UNIVERSIDAD POLITÉCNICA DE MADRID
ESCUELA TÉCNICA SUPERIOR DE INGENIEROS DE
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THE USE OF MEASURE WHILE
DRILLING FOR ROCK MASS
CHARACTERIZATION AND
DAMAGE ASSESSMENT IN
BLASTING

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RESUMEN

Cualquier contrucción subterránea o a cielo abierto desarrollada mediante perforación y voladura, consiste en un conjunto de operaciones, tales como perforación, carga de explosivo, voladura, saneamiento y sostenimiento primario, carga del material y transporte, todas ellas vinculadas en ciclo de producción. La ejecución de las operaciones iniciales (perforación, carga de explosivo y voladura) normalmente preconditionan el desarrollo de la operación aguas abajo en la producción.

La automatización de las diferentes operaciones desarrolladas en minería u obra civil generan nuevas oportunidades de optimizar la operación. Para ello, una detallada caracterización de la roca es uno de los requisitos más importante para llevar a cabo un buen control del mineral a extraer, mejorar los resultados de la voladura y optimizar el proceso de producción. Los métodos clásicos de caracterización de la roca (tales como Rock Quality designation, RQD, Q Index or Rock mass rating) normalmente evalúan de forma global las propiedades del macizo rocoso. Sin embargo, en el transcurso de las operaciones, la aparición de anomalías inesperadas suele influenciar el resultado de las siguientes etapas de producción. El monitoreo de los parámetros de rendimiento de las perforadoras modernas a través del sistema Measurement While Drilling (MWD) registra información real de la roca perforada, proporcionando una herramienta complementaria para la caracterización del macizo rocoso.

El correcto manejo e interpretación de los datos MWD es clave para obtener una caracterización geotécnica del macizo rocoso. Esta tesis lleva a cabo un profundo análisis de esta tecnología con el fin de guiar la perforación y voladura para mejorar la eficiencia de las operaciones mineras y excavaciones de obra civil. Para ello, se ha realizado un detallado análisis del sistema de control de la perforación, teniendo en cuenta cómo se relacionan los parámetros de perforación entre sí y su respuesta ante cambios en la roca. Además, se ha llevado a cabo un exhaustivo proceso de normalización de los parámetros MWD, con el objetivo de eliminar cualquier influencia externa en los datos que pueden inducir a errores en su interpretación y para resaltar comportamientos en los parámetros provocados por la roca.

Esta tesis recopila datos del monitoreo de la perforación para un total de 1285 voladuras y más de 70,000 barrenos de producción, recogidos a partir de tres construcciones en túnel, dos yacimientos en minería subterránea y una cantera. Se han analizado datos de

once máquinas de perforación, incluyendo: tres jumbos XF3C, dos jumbos XL3C, un jumbo E2C para operaciones en túnel, cuatro Atlas Copco SIMBA W6C para trabajos en minería subterránea por subniveles y una perforadora Sandvik Tamrock DX-800 para cantera.

La tecnología MWD se ha analizado en estos tres entornos proporcionando i) un mayor conocimiento de la relación que existe entre los parámetros MWD para maquinaria en túnel, minería subterránea y cantera, (ii) la evaluación de las desviaciones generadas durante la perforación de los barrenos en túnel, (iii) el desarrollo de un índice de roca para detectar zonas potenciales de sobre-excavación durante la excavación de túneles, (iv) un modelo de predicción de problemas de carga de explosivo en los barrenos para minería subterránea y su aplicación a gran escala, (v) desarrollo, en colaboración con Integra Automatización S.L.U., de un sistema propio de registro de parámetros MWD, como solución económica, que permite monitorizar la información de cualquier maquinaria de perforación, y (vi) un índice de fracturación para caracterizar la condición del macizo rocoso en cantera y la validación de sus resultados con registros fotográficos de las paredes del barreno.

ABSTRACT

Any underground or open construction developed by drill and blast consists of several unit operations, such as drilling, charging, blasting, support installation, loading and hauling, all of them linked in the production cycle. The performance of the initial operations (drilling, charging and blasting) pre-conditions the unit operations in the downstream production cycle.

The automatization of different unit operations involved in mining or civil works brings new possibilities to optimize the operation. A detailed rock mass characterization is one of the most important requirements to control the ore to be mined, improve blasting results and optimize the mine to mill performance. Classical methodologies typically assess global rock mass properties (such as Rock Quality designation, RQD, Q Index or Rock mass rating). However, the occurrence of unexpected anomalies will influence the outcome of the following unit operations. Monitoring of performance data of modern drill rigs through Measurement While Drilling (MWD) systems gathers real time information of the penetrated rock mass, providing a complementary tool for rock mass characterization.

The correct management and interpretation of these data is helpful to obtain a geotechnical characterization of the rock mass. The assessment of this technology is carried out in this thesis with the purpose of guiding drilling and blasting towards more efficient mining operations and civil excavation works. To do so, a detailed analysis of the drilling control system, including how monitored parameters relate to each other and to the rock mass conditions, has been performed. A multi-step transformation of the MWD parameters has been also carried out to remove external influences in the data that may lead to a wrong interpretation and to highlight changes in the parameters depending on the rock properties.

This thesis compiles drill monitoring data for a total of 1,285 blasts and more than 71,000 production blastholes, recorded from three tunneling works, two orebodies of an underground mine and one quarry. Data from eleven drilling machines have been analyzed; this comprises: three jumbos XF3C, two jumbos XL3C, one jumbo E2C for tunneling, four Atlas Copco SIMBA W6C drill rigs for sublevel caving and one Sandvik Tamrock DX-800 drilling machine for quarrying.

MWD has been analyzed in these three different environments providing i) further knowledge of the relationship between MWD parameters in jumbo drills in tunneling, underground mining and quarrying, (ii) assessment of drilling deviations in tunneling, (iii) the development of a drilling rock index to detect potential overbreak zones in tunnel blasting, (iv) assessment of chargeability issues in underground mining and its application to the full scale, (v) development, in collaboration with Integra Automatización S.L.U., of an in-house MWD system as a low-cost alternative that allows monitoring the information of any rig while drilling and (vi) a fracturing index to characterize ground rock conditions in quarrying and validation of the last results with photographic records of the blastholes walls.

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Chapter 1. INTRODUCTION

1.1. Background

For an efficient mining operation and civil work, it is essential to have a well-defined characterization of the rock mass to be excavated and the surrounding ground conditions. Any underground or open construction process developed by drilling and blasting consists of several unit operations, such as drilling, charging, blasting, support installation, loading and hauling, all of them linked in the production cycle (Ghosh, 2017). In this way, the performance of the initial operations (drilling, charging and blasting) pre-conditions the unit operations in the downstream production cycle.

The automatization of the different stages involved in mining or civil works bring new possibilities to optimize the whole operation. A proper rock mass characterization is one of the most important requirements for having an efficient control of the production cycle. The rock mass is a combination of materials with different mechanical properties, intersected by discontinuities and fractures. Pre-studies carried out to determine rock mass conditions roughly estimate its properties; however, during the operation, unexpected anomalies may appear, increasing production costs, extending operation times and reducing the safety of work conditions. Since rock characteristics have an important influence on the drilling response (Hatherly et al., 2015; Peng et al., 2005; Schunnesson et al., 2011), technologies based on measuring drill parameters from the performance data of drill rigs may assess changes in the rock mass with high resolution.

The Measurement While Drilling (MWD) technique is a drill monitoring system that collects operational drilling data at predetermined length intervals along the blasthole (Schunnesson, 1997). This technology, able to provide high data resolution and with a minimal disturbance of the production, has made MWD a complementary tool for rock mass characterization and geotechnical ground recognition. However, despite the advantages of this technology, the massive data recorded per blast and the difficulties of interpretation complicate its application as a decision-making tool in the daily routine of construction works.

Although there is a large number of studies focused on the geological and geotechnical interpretation of the rock mass by using MWD (Teale, 1965; Scoble et al., 1988;

Schunnesson, 1996, 1997, 1998; Liu and Karen, 2001; Kaharaman et al., 2003; Peng et al., 2005; Tang, 2006; Hjelme, 2010; Schunnesson et al., 2011; Schunnesson et al., 2012; Naeimipour et al., 2014; Ghosh et al., 2015; Hatherly et al., 2015; Leung et al., 2015; Ghosh, 2017, Ghosh et al., 2018), this technology has not been applied, to the author's knowledge, to guide drilling and blasting towards a more efficient operation.

1.2. Problem statement

Viability studies carried out to determine rock mass properties (Deere et al., 1967; Barton et al., 1974; Bieniawski, 1995), are based on drill cores, making them time consuming and expensive. To reduce these costs, normally a few widely spaced holes are logged and the ground conditions between them are interpolated (Schunnesson, 1997). This may lead to inaccurate interpretations of the geological and geotechnical rock mass properties that will significantly impact the results of the operation.

The performance data of the drill rig through the MWD system shows real time information of the response of the drilling to rock mass. Modern drilling machines allow a full automatization of the drilling operation by installing monitoring systems in the drill rig such as the ABC (Advanced Boom Control, Atlas Copco, 2017), iREDES (Sandvik, 2017) and Bever Control (Bever Control, 2017). To manage the MWD records and to evaluate rock properties, such as hardness and fracturing, different software packages have been developed (Tunnel Manager - Atlas Copco, iSURE - Sandvik and Bever Control in cooperation with AMV). However, difficulties in the interpretation due to the lack of knowledge of how these softwares manage the information and the still unsolved limitations of this technology, complicate its application for guiding drilling and blasting operations.

This thesis carries out a thorough analysis of the MWD technology with the final purpose of developing an engineering tool based on the MWD records interpretation, to be used as guide during the design of the blasts in mining and construction works.

1.3. Objectives

The main objective of this research is to assess rock mass quality through drill-monitoring data to guide blasting in tunneling, underground mining and quarrying. MWD has been assessed in three different environments providing (i) a drilling rock index to detect

potential over-excavation zones in tunnel blasting; (ii) a mining planning index to predict problems during the charging of long blastholes in underground mining; (iii) an in-house MWD system to monitor the information of any rig while drilling and, from the records, a drilling index to characterize rock conditions in quarrying. A list of the objectives fulfilled by this work are:

- Obj 1. Assessment of the drilling control system navigation for jumbo drills in tunneling and underground mining.
- Obj 2. Development, in collaboration with Integra Automatización S.L.U., of an in-house MWD system as a low-cost alternative that allows monitoring the information of any rig while drilling.
- Obj 3. Interpretation and processing of MWD data.
- Obj 4. Assessment of drilling deviations in tunneling.
- Obj 5. Development of a detection of potential overbreak zones (high risk of potential over-excavation zones) prediction model based on the MWD data.
- Obj 6. Development of a geotechnical rock condition block model and its application in full scale in underground mining.
- Obj 7. Analysis of drill monitoring data to assess chargeability in fan shaped long-holes: development and application of a risk of blasthole collapse model.
- Obj 8. Assessment of the MWD logs to define a new fracturing index for geotechnical rock recognition in quarry blasting and validation of the results with photographic records of the blastholes walls.

1.4. Thesis structure

This thesis includes seven chapters. The state of the art in Chapter 2 provides a comprehensive literature review of (i) the different pre and post-blast rock mass characterization methods and (ii) an extensive description of the different drilling modes and the application of the MWD parameters for rock mass characterization. Chapter 3 describes the data set, the geological and geotechnical environments and the different equipment used at the three sites considered. Chapter 4 describes the application of MWD in tunneling to study the relationship between MWD parameters, to assess drilling deviations in tunneling and to develop an engineering tool to predict over-excavation by blasting effect. Chapter 5 deals with the application of MWD to underground mining. It characterizes and predicts geotechnical rock conditions to assess blasthole collapses that

involve charging problems. In it, two block models are developed: one for the geotechnical rock condition assessment of the ore body and the other to predict problems during the charging of long blastholes in sublevel caving. Chapter 6 presents an in-house MWD system, as a low-cost alternative to monitor the information of drilling and the development of a drilling index to characterize ground conditions in quarrying. Lastly, Chapters 7 and 8 highlight the overall conclusions and recommendations for future work. Appendixes 1 to 4 include supplementary material for the analysis developed in chapters 4 and 5.

The accomplishments of this thesis are addressed in 5 journal papers and 2 conference papers, as given in Table 1.1. Table 1.2 shows the objectives reached by these papers. These papers are compiled in Appendices A through G.

Table 1.1. Literature generated by this thesis.

Paper	A	<i>“MWD parameters and Drilling Control system”</i> Navarro, J., Sanchidrián, J. A., Segarra, P., Castedo, R., López, L. M., Cienfuegos, R., 2017. Proceedings of the World Tunnel Congress, Bergen, Norway, pp. 194 – 203.
	B	<i>“Assessment of contour profile quality in D&B tunneling”</i> Costamagna, E., Oggeri, C., Segarra, P., Castedo, R., Navarro, J., 2018. Tunneling and Underground Space Technology, vol. 75, pp. 67 – 80.
	C	<i>“On the mutual relations of drill monitoring variables and the drill control system in tunneling operations”</i> Navarro, J., Sanchidrián, J. A., Segarra, P., Castedo, R., Paredes, C., López, L. M., 2018. Tunneling and Underground Space Technology, vol 72, pp. 294 – 304.
	D	<i>“Detection of potential overbreak zones in tunnel blasting from MWD data”</i> Navarro, J., Sanchidrián, J. A., Segarra, P., Castedo, R., Costamagna, E., López, L. M., 2018. Tunneling and Underground Space Technology, vol 82, pp.504-516.
	E	<i>“Assessment of drilling deviations in underground operations”</i> Navarro, J., Segarra, P., Sanchidrián, J. A., Castedo, R., Lopez, L. M., 2018. Tunneling and Underground Space Technology, vol. 83, pp. 254-261.
	F	<i>“Application of an in-house MWD system for quarry blasting”</i> Navarro, J., Segarra, P., Sanchidrián, J. A., Castedo, R., Pérez Fortes, A. P., Natale, M., Lopez, L. M., 2018. Proceedings of the 12 th International Symposium on Rock Fragmentation by Blasting, Fragblast 12, Luleå, Sweden, pp. 203 – 207.
	G	<i>“Application of drill-monitoring for chargeability assessment in sublevel caving”</i> Navarro, J., Schunnesson, H., Johansson, D., Sanchidrián, J. A., Segarra, P., 2018. International Journal of Rock Mechanics and Mining Sciences. Under review.

Table 1.2. Relevance of appended papers to thesis objectives

Objectives	Papers						
	A	B	C	D	E	F	G
Obj.1. Drilling control system navigation.				X	X		X
Obj. 2. Development of an in-house MWD system.						X	
Obj. 3. Interpretation and processing of MWD data.	X		X	X	X	X	X
Obj. 4. Assessment of drilling deviations in tunneling.					X		
Obj. 5. Development of a detection of potential overbreak zones prediction model based on the MWD data.		X		X			
Obj. 6. Development of a geotechnical rock condition block model.							X
Obj. 7. Analysis of drill monitoring data to assess chargeability.							X
Obj. 8. Assessment of the MWD logs to define a new fracturing index.						X	

1.5. Authors' contribution of the appended papers

The contributions of the authors can be divided into the following activities:

1. Responsible for the work described in the paper
2. Collection of data
3. Analysis of data and results
4. Preparation of the manuscript
5. Revision and final approval of manuscripts

Based on these activities, the authors' contributions to each paper are presented in Table 1.3.

Table 1.3. Authors' contribution

	Papers						
Authors	A	B	C	D	E	F	G
Castedo, Ricardo	5	2,3,5	5	2,5	5	2,5	
Cienfuegos, Raquel	2						
Costamagna, Elisa		1,3,4,5		3			
Johansson, Daniel							5
López, Lina María	5		5	5	5	5	
Natale, Marco						2	
Navarro, Juan	1-5	2,3,5	1-5	1-5	1-5	1-5	1,3,4,5
Oggeri, Claudio		4,5					
Paredes, Carlos	5						
Pérez Fortes, Ana Patricia						2	
Sanchidrian, José Ángel	4,5		4,5	4,5	4,5	4,5	5
Schunnesson, Håkan							2,5
Segarra, Pablo	4,5	2,3,5	4,5	2,4,5	2,4,5	4,5	5

Chapter 2. LITERATURE REVIEW

This chapter provides background information on different techniques used during pre- and post-blast rock mass characterization and gives an extensive description of the different drilling modes and the application of the MWD parameters for rock mass characterization.

2.1. Rock characterization methods: Rock Quality pre-blast

Prior to any blast, it is important to characterize the existing integrity of the rock mass. The definition of the mine's rock mass and their properties may assess a geomechanical classification to provide a basic reference for postblast observations (Brown, 1981). Literature reviews of the most common pre-blast methods for rock mass characterization, such as the Rock Quality Designation (RQD), the Rock Tunneling Quality index (Q index) and the Rock Mass Rating (RMR) are described. Some background on different techniques for geotechnical rock recognition of the blastholes before blasting is also included.

2.1.1. Rock Quality Designation index (RQD)

The RQD index was developed by Deere (Deere et al. 1967) to provide a quantitative estimation of the rock mass quality by using drill cores. At present, RQD is incorporated in some of the rock indexes like the Rock Tunneling Quality Index (Barton et al., 1974) and the Rock Mass Rating (RMR, Bieniawski, 1989). The RQD is defined as “*the percentage of intact core pieces longer than 100 mm in the total length of core*” (Deere et al. 1967). The core should be at least of 54.7 mm diameter and should be drilled with a double-tube core barrel. Figure 2.1 shows an example of the procedure for measuring the length of the core pieces and the RQD calculation. When no core is available, but discontinuities traces are visible on the rock exploration surface, the RQD may be estimated by applying a scanline on the surface and measuring discontinuities passing across the line. Figure 2.2 shows the scanline over a surface, where the “core pieces” greater than 10 cm are shown in black.

Figure 2.1. Procedure for measurement and calculation of RQD (Deere, 1989).

Figure 2.2. Procedure for measurement of RQD on the surface (Palmstrom et al., 2005).

2.1.2. Rock Tunneling Quality Index

The Rock Tunneling Quality Index (Q-value) index was proposed by Barton et al. (1974) as classification of the ground conditions for underground excavations. The Q-value expresses the quality of the rock mass on which the design and the support recommendations for underground excavations are based. The numerical value of the index Q is:

$$Q = \frac{RQD}{J_n} \times \frac{J_r}{J_a} \times \frac{J_w}{SRF} \quad (2-1)$$

where, RQD is the Rock Quality Designation

J_n is the joint set number

J_r is the joint roughness number

J_a is the joint alteration number

J_w is the joint water reduction factor

SRF is the stress reduction factor

The first quotient (RQD/J_n) represents the structure of the rock, the second quotient (J_r/J_a), the roughness and frictional characteristics of the joint walls or filling materials, and the third quotient (J_w/SRF), the rock mass active stress.

Barton et al. (1974) developed a classification for the individual parameters used to obtain the Tunneling Quality Index Q for a rock mass. Values of these individual parameters are given in tables defined by Barton et al. (1974). The estimated support categories, based on the tunneling quality Q index, are shown in Figure 2.3 on a logarithmic scale from 0.001 to 1000. It shows, in the top part, the rock classes (A through G) in which the rock mass is qualitatively classified. To determine the support from Figure 2.3, it is necessary to use an Excavation Support Ratio (ESR) defined in Table 2.1. This parameter is related with the intended use of the excavation and the degree of safety which is demanded.

Table 2.1. ESR values according the excavation category.

Excavation category		ESR
A	Temporary mine openings	3-5
B	Permanent mine openings, water tunnels for hydro power (excluding high pressure penstocks), pilot tunnels, drifts and headings for large excavations.	1.6
C	Storage rooms, water treatment plants, minor road and railway tunnels, surge chambers, access tunnels.	1.3
D	Power stations, major road and railway tunnels, civil defence chambers, portal intersections.	1.0
E	Underground nuclear power stations, railway stations, sports and public facilities, factories.	0.8

Barton et al. (1974) developed a chart to define the estimated support categories in the excavation based on the representation of both the Equivalent dimension (Excavation span, diameter or height / ESR) and the Q Index. Grimstad and Barton (1993) updated

the chart to reflect the increasing use of steel fiber reinforced shotcrete in underground excavation support, see Figure 2.3.

Figure 2.3. Estimated support categories based on the tunneling quality index Q (Grimstad and Barton, 1993).

2.1.3. Rock Mass Rating

The Rock Mass Rating (RMR, Bieniawski, 1989) system is defined as the geo-mechanical classification of the rock, based on the combination of the most significant rock mass characterization parameters used in underground excavations. The RMR is calculated from the following six parameters:

- 1- Uniaxial Compressive Strength of rock material.
- 2- Rock Quality Designation (RQD).
- 3- Spacing of discontinuities.
- 4- Condition of discontinuities.
- 5- Ground water conditions.
- 6- Orientation of discontinuities.

According to the characteristics of the rock mass, the mentioned six parameters are ranked in a numerical rating classification (Table 2.2). The RMR value is calculated as the sum of the six parameters in Table 2.2.

Table 2.2. Rock Mass Rating System Parameters (Bieniawski, 1989).

A. CLASSIFICATION PARAMETERS AND THEIR RATINGS									
Parameter			Range of values						
1	Strength of intact rock material	Point-load strength index	>10 MPa	4 - 10 MPa	2 - 4 MPa	1 - 2 MPa	For this low range - uniaxial compressive test is preferred		
		Uniaxial comp. strength	>250 MPa	100 - 250 MPa	50 - 100 MPa	25 - 50 MPa	5 - 25 MPa	1 - 5 MPa	< 1 MPa
	Rating		15	12	7	4	2	1	0
2	Drill core Quality RQD		90% - 100%	75% - 90%	50% - 75%	25% - 50%	< 25%		
	Rating		20	17	13	8	3		
3	Spacing of discontinuities		> 2 m	0.6 - 2 . m	200 - 600 mm	60 - 200 mm	< 60 mm		
	Rating		20	15	10	8	5		
4	Condition of discontinuities (See E)		Very rough surfaces Not continuous No separation Unweathered wall rock	Slightly rough surfaces Separation < 1 mm Slightly weathered walls	Slightly rough surfaces Separation < 1 mm Highly weathered walls	Slickensided surfaces or Gouge < 5 mm thick or Separation 1-5 mm Continuous	Soft gouge >5 mm thick or Separation > 5 mm Continuous		
	Rating		30	25	20	10	0		
5	Groundwater	Inflow per 10 m tunnel length (l/m)	None	< 10	10 - 25	25 - 125	> 125		
		(Joint water press./ (Major principal σ)	0	< 0.1	0.1, - 0.2	0.2 - 0.5	> 0.5		
		General conditions	Completely dry	Damp	Wet	Dripping	Flowing		
	Rating		15	10	7	4	0		
B. RATING ADJUSTMENT FOR DISCONTINUITY ORIENTATIONS (See F)									
Strike and dip orientations			Very favourable	Favourable	Fair	Unfavourable	Very Unfavourable		
Ratings	Tunnels & mines		0	-2	-5	-10	-12		
	Foundations		0	-2	-7	-15	-25		
	Slopes		0	-5	-25	-50			

The RMR varies in a tighter range than the Q -index. From the RMR results, Bieniawski (1989) defined qualitatively the rock and published a set of guidelines for excavation and support for a 10 m span rock tunnel (Table 2.3).

2.1.4. Geotechnical blasthole condition characterization

Since a hole is an open aperture in the rock, the original properties of the ground condition around the hole are influenced, inter alia, by the stress redistribution (Kwon et al., 2009) and by vibrations from adjacent blast-rounds that may induce stability problems around the blasthole wall. When the surrounding stress exceeds the tensile, the compressive, or the shear strengths of the rock formation, failures in the blasthole wall can be generated (Zhang et al., 2003). As result, rock detachments may occur with possible collapses inside the hole, which later may result in charging problems.

Table 2.3. Guidelines for excavation and support of 10 m span rock tunnels in accordance with the RMR system (Bieniawski, 1989).

Rock mass class	Excavation	Rock bolts (20 mm diameter, fully grouted)	Shotcrete	Steel sets
I - Very good rock <i>RMR: 81-100</i>	Full face, 3 m advance.	Generally no support required except spot bolting.		
II - Good rock <i>RMR: 61-80</i>	Full face , 1-1.5 m advance. Complete support 20 m from face.	Locally, bolts in crown 3 m long, spaced 2.5 m with occasional wire mesh.	50 mm in crown where required.	None.
III - Fair rock <i>RMR: 41-60</i>	Top heading and bench 1.5-3 m advance in top heading. Commence support after each blast. Complete support 10 m from face.	Systematic bolts 4 m long, spaced 1.5 - 2 m in crown and walls with wire mesh in crown.	50-100 mm in crown and 30 mm in sides.	None.
IV - Poor rock <i>RMR: 21-40</i>	Top heading and bench 1.0-1.5 m advance in top heading. Install support concurrently with excavation, 10 m from face.	Systematic bolts 4-5 m long, spaced 1-1.5 m in crown and walls with wire mesh.	100-150 mm in crown and 100 mm in sides.	Light to medium ribs spaced 1.5 m where required.
V - Very poor rock <i>RMR: < 20</i>	Multiple drifts 0.5-1.5 m advance in top heading. Install support concurrently with excavation. Shotcrete as soon as possible after blasting.	Systematic bolts 5-6 m long, spaced 1-1.5 m in crown and walls with wire mesh. Bolt invert.	150-200 mm in crown, 150 mm in sides, and 50 mm on face.	Medium to heavy ribs spaced 0.75 m with steel lagging and forepoling if required. Close invert.

As an alternative to core logging, different methodologies have been applied to record structural features; these can be classified as geophysical, mechanical and optical. These techniques can be used to get a detailed information on blasthole issues.

Among geophysical methods, blasthole radar and auto scanning laser systems have been used to detect cavities and fractures (Haeni et al., 2002), and single-hole reflection and tomography methods have been applied for rock characterization in inclined blastholes at the Grimsel nuclear waste laboratory in Switzerland (Haeni et al., 2002).

The mechanical methodology is based on the monitoring of performance data from jumbos while drilling the blasthole. This is explained in detail in section 2.3.

Optical methods have been applied by Kangas (2007) and Ghosh et al. (2015). They used a mini-video camera in blastholes of the Kiruna and Malmberget mines, Sweden, to assess the condition of the blastholes. They found that typical blastholes problems are generated by deformations of blasthole walss and by blastholes blocked by stones.

The condition of the blastholes can also be analyzed with an optical televiewer, which provides a continuous unwrapped 360° oriented color image of the blasthole walls (Li et al. 2013).

2.2. Rock Quality post-blast

Damage is a change in the rock mass properties which degrades its performance and behavior (Singh, 1992). Blast damage determines an important link between the excavation process and the structural stability of the rock (Singh and Xavier, 2005). Numerous techniques are used to evaluate rock damage:

1. Seismic vibration analysis: models to assess vibration levels from which stress can be estimated are used to build damage contours around the hole.
2. Indirect methods: geophysical methods such as seismic tomography, loose rock detection sensors and ground-penetrating radar, high-frequency crosshole seismic, seismic-refraction tomography.
3. Direct methods based on core drilling and diamond drilled or sawed cuts in the rock.
4. Traditional observation methods give a measure of the macroscopic damage, such as the assessment of the overbreak in the remaining rock mass or the half cast measurements.

2.2.1. Prediction models for damage assessment based on vibration analysis.

Numerous techniques to predict rock damage caused by the blasting effect have been developed; these are based on the explosive charge of the blasthole, the measured particle velocity and the characteristics of the blasted rock.

In relation to those models to assess vibration levels and to assign a stress to these vibrations, Holmberg and Persson (1979) developed an experimental methodology for predicting a perimeter control of the damaged rock based on the peak particle velocity (PPV) generated by detonation of a single linear explosive charge. They located several seismographs at different distances from the blasthole, and PPV values were measured and correlated with the linear charge concentration, the distance between the explosive charge and the seismographs, and the damage extension created by the blasting effect. Based on the Favreau (1969) solution, Hustrulid et al. (1992) developed another approach at the Colorado School of Mines, by dividing a linear explosive column charge into equivalent spherical small charges and by calculating, firstly, the necessary time for the propagation of the detonation wave between the equivalent explosive charges and, secondly, the p-wave velocity in rock mass.

Blair and Minchinton (1996, 2006) used the Heelan (1953) solution, as starting point, to predict the total horizontal and vertical components of the PPV recorded at any monitoring point around a linear explosive charge (in a half space). The Scaled Heelan Model proposed by Blair and Minchinton (1996, 2006) uses full waveform superposition to provide numerical values for the particle velocity at each instant of time. Recently, Blair (2015) has developed a near field vibration damage model and has studied the effect of the velocity of detonation (VoD) of the explosive in the near-field regions. In his work, Blair studied the incident p-wave and s-wave radiation produced by a full column of explosives and predicted that the explosive VoD can be altered to control damage in specific near-field regions. The Heelan solution and Zoeppritz equation (Aki et al., 2002), based on the plane wave, were used for the analytical model. An alternative to validate these models is to apply dynamic finite element model (DFEM) calculations of the radiation from a full column of explosive.

Other models are based on the assignment of a “practical” radius of damage to each blasthole. Hustrulid (1999) explained a method to obtain the damage radius based on the explosive energy, for fully coupled holes, and the rock mass density. Hustrulid (2010) modified Holmberg (1982) damage radius approach, based on the Langefors and Kihlström (1978) explosive energy equation, used to design the burden of lifter and stopping holes. The Spokane Research Lab, NIOSH (Johnson, 2010) developed a new approach to predict practical radius of damage to each blasthole, by estimating five zones around the blasthole (Figure 2.4): the explosive zone, the blasthole decoupled zone, the extensive or crushed damaged zone, the transition or partially damaged zone (cracked zone) and the seismic or non-damaged zone.

Figure 2.4. Representation of crushed, cracked and damage zones. Johnson (2010)

2.2.2. Geophysical methods for rock mass damage control

Geophysical logs, such as gamma-ray, density, p- wave velocity and acoustic reflectivity, are used to gather lithological, structural and rock mechanic information.

The cross-hole sonic logging technique is an ultrasonic integrity testing method to assess in-placed constructions. The technique consists in generating a sonic pulse with one transducer (transmitter) and picking the signal up with another transducer (receiver). Transducers (typically geophones or accelerometers), may e.g. be placed in vertical PVC tubes, filled with water, that act as coupling medium between the transducer and the tube (Niyama et al. 2000). By knowing the distance between the transducers and the arrival time of the sonic wave, the sonic velocity can be calculated. Sonic velocity is thus correlated with rock mass properties. The Spokane Research Lab, NIOSH (Oyler et al., 2008 and 2010), used the cross-hole sonic logging technique to correlate sonic velocity with the Uniaxial Compressive Strength value in different types of rock, and formulated a general exponential equation with a coefficient of determination of 0.72:

$$UCS = 329,100e^{-0.0505t} \quad (2-2)$$

where t is travel time read directly from a sonic log, in $\mu\text{s}/\text{ft}$.

Schepers et al. (2001) combined core images and acoustic log data from wall boreholes for rock structure classification and characterization, and used tomography methods (seismic, geoelectric and radar) for rock mass analysis and geotechnical exploration in boreholes when drill cores were not available. Moreover, Cardarelli et al. (2003) investigated tunnel stability by using an integrated interpretation of three geophysical methods: seismic refraction, seismic tomography and ground penetrating radar. They located discontinuities in the investigated zone by using radar data and estimated the distribution of the mean rock mass elastic properties by analyzing seismic information.

2.2.3. Crack detection methods and crack length prediction models

Several methods to assess the generation of new cracks in the remaining rock mass have been carried out to get a better understanding of the blasting influence in crack development. This allows a better blast design with the purpose of achieving a higher quality of the remaining rock surface. According to Langefors and Kihlström (1973), the breakage of the rock by blasting effect consists of two stages: the first, due to the explosive shock wave, that generates radial cracks around the hole. The second caused by gas expansion, that penetrates in the cracks increasing their length and width.

SveBeFo (Swedish Rock Engineering Research Organization) has made a thorough study on cracks generation in the rock surrounding the blast. Olsson and Bergqvist (1996) studied crack lengths in multiple holes blasted by cutting blocks and spraying them with dye penetrant. Ouchterlony et al. (1999), in the granite quarry tests at Svenneby, and Ouchterlony et al. (2000), in the gneiss at Jakobsdal/Moraberg, explained a technique for crack detection based on the use of dye penetrant that makes cracks more visible. A new procedure based on photographing and sketching the emerging cracks was also designed. Ouchterlony (1997) and Ouchterlony et al. (2002), in the new Swebrec approach, formulated a new equation to predict the length of the radial cracks emanating from the half-cast of the contour holes after cautious blasting, with zero-delay initiation, which is a measure of the extension of the damage zone in the rock. Saiang (2008), based on Olsson and Bergqvist's (1995) work, made another crack pattern classification depending on factors such as: explosive parameters, blast-hole geometry and rock mass properties.

2.2.4. Methods for evaluating the extension of the overbreak after blasting

The damage extension zone generated by blasting can be correlated with overbreak, which is defined as the void created during the excavation, in excess of an established perimeter or pay line (Mahtab et al., 1997).

Among the advantages of minimizing blast damage, the most important are related to: (i) improvement on safety work; (ii) reduction in support, scaling and secondary blasting; (iii) minimizing overbreak and cost related to removal extra mineral; (iv) improvement on roof and wall stability; (v) if the case be, damage prevention of nearby structures by controlling ground vibration

For a cautious blast design, the damage on the perimeter of the excavation is mainly induced by contour and buffer blastholes and it is created by a drop of strength, caused by the opening or shearing of newly generated or existing fractures or cracks (Scoble et al., 1997; Ouchterlony et al., 2001; Costamagna et al., 2018). In tunneling, damage can be categorized as major, minor or no damage, when there is rock falling, chips detachment or no visual damage, respectively (Costamagna et al., 2018). Damage assessment is analyzed through four main indexes: (i) Rock Tunneling Quality Index (Q-value, Barton et al., 1974) as classification of the ground condition for underground excavations; (ii) Blast Damage Factor (Hoek et al., 2002; Hoek, 2012) that estimates the global rock mass strength and the rock mass modulus; (iii) Blast Damage Index (Yu and Vongpaisal, 1996)

that correlates the mechanics and the effects of wave propagation into the rock mass; and (iv) failure approach index (Xu et al., 2017) that quantifies the rock mass damage through numerical simulations for tunnel support design.

Overbreak on the contour perimeter, which is defined as the void created during the excavation in excess of an established perimeter or pay line (Mahtab et al., 1997), is usually correlated with the damage extension zone which measures the quality of the blast. Overbreak and underbreak are mainly influenced by the geotechnical condition of the rock mass (rock disturbances and rock strength) and blast design parameters such as the explosive type, the charge concentration, the blast timing, the drill pattern and the drilling deviations (Ibarra et al., 1996; Oggeri and Ova, 2004; Singh et al., 2003; Singh and Xavier, 2005; Hustrulid, 2010; Johnson, 2010). Blasting affects the rock mass structure because of shock wave propagation (vibrations), gas pressure and stress redistribution (Singh et al., 2003; Hu et al., 2014).

Guidelines on construction have established an overbreak magnitude of 0.15-0.2 m and 0.1-0.15 m in crown and sidewalls, respectively (Mandal and Singh, 2009; Korea Highway Corporation, 2000; Cunningham and Goetzsche, 1990). The maximum overbreak distance allowed depends on each national legislation and special terms can be arranged between the two parts in the contract (Olsson, 2010; Costamagna, 2018). For example, Scandinavian countries present similar regulations for tunneling excavation requirements (Anläggnings-AMA in Sweden, InfraRYL in Finland and the Norwegian Public Roads Administration, NPRA, in Norway; Olsson, 2010; SN, 2004; NPRA, 2012). Consequences of a bad drilling can be short pulls of the rounds, increase of rock reinforcement due to extra overbreak in the rock mass, longer scaling and mucking time and bad control of grouting.

Two methods are mainly used to assess the condition of the void created after blasting: (i) the photogrammetric technique performs a 3D scan of the tunnel by using stereoscopic image pairs, i.e., merging two pictures of the same area taken from different position, and (ii) the Terrestrial Laser Scanner (TLS) equipment provides a measurement of the excavated void and provides a 3D view of the scanned surface in a point cloud. This is typically used to analyze the quality of the excavated void after blasting for tunneling constructions. An overview of several works based on those data follows.

Mahtab et al. (1997) proposed an approach for predicting the overbreak threshold during a tunnel construction with systematic jointed rock, based on a two-dimensional limit-

equilibrium analysis of wedges formed on the periphery of the excavation. Singh and Xavier (2005) studied the influence of rock mass properties, explosive characteristics and blast design parameters on the overbreak zone and proposed a new approach for the perimeter hole pattern and charge concentration to minimize damage and optimize productivity. Kwon et al. (2009) investigated the characteristics of the excavation damage zone (EDZ) in a tunnel construction and carried out a sensitivity analysis between several rock mass parameters obtained from laboratory and in situ tests to estimate the EDZ. Mandal and Singh (2009) analyzed the overbreak magnitude in several tunnels through scanner profiles. For that, both designed and excavated profiles were divided in three sections, left side, right side and crown, 120° each (floor was not considered since it was refilled before doing the scanner profile). They discovered that the highest overbreak was in the crown region, coinciding with the maximum magnitude region of stress generated after blasting. Olsson (2010) estimated an admissible maximum overbreak area of 25% over the theoretical area for a tunnel excavation. Kim and Bruland (2009, 2015) estimated a tunnel contour quality index from overbreak distances of the cross-section, the contour roughness and the longitudinal variation in each blasted round. It was developed to evaluate tunnel and rounds contour quality in drill and blasting.

However, no analytical models to predict the resulting over-excavation from geotechnical rock conditions and explosive characteristics have been found in the literature.

2.3. Measurement While Drilling (MWD).

The Measurement While Drilling (MWD) system was described by Schunnesson (1997) as a drill monitoring system that logs drilling data at predetermined length intervals providing information of the operational parameters involved in drilling.

2.3.1. Drilling methods

In mining and civil work operations, there are three types of drilling methods commonly used:

Rotary Drilling

Rotary drilling is mainly used in surface mining. The necessary energy transmitted to the bit to break the rock is a combination of a rotary force or torque, that spalls the rock due to the tensile failure along already developed fractures, and a feed force or thrust, that

keep the bit in contact with the rock at the bottom of the drill hole and aids to induce material failure due to crushing or compressive failure mechanisms (Khorzoughi, 2013). According to Schunnesson and Kristoffersson (2011), Khorzoughi (2013) and Ghosh et al. (2015) the most important parameters recorded during rotary drilling are divided in:

- Independent parameters: parameters that can be directly controlled by the operator.
 - Feed force or Thrust (N): Measures the hydraulic force necessary to keep the bit in contact with the rock at the bottom of the drill hole and thus, induces material failure due to crushing or compressive failure mechanisms (Khorzoughi, 2013).
 - Rotation speed (rpm): number of rotations per minute.
 - Air pressure (bar): air pressure necessary to remove rock cuttings from the blasthole.
- Dependent parameters: dependent both upon the independent parameters and the geological features of the rock.
 - Penetration rate (m/min): rate of penetration of the drill bit through the rock mass. It is influenced by the geo-mechanical response of the rock mass (Schunnesson and Kristoffersson, 2011).
 - Torque (N-m): is the rotational force multiplied by the blasthole radius to maintain the required rotation (Schunnesson and Kristoffersson, 2011). It depends on the rock type, weight of the bit and bit design (Peck, 1989).
 - Vibration (N-m/s): vibration measure on the drill rig and recorded during drilling through the rock mass (Schunnesson and Kristoffersson, 2011).
- An additional parameter is:
 - Hole depth (m): depth at which each sample of the above parameters is logged.

Rotary-Percussive Drilling

Rotary-percussive drilling is normally the system used by jumbos in tunneling. It is based on the combination of three operations:

- Percussion: impact of the bit against the rock at the bottom of the hole.
- Rotation: rotation movement of the bit at the bottom of the hole while impacting the rock.
- Thrust: hydraulic force necessary to keep the bit in contact with the bottom of the drill hole.

The process consists mainly of three operations (Thuro, 1997):

- The impact of the bit against the rock results in a tensile stress that creates radial cracks in the rock.
- A shear stress induced by the rotation of the bit in contact to the bottom of the hole generates parallel cracks to the surface and chips off the rock through them.
- The cutting chips are transported out to the hole by flushing them with water or air.

According to Schunnesson et al. (2011), the MWD parameters registered during rotary-percussive drilling monitoring are also divided in independent and dependent parameters:

- Thrust force, known as feed pressure, and rotation speed, are, as in rotary drilling, independent parameters:
 - Feed Pressure (bar): Feed pressure is necessary not only to keep the bit in contact with the bottom of the hole throughout the transmission of energy, but also to maintain a minimum force between bit and rock to maximize energy transfer to the rock.
 - Rotation speed (rpm): it is defined as the number of turns of the bit per minute.

Besides, two additional independent parameters are monitored:

- Hammer Pressure (bar) or Percussive Pressure: this is a measure of the impact pressure acting on the piston in the rock drilled.
- Damp Pressure (bar): it measures the pressure absorbed by the drill rig to prevent vibrations or undesired motion in the boom or drill rod.
- In line with rotary drilling, the dependent parameters monitored are:
 - Penetration Rate (m/min): rate of penetration of the drill bit through the rock mass.
 - Rotation Pressure (bar): it is the torque pressure required to rotate the bit at a defined speed.

In addition to hole depth or hole length (already defined for rotary drilling), further parameters that can be collected depending on the drill rig are:

- Water Pressure (bar): it is the pressure of the water used to flush the drill cuttings from the blasthole.
- Water Flow (l/min): it is the rate of water inflow into the drill rod.

- Time (hh:mm:ss): time at which each data point is monitored.

Wassara water-hydraulic ITH

The Wassara water-hydraulic in-the-hole (ITH) technology is used to drill the fan shaped long-holes in LKAB's Kiruna and Malmberget Mines, Sweden. According to this technology, water at high pressure (up to 180 bar) is used to power the impact mechanism of the hammer against the rock. The energy is transmitted through the drill rod in form of pressurized water, mechanical torque and mechanical feed force (Ghosh, 2017). Once the bit strikes the rock, water is released through the bit, removing drill cuttings from the blasthole (LKAB Wassara AB, 2018). Figure 2.5 shows a schematic view of the Wassara ITH hammer components.

Figure 2.5. Wassara ITH hammer components (LKAB Wassara AB, 2018).

Wassara ITH hammer positioning and drilling is fully automatic. In this system, feed pressure, rotation pressure and penetration rate parameters work similar to those for percussive-rotary drilling. Differences can be seen with the percussive pressure, in which water at high pressure is used to power the impact mechanism of the hammer to force the piston to impact with the bit (Ghosh, 2017). In this drilling mode, hammer pressure corresponds to percussive pressure because is the way the MWD files name this parameter for the Wassara water-hydraulic in-the-hole (ITH) technology. Hole depth and time are also recorded at preset sample intervals.

