all possible causes for these incidents were identified and put into the cause and effect diagram. Well over fifty related single causes were considered. The diagram formed the basis for a risk tree, which is the basis of the risk analysis model. The fifty causes were reduced down to nine input nodes as independent probability values, see Figure 4.

The probability values of each of the nine nodes are assessed for each site or case and put in as basic data of the model. For example, different types of explosives will result in different probability values. However, for each blasting site the explosive type is given. The probability value must be assessed on the basis of what is used.

The most important basic data are:

- Drill plan; drilling pattern, borehole diameter and bench height
- Drilling method, drilling accessory equipment design and drilling machine settings
- Charging plan, explosives type and explosives dimensions
- Firing plan, detonator type and coupling routines
- Geology, rock mass jointing and water conditions
- Blasting control routines; measuring borehole deviation, measuring coordinates of borehole top and drill pattern displacement

2.4 Risk Analysis Estimation

There is a relatively large uncertainty in the estimation of the probability parameters. Therefore the input values are given as probability distributions. In the model, a triangle distribution is used. Other distributions are more likely to fit the reality, but the triangle distribution makes the model much easier.

The triangle distribution needs three probability values: Minimum, medium and maximum. In the model it is suggested to use the 10 % and 90 % percentiles for the minimum and maximum values. This means that it is likely that 90 % of the events have a higher value than the minimum value, and 90 % of the events have a lower value than the maximum value. The medium value is the mode value, the value believed to occur most times.

The software Definitive Scenario™ is used in designing the calculation model. Other statistical softwares may be used.

Monte Carlo simulation is used to perform the model calculations [5]. In the analysis, the probability values are randomly picked for the nine input nodes and the analysis calculations are carried out a certain number of times, if necessary up to one million repetitions.

The results from the Monte Carlo simulation present a probability distribution of all the calculations, and its statistical characteristics: Percentiles, average, standard deviation, minimum and maximum values. The distribution shows the possibility for detonation of a misfire, either for the bench cleaning operation or for drilling through the rock debris of the prior sub drilled zone.

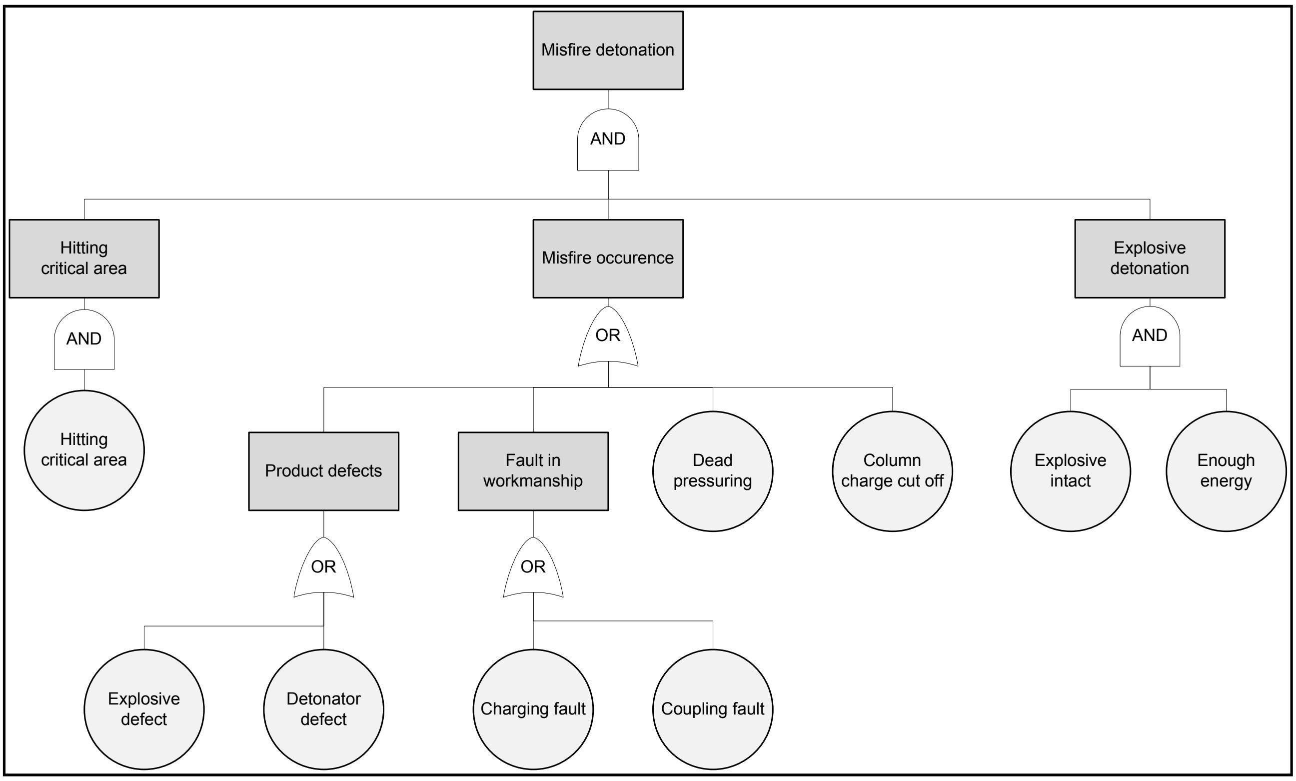

Figure 4 Risk tree used in the analysis model.

2.5 Basic Input Assessments

Misfire Occurrence

Misfires are mainly used in terms of cartridge explosives, or remains of these, and primers, even though non-detonated bulk explosives are embraced by the definition. The model includes all possible misfires. Basically there are two factors causing misfires:

- Malfunction within the initiation system or the initiators.
- The detonation stops or is being cut off in the charge column.

The probability of these two factors will vary between each quarry and will depend on different conditions:

- Geological conditions, e.g. degree of jointing, faults and water conditions.
- Charging and coupling work, e.g. initiation system and plan, bench conditions and the company's control routines.
- Product defects.

Higher degree of jointing and faults increase the possibility for dead pressuring of the explosives or a charge column cut-off in the neighboring holes. Groundwater in the rock mass will make this effect stronger. Also, jointed rock mass will increase the drilling deviation. This can result in poorer blasting conditions and more misfires. Too short distance between holes may lead to dead pressuring and cut-offs.

Electrical and electronic detonators can be controlled electronically before blasting, to test if they are intact. Non-electrical systems do not have this possibility, and will have a higher risk of misfire because of this. This is particularly important when it is necessary to use blasting mats on the top of the bench.

Different firing plans may influence the probability for misfiring by means of delay time and coupling succession. Too long delay time may result in dead pressuring, cut-offs or displacement of the rock mass, and may disturb neighboring holes. In large blasts, good coupling succession will reduce long delay times. Two detonators in each hole will also reduce the risk of misfires, and with coupling backups from the second detonator, even better detonation probability will be ensured. When blasting several rows, coupling backwards from each hole in the first row will be positive in relation to misfire occurrence.

In rough terrain or in very uneven bench conditions, the coupling operator's general view may be weakened. This can lead to poorer quality of the manual work and more misfires. Every company should have control routines that will capture such errors.

Product defects can origin from the producer or from wrong use or treatment from the consumer.

Mathematically the misfire occurrence is dependent of 4 independent events, see Figure 4:

- Product defects
- Fault in workmanship
- Dead pressuring
- Cut-off column charge

Only one of these four events needs to happen for a misfire to occur. The events are connected to the OR operator, as described in Equation (7).

The product defects probability is further related to two independent events: Defect explosive (primer) and defect detonator. Only one of these needs to happen to get a product defect causing misfire. These events are also connected with the OR operator.

The same scenario is applied for the fault in workmanship probability. Fault may occur during charging of the hole or during the coupling work. If one of the events happens the result will be a misfire, and the probability values are connected with the OR operator.

Figure 5 Bench cleaning operation at Visnes Limestone Quarry, Norway. The rock debris from the overlaying blast is removed before drilling. Photo: Vegard Olsen.

<u>The Misfire Gets Hit by Drilling or Bench Cleaning</u>

The probability for hitting a misfire depends primarily on:

- Primer/cartridge dimensions
- Drilling pattern
- Drill bit diameter
- Deviation of overlying boreholes
- Length of sub drilling

For the drilling alternative the parameters indicate how large the possible hit area is, compared to the drilling pattern. Larger cartridges and longer sub drilling give a higher probability of a hit. Misfires in the rock debris are often found in the bottom of the previous boreholes, and the deviation of this hole will affect the hitting area. The more deviation, the more exposed the borehole bottom will be, when drilling the next bench. When cleaning the bench, the misfire's orientation is assumed not to affect the probability.

The drill bit diameter directly affects the specific drilling or drilling pattern area. Larger drill bits give larger drilling pattern. The increase in drilling pattern area is less than the increased drill bit cross section area, and the probability of hitting a misfire will increase for each drilled hole. However, in total the probability will decrease as fewer boreholes are needed to produce the same amount of rock.

In addition, larger drill bits will give straighter holes, and larger drilling pattern will also simplify a displacement relative to the overlying blast. The latter requires a system for measuring the coordinates of the top of each borehole.

Looking at the bench cleaning operation the parameter is based on the assessments of how much of the rock debris will be cleaned. The probability will be close to 1.

To estimate the probability of hitting the misfire, the basic data of the current site are used, see points above. The drilling estimation is exact and performed out-side the model, giving only one input value: Hitting critical area.

The Misfire Detonates

Two main factors decide whether a misfire will detonate or not, when it is hit. These are:

- Sensitivity of the explosive
- The energy transferred to the misfire

Type of explosive and time since the overlying bench was blasted will affect the condition and the sensitivity of the misfire. Bulk explosives will be insensitive in relatively short time, whilst cartridge or primer explosives can be sensitive even after several decades, if the wrapping still is intact.

The energy from both an excavator bucket and a drill bit is sufficient to detonate a misfire [6]. There is a greater chance for detonation, if the misfire is hit by the drill bit than if hit by the bucket. If the misfire is touched by the bucket, it is possible that it will only be disturbed or squeezed without detonation.

When cleaning the bench, it is possible to detect remaining shock tubes or bulk explosives which can indicate misfires. A misfire detection degree factor is used in the model to provide for this.

The probability of misfire detonation is divided into the sum of the probability of detonation of the column charge explosive and the probability of detonation of the primer item. The probability of the latter includes both primer alone and primer with detonator inside. A single detonator will not contain the necessary energy to cause any significant injury. Both events will individually lead to a misfire, and the OR operator connects the two events in the model, ref. Equation (7).

Furthermore, these probabilities are based on two independent input probability values each. The explosive item must be intact and the energy level of the drill bit or the excavator bucket must be high enough to trigger the detonation. Both must happen coincidentally and they are combined with the AND operator, ref. Equation (6).

Result of Basic Model

The result of the model appears by multiplication (AND operator) of the three main probabilities as described above: Misfire is present, misfire is hit and misfire detonates, as all three events must happen to start an unintended incident.

If two different explosives may be in the critical area, e.g. primer and bulk, two separate calculations must be done. Equation (7) is used to find the total probability of e.g. "Misfire detonation; primer" and "Misfire detonation; bulk", see Figure 6.

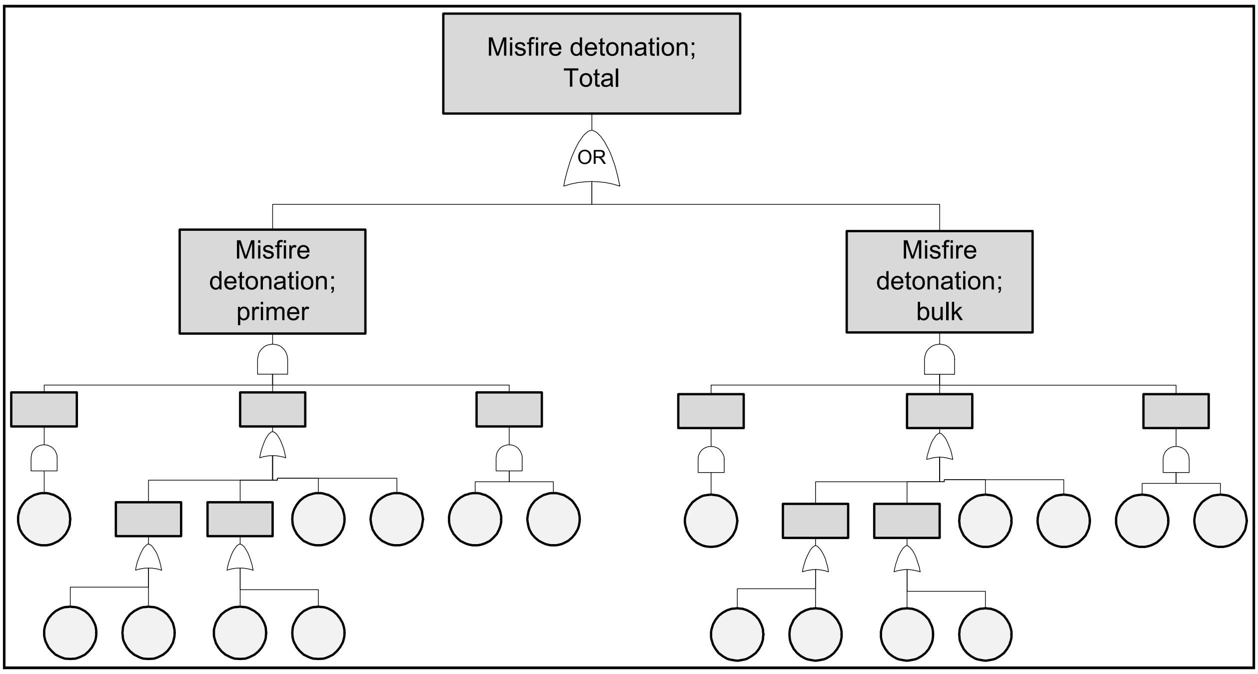

Figure 6 Extended risk tree combining two different explosives types.

3 PRACTICAL APPROACH TO THE MODEL

The basic model describes the probability of a misfire to detonate, however the consequences are not assessed and this is added in the practical part. In general risk assessment analysis fatal injury is the common consequence measure, and this is also used in this paper. This makes it possible to compare the estimation results, not only with other quarrying and mining operations, but also with occupational groups and in general. Alternatively, ranking the severity of injuries is difficult if not impossible and the fatal injury is used as a definite limit.

The model is very detailed concerning some input parameters, and the inputs can be difficult to define. If so, the input can be placed higher in the structure. This is particularly applicable for the occurrence of misfires and the corresponding causes. In practice, the number of occurred misfires per year can be used as the mean value, embracing all input values in the misfire occurrence probability structure. It is important that all detected misfires are recorded. If not the calculations will be based on too low probability values. If no records are available, statistics from the industry can be used, e.g.:

- Public incident records
- Producer complaints records
- Producer quality requirements

3.1 Risk Analysis Methods

There are several ways of analyzing risks. The formulations are different from one country to another, but in fact they are different facets of the same idea. In the following three of the most common are considered:

- The French GAMAB, Globalement Au Moins Aussi Bon (totally at least as good)
- The English ALARP, As Low As Reasonably Practicable
- The German MEM, Minimal Endogeneous Mortality

Below the methods are described briefly, but they are all described thoroughly in the EN 50126-1:1999 [13].

GAMAB assumes an already acceptable system (in this case cleaning the bench), and the new system (drilling through the rock debris) shall in total be at least as good as the already acceptable system.

ALARP assumes an already acceptable and known risk for the society, and the new system shall not increase this risk. GAMAB and ALARP are very much alike.

MEM is based on the fact that the death risk varies for the different age groups in the society and the technological systems all individuals are exposed to (transportation, sports etc.) A new technological system shall not significantly augment the general death risk of the age group of 5 to 15 years, which has the lowest mortality rate in well-developed countries, set to $2 \cdot 10^{-4}$ fatalities/person year. The reference level of MEM is interesting in collaboration to actual fatal injury rates in the industry.

3.2 General Fatal Injury Data

Statistical data concerning fatal injuries are reviewed from the following sources:

- Statistics Norway [15].
- The Norwegian Labour Inspection Authority [16].
- The Directorate for Civil Protection and Emergency Planning (DSB) in Norway [17].
- The Swedish Work Environment Authority (AV) [22].

All sources are public, though DSB's and AV's data are available only on special request. The most important results of the survey are referred below:

- General fatal injury rate in Norway for 0-14 year olds from 2000 to 2004 is $3.8 \cdot 10^{-5}$ [15]. Considerably lower than MEM, $2 \cdot 10^{-4}$ [13].
- General fatal injury rate at work in Norway from 2000 to 2005 is $2 \cdot 10^{-5}$. (Including Road Traffic and Transport Accidents (RTTA)) The rate increases for the mining, building and construction industries to $5 \cdot 10^{-5}$. Narrowing the occupations even more, looking only at blasters and miners, road and heavy construction workers and construction machinery operators, the fatal injury rate increases to $18 \cdot 10^{-5}$ [15].
- In the period from 1991 to 2006 there were 17 fatal injuries due to detonation of explosives in Norway [15]. In average one per year for an annual production of 50 million tons in quarries and mines and 3 million tons tunneling and rock caverns [9], [23], [24] and [25]. (This includes third persons and non-blasting occupations. Reference to number of employees is not applicable.)
- In the period from 1991 to 2006 there were 4 fatal injuries due to detonation of explosives in Sweden. In average 80 million tons per year (fifty-fifty mines and quarries) [26], [27]. No exact data on tunnels and rock caverns, although it is less than the Norwegian production.

3.3 Practical Input Assessments

In the following the background data and the mathematical expressions of the risk assessment parameters are presented. As described above the main assessment issues are:

- Critical area
- Misfire occurrence rate
- Misfire detonation probability
- Fatal accident probability

Hitting Critical Area

The probability of hitting the critical area is expressed by Equation (8), which gives the ratio between the horizontally exposed area of a possible misfire plus the drill bit cross section area and the overlaying drilling pattern.

$$\boldsymbol{P(CA_d)} = \frac{A_m + A_b}{A_d} = \frac{A_{cm} \cdot \sin\alpha + A_{sm} \cdot \cos\alpha + A_b}{A_d} \qquad \textbf{where} \qquad \textbf{(8)}$$

$P(CA_d)$ = probability of hitting critical area when drilling
A_m = misfires horizontal exposure area
A_b = current drill bit cross section area
A_d = previous overlaying blast drilling pattern
A_{cm} = misfire cross section area
A_{sm} = misfire long side area
α = misfire angle in the rock debris (90° is vertical)

The on-going drilling pattern will not influence the *$P(CA_d)$*. It will be included in calculations according to the numbers of holes per year. Larger drilling pattern gives fewer holes per year for the same annual production.

In Table 17 the probability value $P(CA_d)$ for some common parameter combinations are shown. The blast design calculations are based on the NTNU Rock Quarrying blast design model [28] with normal parameters and rock conditions.

Alternatives	Drilling pattern overlaying blast	Current drill bit diameter	Critical area input value
Example 1	5.7 m^2 [76 mm]	89 mm	$3.0 \cdot 10^{-3}$
Example 2	8.0 m^2 [89 mm]	89 mm	$2.6 \cdot 10^{-3}$
Example 3	8.0 m^2 [89 mm]	165 mm	$4.5 \cdot 10^{-3}$
Example 4	23.5 m^2 [165 mm]	165 mm	$1.3 \cdot 10^{-3}$
Example 5	23.5 m^2 [165 mm]	89 mm	$0.7 \cdot 10^{-3}$

Table 17 Probability value for hitting primer area with common combinations of drilling pattern and drill bit size. Calculations based on blast design model NTNU [28].

In the bench cleaning operation the whole bench top area is affected/stirred by the excavator bucket, and the probability value $P(A_{bc})$ will be around 1.0. The misfire can be stirred or it can slip out of the excavator bucket without detonation. The misfire can also be detected during the cleaning operation. This is included in the misfire detonation probability assessments described a little later.

Misfire Occurrence Rate

The misfire occurrence rate is dependant of 6 independent causes, as described in Section 2.5 and shown in Figure 4. A mentioned above it is hard to distinguish the causes and find the probability of each input value, and the misfire occurrence rate is decided by the number of misfires found over a particular period of time.

Studies and interviews on the subject have revealed some statistical "facts".

- Interview with 9 quarry owners, totally 25 quarries and total annual production approximately15 million tons. In average 1 misfire reported per 2 million tons (2000-2004) [9].
- Unintended misfire detonations reported to DSB from 1991-2004 count 21 incidents [9]. In average 1.5 incidents per year and annual production of about 40 to 50 million tons.
- Single quarry with annual production 1.3 million tons had 30 misfires in a two year period originating from 10 blasts (2005-2006). Possible inaccurate coupling procedures. Safety actions introduced, no new misfires reported afterwards [29].
- Single quarry with annual production 4.5 million tons had 4 primers originating from 2 contour blasts (2006). One single blast with twelve undetonated holes (2005). Beside this no reports of misfires from production blasts the last 5 years (2002-2006) [30].
- Single quarry with annual production 1.5 million tons have in some occasions found column cartridges from contour blasts in the muckpile. No primers found the last 8 years (1999-2006) [9].
- Detonator manufacturer's demands are 1 error of 10 000 detonators produced. Actual statistics 1 error of 100 000 [6].
- Botniabanan railway project in Sweden experienced 17 misfire detonations in surface operations in five years and production of 4 million m^3 rock [31].
- Two other large Swedish projects have experienced misfires, but not the in such extent as described at Botiabanan [32].

To a certain point the augment of the numbers of misfires will be discovered as very poor blasting, high rate of misfire findings and more reported incidents of detonation of misfires. Risk reduction actions will be taken at a relatively early point to detect the reasons and reduce the problem.

Mathematically the misfire occurrence rate will be expressed by the following formula:

$$\boldsymbol{P(MO) = \frac{n_m}{n_{dh}}} \quad \textbf{where} \qquad \textbf{(9)}$$

P(MO) = *probability of misfire occurrence*
n_m = *number of misfires*
n_{dh} = *number of drilled holes*

Figure 7 Left: Misfires in the muckpile.
Right: Bottom of hole made visible during bench cleaning. Remaining Nonel shock tube indicating possible misfire. Photos: BEF [33].

<u>Misfire Detonation Probability</u>

As mentioned before, both the drilling and the cleaning operation hold enough percussion energy to detonate misfires. This is also verified by accident and incident reports due to drilling, cleaning and loading operations in surface and underground workings.

Surface related misfire incidents reported to DSB 1991 -2004, [6], [9]and [17].

- Drilling 6 incidents
- Loading 5 incidents
- Bench cleaning 1 incident
- Secondary breaking, scaling 5 incidents

Reported incidents at Botniabanan railway construction in Sweden 2000-2005, [31]. Surface operations. In total 4 million m^3.

- Drilling 1 incident
- Loading 10 incidents
- Secondary breaking 2 incidents
- Crushing 3 incidents

The reported incidents do not necessarily represent all incidents. The numbers have to be used with caution statistically. A comparison of incidents mentioned in the media in Norway and reports submitted to DSB, shows that only 30 % of the incidents are reported [20]. The documentation degree will be higher as the severity of the accidents increases and one can assume that virtually all fatal and heavy injuries are reported. The Botniabanan reports are likely to have a high degree of reporting as the project management focused on the issue.

Direct hit of a misfires without detonation is not very likely. No actual reports of this are found, and it will be natural to assume a probability value around 1, at least for the drilling operation. For the cleaning operation the probability value is likely to be lower than this, as the misfire can be stirred or it can slip out of the excavator bucket without detonation. To set the correct probability value for the cleaning operation is very important in the analysis, as it is directly competitive with the probability of the drilling operation probability.

Probability assessment is in general problematic as incident rates are low or maybe never happened. It may be necessary to use other similar operations with more frequent incidents to get good numbers. The data listed above shows only one detonation by bench cleaning from 1991 to 2004. Hardly any bench cleaning is performed in the Botniabanan project as most of the volume was cut blasting, i.e. one bench and sub drilling remains below loading level. To evaluate the cleaning operation it will be obvious to use the loading operation as a comparison. The operation and the machinery are the same and no big source of error will be made.

The data above shows detonations due to drilling, but none of these are related to drilling through the rock debris. The drilling incidents are related to drifting, trench blasting and drilling in relation to bench toe problems.

To evaluate the data it is important to have in mind the mathematical description of the misfire detonation probability used in the model: Number of detonations per hit misfire. See Equation (10).

According to this we can use the Botniabanan project data, which is relatively well defined according to misfire recording and produced volume, and make a train of thought to figure out the number of detonations per hit:

"Close to all cartridge misfires will detonate in the crusher. This gives 6 out of 16 (10+3+2+1) misfires did not detonate under the loading operation. The detonation due to drilling and secondary breaking is added in the calculations in favor of the bench top cleaning operation.

By these numbers 10 out of 16 misfires (63%) will detonate if caught or stirred by the bucket. Supposing that only half of the misfires have detonated in the crusher, which is highly unlikely, the detonation degree will be 10 out of 19 (53%), and this will not affect the total risk calculations much anyway.

There is very little documentation of misfires found during the bench cleaning or loading. In favor of the bench cleaning process it is assumed that these incidents have a low reporting degree, as no accident has occurred. A reasonable assumption is that 90 % of all misfires are found in the loading or bench cleaning operation or will remain undiscovered misfires in the drillhole bottom, covered by the rock. The rest detonates.

Looking at the Botniabanan project this gives in total 160 misfires in total. Substantial augment in the assumed number of detected misfires would have led to extensive safety assessment actions earlier in the project and probably reduced the amount afterwards. 160 misfires gives approximately 1 misfire per 25 000 m^3".

Comparing the Botniabanan train of thought with the data listed earlier, the assumed misfire occurrence rate is high. The worst case gives 1 misfire per 32 000 m^3 for 89 mm drillholes for production blasting in a quarry. Assuming 76 mm or 89 mm drillholes at Botniabanan these numbers are quite similar. In both examples such amounts of misfires have initiated caution efforts and focus on the matter, and due to the other information available, the level derived above is likely to be conservative in favor of the bench cleaning operation.

$$\boldsymbol{P(MD) = \frac{n_{md}}{n_{mh}}(1 - f)} \quad \textbf{where} \tag{10}$$

P(MD) = *probability of misfire detonation*
n_{md} = *number of misfire detonations*
n_{mh} = *number of misfire hits*
f = *detection degree factor*
$n_{mh}/(1-f) = n_m$ = *total number of misfires*

The detection degree factor describes the possibility of discovering the misfire before a hit occurs. For drilling through the rock debris f will be close to zero. The assumptions in the Botniabanan project give a detection degree factor of 0.9 for the bench cleaning operation.

<u>Fatal Injury</u>

The cause of death due to detonation of misfires is mainly related to flyrocks or other fragments hitting exposed persons. The input probability value is related to how many fatal injuries that will appear if the misfire detonates when hit by drilling or bench cleaning. In general the value will be dependant of the barrier between the energy source (the misfire) and the exposed person (the operator) [34]. Also the concentration and direction of the fragments will influence the value. The barriers will vary from site to site according to each site's machinery and working conditions. In the scope of this work typical barriers will be:

- Operator cabin (glass and steel bars on drilling rig and excavator)
- Machine (e.g. standing behind machine when drilling without cabin)
- Rock debris layer thickness
- Distance and angle to detonation spot (remote drilling)
- Protective equipment (helmet, goggles, clothing)

The calculations are focused on the differences between the rock debris drilling and the bench cleaning operation, rather than the safety level. Safety level and safety reduction actions are discussed shortly later.