2.3.2. Relations between MWD parameters

The monitoring of the drilling performance is carried out by measuring sensors installed in the operational mechanisms related to the rotational, percussive and flushing mechanisms that the drill rig controls to performs the drilling. Sensor readings of several parameters involved in the drilling operation are digitally recorded. The management of the drilling is carried out by a control system that adjusts the values of the drill parameters to optimize the operation, to minimize damages to the drill rig (Schunnesson, 1998;

Schunnesson et al., 2011) and to reduce bit wear (Schunnesson 1997). As previously explained, parameters can be grouped as independent or dependent parameters. Therefore, there exist relations and responses between them as a result of the control system adjustment. The most important relations found from previous works are described below.

Effect of thrust or feed pressure on penetration rate and rotation pressure or torque

The feed pressure or thrust ensures that the bit keeps contact at the end of the hole to obtain a correct penetration rate. Schunnesson (1998) claimed that penetration rate tends to increase with an increase of thrust until a maximum penetration rate is reached (see Figure 2.6). From this moment, further increases in the feed pressure will make the penetration rate decrease and sometimes stall the drilling. On the other hand, sudden drops of the feed pressure values may cause a sharp increase of the penetration rate as a response to a high advance with no energy transfer to the rock. This situation takes place when drilling through a big fracture or discontinuity. Excessive feed pressure in jointed rock causes rod jamming and in weak rock, the bit cannot break the rock effectively due to a deficient cuttings removal by the flushing system, which results in slow penetration (Pearse, 1985; Schunnesson, 1998). The optimum feed pressure was defined by Hustrulid (1971) as the value that provides a penetration rate near to the maximum, so the bit is in contact with the bottom of the hole but without an excessive bit wear.

The torque rotation pressure necessary to drill is normally related to the bit resistance to wear and the friction between the drill rod and the hole walls (Schunnesson, 1998).

Pearse (1985) determined a clear correlation between the feed thrust, the penetration rate and the torque parameters for manual percussive drilling. Figure 2.6 identifies the level of feed thrust to obtain a maximum penetration rate. At this same feed thrust level, the torque force shows a knee shape on its curve, increasing its value with rises in the feed thrust after the maximum penetration.

Figure 2.6. Relationship between feed force, rotational torque and penetration rate (Pearse, 1985).

Effect of torque or rotation pressure on penetration rate

Schunnesson (1998) determined that at high penetration rate values the torque or rotation pressure normally tends to decrease; the weaker the rock and less resistance to penetration, the larger the penetration rate and the lower the rotation pressure needed to break the bottom of the hole. On the other hand, a low penetration rate is associated with hard rock of high resistance to be broken by the bit, which generates an increase of the rotation pressure values. Drilling through a fracture zone normally generates peaks or fluctuations in both penetration and torque parameters.

2.3.3. MWD analysis for Rock Mass Properties

Drill monitoring has been widely studied with the purpose of exploring the in-situ rock mass characteristics. In relation to studies developed for rotary drilling, Teale (1965) introduced the concept of specific energy (SE) as the energy required to excavate a unit volume of rock. He defined the specific energy as the combination of two parts: the work done by the axial feed force (first part of summation in Eq. 2-3) and the work done by the rotational torque (second part of summation in Eq. 2-3). He formulated the Specific energy (SE) as follows:

$$SE = \left(\frac{F}{A}\right) + \left(\frac{2\pi}{A}\right)\left(\frac{NT}{P}\right) \quad (2-3)$$

where, F is feed force (lb), A is cross-sectional area of the drill hole (in²), N is rotation speed (rpm), T is torque (lb·in) and P is penetration rate (in/min). Note that Teale's specific energy has the dimensions of pressure, or energy per unit volume.

Liu and Karen (2001) modified specific energy formulation developed Teale (1965) by introducing a specific surface energy indicator. They described that the specific surface energy is the energy required for breaking the intact rock and forming a new surface of unit area and that it is a property of the material. In this way, to transform the specific surface energy to specific energy it is needed the surface area per unit volume (specific surface area), that is related to the hardness of the material and the external conditions such as pulldown force and rotation speed:

$$E_a = \frac{1}{A_s A_e} \left(F + \frac{2\pi NT}{u} \right) \quad (2-4)$$

where, E_a is specific surface energy (N/m), A_s is specific surface area (m²/m³), A_e is excavation area given by the section of the drill bit (m²), F is pull down force (N) or feed force, N is rotation speed (rps), T is torque (N·m) and u is penetration rate (m/s)

Scoble et al. (1988) studied the variations of the monitored parameters, to define different geological formations. They found that changes in the penetration rate and the rotation torque were related to a variation of the geomechanical rock conditions. They used the Specific energy (Eq. 2-3) to interpret the behavior of the MWD parameters in different rock strength measured from geophysical logs.

According to MWD applications for the estimation of rock mass properties, Hatherly et al. (2015) compared the MWD data with the geological rock conditions, obtained by geophysical logs, and demonstrated that if rotary speed and weight on bit are kept constant, MWD measurements can determine rock properties. Leung and Scheduling (2015) proposed a coal-seam detection model called SEM (modulated specific energy), based only on MWD data. For rotary drilling, the SEM model can distinguish coal and no-coal rock layers with high accuracy by analyzing differences in the rotation-to-thrust power ratio.

Finally, Ghosh et al. (2015) proposed a filter methodology for outlier values found in raw data, based on both frequency analysis and practical experience. In addition, they demonstrated that monitored parameters present significant variations versus hole depth, that are not related to rock mass properties. They found that the calculated specific energy in long holes is highly influenced by the length of the hole.

For percussive and rotary-percussive drilling, Schunnesson (1996) used MWD parameters to develop a method for estimating Rock Quality Designation (RQD, Deere and Miller, 1966) values of the rock mass and for predicting discontinuities or fractures. The method was based not only on the penetration rate and torque parameters, but also on their variation, which shows a close correlation with the presence of large discontinuities, fractures or major faults. His results showed that the rate of penetration and the torque pressure are highly influenced by fractures. The rate of penetration increases when a fracture is found, and a high degree of variability is seen when drilling through an extended fractured zone.

$$TP_{var} = \sqrt{(TP - TP_{av})^2} \quad (2-5)$$

where TP_{var} is the torque pressure variability in a hole, TP is the torque pressure log data measured at 0.01 m intervals and TP_{av} is a moving average of the torque pressure from measurements at 0.5 m intervals.

Schunnesson (1998) introduced a methodology to normalize percussive drilling parameters to delete external influences associated with the length of the hole and the initial stage of drilling the blasthole. A rock dependent torque parameter was obtained, by normalizing it versus blasthole depth, thrust and penetration rate parameters. Similar work was made by Hjelme (2010).

Hydraulic rock mass properties were studied by Schunnesson et al. (2011). They suggested, for a rotary-percussive drilling, a model based on the location and amount of water inflow into the blasthole while drilling. The model correlates inflow water zones with increasing of water pressure and decreasing of water flow and dry fractures zones with decreasing of water pressure and increasing of water flow.

On the other hand, Peng et al. (2005) and Tang (2006) proposed a method for void/fracture detection and for rock mass geological properties prediction based on roof bolts drilling. To understand drill parameters variations in presence of fractures, they carried out several laboratory tests using different rock conditions. They found that the feed pressure parameter is a good detector of anomalies or discontinuities in the rock and a good estimator of the rock mass strength. Peng et al. (2005) designed a new methodology for normalizing the feed pressure based on determining the performance of the machine when drilling in the air (i.e. for no-load condition, outside the rock mass). This assesses how much feed pressure or rotation pressure is required for running the machine itself. Normalized feed pressure can be used to detect significant discontinuities

in the rock and different rock strengths, if the penetration rate and the rotation pressure are constant. Kaharaman et al. (2016) found a strong correlation between the penetration rate and the uniaxial compressive strength (UCS), the Brazilian tensile strength, the point load strength and the Schmidt hammer data. Van Eldert et al. (2018) assessed the extent of the damage zone from MWD parameters and ground penetration radar measurements recorded along the tunnel wall.

Based on correlating MWD data with rock mass mechanical measures, Schunnesson et al. (2012) explained a methodology to assess rock strength ranges based on an MWD hardness index provided by Atlas Copco software. For that, they used Schmidt hammer data to correlate MWD values with empirical rock strength measurements. Naeimipour et al. (2014) proposed a technique to improve void detection and rock strength estimations while drilling roof bolts holes, by using a 3D visualization of the ground and developing hazards maps for the tunnel wall. An optical and acoustic televiewer were used to record inside the blasthole once it was drilled. Results from drilling data were analyzed and correlated with the televiewer recordings.

Wassara water-hydraulic ITH (down-the hole) drilling mode has been studied by Ghosh (2017) and Ghosh et al. (2018). Part of the findings of this work has been taken as starting point for the analysis developed in Chapter 5. Ghosh (2017) analyzed parameters monitored by the specific drilling system used in the in the Kiruna and Malmberget mines. A fully-automated Atlas Copco SIMBA W6C drill rig equipped with Wassara W100 hydraulic ITH hammers was used for the drilling of upward fan shaped rings (LKABWassara AB, 2016). He showed an increase in the variation of the penetration rate and rotation pressure when drilling through a fractured zone. These variations are highlighted and calculated as the sum of the residuals over a defined interval along the blasthole (Schunnesson, 1996; Ghosh, 2017; Ghosh et al., 2018):

$$PR.V_i = \sum_{i=1}^{i+N} \left| \frac{\sum_{i=1}^{i+N} PR_i}{N+1} - PR_i \right|, \text{with } i = 1, 2, \dots, L \quad (2-6)$$

$$RP.V_i = \sum_{i=1}^{i+N} \left| \frac{\sum_{i=1}^{i+N} RP_i}{N+1} - RP_i \right|, \text{with } i = 1, 2, \dots, L \quad (2-7)$$

where, $PR.V_i$ is the penetration rate variability, $RP.V_i$ is the rotation pressure variability, N is the windows size (number of values considered in the interval, for the case under study, $N = 4$), i is the sample number, L is the length of the MWD signal, PR_i and RP_i are the monitored penetration rate and rotation pressure, respectively, in the i sample.

The variability of both penetration rate and rotation pressure ($PR.V$ and $RP.V$) at each i sample were combined, with a 50 % influence each, in a ‘fracturing’ parameter to make a more robust index. For that, the magnitudes of both parameters were scaled with the standard deviation of the raw parameter.

$$Fracturing_i = \frac{1}{5} \left[0.5 \cdot \left(\frac{PR.V_i}{std(PR)} \right) + 0.5 \cdot \left(\frac{RP.V_i}{std(RP)} \right) \right], \text{ with } i = 1, 2, \dots, L \quad (2-8)$$

where, $PR.V_i$ is penetration rate variability, $RP.V_i$ is rotation pressure variability, i is the sample number and L is the length of the MWD signal.

By combining the obtained fracturing index with the penetration rate, feed pressure, percussive pressure and rotation pressure parameters, Ghosh et al. (2018) used principal components analysis (PCA) to determine five classes of rock disturbance. For that, they recorded the blasthole wall along its trajectory with a video camera and identified relations between rock disturbances and fluctuation ranges in the MWD records. This methodology is further explained in section 5.3.

Finally, Atlas Copco AB, Sandvik and Bever Control have developed their own software packages (Tunnel Manager MWD, iSURE and Bever Control respectively), as a tool for planning, administration and evaluation of drill parameters. From the MWD files collected, the blastholes can be represented in 3D and hardness and fracturing maps are provided. The background for this information is however confidential as the companies consider it proprietary.

Chapter 3. DATA OVERVIEW

This chapter describes the data sets, the geological and geotechnical environments and the different equipment used in the five sites considered in this thesis: three tunnel projects, one underground mine and one quarry.

3.1. Tunneling

3.1.1. Site description

MWD data from three underground drill and blast excavation works have been considered in section 4. They are the following:

- The underground extension work of Bekkelaget's water reclamation facility, located in Oslo, Norway, is composed of five caverns of 460 m² section and around 180 m length each, a main access drift of 60 m² section and 850 m length and other sections. Figure 3.1 shows the construction plan, where the access drift (S1) is colored in yellow and the caverns in blue and purple (S2-S6). Other sections of the facility are colored in grey (S7), green (S8, S9) and red (S10) in Figure 3.1.

Figure 3.1. Plane of Bekkelaget's water reclamation facility (courtesy of OSSA Obras subterráneas S.A.).

- A roadway tunnel construction in Sorkjosen, North Norway. The main work involves the construction of one of the tunnel tubes with 4,585 m length and 80 m² cross section.
- A high-speed rail tunnel (El Espiño) in the line Madrid-Galicia, in the province of Orense, Northwest Spain. The tunnel consists of two roadways with 7.9 km length and 80 m² cross section.

The type of available data for the three sites have been collected in Table 3.1, where the files format, a brief explanation of the data and the number of files/rounds are summarized.

Table 3.1. Data available for tunneling case study.

Test site	Section	Type of File	Information	Rounds
Bekkelaget	S1	EXCEL	Geotechnical cond. and supports	172
		AutoCAD	Scanner profile (every 20 cm) from pk 34 to pk 844	4050(pr*)
		MWD	Measurement While Drilling	170
		PDF	Nominal Blast design	172
	S3, S8	MWD	Measurement While Drilling	42, 26
	S2, S5, S6	EXCEL	Geotechnical cond. and supports	23, 20, 28
		MWD	Measurement While Drilling	18, 68, 72
		PDF	Nominal Blast design	36, 86, 64
	S7, S9, S10	MWD	Measurement While Drilling	21, 77, 33
		PDF	Nominal Blast design	16, 27, 19
Tunnel in the North		PDF	Geology reports	53
		AutoCAD	Scanner profile (every 50 cm)	185(pr)
		MWD	Measurement While Drilling	86
El Espiño		PDF	Geotechnical rock conditions	28
		MWD	Measurement While Drilling	86

*pr: profiles

3.1.2. Geological description

Geological files in the underground extension work of Bekkelaget's water reclamation facility provide information related to *Q*-Barton Index and the type of supports used during the construction of the tunnel. A schematic view of the blasting face condition for each round is also drawn, in which discontinuities and rock mass intrusions are sketched.

Figure 3.2 represents a picture of a blasting face and its schematic view. Appendix 2 shows example of the geotechnical report for chainage 383 m.

Figure 3.2. Picture of a blasting face and the schematic view (courtesy of OSSA Obras subterráneas S.A.).

The facility extension work was built on competent rock mass, composed by gneiss with small tonalite and quartzite intrusions. Geological data analyzed for the access drift (S1, Figure 3.1) shows that the excavation was done in a good rock mass condition. Figure 3.3a shows the *RQD* values along the access drift and the *Q* index obtained from Eq. 2-1. Figure 3.3b shows the division of the *Q* index values into the different rock mass classes proposed in Figure 3.3a. The rock class is mainly of a good and fair condition and the *RQD* is nearly constant along the acces drift, ranging between 100-90 in most of the length. This indicates that the rock mass is very competent and that no significant rock mass changes are found along the excavation.

Figure 3.3. Rock Tunneling Quality Index in Bekkelaget; a) Rock Tunneling Quality Index versus chainage; b) classification of the ground condition: poor ($1 < Q \leq 4$), fair ($4 < Q \leq 10$), good ($10 < Q \leq 40$), very good ($40 < Q \leq 100$) according to Figure 2.3.

The roadway tunnel construction in North Norway was done in competent rock mass, composed by metamorphic rock with sedimentary origin (sandstones, slates) and

expansive clays with chlorites. The geological data analyzed show that the excavation was done mainly in a fair rock mass condition. Figure 3.4a shows the RQD values for part of the roadway tunnel and the Q index values from Eq. 2-1. Figure 3.4b shows the division of the Q index into the different rock mass classes proposed in Figure 2.3. It is observed that the rock class is mainly fair. In addition, the RQD varies along the tunnel in a range of values 90-25, which indicates that significant rock mass changes may exist along the excavation.

Figure 3.4. Rock Tunneling Quality Index in Sorkjosen; a) Rock Tunneling Quality Index versus chainage; b) classification of the ground condition: poor ($1 < Q \leq 4$), fair ($4 < Q \leq 10$), good ($10 < Q \leq 40$) according to Figure 2.3.

El Espiño tunnel was developed in sedimentary rock formed by sandstone and slates. Figure 3.5a shows the RMR values and Figure 3.5b shows the division of the RMR values into the different rock mass classes proposed in Table 2.2. It is observed that the rock class is mainly fair. In addition, Figure 3.5a shows that RMR varies along the tunnel in a range of values from 35 to 50, which indicates rock mass changes along the excavation.

Figure 3.5. RMR Index in El Espiño; a) RMR Index versus chainage; b) classification of the ground condition: poor ($21 \leq RMR \leq 40$), fair ($41 \leq RMR \leq 60$), according to Table 2.3.

3.1.3. Survey measurements

During the construction of the access gallery of the underground extension work of Bekkelaget's water reclamation facility, a laser scanner system was used to monitor the excavated void from each blast. It was set in the center of the excavation to be scanned and target spheres were installed along the wall of the main gallery in places with known coordinates. The software of the scanner identifies the position of the spheres in a post-analysis of the 3D cloud of points and trilaterates the location of the scanner by measuring distances from the latter to the spheres. Profiles of the excavated void in a direction perpendicular to the tunnel line are collected at steps of 0.2 m from the 3D cloud of points; each profile is identified by its respective chainage. Figure 3.6 represents a profile of the excavated void in perpendicular direction to the tunnel axis.

Figure 3.6. Profile of the excavated void in perpendicular direction to the tunnel axis.

During the construction of the roadway tunnel, a laser system installed on the top side of the jumbo was used to locate the jumbo inside the tunnel before drilling and to monitor the excavated void from each blast. The surveying of the resultant contour was done perpendicular to the direction of the tunnel axis with a preset density of the points. Measured sections were extracted in AutoCAD (AutoCAD, 2017) format at steps of 0.5 m; each profile is identified by its respective chainage. Figure 3.7 represents a profile of the excavated void in perpendicular direction to the tunnel axis. No data of this type was available for El Espiño tunnel (see Table 3-1 above)

Figure 3.7. Excavated section road tunnel in Sorkjosen.

3.1.4. MWD Data in tunneling works

Table 3.2 summarized the working conditions (wc) and the test site where data was gathered. All jumbos involved in the tunneling works were manufactured by Atlas Copco; five of them (3 jumbos XE3C and 2 jumbos XL3D) had three booms and one (E2C) had two booms. They all used percussive-rotary top hammer drilling mechanism with different levels of ABC (Advanced Boom Control) total system. Data comprises production face drilling holes of short length (4 – 5 m), drilled by using only one rod of 5 m length and 38 mm diameter and a bit of 46 mm diameter. Drill directions are almost parallel with a maximum lookout angle of 10° and 15° for the contour and bottom holes, respectively. The version of the revision control system (RCS) of the jumbo is indicated in Table 3.2 for each working condition (wc) in addition to the jumbo model, the ABC System and the rock type.

The data set comprises 687 blasts and more than 65,000 signals from 11 different working conditions; these are identified with the number assigned to the jumbo followed by a letter when they use a different drilling mode, or if the rock type or the sample interval vary. This occurs, for instance, with Jumbo 2 that worked on both semi-automatic (wc 2a) and manual (wc 2b) levels.

Table 3.2. Description of data characteristics of working conditions of Jumbos.

Jumbo	wc ^a	Jumbo		RCS ^b	Sample interval (m)	Test Site	Rock type	No. blast
		Model	ABC Total System					
1	1	XE3C	Semi-auto	4.9	0.1	Bekkelaget	Gneiss	159
2	2a	XE3C	Semi-auto	3.7	0.1	Bekkelaget	Gneiss	76
	2b	XE3C	Manual	3.7	0.1	Bekkelaget	Gneiss	90
3	3	E2C	Semi-auto	4.8	0.1	Bekkelaget	Gneiss	29
4	4a	XE3C	Semi-auto	4.7	0.1	Bekkelaget	Gneiss	95
	4b	XE3C	Semi-auto	4.7	0.1	Sorkjosen	Sandstone, Slates & Clays	67
5	5a	XL3D	Semi-auto	2.185	0.02	Sorkjosen	Sandstone, Slates & Clays	48
	5b	XL3D	Semi-auto	2.185	0.02	El Espiño	Sandstone & Slates	15
	5c	XL3D	Semi-auto	2.185	0.2	El Espiño	Sandstone & Slates	21
6	6a	XL3D	Semi-auto	4.0	0.1	El Espiño	Sandstone & Slates	49
	6b	XL3D	Semi-auto	4.0	0.2	El Espiño	Sandstone & Slates	37

^(a) wc is working conditions; ^(b) RCS is revision control system

3.2. Underground mine: Malmberget mine

The Luossavaara-Kiirunavaara AB's (LKAB) Malmberget underground iron ore mine is located close to the municipality of Gällivare, Sweden. The mine contains twenty iron orebodies spread over an underground area of about 5 by 2.5 km. Twelve of those orebodies are currently being mined by the mining company Luossavaara Kiirunavaara AB (LKAB). The orebodies are mainly composed by magnetite with hematite intrusions in some areas, mostly in the western field. The current production areas are normally between the previous haulage level at 1000 m and the new one at 1250 m. The dip of the ore bodies varies between 15° and 75°, with an average dip of 45° - 50° (Nordlund, 2013; Ghosh, 2017). The host rock is composed by vulcanite rock, skarn, granite and biotite schist (Quinteiro et al., 2001; Wettainen, 2010; Nordlund, 2013; Umar et al., 2013).

Sublevel caving is the mining method used for ore extraction in this mine. This is a mass mining method based on the utilization of gravity flow of the blasted ore and the caved barren rock (Hartman, 1992). It is normally used for massive, steeply-dipping orebodies with considerable strike length such as that in Malmberget, when open pit is not economically viable. In this method, the ore is extracted from the top of the orebody and it is developed downwards via sublevels at regular intervals. Each sublevel features a layout formed by parallel drifts along or across the orebody, depending on its length and

width (Hustrulid and Bullock, 2001). The sublevel drift normally starts from the footwall in the direction to the hanging wall. The ore section above the drift is drilled in a fan shaped long-blastholes. Blasting is carried out in slices starting from the hanging wall and retreating towards the footwall. To follow a straight front, adjacent drifts should be mined at a similar pace. The blasted ore falls gradually into the drift opening as the material is mucked out. Drilling, charging and blasting operations must be timed to schedule the mine production. Figure 3.8 represents a view of the different stages of the sublevel caving mining method.

Figure 3.8. Stages of the sublevel caving mining method (Britannica, 2018).

3.2.1. MWD data in the Malmberget Mine

Malmberget mine uses transversal and longitudinal sublevel caving mining as the method for ore extraction. Different drill rigs are used for the development of the parallel drifts and the fan shaped long-holes. In the first case, horizontal face drilling holes of short length are necessary, and jumbos are similar to those used in tunneling operations. For the second stage, Atlas Copco SIMBA W6C drill rigs equipped with a hydraulic Wassara ITH hammer are used to drill the fan shaped holes. For the case under study, only data from the latter are used.

Six Atlas Copco SIMBA W6C drill rig equipped with hydraulic Wassara In-The Hole (ITH) hammer are used for production drilling in mine. The production holes are drilled in a fan shaped pattern with holes between 10 and 50 m length. Drilling is done using 102 mm extension tubes with 2.3 m length and 4,5” (115mm) bits. diameter. A 4.5 revision control system (RCS) was installed in the rig. Table 3.3 lists the MWD parameters monitored; it also gives the units and the acronym used for each of them. They are described in section 2.3.3.

Table 3.3. MWD parameters description.

Parameter	Acronym	Unit
Hole Depth	HD	m
Penetration Rate	PR	m/min
Percussive Pressure	PP	bar
Feed Pressure	FP	bar
Rotation Pressure	RP	bar

Two sets of MWD data have been gathered for the analyses. Table 3.4 shows the available data, gathered from five different orebodies, used for the development of the geotechnical rock condition block model and, from it, the risk of collapse model. For that, the underground charging operation at the mine was followed for 102 blastholes (11 rings). The purpose is to compare the predicted risk of collapse in the blasthole with actual problems encountered by the charging personnel. The maximum depth of the charging hose introduced in the blasthole, in relation to the respective blasthole length, gives a measure of whether the charging hose was obstructed in the blasthole or not and, in case of obstruction, the depth at which it occurs.

The application of the two models in full scale is carried out for data from two orebodies (orebody 1 and orebody 4) located at different depth levels. Table 3.5 describes this second data set for the representation in full scale of both geotechnical rock condition and risk of collapse models, which comprises 20 drifts and 5060 fan-holes.

Table 3.4. MWD data for the development of the risk of collapse model.

Orebody	No. rings	No. Blastholes
1.	6	63
2.	1	7
3.	2	18
4.	1	7
5.	1	7

Table 3.5. MWD data for the representation of both geotechnical rock condition and risk of collapse models in full scale.

Orebody	No. drifts	No. Blastholes	Level (m)
1.	9	2662	1052
4.	11	2398	1031 & 1056

3.3. Quarry: El Aljibe

El Aljibe quarry is located in the milonyte band of Toledo, Spain that responds to a ductile shear zone which was developed at the end of the Variscan orogeny. The quarry mines a belt of metamorphic rocks with 1–1.5 km length and 400 m height (Martínez-Salanova et al., 2009). These metamorphic rocks correspond to mylonite and cataclasite series, originated at high pressure and temperature within a ductile fault regime. Lithologies found in the quarry are ultramylonite, orthomylonite, ultracataclasite and protomylonite. However, the dominant lithology detected in the working area has been orthomylonite. Discontinuous strips of ultracataclasite were also detected, but they had not enough thickness and length to be considered representative in the rock mass Figure 3.9 shows the location of the quarry in the geological regional map of the zone.

Figure 3.9. Geological regional map of Aljibe quarry (IGME, 2018).

Chapter 4. MWD APPLICATION FOR TUNNELING

This chapter carries out a thorough analysis of the drill-monitoring data logged by jumbo rig in tunneling works. From the management of data in three sites, Bekkelaget, Sorkjosen and El Espiño, the mutual relation between Measurement While Drilling (MWD) parameters in percussive-rotary drilling is firstly investigated to find the parameters that govern the drilling. This allows to limit the number of variables to be used in a sound rock mass characterization. The navigation system of the jumbo and blasthole positioning are also described.

The quality of the drilling from data gathered in Bekkelaget is also investigated with a view to quantify the error of the MWD system with respect to the actual end position of the blasthole logged and to assess the influence of the rock structure in the drilling path.

Finally, the damage from blasting to the remaining rock mass is analyzed from data in Bekkelaget, with the purpose of developing a drilling index from the MWD parameters, that predicts potential overbreak zones in the contour of a tunnel excavated by blasting.

This chapter is made up of four of the papers listed in Table 1.1. Section 4.1 corresponds to paper A (Appendix A), “MWD parameters and drilling control system”, published as a conference paper in the World Tunneling Congress 2017, and paper C (Appendix C), “On the mutual relations of drill monitoring variables and the drill control system in tunneling operations”, published in the journal Tunneling and Underground Space Technology. Section 4.2 describes the jumbo navigation system, which is included in papers D, E and G (see Table 1.1). Section 4.3 corresponds to the research published in paper E (Appendix E), “Assessment of drilling deviations in underground operations” in the journal Tunneling and Underground Space Technology. Finally, the investigation developed in section 4.4 correspond to paper D (Appendix D), “Detection of potential zones in tunnel blasting from MWD data” and part of it was the starting point of paper B (Appendix B) “Assessment of contour profile quality in D&B tunneling”, both published in the journal Tunneling and Underground Space Technology.

4.1. Drilling control system analysis

The mutual relation between Measurement While Drilling (MWD) parameters in percussive-rotary drilling is investigated here. Nowadays, jumbos manufactured by Atlas Copco allow to automatize the drilling operation by installing systems in the drill rig such

as the ABC (Advanced Boom Control), which helps the operator to follow a predesigned drill pattern and optimizes the drilling. The ABC system enables to program the drill rig in three operational automatization levels: basic, regular and total (Nord and Appelgren, 2001; Atlas Copco, 2010). At the basic level, both positioning and drilling are done manually; the system allows the operator to watch in a control screen and record the collaring and alignment of the boom, whereas monitoring the drilling is not available. At the regular level, the operator follows a predesigned drill plan, by controlling boom and feeder manually; drilling and data logging are done automatically. The total level authorizes the operator to follow a predesigned drill plan, to log the drill hole operation and to switch collaring and drilling between manual, semi-automatic and fully-automatic mode. In the manual mode of the ABC total system, the operator moves the boom and feeder manually and takes control of the drilling process. In the semi-automatic mode, the operator controls also boom and feeder manually, but the drilling is automatic. The fully-automatic ABC total system enables program collaring, alignment and drilling automatically (Atlas Copco, 2010). Other manufactures like Sandvik have similar automatization in their jumbos by installing the iDATA control system in the drill rig, in combination with the Sandvik iSURE software (Sandvik, 2017).

Some authors have provided interpretations of the performance of the control system. Schunnesson (1998) defined a normalized torque pressure as rock-dependent drilling parameter. Schunnesson et al. (2011) indicated that percussive pressure, feed pressure and rotation speed are the rock-independent drilling parameters. Peng et al. (2005) described the data control unit installed in the drilling system, in which some pre-set MWD parameters and their thresholds could be selected. They claimed that penetration rate, rotation speed and feed pressure were the key parameters in the performance of the drilling operation. Danell (1989), Gui et al. (2002) and Cooper et al. (2004) stated that the drilling must hold constant force and rotation speed. This background points out some controversy in the performance of the control system as to what are the mutual relations between MWD parameters and their precedence, hence the need of analyzing the actual signals from a cross-correlation standpoint.

4.1.1. Statistical Tools

Digital signals that measure the operational parameters of the jumbo are monitored as a discrete set of values at equal depth intervals. Statistically, MWD signals may be regarded

as a discrete time series (time being in this case equivalent to depth). A stochastic process may be applied to these series (Box and Jenkins, 1976) to analyze their correlations.

Cross-correlation

The correlation of two signals sequence is a measure of the statistical degree to which the two signals are similar (Box and Jenkins, 1976; Proakis and Manolakis, 1996). Cross-correlation is defined from the cross-covariance function. Let X and Y be two variables available as depth series x_t and y_t of length N , the cross-covariance C_{xy} between both signals is given by (Box and Jenkins, 1976):

$$C_{xy}(k) = \begin{cases} \frac{1}{N} \sum_{t=1}^{N-k} (x_t - \bar{x})(y_{t+k} - \bar{y}), & \text{with } k = 0, 1, \dots, N \\ \frac{1}{N} \sum_{t=1}^{N-k} (x_{t+k} - \bar{x})(y_t - \bar{y}), & \text{with } k = 0, -1, \dots, -N \end{cases} \quad (4-1)$$

$$(4-2)$$

where $\bar{x}$ and $\bar{y}$ are the means and k is the shift or lag parameter.

The cross-correlation r_{xy} is defined as a dimensionless coefficient by scaling the cross-covariance function C_{xy} with the product of the variances S_x and S_y of the two series at $k=0$:

$$r_{xy}(k) = \frac{C_{xy}(k)}{S_x S_y}, \quad \text{with } [k = 0, \pm 1, \pm 2, \dots, \pm N] \quad (4-3)$$

where

$$S_x = \sqrt{C_{xx}(0)} \quad \text{and} \quad S_y = \sqrt{C_{yy}(0)} \quad (4-4)$$

The cross-correlation function is not symmetric about $k=0$ (Box and Jenkins, 1976). Eqs. (4-1) and (4-2) describe the direction in which a signal (i.e. depth series) is shifted over the other. Eq. (4-1) shows the depth series y_t shifted forward k units with respect to the series x_t . In this case, the x_t series leads the y_t series or, equivalently, y_t lags x_t . Eq. (4-2) indicates the opposite situation by shifting x_t forward y_t and analyzes the correlation between the two series when y_t leads x_t (Proakis and Manolakis, 1996). Cross-correlation results are represented in the form of correlograms, where r_{xy} is plotted as a function of the lag k . Since cross-covariance is scaled by the variances, the result of the cross-correlation between two signals is ranked from 1 to 0 for positive correlation and from 0 to -1 for negative correlation, being ± 1 maximum correlation and 0 no correlation. The highest absolute value of r_{xy} indicates the maximum correlation between two time series and the lag (k value) at which this occurs shows the delay between them.

From the results of the cross correlation, the existence and location of a maximum and other peaks in the correlogram may indicate different relationships between time series. Although several works (Yevjevich, 1972; Box and Jenkins, 1976; Hamilton, 1994; Proakis and Manolakis, 1996; Antoniou, 2006; Shumway and Stoffer, 2011) have described the cross-correlation analysis and its applications, the interpretation of the type of correlograms that may be obtained is generally very limited. With the purpose of filling this gap, an interpretation of the common correlograms follows.

Figure 4.1 shows seven possible correlogram types that may be found for two MWD signals X and Y . Since Eqs. (4-1) and (4-2) show different shift directions between the time series, each correlogram has been divided into two zones, one for each sign of the k value. The leadership direction in each zone between variables X and Y is written $X \rightarrow Y$ or $Y \rightarrow X$, i.e., X leads Y for positive lags or Y leads X for negative lags, respectively. The red lines show the 95 % coverage of the cross-correlations, so that correlation values outside this band indicate significant correlation. The interpretation of the seven correlograms in Figure 4.1a to 4.1g is the following:

- a) Significant correlations or fluctuations in the correlogram signal, including the maximum, are found in only one side of the correlogram (Figure 4.1a). This indicates a causal relationship with a delay between the two series and memory in the response, i.e., there are several peaks at different lags, which may point out that the output is influenced by present and past input values. The leadership direction between the two signals falls into the correlogram zone with significant correlations (in the case of Figure 4.1a, Y leads X). This also applies to graphs b (Y leads X) and c (X leads Y) of Figure 4.1.
- b) Maximum correlation on one side of the correlogram but no other significant correlations (Figure 4.1b). This represents a causal relationship with a delay between the two series but no memory in the response, i.e., the two series are correlated only once, in contrast with case a. This may indicate that the output is only influenced by present input values.
- c) Maximum correlation at lag 0 and significant correlations or fluctuations in the correlogram signal on one side of the correlogram. This reveals a causal relationship with instantaneous reaction between input and output and memory in the response (Figure 4.1c).

- d) Maximum correlation at lag 0 but no other significant correlations or fluctuations in the correlogram signal. This indicates correlation between the two series but none of the signals leads the other (Figure 4.1d). This correlogram shows an independent behavior between them and thus a non-causal relationship.
- e) All correlation values within the confidence band with no apparent differences at positive and negative lags. This indicates a non-causal relationship (Figure 4.1e).
- f) Non-significant correlations on one side and flat response on the other side of the correlogram (Figure 4.1f). This points out a non-causal relationship with certain memory in the response and thus, a slight influence of the input on the output response. The leadership direction between the two series belongs to the correlogram zone with higher fluctuation: in the case of Figure 4.1f, X leads Y .
- g) Maximum correlation on one side of the correlogram and significant correlations or fluctuations on the other side (Figure 4.1g). These indicate contradictory results and therefore a non-significant input-output dependence (non-causal system).

Figure 4.1. of cross-correlations between X and Y MWD signals: a) Causal relationship with delay and memory response between parameters; b) Causal relationship with delay but no memory response between parameters; c) Causal relationship with instantaneous reactions and memory in the response; d) Non causal relationship with two correlated signals with no leadership; e) No significant correlation; f) No significant correlation but memory in the response; g) Contradictory results (non-causal system).

Auto-Correlation

The auto-correlation function reveals the degree of correlation between parts of the same signal when it is shifted with respect to itself. In the same way as cross-correlation, the

auto-correlation is defined by the auto-covariance function, scaled by the variance of the series at $k=0$:

$$r_x(k) = \frac{C_{xx}}{S_x S_x}, \text{ with } k = 0, \pm 1, \pm 2, \dots, N \quad (4-5)$$

where, C_{xx} is obtained replacing y_{t+k} by x_{t+k} and $\bar{y}$ by $\bar{x}$ in Eq. (4-1).

The auto-correlation function, which is symmetric about zero, is commonly used for the interpretation of the signal properties (stationarity, periodicity or repeating patterns, frequency) and the detection of non-randomness in data (Box et al., 1994). For the case under study, auto-correlation has no interest for the analysis of the relation between MWD parameters and the performance of the control system, but it is calculated and shown together with the cross-correlation for completeness. Notwithstanding, further tests based on the study of the auto-correlation applied to MWD signals may be an interesting subject of research towards a deeper insight into drilling performance.

4.1.2. Data processing and analysis

Underground drill and blast excavations works described in section 3 have provided data to this study. Table 3.2 shows the working conditions (wc) where data was gathered. Table 4.1 lists the parameters monitored by percussive-rotary jumbos; it also gives the units and the acronym used for each of them. They have been described in detail in section 2.3.1 (pp. 17).

Table 4.1. MWD parameters description.

Parameter	Acronym	Unit
Penetration Rate	PR	m/min
Hammer Pressure	HP	bar
Feed Pressure	FP	bar
Damp Pressure	DP	bar
Rotation Speed	RS	rpm
Rotation Pressure	RP	bar
Water Pressure ⁽²⁾	WP	bar
Water Flow ^{(1) (2)}	WF	l/min

⁽¹⁾ For working condition 5a, this parameter is not available.

⁽²⁾ For working conditions 5b, 5c, 6a and 6b these parameters are not available.

Figure 4.2 shows, as an example, all MWD signals for one hole. Penetration rate, rotation pressure and damp pressure show a drift with depth. Schunnesson (1998) related this dependence with the increase of the frictional resistance between the drill string and the

walls of the blasthole, the reduction of the available pressure over the hammer, the decrease of the flushing efficiency with depth, and the bit wear.

As auto-correlation and cross-correlation require stationary or depth-invariant series (Box et al., 1994), data should be processed (detrended) to compensate the hole depth effect. Data differentiating is often used to remove stochastic trends (Box et al., 1994). For our data, it is enough to calculate the first difference between elements of each signal; considering a discrete depth series Y at a period t , the first difference is defined as $Y_t - Y_{t-1}$; the resulting series has a near-zero mean value. This can be seen in Figure 4.2b, where the detrended signals from Figure 4.2a are plotted. Detrended parameters have been noted by using the same acronyms as in Table 4.1 with an asterisk.

Figure 4.2. Detrending technique to remove hole depth dependence on MWD signals from Jumbo 1 for hole No 57 of the blast located at the chainage 399 of the main gallery of Bekkelaget's construction. a) Raw signals; b) Processed signals. The units of the parameters are given in Table 4.1.

Considering the detrended full signals for each hole, auto-correlations have been calculated for the available MWD parameters, whereas cross-correlation has been calculated for all pairs of parameters. Calculations have been made with Matlab (2016) using a window (equivalent to the range of k in Eqs. (4-1) and (4-2) of 2 m between signals. A smaller window was not possible because some of the working conditions, with 0.2 m logging intervals (wc 5c and 6b, Table 3.2) result in short series that do not allow to get peaks other than the main one.

Figure 4.3 shows, in a correlation matrix, the resulting correlograms and auto-correlograms (main diagonal) from a single hole of one of the working conditions under study (wc 3 in Table 3.2). The MWD parameters in the abscissa and ordinate of the correlation matrix correspond, for each graph, to the X and Y signals as from Eqs. (4-1) and (4-2), respectively. As explained in section 4.1.1 (cross-correlation), the leadership

direction between the two MWD parameters falls to the correlogram zone with significant correlations or the side with higher signal fluctuations. Results from Figure 4.3 can be matched with the interpretation of the correlograms in Figure 4.1: typical correlograms of causal relationships are colored in green in Figure 4.3. There is a delay with memory response (Figure 4.1a) between penetration rate (PR) – hammer pressure (HP) and PR – feed pressure (FP). HP is delayed with respect to FP but with no memory response (Figure 4.1b). FP induces an instantaneous response with memory on damp pressure (DP) and rotation pressure (RP) (Figure 4.1c). Other relations between parameters do not give relevant information about the performance of the control system. For instance, RP is independent of PR (Figure 4.1d) and correlation RP – water pressure (WP) (Figure 4.1e), HP – rotation speed (RS) and FP – RS qualify as non-significant to the 95% level. However, for the last two pairs, there is a clear memory in the response (Figure 4.1f). Contradictory results occur between HP and both DP and RP (Figure 4.1g).

Figure 4.3. Cross-correlation matrix of the MWD parameters for hole 7 drilled with Jumbo 3 at the chainage 99 of cavern 1 of Bekkelaget's construction.

Peaks and fluctuations (at positive and negative lags) that are apparent for a single hole (Figure 4.3), become blurred when representing several holes in the same plot, which complicates the detection of a global pattern between the parameters. To solve this, a new methodology has been developed in this Thesis. First, for each working condition, a mean correlogram per blast is calculated (this can be done as the cross-correlation functions are calculated over the same lag). Secondly, the standard deviation of the correlations on each side of the correlogram are calculated for each hole, which indicates the side of the correlogram with higher fluctuations. In order to consider only secondary correlations,

the maximum peak must be avoided in the standard deviation calculation. As the peak correlation is always within ± 3 lags (see Figure 4.3), the standard deviation is calculated from lag +3 to the maximum lag on the positive side and from lag -3 to the minimum lag on the negative so that, for every pair of signals, one standard deviation is obtained for positive lags and another for negative. The distributions of standard deviations for each working condition are represented in the form of two histograms: one for positive lags and another for negative.

4.1.3. Results and discussion

The analysis of the mean correlogram and the standard deviation histograms shows seven different types of correlations between the MWD parameters. Their characteristics are described in Table 4.2, in line with Figure 4.1. The fluctuations at each side of the correlogram are compared from the histograms of the standard deviation at positive and negative lags; the empirical cumulative distribution functions corresponding to each histogram have been compared by using a two-sample Kolmogorov-Smirnov test (Marsaglia et al., 2003); where significant differences are detected, there is memory response.

Table 4.2. Description of the seven correlation results.

Correlation type (Figure 4.1)	Mean Correlogram		Histograms of the standard deviation	
	Existence of Significant Max. Peak.	Lag of the peak (k)	Existence of Significant differences	Sign of the lags with higher standard deviation ⁽¹⁾
A	Yes	$k \neq 0$	Yes	Same as Max. Peak
B	Yes	$k \neq 0$	No	No differences
C	Yes	$k = 0$	Yes	Positive or Negative
D	Yes	$k = 0$	No	No differences
E	No	-	No	No differences
F	No	-	Yes	Positive or Negative
G	No	$k \neq 0$	Yes	Opposite to Max. Peak

⁽¹⁾ This is the lag of the histogram with significant higher standard deviation.

To illustrate the correlations defined in Table 4.2, data recorded for jumbo 1 (wc 1, Table 3.2) is considered as an example, which comprises 159 blasts. Correlations for the rest of the working conditions are shown in Appendix 1, in line with the explanation that follows here. Figure 4.4 shows the mean correlograms per blast for the MWD parameters recorded. The correlograms are plotted with a different color according to the correlation characteristics between MWD parameters given in Table 4.2, where the letter codes A through G correspond to the cross-correlation types described in Figure 4.1; only

correlation type D does not occur in this case. It is found that for each pair of variables, the 159 blasts follow the same mean correlogram type (Figure 4.4), which validates the consistent behavior of the control system. Figure 4.5 shows the histograms of the standard deviation for each of the two correlogram zones. The p-value of the two-sample Kolmogorov-Smirnov test is also given on the top-right side of each graph; when p is less than 0.05, the distributions of standard deviation are different at a 95 % confidence level. The correlation type is denoted by the same color as in Figure 4.4. In the same way as for Figure 4.3, the MWD parameters in the abscissa and ordinate of the correlation matrix correspond to the X and Y variables, respectively.