As mentioned before, the official incident data bases in Norway and Sweden do not contain reports of incidents with detonation of explosives due to rock debris drilling in quarry production or open-pit mining. The incidents reported are related to tunnels, construction sites and secondary drilling of large blocks and bench toes and contains four fatal injuries.

According to bench cleaning and loading operations several incidents of detonation are reported, but no fatal accidents. By these facts one can assume that the bench cleaning or loading operations have a lower risk of fatal injuries than the drilling operation including rock debris drilling. This however disregards the rock mass and the rock debris as a barrier and an energy absorber, which will reduce the lethality of the misfire detonation. This only to a certain point as the rock debris depth generally will be smaller due to the misfire. It is assumed that the drilling operation will obtain some of the positive barrier effect from the rock mass, which is clearly present for the cleaning and loading operations.

Another issue is whether the excavator bucket will concentrate the detonation energy more directly against the operator cabin than a drilling detonation will do. No documentation on this is found and the analysis does not differentiate this issue.

Today, both the excavator operator and the drill rig operator are place in a cabin, and it is assumed that they will have the same barrier conditions. However, the distance from the excavator bucket to the cabin will in most cases be larger than from the drilling point to the drill rig cabin. The overall assessment will be that the cleaning operation will have a lower fatal injury probability if a misfire detonates.

Though the fatal injury probability level for the operations is difficult to set, the incident surveys above give grounds for some assumptions.

In the Botniabanan project ten detonations during loading occurred without any human injuries. One human injury was reported from secondary breaking operation. In total this gives one injury out of 13 incidents. The data does not contain direct information about what loader that was used, though the rhetoric implies mainly excavator machines. Wheel loader loading will hardly cause any risk upon the operator. Assuming excavator loading is conservative in favor of the bench cleaning operation, as this gives less fatal injury probability.

The DSB data shows 6 misfire detonations due to loading and bench cleaning over a period of 14 years. As mentioned before, this is probably less than the reality. The fact that no fatal injuries has occurred is 100 % true. A doubling of detonations in reality gives the same numbers as the Botniabanan data.

Both sources could have experienced "lucky" circumstances, but according to the information available it is reasonable and conservative to think that 1 out of 10 misfire detonations will lead to a fatal injury for drilling and 1 out of 100 for bench cleaning. This is assumed with no particular safety precautions made.

Mathematically the fatal injury rate is described like this:

$$\boldsymbol{P(FI)} = \frac{\boldsymbol{n_{fa}}}{\boldsymbol{n_{md}}} \qquad \textbf{where} \qquad \textbf{(11)}$$

P(FI) = *fatal injury rate due to misfire detonation*
n_{fa} = *number of fatal accidents*
n_{md} = *number of misfire detonations*

Total Risk of Fatal Accident by Misfire Detonation

The total risk for a fatal accident to happen due to misfire detonation for one drilled hole, or one corresponding area cleaned bench, is calculated as the product of the four independent probability events described above.

$$\boldsymbol{P(FA_{d1}) = P(CA_d) \times P(MO) \times P(MD) \times P(FI)} \quad \textbf{and} \qquad \textbf{(12)}$$

$$\boldsymbol{P(FA_{bc1}) = P(A_{bc}) \times P(MO) \times P(MD) \times P(FI)} \quad \textbf{where} \qquad \textbf{(13)}$$

$P(FA_{d1})$ = Probability of fatal accident for one single hole when drilling
$P(FA_{bc1})$ = Probability of fatal accident for one single hole equivalent when cleaning the bench

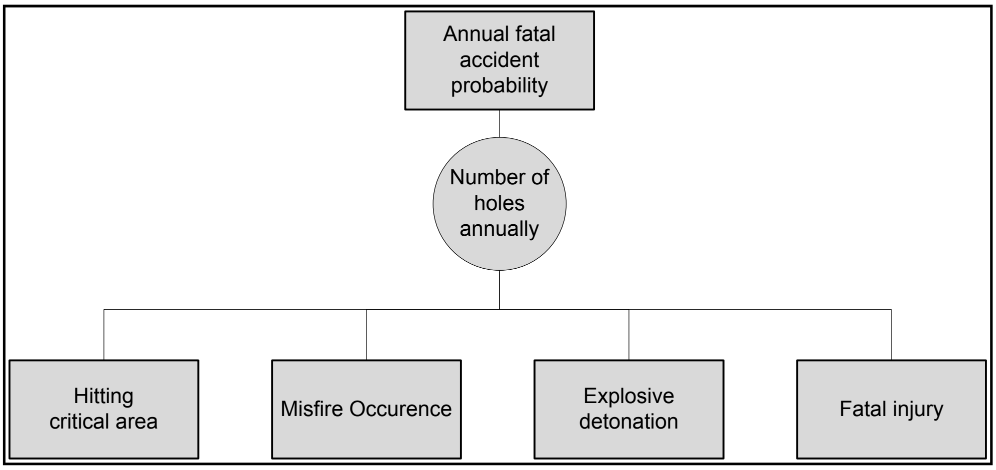

Figure 8 General view of calculation model for annual fatal accident probability.

To be able to compare risks across working operations and other industries it is necessary to evaluate it over a certain period of time, normally per year. The total number of holes drilled and higher annual production will affect the total fatal injury risk over a year.

The fatal injury risk probability includes the one-single incidents plus all the multiple incidents variations, and the calculation is complicated. By turning the equation over and looking at the probability for no accident the calculation is much easier [2]. Equations (14) and (15) describe the turn-over:

$$\boldsymbol{P(NFA_1) = 1 - P(FA_1)} \quad \textbf{where} \qquad \textbf{(14)}$$

$P(NFA_1)$ = Probability of no fatal accident for one single hole
P(FA1) = Probability of fatal accident for one single hole

and

$$\boldsymbol{P(FA) = 1 - P(NFA_1)^{n_{dh}}} \quad \textbf{where} \qquad \textbf{(15)}$$

P(FA) = Annual fatal accident probability
n_{dh} = number of drilled holes

3.4 Example of Risk Analysis Calculations

In the following use of the risk analysis model is exemplified and input values are assessed according to size, quantity and uncertainty. The model is using Monte Carlo simulation and input triangular probability distribution as described in Section 2.4. General assumptions to the risk analysis calculations are:

- It is not considered how the drilling and cleaning operation affects the occurrence of misfires after blasting. The model calculates risk for the situation before blasting.
- General fatal risk of operating the drilling rig and the excavator is not applied in the model.
- The fact that the cleaning operation involves an extra operation and in total more man-hours and a higher general fatal risk probability, compared to drilling through the rock debris, is not included in the model.
- The model does not take into account poorer maneuvering conditions, and e.g. increased risk for tip over, for the drilling rig when the bench is cleaned.
- When cleaning the bench one has to drill afterwards. Undiscovered misfires will still be a hazard to the drilling operator. The extra risk is ignored.
- Production blast with bulk explosives, two primers (bottom/top) and Nonel ignition system.
- No sort of documentation of previous overlaying blasts holes coordinates.
- It is assumed that the operations are carried out with one operator and no other operations or personnel in immediate proximity.
- Annual production of 1 million tons, drillhole diameter 89 mm and 3000 holes drilled per year.

The assumptions in the calculations are tried to be as objective as possible, and derived from the collected facts. When necessary, e.g. with high uncertainty assumptions, they are tried to be conservative in favor of the bench cleaning operation, as this operation is the approved operation according to the regulation text [1].

Hitting Critical Area Assessment

The probability value for hitting critical area is different for the two operations. For both, this value is relatively well determined, though the cleaning values are more analytic than the drilling values.

Operation	Minimum	Medium	Maximum
Drilling through the blasted rock debris	$5 \cdot 10^{-4}$	$2.5 \cdot 10^{-3}$	$5 \cdot 10^{-3}$
Cleaning the bench	0.8	0.9	1.0

Table 18 Probability values for hitting critical area. The drilling values are more or less based on Table 17.

Drilling:

Medium value: Medium rock conditions, overlaying and current drilling pattern due to 89 mm drillhole diameter.

Minimum value: Good rock conditions, overlaying and current drilling pattern respectively 165 mm and 89 mm drillhole diameter.

Maximum value: Poor rock conditions, overlaying and current drilling pattern respectively 89 mm and 165 mm drillhole diameter.

Cleaning:

Medium value: 90 % of all overlaying drillholes will be cleared.

Minimum value: Poor cleaning and 80 % of the drillhole bottoms revealed. Lower value will make the probability for hitting by drilling significant.

Maximum value: Natural to use 100%.

Misfire Occurrence Rate Assessment

The misfire occurrence rate will be the same for both drilling and cleaning.

Operation	Minimum	Medium	Maximum
Drilling and cleaning	$2.2 \cdot 10^{-5}$	$2.2 \cdot 10^{-4}$	$1.1 \cdot 10^{-3}$

Table 19 Probability values for misfire occurrence rate.

Medium rock conditions and overlaying drilling pattern 89 mm.

Medium value: 1 misfire per 1.5 million tons.

Minimum value: 1 misfire per 15 million tons.

Maximum value: 5 misfires per 1.5 million tons.

Explosive Detonation Rate Assessment

The probability value for detonation of explosives, when hit, is different for the two operations. The drilling probability value is well determined. The cleaning value is a bit more difficult to set. Sober assumptions in favor of the cleaning operation are made.

Operation	Minimum	Medium	Maximum
Drilling	0.95	0.99	1.0
Cleaning	0.001	0.05	0.1

Table 20 Probability values for misfire detonation rate, when hit. Values are based on the discussions in Section 0.

Drilling:

Medium value: 99 out of 100 misfires will detonate if hit during drilling.

Minimum value: Old, exposed and less sensitive explosives. The explosive may slip or evacuate from the drill bit. 95 out of 100 are assumed to detonate if hit.

Maximum value: Natural to use 100%.

Cleaning:

Medium value: 10 out of 20 misfires will detonate if stirred during cleaning of the bench. 9 out of 10 will be discovered by the excavator operator or remain hidden in the sub-drilled zone.

Minimum value: Old, exposed and less sensitive explosives (reduced 5 times). Extra attentive operator (99 out of 100 misfires detected).

Maximum value: Doubling the medium value.

Fatal Injury Rate Assessment

The fatal injury rate due to misfire detonation is hard to define, as no direct fatal injuries has occurred within rock debris drilling or bench cleaning. Therefore similar operations are used to assess the fatal injury rate. The probability value for fatal injury is not differentiated between the two operations.

Operation	Minimum	Medium	Maximum
Drilling	0.01	0.1	0.9
Cleaning	0.001	0.01	0.09

Table 21 Probability values for fatal injury rate. Values are based on the assessments in Section 0.

Drilling:

Medium value: 1 out of 10 misfire detonations will cause a fatal injury.

Minimum value: 1 out of 100 misfire detonations will cause a fatal injury.

Maximum value: 9 out of 10 misfire detonations will cause a fatal injury.

Cleaning:

Medium value: 1 out of 100 misfire detonations will cause a fatal injury.

Minimum value: 1 out of 1000 misfire detonations will cause a fatal injury.

Maximum value: 9 out of 100 misfire detonations will cause a fatal injury.

Monte Carlo Simulation Results

The Monte Carlo simulation is based on 1 million repetitions. Table 22, Table 23 and Figure 9 show the simulation results of all probability inputs and the estimated annual fatal accident probabilities for the two analyzed operations.

	Base	2.5%	10.0%	50.0%	90.0%	97.5%	Mean	Std.dev	Min.	Max.
Annual fatal accident probability	$3.0e^{-4}$	$1.3e^{-4}$	$3.1e^{-4}$	$1.4e^{-3}$	$4.7e^{-3}$	$7.7e^{-3}$	$2.1e^{-3}$	$2.1e^{-3}$	$1.3e^{-6}$	$2.3e^{-2}$
Fatal accident probability for one single hole equivalent area	$9.9e^{-8}$	$4.3e^{-8}$	$1.0e^{-7}$	$4.6e^{-7}$	$1.6e^{-6}$	$2.6e^{-6}$	$6.8e^{-7}$	$6.9e^{-7}$	$4.2e^{-1}0$	$7.7e^{-6}$
Hitting critical area	$9.0e^{-1}$	$8.2e^{-1}$	$8.5e^{-1}$	$9.0e^{-1}$	$9.6e^{-1}$	$9.8e^{-1}$	$9.0e^{-1}$	$4.1e^{-2}$	$8.0e^{-1}$	$1.0e^{+0}$
Misfire occurrence	$2.2e^{-4}$	$9.5e^{-5}$	$1.7e^{-4}$	$4.1e^{-4}$	$7.9e^{-4}$	$9.5e^{-4}$	$4.5e^{-4}$	$2.3e^{-4}$	$2.3e^{-5}$	$1.1e^{-3}$
Explosive detonation	$5.0e^{-2}$	$1.2e^{-2}$	$2.3e^{-2}$	$5.0e^{-2}$	$7.8e^{-2}$	$8.9e^{-2}$	$5.0e^{-2}$	$2.0e^{-2}$	$1.0e^{-3}$	$1.0e^{-2}$
Fatal injury	$1.0e^{-2}$	$5.5e^{-3}$	$1.0e^{-2}$	$3.0e^{-2}$	$6.3e^{-2}$	$7.7e^{-2}$	$3.4e^{-2}$	$2.0e^{-2}$	$1.0e^{-3}$	$9.0e^{-2}$

Table 22 Risk analysis model probability results for bench top cleaning.

	Base	2.5%	10.0%	50.0%	90.0%	97.5%	Mean	Std.dev	Min.	Max.
Annual fatal accident probability	$1.6e^{-4}$	$9.0e^{-5}$	$2.0e^{-4}$	$8.2e^{-4}$	$2.7e^{-3}$	$4.3e^{-3}$	$1.2e^{-3}$	$1.1e^{-3}$	$1.6e^{-6}$	$1.2e^{-2}$
Fatal injury probability for one single hole	$5.5e^{-8}$	$2.8e^{-8}$	$6.6e^{-8}$	$2.7e^{-7}$	$8.9e^{-7}$	$1.5e^{-6}$	$3.9e^{-7}$	$3.8e^{-7}$	$5.2e^{-1}0$	$3.9e^{-6}$
Hitting critical area	$2.5e^{-3}$	$9.8e^{-4}$	$1.5e^{-3}$	$2.6e^{-3}$	$3.9e^{-3}$	$4.5e^{-3}$	$2.7e^{-3}$	$9.2e^{-4}$	$5.0e^{-4}$	$5.0e^{-3}$
Misfire occurrence	$2.2e^{-4}$	$9.5e^{-5}$	$1.7e^{-4}$	$4.1e^{-4}$	$7.9e^{-4}$	$9.5e^{-4}$	$4.5e^{-4}$	$2.3e^{-4}$	$2.3e^{-5}$	$1.1e^{-3}$
Explosive detonation	$9.9e^{-1}$	$9.6e^{-1}$	$9.6e^{-1}$	$9.8e^{-1}$	$9.9e^{-1}$	$1.0e^{+0}$	$9.8e^{-1}$	$1.1e^{-2}$	$9.5e^{-1}$	$1.0e^{+0}$
Fatal injury	$1.0e^{-1}$	$5.5e^{-2}$	$1.0e^{-2}$	$3.0e^{-1}$	$6.3e^{-1}$	$7.7e^{-1}$	$3.4e^{-1}$	$2.0e^{-1}$	$1.0e^{-2}$	$9.0e^{-1}$

Table 23 Risk analysis model probability results for rock debris drilling.

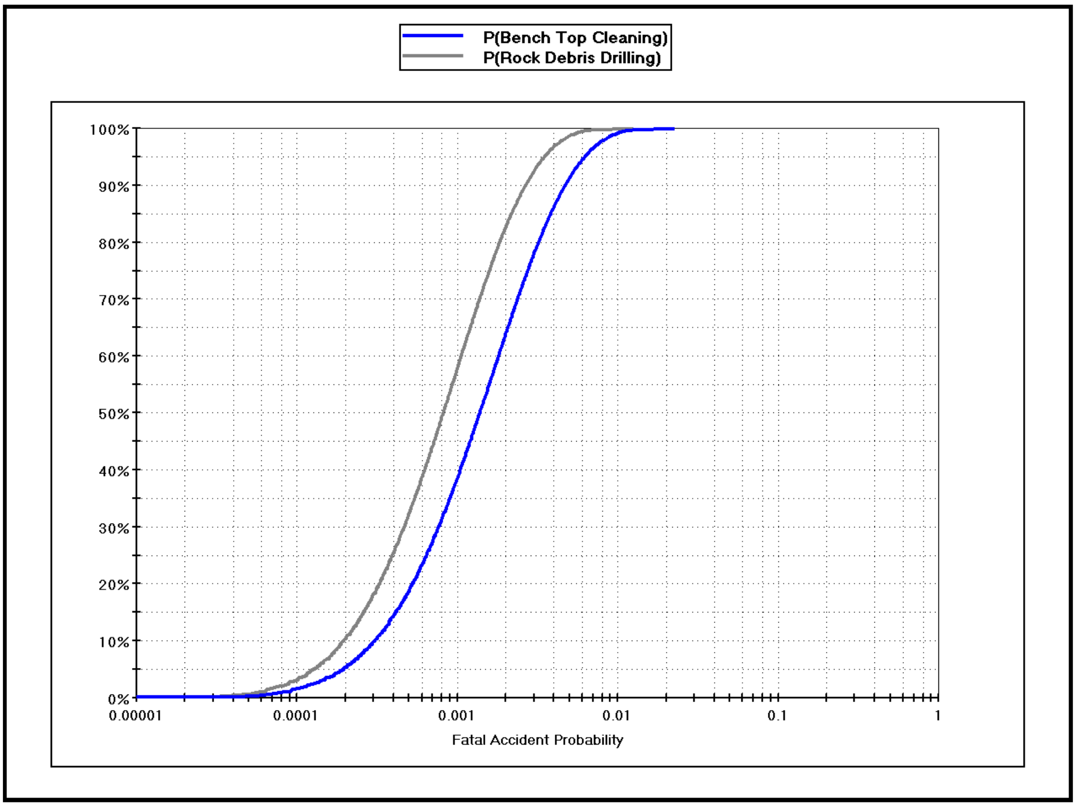

Figure 9 **S-curve for the Monte Carlo simulation results. Right curve (blue) bench top cleaning. Left curve (grey) rock debris drilling.**

As we see from the grey columns in the above tables the expected (means) fatal accident value less for the rock debris drilling alternative than the bench top cleaning alternative, respectively $1.2{\cdot}10^{-3}$ and $2.1{\cdot}10^{-3}$. The P_{50} (median) values are slightly lower for both operations, respectively $8.2{\cdot}10^{-4}$ and $1.4{\cdot}10^{-3}$. Said in other words a fatal accident will most likely happen approximately every 1000 year due to rock debris drilling and every 500 years due to bench cleaning. The ratio between the two is almost constant throughout the whole statistical distribution, see Figure 9, and the rock debris drilling fatal accident probability is about half of the bench top cleaning operation.

4 DISCUSSION

The results show that the rock debris drilling operation is at least as good as the bench top cleaning operation. Considering the fact that the input values are conservative in favor of the bench top cleaning operation, this simulation supports the rock debris drilling operation to be the default operation in given circumstances, and not vice versa as the law regulation text says. In the following a discussion is made to support this statement.

In the model the difference of the two operations depends on three input parameters, as the misfire occurrence rate is equal for both alternatives:

- **The probability for hitting the misfire**, which is reliable for both operations. The drilling operation is dependant of the geometrics of the drill bit, the drilling pattern and the explosive dimensions. These are known values. The uncertainty lies in the placement angle of the explosives in the rock debris, and uncertainty concerning the overlaying drilling pattern. Both these uncertainties can be estimated mathematically and distribution is quite narrow as the worst scenario calculation is barely 5 times the best scenario. For the cleaning operation the critical area hit probability is around 1. The distribution spans from 80 – 100 % cleaning efficiency. Less than 80 % is not applicable as this will lead to higher probability for hitting by drilling afterwards.

- **The misfire detonation probability** if hit by drilling is the most reliable input parameter of the complete model, and the probability is more or less 1. For cleaning it is more uncertain, but the survey results show that detonations occur regularly and speaking of detonation probabilities between 10 % and 100 % make sense. By reducing the detonation values, according to the detection probability during the cleaning operation, this gives a well defined conservative level for detonation probability.

- **The probability of a fatal incident** to occur, if detonation happens, is the most difficult value to set for both alternatives. No fatalities have occurred according to the specified analyzed operations. Does this mean that the probability is zero? By surveying fatal injury statistics from other comparable operations (e.g. tunneling drifting, secondary breaking drilling and muck pile excavation) and also looking at personal injury frequencies, it is clear that the answer is no. The survey show more frequent and more sever accidents related to drilling operations than for loading operations, and the input values are adjusted according to this. The rock debris drilling risk is set 10 times higher than the bench cleaning risk. This is quite a large difference. Nevertheless, it is assessed and used in favor of the current regulation text.

Despite all these conservative assessments, in favor of the bench cleaning process, the rock debris drilling is half as risky as the bench cleaning.

In addition, as this is not clear enough, the analysis model is in general in favor of the bench cleaning operation. The basic risk of doing two operations, instead of only one, is not included in risk calculations. This will be significant knowing that fatal injuries also are related to e.g. squeezing accidents and machine tip over [16]. The latter will also be valid for the drilling operation, as the drill rig's moving safety will be reduced.

The absolute level of fatal accident probability is not very important, as the scope of the work is to find the difference between the two operations. Anyway, by looking at the real fatal accident rates in the Norwegian quarrying and mining industry, we see that the results calculated is not very far from the reality the last 17 years. The example resembles an annual production of 1 million tons. Norway in total has had in average an annual production of 40 million tons at surface in the same period, and in total 17 blast-related fatal injuries. Less than half of them are from surface works. This gives approximately 1 blast-related fatal injury for a production volume of 100 million tons. The simulations of the rock debris drilling and the bench cleaning imply 1 fatal injury for 1000 million tons. This means that 1/10 of the blast related fatal injury risk should be attached to the analyzed operations, which gives a reasonable ratio, though not exact.

Comparing the simulation results (in average a fatality rate of $160 \cdot 10^{-5}$) with the general rate in the blast related industries the recent years ($18 \cdot 10^{-5}$), see Section 0, it is natural to expect that the drill rig operator or the excavator operator have a higher fatality risk than the general worker. The ratio of approximately one power of 10 seems fair, which also supports the model assessments.

The MEM value suggests that technological systems should not be significant to the lowest mortality rate of 5 to 15 year olds in an industrialized society. This is set to $2 \cdot 10^{-4}$ and the significance to 5 % of this, giving $1 \cdot 10^{-5}$. Comparing the reality and the simulation to this risk level, the blast related industries in general have a way to go, which actually is old news. To reduce the risk from the analyzed operations, some easy measures may be taken to reduce the risk for bench top cleaning and rock debris drilling.

- **Shot-proof safety glass** should be standard equipment on all drill rigs and excavators working on previous blasted bench tops, or blasted muckpiles. This will nearly eliminate the operator's fatal injury probability. It is important that no other personnel are working within the risk zone during the operations abolishing the risk effort advantage.

- **Video documentation** of the blasts will increase the probability to identify and locate misfires after ignition. Possible areas with misfires may be cleaned cautiously and the fatal injury probability will be reduced dramatically.

- **Use of electronical detonators** will reduce the risk. Total coupling control before ignition reduces the risk for misfire to occur, and documentation afterwards make it possible to identify and locate non detonated detonators.

- **Drillhole coordinate measuring,** by integrated GPS systems on the drill rig or manual triangulation, will open up for displacing the next bench level drill pattern according to the theoretical drillhole bottom. Logging the drillhole deviation as well, with state-of-the art equipment, the actual drillhole bottom is known, and the risk will be nearly eliminated for the rock debris drilling operation. This will, however, not have any effect on the bench top cleaning operation, nor the muckpile loading operation.

It is important to note that the risk efforts implied must give an overall site increased safety. The latter risk reducing effort alone, will decrease the rock debris drilling fatal accident probability, but the muckpile excavation operation afterwards will inherit basically all the bench cleaning fatal injury probability, as no misfire occurrence reduction efforts are made, disregarding the benefits of reducing total number of working hours and better working conditions for the drill rig.

5 CONCLUSIONS

The purpose of the work has been to compare the fatal injury risk between the operations; bench top cleaning and rock debris drilling, through analytical assessments of all causes related to unintended misfire detonation. The model looks upon a small part of all possible fatal incidents in quarrying work, but it is important to reveal the risk conditions of any part of the work to improve the overall safety. The conclusion of the work is that the rock debris drilling operation has at least half the probability of fatal accidents than the bench top cleaning operation, despite the fact that the latter is demanded in the Norwegian Explosives Regulation. The legal authorities should be aware of this and he author recommends the regulation text to be paraphrased to make the best practice the standard option.

The work show that the fatal risk probability for both operations is low, but significant compared to the real-life accident statistics. The statistical survey also shows that the blasting industry in general has a high fatal injury rate compared to other industries, and any risk reducing suggestion within the industry must be assessed seriously.

The critical part of the issue is mainly the occurrence of misfires, and the causes of that. If no misfires occur, the problem is non-existent. Hence, the main focus must be within reduction of the number of misfires and efforts reducing the occurrence frequency, e.g.:

- Explosives and detonator systems
- Drilling equipment, drilling methods
- Improved blast designs and working routines

However, one can never exclude the possibility of a misfire occurring entirely, and precautions should be taken.

Blast documentation by visual control (video camera) during the blast firing can be helpful to find possible misfires at an early stage. Critical areas can be pointed out and the misfire can be found before drilling or cleaning of the next bench starts.

Recording of blasthole coordinates and displacement of the subsequent drilling pattern according to the coordinate information will reduce the probability of hitting critical areas. Combined with drillhole deviation measurements, the probability of misfire hits is minimized.

The probability of detonation of misfires, if hit, is difficult to influence. Change in explosive type may be a solution where cartridges are used. However, this is not very relevant in larger quarries where drilling through the blasted rock debris is most common.

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APPENDICES

APPENDIX A QUARRIES, CONSTRUCTION SITES AND CONTRACTORS INVOLVED IN THE DATA ACQUISITION

Aitik Surface Mine	Sweden	Drilling and loading	2008
Bitdalen Rock Fill Dam	Rauland	Loading and transport	2006-2007
Brønnøy Limestone Quarry	Brønnøysund	Loading and transport	2003-2004
Feiring Bruk Quarry	Lørenskog	Drilling and deviation	2004
Foss Verk Quarry	Lyngdal	Deviation	2004
Fossberga Quarry	Stjørdal	Drilling and deviation	2005
Frekhaug Sports Area	Bergen	Drilling, loading and transport	2004
Jåbekk Industry Park	Mandal	Deviation	2004
Lia Quarry	Trondheim	Drilling and deviation	2004 - 2007
Melkøya LNG plant	Hammerfest	Drilling, loading and transport	2003
Norsk Stein Quarry	Suldal	Drilling, loading, deviation	2004, 2005
Road Cutting E6 Assurdalen	Oslo	Drilling	2007
Road Cutting E6 Skjeberg	Fredrikstad	Drilling	2007
Road Cutting Rv 21 Skotterud	Glåmdal	Drilling	2007
Road Cutting Rv 4	Raufoss	Loading and transport	2004
Salmar Industry Park	Frøya	Drilling and deviation	2007
Songa Rock Fill Dam	Vinje	Drilling, loading and transport	2002-2004
Stjernøya Syenite Surface Mine	Stjernøya	Drilling, loading and transport	2003-2006
Støleheia Quarry	Kristiansand	Deviation	2004
Titania Surface Mine	Tellnes	Drilling	2007
Tromsdalen Limestone Quarry	Verdal	Drilling and deviation	2006
Vassfjellet Quarry	Trondheim	Drilling and deviation	2001-2007
Visnes Limestone Quarry	Eide	Drilling, loading and deviation	2007

Table A.1 Complete list of quarries and project sites included in the field data acquisition work. See also appendices in the report series (Volume 2 to 6).