Figure 4.4. Representation of the mean correlograms; each graph shows the plot of the 159 mean correlograms for working condition 1; only values at ± 10 lags are given in order to better show the main peak.

Figure 4.5. Histograms of standard deviation at positive and negative lags for each of the correlograms of the 159 blasts drilled in working condition 1. Black histograms are the standard deviation on negative lags and white histograms on positive.

Among the correlations in Table 4.3, only A, B and C types show a clear input-output response between MWD parameters. Figures 4.4 and 4.5 show that feed pressure (FP) leads penetration rate (PR) and hammer pressure (HP) with positive correlation and delay in the output response (A type). FP also leads damp pressure (DP) and rotation pressure (RP) with immediate response, i.e. lag = 0 (C type). HP leads PR (A type) and DP leads PR (B type) and RP (C type); such correlations may be a consequence of the leadership of FP on PR, HP and DP. These results suggest, for jumbo 1 (wc 1), that feed pressure is the parameter used by the control system to lead the adjustment of the other parameters of the jumbo.

The control system is normally based on three main operational modes (Schunnesson, 2017): (i) collaring, (ii) ramp-up, both of which control the increase of the drilling pressure to minimize hole deviations, and (iii) normal drilling, which controls the performance of the parameters to optimize the operation and minimize damages in the boom. As can be seen in Figure 4.2a, the feed pressure shows, initially, a sharp rise until it reaches a pre-set threshold at which it stabilizes. The same analysis has been carried out by applying the correlation to the signals in the ramp up phase on one side and the normal drilling on the other. For that, the signals of the 8 parameters have been divided considering the point at which the feed pressure stabilizes. Values previous to this point form the ramp-up data set and forward values the normal drilling data set. Figures 4.6 and 4.8 show the mean correlograms per blast for the ramp-up and normal drilling phases,

respectively, for working condition 1 (the same working condition that has been analyzed above in Figures 4.4 and 4.5). Due to the limited depth of the records in the ramp-up phase, the cross-correlation in this case has been done only for a window of 4 lags. Figures 4.7 and 4.9 show the histograms of the standard deviation for each of the two correlogram zones for the ramp-up and normal drilling, respectively.

Figure 4.6. Representation of the mean correlograms for the ramp up operational mode for working condition 1; each graph shows the plot of the 159 mean correlograms.

Figure 4.7. Standard deviation at positive and negative lags for each of the correlograms of the 159 blasts for the ramp up operational mode for working condition 1. Black histogram plots the standard deviation on negative lags and white histograms on positive.

Figure 4.8. Representation of the mean correlograms for the normal drilling operational mode for working condition 1; each graph shows the plot of the 159 mean correlograms.

Figure 4.9. Standard deviation at positive and negative lags for each of the correlograms of the 159 blasts for the normal drilling operational mode for working condition 1. Black histogram plots the standard deviation on negative lags and white histograms on positive.

Results of the cross correlation for both ramp-up and normal drilling phases are similar to the results obtained in the analysis of the full signals in all working conditions. For the case shown of working condition 1, slight differences have been found in comparison of Figures 4.4 and 4.5 and Figures 4.6 to 4.9. For the ramp-up mode, differences in the relation HP-PR (B type in ramp-up mode instead of A type for the full signal) and relations between RS, WP, WF and PR, HP (E type in ramp-up mode instead of F type for the full signal) are obtained. For the case of normal drilling, differences are identified

between FP-HP (C type in normal drilling mode instead of A type for the full signal) and DP-RP (D type in normal drilling mode instead of C type for the full signal). These differences might be due to the lesser amount of data in the analysis (especially for the ramp-up case). Notwithstanding, they do not introduce any significant change in the leadership behavior between parameters and evidence that the feed pressure is the parameter with highest influence in the performance of the control system.

Results for the analysis of the full signal for the eleven working conditions under study are summarized in Table 4.3. Cells corresponding to correlation types A, B and C show the leadership direction between parameters, the lag in the response (k) and the sign of the correlation symbol; $r (+)$ indicates a positive correlation and $r (-)$ a negative one. Color code is the same as in Figures 4.4 and 4.5.

The correlation type (C) between feed pressure, damp pressure and rotation pressure (FP-DP and FP-RP) is the same independently of the working conditions involved (with exception of wc 2.b, with manual drilling mode). This outlines that these three parameters are key indicators of the control system in which the feed pressure provides a measure of the hydraulic pressure of the feeder that allows to keep the bit in contact with the rock while it rotates. This induces a torque that is measured by the rotation pressure. Additionally, the pressure provided by the damper should react immediately to changes in the hydraulic pressure to prevent undesired motion of the boom (Peng et al., 2005; Schunnesson et al., 2011).

For jumbos working in semiautomatic mode, relations in which penetration rate is involved HP-PR, FP-PR, DP-PR and relations between FP-HP, qualify as A, B or C types depending on the working condition. This reflects changes only in the lag of the response (i.e. delayed response, A or B types and instantaneous response, C type), whereas the sign of the correlation and the leading parameter do not vary. The relations show that FP always leads PR, HP, RP and DP.

Results for working conditions 3, 4a, 4b and 6a are in line with those for working condition 1. All these cases share the same automatization level setting (semi-automatic) and the sampling rate, whereas the rock drilled varies for some of them. This suggests that the inter-correlations between MWD parameters are independent of rock characteristics. On their side, correlations in working conditions 2a, 5a, 5b, 5c and 6b qualify as C type instead of A, suggesting that the response of the hammer pressure and penetration rate to changes in feed pressure occurs immediately. The rock mass was also

different for some of these working conditions, which confirms the (otherwise expected) independence of the control system from the rock mass. Differences in the version of the revision control system (RCS, Table 3.2) should explain this behavior: RCS versions higher or equal than 4.0 (wc 1, 3, 4.a, 4.b and 6.a) show similar results in contrast to RCS lower than 4.0 (2.a, 5a, 5b, 5c and 6b).

Interestingly, for jumbos working with 0.02 m sampling length (jumbo 5, wc 5a and 5b) no delay in the response is observed. This may indicate a different performance of jumbo 5 or a spurious effect caused by an older RCS version (Table 3.2). For jumbos with logging interval of 0.2 m (wc 5c and 6b), the fact that no delay is observed is a consequence that, for them, a zero lag (no apparent delay) includes any actual delay from 0 to 0.2 m; this corresponds to lags zero and one for logging interval 0.1 m.

For jumbos working in semiautomatic mode, most of the relations are of A or C type for significant correlations (i.e. between PR, HP, FP, DP and RP) and of F type for relations between HP, FP and DP with RS, WP and WF. These relations indicate memory in the response and suggest that the rig control system uses present and past values of the master parameter (feed pressure) to adjust the others.

Different behavior is found in Jumbo 2 (wc 2b) working in manual drilling level, concerning the negative correlation between FP (input) and PR (output) parameters (B type) and the non-causal relationship of HP and RP, contrary to the rest of cases. The former may indicate a lower efficiency in the performance of the drill, which may be caused by the effect of the operator on the drilling. This was explained by Schunnesson (1998): penetration rate tends to increase with an increase of the feed pressure until it reaches a maximum penetration rate; from this moment, further increases in the feed pressure will make the penetration rate decrease and sometimes stall the drilling. Despite these differences, the feed pressure still seems to be the most influential parameter during the drilling operation. In addition, there is no memory between parameters except of the FP – DP relation, which means that the adjustment of the parameters with manual operation only follows present values of the feed pressure and disturbances in the adjustment of the other parameters are controlled by the operator.

Table 4.3. Relationships between MWD parameters for the 11 working conditions in Table 3.2.

wc																
HP	1	A: $HP \rightarrow PR$; $k(1)$; $r(+)$														
	3															
	4a															
	4b															
	6a															
	2a	C: $HP \rightarrow PR$; $k(0)$; $r(+)$														
	5a															
	5b															
	5c															
6b																
2b	E															
FP	1	A: $FP \rightarrow PR$; $k(1)$; $r(+)$				A: $FP \rightarrow HP$; $k(1)$; $r(+)$										
	3															
	4a															
	4b															
	6a															
	2a	C: $FP \rightarrow PR$; $k(0)$; $r(+)$				C: $FP \rightarrow HP$; $k(0)$; $r(+)$										
	5a															
	5b															
	5c															
6b																
2b	B: $FP \rightarrow PR$; $k(1)$; $r(-)$		E													
DP	1	B: $DP \rightarrow PR$; $k(1)$; $r(+)$		G		C: $FP \rightarrow DP$; $k(0)$; $r(+)$										
	3															
	4a	A: $DP \rightarrow PR$; $k(1)$; $r(+)$		B: $HP \rightarrow DP$; $k(1)$; $r(+)$												
	4b															
	6a	B: $DP \rightarrow PR$; $k(1)$; $r(+)$		D												
	2a	D		G												
	5a	C: $DP \rightarrow PR$; $k(0)$; $r(+)$		C: $HP \rightarrow PR$; $k(0)$; $r(+)$												
	5b															
	5c															
6b																
2b	B: $DP \rightarrow PR$; $k(1)$; $r(-)$		E													
RS	1	F														
	3															
	4a															
	4b								E			F				
	6a											F				
	2a				F											
	5a	F		E												
	5b				F											
	5c				F											
6b	E		F													
2b	E		E						E							
RP	1	E		G			F									
	3	D														
	4a															
	4b															
	6a	E							C: $FP \rightarrow RP$; $k(0)$; $r(+)$			C: $DP \rightarrow RP$; $k(0)$; $r(+)$				
	2a	D														
	5a															
	5b								E							
	5c								G							
6b	E		D			E										
2b	E															
WP	1	F														
	3															
	4a															
	4b															
	6a															
	2a	F		F		E		E	E							
	5a	-		-		-		-	-							
	5b	-		-		-		-	-							
	5c	-		-		-		-	-							
6b	-		-		-		-	-								
2b	E		E		E		E	E								
WF	1	F				E			F	E						
	3															
	4a															
	4b															
	6a	E		E							E					
	2a	F		F					E	E						
	5a	-		-		-		-	-	-						
	5b	-		E		E		-	-	-						
	5c	F		F		F		F	E	E						
6b																
2b	E		E		E		E	E	D							
		PR	HP	FP	DP	RS	RP	WP								

According to the results, the feed pressure seems to be the parameter that leads the changes in the other parameters. Considering the three main operational modes: collaring, ramp-up and normal drilling, the feed pressure signal (see Figure 4.2a) shows in the beginning a sharp increase until it reaches a pre-set threshold at which it stabilizes. During this ramp-up operation, the feed pressure takes control of the drilling and adjusts the other parameters. After that, the normal drilling mode starts. In this mode, the feed pressure keeps its influence over the other parameters and adjusts them in order to keep an optimal drilling pressure.

When open fissures in the rock are found, the feed pressure shows a sharp drop to the level of drilling in the air (Finfinger et al., 2002; Peng et al., 2005; Kahraman et al., 2016). Figure 4.10 shows an example of this situation, where discontinuities at about 0.7 and 3.2 m are found. Similar graphs are shown e.g. in Peng et al. (2005) and Kahraman et al. (2016). Schunnesson (1996 and 1998) and Schunnesson et al. (2011) showed that when the drill bit goes through a fracture zone, the rotation pressure increases. This should be the response of a disturbance in the adjustment of the parameters. As can be seen in Figure 4.10, when the drilling reaches a discontinuity, the rotation pressure shows a peak followed by a sharp decrease and later by an increase, connected with the behavior of the feed pressure.

Figure 4.10. MWD signals in rock discontinuities drilled by Jumbo 1 for hole 52 of blast from chainage 747 of Bekkelaget's construction.

In the same way, Figures 4.2a and 4.10 show a peak in the damp pressure where feed pressure stabilizes after the ramp-up. The leadership behavior of the feed pressure is

limited by certain bounds of variation preset by the manufacturer, in order to optimize the performance of the jumbo. Moreover, some parameters led by the feed pressure, such as the rotation pressure and damp pressure, also limit the feed pressure adjustment by using specific bounds preset in the control system. They work as control parameters in order to minimize damages in the mechanism of the boom. In line with Schunnesson (1996, 1998) and Schunnesson et al. (2011), if the rotation pressure bounds are exceeded (as it will likely be upon a discontinuity), the feed pressure ramps down so that they return to their working range. For the damp pressure case, bounds preset in this parameter are mainly used to limit the increase of pressure during the ramp-up operation and minimize undesirable motions in the boom while drilling.

Other parameters are not significantly correlated with any other (relation types E or F) such as rotation speed (RS), water pressure (WP) and water flow (WF). This involves no (E type) or little (F type) effect of the control system on these variables. Some authors indicate that discontinuities can induce variations in water flow and water pressure (Schunnesson et al., 2011). Water flow shows negative peaks in Figure 4.10 where discontinuities appear, confirming its relation with rock variations. On the contrary, RS and WP signals have no significant variation.

4.1.4. Conclusions on the mutual relations of drill monitoring variables and the the influence of the drill control system in tunneling operations

The results obtained lead to the following conclusions:

- Among the eight MWD parameters, penetration rate (PR), hammer pressure (HP), feed pressure (FP), damp pressure (DP) and rotation pressure (RP) show significant mutual correlation.
- For all working conditions studied (involving changes in the rock, automatization level and sample interval), the feed pressure is the parameter that leads the adjustment of the other parameters.
- Feed pressure, damp pressure and rotation pressure show instantaneous response with memory between them for all working conditions when drilling in semi-automatic mode. This reflects the mechanical response of the control system to face disturbances during the drilling. The rotation pressure and the damp pressure limit the feed pressure adjustment by bounds preset in the control system. They work as control parameters in order to minimize damages in the mechanisms of the boom.

- Rotation speed (RS), water pressure (WP) and water flow (WF) are not significantly correlated with any other parameter. These parameters, not influenced by the control system, may be affected by variations in the rock mass.
- Contradictory results about the delay in the response of FP to PR and HP and of HP to PR, for semi-automatic mode, have been found. In five working conditions, the response is instantaneous whereas in the other five there is a delay. More data is required to explain this inconsistency; different RCS versions may be one of the reasons for it.
- Memory in the response between parameters with significant correlation for Jumbos working in semi-automatic drilling mode, suggests that the rig control system uses present and past values of the feed pressure to control the adjustment of the others. On the contrary, for manual drilling mode this adjustment only follows the present values of the feed pressure.
- The results for manual drilling point out a negative correlation between feed pressure and penetration rate, contrary to the semi-automatic drilling mode. According to the relation between feed pressure and penetration rate in section 2.3.2, in this case, the maximum penetration rate peak has been overcome and increases of feed pressure will make the penetration rate to decrease (see Figure 2.6). A relevant conclusion for manual drilling mode is that the influence of the operator hides any relationship between the MWD parameters and how the drill system controls them.
- In all cases, the performance of the control system is independent of the rock type.

Since changes in the rock mass primarily drive variations in the feed pressure when the thresholds of the control parameters are exceeded, and this influences the adjustment of the other parameters, the feed pressure may be considered a potential MWD parameter to be used for rock mass characterization. Unfortunately, information of the threshold values is not provided by the manufacturer. Other parameters not influenced by the control system, such as rotation speed, water pressure and water flow may also reflect rock variations and, being independent of the feed pressure-controlled parameters, could add significance for a MWD-based rock mass characterization.

4.2. Jumbo navigation and blasthole positioning

Navigation is necessary to locate the jumbo inside the tunnel before drilling a new round. For that, the jumbo rig uses three reference systems, sketched in Figure 4.11: (i) an

absolute coordinate system that references the position of the jumbo, in this case the EUREF 89 Norwegian Transverse Mercator (NTM) projection, (ii) a Tunnel Reference System (TRS) with one axis parallel to the tunnel axis and the other two in the plane of the tunnel face of the new round, and (iii) a Drilling Reference System (DRS) defined by two vertical planes X_dY_d , Y_dZ_d and a horizontal X_dZ_d plane. The angles of the TRS axes with the DRS ones are θ , ω and γ (see Figure 4.11).

Figure 4.11. Representation of the three reference systems involved in the jumbo navigation: i. NTM Coordinates System, ii. Tunnel Reference System, iii. Drilling Reference System.

4.2.1. NTM Coordinate System

The position of the jumbo inside the tunnel is first obtained, see Figure 4.11 (i. NTM coordinate System). The jumbo has a laser scanner installed. In addition, target plates, with known coordinates, are located along the tunnel wall at every 5 m distance. The absolute coordinates of the jumbo are calculated by trilateration (i.e. distance measurement from the laser scanner to the target points). In this case study, X_{NTM} , Z_{NTM} coordinates are given in the NTM projection and Y_{NTM} is the height above sea level.

4.2.2. Tunnel Reference System (TRS)

The drill rig is aligned with the tunnel line (perpendicular line to the real face of a new round that defines the orientation of the drilling) in order to follow the design of the construction. For that, two targets are mounted on one of the booms. The laser beam

points to the free face in the direction of the tunnel axis and the boom is rotated until the laser beam passes through both targets (Figure 4.11, ii. TRS). The boom is now aligned with the tunnel axis and the orientation and inclination of the boom are registered in three orthogonal vectors ($\vec{x}_t$, $\vec{y}_t$, $\vec{z}_t$) to create a coordinate system parallel to the tunnel axis and the free face.

The laser scanner also measures the distance from the jumbo to the face of the new round and records the chainage at which it is located inside the tunnel. This chainage is taken as reference plane of the collaring depth position of the blastholes. Negative depth values are assigned to measurements behind this plane, and positive values, to measurements ahead of this plane.

4.2.3. Drilling Reference System (DRS)

The blasthole position measured by each boom is calculated in the Drilling Reference System defined by means of three spherical coordinates, blasthole length (l_b), azimuth or lookout direction (L_D) and inclination or lookout angle (L_I) (see Figure 4.11):

$$X_F = l_b \cdot \sin(L_I) \cos(L_D) \quad (4-6)$$

$$Y_F = l_b \cdot \sin(L_I) \sin(L_D) \quad (4-7)$$

$$Z_F = l_b \cdot \cos(L_I) \quad (4-8)$$

The inclination angle varies between 0 and 90° both for holes drilled upwards or downwards so that the azimuth is between 0 and 180° for holes drilled upwards and between 0 and -180° for holes drilled downwards.

Blasthole positioning data logged by the ABC system uses sensors installed along the boom (outside the blasthole) to measure the azimuth and the inclination angles. The semi-automatic ABC total system installed in the drill rig authorizes the operator to move the boom and feeder manually to follow a predesigned drill plan. Once the boom is placed in the required position and before the drilling starts, the measurements of the azimuth and inclination angles are logged in the DRS. These are considered constant as the boom remains still while drilling the blasthole. The end coordinates of the blasthole are calculated by adding, to their collaring coordinates, the result from Eqs. (4-6), (4-7) and (4-8). Since deviations beyond the collaring point cannot be measured by the MWD technology there is a possible error between the actual end position of the blastholes and the end position given by the MWD system. This is further studied in section 4.3.

4.2.4. Transformation System TRS-DRS

The operations described in sections 4.2.1 and 4.2.2 make the jumbo to be oriented by three angles (γ, θ, ω) , according to the directional vectors of the TRS: horizontal $(\vec{x}_t, \vec{z}_t)$ and vertical $(\vec{z}_t, \vec{y}_t)$ directions of the tunnel axis and the $(\vec{x}_t, \vec{y}_t)$ rotation of the free face, respectively. To calculate the position of the blastholes, the drill rig rotates, according to these three angles (θ, ω, γ) , the planes formed in the tunnel reference system $(X_t Z_t, Y_t Z_t, X_t Y_t)$, to create the drilling reference system defined by the two vertical planes $Y_d Z_d$, $X_d Y_d$ and a horizontal $X_d Z_d$ plane.

For Atlas Copco jumbos, the directional coordinate vectors of the TRS $(\vec{x}_t, \vec{y}_t, \vec{z}_t)$ and the NTM coordinates of the jumbo $(X_{NTM}, Y_{NTM}, Z_{NTM})$ are presented at the end of each MWD file. The three rotation angles to transform the DRS coordinates of a point in a blasthole to the TRS can be seen in Figure 4.11 and are further explained in Figure 4.12.

Figure 4.12. Angles of the transformation system: a) rotation with respect axis $\vec{x}_d$; b) rotation with respect axis $\vec{y}_d$; c) rotation with respect axis $\vec{z}_d$.

The rotation of the blastholes coordinates from the DRS (X_d, Y_d, Z_d) to the TRS (X_t, Y_t, Z_t) is carried out by introducing these three angles in a 3D rotation matrix:

$$\begin{bmatrix} X_t \\ Y_t \\ Z_t \end{bmatrix} = \begin{bmatrix} 1 & 0 & 0 \\ 0 & \cos\theta & -\sin\theta \\ 0 & \sin\theta & \cos\theta \end{bmatrix} \cdot \begin{bmatrix} \cos\gamma & 0 & \sin\gamma \\ 0 & 1 & 0 \\ -\sin\gamma & 0 & \cos\gamma \end{bmatrix} \cdot \begin{bmatrix} \cos\omega & -\sin\omega & 0 \\ \sin\omega & \cos\omega & 0 \\ 0 & 0 & 1 \end{bmatrix} \cdot \begin{bmatrix} X_d \\ Y_d \\ Z_d \end{bmatrix} \quad (4-9)$$

The location of the oriented blastholes in absolute coordinates $(X_{t,NTM}, Y_{t,NTM}, Z_{t,NTM})$ is obtained by adding the NTM coordinates of the jumbo $(X_{NTM}, Y_{NTM}, Z_{NTM})$ to the oriented coordinates of the blastholes (X_t, Y_t, Z_t) in the TRS system.

4.3. Assessment of drilling deviations in tunneling

Nowdays, software packages developed by manufacturers Atlas Copco, Sandvik and Bever Control in cooperation with AMV show the position of the blastholes. However, this is only a theoretical representation since the actual deviation in the hole is not considered. Four variables mainly influence deviations during the drilling (Östberg, 2013): (i) setting out, (ii) collaring and alignment, (iii) drill rod deflection and (iv) rock structure.

Blasthole positioning is monitored from sensors (inclinometers-accelerometers) installed along the boom and thus, outside the blastholes. Since the boom remains still during the drill, it normally measures constant values of its direction. However, the actual path inside the rock is not measured, due to neither the drill rod nor the bit are equipped with any sensor. This suggests an unknown error at the bottom hole position in relation to the intended position (given by the MWD system). Olsson (2010) analyzed this error for the contour blastholes by measuring, with a total station, the end position of the half cast contour holes when they were visible. He determined a mean deviation value of 11.6 ± 6.8 cm (mean $\pm$ standard deviation). To the authors' knowledge, no additional data on deviations of production blastholes in tunneling has been published.

Drill deviations may generate a non-uniform explosive charge concentration: excessive proximity between two blastholes may increase the specific charge (or volume charge concentration) in this zone, whereas a higher distance between them may reduce it, resulting in problems with rock breakage, fragmentation and pull. They may also induce problems in the perimeter excavated. Outwards deviations create an excessive over-excavated zone generating short-term stability problems around the perimeter of the excavation. This also increases production costs due to the retrieval of the extra material blasted and the need of a sturdier primary supports. Inwards deviations cause under-excavation zones in the perimeter, requiring a more intensive scaling to fulfil the pay-line requirements.

4.3.1. Measurements overview

The assessment of the drilling quality has been done with a Pulsar Micro Probe Mk3, manufacturer by geo-koncept, of 0.037 m diameter and 0.30 m length. The probe works as an inertial measure system that logs data registered by an accelerometer, a compass and an inclinometer, each one oriented in one of the 3D axes. This allows to measure the

actual path of both vertical and sub-horizontal production blastholes. The probe is connected to an acquisition data Trimble system as shown in Figure 4.13.

Figure 4.13. Overview of blasthole deviation measurement system.

For the measurement of drill deviations, the probe is introduced in the blasthole and it is pushed gradually inside the hole with a steel bar, with a rubber on its end. The probe records at every 1 m sample interval (there are marks in the wire at every 1 m length), in exception of the first measurement, that corresponds to a length of 0.865 m, including the length of the probe. The azimuth (horizontal angle) and the inclination (vertical angle) are also logged and they are measured with respect to the magnetic North and the horizontal plane, respectively. The accuracy of the azimuth and inclination is $\pm 1^\circ$ and $\pm 0.25^\circ$, respectively. The magnetic declination of the day when the measures were taken (22-12-2015) at Oslo, Norway, is $2^\circ 56.76'$ E; this must be removed from the data gathered to pass from magnetic to geographical North.

Blastholes were measured from two blasts located on the top right section of the Biohall 5 at chainage (ch) 129.5 m and 133 m of the Caverns Biohall 5 of the underground extension work of Bekkelaget's water reclamation facility (see Figure 3.1, S3). They were drilled with the Atlas Copco jumbo described in Table 3.2 as working condition 4. The collaring position of these blastholes are shown in Figure 4.14; all blastholes were drilled by boom 1 except blasthole 32 (chainage 129.5), that was drilled by boom 3. Table 4.4 shows the nominal values of lookout angle and lookout direction from the MWD records.

Figure 4.14. Measured blastholes (highlighted in red) in blast of the upper right side of Biohall 5: a) chainage 129.5; b) chainage 133.

Table 4.4. Nominal lookout angle and lookout direction from the MWD records chainage (ch) 129.5 and 133.

Blasthole	ch 129.5			ch 133	
	75	76	32	76	77
Lookout angle (°)	7.4	5.6	4.8	1.2	3.2
Lookout direction (°)	-157.9	166.6	-30.1	-159.4	-173.2

4.3.2. Data analysis and Results

Drilling deviations

The trajectory of the blasthole is calculated for both MWD data and the probe by using Eqs. 4-6, 4-7 and 4-8. For the probe, L_I and L_D correspond to the inclination and azimuth angles, respectively, and l_b is the length of wire introduced inside the blasthole at each measure.

To transform data obtained with the probe into the drilling reference system, the complete trajectory logged by the probe is rotated to match the first measurement, 0.865 m, with the trajectory direction of the MWD data in the drilling reference system. This presupposes a correct collaring and alignment of the bit and the drill rod for the first measurement of the probe in the blasthole, which may likely occur due to the information provided by the MWD to the jumbo operator. The analysis has been carried out in Matlab (Matlab, 2017).

Figures 4.15 and 4.16 show the comparison of the trajectory followed by the five blastholes as from the MWD data (i.e., theoretical) and the probe records for rounds located in the ch 129.5 (blastholes 75-graph a, 76-graph b and 32-graph c) and ch 133 (blastholes 76-graph a and 77-graph b). For each hole, three graphs are included,

corresponding to the projection of the trajectories in the X_dY_d plane (upper-left side), X_dZ_d plane (right side) and Y_dZ_d plane (bottom-left side). The actual trajectory measured by the probe is coloured different for each hole (dots are the position of the measures taken); the black line represents the trajectory given by the MWD system and the black dot the collaring of the blasthole.

Figure 4.15. Blastholes trajectory for round in chainage 129.5 m, Biohall 5. Projection in the X_dY_d plane (plots at the top-left side), X_dZ_d plane (plots at the right side) and Y_dZ_d plane (plots at the bottom-left side).

Figure 4.16. Blastholes trajectory for round in chainage 133 m, Biohall 5. Projection in the X_dY_d plane (plots on the top-left side), X_dZ_d plane (plots on the right side) and Y_dZ_d plane (plots on the bottom-left side).

As can be seen, the blasthole length in the probe measurements (Figures 4.15 and 4.16) is always shorter than the maximum length given by the MWD (these differences in some cases, Figure 4.15a and 4.16b are nearly 1.5 m). This variation may be caused by the

difficulty to introduce the probe inside the blasthole, especially in the final meters, probably due to sharp changes in the actual hole path and the stiffness of the probe or to the blockage of the blasthole after drilling. It is also interesting that the chainage difference between rounds 129.5 and 133 is 3.5 m while the length of the intended drilled holes is about 5.5 m. Figure 4.17 shows a big overlap between the drilling plan of these two rounds that indicates a wrong performance of the blasting at the chainage 129.5.

Figure 4.17. Overlapping between the drilling plan of rounds 129.5 and 133

Table 4.5 shows the absolute distance between the end position given by the probe and the respective position obtained from the MWD data at the same hole depth, for the horizontal (X_dZ_d plane) and vertical (X_dY_d and Y_dZ_d planes) projections, and for the spatial distance (distance in the 3D). These values are expressed as the distance with respect to the maximum probe length measurement in m/m. The mean value and the standard deviation are also given. In the vertical plane (Y_dZ_d , Figures 4.15 and 4.16), trajectories for blastholes 75 and 76 (ch 129.5) and for blasthole 76 (ch 133) show upwards deviation, and trajectories for blasthole 32 (ch 129.5) and blasthole 77 (ch 133) indicate downwards deviation, with a mean absolute value of 0.07 ± 0.03 m/m (mean and standard deviation, Table 4.5). An interesting result occurs for blasthole 75 (Figure 4.15), where, the blasthole results in a final upwards lookout instead of the designed downwards. Considering the horizontal plane (X_dZ_d , Figures 4.15 and 4.16), there is always a deviation in the direction of the theoretical MWD orientation with a mean absolute deviation value of 0.05 ± 0.02 m/m (Table 4.5). The X_dY_d plane shows for all blastholes with exception of hole 32 (ch 129.5), differences in the actual blasthole trajectory

direction in relation to the one given by the MWD system, with a mean absolute value of 0.08 ± 0.03 m/m. The mean absolute error in the 3-D is 0.11 ± 0.03 m/m.

Table 4.5. Deviation measures at the depth of the probe data.

Chainage	Blasthole	MWD	Max. probe	Absolute deviation (m/m)			
		length (m)	measured length (m)	$X_d Z_d$	$Y_d Z_d$	$X_d Y_d$	3D
129.5	32	5.50	4.00	0.04	0.05	0.06	0.09
	75	5.50	5.02	0.05	0.11	0.12	0.17
	76	5.50	4.40	0.04	0.09	0.10	0.14
133	76	5.50	4.87	0.07	0.06	0.09	0.13
	77	5.50	4.12	0.03	0.03	0.04	0.06
Mean $\pm$ std.				0.05 $\pm$ 0.02	0.07 $\pm$ 0.03	0.08 $\pm$ 0.03	0.11 $\pm$ 0.03

Comparison of drilling deviations with MWD data

The comparison between MWD parameters and deviations measurements will improve the understanding of the drill/rock interaction, showing the impact of the rock structure in the blasthole path generated while drilling. For that, the magnitude and the fluctuation of the MWD signals are considered. Figure 4.18 shows the eight MWD parameters for the five blastholes analysed; signals for each blasthole are coloured in line with their respective measured trajectory in Figures 4.15 and 4.16. According to the MWD parameters description (Peng et al., 2005; Beattie, 2009; Hjelme, 2010; Schunnesson et al., 2011 and Schunnesson and Kristoffersson, 2011), parameters mainly controlled by the drilling control system are (i) the feed pressure, defined as the thrust to keep the bit in contact with the bottom of the hole, (ii) the hammer pressure or impact pressure of the bit against the rock and (iii) the rotation speed that quantifies the number of revolutions of the bit per minute. The rotation pressure or torque and penetration rate, on the other hand, are considered a drilling response to rock conditions. As can be seen, the hammer pressure, rotation speed and rotation pressure signals of blasthole 32, ch 129.5 have a different magnitude range than the signals for the other holes. These values might be related to a different calibration of sensors of the boom 3.

Figure 4.18. Representation of the MWD parameters for the 5 blastholes analysed; where, BH is blasthole and ch is chainage, HD is hole depth, PR is penetration rate, HP is hammer pressure, FP is feed pressure, DP is damp pressure, RS is rotation speed, RP is rotation pressure, WF is water flow and WP is water pressure.

The behaviour of the MWD parameters while drilling through possible disturbance zones has been investigated: Peng et al. (2005) determined the feed pressure and thus hammer pressure as good indicators for void detection and in section 4.1 it has been identified the feed pressure as the parameter that controls the adjustment of the other parameters when boundary values while drilling are exceeded. In addition, Schunnesson (1996, 1997) and Ghosh et al. (2018) claimed that when discontinuities are encountered during the drilling, the rotation pressure shows significant fluctuation, resulting in a noisy signal. These variations are highlighted and calculated as the sum of the residuals over a defined

interval along the blasthole, according to Eq. 2-7. Figure 4.19 represents the variations of the rotation pressure for each one of the five blastholes analyzed, considering only values of the normal drilling mode; the missing data directly after HD=0 represent the ramp-up mode (see section 4.1.3, pp. 48).

Figure 4.19. Representation of the variability of the rotation pressure for the 5 blastholes.

The analysis of both Figures 4.18 and 4.19 give relations between the MWD parameters and the actual blastholes path. Blasthole 32, ch 129.5 (Figure 4.15.a) shows the second smallest deviation values (see Table 4.5). MWD parameters for this hole indicate lower hammer pressure and rotation pressure and higher rotation speed magnitudes. These values might be related to low deviations at the beginning of the drilling due to a lower pressure over the drill rod during the operation. Variabilities in the rotation pressure signal at 3.5 - 4 m and 4 - 4.5 m are correlated with a change in the actual trajectory from a depth of 3 m.

Blasthole 75, ch 129.5 (Figure 4.15.b) shows the highest deviations values (see Table 4.5). In this case, the MWD parameter magnitudes are similar to those of the other blastholes (Figure 4.18). The high deviation can be explained by fluctuations in its signal. In Figure 4.18 (yellow signal), the feed and hammer pressure parameters show a drop between 0.5 and 1 m depth; this coincides with a peak in the RP variability (Figure 4.19). This indicates a possible discontinuity that may have changed the path of the blasthole at the beginning of the drilling. Another possible disturbance zone between 2.5 - 3.5 m depth may be the reason of a new sharp change in the blasthole path deviation at 3 m depth.

Blasthole 76, ch 129.5 (Figure 4.15.c) also shows high deviations (see Table 4.5). Variabilities in the rotation pressure signal at 3 - 3.75 m (Figure 4.19) are correlated with

a change in the actual trajectory of the blasthole between 3 – 4 m depth, similarly as in blasthole 32 ch 129.5.

Blasthole 76, ch 133 trajectory represents a good alignment with the MWD path until 2 m depth (Figure 4.16.a). This is related with the ramp-up drilling mode (low feed and hammer pressure values) until 1.2 m depth. The high rotation pressure magnitude during the normal drilling (purple line, second lower graph, Figure 4.18) may suggest an excessive torque force over the drill rod. It is interesting that, in this case, there is a higher deviation in the X_dZ_d than in the Y_dZ_d (see Table 4.5).

Blasthole 77, ch 133 presents the lowest deviation values (Table 4.5). Changes in the path between 1 - 2 m and 3 - 4 m depth, are associated with peaks in the rotation pressure variability signal between 1.5 - 2 m and 3 - 3.5 m.

The results point out that drilling deviations are highly influenced by the rock structure. Peaks and drops detected in the variability of the rotation pressure and in the signals of feed and hammer pressure, respectively, are highly correlated with changes in the blasthole trajectory; these anomalies in the signal are normally associated with disturbances zones in the rock. In addition, changes in the magnitude of the hammer, feed and rotation pressure parameters variate the pressure of the drill rod against the rock.

4.3.3. Conclusions on drilling deviations in tunneling

The quality of the drilling in underground blasting operations has been investigated with a view to quantify the error of the intended trajectory given by the monitoring system data with respect to the actual end position of the blasthole. For that, a Pulsar Micro Probe Mk3 has been used to measure the actual trajectory of five production blastholes, by measuring inclination and azimuth values along its length.

Measurements analyzed have been carried out in the underground extension work of the municipal wastewater treatment plant in Oslo, Norway. The results indicate deviations, in some cases significant, between the actual trajectory of the blastholes and the position estimated by the MWD system. The mean absolute deviation errors in the vertical and horizontal planes with respect the deepest location measured by the probe are estimated to be 0.07 ± 0.03 m/m and 0.05 ± 0.02 m/m (mean and standard deviation), respectively. Deviations in the X_dY_d plane also show differences in the length and orientation of lookout distance with a mean absolute value of 0.08 ± 0.03 m/m. These values correspond

to the minimum expected errors since it is assumed that a correct collaring and alignment of the of the bit and drill rod.

Drilling deviations may result in undesired variations of charge concentration at depth resulting in poor or excessive rock breakage and under- or over-excavation. The shorter final length of the measurements done with the probe may also indicate a shorter effective drill length and thus, shorter rounds than expected.

The comparison between MWD parameters and deviation measurements point out that drilling deviations are highly influenced by the rock structure. Disturbance zones in the rock detected by peaks and drops in some parameters, such as the variation of the rotation pressure and the feed and hammer pressure, are correlated with changes in the blasthole trajectory. In addition, changes in the magnitude of these parameters may also have influence on the phenomena, since rises in the pressure of the drill rod against the rock may increase the probability of deviations during the drilling.

Although this study sheds some light on the drill deviation issue in tunneling, a higher number of measurements is necessary to draw a general conclusion in relation to the quality of the drilling and to improve the assessment of the MWD to detect cases of deviations in the blastholes due to the rock structure. This would allow the building of e.g. a probability distribution of drilling error, to be applied to the blasthole trajectory given by the MWD system.

4.4. Detection of potential overbreak zones in tunnel blasting from MWD data

Rock excavation in mining and tunneling frequently use cautious blasting techniques. The primary objective of blasting is to fragment rock to allow loading and haulage, without creating extensive damage to the remaining rock mass. As Anderson (1994) defined: *“Cautious blasting is a blasting that does not cause damage to the rock outside of the intended damage distance”*.

Data monitored in the excavation of the gallery S1 in Bekkelaget’s water reclamation facility has been considered (see Figure 3.1 and Tables 3.1 and 3.2). The overbreak from blasting to the remaining rock mass is analyzed with the purpose of developing a drilling index from the MWD parameters, able to predict the area over-excavated in the contour of a tunnel by blasting effect. A new methodology based on the comparison of scanner

profiles of the excavated sections with the position of the contour blastholes, has been developed in this chapter to obtain the excavated mean distance (EMD) between the blasthole and the excavated profiles at each MWD record position, which may be considered as a damage measure.

The charging of the rounds was carried out with emulsion of different linear charge for different types of blasthole: cut, lifter, easer, buffer, contour. String loading method was used for the charging of the contour blastholes, with a design linear charge of 0.5 kg/m. This is assumed to be constant in the analysis. The actual charge of the holes may vary around the design value. However, such variations are assumed to be random and very tight so that, although they are a source of indetermination in the analysis, they will not bias the influence of the other parameters in the overbreak from blasting.

For the calculation of this index, the MWD parameters play an important role as a mean to describe the in-situ rock mass properties before the blast. A thorough normalization of the MWD parameters has been carried out to remove external influences in the data that may lead to a wrong interpretation. 54 blasts, which comprise around 1700 contour blastholes, have been compared with more than 4000 excavated sections to carry out the analysis.

Sources of uncertainty such as drilling deviations, the scaling and primary support done before the excavated section scanning, possible variations (unrecorded) in the explosive linear density, etc., have been assumed to be of random nature, unavoidable in the condition in which such data were measured. A non-linear regression model of the over-excavated mean distance has been developed by combining the rotational, hydraulic flushing and the rate of advance of the drilling, and the confinement of the rock mass with depth. This is cast with the MWD parameters penetration rate, hammer pressure, rotation speed, rotation pressure and water flow, plus the drill lookout distance.

4.4.1. Analysis of the excavated area

4.4.1.1. Superposition of excavated profiles and contour blastholes

As described in section 3.1.3 (pp. 32), a laser scanner system has been used to monitor the final profiles of the excavated void from each blast. An interface AutoCAD-Matlab (AutoCad, 2017; Matlab, 2017) has been created to automatically compare the excavated profiles with the contour blastholes for each round. The profile formed by the contour holes (hereinafter named contour profile) is compared with the scanner profiles of the

excavated sections in order to obtain the excavated mean distance between the blasthole and the scanner section at each depth for which MWD data are logged. This distance is considered as an indicator of the resulting damage (i.e. over-excavation).

For safety reasons, scanning of the excavated section is done after scaling and installation of primary supports. During scaling, non-stable pieces of rock are removed from wall and roof to avoid rock falls and to ensure safe work conditions. Next, rock bolts and shotcrete are applied to reinforce the tunnel walls and to prevent stability problems. These operations obviously modify somewhat the perimeter excavated. Shotcrete was automatically applied to the tunnel surface and the thickness of the shotcrete layer was modified per round according to the geotechnical recognition of the tunnel wall and crown. The thickness of the shotcrete layer was known for every round and it was homogeneous over walls and crown, according to the operation reports. Appendix 2 shows an example of the geotechnical report for one round of the main gallery (chainage 383), where it can be seen the shotcrete thickness established according to the geotechnical interpretation (Q-Barton) and additional supports installed. This thickness has been added to the scanner profiles in the AutoCad files to obtain the actual excavated contour from the blast. However, an uncertainty remains.

For the comparison between excavated and contour profiles we consider the nominal lookout angle and lookout direction values given by the MWD for each blasthole, which make them to be outward oriented (this does not mean that every hole has the same orientation). Since no deviation measures of the blasthole trajectory are available, the intended trajectory of the blastholes given by the MWD system has been used for the analysis. Each blasthole is represented as a straight line according to its nominal values of azimuth or lookout direction and inclination or lookout angle (see section 4.2.3); the intended lookout distance of each blasthole increases linearly with depth. Eqs. 4-6 and 4-7 are used to calculate the theoretical position of the blasthole at each excavated profile depth in the Drilling Reference System (DRS). Variables L_I (lookout angle) and L_F (lookout direction) are obtained from the MWD files and l_b is the length of the blasthole from the collaring position to the respective excavated profile depth (Z_F , see Figure 4.11), obtained with Eq. 4-8. This working procedure will add uncertainty to the results as the nominal trajectory of the holes has been considered.

Irregularities on the free face of a new round cause the collars of the blastholes not to be in the same plane, so they have different collar depths. In addition, some excavated

profiles (mainly profiles at the beginning of the round) are not influenced by all contour holes of the current blast but by blastholes from the previous one. Each blasthole is extended from the foremost collaring hole to the depth of the deepest blasthole of each round to calculate the excavated area for all profiles included in a round. Figure 4.20a represents the contour profiles (cyan lines), the position of the blastholes (black lines) and their extensions (tiny blue dots).

Figure 4.20. Chainage 399: a) Contour holes profiles, position of the blastholes and their extensions; b) Overlapping between the contour profiles and the scanner profiles.