AF Gruppen
Albert Kr. Hæhre Entreprenør AS
Atlas Copco Anlegg-og Gruveteknikk AS
Bernt Jonny Maskin AS
Brønnøy Kalk AS
Devico
DiaTeam
Espevik & Buck
Franzefoss Pukk
Kjell Foss AS
Koren Sprengningsservice AS
Maskineier Willy Melbye AS
Norcem Kjøpsvik
Norsk Stein AS
Norstone Tau
North Cape Minerals
Orica Mining Services, (former Dyno Nobel.)
Øyvind Ødegård Fjellboring AS
Råde Graveservice AS
Sandvik AS
Secoroc
Statkraft
T.S. Stangleand Maskin AS
Terrengtransport AS
Titania
Verdalskalk AS
Visnes Kalk og Eklogitt AS

Table A.2 Complete list of companies contributing with data, equipment and personnel during the thesis work.

APPENDIX B GRINDING EQUIPMENT

	Button diameter						
Button shape	**9 mm**	**10 mm**	**11 mm**	**12 mm**	**13 mm**	**14 mm**	**16 mm**
Spherical	Bit 7*) Grinder 2		Bit 3, 7, 10, 11 Grinder 1	Bit 3, 12 Grinder 2		Bit 12 Grinder 1	Bit 12 Grinder 1
Semi-ballistic		Bit 1, 4 Grinder 2	Bit 1, 2, 4, 5, 6 Grinder 1	Bit 2 Grinder 2	Bit 6 Grinder 2		
Ballistic			Bit 9 Grinder 1	Bit 14 Grinder 1	Bit 13 Grinder 1	Bit 14 Grinder 1	Bit 13 Grinder 1

Table B.1 Grinding machines and cup dimensions used for grinding the different drill bits in the extensive drill bit testing program. *) Not obtained; 10 mm semi ballistic grinding cup was used instead

Figure B.1 Grinding machines.
Left: Sanroc Multi, Sandvik
Right: CME junior; Construction and Mining Equipment

APPENDIX C PICTURES OF THE BITS INCLUDED IN THE EXTENSIVE TESTING PROGRAM

Drill bit 1: 76 mm, drop centre, semi ballistic.

Drill bit 2: 89 mm, drop centre, semi ballistic.

Drill bit 3: 89 mm, drop centre, spherical

Drill bit 4: 89 mm, flat front, semi ballistic.

Drill bit 5: 76 mm, drop centre, semi ballistic.

Drill bit 6: 89 mm, drop centre, semi ballistic

Drill bit 7: 89 mm, drop centre, spherical.

Drill bit 8: 102 mm, drop centre, semi ballistic.

Drill bit 9: 89 mm, drop centre, ballistic

Drill bit 10: 89 mm, drop centre, spherical.

Drill bit 11: 76 mm, drop centre, spherical.

Drill bit 12: 102 mm, drop centre, spherical

Drill bit 13: 102 mm, flat front, ballistic.

Drill bit 14: 89 mm, flat front, ballistic.

APPENDIX D LABORATORY RESULTS FROM THE EXTENSIVE DRILL BIT PROGRAM

	SITE 1	SITE 2	SITE 3 AND 4	SITE 5	SITE 6	SITE 7	SITE 8	SITE 9
Rock type	Greenstone	Granodiorite	Greenstone	Greenstone	Greenstone	Greenstone	Limestone	Marble
Brittleness value [%]	50.3	45.6	32.7	35	49.3	40.6	51.6	66.3
Flakiness value	1.37	1.42	1.36	1.33	1.35	1.34	1.38	1.25
Compaction index	1	0	1	1	1	1	3	1
Density [g/cm^3]	3.05	2.55	2.87	3.05	3.08	3.03	2.72	2.70
Sievers' J value [1/10 mm]	31.96	2.3	16.2	3.6	8.4	7.8	81.5	110.6
Abrasion value [mg]	2.5	11.5	0.5	1.5	2.5	3	0.5	1.0
Abrasion value steel [mg]	3.0	5	0.5	4	4	0.5	0.5	1.0
DRI	56	38	35	30	48	39	62	66
BWI	21	46	22	37	25	34	11	9
CLI	34.4	10.3	52.8	13.3	18.4	39.8	98	85
Sievers J Interseption Point [sec]	11.1	2.1	8.3	3.9	3.2	3.9	24.3	23.5[1)]
Quarz content [%]	0	21	0	0	0	0	0	0
Blastability Index SPR	0.40	0.4	0.38	0.41	0.42	0.42	0.37	-
Cerchar Abrasion Index [1/10 mm]	4.0	4.6	3.2	3.4	2.2	3.2	2.1	-
Brazillian Tensile Strength [Mpa]	-	-	-	-	-	-	8	5
UCS [MPa]	84.3	163.2	60.8	159	149	143	81	67
Point load index PLI_{50} [MPa]	5.2	9.9	8.1	8.3	9.0	10.1	3.9	3.15
Compressive strength from point load [MPa]	114	218	178	182	198	222	86	65
Tensile strength from point load [MPa]	6	12	10	10	11	13	5	4
Sonic velocity [m/s]	3877	5678	4783	5382	4723	4801	6374	5402
Youngs modulus [GPa]	32	53	56	60	40	50	88	66
Transversal contraction/ Poisson	0.14	0.18	0.1	0.1	0.14	0.12	0.254	0.242

Table D.1 Laboratory test results. 1) Different block, same rock.

PRØVE MRK.	SITE 1A	SITE 1B	SITE 2	SITE 3 AND 4	SITE 5	SITE 6	SITE 7	SITE 8	SITE 9
Plagioclas	37 %	40 %	58 %	62 %	36 %	44 %	40 %		
K-feltspar			17 %	9 %	4 %	6 %	5 %		
Quartz			21 %						
Mica			2 %						
Kaolin			1 %						
Amfibole	22 %	26 %		18 %	39 %	38 %	39 %		
Pyrite	Spor								
Epidote	37 %	30 %		5 %	19 %	10 %	11 %		
Clorite	4 %	4 %		6 %	2 %	2 %	5 %		
Chalkopyritt			1						
Calsite								100 %[2)]	100 %[2)]

Table D.2 Semi quantitative mineral analysis. 2) Based on previous tests. Traces of other minerals may be found.

APPENDIX E SUMMARY OF THE EXTENSIVE DRILL BIT PROGRAM MEASUREMENTS

Bit name	Drill bit 1	Drill bit 2	Drill bit 3	Drill bit 4	Drill bit 5	Drill bit 6	Drill bit 7	Drill bit 8	Drill bit 9	Drill bit 10	Drill bit 11	Drill bit 12	Drill bit 13	Drill bit 14	Drill bit X
Bit size	76 mm	89 mm	89 mm	89 mm	76 mm	89 mm	89 mm	102 mm	89 mm	89 mm	76 mm	102 mm	102 mm	89 mm	89 mm
Original bit diameter [mm]	78.3	91.2	91.0	90.2	78.3	90.2	90.7	103.6	90.2	90.8	78.9	105.0	103.6	90.1	-
Original skirt diameter [mm]	74.4	86.5	87.0	86.0	72.5	86.5	86.5	77.0	84.0	63.4	72.0	97.1	98.1	84.5	
Threads	T45	T51	T51	T51	T45	T51	T51	T51	T51	T45	T45	T51	T51	T51	T51/T45
Front design	Drop Center	Drop Center	Drop Center	Flat Front	Drop Center	Drop Center	Flat Front	Drop Center	Drop Center	Drop Center	Drop Center	Drop Center	Flat Front	Flat Front	Drop Center
Button Design	Semi ballistic	Semi ballistic	Spherical	Semi ballistic	Semi ballistic	Semi ballistic	Spherical	Semi ballistic	Ballistic	Spherical	Spherical	Spherical	Ballistic$^{1)}$	Ballistic	Semi ballistic
Site 1															
Diameter before drilling [mm]	-	91.2	91.0	90.2	-	90.2	90.7	103.6	90.2	-	78.9	-	-	-	-
Diameter after drilling [mm]	-	91.1	90.9	90.1	-	90.1	90.6	103.6	90.1	-	78.8	-	-	-	-
Skirt diameter after drilling [mm]		85.6	86.1	85.0		85.4	85.5	77.0	83.5		71.8				
Number of holes	-	4	4	4	-	4	4	4	4	-	4	-	-	-	-
Drilled meters [m]	-	61.8	58.6	60.1	-	60.6	61.4	60.3	60.2	-	58.9	-	-	-	-
Net pen rate 2nd rod P_{10} [cm/min]	-	108	104	121	-	122	113	75	108	-	118	-	-	-	-
Avg. defl. at 14 m [% or cm/m]	-	3.20	2.60	3.45	-	2.68	2.04	4.36	2.88	-	3.58	-	-	-	-
Min [% or cm/m]	-	2.50	2.10	2.30	-	1.10	0.60	2.60	1.30	-	1.40	-	-	-	-
Max [% or cm/m]	-	4.30	3.50	4.70	-	4.40	4.60	6.14	4.10	-	5.70	-	-	-	-
Site 2															
Diameter before drilling [mm]	78.3	90.7	90.0	89.5	78.3	90.1	90.3	-	89.0	90.8	78.4	105	-	-	-
Diameter after drilling [mm]	77.9	90.2	89.6	89.3	77.8	89.9	90.1	-	88.8	90.8	78.2	103.1	-	-	-
Skirt diameter after drilling [mm]	72.6	83.9	84.6	83.6	71.2	85.1	85.2		82.5	63.1	70.6	96.6			
Number of holes	2	2	2	2	2	2	2	-	2	1	2	3	-	-	-
Drilled meters [m]	40.1	34.9	36.9	43.6	29.6	34.7	35.6	-	36.4	13.3	28.3	44.1	-	-	-
Net pen rate 2nd rod P_{10} [cm/min]	109	95	85	83	108	88	85	-	92	84	99	60	-	-	-
Avg. defl. at 12 m [% or cm/m]	20.6	7.2	2.5	11.1	19.7	8.7	6.4	-	10.8	-	7.5	20.6	-	-	-
Min [% or cm/m]	16.7	5.6	2.2	9.5	16.5	8.3	5.1	-	8.2	-	6.5	16.7	-	-	-
Max [% or cm/m]	24.5	8.8	4.0	12.7	22.9	9.1	7.7	-	13.5	-	8.3	24.5	-	-	-
Site 3 and 4															
Diameter before drilling [mm]	-	90.1	89.4	89.0	-	89.6	89.5	103.0	87.9	-	-	102.8	103.6	-	-
Diameter after drilling [mm]	-	$89.2^{2)}$	$89.2^{2)}$	$88.2^{2)}$	-	$89.2^{2)}$	$89.4^{2)}$	102.4	$87.6^{2)}$	-	-	102.8	103.4	-	-
Skirt diameter after drilling [mm]		83.4	84.1	82.9		84.8	84.9	77.0	82.4			96.5	97.9		
Number of holes	-	6	6	6	-	6	6	3	6	-	-	3	3	-	-
Drilled meters [m]	-	95.4	97.2	94.9	-	96.1	94.5	46.3	94.7	-	-	48.5	46.8	-	-
Net pen rate 2nd rod P_{10} [cm/min]	-	91	84	78	-	83	80	73	90	-	-	66	71	-	-
Avg. defl. at 14 m [% or cm/m]	-	3.4	4.0	3.7	-	4.0	3.8	11.3	4.3	-	-	2.6	2.7	-	-
Min [% or cm/m]	-	0.7	1.6	2.4	-	2.6	1.6	7.0	2.6	-	-	2.4	2.3	-	-
Max [% or cm/m]	-	7.4	6.7	4.4	-	5.3	7.0	16.4	7.1	-	-	3.0	3.2	-	-
Site 5															
Diameter before drilling [mm]	77.7	88.9	88.8	87.6	77.5	89.1	88.9	102.3	86.8	90.6	77.9	102.6	-	-	-
Diameter after drilling [mm]	77.6	$87.9^{2)}$	$88.4^{2)}$	$87.2^{2)}$	77.5	$88.8^{2)}$	$88.5^{2)}$	-	$85.9^{2)}$	90.5	77.7	102.6	-	-	-
Skirt diameter after drilling [mm]	71.5	81.9	82.4	82.0	70.5	84.1	84.1		82.0	62.5	70.2	96.5			
Number of holes	4	6	7	5	4	6	6	-	6	4	4	2	-	-	-
Drilled meters [m]	57.8	91.4	107.7	75.4	56.7	90.0	88.2	-	88.0	57.5	57.7	30.8	-	-	-
Net pen rate 2nd rod P_{10} [cm/min]	115	99	93	94	116	97	94	-	99	96	108	70	-	-	-
Avg. defl. at 14 m [% or cm/m]	6.4	5.0	4.8	3.3	5.0	3.6	4.7	-	3.0	7.0	7.1	6.0	-	-	-
Min [% or cm/m]	4.1	3.9	1.1	1.9	3.1	2.0	1.0	-	2.1	6.0	6.0	4.0	-	-	-
Max [% or cm/m]	7.7	6.6	7.9	5.0	7.0	4.4	11.0	-	4.1	8.0	9.2	7.9	-	-	-

Figure E.1 Summary of key data measured in the drill bit test program. (Continues next page)

Site 6															
Diameter before drilling [mm]	-	87.2	88.3	86.8	-	88.5	88.4	-	85.3	90.1	-	102.6[3]	102.4	90.1	-
Diameter after drilling [mm]	-	87.2	88.2	86.8	-	88.4	88.2	-	85.3	90.0	-	102.5	102.3	90	-
Skirt diameter after drilling [mm]		81.1	81.8	81.7		83.6	83.6		81.7	62.0		95.9	97.4	83.6	
Number of holes	-	3	4	2	-	4	4	-	4	4	-	2	5	3	-
Drilled meters [m]	-	36.8	50.4	25.6	-	51.0	52.7	-	52.3	50.3	-	24.9	62.9	38.9	-
Net pen rate 2nd rod P_{10} [cm/min]	-	118	108	85	-	99	99	-	103	85	-	69	70	96	-
Avg. defl. at 12 m [% or cm/m]	-	2.6	2.2	1.2	-	2.3	2.5	-	2.0	2.1	-	1.8	2.3	1.9	-
Min [% or cm/m]	-	2.1	1.4	1.0	-	1.0	1.9	-	1.2	0.9	-	0.6	0.9	0.9	-
Max [% or cm/m]	-	3.3	3.0	1.3	-	3.2	3.6	-	2.8	3.7	-	3.0	3.2	3.2	-
Site 7															
Diameter before drilling [mm]	-	86.7	88.1	86.3	-	87.4	88.1	-	84.4	89.7	-	-	-	89.1	-
Diameter after drilling [mm]	-	86.5	88.1	86.2	-	87.3	87.9	-	84.3	89.5	-	-	-	89.0	-
Skirt diameter after drilling [mm]		80.8	81.6	81.4		83.6	83.5		81.6	62.0				83.5	
Number of holes	-	3	3	3	-	3	3	-	4	3	-	-	-	3	-
Drilled meters [m]	-	40.7	40.1	39.9	-	40.6	40.2	-	53.8	39.5	-	-	-	37.6	-
Net pen rate 2nd rod P_{10} [cm/min]	-	75	70	64	-	75	67	-	78	71	-	-	-	71	-
Avg. defl. at 12 m [% or cm/m]	-	6.5	4.3	4.4	-	3.7	1.9	-	3.0	1.8[4]	-	-	-	5.7	-
Min [% or cm/m]	-	5.3	2.6	2.4	-	2.4	1.1	-	1.3	0.8	-	-	-	3.2	-
Max [% or cm/m]	-	7.9	6.6	7.9	-	4.5	2.7	-	4.3	2.8	-	-	-	8.6	-
Site 8															
Diameter before drilling [mm]	76.6	85.7	87.9	85.9	76.6	86.6	87.8	-	83.7	-	77.4	102.3	102.0	88.4	90.2
Diameter after drilling [mm]	76.6	85.7	87.9	85.9	76.6	86.6	87.8	-	83.7	-	77.4	102.3	102.0	88.3	90.2
Skirt diameter after drilling [mm]	71.5	80.8	81.6	81.4	70.5	83.6	83.5		81.6		70.2	95.9	97.4	83.5	?
Number of holes	2	3	3	3	3	3	2	-	3	-	2	3	3	3	2
Drilled meters [m]	34.8	52.2	52.2	54	52.2	52.2	34.8	-	53.1	-	34.8	52.2	52.2	52.2	34.8
Net pen rate 2nd rod P_{10} [cm/min]	166	125	97	115	150	116	99	-	125	-	131	78	104	140	141
Avg. defl. at 17 m [% or cm/m]	7.8	5.5[4]	8.6[4]	4.7[4]	10.9	1.1[4]	13.6[4]	-	2.8[4]	-	12.1	4.4[4]	9.9[4]	4.0[4]	6.9
Min [% or cm/m]	7.7	4.8	5.4	2.0	4.6	1.0	13.6	-	1.0	-	10.6	3.6	7.6	2.6	6.8
Max [% or cm/m]	7.8	6.0	11.1	8.0	16.3	1.4	13.7	-	4.5	-	13.5	4.9	12.3	4.9	7.0
Site 9															
Diameter before drilling [mm]	-	85.7[3]	87.9[3]	85.9[3]	-	86.6[3]	83.7[3]	-	83.7[3]	-	-	102.3[3]	102.0[3]	88.3[3]	91.2
Diameter after drilling [mm]	-	85.7	87.9	85.9	-	86.6	83.7	-	83.7	-	-	102.3	102.0	88.3	91.2
Skirt diameter after drilling [mm]		80.8	81.6	81.4		83.6	83.5		81.6			95.9	97.4	83.5	?
Number of holes	-	3	3	3	-	3	3	-	3	-	-	3	3	3	2
Drilled meters [m]	-	43.3	43.4	43.8	-	43.8	43.7	-	46.2	-	-	45.2	43.4	43.0	29.6
Net pen rate 2nd rod P_{10} [cm/min]	-	155	136	138	-	150	136	-	144	-	-	113	122	150	172
Avg. defl. at 14 m [% or cm/m]	-	3.2	6.0	3.9	-	2.6	6.5	-	1.6		-	6.8	3.3	3.4	4.5
Min [% or cm/m]	-	2.9	3.6	1.9	-	1.9	4.6	-	1.4		-	5.7	1.5	2.8	4.4
Max [% or cm/m]	-	3.5	9.9	6.9	-	3.1	10.0	-	1.8		-	8.0	6.7	3.9	4.7

1) Originally semi ballistic. fully ballistic after 1st grinding. 2) Inclusive one grinding. 3) Not ground 4) Adapter

Figure E.1 Continued from previous page.

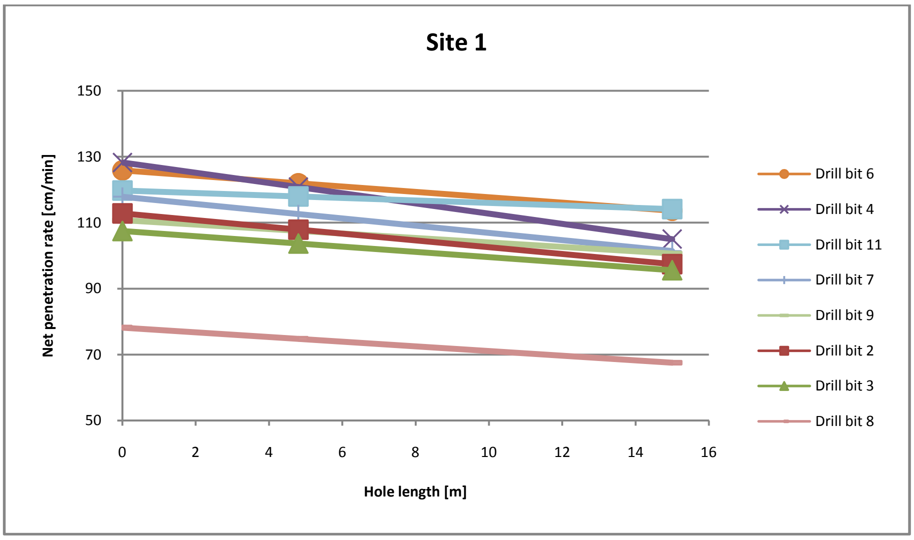
Regression data		Drill bit 1	Drill bit 2	Drill bit 3	Drill bit 4	Drill bit 5	Drill bit 6	Drill bit 7	Drill bit 8	Drill bit 9	Drill bit 10
Site 1	60 cm (P10)		y = -1,0277x + 112,81	y = -0,7925x + 107,47	y = -1,5486x + 128,24		y = -0,8174x + 125,87	y = -1,0958x + 117,85	y = -0,7096x + 78,181	y = -0,6732x + 110,74	
	60 cm (P50)		y = -0,6822x + 144,44	y = -0,4824x + 139,4	y = -1,8455x + 148,18		y = -1,1985x + 150,89	y = -0,8228x + 136,79	y = 0,3991x + 98,232	y = -1,6079x + 142,96	
	Whole rods		y = 0,0072x + 136,58	y = -0,6247x + 139,08	y = -1,4167x + 144,48		y = -1,5907x + 152,99	y = -1,2265x + 140,41	y = 0,5505x + 95,79	y = -1,5867x + 141,86	
Site 2	60 cm (P10)	y = -0,7813x + 112,72	y = -0,5102x + 97,885	y = -0,7839x + 88,322	y = -0,2087x + 84,142	y = -0,5253x + 110,53	y = -0,1741x + 88,982	y = -0,6371x + 88,145		y = 0,0699x + 91,961	y = -0,2867x + 85,082
	60 cm (P50)	y = -0,726x + 118,92	y = -1,5269x + 117,96	y = -0,8981x + 95,94	y = -0,2908x + 95,229	y = -0,0158x + 113,37	y = -0,2743x + 105,72	y = -0,7775x + 100,36		y = 0,0103x + 101,41	y = -1,3878x + 99,233
	Whole rods	y = -0,6414x + 118,41	y = -1,4183x + 115,27	y = -0,8557x + 95,202	y = -0,2602x + 94,184	y = -0,2841x + 115,99	y = -0,5328x + 108,44	y = -0,8069x + 99,673		y = -0,279x + 102,43	y = -1,194x + 98,005
Site 3 and 4	60 cm (P10)		y = -0,9711x + 95,522	y = -1,35x + 90,653	y = -0,385x + 79,439		y = -0,7806x + 86,999	y = -0,7972x + 83,664	y = -0,683x + 76,258	y = -0,6247x + 93,226	
	60 cm (P50)		y = -0,8519x + 112,89	y = -1,679x + 116,22	y = -0,5737x + 95,619		y = -1,6149x + 108,02	y = -1,7391x + 109,22	y = -1,4952x + 94,123	y = -1,0614x + 117,15	
	Whole rods		y = -1,0838x + 114,72	y = -1,8416x + 119,93	y = -0,9916x + 99,834		y = -1,5343x + 109,85	y = -1,6551x + 108,93	y = -1,762x + 95,525	y = -1,0854x + 117,14	
Site 5	60 cm (P10)	y = -0,5037x + 117	y = -0,6767x + 102,63	y = -1,4078x + 99,423	y = -0,7362x + 97,774	y = -0,6124x + 118,73	y = -0,5491x + 100,02	y = -1,4251x + 100,57		y = -0,6155x + 101,43	y = -1,1509x + 101,62
	60 cm (P50)	y = -0,3028x + 131,46	y = -0,9303x + 129,87	y = -1,2162x + 123,89	y = -1,4504x + 126,51	y = -0,959x + 142,84	y = -1,5753x + 134,09	y = -0,6153x + 111,36		y = -0,9825x + 121,24	y = -0,4947x + 111,18
	Whole rods	y = -0,0452x + 129,08	y = -0,6829x + 125,33	= -1,327x + 125,37	y = -1,2316x + 124,3	y = -0,8794x + 141,65	y = -1,3701x + 131,93	y = -1,2708x + 115,87		y = -0,776x + 118,97	y = -0,3935x + 110,62
Site 6	60 cm (P10)		y = -2,622x + 130,19	y = -4,4205x + 129,05	y = -1,3709x + 91,192		y = -1,2035x + 105,2	y = -3,1562x + 114,1		y = -1,5549x + 110,42	y = -1,3463x + 91,679
	60 cm (P50)		y = -2,3041x + 139,61	y = -3,363x + 137,41	y = -3,083x + 125,39		y = -1,6033x + 126,26	y = -3,3541x + 130,75		y = -2,2749x + 128,78	y = -2,3611x + 120,64
	Whole rods		y = -2,4445x + 139,54	y = -4,2579x + 141,66	y = -2,7057x + 121,61		y = -2,0709x + 129,59	y = -3,5644x + 129,91		y = -2,2942x + 128,99	y = -2,7935x + 120,78
Site 7	60 cm (P10)		y = -0,8329x + 79,039	y = -1,0762x + 75,545	y = -0,1749x + 64,482		y = -1,2885x + 80,762	y = 0,1657x + 66,046		y = -0,6433x + 80,942	y = -0,472x + 73,318
	60 cm (P50)		y = -0,9157x + 92,622	y = -1,9088x + 96,097	y = -1,436x + 86,573		y = -1,1344x + 89,81	y = -0,7352x + 83,973		y = -0,4168x + 93,934	y = -1,1934x + 92,728
	Whole rods		y = -0,6813x + 91,358	y = -2,3428x + 100,61	y = -1,781x + 90,465		y = -1,068x + 90,434	y = -0,5942x + 82,329		y = -0,3668x + 94,861	y = -0,9002x + 88,94
Site 8	60 cm (P10)	y = -1,3633x + 172,55	y = 0,0812x + 124,26	y = -0,0682x + 97,305	y = -0,6348x + 117,58	y = -0,7916x + 153,4	y = -0,012x + 115,62	y = -1,4813x + 106,3		y = -0,5079x + 127,69	
	60 cm (P50)	y = -1,6884x + 185,36	y = -0,1722x + 137,01	y = -0,3656x + 111,19	y = -0,7936x + 130,68	y = -0,9558x + 171,13	y = -0,7899x + 137,52	y = -1,6981x + 115,96		y = -0,8928x + 142,42	
	Whole rods	y = -1,855x + 187,33	y = -0,0782x + 135,73	y = -0,9008x + 117,59	y = -1,2467x + 133,32	y = -1,053x + 171,98	y = -0,9244x + 138,92	y = -1,8727x + 117,95		y = -1,1046x + 144,75	
Site 9	60 cm (P10)		y = -0,47x + 156,7	y = -0,2269x + 136,81	y = -1,0036x + 143,24		y = 0,0242x + 150,23	y = -1,8092x + 144,94		y = 0,2049x + 143,38	
	60 cm (P50)		y = -0,0832x + 161,8	y = -0,4091x + 145,45	y = -0,7051x + 149,9		y = -0,3021x + 160,45	y = -1,8092x + 144,94		y = -0,1507x + 153,69	
	Whole rods		y = 0,0336x + 160,88	y = -0,4135x + 145,73	y = -0,7153x + 149,5		y = 0,0408x + 157,46	y = -1,7135x + 144,07		y = -0,4817x + 156,95	