To carry out the analysis, both contour and excavated profiles must be overlaid. Excavated profiles from AutoCAD files are drawn in a vertical xy plane, where the Y coordinate is referred to the Y_{NTM} absolute coordinate and the Z coordinate, i.e. depth of the xy plane, is indicated by the chainage at which it is located. The contour blastholes coordinates in the DRS must be rotated to the TRS. Since the rounds studied belong to the main gallery and they are excavated in a straight line, the xy and xz planes for both TRS and DRS coincide (angles γ and ω are 0); only plane y_tz_t is rotated in case the tunnel axis is uphill (positive θ angle) or downhill (negative θ angle), see Figure 4.12. The rotation of the DRS contour blastholes coordinates (X_d, Y_d, Z_d) to the TRS (X_t, Y_t, Z_t) is obtained by introducing the three angles ($\theta, \gamma=0, \omega=0$) in Eq. 4-9. The translation of the Y_t and Z_t coordinates is carried out by adding the Y_{NTM} and the chainage values of the round studied, respectively. Figure 4.20b sketches the overlapping of both excavated (red lines) and contour (blue lines) profiles for a round.

Excavated Mean Distance from the blasthole

The overbreak created around the blasthole by blasting is caused, among other factors, by the combination of the explosive and the rock mass condition around the blasthole (Hustrulid, 2010; Johnson, 2010). Considering the contour blastholes position per round, an Excavated Mean Distance (EMD) has been defined (Figure 4.21). It corresponds to the area between the midpoints of the spacings on both sides of the hole and the excavated profile, normalized by the sum of the mid-spacings on both sides of the blasthole. When two adjacent holes are on the same side of the excavated profile, the EMD is calculated as EMD 1 by (Figure 4.21, EMD 1).

$$EMD\ 1 = \frac{A_{T1}}{\frac{S_1}{2} + \frac{S_2}{2}} \quad (4-10)$$

where S_1 and S_2 are the spacing between the current blasthole and the adjacent ones (they are generally around 0.7 m); A_{T1} is the area excavated by each blasthole defined by points 1, 2, 3 and 4 in Figure 4.21, EMD 1; it is positive when there is over-excavation (blasthole outside the rock) and negative for under-excavation (blasthole inside the rock).

In case two consecutive blastholes are located one inside and the other outside of the excavated profile, corresponding to over (positive EDM) and under (negative EMD) excavation (Figure 4.21, EMD 2), the area influenced by the blasthole to the excavated contour is obtained as EMD 2 by adding both areas with their respective sign:

$$EMD\ 2 = \frac{A_{T2}}{L_a} + \frac{A_{T3}}{L_b} \quad (4-11)$$

where A_{T2} is the area defined by 5, 6 and p_{int} (Figure 4.21, EMD 2); A_{T3} is the area defined by 7, 8 and p_{int} (Figure 4.21, EMD 2); p_{int} is the intersection point between the excavated profile and the line joining two adjacent blastholes; L_a and L_b are the distances between p_{int} and the mid-spacing between the current blasthole and the adjacent one.

Figure 4.21. Calculation of the Excavated Mean Distance.

The scanner profiles and the EMD values per blasthole are evaluated at 0.2 m intervals. The MWD sample interval is 0.1 m and the collaring chainage of each contour blasthole differs due to irregularities of the free face. Thus, the actual chainage of the MWD logs of each hole varies so that the position of the measurements recorded by the MWD system does not coincide with the depth of the calculated EMD values. A piecewise cubic Hermite interpolating polynomial (Fritsch and Carlson, 1980) is used to interpolate the EMD values at the specific depths of the MWD logs.

As an example, Figure 4.22a shows the area of influence of each blasthole (AT_1 or AT_2+AT_3 in Figure 4.21), calculated every 0.2 m for a round between the chainages 398 and 406 m considering the extension of the blastholes; over- and under- excavated areas are given in different colors and the contour blastholes are marked in black. Figure 4.22b represents the EMD values for one of the blastholes in graph a (that includes the backwards and forwards extensions, as explained above, versus chainage. It shows the EMD calculated (black dots), the cubic interpolation of the EMD calculated (red line), and the estimated EMD values corresponding to the MWD depths logged (blue dots).

Figure 4.22. a) Area excavated along each blasthole and its extension; b) EMD cubic interpolation and EMD value according to the MWD depth.

4.4.2. MWD Data processing

The response of MWD parameters is often affected by external influences different than the rock mass, such as the calibration of the monitoring sensors, the hole length and/or the drill rig performance (Schunnesson, 1998). The control system of the jumbo, during the adjustment of the parameters while drilling, induces systematic variations in the MWD data. All together, they add uncertainty to the data. A multi-step transformation has been developed to highlight changes in the parameters by the influence of rock properties. This process comprises:

- (i) Filtering out of outliers: production data often includes unrealistic high and low performance values of the jumbo, which may lead to a wrong interpretation of the MWD data (Ghosh et al., 2015).
- (ii) Removing of the ramp-up section of the log
- (iii) Correction of systematic variations in the MWD parameters such as the effect of hole length and feed pressure influence
- (iv) Normalization with the standard deviation to account for fluctuations in the signals.

Figure 4.23 shows a flow chart with the methodology developed to filter and correct the raw MWD data, with the acronym used after each step.

Figure 4.23. MWD data processing flow chart.

Filtering of outliers

For the filtering, the empirical probability distribution function of each MWD parameter is built from the complete data set values (54 blasts, comprising more than 6500 blastholes). Figure 4.24 shows the cumulative distribution functions (CDF) of the eight parameters. Parameters below and above the percentiles 2.5 and 97.5 (lower and upper tails of the 95% coverage), respectively, are discarded from the analysis. The black dashed horizontal lines mark the percentiles 2.5 and 97.5 and the vertical lines mark the corresponding limits of the parameters, given in Table 4.6.

Figure 4.24. Probability distribution function.

Table 4.6. Range of reasonable values of the MWD parameters as from their 95 % coverage.

Parameter	Units	Ranges
Penetration Rate	m/min	1 - 3.8
Hammer Pressure	bar	130 - 235
Feed Pressure	bar	20 - 80
Damp Pressure	bar	45 - 100
Rotation Speed	rpm	170 - 310
Rotation Pressure	bar	35 - 80
Water Pressure	bar	12 - 22
Water Flow	l/min	54 - 175

Removing of ramp-up operation mode

As can be seen in Figure 4.25, the feed pressure (FP) shows, initially, a sharp rise (ramp-up mode) until it reaches a pre-set threshold at which it stabilizes (normal drilling mode), see section 4.1.3 (pp. 48). Signals of the 8 parameters have been divided considering the point at which the feed pressure stabilizes. Only values included in the normal drilling mode are considered for the analysis, as data in the ramp-up mode are not representative of changes in the rock mass conditions. The former will be referred as MWD_{c1}.

Figure 4.25. Drilling operation modes in MWD signal from blasthole 71 at chainage 349. Units are given in Table 4.6.

Hole length and feed pressure corrections

Systematic variations generated by the drilling system and other parameters can be estimated and removed by averaging, for a large amount of data, the response of the parameters (Schunnesson, 1998; Hjelme, 2010). The average value of the eight MWD_{c1} parameters at every 0.1 m hole length has been calculated for the entire data for the 54 blasts analyzed. This result in an average signal for each parameter.

For the rounds analyzed, booms 1 and 3 always drilled the contour and buffer holes. According to incidence reports, the sensors that monitor the position of boom 3 were out of calibration in most of the rounds, thus the blastholes drilled by this boom had a lower lookout and lookout direction angles than indicated in the MWD data files. A more intensive scaling, resulting in a significant distortion of the excavated profile, was then necessary to meet the contour requirement. This would hide the influence of rock mass characteristics (described by MWD) on the resulting over break. Since the precise blasts for which boom 3 was out of calibration are not available, all MWD data for boom 3 are discarded in the analysis.

The correction of the hole length influence (to obtain a signal MWD_{c2} , see Figure 4.23) is done by:

$$MWD_{c2}^i = [MWD_{c1}^i - MWD_{fit,HL}^i] + MWD_{fit,HL}^1; \text{ with } i = 1, 2, \dots, N \quad (4-12)$$

where i indicates the logs of each blasthole, N being the number of these. $MWD_{fit,HL}$ is a polynomial regression (grade 3 to 5) with hole length, of the average signal of each MWD_{c1} parameter at every 0.1 m hole length for the entire data. The determination coefficient of the fit R^2 is always higher than 0.95. $MWD_{fit,HL}^1$ is the intercept of the fit, i.e. the value where ramp-up ends and normal drilling starts.

Figure 4.26 shows the average MWD_{c1} signal (blue lines), the polynomial regression ($MWD_{fit,HL}$, green lines) and the hole length transformed average signal (MWD_{c2} , red lines) at every 0.1 m hole length for boom 1. As can be seen in Figure 4.26, there is no noticeable effect of hole length in the average signal of hammer pressure ($Av.HP_{c1/c2}$) and water flow ($Av.WF_{c1/c2}$), thus the hole length transformation is not applied for these two parameters.

Figure 4.26. Correction of hole length influence in MWD parameters (pass from MWD_{c1} to MWD_{c2} , see Figure 4.23); blue lines: average signals, MWD_{c1} (obtained after filtering outliers and removing ramp-up section, see Figure 4.23); green lines: polynomial regression; red lines: average transformed signals, MWD_{c2} . Units and acronyms of the parameters in the ordinates are given in Table 4.6.

The analysis in section 4.1 shows that penetration rate, hammer pressure, damp pressure and rotation pressure are influenced by the feed pressure. According to this, the feed pressure generates systematic variations in these parameters that may hide the rock dependence on them. On the contrary, rotation speed, water flow and water pressure are little influenced by the feed pressure, thus being considered independent.

The same methodology followed for the hole length influence is now used to correct the feed pressure influence. The average value of the seven MWD_{c2} parameters (feed pressure is not included) is calculated for steps of 1 bar feed pressure value for the 54 blasts for boom 1. Similar to Eq. 4.13, the correction of the feed pressure influence (to obtain a signal MWD_{c3} , see Figure 4.23) is done by:

$$MWD_{c3}^i = [MWD_{c2}^i - MWD_{fit,FP}^i] + MWD_{fit,FP}^1; \text{ with } i = 1, 2, \dots, N \quad (4-13)$$

where i indicates the logs of each blasthole, N being the number of these. $MWD_{fit,FP}$ is a polynomial regression (grade 3 to 5) with the feed pressure, of the average signal of each MWD_{c2} parameter at every 1 bar feed pressure for the entire data. The determination coefficient of the fit R^2 is always higher than 0.95. $MWD_{fit,FP}^1$ is the intercept of the fit, i.e. the value at the minimum feed pressure.

Figure 4.27 shows the average MWD_{c2} signal (blue lines), the polynomial regression ($MWD_{fit,FP}$, green lines) and the feed pressure corrected average signal (MWD_{c3} , red lines) at every 1 bar for boom 1. In line with the results from section 4.1, penetration rate (PR), hammer pressure (HP), damp pressure (DP) and rotation pressure (RP) parameters have a strong influence on the feed pressure (FP). For the case of rotation speed (RS), water pressure (WP) and water flow (WF), the influence of feed pressure is considerably less, and these data are not transformed for the subsequent analysis.

Figure 4. 27. Correction of the feed pressure influence in MWD parameters (MWD_{c2} conversion to MWD_{c3} , see Figure 4.23); blue lines: average signals, MWD_{c2} (corrected for hole length influence, see Figure 4.23); green lines: polynomial regression; red lines: average transformed signals, MWD_{c3} . Units and acronyms of the parameters are given in Table 4.6.

The MWD_{c2} data ranges for the feed pressure (red curve) in Figure 4.26 are not the same that the ones represented in the blue curves of Figure 4.27. In relation with the feed pressure, in Figure 4.26 the average signal is calculated with respect the hole length and thus, drops found during the drilling due to local discontinuities are softened. The average signals calculated with respect the feed pressure will include measures at these lower feed pressure values (Figure 4.27).

Analysis of fluctuations in the MWD signals

Schunnesson (1996, 1997) claimed that when discontinuities in rock are drilled, penetration rate and rotation pressure parameters show significant variation, resulting in a noisy signal. Following this reasoning, parameters involved in the rotational mechanism of the jumbo (rotation pressure and rotation speed) and the penetration rate have been

normalized by the standard deviation. The purpose is to add to the magnitude of these parameters the effect of the variation of their signals related to discontinuities, fracture or softer rock zones. The procedure is carried out for the signal of each hole individually, where the MWD_{c3} values of penetration rate (PR_{c3}), rotation speed (RS_{c3}) and rotation pressure (RP_{c3}) are divided by the respective standard deviation of the entire signal of that hole. The resulting signals for these three parameters are MWD_{c4} .

$$MWD_{c4}^i = \frac{MWD_{c3}^i}{std(MWD_{c3})}; \text{ with } i = 1, 2, \dots, N \quad (4-14)$$

where, i indicates the logs of the hole, being N the number of these.

Since all parameters are not subjected to the same number of processing steps, a summary of the corrections made for each parameter is shown in Table 4.7; the last transformation for each parameter is in bold type.

Table 4.7 Summary of the corrections made for each parameter

	i.Filtering of outliers ii.Removing of ramp-up	iiia. HL correction	iiib. FP correction	iv.Signal variaton
	MWD_{c1}	MWD_{c2}	MWD_{c3}	MWD_{c4}
Penetration Rate	PR_{c1}	PR_{c2}	PR_{c3}	PR_{c4}
Hammer Pressure	HP_{c1}	-	HP_{c3}	-
Feed Pressure	FP_{c1}	FP_{c2}	-	-
Damp Pressure	DP_{c1}	DP_{c2}	DP_{c3}	-
Rotation Speed	RS_{c1}	RS_{c2}	-	RS_{c4}
Rotation Pressure	RP_{c1}	RP_{c2}	RP_{c3}	RP_{c4}
Water Pressure	WP_{c1}	WP_{c2}	-	-
Water Flow	WF_{c1}	-	-	-

4.4.3. Detection of potential overbreak zones model

The response of the processed penetration rate (PR_{c4}), hammer pressure (HP_{c3}), rotation speed (RS_{c4}), rotation pressure (RP_{c4}), water pressure (WP_{c2}) and water flow (WF_{c1}) parameters can detect variations in the rock and for equal blasting conditions they may explain variations in EMD data. The feed pressure is not considered as it has been used during the processing steps.

The excavation quality is mainly influenced by the geo-mechanical condition of the rock mass and blast characteristics such as the explosive type, the charge concentration, the blast timing, the drill pattern and the drilling deviations. Another variable to be considered

is the confinement of the explosive and the rock with depth that will influence the capacity of the explosive to break the rock (the deeper in the rock the more difficult to break it). This is normally solved by changing the explosive charge concentration with depth, being more powerful at the bottom of the blasthole and letting some part uncharged at the beginning (stemming).

The lookout distance (i.e. distance from the collaring to the position of the hole in the XY plane at each depth, see Figure 4.11) gives a measurement of the position and inclination of the hole in the rock at each MWD log and it is unsystematic as it changes in each blasthole and thus, it could be considered as a different parameter. In addition, it reflects the confinement of the explosive charge vs depth. If we consider the scale distance of the explosive as $\frac{Q}{L_{dist}}$, it will vary with the L_{dist} as we consider the explosive mass roughly constant from blast to blast according to the nominal blast reports, and thus, it is necessary to include the L_{dist} in the model.

Figure 4.28a shows, as an example, variations of EMD with the hole length and lookout distance for the holes of the blast located at the chainage 500. As can be seen, there is a negative influence of the lookout in the EMD which means that the excavated area in relation with the blasthole position decreases with an increasing of the lookout distance; this EMD relation with lookout distance is related to the hole length since it is assumed that the lookout distance increases linearly with depth. An increase of confinement with lookout distance is associated with an increase of the difficulty in breaking the rock, resulting then in a decrease of the over-excavated area, hence the EMD, which may be even negative (under-excavation). Therefore, the larger the lookout distance and the deeper the drilling, the more under-excavation is created in relation with the position of the contour blastholes.

Figure 4.28. Preselection of the data for blastholes of boom 1 at the chainage 591; a) EMD vs. hole length and lookout distance; b) MWD signals for blastholes 6, 7 and 11. See Table 4.6 for the acronyms of MWD parameters; subscripts c_3 and c_4 refer to the corrections shown in Figure 4.23.

In some cases (data for holes 6, 8 and 11, Figure 4.28a), the general trend is not followed. Figure 4.28b shows the respective MWD_{c1-c4} logging for holes 6, 8 and 11 (see Table 4.7). One may think that local rock conditions may cause such results and that MWD parameters should show a peak or some kind of variation in the signal. This is effectively observed for hole 6, where MWD_{c1-c4} parameters show a significant peak or signal fluctuation at depth 3 m to 4 m, in line with the over-excavation peak represented in Figure 4.28a for this hole. However, no distinctive variation appears in the MWD_{c1-c4} records of holes 8 and 11. Hole 8 shows an over-excavated section at 2.5 m to 3.5 m, whereas hole 11 indicates under-excavation at depths from 1 m to 2 m, though no significant changes can be observed in the MWD signal of these two holes at any depth. Other causes like drill deviations, malfunctioning of blasting, scaling, etc. may be behind such EMD outlier profiles, uncorrelated with the MWD logs; these outliers, i.e. holes like no. 8 and 11, that happen at most in one or two holes per blast (many blasts do not show any) are removed from the analysis.

Table 4.8 shows the statistics for the MWD_{c1-c4} parameters, the lookout distance, and the EMD values for the contour holes drilled by boom 1 (54 rounds and 768 blastholes).

Table 4.8. Statistics of the MWD_{c1-C4} parameters for boom 1.

Parameter	Units	Mean	Std. ¹	Min.	Max.
Penetration Rate _{c4}	m/min	13.01	9.40	0.04	57.10
Hammer Pressure _{c3}	bar	211.81	15.04	63.00	230.0
Feed Pressure _{c2}	bar	73.94	5.71	21.20	79.89
Damp Pressure _{c3}	bar	65.31	12.76	48.23	86.63
Rotation Speed _{c4}	rpm	34.73	30.91	0.18	98.58
Rotation Pressure _{c4}	Bar	26.55	12.99	0.09	128.05
Water Flow _{c1}	l/min	81.99	28.20	53.70	145.56
Water Pressure _{c2}	bar	16.43	4.87	13.57	21.84
Lookout	m	0.26	0.15	0.02	0.94
EMD	m	0.14 ²	0.30	-0.63	1.09

¹Std. is standard deviation.

Different equations have been tried using only MWD parameters on one side (trials 1-2) and adding the L_{dist} on the other (trials 3-9). From them, the most significant equations are shown in Appendix 3, Table A3.1, where the equation type, the parameters included, the determination coefficient (R^2), the coefficient values and the p-value are described. Figure A3.1 in Appendix 3 shows the results of the equations described in Table A3.1. The best fit model obtained to predict the EMD is a power function of the MWD parameters and the Lookout distance (Trial 6, Table A3.1):

$$EMD = A_0 + \left(\frac{PR_{C4}}{13.01}\right)^{A_1} \cdot \left(\frac{RS_{C4}}{34.73}\right)^{A_2} \cdot \left(\frac{RP_{C4}}{26.55}\right)^{A_3} \cdot \left(\frac{WF_{C1}}{81.99}\right)^{A_4} \cdot \left(\frac{L_{dist}}{0.26}\right)^{A_5} \quad (4-15)$$

where L_{dist} is the lookout distance.

The effect of the differences in the magnitude of the variables of the model has been minimized to avoid that the large parameter magnitudes have a major determining factor in the model. For that, the model has been normalized, dividing the transformed parameters (MWD_{C1-C4}) from Eq. 4-15 by the mean value of their already transformed data set (Table 4.8). The use of normalized variables has the advantage that the product factor becomes independent of the units.

Since EMD has positive and negative values, an additive constant A_0 has been included. The transformed hammer pressure (HP), damp pressure (DP_{c3}) and water pressure (WP_{c2}) have been removed since their contributions are minimal and their exponents (for the DP and WP) are non-significant. The non-linear regression model has been programed with Matlab 2016b (Matlab, 2016) by an ordinary nonlinear least square method. The model coefficients are given in Table 4.9.

Table 4.9. Non-linear regression model coefficients.

	PR _{c4}		RS _{c4}	RP _{c4}	WF _{c1}	L _{dist}
	A ₀	A ₁	A ₂	A ₃	A ₄	A ₅
Coefficient	-0.8966	-0.1001	0.0985	-0.0231	0.1891	-0.1598

The determination coefficient of the fit R^2 is 0.73. Coefficient values estimated for each parameter are significantly different from zero, i.e. their p-value is <0.0001 for all cases, which means that the analysis is statistically significant. According to the value and sign of the coefficients, the MWD parameters affect the EMD differently:

- The normalized penetration rate (PR_{c4}) and rotation pressure (RP_{c4}) show negative correlation (negative sign in A₁ and A₄) with EMD. According to Schunnesson (1996, 1997), high fluctuations in the PR and RP signals indicate soft/fractured rock. High fluctuations mean high standard deviations so that, according to Eq. 4-14, PR_{c4} and RP_{c4} show lower values. This means that low PR_{c4} and RP_{c4} values indicate soft/fractured rock, prone to suffer over-excavation.
- The normalized rotation speed (RS_{c4}) shows positive correlation with EMD (positive sign in A₃). Schunnesson (1998) claimed that at high feed pressure, the rotation pressure required for the rotation of the bit increases and sometimes a reduction in the rotation speed can be seen. The higher resistance to rotate may also translate into an increase in the signal fluctuation. According to Eq. (4.14), the higher the standard deviation of the RS parameter, the lower the RS_{c4} value and the lower the excavated area that is created. For the case of discontinuities, Hustrulid (1968) showed that at low feed pressure, the bit will not be in constant contact with the bottom of the hole, resulting in a free rotation of the bit that may show a lower fluctuation in the rotation speed. This way, the lower the standard deviation of the RS parameter, the higher the RS_{c4} and the greater the excavated area.
- The normalized water flow (WF_{c1}) has a positive correlation with EMD (positive sign in A₅). This can be explained by the fact that, when discontinuities or soft rock are drilled, the control system requires a higher water flow due to the higher amount of drill cuttings or to the water leakage into the discontinuities.
- The lookout distance presents a negative correlation with EMD, which means, as discussed before, that the excavated area in relation with the blasthole position decreases with an increase of lookout.

The results of the EMD predicted with our model versus the EMD data are plotted in Figure 4.29a. The linear regression obtained has a slope of one with a zero-constant term. Figure 4.29a also shows the upper and lower prediction band at a 95% confidence level. The residuals are normally distributed.

Figure 4.29. Result of the EMD predicted model: a) predicted versus measured EMD values (upper graph: residuals); b) Box-plot of the RMSE values per blast.

Figure 4.29b shows the distribution of the root mean square error (RMSE) for each of the 54 blasts. The median and the 25th and 75th percentiles RMSE are 0.146 m, 0.121 m and 0.173 m, respectively. An illustration of the application of the model is shown in Figure 4.30 for three blasts, representing the 75 % (large error, Figure 4.30a), 50 % (expected error, Figure 4.30b) and 25 % (small error, Figure 4.30c) of the RMSE values; They correspond to blasts located at the chainages 388, 500 and 591 m, respectively. In the three cases, the EMD predicted is compared with the EMD measured for blastholes of boom 1. Five different over-excavation ranges have been defined. For high RMSE values (Figure 4.30a), visual differences between the predicted EMD and the EMD measured are apparent; these are considerably reduced when representing medium and low RMSE values (Figures 4.30b and 4.30c). Figure 4.30c shows a predicted EMD generally in line with the EMD measured though still some light differences exist. Considering the noise of the MWD data due to the uncertainty brought by drilling deviations, the scaling and primary support done before scanning the excavated section, sensors potentially out of

calibration, possible variations in the explosive linear density, etc., the quality of the fit is shows very good results.

Figure 4.30. Representation of the predicted EMD model (right graph) and the EMD measured (left graph) for blastholes of the boom 1: a) blast in chainage 388, 75th percentile of the RMSE; b) blast in chainage 500, median value of the RMSE; c) blast in chainage 591, 25th percentile of the RMSE.

4.4.4. Conclusions of rock mass over-excavation prediction from MWD data

The overbreak of the remaining rock mass in tunnel blasting has been analyzed in the light of MWD records, with the purpose of developing a prediction model of over- and under-excavation depths from blasting. Such a predictive model may also be seen as a drill or rock index that could be used to identify zones of potentially high geotechnical risk (those for which the over-excavation prediction is high). By comparison of scanner profiles of the excavated sections with the blasthole positions, a methodology has been developed to obtain an Excavated Mean Distance (EMD) between the blasthole and the excavated profile. This parameter may be considered a measure for quality control of the contour excavated. Quantitative predictions for different conditions would require a re-calibration of the model for the new site, following the methodology described here.

Given that blasting factors are constant (as from blast reports) the overbreak and underbreak are considered mainly influenced by the geotechnical condition of the rock mass and the location of blasthole (lookout), that will introduce the effect of the confinement of the explosive and the rock with depth that will influence the capacity of the former to break the rock. Such rock mass properties are assessed from MWD parameters that show the response of the jumbo to the rock before blasting. A multi-step transformation of the MWD logs has been carried out to filter out systematic variations due to the nature of the drilling process and to highlight the dependence of the rock in the parameters. This procedure includes: (i) filtering out of outliers, (ii) removal of the ramp-up section of the logs, (iii) correction of systematic variations in the MWD parameters caused by the influence of hole length and feed pressure and (iv) normalization with the standard deviation to account for fluctuations in the signals.

A non-linear power-form model has been developed that predicts the excavated mean distance as function of the normalized penetration rate, rotation speed, rotation pressure and water flow parameters, and the lookout distance. They combine the rotational, hydraulic flushing and the rate of advance of the drilling, and the confinement of the rock mass by depth. The model has a determination coefficient of 0.73, with the coefficients of the model strongly significant. Residuals are essentially normally distributed. The signs of the exponents indicate that normalized penetration rate, hammer pressure, rotation pressure and the lookout distance inversely influence the excavated distance i.e. high values for them reflect hard, unaltered rock. On the other hand, normalized rotation

speed and water flow are directly correlated with the excavated distance, so that high values for them indicate soft, fractured rock.

Chapter 5: MWD APPLICATION IN UNDERGROUND MINING PRODUCTION BLASTS

Nowadays, the charging procedure is carried out with no prior information of the rock mass condition, which limits the reaction to unexpected rock issues that may collapse the blasthole. This results in charging problems and, in consequence, bad fragmentation of the rock after blasting. Too coarse material (likely due to a small explosive concentration in the zone) may cause gravity flow problems and disturb ore extraction (Hartman, 1992), hampering ore loading and transportation. The performance data of jumbos through the MWD system can identify issues in the rock condition before charging.

This section demonstrates the use of the MWD technique in the Mamberget mine. The work done by Ghosh et al. (2018) has been used as starting point to classify the geomechanical rock condition in five classes (solid rock, fractured rock, cave-in, minor cavity and major cavity). From it, two applications have been developed: one for geotechnical rock condition of orebodies and the other for predicting the risk of collapse in blastholes. Ghosh et al. (2018)'s work has been improved into a rock condition block model to simplify the quantitative assessment and automatic recognition of trends in the rock condition. A multi-step transformation of the MWD parameters has been also applied to minimize external influences. From it, a model for the risk of blasthole collapse has been developed by comparing different situations of the geotechnical rock condition block model with the charging length of 102 production fan-holes (see section 3.2.1, pp. 36). The assessment of the number of blocked and non-blocked fan-holes and the ratio of charging length/blasthole length has been used to assign a high, medium or low risk of collapse to different situations. The results indicate collapses in the first half of the fan-holes for the high risk, collapses in the second half of the fan-hole for the medium risk and no collapses along the hole for the non-risk holes. The two models have been applied to the full scale for part of two orebodies in the Malmberget mine, which comprise 20 drifts and 5060 fan shaped long-holes.

The two models were developed during a PhD stay of the author in the Luleå University of Technology, Sweden and a full paper is being prepared for the journal "Rock

Mechanics and Rock Engineering” with the name of “Application of drill monitoring for chargeability assessment in sublevel caving”, see Appendix G.

5.1. Data analysis

5.1.1. MWD data processing

One general problem with MWD data analysis is that the logged response is a mixture of the response from varying rock mass conditions, from the drill rig control system and from the operator’s intervention (Schunnesson, 1998). All factors add uncertainty to the data that must be transformed to highlight changes in the parameters depending on the rock properties. This process is described next and comprises: (i) filtering of outliers, (ii) removing of systematic peaks due to the addition of a new rod, and (iii) correction of the hole depth influence.

Filtering of unrealistic values

Unrealistic values found in the MWD raw data are filtered out according to the criteria established by Ghosh et al. (2018), based on both frequency and practical experience in the Wassara water-hydraulic ITH technology. Table 5.1 describes the threshold values used for the analysis:

Table 5.1. Selection of intervals for filtering raw data.

Parameter	Units	Thresholds
Penetration Rate (PR)	m/min	$0.1 < PR \leq 4$
Percussive pressure (PP)	bar	$5 < PP \leq 200$
Feed pressure (FP)	bar	$35 < FP \leq 100$
Rotation pressure (RP)	bar	$25 < RP \leq 125$

Removal of systematic peaks due to the addition of a new rod

For long holes, systematic variations are related to the parameters response when a new rod is added; when the rock drill reaches the end of the feeder, a new rod must be added to continue the drilling. During this procedure, percussive pressure, feed pressure and rotation pressure parameters are shut down, the drill rod is pulled back and the new rod is added to the end of the last one. The logging system starts recording values again when the bit passes the last depth measure. Figure 5.1a represents the raw signal values of MWD parameters for a single hole, where one can see systematic drops at every 2.3 m,

coinciding with the addition of a new rod. Such systematic variations, that do not reflect any information of the rock conditions, are automatically filtered out here in a post analysis through a routine in Matlab (Matlab, 2017). Figure 5.1b shows the filtered signal from graph a, in which all peaks associated with rod changes have been removed.

Figure 5.1. Filtering to remove variations in the signal: a) Raw signals; b) Filtered signals.

Correction of hole depth influence

Figure 5.2 shows the average value of penetration rate (Av.PR), percussive pressure (Av.PP), feed pressure (Av.FP) and rotation pressure (Av.RP) at every 1 m hole length for the 5162 fan-holes analyzed. The calculation is shown for four drill rigs separately (represented with different color), to see whether there is any other systematic variation caused by the calibration of the rig sensors. From the four graphs, feed pressure (Av.FP) shows a well-defined increase with hole depth (HD) for all drill rigs. The correction of the hole length influence is calculated, for this parameter, for the four rigs together, since all of them follow the same trend (Figure 5.2a, Av.FP). Blastholes are normally drilled between 10 and 50 m length, corresponding the latter to blastholes in the middle of the fan. Since measures over 40 m are limited, they may produce deviations in the trend of the average signal at high hole depth values. These values are removed statistically from the analysis. For that, the empirical probability distribution function of the hole depth (HD) is built with the complete data set values and data above the percentile 97.5 (upper tail) are discarded. Since data in the ramp-up mode are not representative of changes in the rock mass conditions, the first drilling meter is excluded from the analysis to retain only values from the normal drillin.

Figure 5.2. Average variation for penetration rate (Av.PR), percussive pressure (Av.PP), feed pressure (Av.FP) and rotation pressure (Av.RP) with depth; data are measured at every 1 meter depth for the 5162 data gathered (each color corresponds to a different rig). HD is hole depth.

The average signal of the feed pressure (Av.FP) is calculated within 1 m to 36 m hole depth range for the hole depth trasformation analysis to exclude data from ramp-up operation and the upper tail (above percentile 97.5) of the CDF (see black dashed lines in Figure 5.2, Av.FP). Penetration rate, percussive pressure and rotation pressure show no influence by the hole depth and they are not transformed.

The correction of the hole length influence (to obtain a signal FP_n) is done by:

$$FP_n^i = [FP^i - FP_{fit}^i] + FP_{fit}^1; \text{ with } i = 1, 2, \dots, N \quad (5-1)$$

where i indicates the logs of each blastholes, being N the number of these. FP_{fit} is a linear regression with hole depth of the average signal at every 0.1 m hole length for the entire data. The determination coefficient of the fit R^2 is 0.92. FP_{fit}^1 is the intercept of the fit, i.e. the value at depth zero.

5.1.2. Geo-mechanical model

Ghosh et al. (2018) used Principal Component Analysis (PCA) to correlate penetration rate, percussive pressure, normalized feed pressure and rotation pressure parameters, plus the calculated penetration rate variability (Eq. 2-6), rotation pressure variability (Eq. 2-7) and fracturing index (Eq. 2-8). As shown in Figure 5.2a (Av.PR and Av.RP), the range of values for the penetration rate and rotation pressure is different between the four jumbos, maybe due to a different calibration of the sensors installed in each jumbo. To solve this,

the PR_{var} and RP_{var} values of the fracturing index (Eq. 2-8) are scaled with the standard deviation of their respective jumbo.

Based on the results from filming of the interior of a number of test holes, Ghosh et al. (2018) assessed first principal component (PC1) values to different rock properties and ranked these values into four different rock classes (solid rock, fractured rock, cave-in and cavity). To obtain these ranks, they represented, as probability distribution functions, the first principal component values for these four studied geotechnical categories, obtaining 4 curves, see Figure 5.3. Based on the shape of these curves and the relation between them, Ghosh et al. (2018) estimated five different rock classes ranks from the values of the 1st principal components (C1 is solid rock, C2 is fractured rock, C3 is cave-in, C4 is minor cavity, C5 is major cavity) and described their geotechnical features and their possible effect on blasthole chargeability. This work uses the estimated rock classes defined by Ghosh et al. (2018) as starting point to build the geotechnical rock condition block model and, from it, to develop the risk of collapse model.

Figure 5.3. Proposed geo-mechanical model based on the probability density function of the 1st principal component values of each rock class (Ghosh et al., 2018). C1 is solid rock, C2 is fracture rock, C3 is cave-in, C4 is minor cavity, C5 is major cavity.

Figures 5.4a and 5.5a represent the five rock classes estimated from the PC1 interpretation, based on Ghosh et al. (2018) methodology, for two expected bad and good rock condition rings, respectively. Appendix 3 shows the results for the rest of the blastholes described in Table 3.4. The rock classification is given in different colors as can be seen from the legend. The black line at the contour of the fan-holes represents the

measured charging length (see section 3.2.1, pp. 33). Charging lengths shorter than the total fan-hole length point out a collapse in hole at the end of this line, hence charging problems. In addition, no charging length drawn indicates a collapse at the collar position, leaving this hole uncharged. Although there is always an uncharged part at the beginning of the hole, the charging length data is drawn from the collaring position of the blasthole to determine the location where it is collapsed.

Figure 5.4. Analysis for an expected problematic ring (Orebody 1, ring 4): a) Classification based on a ranking from first principal components interpretation (Ghosh et al.'s, 2018 methodology); b) Rock condition block model developed here (see section 5.3); c) Risk of collapse planning model developed here (see section 5.4).

Figure 5.5. Analysis for an expected good ring (Orebody 3, ring 2): a) Classification based on a ranking from first principal components interpretation (Ghosh et al.'s, 2018 methodology); b) Rock condition block model developed here (see section 5.3); c) Risk of collapse planning model developed here (see section 5.4).

5.2. Drill rig navigation

Navigation in the Atlas Copco SIMBA W6C drill rigs is similar to jumbos for tunneling constructions (see section 4.2); they use three reference systems to locate the drilling machine inside the drift to carry out the drilling of the fan shaped long-holes:

- i. An Absolute Coordinate System (ACS) that references the position of the drill rig by trilateration, i.e, the drill rig uses a laser scanner to measure the distance from the scanner to target points with known coordinates located along the wall of galleries and drifts (see Figure 5.6a). For the case under study, the ACS is measured in the EUREF 89 Norwegian Transverse Mercator (NTM) projection.
- ii. A Total Reference System (TRS) composed of one axis parallel to the drift line (Z_t) and the other two (X_t and Y_t) in the cross section of the drift (see Figure 5.6b). To obtain the TRS, the feeder is aligned with the drift line; the laser beam of the scanner points in the direction of the drift line and the feeder is rotated until the laser beam passes through two targets mounted on it.
- iii. A Drilling Reference System (DRS) defined by two vertical planes X_dY_d , Y_dZ_d and a horizontal X_dZ_d plane. The drill rig monitors, in the Drilling Reference System, the collaring and end position of the fan shaped holes before drilling starts; the final position is automatically calculated by means of three spherical coordinates: the hole length, the azimuth and the inclination angles of the feeder (see section 4.2).

Figure 5.6. Drill rig navigation; a) Absolute Coordinate System (ACS); b) Alignment with drift line to measure the Total Reference System (TRS).

During the navigation operation, the drill rig transforms the DRS coordinates of a point in the fan-hole to the TRS by means of three rotation angles (γ , θ , ω). For Atlas Copco SIMBA W6C drill rigs, the directional coordinate vectors of the TRS ($\vec{x}_t$, $\vec{y}_t$, $\vec{z}_t$) and the NTM coordinates of the rig (X_{NTM} , Y_{NTM} , Z_{NTM}) are presented at the beginning of each

MWD file. This transformation system is similar than the use for tunneling operations and it is described in section 4.2.4 in Figure 4.12.

5.3. Result 1: Geotechnical rock condition block model

In ranking from first principal components interpretation developed by Ghosh et al. (2018) (Figures 5.4a and 5.5a), it can sometimes be difficult to interpret the graphs due to the large number of different rock classes that result in many short intervals (mix of colors along the fan-hole). This complicates the quantitative assessment or the automatic recognition of rock condition trends and chargeability. To solve this, the first principal component results have been divided in zones based on abrupt changes of the mean value in the signal. The procedure divides the signal in a pre-set number of sections and estimates the mean value for each one. These sections are varied iteratively in length until the total squared error attains a minimum (Lavielle et al., 2005; Killick et al., 2012). Considering that the first principal component is a signal x of length N , the iteration finds k (for a two-section case) such that J , defined in Eq. 5-2, is smallest. Eq. 5-2 can be easily generalized to any number of sections n and subdivisions k_1 to k_n . The minimum of the residual squared error is obtained by using Gaussian log-likelihood.

$$J = \sum_{i=1}^{k-1} \left(x_i - \frac{1}{k-1} \sum_{r=1}^{k-1} x_r \right)^2 + \sum_{i=k}^N \left(x_i - \frac{1}{N-k+1} \sum_{r=k}^N x_r \right)^2 \quad (5-2)$$

The analysis has been programmed in Matlab (Matlab, 2017). For the test fan-holes, the maximum number of sections is pre-set to six ($n = 6$). Figure 5.7 shows an example of this procedure for blasthole 9 of the Orebody 1, drift 8 (Table 3.5), where the green dot line indicates abrupt changes in the mean value of the signal and red dot line represents the average of the values within these sections.

Figure 5.7. División of the 1st PC in six zones based on abrupt changes in the signal

A new set of limits has been defined from the average of the first principal component in each section for the test fan-holes (Table 3.4), to match the five rock classes estimated by Ghosh et al. (2018); this ranking is given in Table 5.2. Figures 5.4b and 5.5b show the geotechnical block model application to the first principal components results from Figures 5.4a and 5.5a. They are also shown in Appendix 3 for the rest of the blastholes from Table 3.4.

Table 5.2. New rank to assess the five rock classes in the geotechnical block model.

Rock class	Threshold
Solid rock	$x^1 \leq 0.44$
Fractured rock	$0.44 < x \leq 2.5$
Cave-in rock	$2.5 < x \leq 5$
Minor cavity	$5 < x \leq 8$
Major cavity	$8 < x$

¹ x is the average value of the first principal component in a given section.

5.2.1. Full scale application of the block model

The application of the rock condition block model analysis on larger mining areas (data in Table 3.5) is done following the above explained procedure. The representation of the corresponding fan shaped long-holes in a 3D plot is carried out by considering both collaring and end coordinates of the hole in the DRS, the alignment of the drill rigs (TRS) and its absolute coordinates when starting to drill a new ring. The rotation from the DRS coordinates to the TRS is also considered. For orebody 1, the fan-hole coordinates in the local DRS are rotated 212.5° around axis Y (i.e., rotation of X_dZ_d plane, $\gamma = 212.5^\circ$) with respect the TRS, being $\theta = 0$ and $\omega = 0$. For orebody 4, DRS and TRS reference system match, so θ , γ and ω angles are 0°. The translation to the absolute coordinates is obtained by adding the NTM coordinates of the drilling machine to the oriented coordinates (TRS) of the fan-holes. Figures 5.8 and 5.9 show the full scale representation in 3D of the geotechnical block model for orebody 1 and 4, respectively. They represent the parallel drifts layout and the fan shaped long-holes of each blast, drilled above the drifts. Rock classification, consolidated as solid rock, fracture rock, cave-in rock, minor cavity and mayor cavity, is colored in dark blue, green, red, cyan and yellow, respectively; the drift layout of each orebody level is represented in black.

The rock condition block model for orebody 1 (Figure 5.8) is developed for one level, with nine drifts drilled at 1052 m level; a clear trend of fracture and cave-in rock can be identified (see red circle, Figure 5.8), which might be related with geotechnical problems

found, as from internal reports from LKAB (LKAB, 2017). Some cave-in and cavity rock classes are found at the end of the fan-holes located in the middle of each ring, which may be related to the drilling of already fractured rock in upper levels. The upper part of these fan-holes often drills rock mass affected by adjacent blasts from upper levels matching these features (see yellow circles, Figure 5.8b).

Figure 5.8. Rock condition block model in 3D for orebody 1: a) top-isometric view; b) isometric view

Full scale geotechnical block model for orebody 4 (Figure 5.9a) includes two levels; the lower level comprises nine drifts drilled at 1056 m and the upper level encompasses two drifts at the 1031 m level. On the left side of the plot (see red circle), there is an area with high density of green and red color that corresponds to a fracture and cave-in zone; this belongs to a steep zone with softer or more fractured rock going through the two levels

(LKAB, 2017). Figure 5.9b shows a zoom-in representation of the problematic area represented in Figure 5.9a (see red circle).

Figure 5.9. Rock condition block model in 3D for orebody 4: a) top-isometric view; b) Zoom in of the rock condition block model in 3D for orebody 4: Front view (left), Top view (centre), isometric view (right).

5.4. Results 2: Risk of blasthole collapse planning model

Geotechnical issues in the rock surrounding fan-holes are close related to blasthole collapses. Since a hole is an open aperture in the rock, the long period of time between drilling and blasting further exposes the original properties of the ground condition around the fan-holes to blast vibrations and to stress redistributions (Kwon et al. 2009). When the surrounding stress exceeds the tensile, the compressive, or the shear strengths of the rock formation, failures in the fan-hole wall can be formed (Zhang et al. 2003). As result, rock detachments may be generated with possible collapses inside the hole, which later may result in charging problems.

As explained in section 5.3, the rock condition block model can characterize geotechnical issues around the blastholes. Figure 5.4b shows that problems in the measured charging length (black line drawn at the contour of the hole) are often generated when a large fracture zone is found (see green zones in fan-holes 6 and 7 starting from left) or when a fracture zone follows a cave-in zone or vice versa (fan-holes 3 and 8 starting from left), even if there is a short solid rock zone between them (fan-holes 5 and 9 starting from left). Figure 5.5b demonstrates, on the other hand, that when solid rock conditions are dominant, no charging issue is expected.