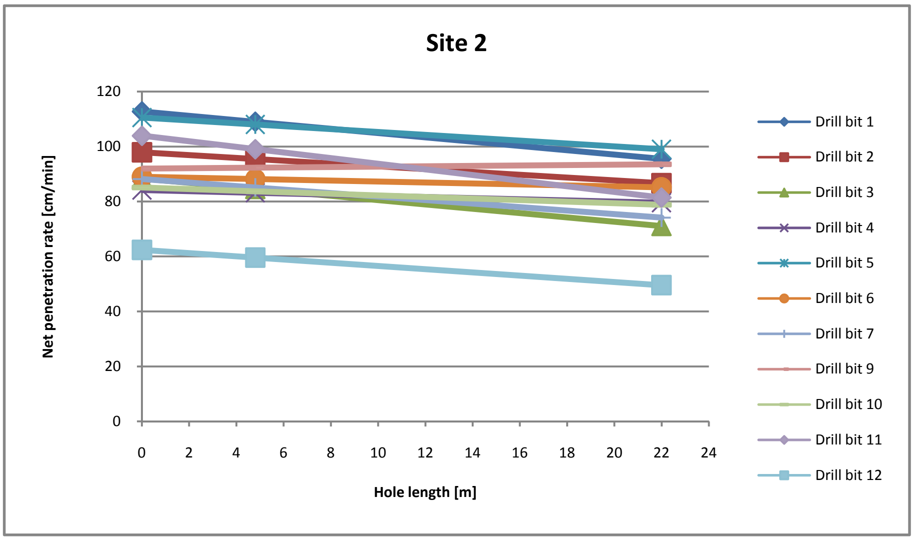
Regression data		Drill bit 11	Drill bit 12	Drill bit 13	Drill bit 14	Drill bit X
Site 1	60 cm (P10)	y = -0,3766x + 119,74				
	60 cm (P50)	y = -0,6404x + 154,68				
	Whole rods	y = 0,0087x + 146,73				
Site 2	60 cm (P10)	y = -1,0225x + 103,92	y = -0,5831x + 62,36			
	60 cm (P50)	y = -1,0412x + 114,95	y = -1,0257x + 73,339			
	Whole rods	y = -1,3226x + 117,15	y = -0,9536x + 72,845			
Site 3 and 4	60 cm (P10)		y = -0,83x + 69,651	y = -0,3992x + 72,529		
	60 cm (P50)		y = -0,8453x + 79,375	y = -0,0819x + 82,951		
	Whole rods		y = -0,5544x + 77,094	y = -0,3933x + 90,083		
Site 5	60 cm (P10)	y = -0,5105x + 110,7	y = -0,4201x + 71,66			
	60 cm (P50)	y = -0,6385x + 131,58	y = -0,3686x + 81,487			
	Whole rods	y = -0,5891x + 130,04	y = -0,5458x + 82,247			
Site 6	60 cm (P10)		y = -0,7498x + 72,56	y = -0,8143x + 73,874	y = -2,4277x + 107,95	
	60 cm (P50)		y = -1,9346x + 95,189	y = -1,3845x + 96,516	y = -3,788x + 136,5	
	Whole rods		y = -1,7888x + 91,985	y = -1,7498x + 98,786	y = -3,7737x + 134,39	
Site 7	60 cm (P10)				y = -0,4473x + 72,601	
	60 cm (P50)				y = -0,3197x + 88,065	
	Whole rods				y = 0,186x + 83,582	
Site 8	60 cm (P10)	y = -0,9538x + 138,23	y = -0,5708x + 80,889	y = -0,9041x + 108,32	y = -0,874x + 144,18	y = -1,36x + 147,63
	60 cm (P50)	y = -0,8963x + 147,01	y = -0,6093x + 87,663	y = -0,4757x + 110,33	y = -1,3125x + 158,36	y = -1,6612x + 162,8
	Whole rods	y = -0,841x + 146,42	y = -0,6747x + 87,965	= -0,6188x + 111,98	y = -1,2993x + 158,41	y = -1,6093x + 161,48
Site 9	60 cm (P10)		y = -0,7353x + 116,34	y = -0,3873x + 123,78	y = 0,0453x + 150,04	y = -0,2478x + 172,78
	60 cm (P50)		y = -0,4022x + 120,63	y = -0,5965x + 131,71	y = -0,4051x + 162,68	y = -0,0233x + 178,3
	Whole rods		y = -0,4267x + 120,4	y = -0,4608x + 130,54	y = -0,1727x + 160,85	y = 0,2658x + 175,17

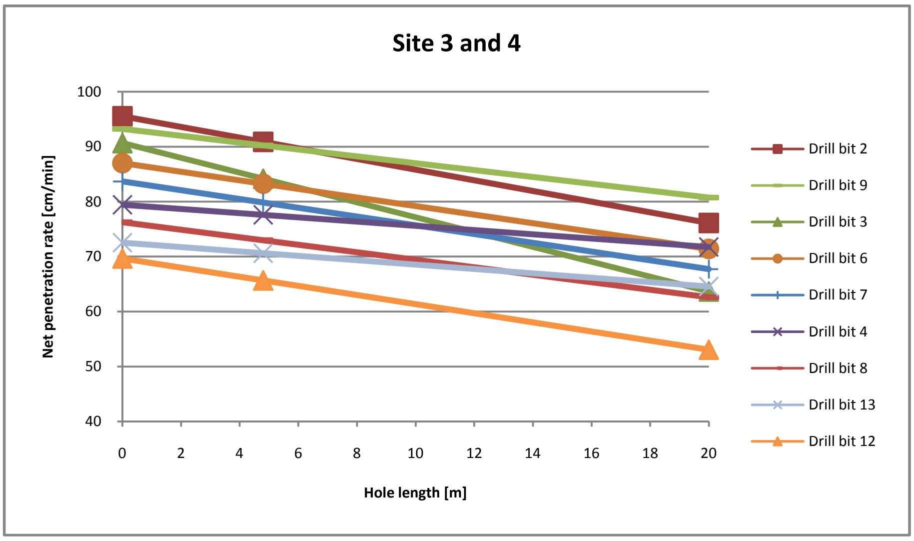
Figure E.2 Summary of the regression curves used in the drill bit testing program.

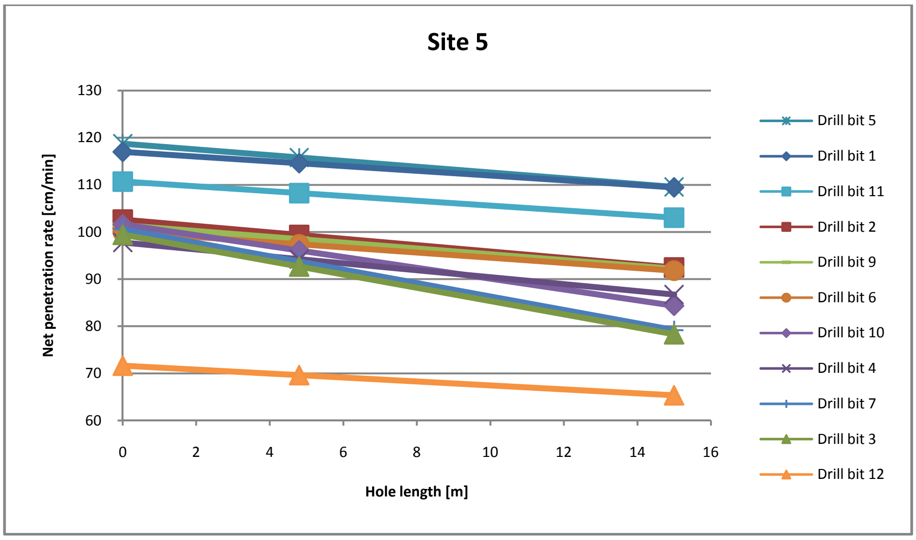
APPENDIX F NET PENETRATION RATE REGRESSION, HOMOGENOUS ROCK (P_{10})

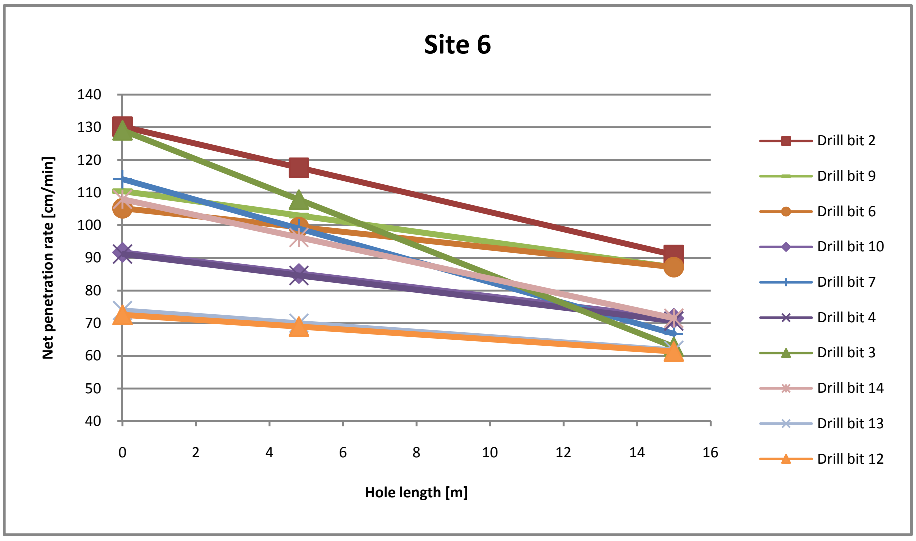
Curves based on 60 cm (P_{10}) in Appendix E, Figure E.2

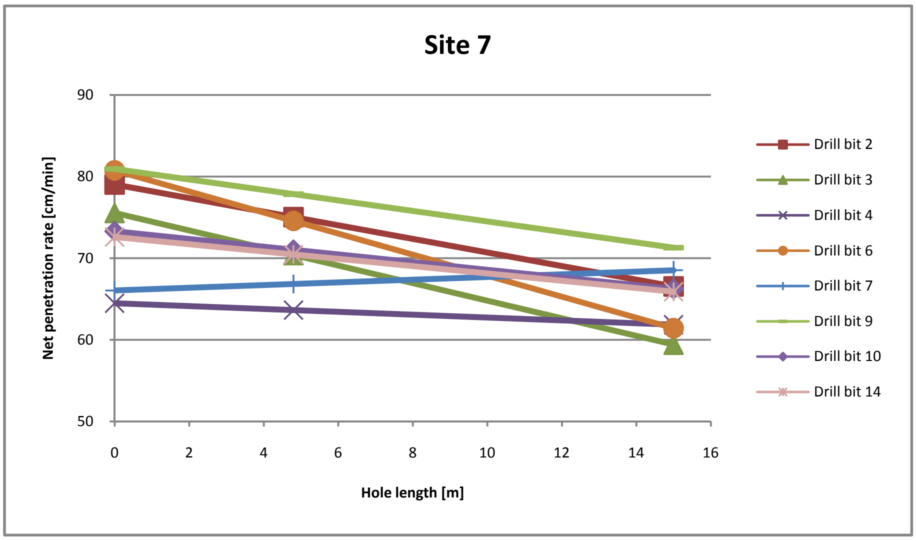


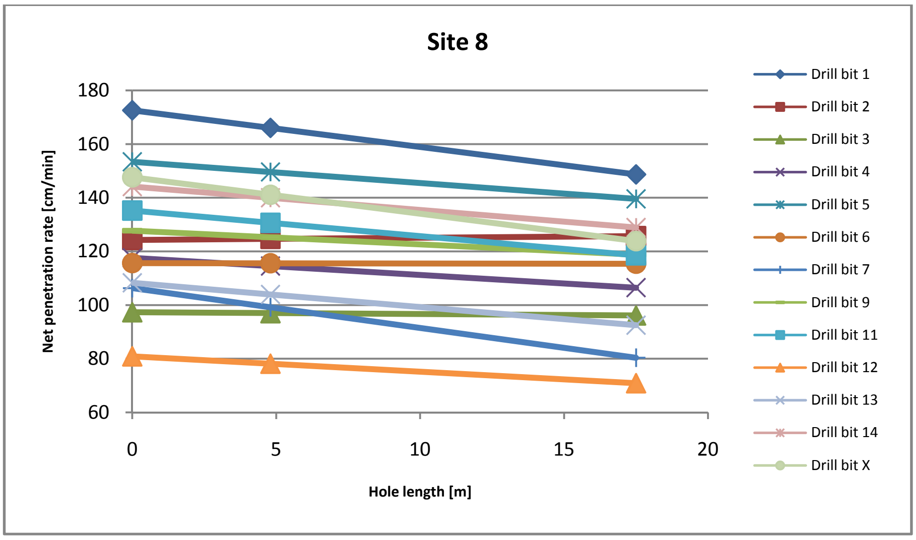




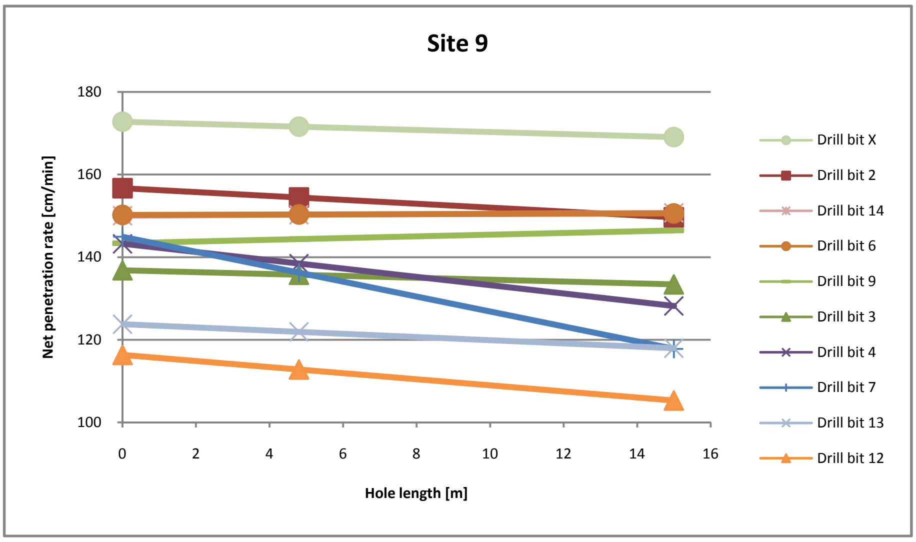

Site 5
Net penetration rate [cm/min]
130
120
110
100
90
80
70
60
0
2
4
6
8
10
12
14
16
Hole length [m]
Drill bit 5
Drill bit 1
Drill bit 11
Drill bit 2
Drill bit 9
Drill bit 6
Drill bit 10
Drill bit 4
Drill bit 7
Drill bit 3
Drill bit 12


Site 6
Net penetration rate [cm/min]
140
130
120
110
100
90
80
70
60
50
40
0
2
4
6
8
10
12
14
16
Hole length [m]
Drill bit 2
Drill bit 9
Drill bit 6
Drill bit 10
Drill bit 7
Drill bit 4
Drill bit 3
Drill bit 14
Drill bit 13
Drill bit 12


Site 7
Net penetration rate [cm/min]
90
80
70
60
50
0
2
4
6
8
10
12
14
16
Hole length [m]
Drill bit 2
Drill bit 3
Drill bit 4
Drill bit 6
Drill bit 7
Drill bit 9
Drill bit 10
Drill bit 14



Site 8
Net penetration rate [cm/min]
180
160
140
120
100
80
60
0
5
10
15
20
Hole length [m]
Drill bit 1
Drill bit 2
Drill bit 3
Drill bit 4
Drill bit 5
Drill bit 6
Drill bit 7
Drill bit 9
Drill bit 11
Drill bit 12
Drill bit 13
Drill bit 14
Drill bit X



Site 9
Net penetration rate [cm/min]
180
160
140
120
100
0
2
4
6
8
10
12
14
16
Hole length [m]
Drill bit X
Drill bit 2
Drill bit 14
Drill bit 6
Drill bit 9
Drill bit 3
Drill bit 4
Drill bit 7
Drill bit 13
Drill bit 12

APPENDIX G NET PENETRATION RATE VS. DRILL BIT DIAMETER

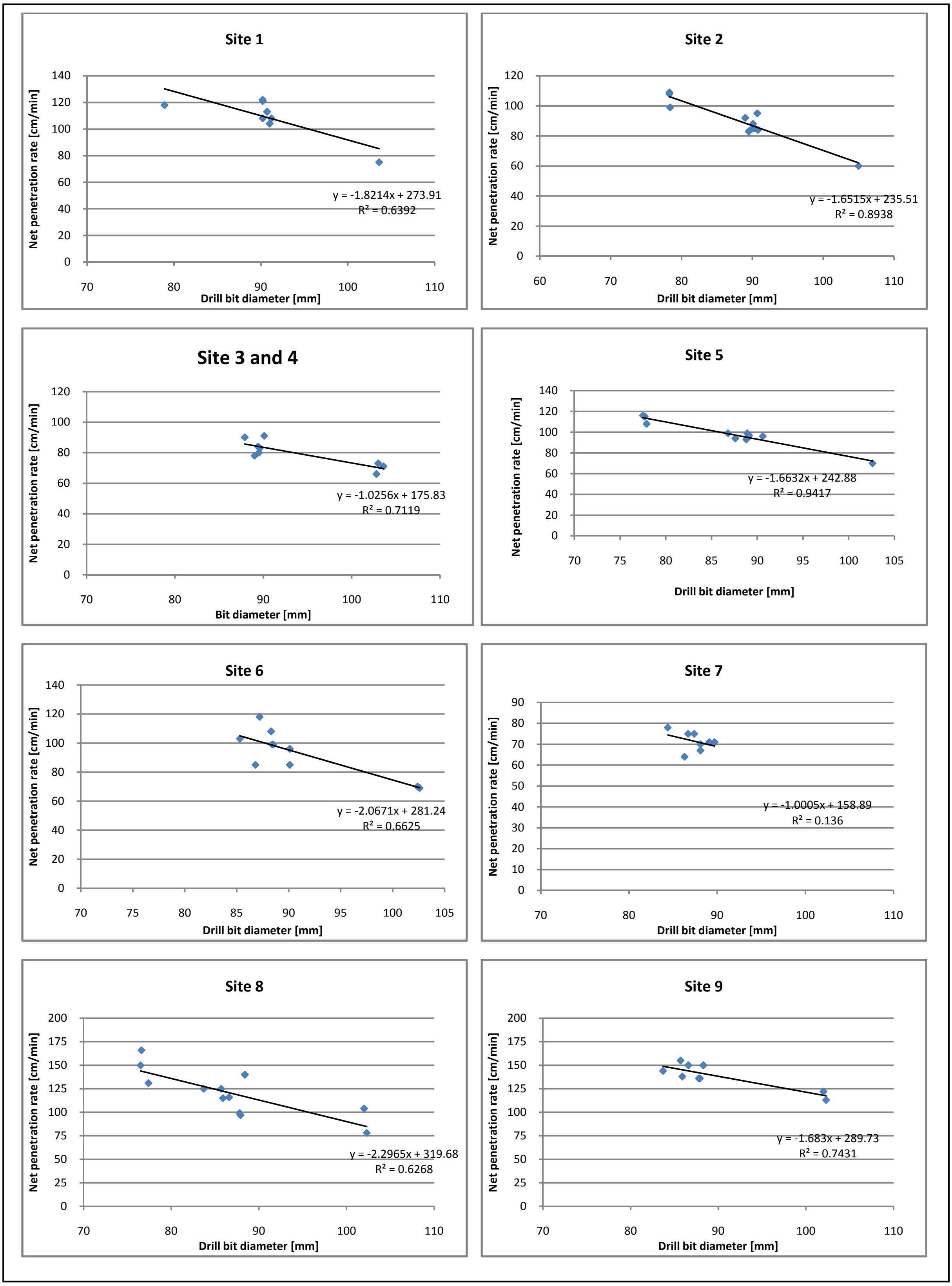


Figure G.1 **Net penetration rate vs. drill bit diameter for all drill bits in the testing program. Each point represents the net penetration rate for the 2^{nd} rod for each bit. (P_{10} regression value at 4.8 m hole depth).**

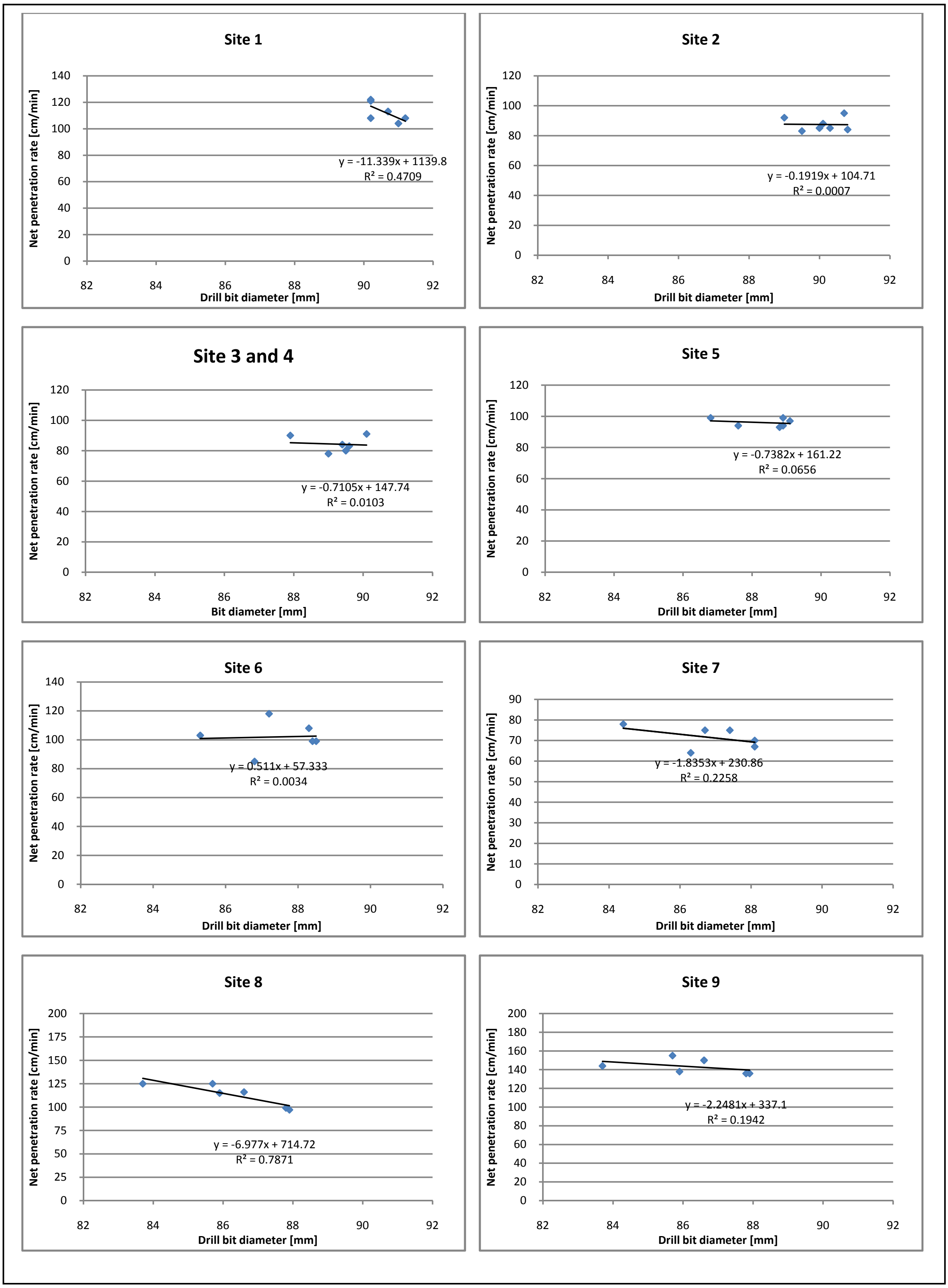


Figure G.2 **Net penetration rate vs. drill bit diameter for the 89 mm category drill bits. Each point represents the net penetration rate for the 2nd rod for each bit. (P_{10} regression value at 4.8 m hole depth).**

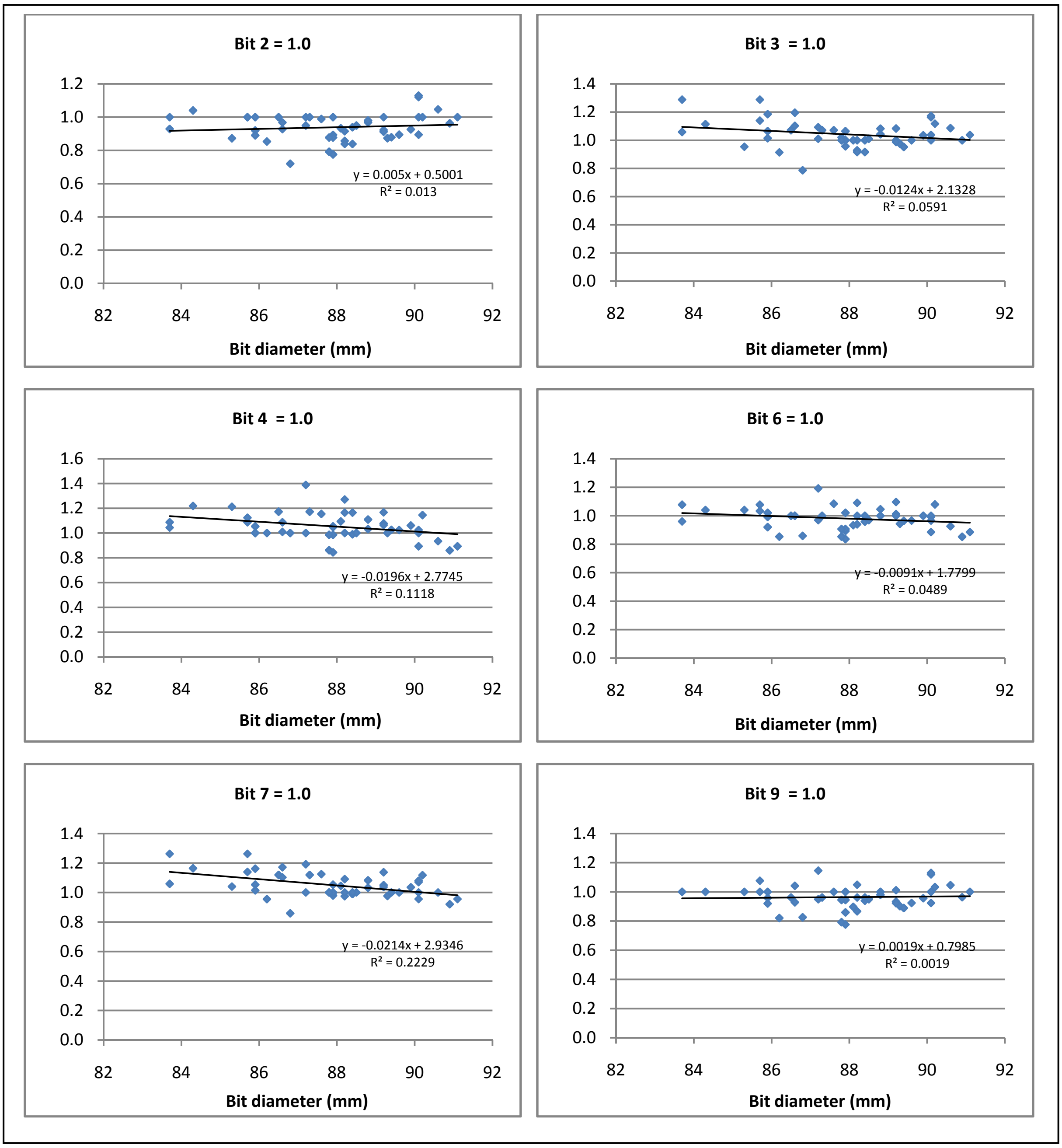
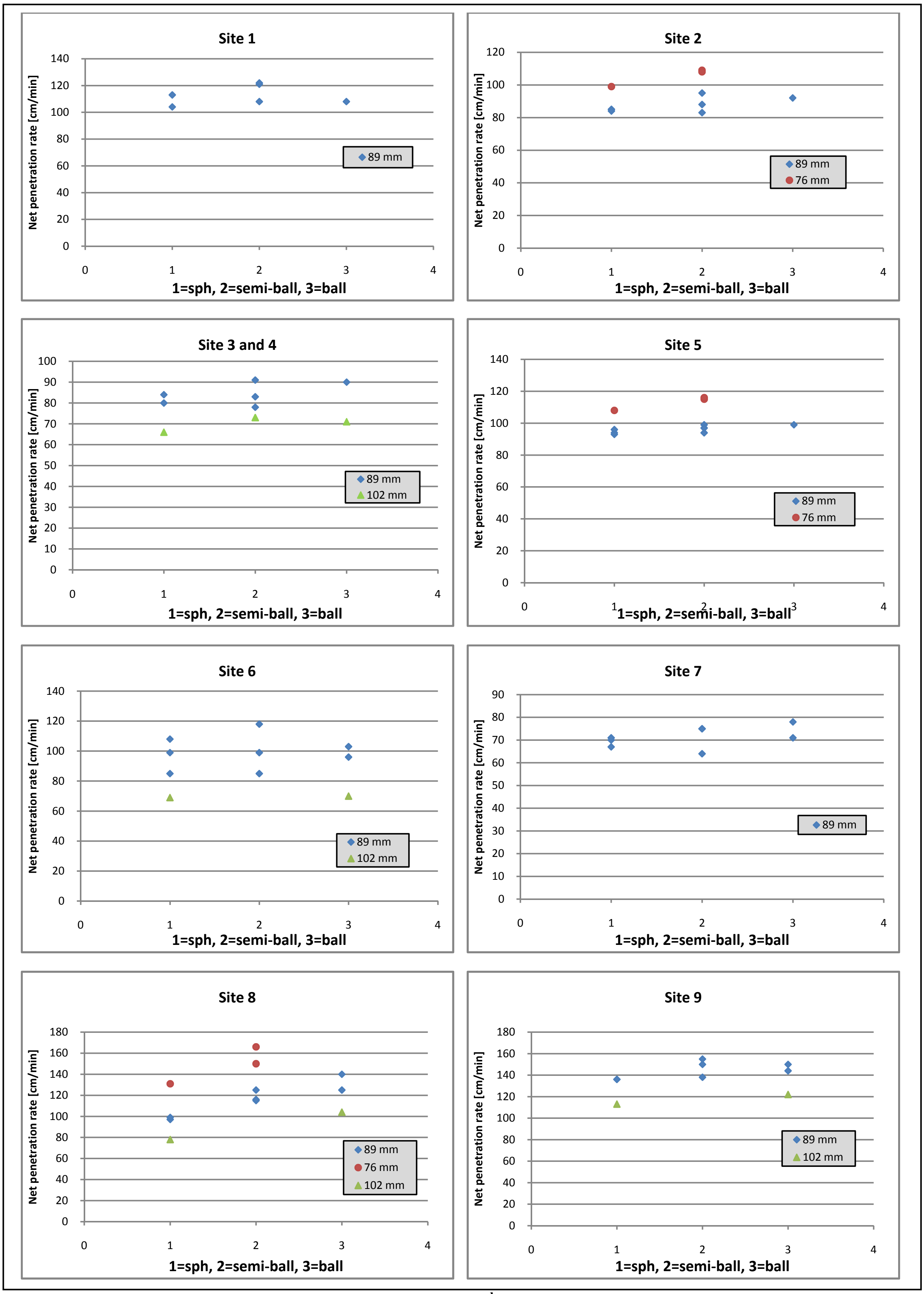


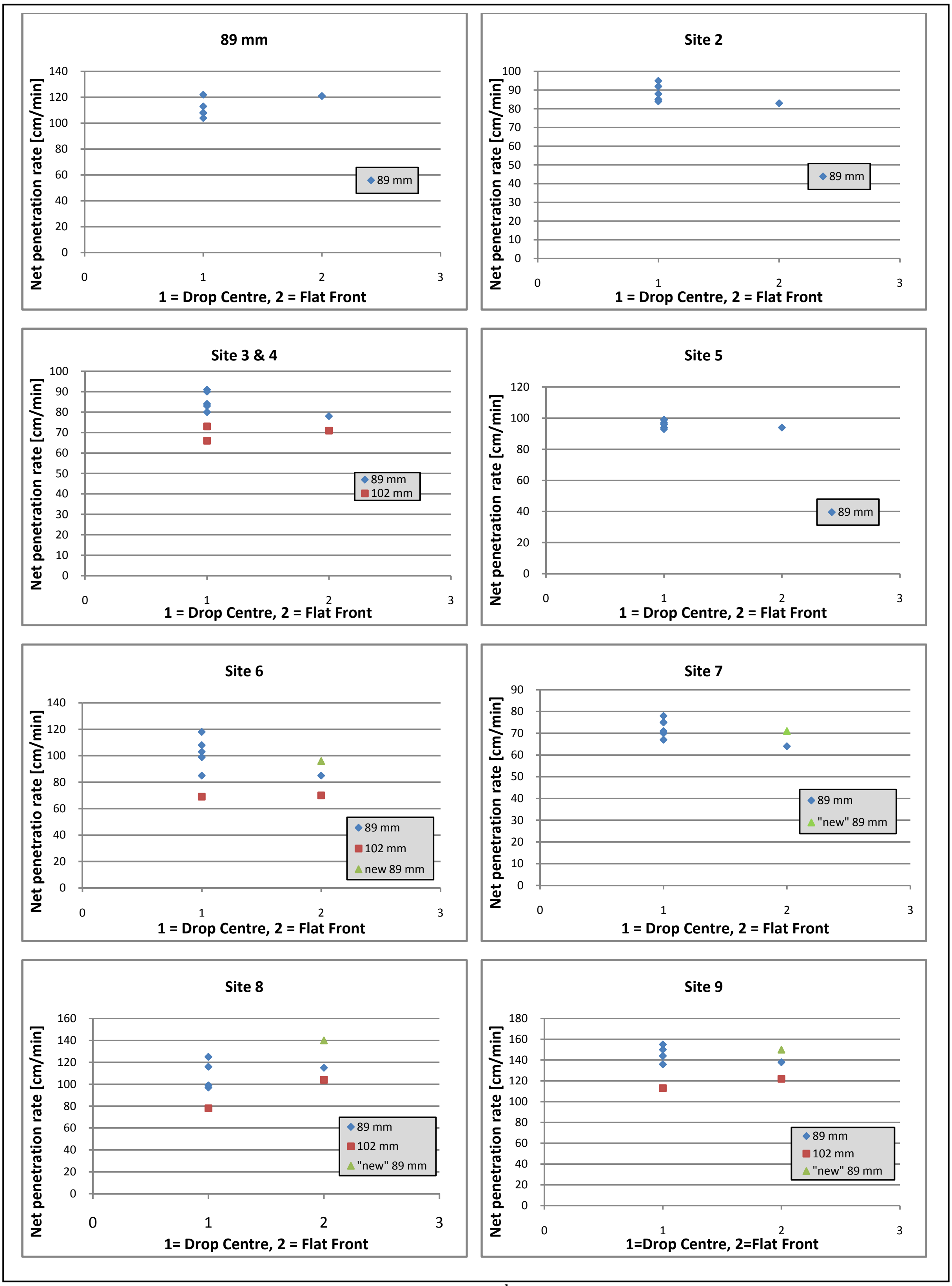
Figure G.3 **Relative net penetration rate values as a function of drill bit diameter for the 89 mm category drill bits for all sites together. Each graph represents each drill bit as the 1.0 reference level. Each point represents the net penetration rate for the 2nd rod for each bit. (P_{10} regression value at 4.8 m hole depth).**

APPENDIX H NET PENETRATION RATE VS. BUTTON SHAPE



Each point represents the net penetration rate for the 2nd rod for each bit.
(P_{10} regression value at 4.8 m hole depth).

APPENDIX I NET PENETRATION RATE VS. FRONT DESIGN



Each point represents the net penetration rate for the 2nd rod for each bit.
(P_{10} regression value at 4.8 m hole depth).

APPENDIX J OVERVIEW OF SUPPLEMENTARY DEVIATION MEASUREMENTS

Location data		Avg. hole length	No. holes	Drill bit diameter	Avg. deflection	Std. deviation
Quarry 1	Blast 1 - DTH	14.2 m	25	140 mm	0.7 %	0.4 %
	Blast 2 - Top hammer	13.5 m	12	89 mm	4.0 %	2.0 %
	Blast 3 - Top hammer	20.3 m	25	89 mm	5.7 %	3.2 %
Quarry 2	Top hammer	11.5 m	16	70 mm	4.3 %	2.0 %
Quarry 3	Blast 1 - Top hammer	15.8 m	57	76 mm	4.1 %	3.0 %
	Blast 2 - Top hammer	20.0 m	32	76 mm	3.9 %	2.4 %
Quarry 4	Top hammer	16.1 m	46	102 mm	3.6 %	2.1 %
	DTH	15.1 m	5	100 mm	2.4 %	2.0 %
Quarry 5	Top hammer	11.0 m	71	89 mm	4.5 %	2.1 %
	Top hammer	14.0 m	14	89 mm	4.9 %	2.5 %
Quarry A (Site 2)	Coprod	18.5 m	38	105 mm	3.7 %	2.1 %
Quarry B (Site 7)	Top hammer	14.5 m	54	89 mm	6.7 %	2.5 %
	Top hammer	15.0 m	52	89 mm	6.8 %	2.5%
	Top hammer	16.0 m	41	89 mm	5.8 %	2.9 %
Quarry B (Site 3 and 4)	Top hammer	16.0 m	44	89 mm	5.0 %	2.7 %
Quarry C (Site 1,5 and 6)	Top hammer	16.5 m	69	89 mm	3.9 %	1.8 %

Table J.1 Deflection at the bottom the holes in percentage of hole depth.
Measurements carried out parallel to the extensive drill bit testing program.
Quarries A, B and C correspond to the drill bit test sites.

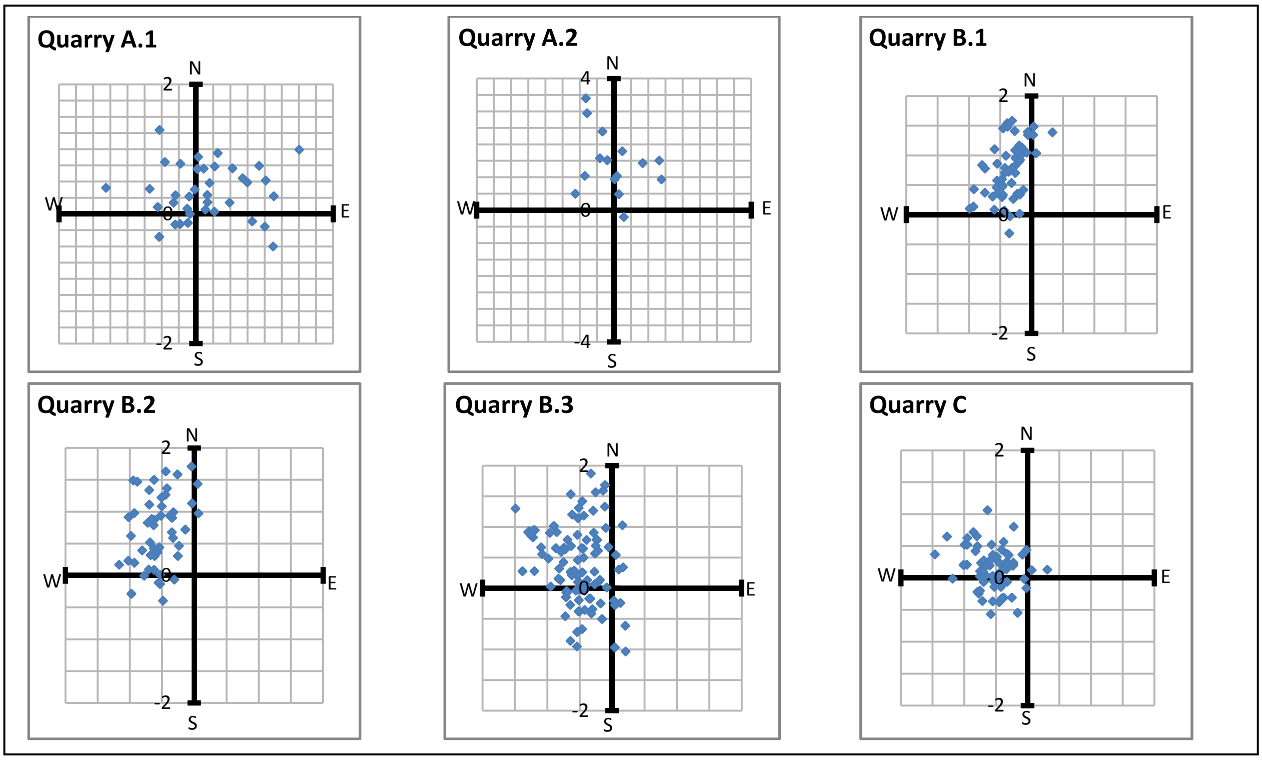


Table J.2 Bulls' Eye-plots of supplementary measurements. N equals Magnetic North. Deflection in metres. The grid has the same scale in both directions.

APPENDIX K SUMMARY OF THE TESTING PROGRAM DEFLECTION MEASUREMENTS

<u>Site 1</u>

Site 1	Bit 2	Bit 3	Bit 4	Bit 6	Bit 7	Bit 8	Bit 9	Bit 11
Number of holes	4	4	4	4	4	4	4	4
Average deflection % [cm/m]	3.20	2.60	3.45	2.68	2.04	4.36	2.88	3.58
Min deflection % [cm/m]	2.50	2.10	2.30	1.10	0.60	2.60	1.30	1.40
Max deflection % [cm/m]	4.30	3.50	4.70	4.40	4.60	6.14	4.10	5.70

Table K.1 Deflection results in percentage of hole depth at 14 m depth at site 1.

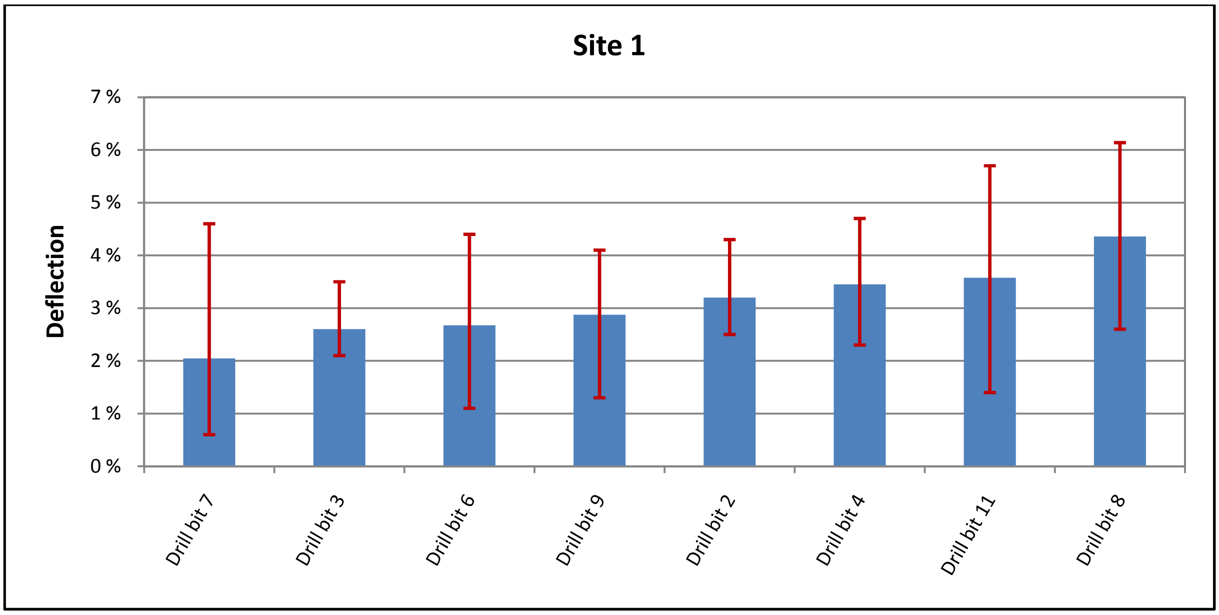


Figure K.1 Test bits' average and max-min deflection values at 14 m hole depth for site 1, in ascending order for average values.

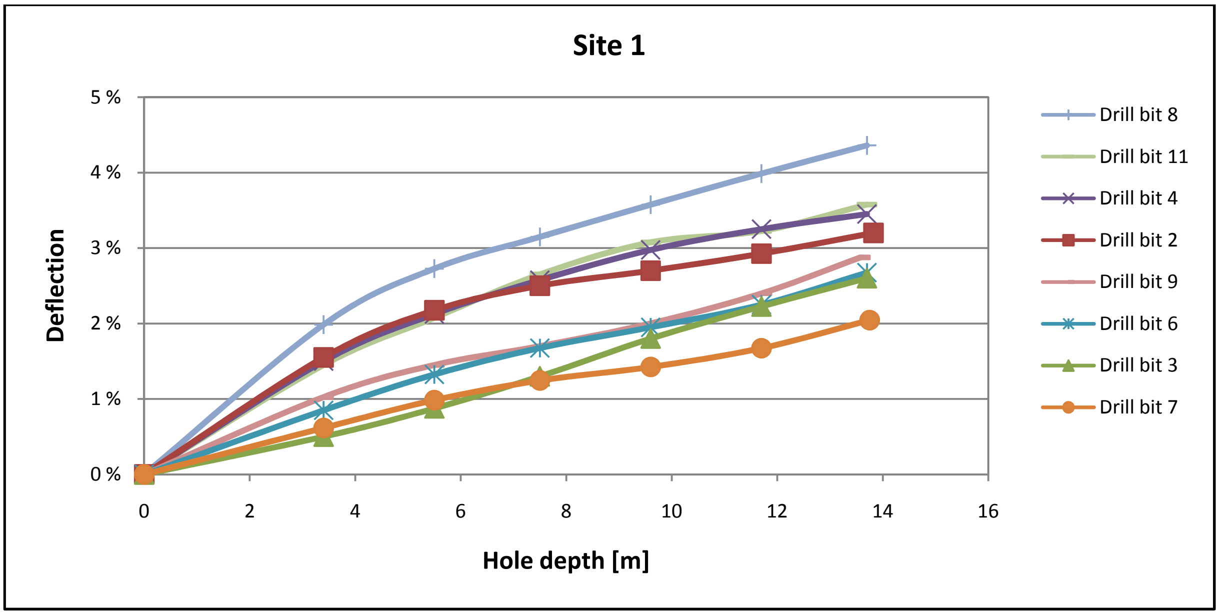


Figure K.2 Test bits' average deflection propagation down the hole for site 1.

Site 2

Site 2	Bit 1	Bit 2	Bit 3	Bit 4	Bit 5	Bit 6	Bit 7	Bit 9	Bit 11	Bit 12
Number of holes	2	2	2	2	2	2	2	2	2	3
Average deflection % [cm/m]	20.6 %	7.2 %	2.5 %	11.1 %	19.7 %	8.7 %	6.4 %	10.8 %	4.1 %	7.5 %
Min deflection % [cm/m]	16.7 %	5.6 %	2.2 %	9.5 %	16.5 %	8.3 %	5.1 %	8.2 %	3.2 %	6.5 %
Max deflection % [cm/m]	24.5 %	8.8 %	4.0 %	12.7 %	22.9 %	9.1 %	7.7 %	13.5 %	5.0 %	8.3 %

Table K.2 Deflection results in percentage of hole depth at 12.5 m depth at site 2.

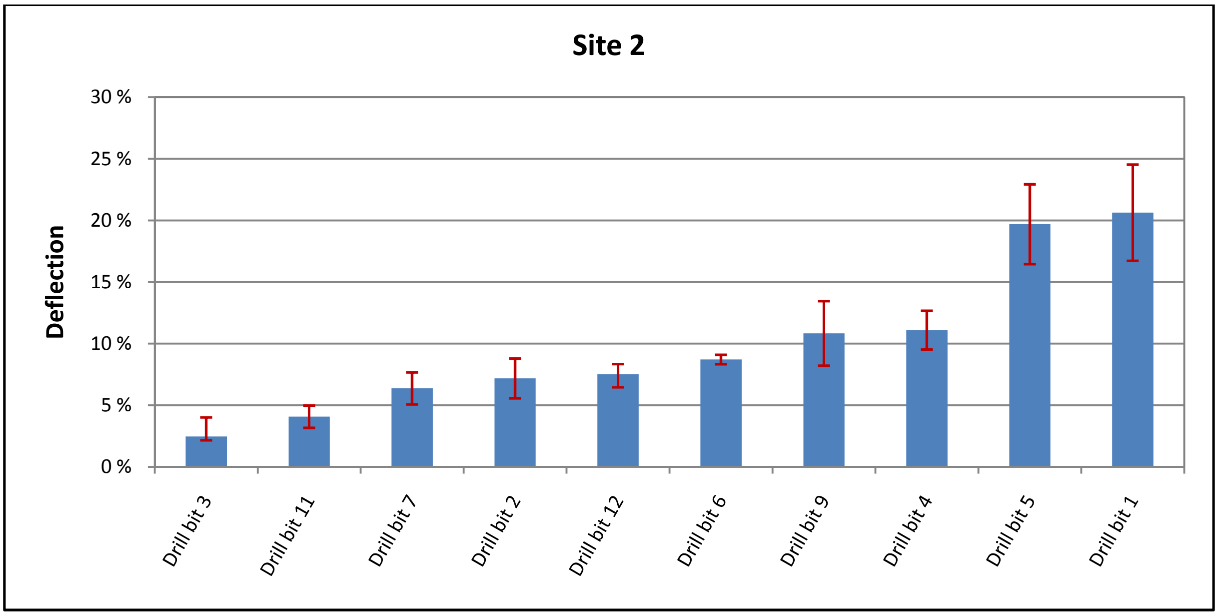


Figure K.3 Test bits' average and max-min deflection values at 12-13 m hole depth for site 2, in ascending order for average values.

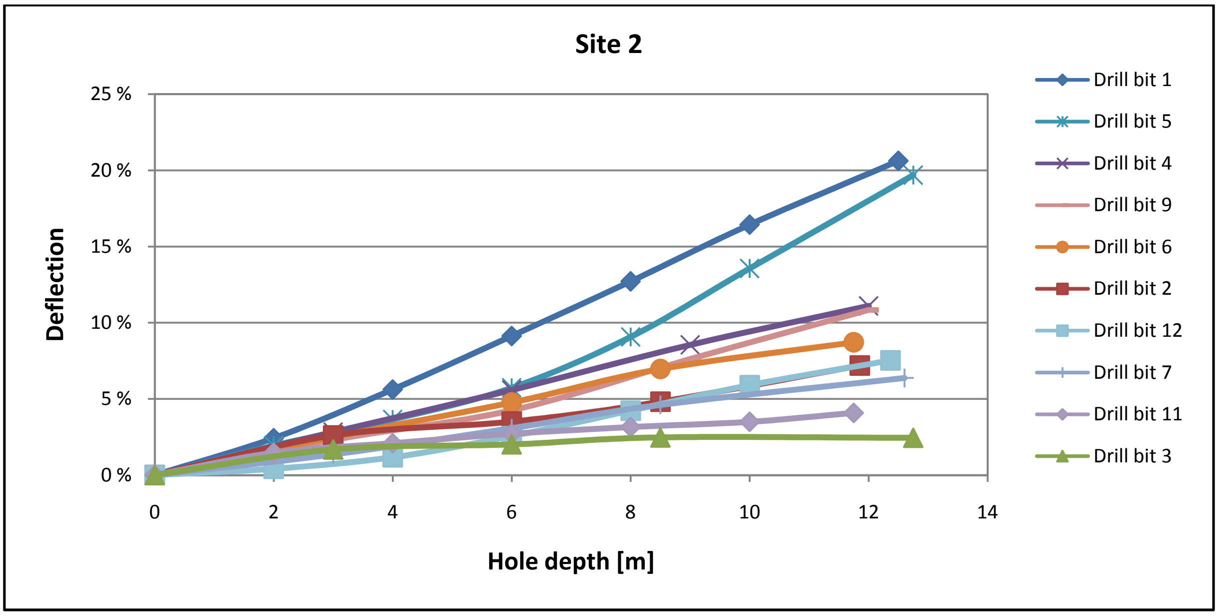


Figure K.4 Test bits' average deflection propagation down the hole for site 2.

Site 3 and 4

Site 3 and 4	Bit 2	Bit 3	Bit 4	Bit 6	Bit 7	Bit 8	Bit 9	Bit 12	Bit 13
Number of holes	6	6	6	6	6	3	6	3	3
Average deflection % [cm/m]	3.4 %	4.0 %	3.7 %	4.0 %	3.8 %	11.3 %	4.3 %	2.6 %	2.7 %
Min deflection % [cm/m]	0.7 %	1.6 %	2.4 %	2.6 %	1.6 %	7.0 %	2.6 %	2.4 %	2.3 %
Max deflection % [cm/m]	7.4 %	6.7 %	4.4 %	5.3 %	7.0 %	16.4 %	7.1 %	3.0 %	3.2 %

Table K.3 Deflection results in percentage of hole depth at 14 m depth at site 4.

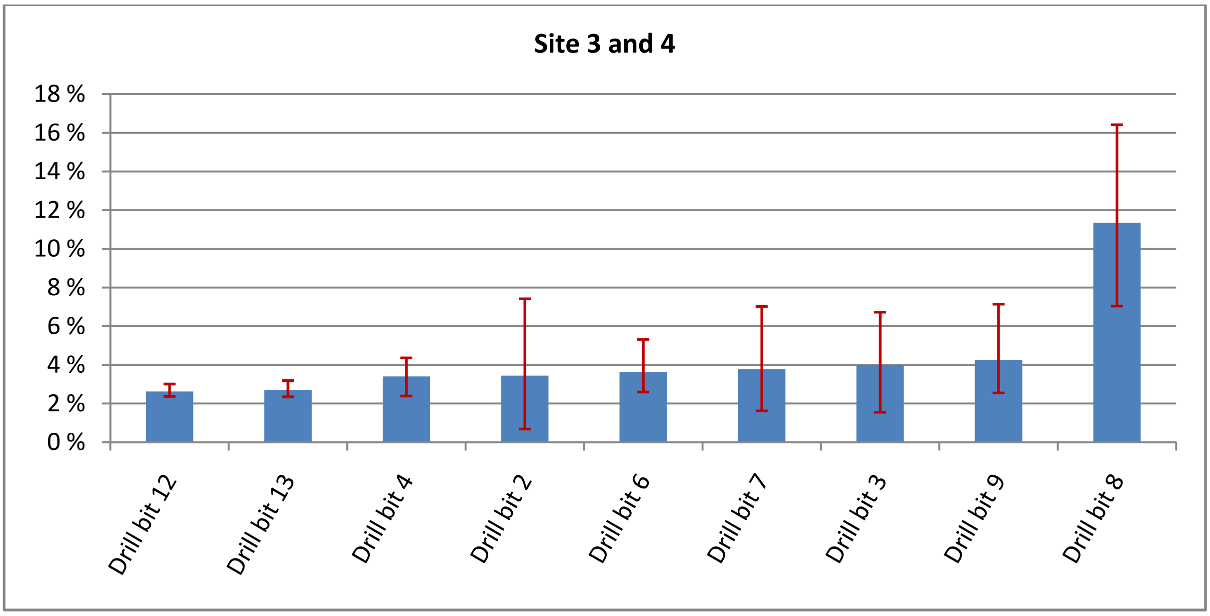


Figure K.5 Test bits' average and max-min deflection values at 14 m hole depth for site 3 and 4, in ascending order for average values.