The prediction model for the probability of risk of collapse in the blasthole is carried out for the test fan-holes described in Table 3.4. Fan-holes have been grouped based on the existence (according to the developed block model) and length of zones where solid rock, fractured rock, cave-in and cavities along the hole are defined. In this case, minor and major cavities are considered as one class, hence, the analysis is done only for four different rock classes. The charging length (see section 3.2.1, pp. 36) has been used to estimate a mean charging length/blasthole length (L_c/L_b) ratio for each group and to assess hole chargeability. Two different types of classification have been carried out: Table 5.3 (cases 1 to 4) shows the situation when only solid rock or when solid rock plus one of the classes fracture, cave-in or cavity rock appear in the fan-hole and Table 5.4 (cases 5 to 8) describes the situation when there are more than two rock classes in a fan-hole. Cells highlighted in the left part indicate the rock class and the corresponding length interval. The number of blocked and non-blocked blastholes, the mean charging length/blasthole length (L_c/L_b) ratio and the estimated risk of collapse are also shown for each situation. The latter is assessed from the value of the L_c/L_b ratio for each situation considering the amount of blocked or non-blocked blastholes. In this way, the smaller the L_c/L_b ratio and the higher the number of blocked blastholes, the higher the risk of collapse for each situation and vice versa.

According to Table 5.3, the existence of a large fractured zone (length, $L > 15$ m) or a large cave-in zone ($L > 2$ m) are always related with blocked blastholes. The mean L_c/L_b ratio in these situations is low (0.24 and 0.45, respectively), which indicates that there is a high risk that the hole collapses in the first half of the fan-hole. The presence of one of these two situations in a fan-hole will indicate high risk of collapse. On the other hand, only solid rock or the existence of short fractured zones ($L < 3$ m) or cave-in zones ($L < 2$ m) normally show no collapse in the blasthole, with a mean L_c/L_b ratio above 0.90.

These situations thus predict no risk of collapse in the fan-hole. An intermediate result is found for fractured zones between 3 and 15 m. Although the number of blocked blastholes for this situation exceed the non-blocked holes, there are also many cases where the blasthole was not blocked. In this way, a medium risk of collapse has been considered. No situation of solid rock and cavity classes has been found in the test fan-holes analyzed. A medium risk of collapse has been assigned when there is a large cavity zone ($L > 2$ m) and no risk of collapse for small cavities ($L < 2$ m). This is made because the existence of a large cavity may hinder the charging of the hole, since the hose used to charge the explosive into the hole may get stuck. However, further measurements for these two situations are necessary to validate the assumption.

Table 5.3. Situation when only solid rock or when solid rock plus either fractured, cave-in or cavity rock classes appear in the blasthole. L is length (m). Grids for solid rock, fractured rock, cave-in and cavities are highlighted in blue, green, red and yellow, respectively.

Case	Solid rock	Fractured rock			Cave-in		Cavity		No. Blasthole		Ratio L_c/L_b^1 (mean $\pm$ std)	Risk of collapse
		$L < 3$	$3 < L < 15$	$L > 15$	$L < 2$	$L > 2$	$L < 2$	$L > 2$	Blocked	Non-blocked		
1		✓							0	7	0.98 ± 0.02	No risk
2	a	✓		✓					10	0	0.24 ± 0.27	High
	b	✓	✓						12	9	0.7 ± 0.4	Medium
	c	✓	✓						2	23	0.96 ± 0.2	No risk
3	a	✓				✓			2	0	0.45 ± 0.35	High
	b	✓			✓				0	3	0.95 ± 0.03	No risk
4	a	✓						✓	-	-	-	Medium
	b	✓					✓		-	-	-	No risk

¹Mean value of ratios charging length/ blasthole length for each blasthole.

The risk of collapse assignments listed in Table 5.4 are based on the same rock conditions and rock class lengths than in Table 5.3. For cases 6, 7 and 8, the number of test fan-holes found is limited thus the risk of collapse assignment has been made based on the results from Table 5.3. Any situation with a large fractured zone (length, $L > 15$ m) and/or a large cave-in zone ($L > 2$ m) is correlated with high risk of collapse; since any of these situations separately normally block the blasthole, their situation with another rock class type will increase this risk of collapse. This occurs for cases 5a, 5b, 5c, 5e, 6a, 6b, 7a, 7b, 8a, 8b, 8c, 8d, 8e, 8f, 8i and 8j (Table 5.4). Of these, only cases 5a, 5b, 5c, 5e, 6b, 7a, 8d, 8e have been found in the test fan-holes. These situations normally show a mean L_c/L_b ratio lower than 0.5 and the majority of blastholes have collapsed.

Table 5.4. Situation when more than two rock classes appear in the blasthole. L is length (m). Grids for solid rock, fractured rock, cave-in and cavities are highlighted in blue, green, red and yellow, respectively.

Case	Solid rock	Fractured rock			Cave-in		Cavity		No. Blasthole		Mean ratio L_c/L_b^1	Risk of collapse
		$L < 3$	$3 < L < 15$	$L > 15$	$L < 2$	$L > 2$	$L < 2$	$L > 2$	Blocked	Non-blocked		
5	a	✓		✓		✓			2	0	0.15 ± 0.21	High
	b	✓		✓	✓				4	0	0.35 ± 0.4	High
	c	✓	✓			✓			6	0	0.65 ± 0.08	High
	d	✓	✓		✓				5	4	0.79 ± 0.20	Medium
	e	✓	✓			✓			2	0	0.8 ± 0.2	High
	f	✓	✓		✓				0	6	0.98 ± 0.02	No risk
6	a	✓		✓				✓	-	-	-	High
	b	✓		✓			✓		1	0	0.8	High
	c	✓	✓					✓	-	-	-	Medium
	d	✓	✓				✓		-	-	-	Medium
	e	✓	✓					✓	-	-	-	Medium
	f	✓	✓				✓		-	-	-	No risk
7	a	✓				✓		✓	1	0	0	High
	b	✓				✓	✓		-	-	-	High
	c	✓			✓			✓	-	-	-	Medium
	d	✓			✓		✓		0	1	1	No risk
8	a	✓		✓		✓		✓	-	-	-	High
	b	✓		✓		✓	✓		-	-	-	High
	c	✓		✓	✓			✓	-	-	-	High
	d	✓		✓	✓		✓		1	0	0.25	High
	e	✓	✓			✓		✓	1	0	0.1	High
	f	✓	✓			✓	✓		-	-	-	High
	g	✓	✓		✓			✓	1	0	0.8	Medium
	h	✓	✓		✓		✓		-	-	-	Medium
	i	✓	✓			✓		✓	-	-	-	High
	j	✓	✓			✓	✓		-	-	-	High
	k	✓	✓		✓			✓	-	-	-	No risk
	l	✓	✓		✓		✓		0	1	1	No risk

¹Mean value of ratios charging length/ blasthole length for each blasthole.

An exception occurs when a fractured zone (medium or large) is followed by a large cave-in zone or when just a large cave-in zone appears. In this case, the blasthole is collapsed at the end or along the cave-in zone, independently of its location. Large cavities and/or cave-in zones at the collar or large fractured zones along the hole normally collapse the blasthole from the collar giving a zero L_c/L_b ratio. Cases 6a, 7b, 8a, 8b, 8c, 8f, 8i and 8j

have not been found in the test fan-holes and their risk of collapse classification has been assigned based on the results from Table 5.3.

On the other hand, the existence of any combined situation based on short fractured zones ($L < 3$ m), cave-in zones ($L < 2$ m) and/or cavity ($L < 2$ m), normally shows no blasthole blocked, and has a mean L_c/L_b ratio above 0.90. These situations indicate no risk of collapse and comprise cases 5f, 6f, 7d, 8k, 8l (Table 5.4). Medium risk of collapse is assigned to situations with fractured zones between 3 and 15 m, short cave-in zones ($L < 2$ m) and/or both large or short cavities (cases 5d, 6c, 6d, 6e, 7c, 8g and 8h, Table 5.4). The measured L_c/L_b ratio for these situations lies between 0.7 and 0.9.

Examples of the risk of collapse results from the rock condition block model are plotted in Figures 5.4c and 5.5c and in Appendix 3 (c graphs) where all blastholes analyzed (those from Table 3.4) are represented. In these graphs, the respective situation case number from tables 5.3 and 5.4 are added to each fan-hole. As can be seen, most of the situations studied are related to cases 1, 2 and 5, i.e., to the existence of solid rock and solid rock plus fractured and/or cave-in zones in the blastholes.

5.4.1. Full scale application of the risk of blasthole collapse model

In line with the procedure followed in section 5.3, the risk of collapse model can also be used for evaluation of larger mining areas. Since the model indicates the risk of collapse along the whole fan-hole it can be represented in a 2D map with the collaring coordinates of the horizontal XZ plane plotted in the absolute projection. Figures 5.10a and 5.10b show the full scale representation of the risk of collapse model for orebodies 1 and 4 respectively. The prediction model is represented over the mining layout for each level, allowing to determine potentially problematic areas at the operational level. Figure 5.10a represents the mining layout at level 1052 in orebody 1 and Figure 5.10b shows the mining plan design at level 1056 m in orebody 4. The latter also shows, to the left, two drifts lying outside the layout which corresponds, as commented above, to data from the upper level at 1031 m. Figures 5.10a and 5.10b show the collaring positions of the fan shaped long-holes drilled to blast the ore section above the drifts, in orebodies 1 and 4, respectively. The color of each collaring position indicates their predicted risk of fan-hole collapse. Similar to Figure 5.8, orebody 1 (Figure 5.10a), shows a potential problematic zone on the left side of the plot with high and medium risk of collapse in the fan-holes. This matches internal reports of the mine (LKAB, 2017) where, in this zone, production has stopped due to the impossibility to charge the holes (being most of them blocked) and

due to stability problems of the roofs and walls of the drifts. For the case of orebody 4 (Figure 5.10b), the problematic zone described in Figures 5.9a and 5.9b indicate a zone with medium and high risk of collapse in the holes, where charging problems are expected.

Figure 5.10. Risk of collapse planning model: a) orebody 1; b) orebody 4.

The presented model of risk of blasthole collapse provides an early information of expected charging problems before starting the blasting operation. The model can be used as a decision-making tool in the daily routine for production planning and to minimize unexpected problems during the charging of the fan-holes. These features have the potential of improving rock fragmentation, with the final purpose of optimizing production.

5.5. Conclusions of MWD applications in underground mining

The development of a geotechnical rock condition block model and a predicting risk of blasthole collapse model has been carried out, with reference to MWD records, aiming at minimizing problems during the charging of holes. Such problems are closely related with non-optimal rock fragmentation after blasting. The study has been applied to individual blasts and to full scale, comprising 11 drifts with 102 fan-holes from five orebodies, in the first case, and 20 drifts with 5060 fan-holes from two orebodies in the second, located in

the Luossavaara-Kiirunavaara AB's (LKAB) Malmberget underground iron ore mine, Sweden.

The analysis carried out by Ghosh et al. (2018) has been used as starting point of this work to classify the geotechnical rock condition in five classes: solid rock, fractured rock, cave-in rock, minor cavity and major cavity. Two applications have been developed: one for geotechnical rock condition of orebodies and the other for predicting the risk of collapse in blastholes. Ghosh et al. (2018)'s work has been improved into a rock condition block model to simplify the quantitative assessment and automatic recognition of trends in the rock condition. For that, a multi-step transformation of the MWD parameters has been carried out in order to minimize external influences other than the rock mass. This comprises: (i) filtering of unrealistic values, (ii) removing of systematic peaks due to the addition of a new rod, and (iii) correction of the hole depth influence.

The risk of blasthole collapse model has been developed by comparing different situations of rock classes from the geotechnical block model with the charging lengths of 102 production fan-holes. The model considers the existence and length of solid rock, fractured rock, cave-in and cavities along the hole. The assessment of the number of blocked and non-blocked fan-holes and the charging length/blasthole length ratio has been used to assign high, medium or low risk of collapse to each situation. The results predict collapses in the first half of the fan-holes for the high-risk parts, collapses in the second half of the fan-holes for the medium risk and no collapses along the hole for the non-risk holes. However, further measures are necessary to validate this model.

The two models have been applied to full scale for two orebodies in the Malmberget mine, which comprises 20 drifts and 5060 fan shaped long-holes. Considering that, presently, the charging procedure is carried out with no prior information of the rock mass condition, the two models presented here provide an early information of the rock mass condition around the blastholes and a potential prediction of expected charging problems before starting the blasting operation. This could be used as a decision-making tool in the daily routine for production planning and to minimize unexpected problems during the charging of the fan-holes. These features have the potential of improving rock fragmentation, with the final purpose of optimizing production.

Chapter 6: MWD APPLICATION IN QUARRIES

The high cost and the difficulty to retrofit a drill monitoring system on old and existing drill rigs, restrain the use of this technology in numerous small quarries and mines. In addition, the existing interpretation software packages are tailored for the supplier's drill rigs and do not reveal the details on how they manage the MWD records. This complicates their application as a decision-making tool.

An alternative low-cost MWD system to monitor and record the information and performance of any drilling system is presented in this chapter and in Appendix F. For that, the digitization and automatic sampling of the analog signals from the sensors involved in the rig control have been carried out. With the purpose of developing an engineering tool for geotechnical rock recognition, a fracturing index has been developed by combining both the variation and the magnitude of several parameters. The results have been validated with photographic records from an optical televiewer.

This work is a part of the Sustainable Low Impact Mining project's solution for exploitation of small mineral deposits based on advanced rock blasting and environmental technologies (SLIM Project) funded by the European Commission under H2020 research and innovation program (Sanchidrián, 2018) and has been recently published under the title "Application of an in-house MWD system for quarry blasting" in the Fragblast 12 conference; this corresponds to paper F described in Table 1.1 and can be seen in Appendeix F.

6.1. Description of the in-house prototype

The available drilling machine in El Aljibe quarry (see section 3.3) is a Tamrock DX-800 surface drill rig from 2008, equipped with a top hammer rotary-percussive rock drill. The rig uses 3.2 m rods with a diameter of 58 mm and 89 mm bits. Since the rig supplier does not provide any MWD system for this rig, an in-house MWD system was developed as a low-cost and efficient alternative to monitor the information obtained from the rig while drilling.

Integra Automatización S.L.U. did the design and the implementation of the system. It records and digitizes the analogical signals of the existing sensors on the drill rig. PLC

(Programmable Logic Controller) and HMI (Human Machine Interface) units were developed to fulfil the following actions:

- Design of a graphical user interface to represent live data of depth and pressure parameters on a 7" screen installed on the drill rig.
- Conversion from the existing analog signals to digital data.
- Data logging and storing.

Drill parameters are digitized with a parallel connection to the sensors already installed on the rig for the analog logging of the required parameters that control the drilling. The system monitors information at every sample interval value of depth. For this, the rotary in-home encoder installed in the rig was used to detect an increase of depth during drilling by transforming the encoder pulses into linear displacement; at every pulse, the system records information of the parameters when there is an increase in the hole depth count. The sample interval was preset at 0.01 m. The parameters monitored by the system are, with their acronyms and units:

- Hammer pressure (HP, bar)
- Feed pressure (FP, bar)
- Rotation pressure (RP, bar)
- Air pressure (AP, bar)
- Damp pressure (DP, bar)
- Hole inclination (°)
- Hole depth (HD, m)

Two inclinometers were installed on the mast, to measure the vertical (inclination) and horizontal (azimuth) angles with respect to the direction defined by the mast of the drill rig in vertical position.

The technical specifications of the electronic sensors used to digitize the analog signal of the parameters and the inclination of the blasthole are:

- a. Hammer pressure: electronic pressure sensor with a range of 0 to 600 bar (maximum pressure 2500) with a resolution of 0.0183 bar.
- b. Rotation pressure, Feed pressure and Damp pressure: electronic pressure sensor with a range of 0 to 250 bar (maximum pressure 1200) with a resolution of 0.0066 bar.

- c. Air pressure: electronic pressure sensor with a range of 0 to 10 bar (maximum pressure 300) with a resolution of 0.0003 bar.
- d. Inclinometers: they gather the information of local inclination and azimuth angles with respect to the direction defined by the mast of the drill rig in vertical position.

Other technical common characteristics of the sensors are: working temperature range (-40 to 90 °C), resistance to the vibrations (20 g, in a bandwidth of 10 to 2000 Hz), accuracy (<0.5%) and response time (1 ms).

Data logging and storage are carried out as follow: the PLC receives electrical signals from the pressure, inclination and depth sensors which are handled by the program and they are recorded every time the encoder increases a pulse (i.e., in a length step of 0.01 m) and, in addition, the following boundary pressure conditions are fulfilled:

- Hammer pressure greater than 80 bar
- Feed pressure greater than 30 bar
- Rotation pressure greater than 25 bar
- Air pressure greater than 3 bar.

The visualization of the logging data is done through a 7" touchscreen from which the operation can be controlled. Specific events, such as water in the hole, addition of a new rod or a new bit, drilling in manual or automatic mode, a visible fracture, etc., can be manually entered into the system and visualized on the screen. This will help to understand the monitored data in a subsequent analysis. Figure 6.1a shows the MWD logger of the system in the cabin of the Tamrock DX-800 and Figure 6.1b represents a view of the monitor screen, where all parameters explained above are represented.

Once the blast is drilled, the data is saved in a text file and can be exported with a memory stick.

Figure 6.1. Installation of the in-house drill monitoring system. a) operator's cab; b) view of the monitor screen.

6.2. Data overview

The developed drill monitoring prototype installed on a Tamrock DX-800 rig was used for the monitoring of the first campaign of full scale tests (included in the SLIM project) in El Aljibe quarry (Segarra et al., 2018). The rock quarried in El Aljibe is mylonite, which is a fine-grained metamorphic rock formed by ductile deformation under shear stress (see section 3.3).

Drilling was done in automatic mode, i.e., parameters are automatically adjusted by the drill rig system during the operation. The study comprises six blastholes from one blast drilled by the prototype.

The condition of the blastholes was analyzed with an optical televiewer manufactured by Advanced Logic Technology. The televiewer provides a continuous unwrapped 360° oriented color image of the blasthole walls (Li et al. 2013), and it is mainly composed by:

- The logging tool quick link QL40 OBI-2G (Figure 6.2), of 1.47 m length, diameter of 0.40 m and mass of 5.3 kg has a digital image sensor at the bottom end with an active pixel array of 1.2 Mega Pixel and fisheye matching optics. The light source is provided by ten LEDs. This tool also includes hole deviation sensors in the central part, a connector to the wireline in the top part and two centralizers.
- The data acquisition system (BBox).

- The mini-winch with 200 m of 1/8'' wireline consisting in a steel armored cable with an insulated conductor at the center that gives power to the tool, allows data exchange with the surface, and moves the logging tool along the blasthole a constant velocity.
- A computer with the software ATL Logger Suite 11.2.

Figure 6.2. General aspect of the QL40 OBI-2G (User Guide OBI, 2017).

6.3. Data analysis

The comparison between the monitored MWD parameters and the televiewer records will improve the understanding of the drill/rock interaction, with a final purpose to develop a model to characterize the rock condition. Figure 6.3 shows an example of the monitoring data for blasthole 5 of the blast and the corresponding televiewer records. For the first 4.5 m, negative peaks in the hammer pressure (HP) and feed pressure (FP) and a significant variation in the rotation pressure (RP) can be seen, that may be related to a possible zone of disturbed rock (a different color is depicted in the televiewer records). In the same way, fluctuations in the MWD signals at 11.2 m may show another possible disturbance zone. Schunnesson (1996, 1997) claimed that when discontinuities are encountered, the rotation pressure shows significant fluctuation, resulting in a noisy signal. Peng et al. (2005) determined the feed pressure and thus hammer pressure as good indicators for void detection. In addition, as demonstrated in section 4.1, the feed pressure has been identified as the parameter that controls the adjustment of the other parameters when variations while drilling occur. Damp pressure and air pressure, on the other hand, barely show changes found in any of the other parameters and thus, they will not be considered for the subsequent analysis.

Figure 6.3. Signal of the monitoring system for blasthole no. 5 and televiewer record of the wall condition.

For a proper measurement, the entire logging tool must be introduced in the hole to ensure that the two centralizers are in contact with the rock and the camera is located in the center of the hole. The first 1.5 m that correspond to the length of the logging tool cannot be measured. For blasthole no. 5 (Figure 6.3), the last 1.4 m were not measured with the televiewer as the blasthole was blocked. This may be related to a zone of disturbances indicated by the noisy MWD signals at this depth. The rest of the parameters' signal traces are mainly flat and indicate solid rock.

Once the feeder reaches the end of the rod, a new one must be added in order to continue the drilling. At this point systematic variations in the parameter responses (systematic peaks) are observed, which occurs at intervals of 3.2 m. During this procedure, hammer pressure, feed pressure and rotation pressure parameters are shut down, the drill rod is slightly pulled back to avoid the contact between the bit and the bottom of the hole and the new rod is added to the end of the last one. The logging system starts to record values again when the new length measurement passes the previously deepest measure. Figure

6.3 also shows the depths where a new rod is added (bottom plot, before televiewer log). Systematic variations due to the addition of a new rod are filtered out in a post analysis, since these obviously do not reflect any information of the rock (see section 5.1.1, pp. 90).

6.4. MWD fracture index

According to Schunnesson (1996, 1997), Peng et al. (2005) and section 4.1, fluctuations in the signals of feed pressure, hammer pressure and rotation pressure parameters indicate changes in the geotechnical rock condition. These variations are highlighted and calculated as the sum of the absolute residuals over a defined interval along the blasthole. For this, Eq. (2-6) has been used to calculate variations in the rotation pressure (RP_{var}) and the same equation has been applied to the hammer pressure (HP_{var}) and feed pressure parameters (FP_{var}) for the same purpose.

The variations of the feed pressure, hammer pressure and rotation pressure (HP_{var} , FP_{var} , RP_{var}) at each i sample have been combined, with the same influence, in a new fracturing index. Unlike Ghosh et al. (2018) (Eq. 2-8 above), the magnitudes of the variations of each parameter have been scaled with the mean value of the raw parameter log per hole, with the purpose of minimizing differences between the magnitude of these three parameters. Each scaled parameter is square-root transformed in order to highlight smaller variations of interest to be compared with very high peak values:

$$\text{Fract. Index} = \left(\frac{HP_{var}}{HP} \right)^{1/2} + \left(\frac{FP_{var}}{FP} \right)^{1/2} + \left(\frac{RP_{var}}{RP} \right)^{1/2} \quad (6-1)$$

6.5. Results

The fracturing index from Eq. 6-1 has been compared with the televiewer records. As an example, Figure 6.4 represents, for blasthole no.5, the fracturing index and two pictures of the wall condition inside the blasthole (a 2D 360° scan of the wall and a 3D reconstruction). The trace of the different rock features is marked; seven different types of rock disturbances have been found: open fracture, continuous closed fracture, discontinuous fracture, filled fracture, weakness area, void and change of the lithology. The fracturing index from this blasthole is plotted in Figure 6.4. It has peaks and

variations when the bit reaches a rock disturbance. As can be seen in Figure 6.4, fracturing index values greater than 1 normally indicate significant rock disturbances.

Figure 6.4. Comparison between fracture index and televiewer records, blasthole 5 (section and 3D core reconstruction).

Table 6.1 describes the type of disturbance that creates each signal variation in the fracturing index of Figure 6.4. Different ranges of fracture values can be distinguished according the complexity of the rock disturbance. The more complex the disturbance, which normally includes voids, large fractures and weakness areas, the higher and wider the fracturing index peak.

Table 6.1. Description of the rock disturbances that creates changes in the fracturing index of hole N° 5, blast 1.

Hole depth (m)	Fracturing Index (range of values)	Rock disturbances
1.4 – 2.3	2 – 3.5	Open fracture Cont. closed fracture Weakness area Void
2.65 – 2.85	2	Open fracture Cont. closed fracture Weakness area
3.5 - 4	1.5 – 3	Open fracture Weakness area Void
4.3 – 4.6	1.5 – 2	Cont. closed fracture Weakness area
5 – 5.2	2.8	Cont. closed fracture Filled fracture Change of lithology
5.4 – 5.6	1.2	Cont. closed fracture Discontinuous fracture
7.6 – 7.7	1.5	Cont. closed fracture Discontinuous fracture
11.1 – 11.8	2 – 3	Filled fracture

Although significant variations in the fracturing index are always related to rock disturbances, not all disturbances found in the televiewer records can be noticed in the MWD records. This is the case of some filled fractures and isolated continuous closed fractures found at hole depths 6.2 m (filled fracture), 6.95 – 7.05 m (filled fracture and continuous closed fracture), 8.8 m and 10.78 m (isolated continuous closed fracture). However, most of these isolated continuous closed fractures are almost horizontal with minimum influence during the drilling and the filled fractures are probably tight with recrystallized material and may not have significantly different properties of the rock that could possibly be noticed by the monitoring system.

The fracturing index shows a visual correlation between the geotechnical classification of the blasthole condition and the MWD parameters (Figure 6.4). In line with the

geotechnical rock classification of the fan-holes described in section 5.3, the fracturing index per hole has been divided in zones based on abrupt changes of signal variations. The procedure in this case consists of dividing the signal in a preset number of sections and estimating the standard deviation for each one. These sections are varied iteratively to minimize the sum of the residual error of each region in the standard deviation by using Gaussian log-likelihood (Lavielle et al., 2005; Killick et al., 2012). Considering the long length of the signals (more than 1200 points per blasthole) and the width of the fluctuations, the maximum number of sections is preset to 30 per blasthole (Figure 6.5). The analysis has been programmed in Matlab (Matlab, 2017).

Figure 6.5. Division of the fracturing index in zones based on abrupt changes of variability in the signal, blasthole 5.

The mean fracturing index in each section is calculated and classified in five different ranges of values, correlating them with a new definition of rock condition:

- Solid rock zone (SRZ): intact or massive rock.
- Slight perturbation zone (SPZ): rock with small filled, continuous or discontinuous fractures.
- Fractured zone (FZ): fracture zone composed by several fractures together (filled fractures, open fractures, continuous fractures or discontinuous fractures) or small voids.
- Blocky zone (BZ): zone compose by a weakness area, a large fracture (open fracture, filled fracture) and/or a medium side void.
- Low competence zone: zone of heavily broken rock mass, voids and/or large fracture (open fracture, filled fracture).

Figure 6.6 compares the rock condition classes estimated from the processed fracturing index with the analyzed televiewer records for the six blastholes under study.

Figure 6.6. Comparison between of rock condition predictor televiewer records for blastholes of blast 1.

As can be seen, most of the important rock conditions (low competence zone, blocky zone and fractured zone) are predicted; however, as with Figure 6.4, not all disturbances found in the televiewer records are noticed during the drilling performance. Sometimes, isolated or narrow fractures (filled, open, continuous or discontinuous) are difficult to sense while drilling. The same situation occurs when a change in lithology does not include a significant variation in the rock properties.

6.6. Conclusions from the application of an in-house MWD system for quarry blasting

An in-house MWD system is presented here as a low-cost alternative, that can be retrofitted, to monitor and record information of any drilling system. The system has been developed within the European Union SLIM project as a tool to help small operations lacking state-of-the-art drills to monitor the performance of their drilling.

Of the parameters monitored, the value and variation of the feed pressure, the hammer pressure and the rotation pressure have been combined to define a fracturing index. This index shows peaks and fluctuations when the bit drills through a rock disturbance and can be used as an engineering tool for geotechnical rock property recognition.

A qualitative validation of this index has been done, using photographic records of the blastholes walls obtained with an optical televiewer. From the geotechnical analysis of these records, five different types of rock disturbances have been found: open fracture, continuous closed fracture, discontinuous fracture, filled fracture, weakness area, void and change of lithology.

The results point to a good visual correlation between the fracturing index and the geotechnical features found in the rock and the variations in the fracturing index are found to always correlate with rock disturbances. It is observed that the larger and the more complex the disturbance is, the higher and wider the fracturing index variation. However, since it is difficult to assign values of the fracturing index to a specific rock disturbance, the index signal per blasthole has been divided in zones based on abrupt changes of variability of its values. This simplifies the fracturing index logs and helps their interpretation as to rock characterization. Preliminary results so far are promising and will be validated with a more extensive test program during the second campaign of full scale tests in El Aljibe quarry.

Chapter 7. GENERAL CONCLUSIONS

This thesis compiles and analyzes drill monitoring data for a total of 1,285 blasts and more than 71,000 production blastholes, recorded from three tunneling works, two orebodies of an underground mine and a quarry. Data for eleven drilling machines were analyzed; this comprises: three jumbos XF3C, two jumbos XL3C, one jumbo E2C for tunneling, four Atlas Copco SIMBA W6C drill rigs for sublevel caving and one Sandvik Tamrock DX-800 drilling machine for quarrying.

The assessment carried out of the drill monitoring systems for different drilling machines aims at guiding drilling and blasting towards more efficient mining operations and civil works. For this, a detailed analysis of the drilling system and the drilling control including how monitored parameters relate to each other and to the rock mass conditions, has been performed. A multi-step transformation of the MWD parameters has been carried out to remove external influences in the data in order to highlight the response of the rig to rock mass properties. The correct management of this data can provide an accurate and detailed characterization of the rock mass properties. Drilling indices tailored for different mining environments, machines and end purposes have been defined, that should help to overcome difficulties in the early stages of the production chain (drilling, charging and blasting), hence improving production. The indices developed are: (i) a drilling rock index to predict potential zones with over-excavation by blasting effect in tunneling; (ii) an index to predict problems during the charging of long blastholes in underground mining; (iii) an in-house MWD system to monitor the information of any rig while drilling and, from the records, a drilling index to characterize ground conditions in quarrying.

The results obtained through this investigation lead to the following conclusions:

- Tunneling

The mutual relations between Measurement While Drilling (MWD) parameters shows that the feed pressure is the parameter that leads the adjustment of the other parameters during rotary-percussive drilling. According to the results, the feed pressure shows, for semi-automatic drilling mode, a significant influence on the penetration rate, hammer pressure, damp pressure and rotation pressure parameters. However, rotation speed, water pressure and water flow are not significantly correlated with any other parameter, and hence not influenced by the control system. This suggests that changes in these

parameters may be affected by variations in the rock mass. The analysis also shows a different behavior between the parameters when drilling in semi-automatic or manual modes. This is a key point to consider in any drilling analysis since manual modes are always influenced by the operator. The only way to objectively assess the response of the drilling to the rock condition is through semi-automatic or automatic drilling modes.

The quality of the drilling in underground blasting operations has been investigated with the aim to quantify the error of the monitoring system with respect to the actual end position of the blasthole logged. The results point out a mean absolute deviation error with respect to the deepest location measured by the probe of 0.07 ± 0.03 m/m and 0.05 ± 0.02 m/m (mean and standard deviation) in the vertical and horizontal planes, respectively. The mean absolute error in the 3-D is 0.08 ± 0.03 m/m. This highlights the importance of knowing the actual trajectory of the blasthole before charging in order to apply correction measures and to prevent undesired variations of charge concentration at depth resulting in poor or excessive rock breakage and under- or over-excavation. The comparison between MWD parameters and deviation measurements show that drilling deviations are highly influenced by the rock structure. Disturbance zones in the rock detected by peaks and drops in some parameters are highly correlated with changes in the blasthole trajectory and changes in the magnitude of these parameters may also have influence in the phenomena; and increase in the force of the drill rod against the rock have been found to increase the probability of deviations during the drill

The damage to the remaining rock mass from tunnel blasting has been analyzed with the help of MWD records and with the purpose of developing a drilling or a rock index that can be used to predict potential zones of over and under excavated areas in the void created after blasting. For that, a new methodology based on the comparison of scanner profiles of the excavated sections with the position of the contour blastholes, has been developed to obtain the excavated mean distance (EMD). A multi-step transformation of the MWD parameters has been carried out to remove external influences, resulting in a set of transformed parameters that are more rock dependent and less influenced by systematic variations due to the nature of the drilling process. This comprises: (i) filtering out of outliers, (ii) removal of the ramp-up section of the logs, (iii) correction of systematic variations in the MWD parameters so that the effect of hole length and the influence of feed pressure are removed and (iv) normalization with the standard deviation to account for fluctuations in the signals.

A non-linear power-form model that predicts the excavated mean distance as function of the transformed penetration rate, rotation speed, rotation pressure and water flow parameters, and the lookout distance, has been developed. These parameters combine the rotational, hydraulic flushing and rate of advance of the drilling, and the confinement of the rock mass by depth. The model has a determination coefficient of 0.73, with the coefficients of the model being statistically significant. Residuals are essentially normally distributed. The signs of the exponents indicate that the transformed penetration rate, hammer pressure, rotation pressure and the lookout distance inversely influence the excavated distance i.e. high values for them reflect hard, unaltered rock. On the other hand, transformed rotation speed and water flow are directly correlated with the excavated distance, so that high values for them indicate soft, fractured rock

- Underground Mining

In the underground mine of Mamberget, Sweden, a geotechnical rock condition block model and a predicting risk of blasthole collapse model has been constructed, based on MWD records, aiming at minimizing problems during the charging of holes that are closely related with a proper rock fragmentation after blasting.

The analysis carried out by Ghosh et al. (2018) has been used as starting point of this work to classify the geotechnical rock condition in five classes: solid rock, fractured rock, cave-in rock, minor cavity and major cavity. Two applications have been developed: one for geotechnical rock condition of orebodies and the other for predicting the risk of collapse in blastholes. Ghosh et al. (2018)'s work has been improved into a rock condition block model to simplify the quantitative assessment and automatic recognition of trends in the rock condition. For that, a multi-step transformation of the MWD parameters has been applied to minimize external influences different than the rock mass. This comprises: (i) filtering of outliers, (ii) removing of systematic peaks due to the addition of a new rod, and (iii) correction of the hole depth influence.

The risk of blasthole collapse model has been developed by comparing different situations of rock classes from the geotechnical block model with the charging length of 102 production fan-holes. The model considers the existence and length of solid rock, fracture rock, cave-in and cavities along the hole. The assessment of the number of blocked and non-blocked fan-holes and the charging length/blasthole length ratio has been used to assign high, medium or low risk of collapse to each situation. The results outline a higher probability of collapse in the first half of the fan-holes for the high-risk rock class,

collapses in the second half of the fan-hole for the medium risk class and no collapses along the hole for the non-risk class blastholes.

The two models have been applied to full scale conditions for two orebodies in the Malmberget mine, Sweden, which comprise 20 drifts and 5060 fan shaped long-holes. Considering that, presently, the charging procedure is carried out with no prior information of the rock mass conditions, the two models presented here provide an early information of the rock mass conditions around the blastholes and the prediction of expected charging problems before starting the charging and blasting operation. This could be used as a decision-making tool in the daily routine for production planning and to minimize unexpected problems during the charging of the fan-holes. These features have the potential of improving rock fragmentation, with the final purpose of optimizing production.

- Quarry

An in-house MWD system is presented for quarry operations as a low-cost alternative to monitor and record the information of any drilling system. Of the parameters monitored, the feed pressure, the hammer pressure and the rotation pressure have been combined, considering both their variation and their values, to define a fracturing index to be used as an engineering tool for geotechnical rock recognition. A validation of this index has been done qualitatively with photographic records of the blastholes walls carried out with an optical televiewer. From the geotechnical analysis of these records, five different types of rock disturbances have been found: open fractures, continuous closed fractures, discontinuous fractures, filled fractures, weakness areas, voids and changes of lithology.

The results point out a good visual correlation between the fracturing index and the geotechnical disturbances found in the rock; variations in the fracturing index are always related with rock disturbances. In addition, it is observed that the larger and the more complex the disturbance, the higher and wider the fracturing index variation. This index may be used for blast design, e.g. using lower density explosives, or decked charges, in high fractured block zones determined from the analysis of drilling parameters.

Chapter 8. FUTURE WORK

This thesis has assessed the measure while drilling information to guide blasting in three different environments (tunneling, underground mining and quarrying). Although the findings are limited to three case studies, the methodology described can be used to guide the operations of any mine or civil work project developed by drilling and blasting.

Potential fields for further research include, but are not limited to, the following:

- For tunneling works:
 - Better understanding of drilling parameters and their effect on drillhole deviations.
 - Development of systems to monitor hole deviations with limited disruption of production.
 - The results obtained for the rock mass over-excavation prediction model are encouraging and could probably be upgraded to develop a blast design model that determines the optimum linear explosive charge per hole and, from it, to predict the amount of primary support required to fulfill the requirements of the contour design.
- For underground mining production with fan shaped blasting rounds.
 - Improvement of the risk of collapse model to specify potential zones where the blasthole can be blocked based on the geotechnical block model.
 - New campaign in the mine to gather more charging length data to validate the already developed risk of collapse model.
 - Application of the risk of collapse model in the daily routine of production to improve chargeability.
- For quarry operations
 - Quantitative validation of the developed fracture index from new data gathered in the present quarry.
 - Development of a spacing-between-fractures index and correlation with the spacings observed in the televiewer records.
 - Development of a blast design model that determines the optimum linear explosive charge per hole.
 - Extend the procedure used in tunneling to predict over-excavation from bench blasting.

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APPENDIX 1. Figures of drilling control system analysis

Figure A.1.1. Representation of the mean correlograms; each graph shows the 76 mean correlograms for working condition 2a; only values at ± 10 lags are given in order to better show the main peak.

Figure A.1.2. Histograms of standard deviation at positive and negative lags for each of the correlograms of the 76 blasts drilled in working condition 2a. Black histograms are the standard deviation on negative lags and white histograms on positive.

Figure A.1.3. Representation of the mean correlograms; each graph shows the 90 mean correlograms for working condition 2b; only values at ± 10 lags are given in order to better show the main peak.

Figure A.1.4. Histograms of standard deviation at positive and negative lags for each of the correlograms of the 90 blasts drilled in working condition 2b. Black histograms are the standard deviation on negative lags and white histograms on positive.

Figure A.1.5. Representation of the mean correlograms; each graph shows the 29 mean correlograms for working condition 3; only values at ± 10 lags are given in order to better show the main peak.

Figure A.1.6. Histograms of standard deviation at positive and negative lags for each of the correlograms of the 29 blasts drilled in working condition 3. Black histograms are the standard deviation on negative lags and white histograms on positive.

Figure A.1.7. Representation of the mean correlograms; each graph shows the 95 mean correlograms for working condition 4a; only values at ± 10 lags are given in order to better show the main peak.

Figure A.1.8. Histograms of standard deviation at positive and negative lags for each of the correlograms of the 95 blasts drilled in working condition 4a. Black histograms are the standard deviation on negative lags and white histograms on positive.

Figure A.1.9. Representation of the mean correlograms; each graph shows the 67 mean correlograms for working condition 4b; only values at ± 10 lags are given in order to better show the main peak.

Figure A.1.10. Histograms of standard deviation at positive and negative lags for each of the correlograms of the 67 blasts drilled in working condition 4b. Black histograms are the standard deviation on negative lags and white histograms on positive.

Figure A.1.11. Representation of the mean correlograms; each graph shows the 48 mean correlograms for working condition 5a; only values at ± 10 lags are given in order to better show the main peak.

Figure A.1.12. Histograms of standard deviation at positive and negative lags for each of the correlograms of the 48 blasts drilled in working condition 5a. Black histograms are the standard deviation on negative lags and white histograms on positive.

Figure A.1.13. Representation of the mean correlograms; each graph shows the 15 mean correlograms for working condition 5b; only values at ± 10 lags are given in order to better show the main peak.

Figure A.1.14. Histograms of standard deviation at positive and negative lags for each of the correlograms of the 15 blasts drilled in working condition 5b. Black histograms are the standard deviation on negative lags and white histograms on positive.

Figure A.1.15. Representation of the mean correlograms; each graph shows the 21 mean correlograms for working condition 5c; only values at ± 10 lags are given in order to better show the main peak.

Figure A.1.16. Histograms of standard deviation at positive and negative lags for each of the correlograms of the 21 blasts drilled in working condition 5c. Black histograms are the standard deviation on negative lags and white histograms on positive.

Figure A.1.17. Representation of the mean correlograms; each graph shows the 49 mean correlograms for working condition 6a; only values at ± 10 lags are given in order to better show the main peak.

Figure A.1.18. Histograms of standard deviation at positive and negative lags for each of the correlograms of the 49 blasts drilled in working condition 6a. Black histograms are the standard deviation on negative lags and white histograms on positive.

Figure A.1.19. Representation of the mean correlograms; each graph shows the 37 mean correlograms for working condition 6b; only values at ± 10 lags are given in order to better show the main peak.

Figure A.1.20. Histograms of standard deviation at positive and negative lags for each of the correlograms of the 37 blast drilled in working condition 6b. Black histograms are the standard deviation on negative lags and white histograms on positive.

APPENDIX 2. Geotechnical reports for tunneling operations

Bekkelaget Treatment Plant Extension (UBRA) Elemento: T090 Q - System: evaluación y medidas de refuerzo y/o soporte.		
Voladura	PK	383	Nº hojas	1 de 2
FECHA				
Mapeado				
FECHA	23/06/2015	HORA Inicial	17:30	HORA Final
			17:50	Autor: MMR
			OSSA:	

RQD= 100

Jn = 9

Jr = 3

Ja = 2

Jw = 1

SRF = 1

$$Q = \frac{RQD}{J_n} \times \frac{J_r}{J_a} \times \frac{J_w}{SRF}$$

ESR = 1

Valor Q básico 16,7

Valor Q Ajustado

soporte hastiales 83,3333333

MEDIDAS DE REFUERZO y SOPORTE

	Clase	Espesor (cm)
Gunita	E=1000J	
	E=700J	8
	E=500J	

	Tipo	c/c (m) (1)	L (m)	Dia (mm)	No.
Bulones	K	2,5	3	20	7

⁽¹⁾ Usar SB "spot" para bulones aislados

	c/c (m)	Armado S/D ⁽²⁾	Dia. Rebar	Código RRS (NGI ⁽³⁾)
Arcos armados				

⁽²⁾ S= Una capa rebar.
D= Doble rebar.

⁽³⁾ NGI - Norwegian Geotechnical Institute

OBSERVACIONES

Gunitado no debe extenderse hasta el suelo, dejar 1,5-3m visible desde la base.

LITOLOGIA

Gneis milonítico (Berkseje)	
Meta Tonalita	
Gneis bandeado (Vekslende)	

DISCONTINUIDADES

	Set	Tipo	Buz. Y Dir. Buza.
1er Set	S1	Frac.	56/246
2º Set	S2	Frac.	84/320; 90/320
3º Set	S3	Frac.	56/340
4º Set	S4	Dispersa	80/080

Figure A.2.1. Geotechnical reports for tunneling operations (part 1)

Elemento: T090 Q - System: evaluación y medidas de refuerzo y/o soporte.	
Voladura FECHA: 00/01/1900	PK: 383	Nº hojas: 2 de 2	
Mapeado FECHA: 23/06/2015 HORA: 17:30 HORA Final: 17:50		Autor: MMR Checker OSSA:	

Observaciones

Gneis gris oscuro verdos se hace muy quebradizo, se diría granular, aunque las partciulas son tabulares- El gneis micaceous, en contra de observaciones anteriores, parece en comparación más competente.

En la base del gneis gris oscuro se encuentra aún la banda conteniendo fenocristo de granate.