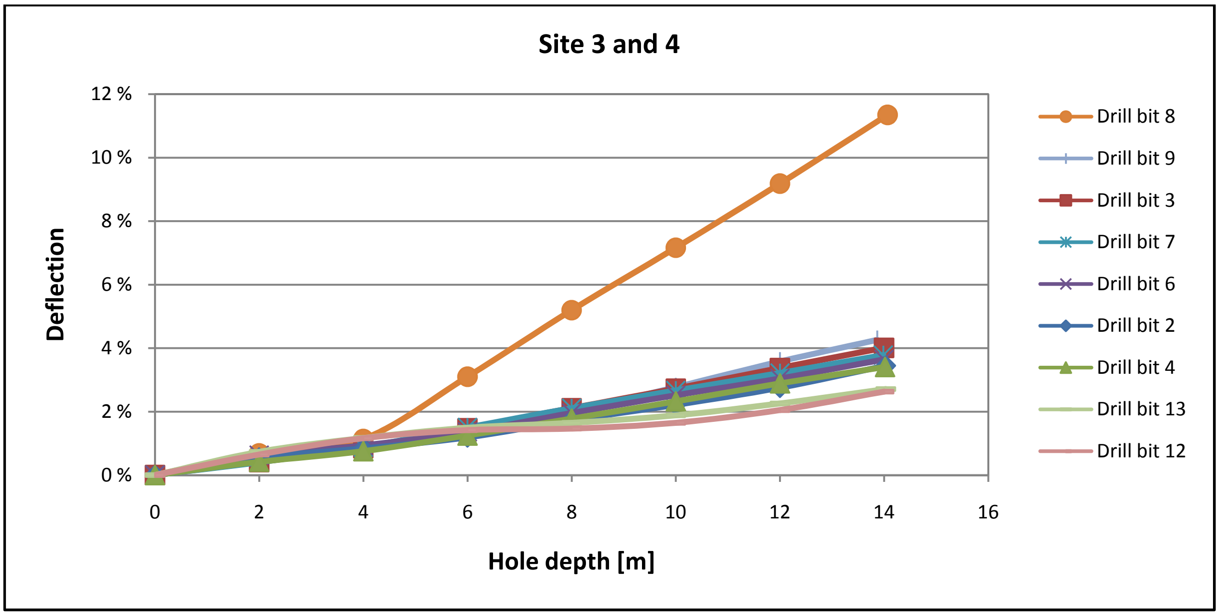


Figure K.6 Test bits' average deflection propagation down the hole for site 3 and 4.

Site 5

Site 5	Bit 1	Bit 2	Bit 3	Bit 4	Bit 5	Bit 6	Bit 7	Bit 9	Bit 10	Bit 11	Bit 12
Number of holes	4	5	6	4	4	5	5	5	4	4	2
Average deflection % [cm/m]	6.4 %	5.0 %	4.8 %	3.3 %	5.0 %	3.6 %	4.7 %	3.0 %	7.0 %	7.1 %	6.0 %
Min deflection % [cm/m]	4.1 %	3.9 %	1.1 %	1.9 %	3.1 %	2.0 %	1.0 %	2.1 %	6.0 %	6.0 %	4.0 %
Max deflection % [cm/m]	7.7 %	6.6 %	7.9 %	5.0 %	7.0 %	4.4 %	11.0 %	4.1 %	8.0 %	9.2 %	7.9 %

Table K.4 **Deflection results in percentage of hole depth at 14 m depth at site 5.**

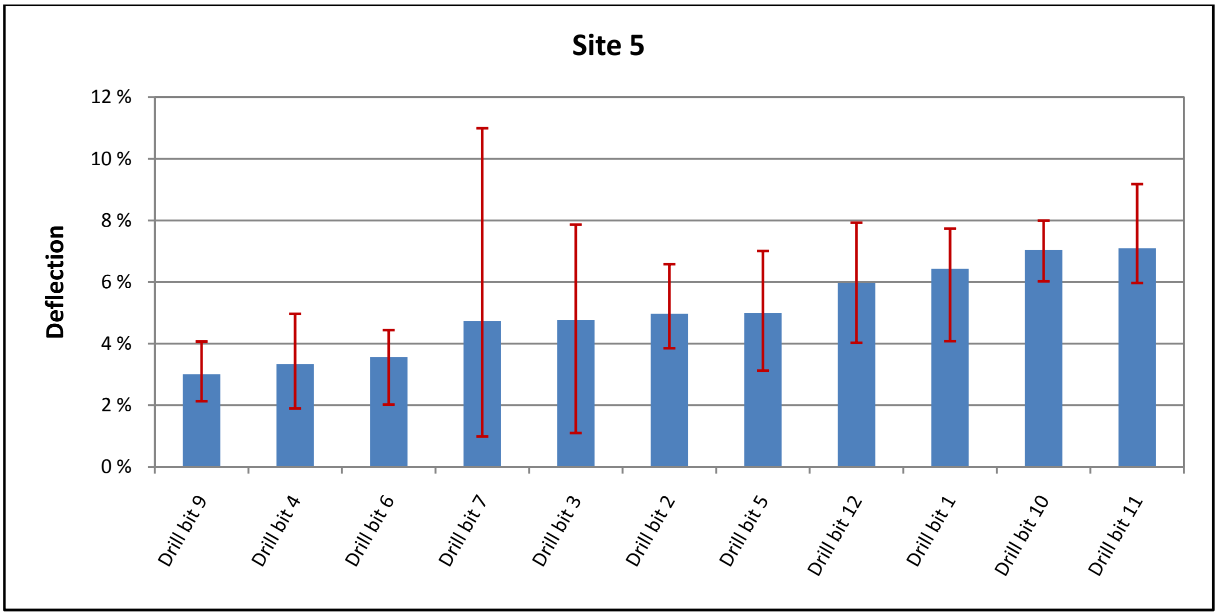


Figure K.7 **Test bits' average and max-min deflection values at 14 m hole depth for site 5, in ascending order for average values.**

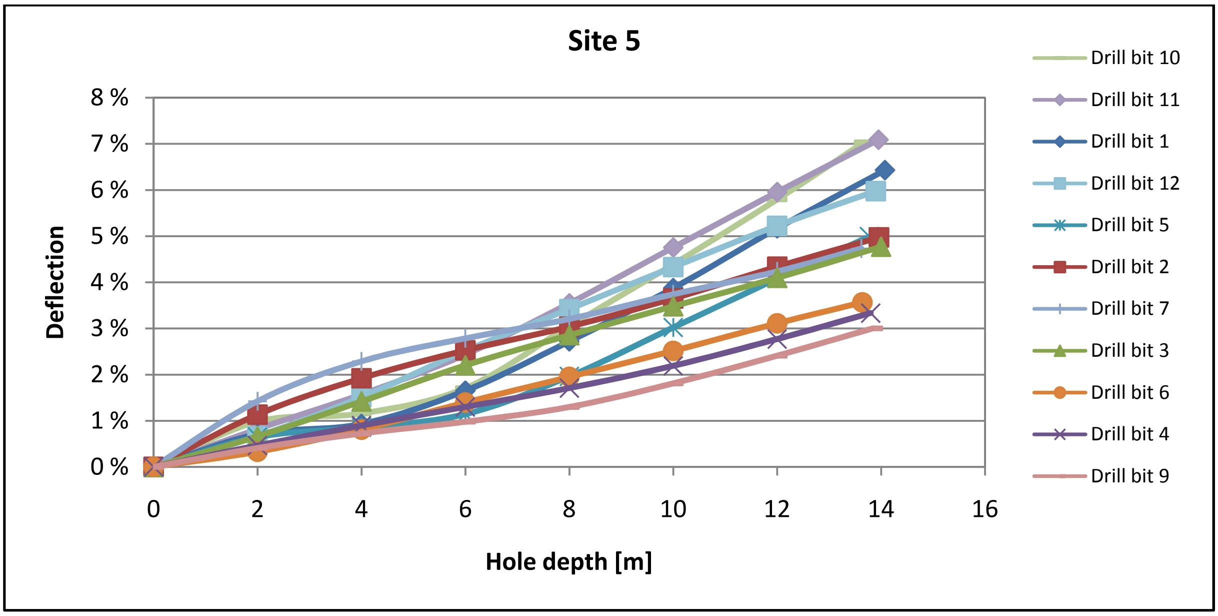


Figure K.8 **Test bits' average deflection propagation down the hole for site 5.**

Site 6

Site 6	Bit 2	Bit 3	Bit 4	Bit 6	Bit 7	Bit 9	Bit 10	Bit 12	Bit 13	Bit 14
Number of holes	3	4	2	4	4	4	4	2	5	3
Average deflection % [cm/m]	2.6 %	2.2 %	1.2 %	2.3 %	2.5 %	2.0 %	2.1 %	1.8 %	2.3 %	1.9 %
Min deflection % [cm/m]	2.1 %	1.4 %	1.0 %	1.0 %	1.9 %	1.2 %	0.9 %	0.6 %	0.9 %	0.9 %
Max deflection % [cm/m]	3.3 %	3.0 %	1.3 %	3.2 %	3.6 %	2.8 %	3.7 %	3.0 %	3.2 %	3.2 %

Table K.5 Deflection results in percentage of hole depth at 12 m depth at site 6.

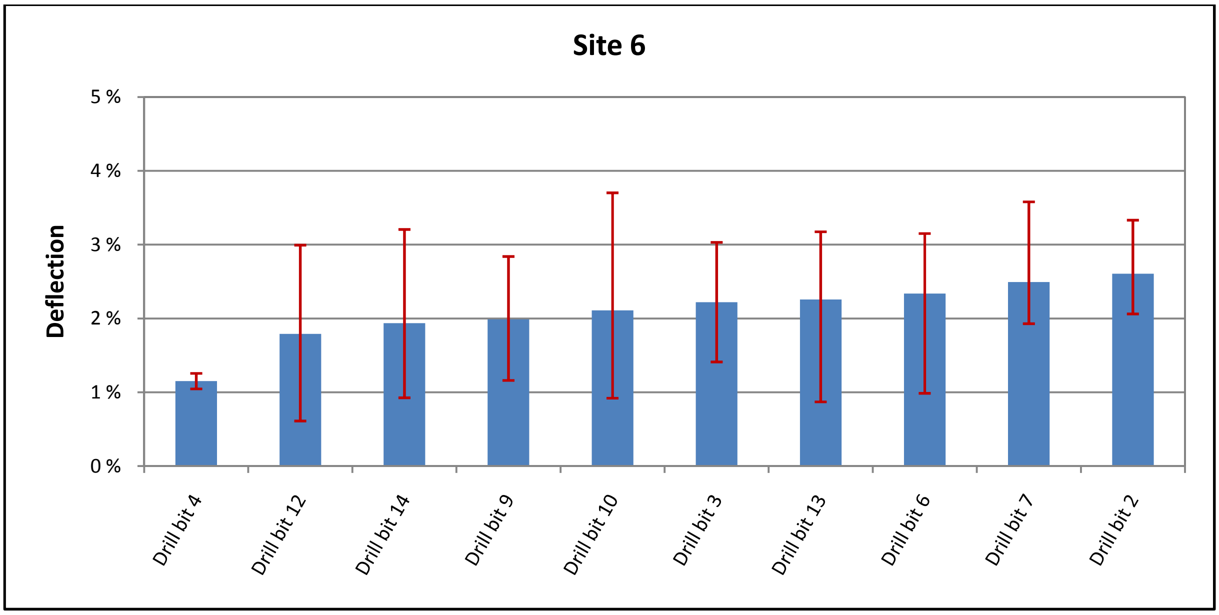


Figure K.9 Test bits' average and max-min deflection values at 12 m hole depth for site 6, in ascending order for average values.

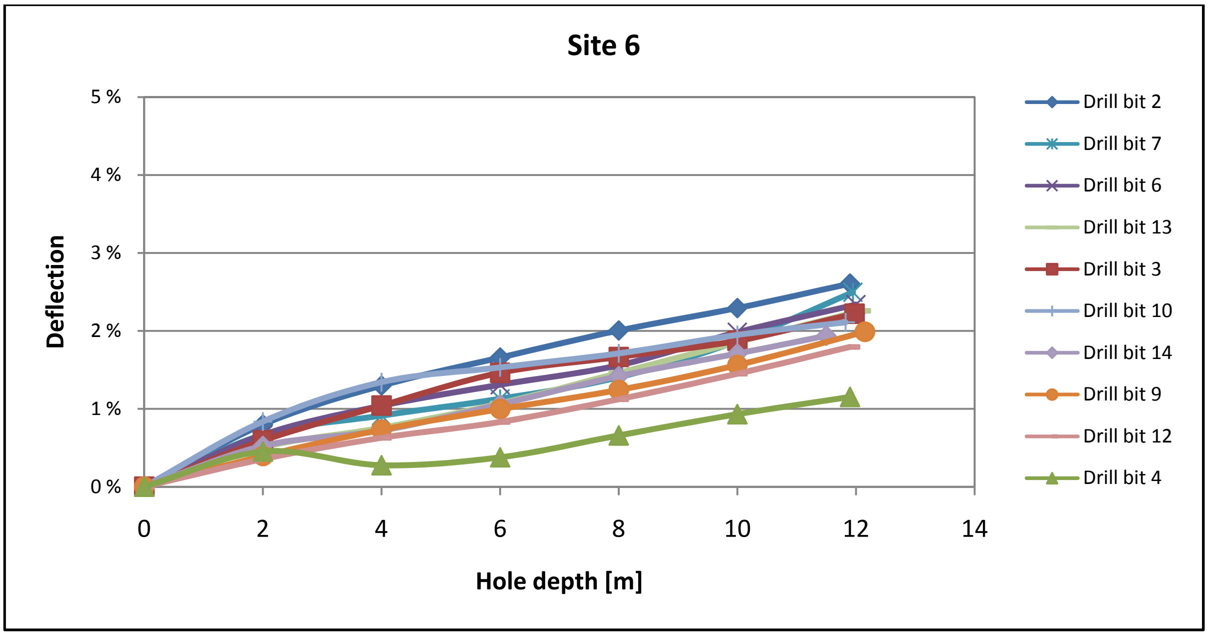


Figure K.10 Test bits' average deflection propagation down the hole for site 6.

Site 7

Site 7	Bit 2	Bit 3	Bit 4	Bit 6	Bit 7	Bit 9	Bit 10	Bit 14
Number of holes	3	3	3	3	3	4	3	3
Average deflection % [cm/m]	6.5 %	4.3 %	4.4 %	3.7 %	1.9 %	3.0 %	1.8 %	5.7 %
Min deflection % [cm/m]	5.3 %	2.6 %	2.4 %	2.4 %	1.1 %	1.3 %	0.8 %	3.2 %
Max deflection % [cm/m]	7.9 %	6.6 %	7.9 %	4.5 %	2.7 %	4.3 %	2.8 %	8.6 %

Table K.6 Deflection results in percentage of hole depth at 12 m depth at site 7.

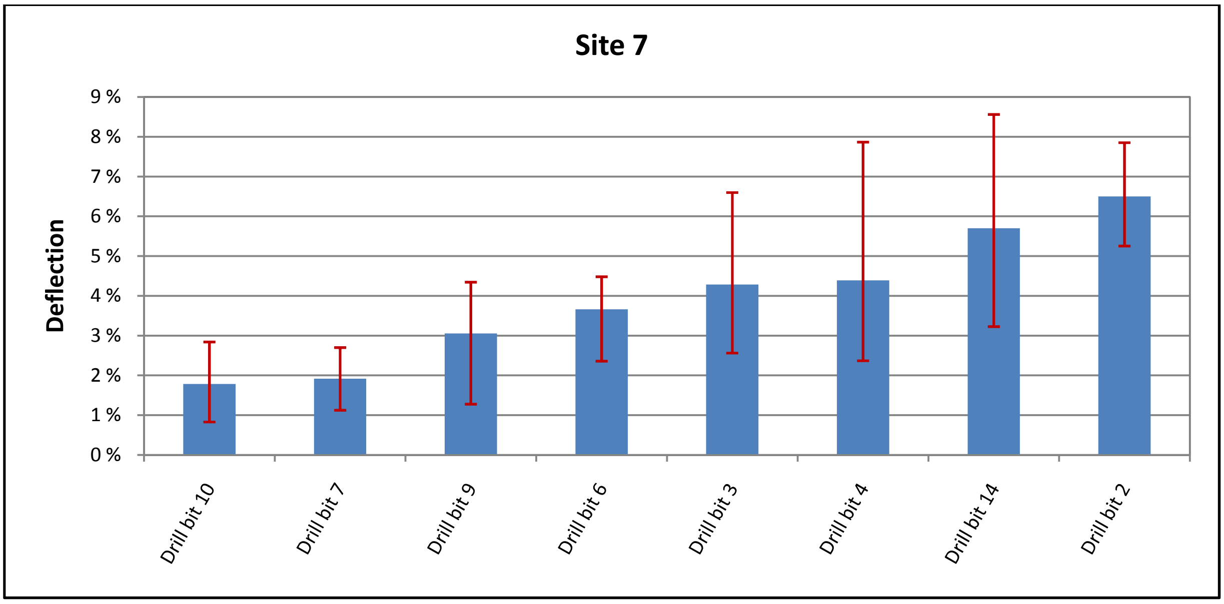


Figure K.11 Test bits' average and max-min deflection values at 12 m hole depth for site 7, in ascending order for average values.

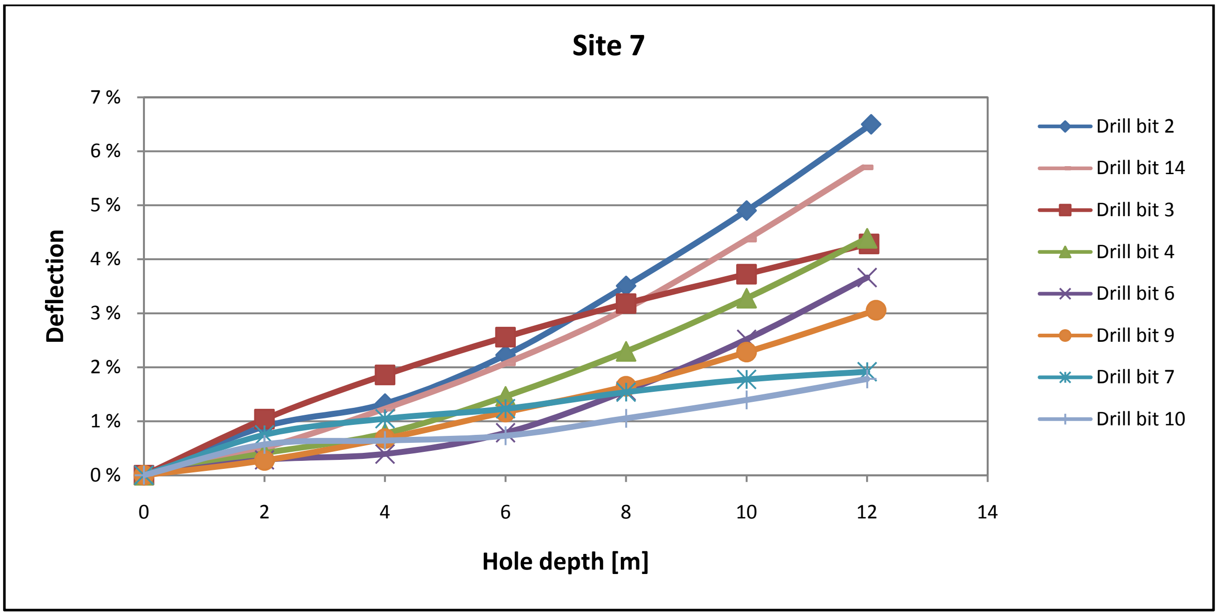


Figure K.12 Test bits' average deflection propagation down the hole for site 7.

Site 8

Site 8	Bit 1	Bit 2	Bit 3	Bit 4	Bit 5	Bit 6	Bit 7	Bit 9	Bit 11	Bit 12	Bit 13	Bit 14	Bit X
Number of holes	2	3	3	3	3	3	2	3	2	3	3	3	2
Average deflection % [cm/m]	7.8 %	5.5 %	8.6 %	4.7 %	10.9 %	1.1 %	13.6 %	2.8 %	12.1 %	4.4 %	9.9 %	4.0 %	6.9 %
Min deflection % [cm/m]	7.7 %	4.8 %	5.4 %	2.0 %	4.6 %	1.0 %	13.6 %	1.0 %	10.6 %	3.6 %	7.6 %	2.6 %	6.8 %
Max deflection % [cm/m]	7.8 %	6.0 %	11.1 %	8.0 %	16.3 %	1.4 %	13.7 %	4.5 %	13.5 %	4.9 %	12.3 %	4.9 %	7.0 %

Table K.7 Deflection results in percentage of hole depth at 17 m depth at site 8.

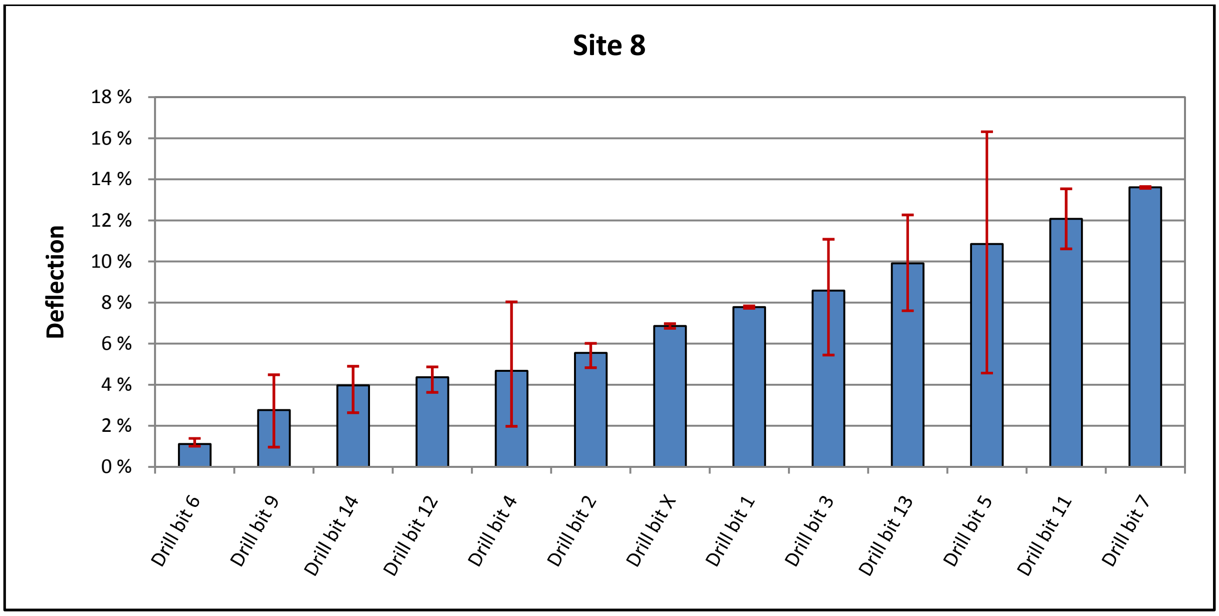


Figure K.13 Test bits' average and max-min deflection values at 17 m hole depth for site 8, in ascending order for average values.

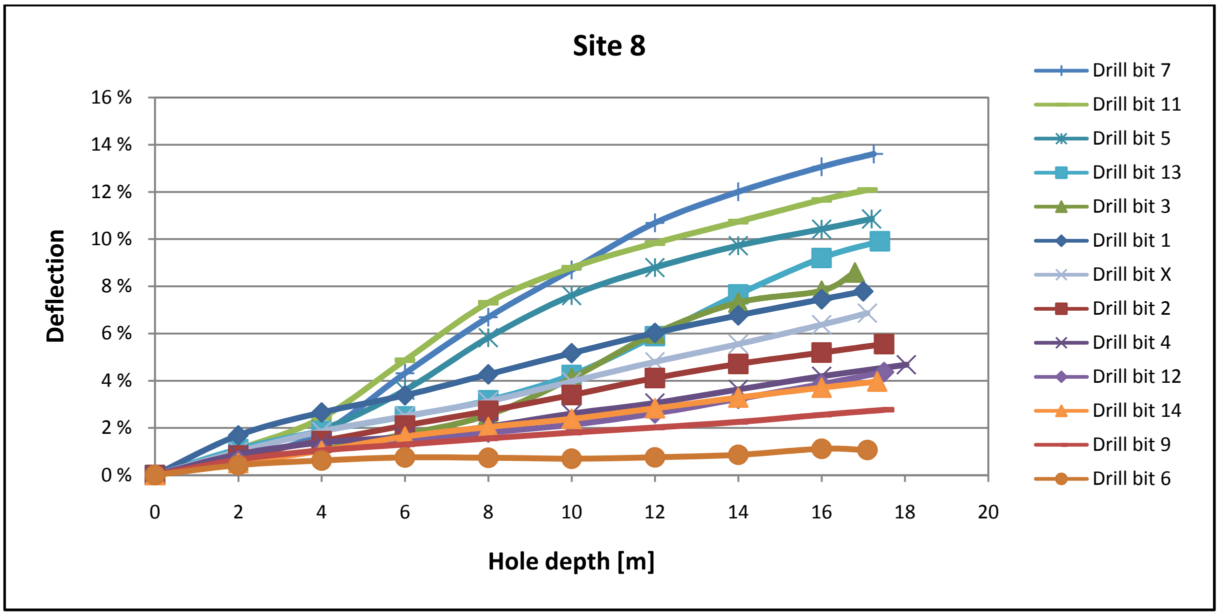


Figure K.14 Test bits' average deflection propagation down the hole for site 8.

Site 9

Site 9	Bit 2	Bit 3	Bit 4	Bit 6	Bit 7	Bit 9	Bit 12	Bit 13	Bit 14	Bit X
Number of holes	3	3	3	3	3	3	3	3	3	2
Average deflection % [cm/m]	3.2 %	6.0 %	3.9 %	2.6 %	6.5 %	1.6 %	6.8 %	3.3 %	3.4 %	4.5 %
Min deflection % [cm/m]	2.9 %	3.6 %	1.9 %	1.9 %	4.6 %	1.4 %	5.7 %	1.5 %	2.8 %	4.4 %
Max deflection % [cm/m]	3.5 %	9.9 %	6.9 %	3.1 %	10.0 %	1.8 %	8.0 %	6.7 %	3.9 %	4.7 %

Table K.8 Deflection results in percentage of hole depth at 14 m depth at site 9.

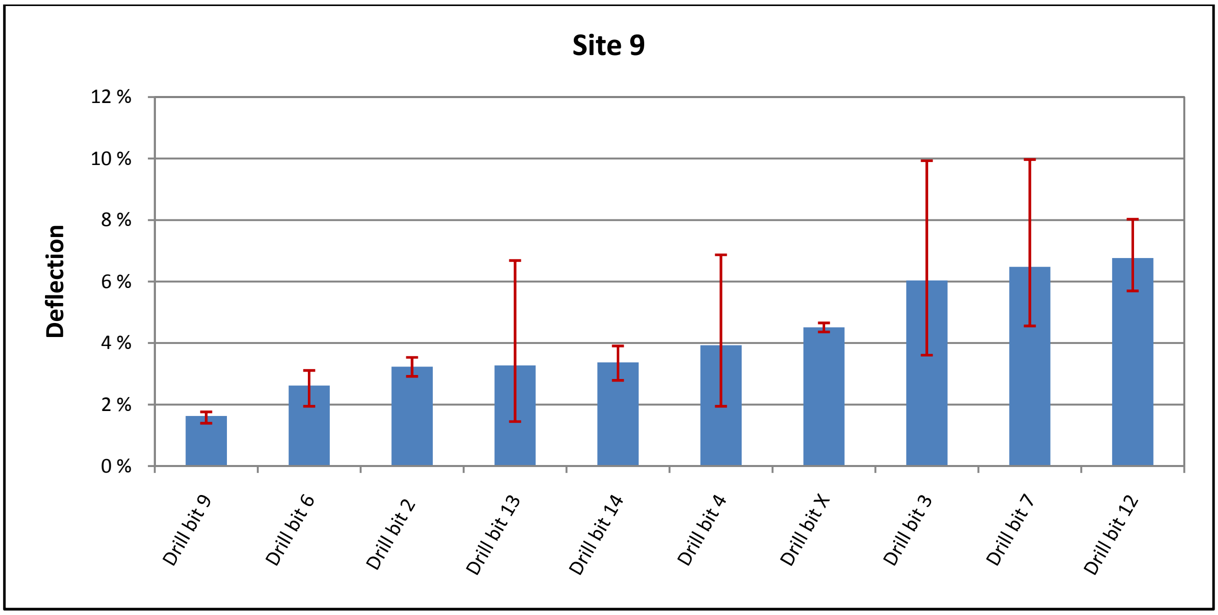


Figure K.15 Test bits' average and max-min deflection values at 14 m hole depth for site 9, in ascending order for average values.

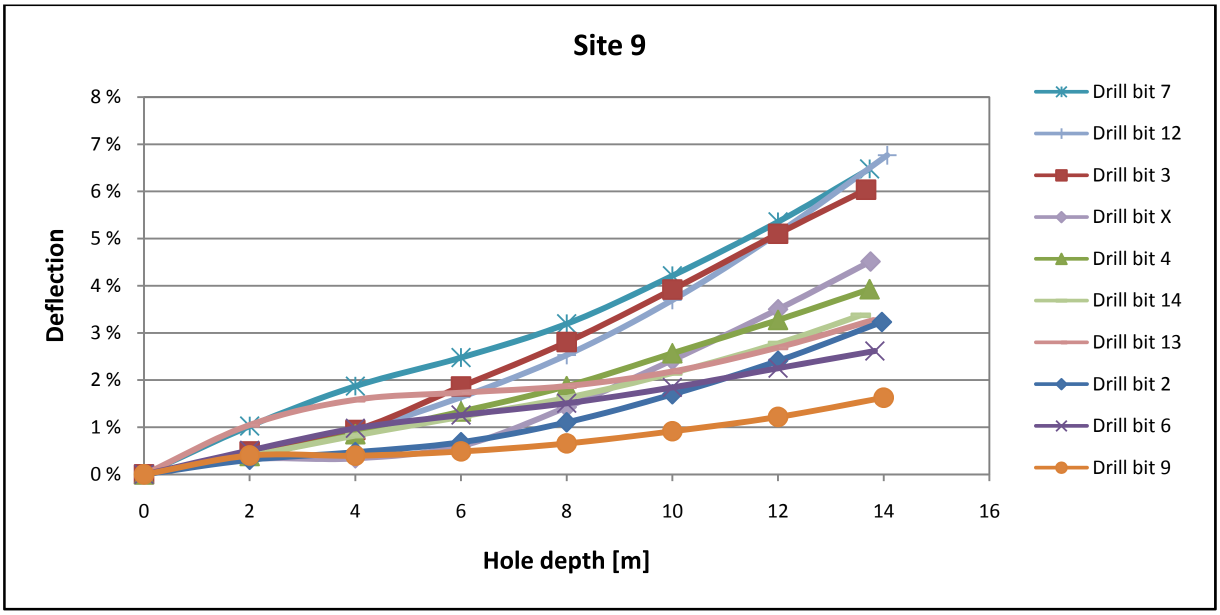


Figure K.16 Test bits' average deflection propagation down the hole for site 9.

APPENDIX L DEFLECTION VS. DRILL BIT AND ROD DIAMETER

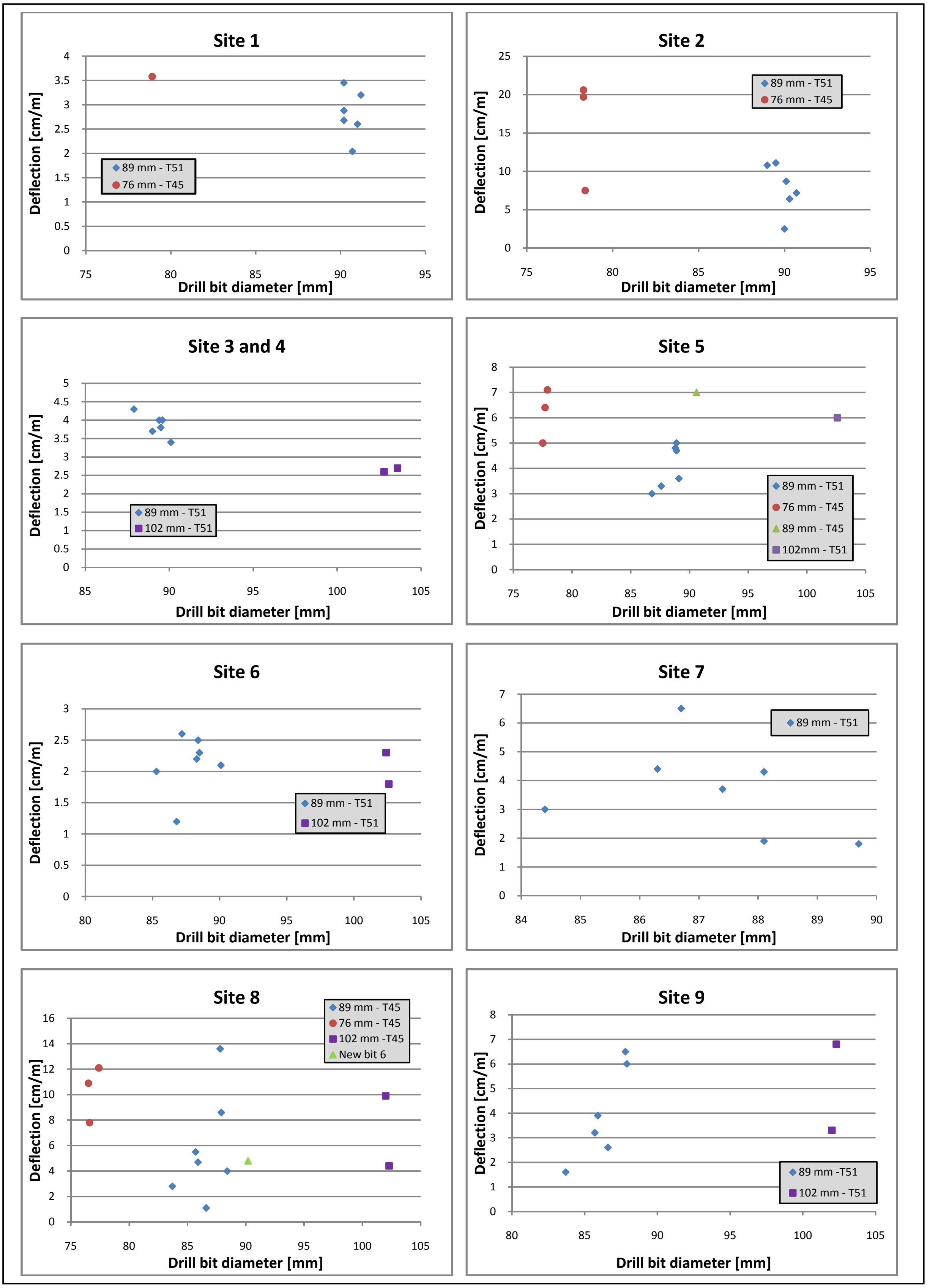


Figure L.1 **Deflection as a function of drill bit diameter and rod diameter (in the text boxes). Each point represents the average cumulated bottom deflection of each bit.**

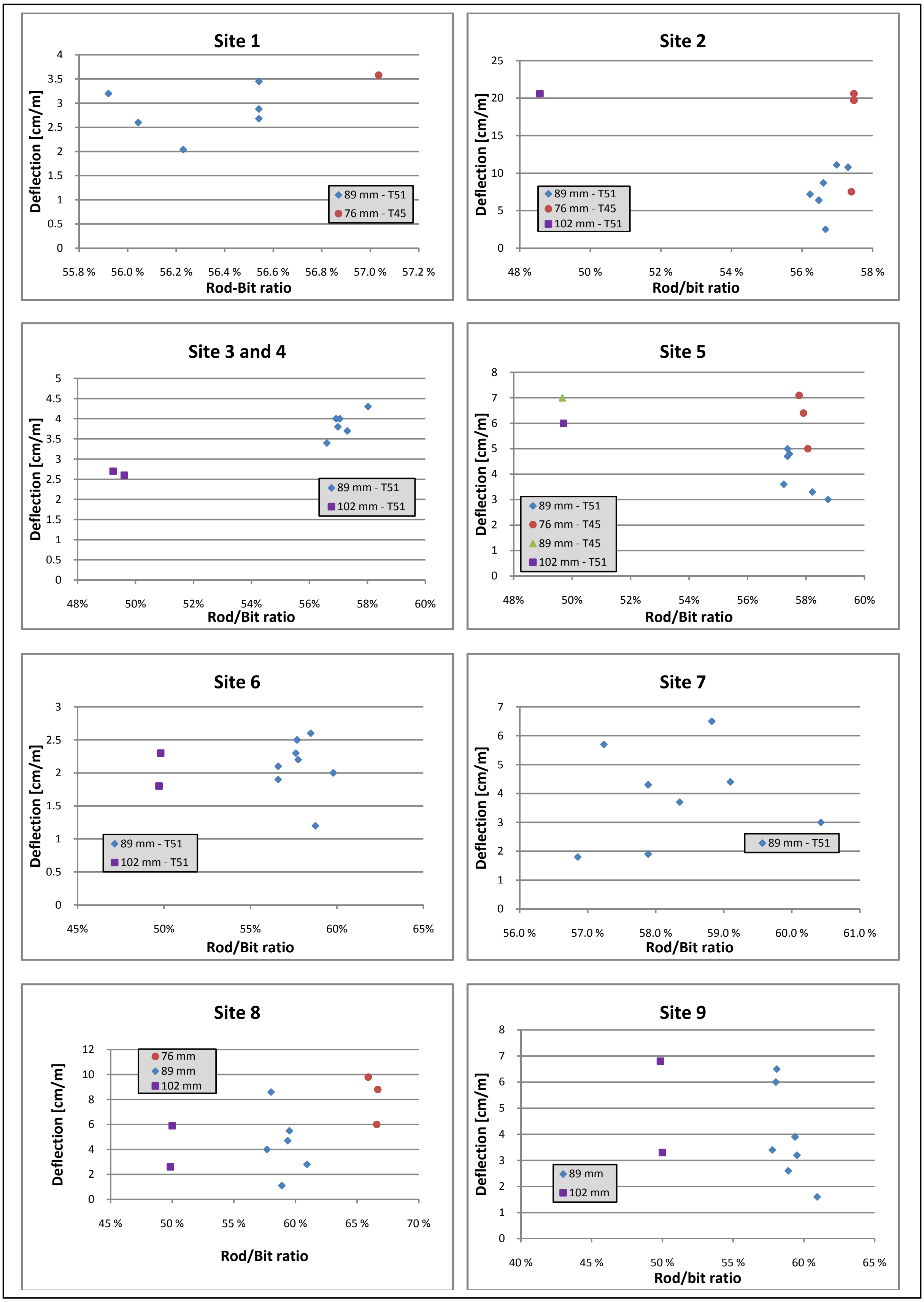
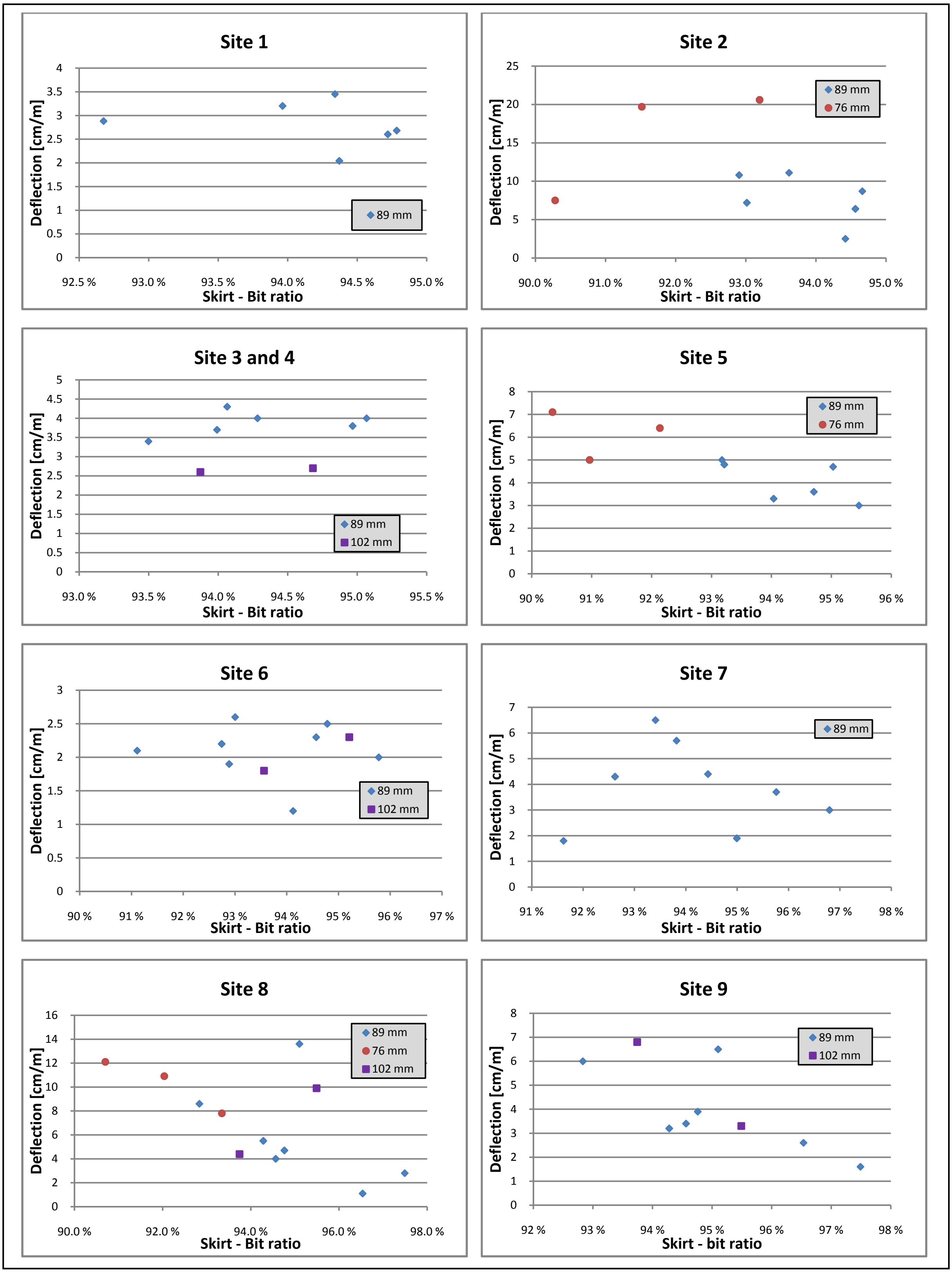


Figure L.2 **Deflection as a function of the rod-bit ratio (i.e. rod diameter divided by bit diameter). Each point in represents the average cumulated bottom deflection of each bit.**

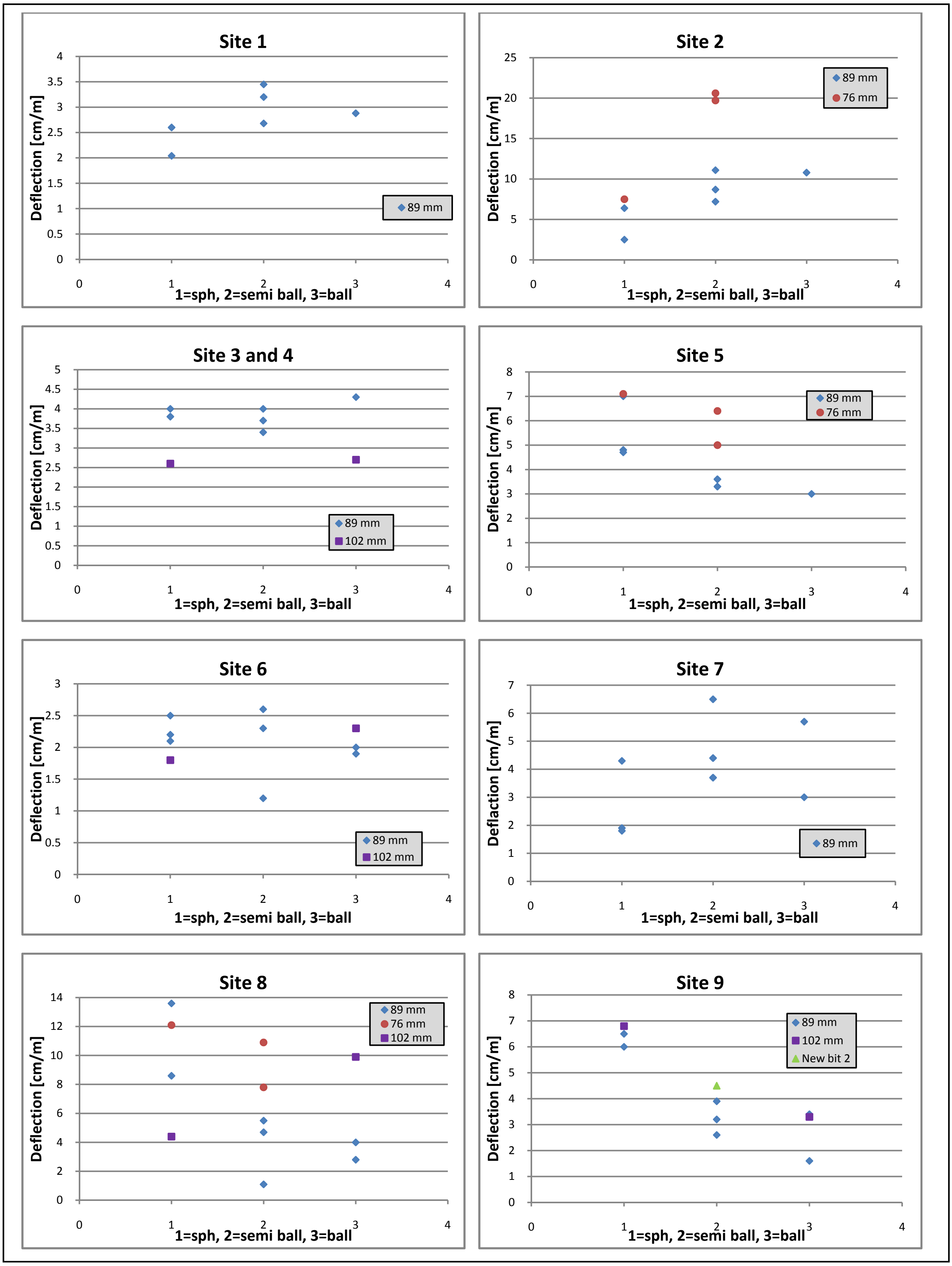
APPENDIX M DEFLECTION VS. THE SKIRT AND BIT DIAMETER RATIO



The skirt-bit ratio equals the skirt diameter divided by the actual bit diameter.

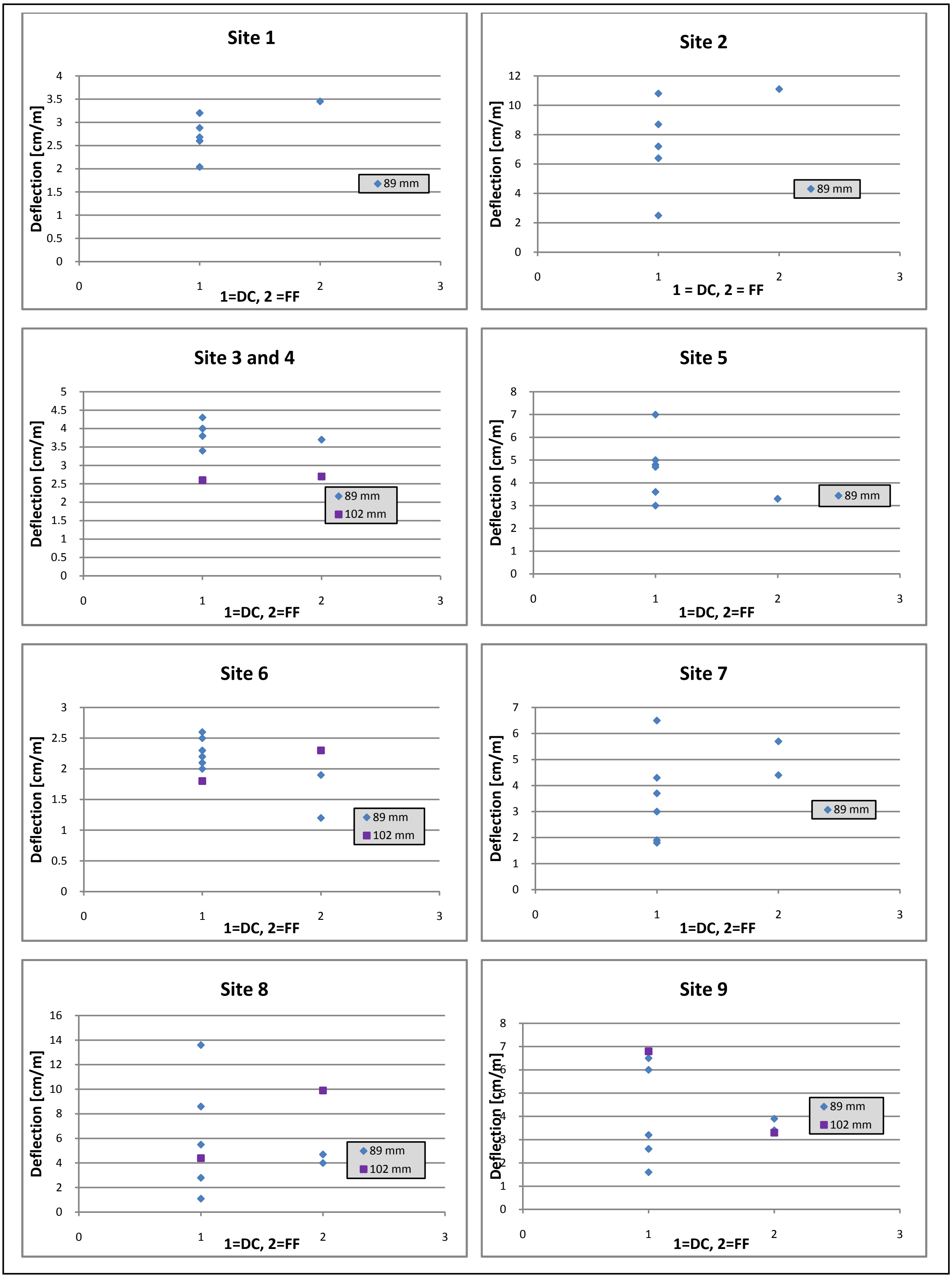
Each point represents the average cumulated bottom deflection at of each bit.

APPENDIX N DEFLECTION VS. BUTTON SHAPE



The button shape varies between spherical, semi ballistic and ballistic, respectively the categories 1, 2 and 3 in the figures. Each point in the figures represents the average bottom deflection of each bit.

APPENDIX O DEFLECTION VS. DRILL BIT FRONT DESIGN



DC = drop centre, FF = Flat front.

Each point in the figures represents the average cumulated bottom deflection of each bit.

APPENDIX P OVERVIEW OF ALL COLLARING AND ALIGNMENT DEVIATION MEASUREMENTS

	Site 1 (Quarry C)	Site 2	Site 3 (Quarry B)		Site 4 (Quarry B)	Site 5 (Quarry C)		Site 6 (Quarry C)	Site 7 (Quarry B)	Site 8	Site 9	Quarry B Blast 1	Quarry B Blast 2	Quarry C	Quarry X[2)]
Floor conditions	Cleaned	Cleaned	Rock debris	Cleaned	Rock debris[1)]	Rock debris	Cleaned	Cleaned	Cleaned	Terrain	Cleaned	Cleaned	Cleaned	Rock Debris	Rock Debris
Number of holes	32	32	21	17	33	30	30	35	31	35	29	54	50	69	193
COLLARING DEVIATION															
BURDEN Planned [m]	-	-	2.50		2.50	2.90		2.80	2.50	-	3.00	2.50	2.50	2.90	81
Average deviation [m]	-	-	±0.12		±0.29	±0.15		±0.09	±0.16	-	±0.18	±0.14	±0.11	±0.21	7.0 m
Maximum deviation	-	-	2.79		2.83	3.39		3.00	2.85	-	3.36	2.94	2.83	3.42	±0.11
Minimum deviation	-	-	2.23		2.21	2.67		2.58	2.30	-	2.69	2.03	2.30	2.29	7.29 m
Standard deviation	-	-	0.14		0.24	0.17		0.11	0.16	-	0.22	0.19	0.12	0.28	6.76 m
SPACING Planned [m]	-	-	3.20		3.20	3.80		3.50	3.20	-	4.00	3.20	3.20	3.8	0.14 m
Average deviation [m]	-	-	±0.07		±0.09	±0.07		±0.07	±0.20	-	±0.12	±0.07	±0.04	±0.06	9.0 m
Maximum deviation	-	-	3.33		3.47	3.99		3.66	3.71	-	4.32	3.36	3.28	3.97	±0.25m
Minimum deviation	-	-	2.94		2.98	3.56		3.32	2.84	-	3.71	3.04	3.11	3.68	9.67 m
Standard deviation	-	-	0.09		0.11	0.09		0.09	0.24	-	0.15	0.08	0.05	0.08	8.18 m
DRILL PATTERN AREA Planned [m^2]	-	-	8.0		8.0	11.0		9.8	8.0	-	12.0	8.0	8.0	11.0	0.32 m
Average deviation	-	-	±6.5 %		±8.1 %	±5.9 %		±4.3 %	±4.6 %	-	±6.4 %	±5.8 %	±4.6 %	±5.9 %	4.3 %
Maximum deviation	-	-	9.6 %		16.4 %	10.3 %		6.3 %	11.5 %	-	12.2 %	17.4 %	12.5 %	12.9 %	9.2 %
Minimum deviation	-	-	-14.2 %		1.9 %	-11.1 %		-14.0 %	-8.7 %	-	-11.3	-15.8%	-8.2 %	-17.3 %	12.7 %
Standard deviation	-	-	7.0 %		4.8 %	6.2 %		5.9 %	6.2 %	-	7.7	7.3 %	5.1 %	7.7 %	4.3 %
ALIGNMENT DEVIATION															
INCLINATION Planned [deg]	-80	-87	-80	-80	-80	-80	-80	-80/-82	-80	-80	-75	-80	-80	-80	-
Average deviation [cm/m]	1.6	2.3	2.1	2.9	1.6	2.9	5.0	1.9	1.3	1.5	0.9	1.9	1.5	2.8	-
Maximum deviation [cm/m]	6.0	6.1	6.3	10.2	7.2	7.8	8.2	5.7	3.1	4.8	2.4	4.8	5.8	7.0	-
Standard deviation	1.5	1.5	1.7	2.4	1.6	2.1	1.8	1.5	1.0	1.3	0.8	1.3	1.3	1.5	-
Inclination angle span	5.7°	6.1°	5.2°	8.2°	5.7°	5.5°	3.6°	4.1°	3.5°	5.2°	1.8°	5.2°	5.6°	6.5°	-
HORIZONTAL DIRECTION (azimuth)															
Average deviation [cm/m]	2.3	2.2	1.7	3.1	1.6	1.1	3.0	1.4	2.1	2.0	1.2	2.4	1.0	1.3	-
Maximum deviation [cm/m]	8.8	9.4	5.4	9.9	4.1	5.0	9.0	4.6	5.0	6.2	5.1	8.2	4.3	4.0	-
Standard deviation	2.1	2.2	1.5	2.7	1.2	0.9	1.9	1.1	1.4	1.7	1.1	1.8	0.9	0.9	-
Azimuth span	59°	180°	25°	46°	26°	34°	35°	25°	30°	46°	18°	40°	19°	30°	-
TOTAL ALIGNMENT															
Average deviation [cm/m]	3.1	3.6	2.9	4.6	2.4	3.3	6.0	2.6	2.6	2.8	1.7	3.4	2.0	2.5	-
Maximum deviation [cm/m]	8.8	11.2	6.7	14.2	7.4	7.9	9.7	6.5	5.3	6.9	5.2	8.3	5.9	6.9	-
Standard deviation	2.1	2.1	1.8	3.2	1.7	2.0	1.7	1.6	1.4	1.7	1.1	1.6	1.4	1.3	-

1) Six holes on cleaned bench 2) New Boliden, Gällivarre, Sweden (GPS).