La excavación está seca.

Foto frente tunel
 Mirando hacia (rumbo):
 PK: 383

Croquis frente túnel

S1
S2
S3
Cuarcita

Figure A.2.2. Geotechnical reports for tunneling operations (part 2)

APPENDIX 3. Models proposed to predict EMD

Table A.3. 1 Models proposed to predict EMD

Trial 1	Figure A.3.1a	Equation	$\text{EMD} = A_0 \cdot \text{PR}_{c4}^{A1} \cdot \text{HP}_{c3}^{A2} \cdot \text{RS}_{c4}^{A3} \cdot \text{RP}_{c4}^{A4} \cdot \text{WF}_{c1}^{A5} \cdot \text{WP}_{c2}^{A6}$						
		R ²	0.59						
		Coefficient	A ₀	A ₁	A ₂	A ₃	A ₄	A ₅	A ₆
			3.757·10 ⁻⁹	-0.5197	1.3698	0.4340	-0.3004	3.7939	-2.3068
Trial 2	Figure A.3.1b	p-value	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵
		Equation	$\text{EMD} = A_0 + \left(\frac{\text{PR}_{c4}}{13.01}\right)^{A1} \cdot \left(\frac{\text{HP}_{c3}}{211.8}\right)^{A2} \cdot \left(\frac{\text{RS}_{c4}}{34.73}\right)^{A3} \cdot \left(\frac{\text{RP}_{c4}}{26.55}\right)^{A4} \cdot \left(\frac{\text{WF}_{c1}}{81.99}\right)^{A5} \cdot \left(\frac{\text{WP}_{c2}}{16.43}\right)^{A6}$						
		R ²	0.60						
		Coefficient	A ₀	A ₁	A ₂	A ₃	A ₄	A ₅	A ₆
Trial 3	Figure A.3.1c		-0.8776	-0.1215	0.3871	0.0823	-0.0706	0.3257	-0.1256
		p-value	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵
		Equation	$\text{EMD}/L_{\text{dist}} = A_0 + \text{PR}_{c4}^{A1} \cdot \text{HP}_{c3}^{A2} \cdot \text{RS}_{c4}^{A3} \cdot \text{RP}_{c4}^{A4} \cdot \text{WF}_{c1}^{A5}$						
		R ²	0.43						
Trial 4	Figure A.3.1d	Coefficient	A ₀	A ₁	A ₂	A ₃	A ₄	A ₅	
			-0.9505	-	0.2142	0.3067	-0.6730	0.2652	
		p-value	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵	
		Equation	$\text{EMD} = A_0 + \text{PR}_{c4}^{A1} \cdot \text{HP}_{c3}^{A2} \cdot \text{RS}_{c4}^{A3} \cdot \text{RP}_{c4}^{A4} \cdot \text{WF}_{c1}^{A5} \cdot \text{WP}_{c2}^{A6} \cdot L_{\text{dist}}^{A7}$						
Trial 5	Figure A.3.1e	R ²	0.74						
		Coefficient	A ₀	A ₁	A ₂	A ₃	A ₄	A ₅	A ₆
			-2.8015	-0.0346	0.1197	0.0356	-0.0107	0.0957	-0.0259
		p-value	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵	0.04
Trial 6	Figure A.3.1f	Equation	$\text{EMD} = A_0 + \text{PR}_{c4}^{A1} \cdot \text{RS}_{c4}^{A2} \cdot \text{RP}_{c4}^{A3} \cdot \text{WF}_{c1}^{A4} \cdot L_{\text{dist}}^{A5}$						
		R ²	0.73						
		Coefficient	A ₀	A ₁	A ₂	A ₃	A ₄	A ₅	
			-1.6898	-0.0563	0.0585	-0.0158	0.1024	-0.096	
Trial 7	Figure A.3.1g	p-value	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵	
		Equation	$\text{EMD} = A_0 + \left(\frac{\text{PR}_{c4}}{13.01}\right)^{A1} \cdot \left(\frac{\text{RS}_{c4}}{34.73}\right)^{A2} \cdot \left(\frac{\text{RP}_{c4}}{26.55}\right)^{A3} \cdot \left(\frac{\text{WF}_{c1}}{81.99}\right)^{A4} \cdot \left(\frac{L_{\text{dist}}}{0.26}\right)^{A5}$						
		R ²	0.73						
		Coefficient	A ₀	A ₁	A ₂	A ₃	A ₄	A ₅	
Trial 8	Figure A.3.1h		-0.8966	-0.1001	0.0985	-0.0231	0.1891	-0.1598	
		p-value	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵	
		Equation	$\text{EMD} = A_0 \cdot \text{PR}_{c4}^{A1} \cdot \text{RS}_{c4}^{A2} \cdot \text{RP}_{c4}^{A3} \cdot \text{WF}_{c1}^{A4} \cdot (A_5 - L_{\text{dist}})^{A6}$						
		It does not work							
Trial 9	Figure A.3.1i	Equation	$\text{EMD} = A_0 + \text{PR}_{c4}^{A1} \cdot \text{RS}_{c4}^{A2} \cdot \text{RP}_{c4}^{A3} \cdot \text{WF}_{c1}^{A4} \cdot (A_5 - L_{\text{dist}})^{A6}$						
		It does not work							
		Equation	$\text{EMD} = A_0 + \text{PR}_{c4}^{A1} \cdot \text{RS}_{c4}^{A2} \cdot \text{RP}_{c4}^{A3} \cdot \text{WF}_{c1}^{A4} \cdot (A_5 - L_{\text{dist}})$						
		R ²	0.70						
Trial 10	Figure A.3.1j	Coefficient	A ₀	A ₁	A ₂	A ₃	A ₄	A ₅	
			-102.75	-0.0512	0.0646	-0.0208	0.1167	101.44	
		p-value	0	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵	<10 ⁻⁵	0

Figure A.3. 1. Figure A.3.1. Representation of functions proposed in Table A.3.1. a) Trial 1, b) Trial 2, c) Trial 3, d) Trial 4, e) Trials 5 and 6, f) Trial 9.

APPENDIX 4. Figures MWD application in underground mining

Figure A.4.2. Analysis for Orebody 1, ring 1: a) Classification based on a ranking from first principal components interpretation (Ghosh et al.'s, 2018 methodology); b) Rock condition block model developed here (see section 5.3); c) Risk of collapse planning model developed here (see section 5.4).

Figure A.4.3. Analysis for Orebody 1, ring 2: a) Classification based on a ranking from first principal components interpretation (Ghosh et al.'s, 2018 methodology); b) Rock condition block model developed here (see section 5.3); c) Risk of collapse planning model developed here (see section 5.4).

Figure A.4.4. Analysis for Orebody 1, ring 3: a) Classification based on a ranking from first principal components interpretation (Ghosh et al.'s, 2018 methodology); b) Rock condition block model developed here (see section 5.3); c) Risk of collapse planning model developed here (see section 5.4).

Figure A.4.5. Analysis for Orebody 1, ring 5: a) Classification based on a ranking from first principal components interpretation (Ghosh et al.'s, 2018 methodology); b) Rock condition block model developed here (see section 5.3); c) Risk of collapse planning model developed here (see section 5.4).

Figure A.4.6. Analysis for Orebody 1, ring 6: a) Classification based on a ranking from first principal components interpretation (Ghosh et al.'s, 2018 methodology); b) Rock condition block model developed here (see section 5.3); c) Risk of collapse planning model developed here (see section 5.4).

Figure A.4.7. Analysis for Orebody 2, ring 1: a) Classification based on a ranking from first principal components interpretation (Ghosh et al.'s, 2018 methodology); b) Rock condition block model developed here (see section 5.3); c) Risk of collapse planning model developed here (see section 5.4).

Figure A.4.10. Analysis for Orebody 5, ring 1: a) Classification based on a ranking from first principal components interpretation (Ghosh et al.'s, 2018 methodology); b) Rock condition block model developed here (see section 5.3); c) Risk of collapse planning model developed here (see section 5.4).

APPENDIX A. Paper A

MWD parameters and Drilling Control system

Navarro, J., Sanchidrián, J.A., Segarra, P., Castedo, R., López, L.M., Cienfuegos, R., 2017. MWD parameters and Drilling Control system. Proceedings World Tunnel Congress 2017, Bergen, Norway, 9–15 June 2017, pp 194–20

MWD parameters and Drilling Control system.

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ABSTRACT: Drill monitoring systems use sensors installed along the drill string to provide digital signals that measure the performance of the drilling machine. Drill parameters are automatically adjusted by a control system in order to optimize the operation. Understanding their mutual relations and how the drilling machine controls them is required for a proper rock mass characterization. This paper aims to analyze the relationship between Measuring While Drilling (MWD) parameters and the influence of rock mass condition in their response. Auto-correlation and cross-correlation studies of MWD from four drilling machines, working on two different rock types, have been carried out. From the results, a complete interpretation of the behavior of the drilling machine control system is obtained. From it, the feed pressure seems to be the preset parameter used for the adjustment of the control system and thus, the one influenced by variations in the rock mass.

1 INTRODUCTION

Although the Measurement While Drilling (MWD) technique has become a complementary tool used for geotechnical ground recognition, there is still limited information related to the drill monitoring operation that may lead to a misinterpretation of the MWD data.

In modern drilling machines, the drill rig is equipped with the ABC system (Advanced Boom Control system), which eases the operator to follow the predesigned drill pattern. The ABC system is available in two levels of automation: regular and total (Nord & Appelgren, 2001, Östberg, 2013). In the regular system, the operator is authorized to follow a predesigned drill plan, by controlling boom and feeder manually and drilling is automatic. The total system allows the operator to switch the positioning and drilling between manually, semi-automatic and fully automatic (Atlas Copco, 2010). In the ABC total manual system, the operator moves the booms manually and takes control of the drilling process; in the semi-automatic system, the operator also controls the booms manually but the drilling process is done automatically; with the fully-automatic system,

both the positioning and the drilling are held automatically.

Drill monitoring systems use sensors installed along the drill string to provide digital signals that measure the parameters applied by the drilling machine. The drill parameters are adjusted by a control system in order to avoid drilling problems, optimize the operation (Schunnesson, 1998, Schunnesson *et al.*, 2011) and reduce bit wear (Schunnesson, 1997). In this adjustment process, there are one (or few) parameters that govern the control system and drive changes to the other parameters. Schunnesson (1998) claimed that the rock-dependent drilling parameter in percussive drilling was a normalized torque pressure. Schunnesson *et al.* (2011) also explained the rock-independent drilling parameters, such as percussive pressure, feed force and rotation speed that are governed by the control system. On the other hand, Peng *et al.* (2005) described a data control unit installed in the drilling system that allows to preset threshold values on the MWD parameters (penetration rate, rotation rate and feed pressure) as inputs for the control of the drilling. This matter is investigated in detail by this paper for the case of percussive-

rotary drilling, commonly used nowadays by underground drilling jumbos.

This paper aims to understand the mutual relations between the MWD parameters and the procedure follow by the drilling rig to control them, which is required for a proper rock mass characterization. Empirical data from four drilling machines working on two different rock types have been used for a back analysis of the actual drilling control operational system.

Two statistical models, auto-correlation and cross-correlation, are used to analyze the relationship between the eight MWD parameters, with the ultimate goal of defining the master MWD parameter to be used for rock mass characterization.

2 DESCRIPTION OF THE STATISTICAL MODELS

The MWD parameters are recorded as a discrete set of values, where observations are made at equal depth intervals. MWD signals monitored may be considered as a discrete time series (where time is equivalent to depth), by representing them over a ‘period’ of depth. Since the MWD parameters show the evolution of the control system represented by random variables indexed in depth, a stochastic process may be used to determine the correlation between the series monitored (Box & Jenkins, 1976).

2.1 Cross-Correlation

The cross-correlation function is a measure of the statistical degree to which two series are correlated. In order to understand the cross-correlation, it is helpful to explain the definition of the cross-covariance function. Considering N pairs of observations in two time series, x_t and y_t , the sample cross-covariance is given by (Box & Jenkins, 1976):

$$C_{xy}(k) = \begin{cases} \frac{1}{N} \sum_{t=1}^{N-k} (x_t - \bar{x})(y_{t+k} - \bar{y}), & k \geq 0 \\ \frac{1}{N} \sum_{t=1}^{N-k} (x_{t+k} - \bar{x})(y_t - \bar{y}), & k \leq 0 \end{cases} \quad (1) \quad (2)$$

where k is the lag between signals and it is $[0, 1, \dots, N]$ for Equation 1 and $[0, -1, \dots, -N]$ for Equation 2; N is the maximum windows lag; $\bar{x}$ and $\bar{y}$ are the mean values of the x series and y series, respectively.

The cross correlation function $r_{xy}(k)$ is the cross covariance, $C_{xy}(k)$, scaled by the variances of the two series at $k=0$:

$$r_{xy}(k) = \frac{C_{xy}(k)}{S_x S_y} \quad [k = 0, \pm 1, \pm 2, \dots, N] \quad (3)$$

where

$$S_x = \sqrt{C_{xx}(0)} \text{ and } S_y = \sqrt{C_{yy}(0)} \quad (4)$$

The cross-correlation function is asymmetrical with respect to zero and Equations 1 and 2 can be described in terms of “lead” and “lag” relationship. Equation 1 is referred to y_t shifted forward to x_t , in which the x_t series leads the y_t series or, equivalently, y_t lags x_t . Equation 2 describes the opposite situation and correlates the variables when y_t leads x_t or x_t lags y_t . Cross correlation results are given in the form of correlograms where cross-correlation values are plotted as function of the lag. Cross-correlation (r_{xy}) between two signals can take values from 0 to 1 for positive correlation and from 0 to -1 for negative correlation, where ± 1 values indicate maximum correlation and 0 no correlation. The maximum r_{xy} value describes the highest correlation between the two series and the lag position (k value) where this maximum takes place, the time delay between them.

Different behaviors between two time series may be identified from the result of the cross-correlation function. Not much information exists about the interpretation of the cross-correlation representation (hereinafter referred to as correlogram). Box & Jenkins (1976), Yevjevich (1972), Hamilton (1994) and Antoniou (2006), among others, explain the cross correlation function but they barely provide an interpretation of the correlograms. In order to fill this gap, Figure 1 shows five correlograms representative of common relationship between two time series x and y . Each correlogram is divided in two correlation zones as function of the sign of the k value. For each zone the notation “X→Y” indicates the leadership direction ‘X leads Y’. Variables x and y from Equations 1 and 2 indicate this leadership on both negative and positive lags. The horizontal red lines mark the 95% confidence band of cross-correlation; values outside this band are deemed significant. The interpretation of the correlograms in Figure 1 are described.

Values inside the confidence band indicate no significant correlation between the series (Figure 1.a). Significant correlations extended over several positive and negative lags and maximum correlation at lag 0 may imply uncorrelated series and thus, a non-causal system (Figure 1.b), i.e., the response of the system does not depend on past or present values of the input.

Significant correlations or significant signal fluctuations (positive or negative) on negative lags (included the maximum correlation) and no significant correlations on positive lags, and vice versa, indicate a causal system between the two series. The leadership direction falls to the correlogram zone with significant correlations. The location of the maximum correlation indicates the lag between the two series (Figure 1.c).

Maximum correlation at lag 0 and significant correlations at only positive or negative lags, indicate causal system, i.e., the output depends on present and past input values with instantaneous response between input and output (Figure 1.d). The leadership direction corresponds to the correlogram zone with significant correlations. Maximum correlation only at lag 0 but no other significant correlation or signal fluctuation at positive or negative lag, indicates that the two series are correlated but none of the signals lead the other. This correlation shows an independent behavior between them and thus a non-causal system (Figure 1.e).

2.2 Auto-Correlation.

The auto-correlation is the correlation between members of the same time series arranged in time. It is defined by:

$$r_x(k) = \frac{C_{xx}(k)}{S_x S_x} \quad [k = 0, \pm 1, \pm 2, \dots, N] \quad (5)$$

where,

$$C_{xx}(k) = \frac{1}{N} \sum_{t=1}^{N-k} (x_t - \bar{x})(x_{t+k} - \bar{x}) \quad (6)$$

There is abundant literature about the interpretation of the auto-correlation function (stationarity and periodicity of the signal) and the application of different regression models

Figure 1. Examples of Cross-correlations between x and y signals: a) No significant correlation between series; b) Uncorrelated series; c) Causal system with delay response between the two series; d) Causal system with instantaneous response; e) Two independent series (any signal leads into the other).

(ARIMA, ARMA...) for removing the noise from the signal according to the shape of the auto-correlogram (e.g. Box & Jenkins, 1976, Yevjevich, 1972, Hamilton, 1994 and Antoniou, 2006). However, this has not been followed here as any treatment of the signal noise might distort the correlation between the MWD parameters signals, so only detrend techniques have been applied (see Section 3.1).

Since this paper is focused on the relation between MWD parameters, the auto-correlation of the signal itself has no interest. However, it is calculated and shown together with the cross-correlation for completeness. Notwithstanding, further tests on the auto-correlation applied to MWD signals may be an interesting subject of research towards interpreting each MWD parameter.

3 DATA DESCRIPTION

3.1 Test sites

Drilling data from two different work constructions located in Bekkelaget and Sørkjosen (Norway) have been analyzed.

The underground extension work of Bekkelaget's municipal wastewater treatment plant, located in Oslo, Norway, is composed of five caverns and a main access drift of about 850 m length. The construction is carried out in competent rock mass, composed by gneiss with small tonalite and quartzite intrusions.

The work in Sørkjosen consists of a roadway tunnel construction of about 4,500 m length and 80 m² cross section. The excavation was done in competent rock mass, including sandstones, slates and expansive clays with chlorites.

3.2 MWD parameters

Three Atlas Copco XE3C Jumbos with three booms and one Atlas Copco E2C Jumbo with two booms have been used with ABC total system. Jumbos analyzed use a percussive-rotary drilling mechanism, which is based on the combination of three operations:

- Percussion: impact of the bit against the rock at the bottom of the hole.
- Rotation: rotation movement of the bit at the bottom of the hole when impacting the rock.
- Thrust: hydraulic force necessary to keep the bit in contact with the bottom of the drill hole.

Four Jumbo drills have been used in six cases shown in Table 1. Jumbo 2 worked on both manual and semi-automatic mode (cases 2a and 2b) and Jumbo 3 was used in two different rock types (cases 3a and 3b).

The MWD parameters recorded during percussive-rotary drilling for every single hole of the blast are described next (Schunnesson *et al.*, 2011); the acronym and the units for each parameters are given in brackets. All these

parameters have been recorded at equal depth intervals of 10 cm.

Table 1. Description of data characteristics of working conditions of Jumbos.

Jumbo & Case	Rock Type	ABC Total System	Jumbo Model	No. blasts
1	Gneiss	Semi-auto	XE3C	159
2a	Gneiss	Semi-auto	XE3C	76
2b	Gneiss	Manual	XE3C	90
3a	Gneiss	Semi-auto	XE3C	95
3b	Sandstone & Slates	Semi-auto	XE3C	67
4	Gneiss	Semi-auto	E2C	29

- Feed Pressure (FP, bar): it is a measure of the hydraulic pressure inside the cylinders necessary to keep the bit in contact with the bottom of the hole.

- Hammer Pressure or Percussive Pressure (HP, bar): measurement of the impact pressure of the bit against the rock mass.

- Damp Pressure (DP, bar): it measures the hydraulic pressure that the jumbo absorbs, in order to avoid vibration or undesired motion of the boom or drill string.

- Rotation speed (RS, rpm): it is defined as the number of turns of the bit per minute.

- Penetration Rate (PR, m/min): rate of penetration of the drill bit through the rock mass. It is influenced by the geo-mechanical response of the rock mass and, according to Schunnesson & Kristoffersson (2011), is also strongly influenced by independent parameters, such as feed pressure, percussive pressure and rotation speed.

- Rotation Pressure (RP, bar): it is defined as the pressure of the bit against the rock to maintain the required rotation. It is influenced by the rotation speed, the rock type and the design of the bit (Schunnesson & Kristoffersson, 2011, Peck, 1989).

- Water Pressure (WP, bar): water pressure necessary for removing drill cuttings from the borehole.

- Water Flow (WF, l/min): amount of water inflow into the borehole to remove detritus.

Traditionally, MWD parameters are grouped in two sets: independent parameters (HP, FP and RS), which are controlled by the operator or the control system, and dependent parameters (PR and RP) that depend on both independent

parameters and rock variations (Beattie, 2009, Hjelme, 2010, Schunnesson *et al.*, 2011, Schunnesson & Kristoffersson, 2011).

4 MWD DATA PROCESSING AND ANALYSIS

Calculations has been carried out for each hole individually. For the autocorrelation and cross-correlation analysis, associated with a stochastic process, the series should be time-invariant or, in the case under study, depth-invariant (Box *et al.*, 1994), i.e., its properties must not change with depth.

Penetration rate, damp pressure and rotation pressure are the parameters with greater dependence with hole depth as can be seen in Figure 2. This result is caused by an increase of the frictional resistance between the drill string and the walls of the borehole (Schunnesson, 1998). The hole depth effect is compensated by a normalization procedure in which first differences between adjacent elements of the signal values are calculated (Matlab, 2016a). Figure 3 represents the result of this procedure on the record of Figure 2.

The auto-correlations have been calculated for each series of the eight parameters, while the cross-correlation has been calculated for all pairs of series of the eight parameters.

Figure 4 shows a correlation matrix with the correlograms and auto-correlograms (main diagonal) of the MWD data obtained for a single hole. The leadership direction between variables is $X \rightarrow Y$ in the case of positive lags and $Y \rightarrow X$ for negative lags. The x variable from Equations 1 and 2 corresponds to the MWD parameters in the abscissa of the correlation matrix. The y variable corresponds to the parameters in the ordinate.

The length of each hole is between 3 and 5.5 meters; thus, the signal points registered per hole is normally greater than 30. The study has been done for a window of ± 10 values of k . This window size corresponds to a shift of 1 m, which is enough to assess relations between MWD data as changes and settings of the control system are almost simultaneous.

Calculations have been programed with Matlab (2016a). As can be observed in Figure 4, some of the correlograms show a peak, followed by positive and negative signal fluctuations. When representing several holes in the same plot, variations in the signals (e.g. the peaks that

Figure 2. MWD signals for hole 100 of blast from Kilometric point 350 with Bekkelaget with Jumbo 1.

Figure 3. Hole depth normalization of the MWD in Figure 2.

are so apparent in the single borehole plots, Figure 4) are blurred so that finding a pattern becomes extremely difficult. To solve this, clarify the results and find patterns that may explain the performance of the jumbo control system, a mean correlogram per blast has been obtained to reduce variations in the signal. Since correlograms are obtained over the same lag number, the mean value of every correlogram at each lag is calculated. The result is a correlogram per blast, which represents the mean correlation value at each lag. This procedure filters out the noise in the signal in a simple way.

As discussed in Section 2.1, significant correlations or significant signal fluctuations at positive or negative lags, where the maximum peak is located, indicate a causal system between the two series and thus, a clear input-output relation. The standard deviation of the

Figure 4. Cross-correlation matrix of the MWD parameters for a single hole.

Figure 5. Mean correlogram Jumbo 1.

Figure 6. Standard deviation analysis Jumbo 1

correlogram is calculated on each side in order to study on which side of the correlogram the signal fluctuation is higher.

Since the maximum correlation is always located within ± 4 lag interval, the standard deviation of the correlogram is calculated from lag +4 to +10 on one side, and from lag -4 to -10 on the other, in order to avoid the peak. The calculation is carried out for every hole and the distribution is represented by two histogram plots, one for the positive lags and the other for the negative.

The results for the six cases under study (see Table 1) are discussed below. For the sake of simplicity, MWD data from Jumbo 1, working in semi-automatic mode in gneiss, is shown here as an example. Figure 5 shows the mean correlograms per blast for the mean MWD parameters recorded by Jumbo 1 in 159 blasts. Figure 6 represents the histograms of the standard deviation of the correlograms; the red histogram gives standard deviations on positive lags and the green one on negative lags. In the same way that for Figure 4, the leadership direction between x and y variables is indicated by the MWD parameters in the abscissa and ordinate of the correlation matrix.

5 RESULTS AND DISCUSSION

By analyzing both the mean correlograms and the standard deviation of the signal variations, five different results (R) have been found; the correlogram type, from Figure 1, representative of each result is given next to the abbreviation used. Table 2 summarizes the results for the six working conditions under study.

- R1 (Figure 1.c): Significant maximum mean correlations concentrated at only one side of the correlogram. The histograms of the standard deviation may overlap or not. For the first case, no significant differences can be assessed, whereas for the second, significant differences on standard deviation occur and the histogram that spams in higher correlation values matches with the side of the correlogram where the maximum correlations are found. These results are identified, for instance, between PR – HP, PR – FP and HP – FP.

- R2 (Figure 1.d): Significant maximum mean correlations concentrated at lag 0 and significant differences between the histograms of the standard deviation (the histogram that spams in higher correlation values corresponds to the

leader parameter). These conditions indicate a significant input-output dependence (causal system) with an immediate response time; they occur between FP – DP, FP – RP.

- R3 (Figure 1.d): Significant maximum mean correlations concentrated at lag 0 with no significant differences between the histograms of the standard deviation (both histograms overlap). These conditions may indicate correlation between the parameters but show an independent behavior (non-causal system). Jumbo 1 does not show this result, but it is observed for Jumbo 3, between PR – RP (see Table 2).

- R4 (Figure 1.a): Maximum mean correlations below the confidence level. This result indicates a non-significant input-output dependence (non-causal system). Rotation speed (RS), water flow (WF) and water pressure (WP) show this condition when they are cross-correlated with the other parameters.

- R5: Significant maximum mean correlations concentrated in a specific lag, at only one side of the correlogram. Additionally, the histograms of the standard deviation show ranges significantly different, the histogram that spams in higher correlation values corresponds to the opposite side of the correlogram where the maximum correlations are concentrated. These conditions indicate contradictory results and therefore a non-significant input-output dependence (non-causal system). Relations between HP – DP and HP – RP fit this result.

Among the results, only R1 and R2 indicate a significant relation between MWD parameters and determine the performance of the control system. Cells with results R1, R2, R3, R4 and R5 are highlighted in Table 2 in red, green, blue, orange and magenta respectively. Cells for R1 and R2 show the leadership direction between parameters, the delay in the response (k) and the symbol of the correlation (r).

Cells for Jumbo 1 in Table 2 summarize the results of the analyses corresponding to Figures 5 and 6: penetration rate, hammer pressure, feed pressure, damp pressure and rotation pressure parameters show correlation. Among these relations, FP (input) leads PR and HP (outputs) with a lag of 1, i.e., there is a delay in the output response. It also leads DP and RP (outputs) with no delay, i.e., there is an immediate response between input and output. Because the sample interval resolution is 10 cm, data recorded is a mean value of the drilling parameters within this depth interval. A lag = 1 indicates a delay

between 10 – 20 cm, whereas when correlation shows lag = 0, the immediate response points out a delay between 0 – 10 cm. It is also observed that HP leads PR and that DP leads RP. However, these correlations may be a consequence of the leadership correlation induced by FP in these parameters, which outline that the control system for the Jumbo 1 adjusts the MWD parameters according to feed pressure.

As shown in Table 2, the performance of the control system for Jumbos 2 (case 2a), 3 (cases 3a and 3b) and 4, also working in semi-automatic, are in line with the results for Jumbo 1. FP (input) leads PR, HP, DP and RP (outputs). However, FP parameter for case 2a shows an immediate response over HP and PR, which may be related to the sample interval resolution. By and large, the results conclude that the control system adjusts the drilling according to the values of feed pressure. Similar results are also found in different rock types, suggesting that the rock is independent on the behavior of the control system.

The results from Jumbo 2 (case 2b) represent the behavior of the parameters when drilling in manual ABC total system, where it is found that the relation between MWD parameters is different. FP (input) parameter has a significant positive correlation over DP and RP (outputs) parameters with immediate response, and significant negative correlation with PR parameter with a lag of 1. In addition HP shows no significant correlation with the other parameters. The odd negative correlation between FP and PR may indicate a lower efficiency of the drilling operation, which highlights the importance of an automatic drilling and the role that the control system plays in it. Despite the different behavior of the control system performance for manual ABC total system, the feed pressure parameter is still the most influential MWD parameter.

6 CONCLUSIONS

An exhaustive analysis of the rig control system has been carried out with the purpose of understanding the mutual relations between MWD parameters in order to get some insight for a proper rock mass characterization.

For the study of the performance of the control system, two statistical models, auto-

Table 2. Summary of the relation between MWD parameters.

HP	Jumbo 1	R1: HP→PR k(1); r (+)						
	Jumbo 2a	R2: HP→PR k(0); r (+)						
	Jumbo 2b	R4						
	Jumbo 3a	R1: HP→PR k(1); r (+)						
	Jumbo 3b	R1: HP→PR k(1); r (+)						
	Jumbo 4	R1: HP→PR k(1); r (+)						
FP	Jumbo 1	R1: FP→PR k(1); r (+)	R1: FP→HP k(1); r (+)					
	Jumbo 2a	R2: FP→PR k(0); r (+)	R2: FP→HP k(0); r (+)					
	Jumbo 2b	R1: FP→PR k(1); r (-)	R4					
	Jumbo 3a	R1: FP→PR k(1); r (+)	R1: FP→HP k(1); r (+)					
	Jumbo 3b	R1: FP→PR k(1); r (+)	R1: FP→HP k(1); r (+)					
	Jumbo 4	R1: FP→PR k(1); r (+)	R1: FP→HP k(1); r (+)					
DP	Jumbo 1	R4	R5	R2: FP→DP k(0); r (+)				
	Jumbo 2a	R4	R5	R2: FP→DP k(0); r (+)				
	Jumbo 2b	R1: DP→PR k(1); r (-)	R4	R2: FP→DP k(0); r (+)				
	Jumbo 3a	R4	R5	R2: FP→DP k(0); r (+)				
	Jumbo 3b	R4	R5	R2: FP→DP k(0); r (+)				
	Jumbo 4	R4	R5	R2: FP→DP k(0); r (+)				
RS	Jumbo 1	R4	R4	R4	R4			
	Jumbo 2a	R4	R4	R4	R4			
	Jumbo 2b	R4	R4	R4	R4			
	Jumbo 3a	R4	R4	R4	R4			
	Jumbo 3b	R4	R4	R4	R4			
	Jumbo 4	R4	R4	R4	R4			
RP	Jumbo 1	R4	R5	R2: FP→RP k(0); r (+)	R2: DP→RP k(0); r (+)	R4		
	Jumbo 2a	R4	R5	R2: FP→RP k(0); r (+)	R2: DP→RP k(0); r (+)	R4		
	Jumbo 2b	R4	R4	R2: FP→RP k(0); r (+)	R4	R4		
	Jumbo 3a	R3	R5	R2: FP→RP k(0); r (+)	R2: DP→RP k(0); r (+)	R4		
	Jumbo 3b	R3	R5	R2: FP→RP k(0); r (+)	R2: DP→RP k(0); r (+)	R4		
	Jumbo 4	R4	R5	R2: FP→RP k(0); r (+)	R2: DP→RP k(0); r (+)	R4		
WF	Jumbo 1	R4	R4	R4	R4	R4	R4	
	Jumbo 2a	R4	R4	R4	R4	R4	R4	
	Jumbo 2b	R4	R4	R4	R4	R4	R4	
	Jumbo 3a	R4	R4	R4	R4	R4	R4	
	Jumbo 3b	R4	R4	R4	R4	R4	R4	
	Jumbo 4	R4	R4	R4	R4	R4	R4	
WP	Jumbo 1	R4	R4	R4	R4	R4	R4	R4
	Jumbo 2a	R4	R4	R4	R4	R4	R4	R4
	Jumbo 2b	R4	R4	R4	R4	R4	R4	R3
	Jumbo 3a	R4	R4	R4	R4	R4	R4	R4
	Jumbo 3b	R4	R4	R4	R4	R4	R4	R4
	Jumbo 4	R4	R4	R4	R4	R4	R4	R4
		PR	HP	FP	DP	RS	RP	WP

correlation and cross-correlation, have been used to analyze the relationship between the MWD parameters in order to find the master parameters that drive the automatic adjustment of the other parameters while drilling. The study has been carried out for four drilling machines and two different underground excavations with different rock mass conditions, totaling six analyses. MWD data for 516 blasts have been used.

From the results obtained, the following conclusions can be drawn:

- Penetration rate, hammer pressure, feed pressure, damp pressure and rotation pressure are the MWD parameters with significant mutual correlations.
- Rotation speed, water flow and water pressure parameters are not correlated with any parameter and thus, should be considered independent parameters.
- The analysis was carried out by using a constant sample interval of 0.1 m obtaining similar results. Although this interval is greater than the minimum possible resolution, it has been found to be sufficient to detect influences between parameters. Further analyses with different logging sample intervals are in progress in order to study whether the resolution of the data recording has an influence on the resulting relations between the MWD parameters.
- The feed pressure seems to be, for semi-automatic and manual drilling mode, the parameter that leads the rest of the parameters and thus, the one used by the control system for the automatic adjustment of the drilling.
- Analyses for manual drilling operation point out a negative relation between feed pressure and penetration rate, contrary to the semi-automatic drilling mode results. This may be an indication of a lower drilling efficiency in manual mode.
- The results for the six drills show that the rock type is independent of the behavior of the control system while drilling.
- Since changes in the rock mass condition primarily induce variations on the preset parameter (feed pressure), and these variations influence the values of the other parameters, such parameter is likely the one to be used for a characterization of the rock mass.

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APPENDIX B. Paper B

Assessment of contour profile quality in D&B tunneling

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Assessment of contour profile quality in D&B tunnelling

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ABSTRACT

Contour profile quality affects tunnel excavation costs, in terms of operational safety, support materials and construction time. In drill and blast tunnelling, under/over-excavation and rock mass damage arising from excavation phase can be evaluated by means of the elaboration of survey data and geophysical testing or coring, before and after the blast. As far as the quality of the profile is concerned, some indices can be used to define the contour and for the rock mass in the boundary as well.

This paper proposes a methodology well applicable to rock tunnelling, and a case study based analysis to correlate the over-excavation and the rock mass conditions is discussed to validate the procedure. Profiles and geological parameters have been processed with automatic code specifically developed for the study. Over-excavation distance and Tunnel Contour Quality Index are evaluated and compared with Q-system values. The results have been discussed, compared with other literature cases and validated for engineering applications.

1. Introduction

The quality of the excavated contour in underground tunnel directly affects final costs of the infrastructural facilities (Scoble et al., 1997; Hu et al., 2014). Poor contouring can produce under or over-excavation and artificial fractures into the rock mass. These factors produce many unfavourable consequences: scaling or specific supports are required, advancing rate decreases, convergences may increase, time schedule increases and safety is compromised. Directly related to the convergences and safety, also static approval tests are facilitated by a good contour profiling: in fact, both first phase lining and final lining are affected in terms of thickness, strength and durability (Pelizza et al., 2000a, 2000b).

Rock mass conditions are an essential factor in choosing the adequate excavation method (Mahdevvari et al., 2013); drill and blast (D&B) technique is the most appropriate in rock masses that present high compressive strength and that are abrasive (Cardu et al., 2004). Contour quality in D&B tunnelling depends on many factors: geological properties and conditions (e.g. rock mass quality and stress), blast design and drilling pattern execution (Oggeri and Ova, 2004; Singh and Xavier, 2005; Singh et al., 2003). Initial rock mass conditions depend on the site geology, but drilling operations and blasting round affect the rock mass structure because of vibrations, shock wave propagation, gas pressure and stress redistribution (Singh et al., 2003; Hu et al., 2014). These factors act on the rock mass depending on the microstructural fabric orientation (Nasseri et al., 2011) and pre-existing fractures.

Charge per delay and total charge per round must be adequately set to preserve rock mass integrity or avoid previous fractures worsening. Charge limit criteria cannot be based on the peak particle velocity (PPV) values as it happens for the man-made structures, because the limit charge is usually determined to control excessive vibration consequences at distance (Cardu et al., 2004). However, even if approximated from elastic media and pure compression waves, PPV relates the acoustic impedance with the stress level that the blast produces because of rock type, stress conditions, rock properties (i.e. density, porosity, anisotropy), water content and temperature (Singh et al., 2003). Blast sequence directly affects the extension of induced fractures; all blasting (contour, production, smooth) in each round produce a cumulative damage effect, both with smooth blasting or pre-splitting method. However, the two methods present some differences in the orientation and intensity of the damage that they generate. The smooth blasting produces both columnar shaped elements finely spaced and also widespread micro cracks; while in the pre-splitting the formation of columnar steep elements is more extended (Hu et al., 2014).

Taking into account the importance of the determination of rock damage and contour conditions after a D&B tunnelling, related to rock mass geology, geostructural features, drilling pattern and blasting sequence, this paper focuses on the assessment on the quality of the tunnel profile by means of some indices. This can be done using quick, easy to find and reliable profile survey techniques, properly adjusted and whose data can be processed to let a practical tool available for technical control and also to limit contractual disputes.

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2. Damage indices

Damage in rock mass means a drop of strength, caused by the opening or shearing of new or extended cracks and joints (Scoble et al., 1997). It can affect both underground and open pit excavations and it is related to the previous discontinuities conditions. The blast produces a direct damage around the blastholes and also an indirect damage due to vibration and rock block dislocation. Vibrations and explosive detonation products can propagate fractures into the rock mass and open existing joint, and this can induce an excavation disturbed zone (EDZ). This zone is the resulting volume around the tunnel boundary, whose extension depends on the excavation method, also valid for the extent of non-blasting methods (Barton, 2007), affected by damages due to excavation and disturbance due to stress state modification. Considering underground tunnelling, the damage can be generally divided in three classes:

- Major damage: when there is rock falling from tunnel roof and/or pillar.
- Minor damage: when there is chips detachment from tunnel roof and/or pillar.
- No damage: when there is not visual damage.

Various techniques can be used for the rock damage evaluation, some were developed for particular studies, and others are used during excavation routine (Scoble et al., 1997; Singh and Xavier, 2005):

- Assessing pre-blast: the inherent damage is evaluated, constructing a geomechanical classification (i.e. Bieniawski's classification) in order to build a base reference for post blast.
- Visual inspection and survey: provide qualitative information on pre/post blast damage and a rock mass classification. Also a borehole camera can be used for core assessment.
- Traditional observation methods: give an indirect measurement of damage. Usually the Half-Cast Factor (HCF) or scaling time is used.
- Rock mass classification methods: empirical rock mass quality rating systems (e.g. Q-system), inherent-damage index and blast-induced damage (e.g. Blast Damage Factor, Blasting Damage Index).
- Geophysical methods: such as seismic tomography, loose rock detection sensors and ground-penetrating radar, high-frequency cross-hole seismic, seismic-refraction tomography.
- Vibration analysis: the damage in the near-field is evaluated from peak particle velocity (PPV) values and rock mass strength.

Four main indices are available for this evaluation: Blast Damage Factor, Blast Damage Index, Failure Approach Index and Tunnel Quality Index, that are briefly illustrated in the following sections. They do not describe the geometrical condition of the excavated contour, which depends on the comparison with the design profile, but they focus on the rock mass damage. During an underground excavation, each blast round is individually mapped, in order to evaluate or update the required support (Barton et al., 1995) and to modify, if necessary, drilling pattern and blast design.

The Q index has been the one used in this study because of the available data. Anyway, the others are presented here in order to provide a more complete overview on the available indices. These could be used in further work if the data collection will take their parameters into account.

2.1. Blast damage factor

The Blast Damage Factor D (Hoek et al., 2002; Hoek, 2012) is a parameter introduced in 2002 into the Hoek-Brown failure criterion. It estimates the global rock mass strength and the rock mass modulus. Its range falls between 0 (undisturbed rock mass) and 1 (highly disturbed rock mass). This parameter must be set only for the actual zone of

Fig. 1. Primolano tunnel (Italy). High quality of the tunnel contour, half blasthole clearly evident at ribs and crown. Suggested $D = 0$. (Courtesy Italesplosivi).

damage, not for the entire rock mass surrounding the excavation and the definition of this extension represents a meaningful assessment. Ideally, the volume between front and undisturbed rock mass can be divided into a number of layer with different values of D using numerical modelling, but usually a single D -value is set for practical reasons. The production blasting data help to determine the actual damaged volume; some outlines (Hoek et al., 2002) suggest the right D -value by giving a description of the rock mass and its appearance. Figs. 1–4 show some examples for D&B tunnelling (and also one example of mechanized underground excavation).

2.2. Blast damage index

Blast Damage Index (BDI – Eq. (1)) was developed by Yu and Vongpaisal (1996) for mining works. This relation takes into account the mechanics and the effects of wave propagation into the rock mass: the compression wave arrives at the free surface and is reflected as a tensile stress wave that causes the damage (Barton, 2007). They analysed how much mining work affects slope walls and roof stability. Cardu et al. (2004) used this index to assess rock slope induced damage along mountainsides, when the advancing face of a tunnel approaches the external slope.

$$BDI = \frac{IS}{DR} = \frac{VdC}{KT} \quad (1)$$

where

Fig. 2. Irregular tunnel contour after D&B; shotcrete for the first phase support is smoothing asperities and over excavation, but nominal profile is not obtained yet. Suggested $D = 0.7$. (Anonymous).

Fig. 3. Hydropower tunnel in Northern Italy. Local damage and irregular profile at rib is due to spalling in metamorphic rock mass and anisotropic state of stress, even if mechanised tunnelling with a full face open TBM has been adopted. Suggested $D = 0.7$. (Photo by C. Oggeri).

Fig. 4. Very irregular profile after D&B tunnelling due to rock joint pattern and poor contouring techniques. Suggested $D = 0.8$. (Photo by C. Oggeri).

- IS : induced stress.
- DR : damage resistance.
- V : vector sum of PPV (mm/s).
- d : specific gravity for rock mass (kg/m^3).
- C : compression wave velocity of rock mass (mm/s).
- K : site quality constant.
- T : dynamic tensile strength of rock mass (N/m^2).

Yu and Vongpaisal (1996) and Singh et al. (2003) assume the value of RMR (Rock Mass Rating) as site quality constant K , that is the most adopted.

Table 1 shows a comparison between BDI ranges. Mining works presents higher limits of BDI than mountainside places (Cardu et al., 2004) but in both cases varies from zero to one. Singh et al. (2003) recommended very different limits in a coal mine situation.

Table 1

Comparison of BDI s in mining works Yu and Vongpaisal (1996), mountainside cases (Cardu et al., 2004) and coal mine Singh et al. (2003).

BDI	Mining	Mountainside	Coal Mine
Absolutely safe	< 0.125	< 0.060	< 1
No noticeable damage/falls seldom	< 0.250	< 0.200	< 2
Serious problems	> 0.250	> 0.200	> 2

2.3. Failure approach index

Failure Approach Index (FAI) proposes a quantification of the rock mass damage when numerical simulations are used in tunnel support design (Xu et al., 2017). At the beginning it was developed for plastic behaviour but then it was improved (Xu et al., 2017) for an elastic-plastic model that takes into account the relation between stress and strain, the strength criterion and also the post-failure response, considering isotropic conditions. This index is proposed for interlayered rock (FAI_m) and bedding plane (FAI_b) in order to find the layered rock mass FAI, which is the maximum between those two.

2.4. Q-system

Tunnel Quality Index (Q-system) is a consolidated and suitable rock mass classification system developed by Barton (1974). It is extensively used in underground rock engineering application and it allows also some correlations to empirically estimation of rock mass properties.

There are some parameters that need careful evaluation in order to improve the accuracy of Q-system; among the others, joint orientation is related to tunnel axis orientation but it is not numerically ranked. In fact the numerical evaluation of this parameter would make the classification less general. Moreover, joints and their characteristics are often difficult to be correctly determined: they form complicated three-dimensional patterns in the crust, while surveys are made in surfaces (two-dimensional) or boreholes (one-dimensional) (Palmström, 2005).

3. Overbreak evaluation

Tunnel excavation quality depends also on the contour geometry. Overbreak or bad profiling directly affects construction costs: more supports are required to avoid that some rock falls and more concrete is necessary to fill up empty spaces in order to help covering layer installation (Scoble et al., 1997). Furthermore, the type and quantity of supports (preliminary and long term layer) affects static approval tests, both during construction stage and long term monitoring (Pelizza et al., 1999, 2000a, 2000b).

Some key indicators can be used:

1. Overbreak area (Mahtab et al., 1997; Mandal and Singh, 2009). It is the excavated section area that exceeds the design (or paid) tunnel section. It is evaluated on a percentage on the design section area.
2. Overbreak distances (Kim, 2009; Olsson, 2010). It is the distance between design and excavated contour.
3. Tunnel Contour Quality Index (Kim, 2009; Kim and Bruland, 2010, 2015). This index relates overbreak distances, contours ratio and longitudinal variation in each blasted round. It can also be evaluated for the entire tunnel.

3.1. Overbreak area indicator

The magnitude of overbreak can be defined as the difference between design and excavated sections. It allows to evaluate the volume of rock that exceeds the planned mucking. In order to consider comparable data, the overbreak area ($O_v \text{ area}$) is evaluated in percentage (Mahtab et al., 1997; Mandal and Singh, 2009) as the difference

between excavated (A_e) and design (A_d) tunnel section, normalized on the design section area (Eq. (2):

$$O_{vareu} = \frac{A_e - A_d}{A_d} \times 100 \quad (2)$$

Mandal and Singh (2009) also propose to divide the cross-section in three zones, in order to evaluate the impact of different stress path and blast design effects. This approach demonstrates that the crown is more affected by overbreak, due to its stress conditions, claiming for a particular attention on the drill plan and the blast design of this zone.

3.2. Overbreak distances and damage distances indicator

In construction manual guidelines and contractual claim, the overbreak is generally evaluated as the distance between design and excavated contour (Mahtab et al., 1997; Scoble et al., 1997; Kim and Moon, 2013; Konkan Railway Corporation, 2012). This approach allows to elaborate directly the topographic mapping data, which is more intuitive than the overbreak area approach. The admitted overbreak distance depends on the position of the section in the blasting round. In fact, the drilling look out angle makes the excavated contour bigger at the end of the round than at the beginning. The admissible overbreak distance can be evaluated as a mean of the distance at round beginning and at round end (Olsson, 2010 – Fig. 5).

The maximum overbreak distance (O_v) depends on each national legislation and special conditions can be arranged between the parts in the contract (Olsson, 2010; Konkan Railway Corporation, 2012). Scandinavian countries present similar values of the admissible overbreak distances. Table 2 shows a comparison of the excavation classes used in Sweden (Anlaggnings AMA) and Finland (InfraRyl) (Olsson, 2010).

Norwegian and Italian legislations come from the Swiss one (SN, 2004; NPRA, 2012). In these countries the overbreak (O_v) depends on the theoretical excavated area (A_d) using Equation (3) as shown in Fig. 6:

$$O_v = 0.07 \times \sqrt{A_d} \quad (3)$$

All the evaluations on overbreak distance consider it outside the design contour because no rock within the design profile is admissible (Mahtab et al., 1997; Olsson, 2010).

3.3. Tunnel contour quality index

This index was developed (Kim, 2009; Kim and Bruland, 2010,

Fig. 5. Scheme of plan view. Excavated contour compared with design contour all along one round (modified from: Olsson, 2010). The start cross section is smaller than the end one due to drilling lookout that is necessary to have enough operative space.

Table 2

Excavation classes of tolerance in Sweden and Finland (from: Olsson, 2010).

Excavation tolerance classes	Maximum admissible distance expressed as average of round beginning and round end [m]		
	AMA – Sweden	InfraRyl – Finland	
		Walls and crown	Floor
1 – Special class	0.30	< 0.20	< 0.20
2 – Normal class	0.35	< 0.40	< 0.60
3 – Tunnel access (first 10 m)	0.40	< 0.60	–
4 – Special cases		No demands	No demands

Fig. 6. Contractual profiles: the design profile presents a tolerance (light blue area) but the admissible over-excavation is measured from the design profile (not from the tolerance) and depends on the minimum between Eq. (3) and (0.4) m. Outside the maximum admissible contour the over-excavation costs relapse on the contractor but the over-excavation costs due to geological condition relapses on the client (green area) (modified from SN, 2004). (For interpretation of the references to colour in this figure legend, the reader is referred to the web version of this article.)

2015) in order to evaluate tunnel and rounds contour quality in D&B context. This index takes into account overbreak distances of each cross-section (O_v), contour roughness as ratio of contour length (RCL) and longitudinal overbreak variation (V_0), as shown in Fig. 7.

Eq. (4a) relates these parameters for the entire tunnel where more than five consecutive rounds are available, Eq. (4b) can be applied in each single round.

$$TCI_{tunnel} = \frac{C_r}{W_1 C_1 \overline{O_v} + W_2 C_2 \overline{RCL} + W_3 C_3 V_0} \quad (4a)$$

$$TCI_{round} = \frac{C_r}{W_1 C_1 O_{v_round} + W_2 C_2 RCL_{round}} \quad (4b)$$

where

- C_r : constant of adjustment.
- W_1 : importance of additional mucking.
- C_1 : overbreak correction factor.
- $\overline{O_v}$: average of the rounds overbreak distance [cm].
- W_2 : importance of additional shotcrete.
- C_2 : contour length correction factor.
- $\overline{RCL}$: average of the round contour ratio.
- W_3 : importance of longitudinal variation.
- C_3 : longitudinal overbreak correction factor.
- V_0 : longitudinal overbreak variation [cm], which is the round overbreak standard deviation.

Fig. 7. Different effects of the parameters that affect the TCI (modified from: Kim, 2009). Excessive overbreak directly affects muck volumes and final lining to reach the design contour; contour roughness influences lining and supports and can cause under-excavation; longitudinal variation affects the operations of lining placement.

The total overbreak is calculated with the following steps:

1. Ov_{O_v} : distances between excavated contour and design contour in many points of the same cross-section.
2. $Ov_{section}O_{v_section}$: average value of overbreak distances (in cm) of each scanned section.
3. $Ov_{round}O_{v_round}$: average value of $Ov_{section}O_{v_section}$ considering at least two sections in each round.
4. $Ov_{\overline{O_v}}$: average value of Ov_{round} to consider the entire tunnel.

Fig. 8 shows the practical way of the procedure.

Coefficients of adjustment have been calculated following the procedure well explained in Kim (2009). Eq. (5) gives an example of their structure showing the overbreak coefficient equation.

$$C_1 = 1 / \left[\left(\frac{1}{n}, \sum_1^n, O_{v_round} \right), +, 5, \times, std, (O_{v_round}) \right] \quad (5)$$

3.4. Profile survey

After rounds blasting, the geometry documentation is important both for owner and contractor, in order to evaluate excavation quality, excavated volumes and supports (Gikas, 2012). Contact (finger probes, tape extensometer and section profiler) and non-contact instruments (theodolite, total stations, photogrammetry, optical triangulation and Terrestrial Laser Scanning – TLS) permit data acquisition (Pejić, 2013).

The methods that are mainly used are photogrammetric techniques, Terrestrial Laser Scanner (TLS) or conventional survey with total

station (Olsson, 2010; Gikas, 2012).

The photogrammetric techniques can give a 3D scan of the tunnel tube collecting each surface point at least in two photographs. It is a quite low cost technique but is not common in underground works because the surface is irregular and there is not enough light for taking quality pictures.

The Terrestrial Laser Scanner (TLS) can rapidly locate points with high accuracy (e.g. thirty meters can be scanned in ten minutes) and it provides a point cloud. The presence of reflective objects (e.g. equipment and water) can affect the recognition of targets (Gikas, 2012). Data can be shown in a virtual reality model and, if texture information are available, it is possible to render a photorealistic VR model (Chmelina, 2010).

The total station needs a calibration and some starting parameters are set manually: profiles interval, measuring angle, beginning and ending chainages; then total station could reveal points automatically with iterations. The instrument should be located as near as possible to the symmetry axis, in order to equilibrate the density of the points on the contour; this surveying method took about one hour each ten meters of tunnel. This procedure is usually done after scaling and shotcreting for safety reasons. Also the total station survey can be affected by the presence of reflective objects (e.g. equipment and water) as the TLS can be.

A profile-image method can also be used in tunnel works (Wang et al., 2009, 2010), joining laser profiling with photogrammetry to obtain a more accurate survey.

Fig. 8. Scheme of the procedure that is necessary to calculate average overbreak, one of the main TCI parameters. The procedure to obtain rounds and tunnel RCL is almost the same.

Table 3
Basic data for the main joint sets.

Orientation of the main joint sets observed	
Joint set (S)	Dip/dip direction (°)
S1	45–70/235–255
S2	60–80/010–020
S3	40–65/095–120

4. Methods

4.1. Data gathering

The study case is a roadway tunnel excavated in one of the North provinces of Norway. The tunnel is 4585 m long with a face of about 80 m² of surface and lies under 300 m of overburden. The work was planned for a period of nine months (from August 2014 until April 2015). The tunnel was excavated by D&B using the Norwegian Tunnel Method of Tunnelling (NTNU, 1995) – NTM – that is the application of New Austrian Tunnelling Method – NATM-on hard rock. The construction was developed through competent metamorphic rock mass, composed by sandstones, slates and expansive clays with chlorites. The Q index, obtained from visual inspection of the tunnel face, was used as classification of the ground condition. The available data from geological site survey list 54 rounds located from kilometric point KP 5561.9 to 5781.8 (47 surveys) for Portal 1 and from KP 10127.5 to 10107.5 (7 surveys) for Portal 2 of the tunnel. Three main joint set families were observed along the rounds excavated (as from the geotechnical report). Table 3 describes the relative range of dip and dip direction of these main joint sets with respect the tunnel axis.

Two jumbos Atlas Copco XE3C and XE3D of three booms each, equipped with percussive-rotary top hammer drilling mechanism, working in semi-automatic ABC total system were used to drill the analysed blast rounds. Available data concern production face drilling holes of short length (4.0 – 5.5 m), drilled by using only one rod (5.5 m length and 38 mm diameter) and a bit of 46 mm diameter.

The charging of the blastholes was carried out with emulsion of different lineal charge according to the type of blasthole; nominal charging information estimate theoretical lineal charges of 1.6 kg/m, 1.2 kg/m, 0.85 kg/m and 0.5 kg/m for cut/lifter, stopping, second contour and contour holes, respectively. The nominal number of blastholes per round was about 140; this counted about 16 cut holes, 12 lifter holes, 57 stopping holes, 24 s contour holes and 32 contour holes. Stemming was estimated in 0.4 m for all blastholes with exception of contour holes that were not stemmed. The firing was bottom initiated with booster and non-electric detonators; nominal timing reports indicate the use of LP non-electric detonators from numbers 0 to 60. Round progress was 93% of the drilled length and production was estimated at 1.6 blasts per day; this is a progress of about 7.2 m per day. The excavation was made simultaneously from the two sides of the tunnel.

Portal 1 and 2 were located at 105 m a.s.l. and 6.75 m a.s.l. respectively. Starting from Portal 1, the tunnel was upwards oriented with a slope of 1.5% for about 617 m, followed by a downhill of a –2.5% slope until Portal 2. Tunnel cross section dimensions were decided considering the estimation of traffic volume twenty years after the opening (Annual Average Daily Traffic volume – AADT) and the tunnel length (NPRA, 2004); AADT was estimated between 7500 and 9500 units. In order to fulfil this traffic volume, most of the cross-sections follow the Norwegian type section T9.5 (theoretical excavated area 74 m²) apart a widening zone of the tunnel that follows the cross-section T12.5 zone (theoretical excavation area 100 m²), for a length of about 30 m. The face area of the two transition zones, before and after the widening, 30 m long each, was increased (or decreased) regularly until matching the respective cross-section, namely that of T9.5 and

Table 4
Geometric measurements for tunnel cross-section that was used in this case study (from: NPRA, 2004; Tunnel project documents).

Geometric measurement	Section T9.5	Section T12.5
Total width (B _T)	9.50 m	12.50 m
Carriage way width (B _K)	7.00 m	10.00 m
Lanes	2 × 3.50 m	3 × 3.50 m
Shoulder (in verge area)	2 × 1.00 m	2 × 1.00 m
Sidewalk (in verge area)	2 × 0.25 m	2 × 0.25 m
Centre point wall radius (X)	0.44 m	3.44 m
Centre height wall radius (Y _V)	1.57 m	1.57 m
Wall radius (R _V)	4.79 m	4.79 m
Centre height lining radius (Y _H)	1.22 m	– 0.46 m
Lining radius (R _H)	5.20 m	7.45 m
Vertical clearance	4.60 m	4.60 m
Nominal area	66.53 m ²	91.23 m ²
Concrete upholstery	0.60 m	0.60 m
Excavation area	74 m ²	100 m ²

T12.5.

Table 4 shows the measurements for the used cross sections, referred to Fig. 9. The design area starts to increase from chainage (KP) 5785 until KP 5815 to reach the T12.5 section between KP 5815 and KP 5845. Then it decreases until KP 5875.

Topographical mapping of the excavated void after blasting was surveyed with a total station Leica Viva on the shotcreted surface; set angle was chosen to reveal approximately one point each 50 cm on the contour. Typical thickness of the shotcrete liner lies between 80 and 100 mm.

Cross-section profiles perpendicular to the direction of the tunnel axis of the excavated face were extracted at every 1 m in AutoCAD files. Each profile was identified by its respective kilometric point; this comprises a total of about 500 excavated profiles measured.

4.2. Data analysis

The aim of the survey is to assess the quality of the resulting contour from the excavated profiles compared with the theoretical section intended. Since the actual lineal charge of the holes is not available, the explosive is considered as a constant variable. Therefore, differences in the excavation sections (over-excavation and under-excavation) should be mainly generated by a variation in the geotechnical rock characteristics. The work developed by Costamagna (2016) is applied here in order to evaluate round blast results in terms of overbreak and TCI. For this reason, a Matlab script has been developed in order to automatically make uniform and treat three kinds of data (Fig. 10):

- (1) Scanned profiles (about 500 .dxf files)
- (2) Geotechnical characterization (54 surveys)
- (3) Drilling data (measurements while drilling, about 11,700 .MWD files)

The data collected by means of the geostructural survey allowed to set the Q-system parameters. The other rock mass damage indices (i.e. BDF, BDI, FAI) could not be evaluated in this case study.

In order to evaluate round results and compute some of the parameters used to assess TCI, it is necessary to identify each round and the cross-sections that belong to each of them. The KP of a new round is measured topographically. In the case study, the KP was measured both manually, by using a total station (used to reference the geotechnical reports), and automatically through the MWD system. The drilling jumbo has a laser scanner installed in its front side. During the positioning of the jumbo and before the drilling of a new round starts, the jumbo is aligned with the tunnel axis, by making pass through two targets located in one of the boom a laser beam aimed in the direction of the tunnel axis. The laser scanner also measures the distance to the

Fig. 9. Geometry of tunnel cross-section T9.5 and T12.5 (NPRA, 2004).

Fig. 10. Analysed data as function of the chainage. The blue crosses show the scanned cross-sections, red circles show the beginning KP of the geological surveys and black triangles show the nominal position of start drilling sections where MWD data are collected. (For interpretation of the references to colour in this figure legend, the reader is referred to the web version of this article.)

face of the new round and records the kilometric point (here intended as the nominal KP) at which it is located inside the tunnel.

When MWD has been correctly measured and recorded, the mode of all z coordinates (borehole position along the longitudinal axis referenced to the nominal KP) in each drilled section is added to their nominal KP, in order to consider also the irregular face surfaces.

In case no MWD data is available for adjacent rounds, the KP taken from the geotechnical reports is used. KPs from the MWD system in which z coordinates records have failed for all boreholes and taken from the geotechnical reports may induce some error in the beginning of the round (due to irregularities in the face) and also occasional overlapping between two adjacent rounds (as for the 93% round progress). Since cross-sections are scanned at every 1 m depth, a correction for clustering excavated areas between two adjacent rounds has been carried out by adding a length of 0.5 m to the initial KP, in order to reduce these KP errors. In addition, rounds shorter than three meters have been rejected because at least three cross-sections in each round are necessary for the round analysis and the maximum length per round has been identified by using a robust variance estimator (Miller and Miller, 2010) and considering site work conditions. Therefore, rounds between 3 and 5.5 m lengths have been considered for the analysis; this comprises 84 available rounds. Scanned cross-sections within the kilometric points of two adjacent rounds of the MWD system will be framed in their respective round. Finally, the first and the last profile of each round have been discarded because their blasting results can be affected by the above errors commented (Fig. 11).

The overbreak has been evaluated based on distances and areas. The overbreak is defined in two ways:

- Over-excavation: it is the extra void outside the design contour line. It is evaluated as positive overbreak.
- Under-excavation: it is the void inside the design contour line. It is evaluated as negative overbreak.

This distinction is necessary for the correct evaluation of the overbreak. In fact, the over-excavation affects the shotcrete thickness, the rock support and the mucking; the under-excavation is not admitted in the contracts and it affects the scaling.

Fig. 12 resumes in four steps the data treatment and matching.

In this paper, cross-section contour have been analysed on the 65% upper part as shown in Fig. 13. This choice depends on the low quality of the floor and corner profile survey: these bad data corrupt the results

showing an unrealistic under-excavation, but the horizontal cut erases their contribution in almost every section.

5. Results and discussion

The automatic analysis mentioned in the previous section collects, selects and treats a huge number of data. This section presents the results validated in a case history focusing on the overbreak (both in area and distance evaluation), trying to correlate them with the Q-system classification. Based on available data and on Eqs. (4a) and (4b), TCI values are consequently processed. This index could allow time and costs reduction because it mainly uses profile scanning data that can be collected and analysed automatically. Anyway, to make its use easy and efficient, an extensive casuistry is required to build a TCI-value classification. Data reported in this study can improve the database and the evaluations required to build a related classification.

As said, all the evaluations on cross-section contour have been done on the 65% upper part (hereafter no more specified), in order to avoid survey inaccuracy. In fact, the survey was probably affected by the presence of muck left in place, disturbance due to ventilation system or water particles that reflected the laser ray in a wrong way (Gikas, 2012).

5.1. Tunnel quality indicator: overbreak

The beginning of the round corresponds to minimum peaks of the excavated area. In facts the end of each round must be larger than round beginning due to the drilling lookout. This trend is confirmed as shown in Fig. 14.

Most of the cross-sections do not present under-excavation, as it should be. Anyway some exceptions, especially section from KP 5570 to KP 5603 and from KP 10004 to KP 10013 present under-excavation due to errors during the contour scanning. However, the actual area is still bigger than the design one because the over-excavation compensates the under-excavation area. Anyway, the following results concern the over-excavation and omit any deeper consideration on under-excavation because, as said, it is not admissible and affects scaling costs.

The over-excavation is measured as distance during the surveys but the over-excavation additional costs depend on volume of additional muck and voids that needs to be shotcreted. These volumes can be calculated a posteriori from the over-excavation area of each cross-section. In this case study, the developed code measures over-excavation area (that correspond to a volume for unit of advance – 1 m) from polygons between design and actual contour but it can be correlated to the average over-excavation distance and the design contour length, as shown in Fig. 15. The two values are linearly correlated: 1.50 slope in a range of [1.45; 1.55], 0.20 intercept in a range of [–0.09; 0.50], 0.88 of R^2 value.

This relation allows to work only on over-excavation distance, as most of the literature does. However, in some cases (Mahtab, 1997; Mandal and Singh, 2009; Olsson, 2010) the over-excavation normalized area (as shown in Eq. (2)) is used. In this case study the average value of the over-excavated ratio is quite similar for the sections from Portal 1 (18.5%) and from Portal 2 (17.1%). Considering all section from both portals, the value is $18.1 \pm 7.7\%$ (mean and standard deviation), in a range between 1% and 46%. Values from literature are in a range between 7.3% up to 51.9% (Mahtab, 1997; Mandal and Singh, 2009; Olsson, 2010) and Olsson (2010) proposes an admissible overbreak ratio of 25% as upper limit.

According to the Norwegian regulation the admissible over-excavation (O_v) is the minimum value between 0.4 m and the value calculated as function of cross-section area (Eq. (3), Table 5). A third limit is interpolated and used to evaluate all the cross-section in which the area progressively increases/decreases due to switch between theoretical section T9.5 and T12.5. No references for these sections were found in literature.

Fig. 11. Round plan view. Each scanned profile (green line) is assigned to its round, using beginning and ending round KP. First and last profiles (orange lines) of each round were discarded because their blasting results can be affected by drilling operations or blast results. (For interpretation of the references to colour in this figure legend, the reader is referred to the web version of this article.)

Fig. 12. Flow chart of the first three step of the analysis. Step 1 elaborates data from profile survey, working from .dxf files and return a structure that contain, among others, statistical evaluations on overbreak (that are used after). Step 2 works on geological data (from .xls file) and evaluates Q system factors. Step 3 calibrates round beginning-end kilometeric point proceeding from drilling data (.MWD). Step 4 matches each scanned profile to its own round and calculate TCI. It also evaluates the correlation TCI – Q value and over-excavation – Q in each round.

Fig. 13. Section KP = 5799. The horizontal line is set at 35% of tunnel height to keep out the corner of the cross-section. All the data about over-excitation that are used for the results and discussion comes from the contour above the horizontal cut. In this way the most of profile scanning errors are not considered.

Fig. 16 shows the average value of the over-excitation distance for each scanned section and the threshold values. Average values of over-excitation of each cross-section from both Portals are considered to obtain average, maximum and minimum values for the entire tunnel (Table 6).

5.2. Correlation between geological and topographical survey

The main values for all the Q-system parameters suggest a fair-good rock (RQD) with an irregular and smooth undulating surface (J_r), usually two or three joint sets were surveyed plus some random joint (J_n) and the most of these joint were slightly altered (J_a); the rock mass is medium stressed (SFR). The surveyed rock mass is dry, and the value J_w is constantly set on 1. As summary, the rock mass can be considered very poor – poor – fair (classes IV, V, VI) quality all along the tunnel.

Under the assumption that every drill an blast cycle was done with the same technique and the same equipment, available geology data are compared with overbreak distances.

Fig. 17 shows Q progression along the tunnel axis and overlaps average over-excitation of each scanned profile.

Average over-excitation of each round is compared with the Q value trying to demonstrate a relation between Q index and excavation results. 18 rounds present enough data to be analysed in this case; no linear correlation was found between the average over-excitation distance and Q values. Table 7 shows the range of over-excitation distances for each Q classes. These ranges are similar for all the classes and they cannot be used to predict the blasting results in each round.

This lack of correlation could depend on shotcreting phase, as it modifies in sense of smoothing the contour profile. In fact, best rock mass quality should require less shotcrete (and the opposite on low

Fig. 14. (left) Areas of the 65% upper zone of the cross section surveyed from Portal 1. (right) Areas of the 65% upper zone of the cross section surveyed from Portal 2. Blue line shows the design area above the horizontal cut and interpolates the theoretical values in the increasing/decreasing segments. Black points show the beginning KP of the analysed rounds.

Fig. 15. Over-excavation area can be calculated as function of average overbreak distance (O_v) and design contour length (D_l), using the relation $O_v \text{ calculated area} = O_v \times D_l$. This relation allows to consider only distance values.

Table 5

Maximum admitted O_v depending on cross-section area and evaluation of case study available sections.

Theoretical cross-section	Area [m ²]	Maximum admissible O_v [m]	No. of sections	Compliant sections (% on their group)
T9.5	74	0.60	400	84
T12.5	100	0.70	28	79
Interpolation	[74; 100]	0.65	59	92

quality surfaces), thus over-excavation results are emphasised (or mitigated). But available data do not allow confirming this hypothesis, thus shotcreting thickness has been considered as a relatively constant value.

5.3. Tunnel quality indicator: Tunnel Contour Quality Index (TCI)

The Tunnel Contour Quality Index – TCI (3.3) can be evaluated for each round.

Table 6

Evaluation on overbreak distances.

Values on sections from Portal 1 and Portal 2	O_v
Average [m]	0.46
Standard deviation [m]	0.19
Maximum [m]	1.10
Minimum [m]	0.10

Eq. (4b) has been used, following the procedure well explained in Kim (2009) for the coefficient determination. Table 8 shows the values used in this case study to calculate TCI in each round and for the whole tunnel.

TCI obtained values vary from 40 up to 86; the two Portals do not present different ranges of this value, and they can be analysed together (Fig. 18).

The distribution of round TCI is shown in Fig. 19 according to the classes that were proposed in literature (Kim, 2009). The largest frequency is found between 62 and 65; the whole range is between 38 and 77. The case study presents good round TCI compared with literature

Fig. 16. (left) Average over-excavation distance in each section from Portal 1 compared with admissible values. (right) Average over-excavation distance in each section from Portal 2 compared with admissible values.

Fig. 17. (left) Over-excavation distance and Q of each scanned or surveyed section from Portal 1. (right) Over-excavation distance and Q of each scanned or surveyed section from Portal 2.

Table 7
Over-excavation ranges in Q classes.

	Over-excavation [m]			
	Min	Max	Mean	Std
$Q \leq 4$	0.35	1.50	0.75	0.36
$4 < Q \leq 10$	0.16	1.23	0.64	0.26
$10 < Q \leq 40$	0.42	0.94	0.64	0.20

Table 8
Values of TCI coefficients and weight.

Coefficient of adjustment	Weight ^a		
C_1^b	0.0049	W_1	4.5
C_2^b	0.6765	W_2	4.5
C_3^b	0.0206	W_3	1.0
C_r^a	300		

^a Values obtained by Kim (2009).

^b Values calculated by using this case data and following Eq. (5).

case studies (Table 9).

TCI can be evaluated on the whole tunnel if more than five rounds are available (Kim, 2009). Table 10 shows the values of the index parameters in the case study, comparing the resultant TCI with literature examples (Kim, 2009; Kim and Bruland, 2010; Kim and Bruland, 2015). The TCI value shows a good tunnel contour quality, quite near to the result in Misiryeong tunnel that was considered to have a very good excavation result.

6. Conclusions

In this paper, tunnel contour evaluation is obtained in terms of overbreak and related to rock mass conditions. Over-excavation affects timing and costs (i.e. mucking and concrete/shotcrete costs), because the additional excavated volume usually needs to be replaced by additional shotcreting and other reinforcing works. In order to achieve a reliable procedure, the quality of almost 500 m of a roadway tunnel has been evaluated. For this purpose, measurements from three different sources are automatically processed: topographical measurements of excavated contour, geological mapping of the rounds and Measuring

While Drilling (MWD) data.

According to the discussion, the work is related with the analysis of resulting profiles from D&B tunnelling and has provided the following main results:

1. A detailed analysis of large data sets is only possible by processing automatically the data. A code developed in Matlab environment has been created to quantify the overbreak caused by blast; it processes automatically MWD, geological and topographic data and stores them in the same numerical format. This code can be easily adapted to other case studies.
2. Results from the analysis are mainly focused on the characteristics of the excavated contour in comparison with the theoretical section, considering over-excavation in terms of area and distance separately and demonstrates that they are close linearly correlated. The study case exhibits an over-excavated distance of 0.46 ± 0.19 m (mean and standard deviation). This value is in general under the admissible Norwegian limit of 0.6 up to 0.7 m calculated on the theoretical area.
3. The quality of data in the study case allows to focus only on geological causes and Q index is adopted to describe rock mass conditions. Q index exhibit from a very poor to fair quality. No strong numerical correlation between Q and over-excavation has been found. This result probably suggests that drill operations and blast design influence are stronger on contour quality control.
4. Tunnel quality has been also evaluated using the engineering index TCI. In the case study TCI has a value of 60.5 ± 8.4 ; the largest frequency is found between 62 and 65; the whole range between 38 and 77. These values are relatively well with those for other tunnels in literature. As more than five rounds are available, the index can be evaluated for the entire tunnel: overall TCI value is 57.4 and shows a quite good quality of the excavation (if compared with the studies of the index developer).

Further works based on the application of TCI to other case studies could enlarge the records and validate the efficiency of the index itself. The impact of shotcrete thickness should be deeply evaluated when the survey is done after shotcreting (as in this case) and more parameters should be recorded during the blast (e.g. PPV) in order to calculate other damage indices (e.g. BDI) and find out the most representative in terms of correlation with over-excavation. In the field of engineering application, the TCI index can provide a tool to evaluate tunnel contour

Fig. 18. TCI in each round from Portal 1 and Portal 2. The most of the values are between 45 and 75 in both Portals.

Fig. 19. Distribution and cumulative distribution of round TCI based on case study data.

Table 9

Comparison with round TCI results from literature (Kim, 2009; Kim and Bruland, 2010) and case study tunnel.

Tunnel	Minimum round TCI	Maximum round TCI	Largest frequency
LS02 (Norway)	38	62	47–50
Marienberg (Norway)	47	58	53–56
Misiryung (Korea)	56	75	62–71
Case study	38	77	62–65

Table 10

Comparison of TCI in different studies.

TCI parameter	Case study
C ₁	0.0049
C ₂	0.6765
C ₃	0.0206
Average O _v [cm]	70.2
Average RCL	1.2
V ₀	5.4
TCI	57.4
<i>Tunnel TCI from literature</i>	
LS02 (four segments)	48.2
Marienberg	52.1
Misiryung	62.4

quality in a unique way, simplifying the contractual requirements.

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APPENDIX C. Paper C

On the mutual relations of drill monitoring variables and the drill control system in tunneling operations

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On the mutual relations of drill monitoring variables and the drill control system in tunneling operations

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ABSTRACT

The mutual relation between Measurement While Drilling (MWD) parameters in percussive-rotary drilling is investigated in order to limit the number of MWD variables, and to select the more significant ones, that are required for a sound rock mass characterization. Cross-correlation of the MWD signals is used to determine their relations, the analysis of which brings a thorough understanding of the way that the rig control system governs the different parameters. Data recorded from six drilling machines, working in semi-automatic and manual mode, on three different rock types with different logging interval, have been analyzed. The results point out the feed pressure as the lead parameter that drives the adjustment of the other variables to optimize the drilling, thus being the upfront response to variations in the rock mass. Control parameters that limit this adjustment to minimize disturbances during the operation are also discussed. Finally, the influence of the rock mass condition on the MWD parameters response is studied.

1. Introduction

The automatization of the different stages involved in a tunnel or underground construction brings new possibilities to optimize the operation. One of the most important requirements for the development of an underground work consists of a proper rock mass characterization for optimizing the stages of the drilling, blasting and support cycle. Conventional methods for geological and geotechnical ground characterization estimate a global value for each round (Barton et al., 1974; Bieniawski, 1989), according to the visual interpretation of the rock face or drilling cores (Palmström, 1996). Since rock characteristics have an important influence in the drilling response (Peng et al., 2005; Schunnesson et al., 2011), technologies based on measuring drill parameters may evaluate changes in the rock mass with higher resolution than conventional methods.

The Measurement While Drilling (MWD) technique is a drill monitoring system that collects operational drilling data at predetermined length intervals along the blasthole (Schunnesson, 1997). This real time information of the rock mass has made MWD a complementary tool for rock mass characterization and geotechnical ground recognition. However, despite the advantages of this technology, the massive data recorded per blast and the difficulties of interpretation complicate its application as a decision-making tool in the daily routine of construction works.

In underground works excavated by drilling and blasting, holes are

typically drilled by hydraulic percussive-rotary machines. Generally, jumbos with MWD technology have sensors installed along the operational mechanisms of the boom and the drill string to record digital signals of several parameters involved in the drilling operation. The management of the drilling is carried out by a control system that adjusts the values of the drill parameters in order to reach an optimum operation of the rig (Schunnesson, 1998; Schunnesson et al., 2011) and reduce bit wear (Schunnesson, 1997). The adjustment of these parameters must be led by one (or few) pre-set parameters that run the drilling and drive changes in the other parameters according to the requirements found along the blasthole.

In order to improve drill accuracy, a proper collaring and alignment must be ensured (Östberg, 2013). Nowadays, jumbos like those manufactured by Atlas Copco allow to automatize the drilling operation by installing systems in the drill rig such as the ABC (Advanced Boom Control), which helps the operator to follow a predesigned drill pattern and optimizes the drilling. The ABC system enables to program the drill rig in three operational automatization levels: basic, regular and total (Nord and Appelgrm, 2001; Atlas Copco, 2010). In the basic level, both positioning and drilling are done manually; the system allows the operator to watch in a control screen and record the collaring and alignment of the boom, whereas monitoring the drilling is not available. In the regular level, the operator follows a predesigned drill plan, by controlling boom and feeder manually; drilling and data logging are done automatically. The total level authorizes the operator to follow a

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predesigned drill plan, to log the drill hole operation and to switch collaring and drilling between manual, semi-automatic and fully-automatic mode. In manual ABC total system, the operator moves the boom and feeder manually and takes control of the drilling process. In semi-automatic system, the operator controls also boom and feeder manually but the drilling is automatic. The fully-automatic ABC total system, enables to program collaring, alignment and drilling automatically (Atlas Copco, 2010). Other manufacturers like Sandvik have similar automatization in their jumbos by installing the iDATA control system in the drill rig, in combination with the Sandvik iSURE software (Sandvik, 2017).

Some authors have provided interpretations of the performance of the control system. Schunnesson (1998) defined a normalized torque pressure as rock-dependent drilling parameter. Schunnesson et al. (2011) indicated that percussive pressure, feed pressure and rotation speed are the rock-independent drilling parameters. Peng et al. (2005) described the data control unit installed in the drilling system, in which some pre-set MWD parameters and their thresholds could be selected. They claimed that penetration rate, rotation speed and feed pressure were the key parameters in the performance of the drilling operation. Danell (1989), Gui et al. (2002) and Cooper et al. (2004), and stated that the drilling must hold constant force and rotation speed.

Our goal is to select the parameters that actually govern the drilling in order to limit the number of variables to be used in a sound rock mass characterization from MWD data. The above background points out some controversy in the performance of the control system as to what are the mutual relations between MWD parameters and their precedence, hence the need of analyzing the actual signals from a cross-correlation standpoint. Cross-correlations tools have been used to analyze the relationship between the MWD signals (Navarro et al., 2017). They preliminary showed that feed pressure is the master parameter that defines the adjustment of the others, thus the one sensitive to variations in the rock mass. This work builds on that and extends the study to other cases with different drills, rocks and logging intervals, resulting in a wide body of data from which significant conclusions can be drawn.

2. Statistical tools

Digital signals that measure the operational parameters of the jumbo are monitored as a discrete set of values at equal depth intervals. Statistically, MWD signals may be regarded as a discrete time series (time being in this case equivalent to depth). A stochastic process may be applied to these series (Box and Jenkins, 1976) to analyze their correlations.

2.1. Cross-Correlation

The correlation of two signals sequence is a measure of the statistical degree to which the two signals are similar (Box and Jenkins, 1976; Proakis and Manolakis, 1996). Cross-correlation is defined from the cross-covariance function. Let X and Y be two variables available as depth series x_t and y_t of length N , the cross-covariance C_{xy} between both signals is given by Box and Jenkins (1976):

$$C_{xy}(k) = \begin{cases} \frac{1}{N} \sum_{t=1}^{N-k} (x_t - \bar{x})(y_{t+k} - \bar{y}); & k = 0, 1, \dots, N \\ \frac{1}{N} \sum_{t=1}^{N-k} (x_{t+k} - \bar{x})(y_t - \bar{y}); & k = 0, -1, \dots, -N \end{cases} \quad (1)$$

where $\bar{x}$ and $\bar{y}$ are the means and k is the shift or lag parameter.

The cross-correlation r_{xy} is defined as a dimensionless coefficient by scaling the cross-covariance function C_{xy} with the product of the variances S_x and S_y of the two series at $k = 0$:

$$r_{xy}(k) = \frac{C_{xy}(k)}{S_x S_y}; \quad k = 0, \pm 1, \dots, \pm N \quad (2)$$

where

$$S_x = \sqrt{C_{xx}(0)} \text{ and } S_y = \sqrt{C_{yy}(0)} \quad (3)$$

The cross-correlation function is not symmetric about $k = 0$ (Box and Jenkins, 1976). Eq. (1)a and b describe the direction in which a signal (i.e. depth series) is shifted over the other. Eq. (1)a shows the depth series y_t shifted forward k units with respect to the series x_t . In this case, the x_t series leads the y_t series or, equivalently, y_t lags x_t . Eq. (1)b indicates the opposite situation by shifting x_t forward y_t , and analyzes the correlation between the two series when y_t leads x_t (Proakis and Manolakis, 1996; Navarro et al., 2017). Cross-correlation results are represented in the form of correlograms, where r_{xy} is plotted as a function of the lag k . Since cross-covariance is scaled by the variances, the result of the cross-correlation between two signals is ranked from 1 to 0 for positive correlation and from 0 to -1 for negative correlation, being ± 1 maximum correlation and 0 no correlation. The highest absolute value of r_{xy} indicates the maximum correlation between two time series and the lag (k value) at which this occurs shows the delay between them.

From the results of the cross correlation, the existence and location of a maximum and other peaks in the correlogram may indicate different relationships between time series. Although several works (Yevjevich, 1972; Box and Jenkins, 1976; Hamilton, 1994; Proakis and Manolakis, 1996; Antoniou, 2006; Shumway and Stoffer, 2011) have described the cross-correlation analysis and its applications, the interpretation of the type of correlograms that may be obtained is generally very limited. With the purpose of filling this gap, an interpretation of the common correlograms follows.

Fig. 1 shows seven possible correlogram types that may be found for two MWD signals X and Y . Since Eq. (1)a and b show different shift directions between the time series, each correlogram has been divided into two zones, one for each sign of the k value. The leadership direction in each zone between variables X and Y is written $X \rightarrow Y$ or $Y \rightarrow X$, i.e., X leads Y for positive lags or Y leads X for negative lags, respectively. The red lines show the 95% coverage of the cross-correlations, so that correlation values outside this band indicate significant correlation. The interpretation of the seven correlograms in Fig. 1a–g is the following:

- Significant correlations or fluctuations in the correlogram signal, including the maximum, are in only one side of the correlogram (Fig. 1a). This indicates a causal relationship with a delay between the two series and memory in the response, i.e., there are several peaks at different lags, which may point out that the output is influenced by present and past input values. The leadership direction between the two signals falls into the correlogram zone with significant correlations (in the case of Fig. 1a, Y leads X). This also applies to graphs b (Y leads X) and c (X leads Y) of Fig. 1.
- Maximum correlation on one side of the correlogram but no other significant correlations (Fig. 1b). This represents a causal relationship with a delay between the two series but no memory in the response, i.e., the two series are correlated only once, in contrast with case a. This may indicate that the output is only influenced by present input values.
- Maximum correlation at lag 0 and significant correlations or fluctuations in the correlogram signal on one side of the correlogram. This reveals a causal relationship with instantaneous reaction between input and output and memory in the response (Fig. 1c).
- Maximum correlation at lag 0 but no other significant correlations or fluctuations in the correlogram signal. This indicates correlation between the two series but none of the signals leads the other (Fig. 1d). This correlogram shows an independent behavior between them and thus a non-causal relationship.
- All correlation values within the confidence band with no apparent differences at positive and negative lags. This indicates a non-causal

Fig. 1. Examples of cross-correlations between X and Y MWD signals: (a) Causal relationship with delay and memory response between parameters; (b) Causal relationship with delay but no memory response between parameters; (c) Causal relationship with instantaneous reactions and memory in the response; (d) Non causal relationship with two correlated signals with no leadership; (e) No significant correlation; (f) No significant correlation but memory in the response; (g) Contradictory results (non-causal system).

relationship (Fig. 1e).

- (f) Non-significant correlations on one side and flat response on the other side of the correlogram (Fig. 1f). This points out a non-causal relationship with certain memory in the response and thus, a slight influence of the input on the output response. The leadership direction between the two series belongs to the correlogram zone with higher fluctuation: in the case of Fig. 1f, X leads Y .
- (g) Maximum correlation on one side of the correlogram and significant correlations or fluctuations on the other side (Fig. 1g). These indicate contradictory results and therefore a non-significant input-output dependence (non-causal system).

2.2. Auto-correlation

The auto-correlation function reveals the degree of correlation between members of the same signal when it is shifted with respect to itself. In the same way as cross-correlation, the auto-correlation is defined by the auto-covariance function, scaled by the variance of the series at $k = 0$ (Eq. (4)).

$$r_x(k) = \frac{C_{xx}(k)}{S_x S_x}; \quad k = 0, \pm 1, \dots, \pm N \quad (4)$$

where, C_{xx} is obtained replacing y_{t+k} by x_{t+k} and $\bar{y}$ by $\bar{x}$ in Eq. (1a).

The auto-correlation function, which is symmetric about zero, is commonly used for the interpretation of the signal properties (stationarity, periodicity or repeating patterns, frequency) and the detection of non-randomness in data (Box et al., 1994). For the case under study, auto-correlation has no interest for the analysis of the relation between MWD parameters and the performance of the control system, but it is calculated and shown together with the cross-correlation for completeness. Notwithstanding, further tests based on the study of the auto-correlation applied to MWD signals may be an interesting subject of research towards a deeper insight into drilling performance.

3. Data overview

Three underground drill and blast excavations works have provided data to this study:

- Five caverns and a main access gallery of 850 m excavated to extend the existing municipal wastewater treatment plant in Bekkelaget (Oslo, Norway). The rock mass was composed by gneiss with small tonalite and quartzite intrusions.
- A roadway tunnel construction of 4500 m length and 80 m² cross section excavated through sandstones, slates and expansive clays with chlorites in North Norway.
- A high-speed rail tunnel in the line Madrid-Galicia, in the province of Orense, Northwest Spain. The excavation was developed in sandstones and slates, with 7900 m length and 80 m² cross section.

Table 1 shows the working conditions (wc) where data was gathered. All jumbos were manufactured by Atlas Copco; two models with three booms and one with two booms were used. They all use a percussive-rotary top hammer drilling mechanism with different levels of ABC total system. Data comprises production face drilling holes of short length (4–5 m), drilled by using only one rod of 5 m length and 38 mm diameter and a bit of 46 mm diameter. Drill directions are almost parallel with a maximum lookout angle of 10° and 15° for the contour and bottom holes, respectively. The version of the revision control system (RCS) of the jumbo is indicated in Table 1 for each working condition.