APPENDIX Q DRILL BIT WEAR

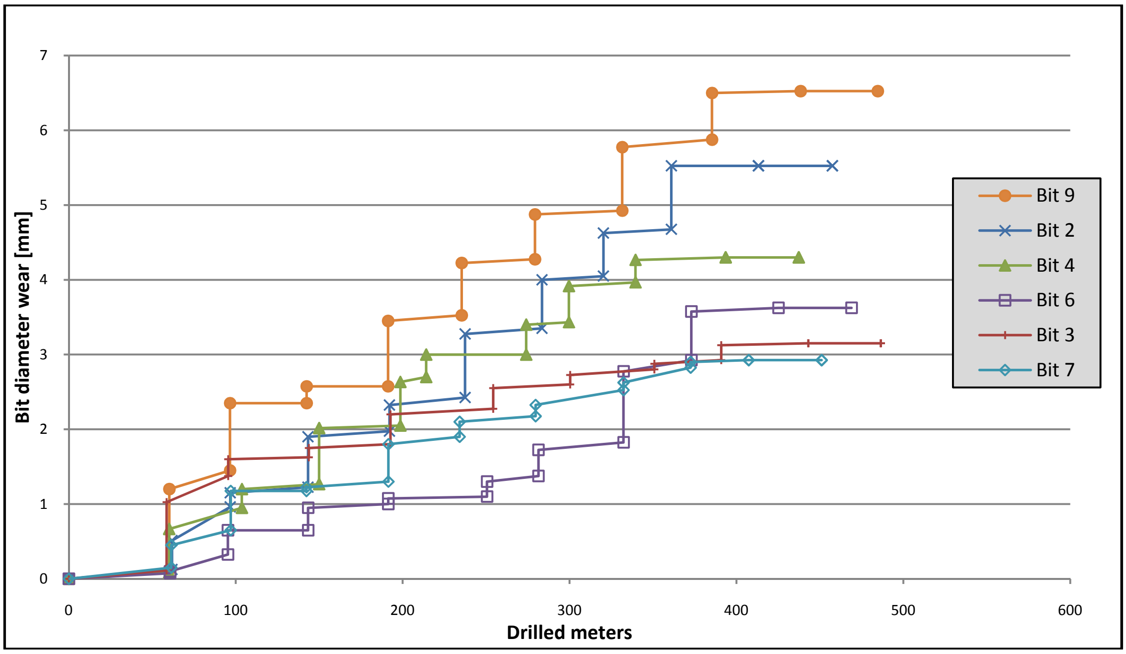


Figure Q.1 Bit diameter wear versus drilled meters, 89 mm category bits

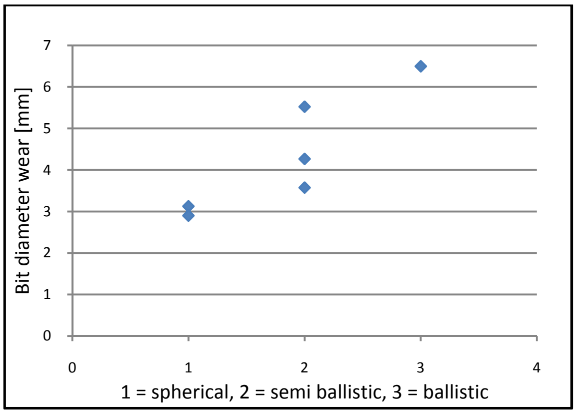


Figure Q.2 Bit wear vs. button shape

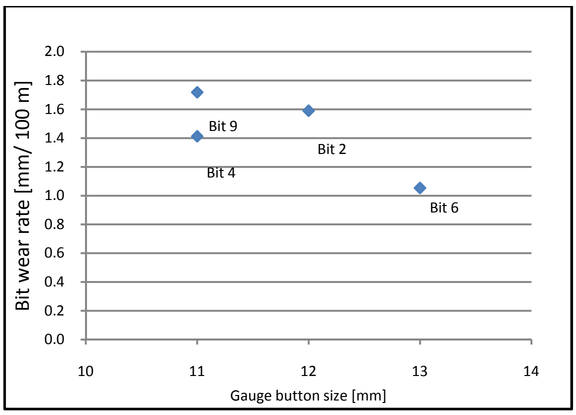


Figure Q.3 Bit wear rate vs. button size

Ballistic and semi ballistic bits.

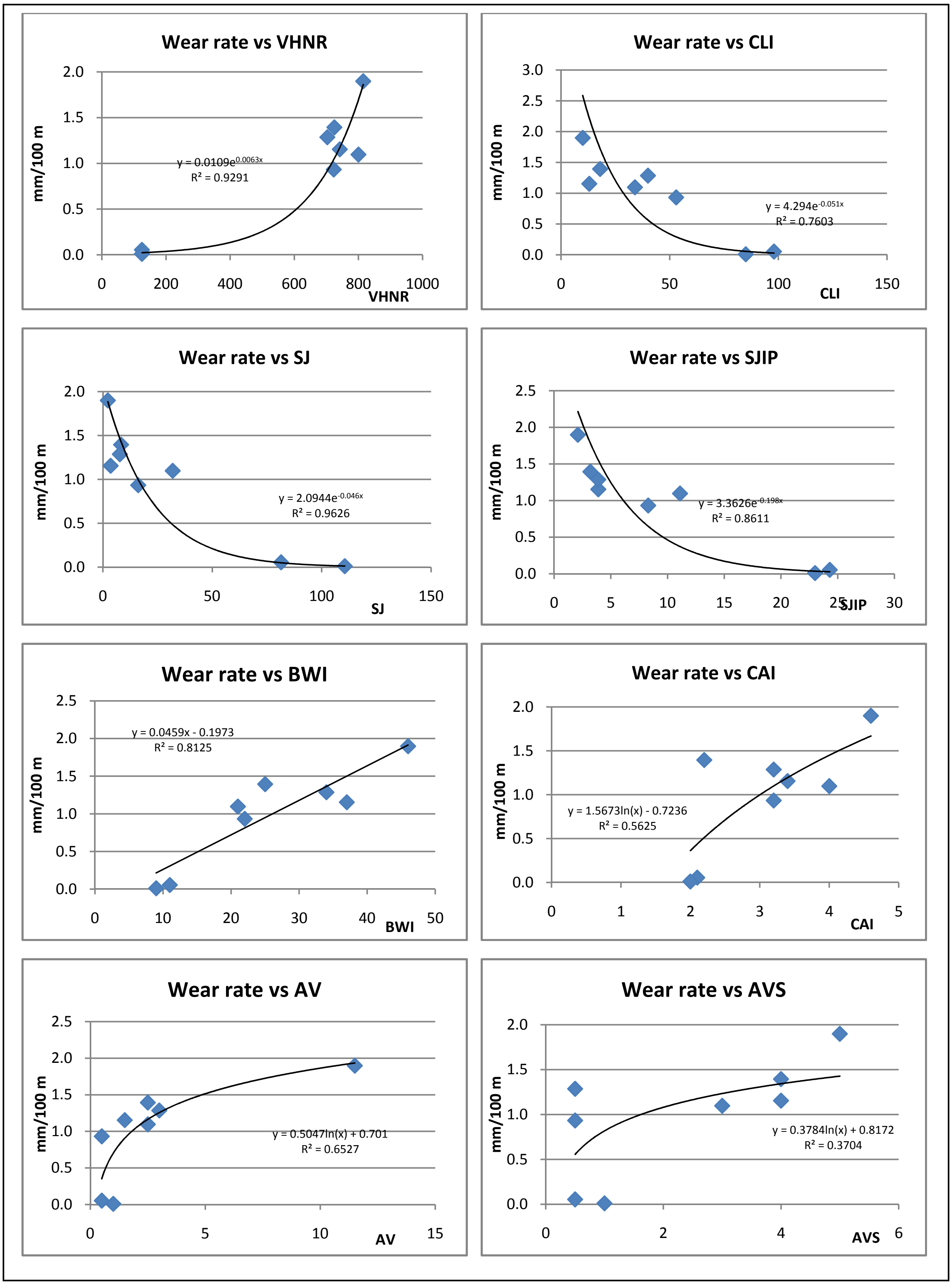


Figure Q.4 **Drill bit wear rate as a function of various rock wear parameters.**

APPENDIX R DEFLECTION VS. DRILL BIT LENGTH

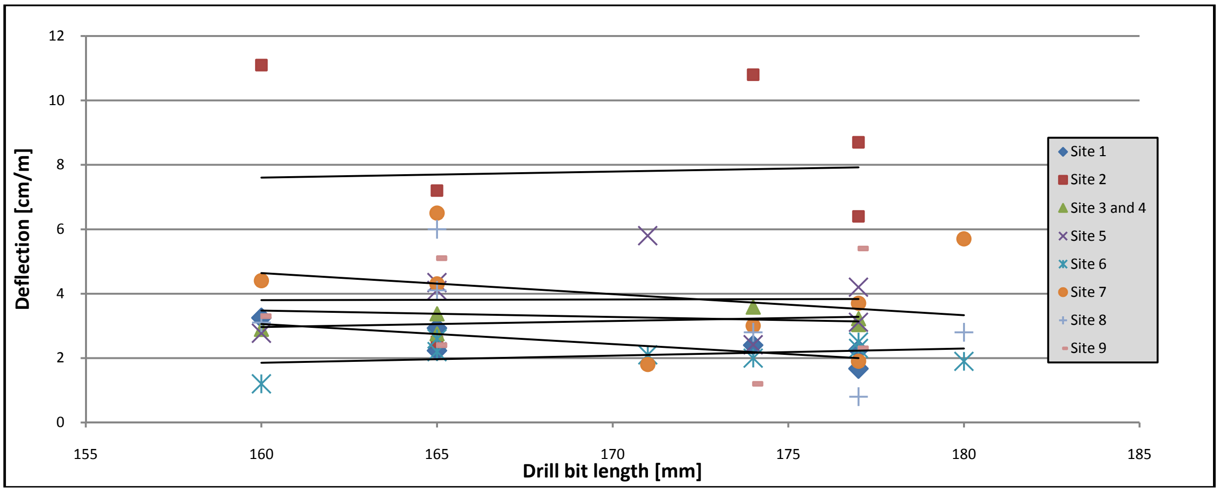


Figure R.1 **89 mm bits and linear regression lines for all sites. No significant connection visible across the drill bit designs.**

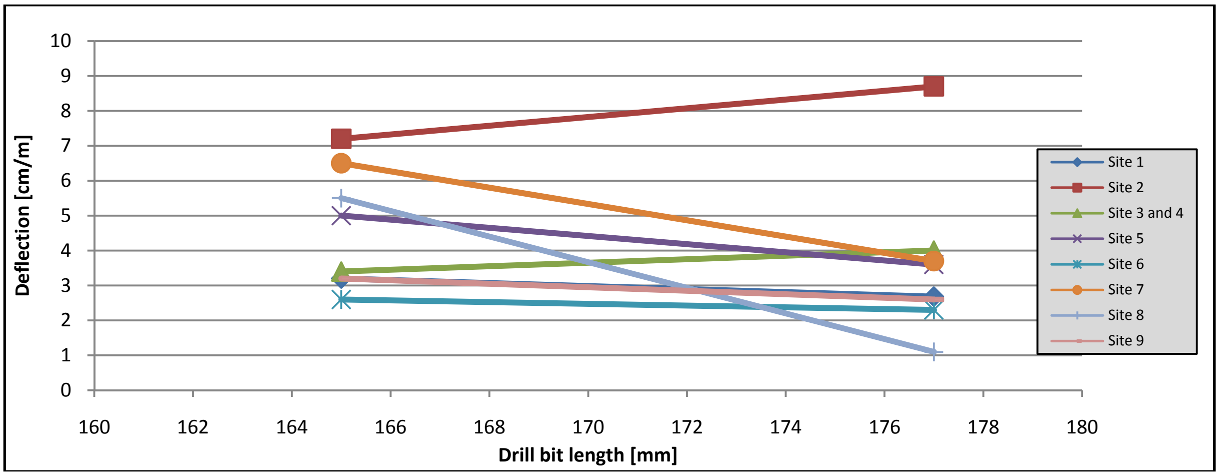


Figure R.2 **Bit 2 vs. Bit 6. Besides the length difference the bits are almost identical. Slightly decreased deflection with increased length.**

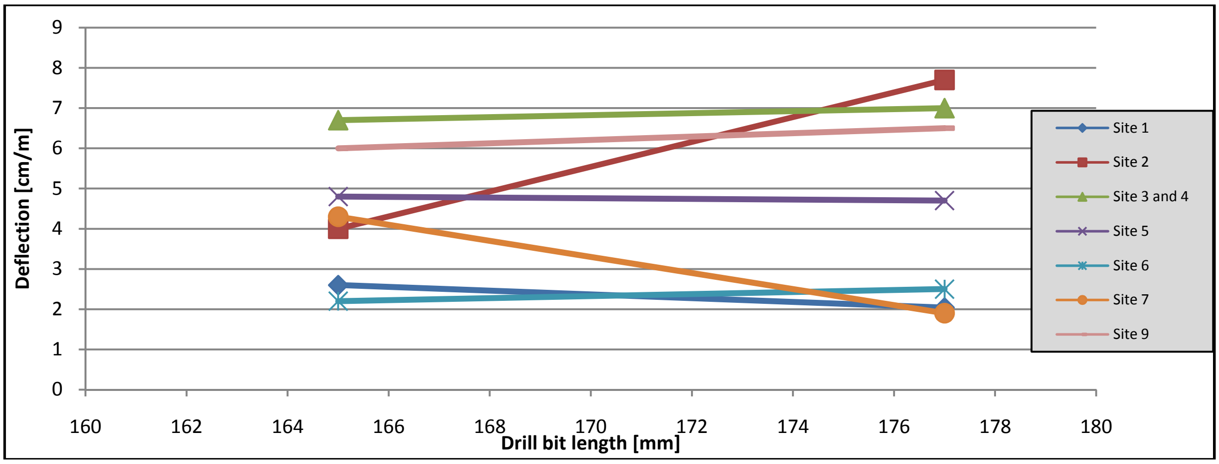


Figure R.3 **Bit 3 vs. Bit 7. Beside the length difference the bits are almost identical. No dependency visible between bit length and deflection.**

APPENDIX S DEFLECTION VS. NET PENETRATION RATE, BIT DIAMETER WEAR AND BUTTON DIAMETER.

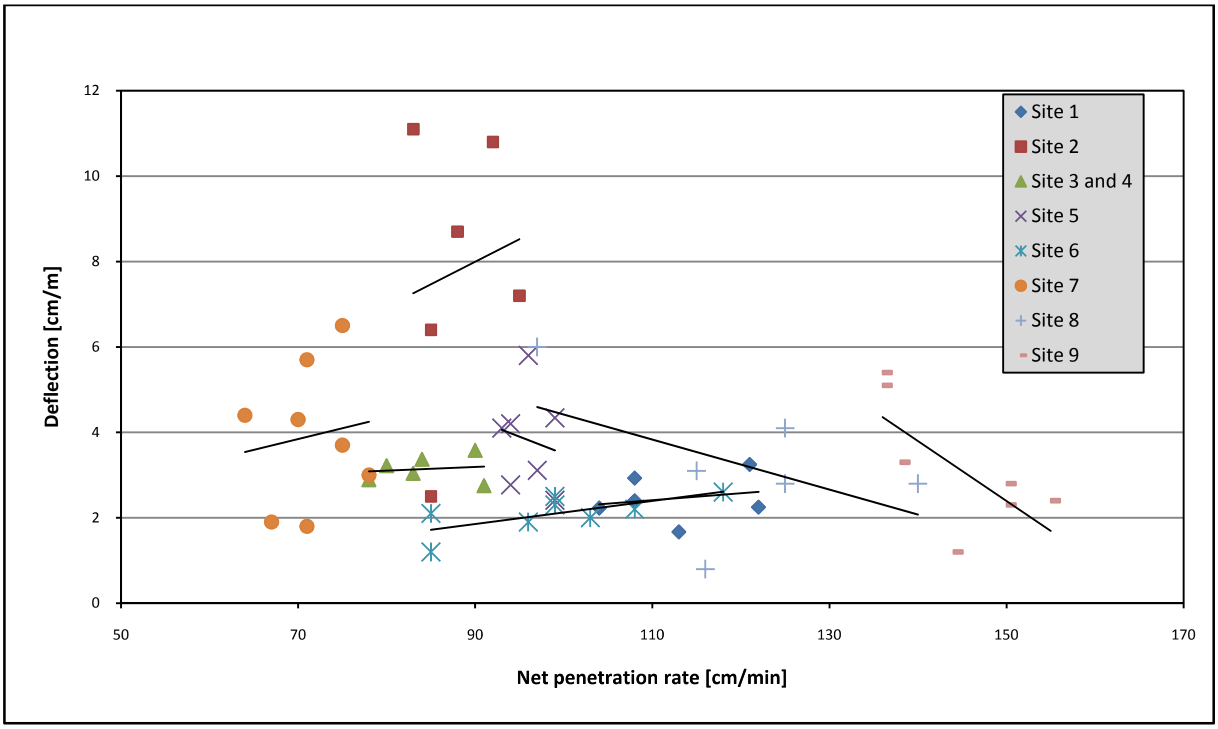


Figure S.1 Average bottom hole deflection versus net penetration rate (89 mm bits) for all sites. Linear regression lines for each site included. Apparently no unique trend.

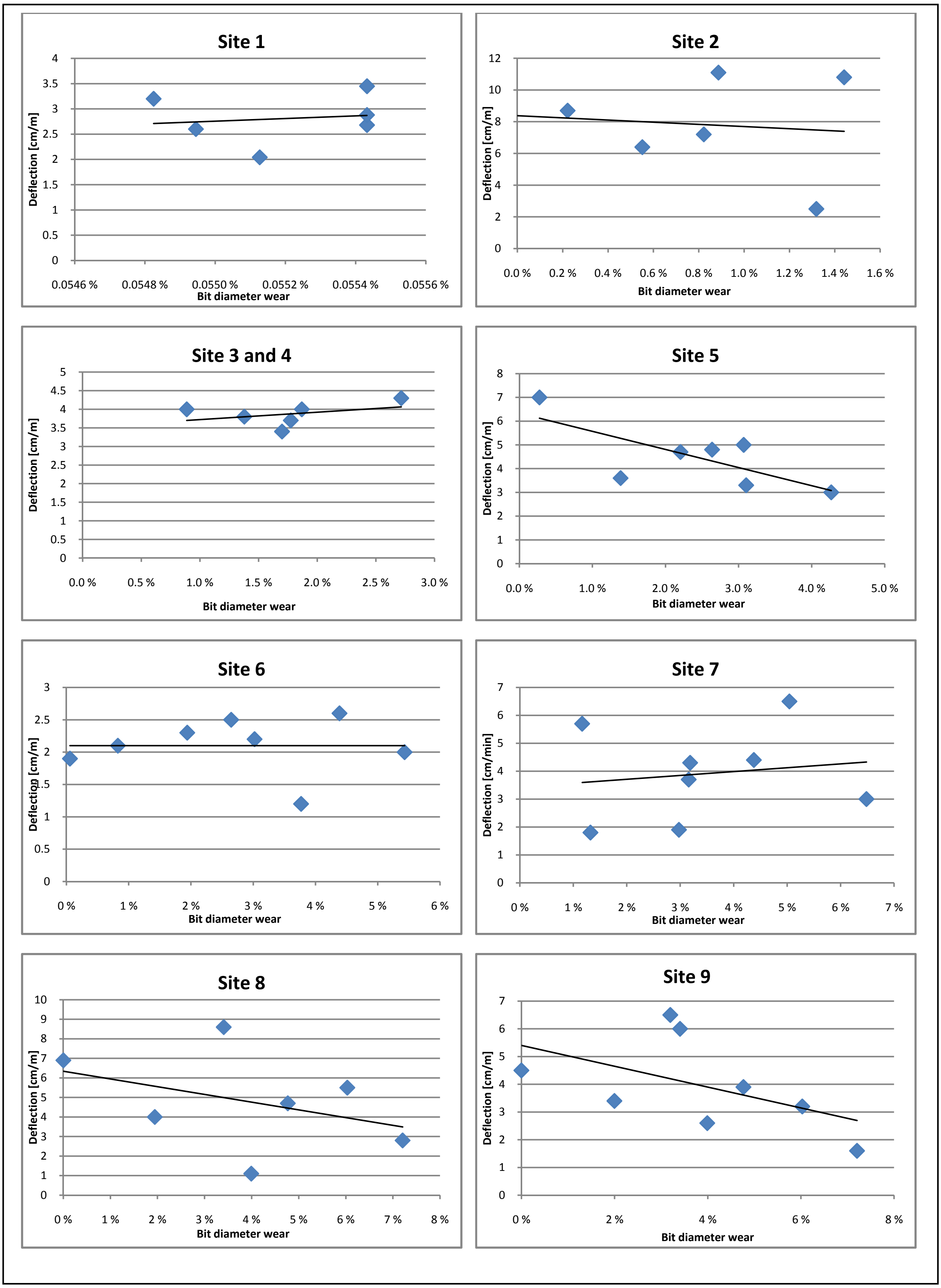


Figure S.2 Average bottom hole deflection versus drill bit diameter wear in percentage of original bit diameter. All 89 mm bits

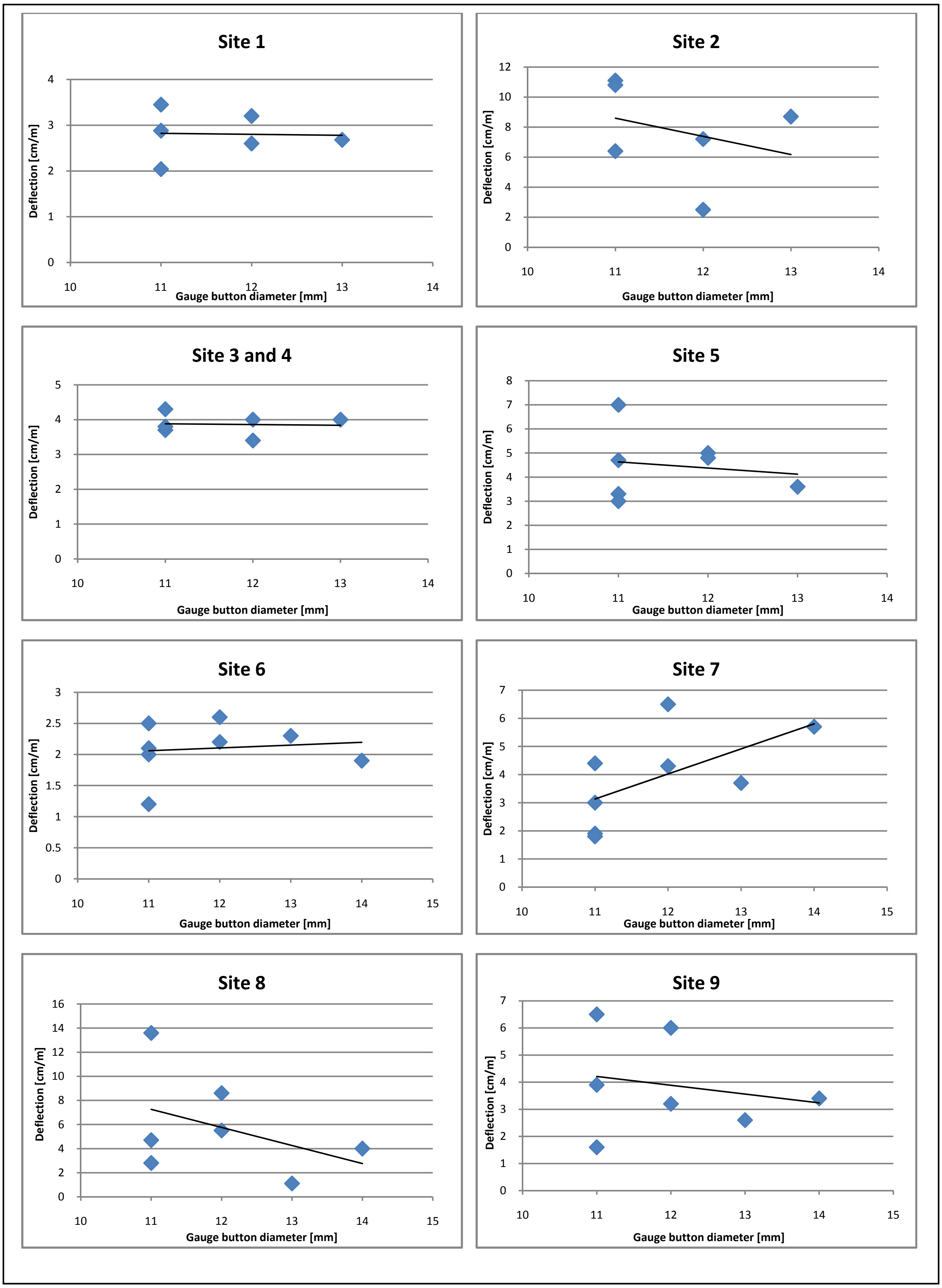


Figure S.3 Average bottom hole deflection versus gauge button diameter. All 89 mm bits

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