The data set comprises 687 blasts and more than 65,000 signals from 11 different working conditions; these are identified with the number assigned to the jumbo followed by a letter when they use a different drilling mode, or if the rock type or the sample interval vary. This occurs, for instance, with Jumbo 2 that worked on both manual (wc 2b) and semi-automatic levels (wc 2a).

Table 2 lists the parameters monitored by percussive-rotary jumbos; it also gives the units and the acronym used for each of them. They have been described in detail by Peng et al. (2005), Beattie (2009), Hjelme (2010), Schunnesson et al. (2011) and Schunnesson and Kristoffersson (2011).

4. Data processing and analysis

Fig. 2 shows, as an example, all MWD signals for one hole. Penetration rate, rotation pressure and damp pressure show a drift with

Table 1
Description of data characteristics of working conditions of Jumbos.

Jumbo	wc ^a	Jumbo		RCS ^b	Sample interval (m)	Rock type	No. blast
		Model	ABC Total System				
1	1	XE3C	Semi-auto	4.9	0.1	Gneiss	159
2	2a	XE3C	Semi-auto	3.7	0.1	Gneiss	76
	2b	XE3C	Manual	3.7	0.1	Gneiss	90
3	3	E2C	Semi-auto	4.8	0.1	Gneiss	29
4	4a	XE3C	Semi-auto	4.7	0.1	Gneiss	95
	4b	XE3C	Semi-auto	4.7	0.1	Sandstone, Slates & Clays	67
	5a	XL3D	Semi-auto	2.185	0.02	Sandstone, Slates & Clays	48
5	5b	XL3D	Semi-auto	2.185	0.02	Sandstone & Slates	15
	5c	XL3D	Semi-auto	2.185	0.2	Sandstone & Slates	21
	6a	XL3D	Semi-auto	4.0	0.1	Sandstone & Slates	49
6	6b	XL3D	Semi-auto	4.0	0.2	Sandstone & Slates	37

^a wc is working conditions.

^b RCS is revision control system.

Table 2
MWD parameters description.

Parameter	Acronym	Unit
Penetration Rate	PR	m/min
Hammer Pressure	HP	bar
Feed Pressure	FP	bar
Damp Pressure	DP	bar
Rotation Speed	RS	rpm
Rotation Pressure	RP	bar
Water Pressure ^b	WP	bar
Water Flow ^{a,b}	WF	l/min

^a For working condition 5a, this parameter is not available.

^b For working conditions 5b, 5c, 6a and 6b these parameters are not available.

depth. Schunnesson (1998) related this dependence with the increase of the frictional resistance between the drill string and the walls of the borehole, the reduction of the available pressure over the hammer, the decrease of the flushing efficiency with depth, and the bit wear.

As auto-correlation and cross-correlation require stationary or depth-invariant series (Box et al., 1994), data should be processed (detrended) to compensate the hole depth effect. Data differentiating is often used to remove stochastic trends (Box et al., 1994). For our data, it is enough to calculate the first difference between elements of each signal; considering a discrete depth series Y at a period t , the first difference is defined as $Y_t - Y_{t-1}$; the resulting series has a near-zero mean value. This can be seen in Fig. 2b, where the detrended signals from

Fig. 2a are plotted. Detrended parameters have been noted by using the same acronyms as in Table 2 with an asterisk.

Considering the detrended full signals for each hole, auto-correlations have been calculated for the available MWD parameters, whereas cross-correlation has been calculated for all pairs of parameters. Calculations have been made with Matlab (2016) using a window (equivalent to the range of k in Eq. (1)a and b) of 2 m between signals. A smaller window was not possible because some of the working conditions, with 0.2 m logging intervals (wc 5c and 6b, Table 1) result in short series that do not allow to get peaks other than the main one.

Fig. 3 shows, in a correlation matrix, the resulting correlograms and auto-correlograms (main diagonal) from a single hole of one of the working conditions under study (wc 3 in Table 1). The MWD parameters in the abscissa and ordinate of the correlation matrix correspond, for each graph, to the X and Y signals as from Eq. (1)a and b, respectively. As explained in section 2, the leadership direction between the two MWD parameters falls to the correlogram zone with significant correlations or the side with higher signal fluctuations. Results from Fig. 3 can be matched with the interpretation of the correlograms in Fig. 1: typical correlograms of causal relationships are colored in green in Fig. 3. There is a delay with memory response (Fig. 1a) between penetration rate (PR) – hammer pressure (HP) and PR – feed pressure (FP). HP is delayed with respect to FP but with no memory response (Fig. 1b). FP induces an instantaneous response with memory on damp pressure (DP) and rotation pressure (RP) (Fig. 1c). Other relations between parameters do not give relevant information about the

Fig. 2. Detrending technique to remove hole depth dependence on MWD signals from Jumbo 1 for hole No 57 of the blast located at the kilometric point 399 of the main gallery of Bekkelaget's construction. (a) Raw signals; (b) Processed signals. The units of the parameters are given in Table 2.

Fig. 3. Cross-correlation matrix of the MWD parameters for hole 7 drilled with Jumbo 3 at the kilometric point 99 of cavern 1 of Bekkelaget's construction.

performance of the control system. For instance, RP is independent of PR (Fig. 1d) and correlation RP – water pressure (WP) (Fig. 1e), HP – rotation speed (RS) and FP – RS qualify as non-significant to the 95% level. However, for the last two pairs, there is a clear memory in the response (Fig. 1f). Contradictory results occur between HP and both DP and RP (Fig. 1g).

Peaks and fluctuations (at positive and negative lags) that are apparent for a single hole (Fig. 3), become blurred when representing several holes in the same plot, which complicates the detection of a global pattern between the parameters. In order to solve this, the technique developed by Navarro et al. (2017) has been followed. First, for each working condition, a mean correlogram per blast is calculated (this can be done as the cross-correlation functions are calculated over the same lag). Secondly, the standard deviation of the correlations on each side of the correlogram are calculated for each hole, which indicates the side of the correlogram with higher fluctuations. In order to consider only secondary correlations, the maximum peak must be avoided in the standard deviation calculation. As the peak correlation is always within ± 3 lags (see Fig. 3), the standard deviation is calculated from lag +3 to the maximum lag on the positive side and from lag -3 to the minimum lag on the negative so that, for every pair of signals, one standard deviation is obtained for positive lags and another for negative. The distributions of standard deviations for each working condition are represented in the form of two histograms: one for positive lags and another for negative.

5. Results and discussion

The analysis of the mean correlogram and the standard deviation histograms shows seven different types of correlations between the MWD parameters. Their characteristics are described in Table 3, in line with Fig. 1. The fluctuations at each side of the correlogram are compared from the histograms of the standard deviation at positive and negative lags; the empirical cumulative distribution functions corresponding to each histogram have been compared by using a two-sample Kolmogorov-Smirnov test (Marsaglia et al., 2003); where significant differences are detected, there is memory response.

In order to illustrate the correlations defined in Table 3, data recorded for jumbo 1 (wc 1, Table 1) is considered as an example; the

Table 3

Description of the seven correlation results.

Correlation type (Fig. 1)	Mean Correlogram		Histograms of the standard deviation	
	Existence of Significant Max. Peak.	Lag of the peak (k)	Existence of Significant differences	Sign of the lags with higher standard deviation ^a
A	Yes	$k \neq 0$	Yes	Same as Max. Peak
B	Yes	$k \neq 0$	No	No differences
C	Yes	$k = 0$	Yes	Positive or Negative
D	Yes	$k = 0$	No	No differences
E	No	–	No	No differences
F	No	–	Yes	Positive or Negative
G	No	$k \neq 0$	Yes	Opposite to Max. Peak

^a This is the lag of the histogram with significant higher standard deviation.

data set comprises 159 blasts. Fig. 4 shows the mean correlograms per blast for the MWD parameters recorded. The correlograms are plotted with a different color according to the correlation characteristics between MWD parameters given in Table 3, where the letter codes A through G correspond to the cross-correlation types described in Fig. 1; only correlation type D does not occur in this case. It is found that for each pair of variables, the 159 blasts follow the same mean correlogram type (Fig. 4), which validates the consistent behavior of the control system. Fig. 5 shows the histograms of the standard deviation for each of the two correlogram zones. The p-value of the two-sample Kolmogorov-Smirnov test is also given on the top-right side of each graph; when p is less than 0.05, the distributions of standard deviation are different at a 95% confidence level. The correlation type is denoted by the same color as in Fig. 4. In the same way as for Fig. 3, the MWD parameters in the abscissa and ordinate of the correlation matrix correspond to the X and Y variables, respectively.

Among the correlations in Table 3, only A, B and C types show a clear input-output response between MWD parameters. Figs. 4 and 5 show that feed pressure (FP) leads penetration rate (PR) and hammer

Fig. 4. Representation of the mean correlograms; each graph shows the 159 mean correlograms for working condition 1; only values at ± 10 lags are given in order to better show the main peak.

pressure (HP) with positive correlation and delay in the output response (A type). FP also leads damp pressure (DP) and rotation pressure (RP) with immediate response, i.e. lag = 0 (C type). HP leads PR (A type) and DP leads PR (B type) and RP (C type); such correlations may be a consequence of the leadership of FP on PR, HP and DP. These results suggest, for jumbo 1 (wc 1), that feed pressure is the parameter used by the control system to lead the adjustment of the other parameters of the jumbo. These results are in line with those obtained by Navarro et al. (2017) using a 1 m window to compute the cross-correlations.

The control system is normally based on three main operational modes (Schunnesson, 2017): (i) collaring, (ii) ramp-up, both of which control the increase of the drilling pressure to minimize hole deviations, and (iii) normal drilling, which controls the performance of the parameters to optimize the operation and minimize damages in the boom. As can be seen in Fig. 2a, the feed pressure shows, initially, a sharp rise until it reaches a pre-set threshold at which it stabilizes. The same analysis has been carried out by applying the correlation to the signals in the ramp up phase on one side and the normal drilling on the other.

Fig. 5. Histograms of standard deviation at positive and negative lags for each of the correlograms of the 159 blast drilled in working condition 1. Black histograms are the standard deviation on negative lags and white histograms on positive.

Fig. 6. Representation of the mean correlograms for the ramp up operational mode for working condition 1; each graph shows the 159 mean correlograms.

Fig. 8. Representation of the mean correlograms for the normal drilling operational mode for working condition 1; each graph shows the 159 mean correlograms.

For that, the signals of the 8 parameters have been divided considering the point at which the feed pressure stabilizes. Values previous to this point form the ramp-up data set and forward values the normal drilling data set. Figs. 6 and 8 show the mean correlograms per blast for the ramp-up and normal drilling phases, respectively, for working condition 1 (the same working condition that has been analyzed above in Figs. 4 and 5). Due to the limited depth of the records in the ramp-up phase, the cross-correlation in this case has been done only for a window of 4 lags. Figs. 7 and 9 show the histograms of the standard deviation for each of the two correlogram zones for the ramp-up and normal drilling, respectively.

Results of the cross correlation for both ramp-up and normal drilling phases are similar to the results obtained in the analysis of the full

signals in all working conditions. For the case shown of working condition 1, slight differences have been found in comparison of Figs. 4 and 5 and Figs. 6–9. For the ramp-up mode, differences in the relation HP-PR (B type in ramp-up mode instead of A type for the full signal) and relations between RS, WP, WF and PR, HP (E type in ramp-up mode instead of F type for the full signal) are obtained. For the case of normal drilling, differences are identified between FP-HP (C type in normal drilling mode instead of A type for the full signal) and DP-RP (D type in normal drilling mode instead of C type for the full signal). These differences might be due to the lesser amount of data in the analysis (especially for the ramp-up case). Notwithstanding, they do not introduce any significant change in the leadership behavior between parameters and evidence that the feed pressure is the parameter with

Fig. 7. Standard deviation at positive and negative lags for each of the correlograms of the 159 blasts for the ramp up operational mode for working condition 1. Black histogram plots the standard deviation on negative lags and white histograms on positive.

Fig. 9. Standard deviation at positive and negative lags for each of the correlograms of the 159 blasts for the normal drilling operational mode for working condition 1. Black histogram plots the standard deviation on negative lags and white histograms on positive.

highest influence in the performance of the control system.

Results for the analysis of the full signal for the eleven working conditions under study are summarized in Table 4. Cells corresponding to correlation types A, B and C show the leadership direction between parameters, the lag in the response (k) and the sign of the correlation symbol; r (+) indicates a positive correlation and r (–) a negative one. Color code is the same as in Figs. 4 and 5.

The correlation type (C) between feed pressure, damp pressure and rotation pressure (FP-DP and FP-RP) is the same independently of the working conditions involved (with exception of wc 2.b, with manual drilling mode). This outlines that these three parameters are key indicators of the control system in which the feed pressure provides a

measure of the hydraulic pressure of the feeder that allows to keep the bit in contact with the rock while it rotates. This induces a torque that is measured by the rotation pressure. Additionally, the pressure provided by the damper should react immediately to changes in the hydraulic pressure to prevent undesired motion of the boom (Peng et al., 2005; Schunnesson et al., 2011; Navarro et al., 2017).

For jumbos working in semiautomatic mode, relations in which penetration rate is involved HP-PR, FP-PR, DP-PR and relations between FP-HP, qualify as A, B or C types depending on the working condition. This reflects changes only in the lag of the response (i.e. delayed response, A or B types and instantaneous response, C type), whereas the sign of the correlation and the leading parameter do not

Table 4
Relationships between MWD parameters for the 11 working conditions in Table 1.

wc								
HP	1	A: HP → PR; k (1); r (+)						
	3							
	4a							
	4b							
	6a							
	2a	C: HP → PR; k (0); r (+)						
	5a							
	5b							
	5c							
	6b							
	2b	E						
FP	1	A: FP → PR; k (1); r (+)						
	3							
	4a							
	4b							
	6a							
	2a	C: FP → PR; k (0); r (+)						
	5a							
	5b							
	5c							
	6b							
	2b	B: FP → PR; k (1); r (-)						
DP	1	B: DP → PR; k (1); r (+)						
	3							
	4a							
	4b							
	6a							
	2a	B: DP → PR; k (1); r (+)						
	5a							
	5b							
	5c							
	6b							
	2b	B: DP → PR; k (1); r (-)						
RS	1	F						
	3							
	4a							
	4b							
	6a							
	2a	F						
	5a							
	5b							
	5c							
	6b							
	2b	E						
RP	1	E						
	3							
	4a							
	4b							
	6a							
	2a	C: FP → RP; k (0); r (+)						
	5a							
	5b							
	5c							
	6b							
	2b	E						
WP	1	F						
	3							
	4a							
	4b							
	6a							
	2a	F						
	5a							
	5b							
	5c							
	6b							
	2b	E						
WF	1	F						
	3							
	4a							
	4b							
	6a							
	2a	F						
	5a							
	5b							
	5c							
	6b							
	2b	E						
		PR	HP	FP	DP	RS	RP	WP

vary. The relations show that FP always leads PR, HP, RP and DP.

Results for working conditions 3, 4a, 4b and 6a are in line with those for working condition 1. All these cases share the same automatization level setting (semi-automatic) and the sampling rate, whereas the rock drilled varies for some of them. This suggests that the inter-correlations between MWD parameters are independent of rock characteristics. On their side, correlations in working conditions 2a, 5a, 5b, 5c and 6b qualify as C type instead of A, suggesting that the response of the hammer pressure and penetration rate to changes in feed pressure occurs immediately. The rock mass was also different for some of these working conditions, which confirms the (otherwise expected) independence of the control system from the rock mass. Differences in the version of the revision control system (RCS, Table 1) should explain this behavior: RCS versions higher or equal than 4.0 (wc 1, 3, 4.a, 4.b and 6.a) show similar results in contrast to RCS lower than 4.0 (2.a, 5a, 5b, 5c and 6b).

Interestingly, for jumbos working with 0.02 m sampling length (jumbo 5, wc 5a and 5b) no delay in the response is observed. This may indicate a different performance of jumbo 5 or a spurious effect caused by an older RCS version (Table 1). For jumbos with logging interval of 0.2 m (wc 5c and 6b), the fact that no delay is observed is a consequence that, for them, a zero lag (no apparent delay) includes any actual delay from 0 to 0.2 m; this corresponds to lags zero and one for logging interval 0.1 m.

For jumbos working in semiautomatic mode, most of the relations are of A or C type for significant correlations (i.e. between PR, HP, FP, DP and RP) and of F type for relations between HP, FP and DP with RS, WP and WF. These relations indicate memory in the response and suggest that the rig control system uses present and past values of the master parameter (feed pressure) to adjust the others.

Different behavior is found in Jumbo 2 (wc 2b) working in manual drilling level, concerning the negative correlation between FP (input) and PR (output) parameters (B type) and the non-causal relationship of HP and RP, contrary to the rest of cases. The former may indicate a lower efficiency in the performance of the drill, which may be caused by the effect of the operator on the drilling. According to Schunnesson (1998), the penetration rate increases with the feed pressure until the former reaches a peak value. Further pressure increases will react with a decrease of the penetration rate. Despite these differences, the feed pressure still seems to be the most influential parameter during the drilling operation. In addition, there is no memory between parameters except of the FP – DP relation, which means that the adjustment of the parameters with manual operation only follows present values of the feed pressure and disturbances in the adjustment of the other parameters are controlled by the operator.

By and large, the feed pressure seems to be the parameter that leads the changes in the other parameters. Considering the three main operational modes: collaring, ramp-up and normal drilling, the feed pressure signal (see Fig. 2a) shows in the beginning a sharp increase until it reaches a pre-set threshold at which it stabilizes. During this ramp-up operation, the feed pressure takes control of the drilling and adjusts the other parameters. Once the feed pressure reaches a pre-set threshold value, it stabilizes and the normal drilling mode starts. In this mode, the feed pressure keeps its influence over the other parameters and adjusts them in order to keep an optimal drilling pressure.

When open fissures in the rock are found, the feed pressure shows a sharp drop to the level of drilling in the air (Finfinger et al., 2002; Peng et al., 2005; Kahraman et al., 2016). Fig. 10 shows an example of this situation, where discontinuities at about 0.7 and 3.2 m are found. Similar graphs are shown e.g. in Peng et al. (2005) and Kahraman et al. (2016). Schunnesson (1996 and 1998) and Schunnesson et al. (2011) showed that when the drill bit goes through a fracture zone, the rotation pressure increases. This should be the response of a disturbance in the adjustment of the parameters. As can be seen in Fig. 10, when the drilling reaches a discontinuity, the rotation pressure shows a peak followed by a sharp decrease and later by an increase, connected with

Fig. 10. MWD signals in rock discontinuities drilled by Jumbo 1 for hole 52 of blast from Kilometric point 747 of Bekkelaget's construction.

the behavior of the feed pressure. In the same way, Figs. 2a and 10 show a peak in the damp pressure where feed pressure stabilizes after the ramp-up. The leadership behavior of the feed pressure is limited by certain bounds of variation preset by the manufacturer, in order to optimize the performance of the jumbo. Moreover, some parameters led by the feed pressure, such as the rotation pressure and damp pressure, also limit the feed pressure adjustment by using specific bounds preset in the control system. They work as control parameters in order to minimize damages in the mechanism of the boom. In line with Schunnesson (1996, 1998) and Schunnesson et al. (2011), if the rotation pressure bounds are exceeded (as it will likely be upon a discontinuity), the feed pressure ramps down so that they return to their working range. For the damp pressure case, bounds preset in this parameter are mainly used to limit the increase of pressure during the ramp-up operation and minimize undesirable motions in the boom while drilling.

Other parameters are not significantly correlated with any other (relation types E or F) such as rotation speed (RS), water pressure (WP) and water flow (WF). This involves no (E type) or little (F type) effect of the control system on these variables. Some authors indicate that discontinuities can induce variations in water flow and water pressure (Schunnesson et al., 2011). Water flow shows negative peaks in Fig. 10 where discontinuities appear, confirming its relation with rock variations. On the contrary, RS and WP signals have no significant variation.

6. Conclusions

A thorough analysis of the drilling control operational system of a number of rigs has been carried out, aimed at understanding the mutual relations between their MWD parameters. The ultimate goal is to determine the master parameters that most significantly reflect the variations in the rock mass, hence being the preferred ones for a proper rock mass characterization. Two statistical tools, auto-correlation and cross-correlation, have been applied to analyze data from six drilling machines in three different underground excavations with different rock type; 687 blasts and more than 65,000 boreholes have been involved in the study.

The results obtained lead to the following conclusions:

- Among the eight MWD parameters, penetration rate (PR), hammer pressure (HP), feed pressure (FP), damp pressure (DP) and rotation pressure (RP) show significant mutual correlation.
- For all working conditions studied (involving changes in the rock,

automatization level and sample interval), the feed pressure is the parameter that leads the adjustment of the other parameters.

- Feed pressure, damp pressure and rotation pressure show instantaneous response with memory between them for all working conditions when drilling in semi-automatic mode. This reflects the mechanical response of the control system to face disturbances during the drill. The rotation pressure and the damp pressure limit the feed pressure adjustment by bounds preset in the control system. They work as control parameters in order to minimize damages in the mechanism of the boom.
- Rotation speed (RS), water pressure (WP) and water flow (WF) are not significantly correlated with any other parameter. These parameters, not influenced by the control system, may be affected by variations in the rock mass.
- Contradictory results about the delay in the response of FP to PR and HP and of HP to PR, for semi-automatic mode, have been found. In five working conditions, the response is instantaneous whereas in the other five there is a delay. More data is required to explain this inconsistency; different RCS versions may be one of the reasons for it.
- Memory in the response between parameters with significant correlation for Jumbos working in semi-automatic drilling mode, suggests that the rig control system uses present and past values of the feed pressure to control the adjustment of the others. On the contrary, for manual drilling mode this adjustment only follows the present values of the feed pressure.
- The results for manual drilling point out a negative correlation between feed pressure and penetration rate, contrary to the semi-automatic drilling mode. This may indicate a lower efficiency of the drilling when operating in manual mode.
- In all cases, the performance of the control system is independent of the rock type.

Since changes in the rock mass primarily drive variations in the feed pressure when the thresholds of the control parameters are exceeded, and this influences the adjustment of the other parameters, the feed pressure is a potential MWD parameter to be used for rock mass characterization. Control parameters such as the rotation pressure can also indicate disturbances while drilling. Finally, other parameters not influenced by the control system, such as rotation speed, water pressure and water flow may reflect rock variations and, being independent of the feed pressure-controlled parameters, could add significance for a MWD-based rock mass characterization.

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APPENDIX D. Paper D.

Detection of potential overbreak zones in tunnel blasting from MWD data

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Detection of potential overbreak zones in tunnel blasting from MWD data

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Rock blasting

ABSTRACT

The damage from blasting to the remaining rock mass is analyzed with the purpose of developing a drilling index from measure while drilling (MWD) parameters, able to predict high risk of potential over- and under-excavated zones produced by blasting in the contour of a tunnel. A new methodology based on the comparison of scanner profiles of the excavated sections with the position of the contour blastholes, has been developed to obtain the excavated mean distance (EMD) between the blasthole and the excavated profiles at each MWD record position, which may be considered as a damage measure. MWD parameters, that describe the in-situ rock mass properties before the blast, are thoroughly normalized to remove external influences that may hide the actual response of the rig to rock mass properties and lead to wrong interpretations. 54 blasts, which comprise around 1700 contour blastholes, have been compared with more than 4000 excavated sections. A non-linear multiple-variable power-form model has been developed to predict the excavated mean distance as function of the normalized penetration rate, hammer pressure, rotation speed, rotation pressure and water flow parameters, and the lookout distance. These parameters combine the rotational, hydraulic and percussive mechanisms of the drill, and the confinement of the explosive charge with depth. Sources of uncertainty, unavoidable in the harsh condition in which the data were measured, such as drilling deviations, the scaling and primary support prior to scanning the excavated section, possible variations (unrecorded) in the explosive linear density, etc., have been assumed to be of random nature.

1. Introduction

Rock excavation in mining and tunneling frequently use cautious blasting techniques. The primary objective of blasting is to fragment rock to allow loading and haulage, without creating extensive damage to the remaining rock mass. As Andersson (1994) defined: “*Cautious blasting is a blasting that does not cause damage to the rock outside of the intended damage distance*”.

For a cautious blast design, the damage on the perimeter of the excavation is mainly induced by contour and buffer blastholes and it is created by a drop of strength, caused by the opening or shearing of newly generated or existing fractures or cracks (Scoble et al., 1997; Ouchterlony et al., 2002; Costamagna et al., 2018). In tunneling, damage can be categorized as major, minor or no damage, when there is rock falling, chips detachment or no visual damage, respectively (Costamagna et al., 2018). Damage assessment is analyzed through four main indexes: (i) Rock Tunneling Quality Index (Q-value, Barton et al., 1974) as classification of the ground condition for underground excavations; (ii) Blast Damage Factor (Hoek et al., 2002; Hoek, 2012) that estimates the global rock mass strength and the rock mass modulus; (iii)

Blast Damage Index (Yu and Vongpaisal, 1996) that correlates the mechanics and the effects of wave propagation into the rock mass; and (iv) failure approach index (Xu et al., 2017) that quantifies the rock mass damage through numerical simulations for tunnel support design.

Overbreak on the contour perimeter, which is defined as the void created during the excavation in excess of an established perimeter or pay line (Mahtab et al., 1997), is usually correlated with the damage extension zone which measures the quality of the blast. Overbreak and underbreak are mainly influenced by the geotechnical condition of the rock mass (rock disturbances and rock strength) and blast design parameters such as the explosive type, the charge concentration, the blast timing, the drill pattern and the drilling deviations (Ibarra et al., 1996; Oggeri and Ova, 2004; Singh et al., 2003; Singh and Xavier, 2005; Hustrulid, 2010; Johnson, 2010). Blasting affects the rock mass structure because of shock wave propagation (vibrations), gas pressure and stress redistribution (Singh et al., 2003; Hu et al., 2014).

Guidelines on construction have established an overbreak magnitude of 0.15–0.2 m and 0.1–0.15 m in crown and sidewalls, respectively (Mandal and Singh, 2009; Cunningham and Goetzsche, 1990). The maximum overbreak distance allowed depends on each national

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legislation and special terms can be arranged between the two parts in the contract (Olsson, 2010; Costamagna et al., 2018). For example, Scandinavian countries present similar regulations for tunneling excavation requirements (Anläggnings-AMA in Sweden, InfraRYL in Finland and the Norwegian Public Roads Administration, NPRA, in Norway; Olsson, 2010; SN, 2004; NPRA, 2004). Consequences of a bad drilling can be short pulls of the rounds, increase of rock reinforcement due to extra overbreak in the rock mass, longer scaling and mucking time and bad control of grouting.

To analyze the extension of the overbreak, studies have been carried out by comparing the laser profile of the excavated perimeter with the designed tunnel profile. Kwon et al. (2009) investigated the characteristics of the excavation damage zone (EDZ) in a tunnel construction and carried out a sensitivity analysis of the predicted EDZ with several rock mass parameters obtained from laboratory and in situ tests. They determined that in situ stress ratio, Young's modulus and EDZ were the three main parameters in rock mass behavior after blasting. Mandal and Singh (2009) measured the overbreak, dividing the contour profile in three sections: left wall, right wall and crown. They found that the crown is more affected by overbreak, due to the stress conditions in this zone. Kim and Bruland (2009, 2015) estimated a tunnel contour quality index (TCI) based on overbreak distances of cross-section scanners, contour roughness and longitudinal overbreak variation in each blasted round. Costamagna et al. (2018), used scanned tunnel profiles to assess the overbreak of the excavated void in relation to the intended theoretical section, and correlated this with the rock mass conditions of each round.

Rock excavation techniques are highly influenced by the geo-mechanical properties of the rock mass (Oggeri and Ova, 2004; Singh et al., 2003; Singh and Xavier, 2005; Mahdevari et al., 2013). A site investigation for a tunneling project generally includes a description of the rock condition and a rough estimation of the rock mass structural properties; however, during the operation, unexpected anomalies that may influence the results of the operation often occur. Such anomalies can be detected by Measure While Drilling (MWD) system on modern jumbos. This has been described by Schunnesson (1997) as a drill monitoring system which logs drilling data at predetermined length intervals providing information of the operational parameters involved in drilling. For rotary drilling, Teale (1965) and Liu and Yin (2001) introduced the concept of specific energy (SE) as the energy required to excavate a unit volume of rock. Scoble et al. (1989) studied the variations of monitored parameters to define different geological formations. Hatherly et al. (2015) compared the MWD data with the geological rock conditions, obtained by geophysical logs, and demonstrated that if rotary speed and weight on bit are kept constant, MWD measurements can determine rock properties. Leung and Scheduling (2015) have developed a coal-seam detection model called SEM (modulated specific energy).

In relation to studies developed for percussive and rotary-percussive drilling, Schunnesson (1996) used the logged parameters of percussive drilling to develop a method for estimating Rock Quality Designation (RQD, Deere and Miller, 1966) based not only on the penetration rate and torque parameters, but also on their variation, which shows a close correlation with the presence of large discontinuities, fractures or major faults. Schunnesson (1998) and Hjelm (2010) introduced a new methodology to normalize percussive and rotary-percussive drilling parameters, to remove external influences generated by the blasthole length and the drill rig performance. Peng et al. (2005) and Tang (2006) proposed a method for void/fracture detection and for prediction of the rock mass properties based on drilling for roof bolting. They found that the feed pressure is a good detector of anomalies or discontinuities in the rock and is a good estimator of the rock mass strength. In addition, Peng et al. (2005) designed a new methodology for normalizing MWD parameters based on determining the performance of the machine when drilling the air (i.e. for no-load condition, outside the rock mass). This assesses how much feed pressure or rotation pressure is required for

running the machine itself. From the correlation of MWD data with rock mass geo-mechanical measures, Schunnesson et al. (2011) suggested a model for the hydraulic properties of the rock mass, based on the monitored water flow and water pressure during rotary-percussive drilling. Schunnesson et al. (2012) explained a methodology to assess rock strength ranges based on a MWD hardness index provided by Atlas Copco software. For that, they used Schmidt hammer to correlate MWD values with empirical rock strength measurements. Naeimipour et al. (2014) developed a void detection algorithm based on MWD parameters. The algorithm was calibrated on full scale experimental tests in concrete blocks with various strengths. Kahraman et al. (2016) found a strong correlation of the penetration rate with the uniaxial compressive strength (UCS), the Brazilian tensile strength, the point load strength and the Schmidt hammer. Van Eldert et al. (2018) assessed the extent of the damage zone from MWD parameters and ground penetration radar measurements recorded along the tunnel wall.

Finally, Atlas Copco AB, Sandvik and Bever Control have developed their own software (Tunnel Manager MWD, iSURE and Bever Control, respectively), as a tool for planning, administration and evaluation of drilling. From the MWD files collected, the blastholes can be represented in 3D, and hardness and fracturing maps are provided. The theoretical background for this information is however confidential.

Although there are many studies focused on the overbreak control by blasting effect on the one hand and on geological and geo-mechanical interpretation of the rock mass by using MWD on the other, no relation between MWD parameters and overbreak (i.e., under or over excavation with respect to the theoretical tunnel contour line) from blasting exists. This paper aims at developing an engineering tool, based on MWD parameters, able to predict high-risk of overbreak potential zones generated on the perimeter of a tunnel face excavated by blasting. For that, scanner profiles of excavated sections have been compared with the contour blastholes position, to obtain the excavated mean distance (EMD) created by each hole. Given that blasting variables (mainly explosive charge and timing) are constant in contour blastholes (as from blast reports), the overbreak and underbreak are considered mainly influenced by the geotechnical condition of the rock mass. Since the MWD system can characterize the rock mass condition of the blastholes, these measures can be correlated with the excavated mean distance (EMD) calculated along the depth of each blasthole. The analysis will match high over-excavated zones with highly fractured or softer rock masses before the blast, and low excavated measures with competent rock.

2. Data overview

The study has been developed in the underground extension work of the municipal wastewater treatment plant in Bekkelaget (Oslo), Norway. The facility comprises five caverns, a main access gallery of about 850 m length and other sections. The construction was developed in competent rock mass, composed by gneiss with small tonalite and quartzite intrusions. The Rock Tunneling Quality Index (Q-value; Barton et al., 1974), obtained from visual inspection of the tunnel face, was used as classification of the ground condition. Fig. 1 plots the Q-value versus the chainage for the rounds analyzed in the study. They comprise 54 rounds located between chainages 294–518 and 560–772 of the main gallery (making up more than 400 m). Fig. 1 also shows the classification of the rock mass based on the Q-value index; rock mass condition generally qualified as good and fair.

Data from the rig have been used to locate the contour blastholes and to compare them with the excavated profiles. A three-boom jumbo XE3C, manufactured by Atlas Copco, equipped with percussive-rotary top hammer drill, using semi-automatic ABC (Advanced Boom Control) was used to drill the analyzed blasts. Data comprises production face drilling holes of short length (4–5.5 m), using single rod (5.5 m length and 38 mm diameter) and 46 mm bit. Eight MWD parameters were logged during the drilling operation with a sampling interval of 0.1 m.

Fig. 1. Rock Tunneling Quality Index; Rock Tunneling Quality Index versus chainage and classification of the ground condition: poor ($1 < Q \leq 4$), fair ($4 < Q \leq 10$), good ($10 < Q \leq 40$).

These are described next as an extension of Peng et al. (2005), Beattie (2009), Hjelme (2010), Schunnesson et al. (2011), Schunnesson and Kristoffersson (2011) and Navarro et al. (2017) interpretations; the acronym and the units for each parameter are given in brackets.

- Feed Pressure (FP, bar): measure of the hydraulic pressure inside the cylinders. Feed pressure is required not only to keep the bit in contact with the bottom of the hole throughout the transmission of energy, but also to maintain a minimum force between bit and rock to maximize energy transfer to the rock.
- Hammer Pressure or Percussive Pressure (HP, bar): this is a measure of the impact pressure acting on the piston in the rock drill.
- Damp Pressure (DP, bar): it measures the pressure absorbed by the drill rig to prevent vibrations or undesired motion in the boom or drill rod.
- Rotation speed (RS, rpm): it is defined as the number of turns of the bit per minute.
- Penetration Rate (PR, dm/min): rate of penetration of the drill bit through the rock mass.
- Rotation Pressure (RP, bar): it is the torque pressure required to rotate the bit at a defined speed.
- Water Pressure (WP, bar): it is the pressure of the water used to flush the drill cuttings from the blasthole.
- Water Flow (WF, l/min): it is the rate of water inflow into the drill rod
- Hole Length (HL, m): depth at which each sample of the above parameters is logged.

The charging of the rounds was carried out with emulsion of different linear charge for different types of blasthole: cut, lifter, easer, buffer, contour. String loading method was used for the charging of the contour blastholes, with a design linear charge of 0.5 kg/m. This is assumed to be constant in the analysis. The actual charge of the holes may vary from the design value. However, such variations are assumed to be random so that, although they are a source of indetermination in the analysis, they will not bias the influence of the other parameters in the overbreak from blasting.

3. Analysis of the excavated area

3.1. Jumbo navigation

Navigation is necessary to locate the jumbo inside the tunnel before drilling a new round. For that, the jumbo rig uses three reference systems, sketched in Fig. 2: (i) an absolute coordinate system that references the position of the jumbo, in this case the EUREF 89 Norwegian Transverse Mercator (NTM) projection, (ii) a Tunnel Reference System (TRS) with one axis parallel to the tunnel axis and the other two in the plane of the tunnel face of the new round, and (iii) a Drilling Reference

System (DRS) defined by two vertical planes X_dY_d , Y_dZ_d and a horizontal X_dZ_d plane. The angles of the TRS axes with the DRS ones are θ , ω and γ (see Fig. 2).

3.1.1. NTM coordinates system

The position of the jumbo inside the tunnel is first obtained, see Fig. 2 (i. NTM coordinate System). The jumbo has a laser scanner installed. In addition, target plates, with known coordinates, are located along the tunnel wall at every 5 m distance. The absolute coordinates of the jumbo are calculated by trilateration (i.e. distance measurement from the laser scanner to the target points). In this case study, X_{NTM} , Z_{NTM} coordinates are given in the NTM projection and Y_{NTM} is the height above sea level.

3.1.2. Tunnel Reference System (TRS)

The drill rig is aligned with the tunnel line (perpendicular line to the face of a new round that defines the orientation of the drilling) in order to follow the design of the construction. For that, two targets are mounted on one of the booms. The laser beam points to the free face in the direction of the tunnel axis and the boom is rotated until the laser beam passes through both targets (Fig. 2, ii. TRS). The boom is so aligned with the tunnel axis and the orientation and inclination of the boom are registered in three orthogonal vectors ($\vec{x}_t$, $\vec{y}_t$, $\vec{z}_t$) to create a coordinate system parallel to the tunnel axis and the free face.

The laser scanner also measures the distance from the jumbo to the face of the new round and records the chainage at which it is located inside the tunnel. This chainage is taken as reference plane of the collaring depth position of the blastholes. Negative depth values are assigned to measurements behind this plane, and positive values, to measurements ahead of this plane.

3.1.3. Drilling Reference System (DRS)

The blasthole position measured by each boom is calculated in the Drilling Reference System defined by means of three spherical coordinates: blasthole length (l_b), azimuth or lookout direction (L_D) and inclination or lookout angle (L_I) (see Fig. 2):

$$X_F = l_b \cdot \sin(L_I) \cos(L_D) \quad (1)$$

$$Y_F = l_b \cdot \sin(L_I) \sin(L_D) \quad (2)$$

$$Z_F = l_b \cdot \cos(L_I) \quad (3)$$

The inclination angle varies between 0 and 90° both for holes drilled upwards or downwards so that the azimuth is between 0 and 180° for holes drilled upwards and between 0 and –180° for holes drilled downwards.

Blasthole positioning data logged by the ABC system uses sensors installed along the boom (outside the blasthole) to measure the azimuth and the inclination angles. The semi-automatic ABC total system installed in the drill rig authorizes the operator to move the boom and feeder manually to follow a predesigned drill plan (Navarro et al., 2018). Once the boom is placed in the required position and before the drilling starts, the measurements of the azimuth and inclination angles are logged in the DRS. These are considered constant as the boom remains still while drilling the blasthole. The end coordinates of the blasthole are calculated by adding, to their collaring coordinates, the result from Eqs. (1), (2) and (3). Since deviations beyond the collaring point cannot be measured by the MWD technology there is a possible error between the actual end position of the blastholes and the end position given by the MWD system. Drill deviations have not been monitored so they are a source of indetermination in our analysis that, as those mentioned in Section 2, is assumed to be of random nature.

3.1.4. Transformation system TRS-DRS

For Atlas Copco jumbos, the directional coordinate vectors of the TRS ($\vec{x}_t$, $\vec{y}_t$, $\vec{z}_t$) and the NTM coordinates of the jumbo (X_{NTM} , Y_{NTM} ,

Fig. 2. Representation of the three reference systems involved in the jumbo navigation: i. NTM Coordinates System, ii. Tunnel Reference System, iii. Drilling Reference System.

Fig. 3. Transformation from drilling reference system ($\vec{x}_d, \vec{y}_d, \vec{z}_d$) to tunnel reference system ($\vec{x}_t, \vec{y}_t, \vec{z}_t$). Left: Rotation towards right and left, or bearing angle, over $\vec{y}_t$; center: uphill-downhill inclination, or elevation angle, rotation over $\vec{x}_t$; right: lateral inclination, roll or bank angle, rotation over $\vec{z}_t$.

Z_{NTM}) are presented at the end of each MWD file. The three rotation angles to transform the DRS coordinates of a point in a borehole to the TRS can be seen in Fig. 2 and are further explained in Fig. 3.

This transformation is required below in order to know the exact position of the blasthole collars and their orientation. The location of the oriented blastholes in absolute coordinates ($X_{t,NTM}, Y_{t,NTM}, Z_{t,NTM}$) is obtained by adding the NTM coordinates of the jumbo ($X_{NTM}, Y_{NTM}, Z_{NTM}$) to the oriented coordinates of the blastholes (X_b, Y_b, Z_b) in the TRS system.

3.2. Superposition of excavated profiles and contour blastholes

A laser scanner system has been used to monitor the final profiles of the excavated void from each blast. The laser was located in the center of the excavation to be scanned and target spheres were installed along the wall of the main gallery in places with known coordinates. The software of the scanner identifies the position of the spheres in a post-analysis of the 3D cloud of points and trilaterates the location of the scanner by measuring distances from the later to the spheres. Profiles of the excavated void in a direction perpendicular to the tunnel line are collected at steps of 0.2 m from the 3D cloud of points; each profile is

identified by its respective chainage. An interface AutoCAD-Matlab (AutoCad, 2017; Matlab, 2017) has been created to automatically compare the excavated profiles with the contour blastholes for each round. The profile formed by the contour holes (hereinafter named contour profile) is compared with the scanner profiles of the excavated sections in order to obtain the excavated mean distance between the blasthole and the scanner section at each depth for which MWD data are logged. This distance is considered as an indicator of the resulting damage (i.e. over-excavation).

For safety reasons, scanning of the excavated section is done after scaling and installation of primary supports. During scaling, non-stable rocks are removed from wall and roof to avoid rock falls and to ensure safe work conditions. Next, rock bolts and shotcrete are applied to reinforce the tunnel walls and to prevent stability problems. These operations obviously modify somewhat the perimeter excavated. Shotcrete was automatically applied to the tunnel surface and the thickness of the shotcrete layer was modified per round according to the geotechnical recognition of the tunnel wall and crown. The thickness of the shotcrete layer was known for every round and it was homogeneous over walls and crown, according to the operation reports. This thickness has been added to the scanner profiles in the AutoCad files to obtain the

Fig. 4. Chainage 399. (a) Contour holes profiles and position of the blastholes; (b) Overlapping between the contour profiles and the excavated section profiles.

actual excavated contour from the blast. However, an unknown uncertainty still remains.

The comparison between excavated and contour profiles must take into account the lookout angle and lookout direction values for each blasthole, which make them to be outward oriented. Since no deviation data of the blasthole is available, it is assumed that the lookout increases linearly with depth. Eqs. (1) and (2) are used to calculate the theoretical position of the blasthole at each excavated profile depth in the local Drilling Reference System (DRS). Variables L_I and L_F are obtained from the MWD files and l_b is the length of the blasthole from the collaring position to the respective excavated profile depth (Z_F , see Fig. 2), obtained with Eq. (3).

Irregularities on the free face of a new round cause the collars of the blastholes not to be in the same plane, so they have different collar depths. In addition, some excavated profiles (mainly profiles at the beginning of the round) are not influenced by all contour holes of the current blast but for from the previous one. Each blasthole is extended from the foremost collaring hole to the depth of the deepest blasthole of each round to calculate the excavated area for all profiles included in a round. Fig. 4a represents the contour profiles (cyan lines), the position of the blastholes (black lines) and their extensions (tiny blue dots).

To carry out the analysis, both contour and excavated profiles must be overlaid. Excavated profiles from AutoCAD files are drawn in a vertical xy plane, where the Y coordinate is referred to the Y_{NTM} absolute coordinate and the hypothetical Z coordinate, i.e. depth of the xy plane, is indicated by the chainage at which it is located. The contour blastholes coordinates in the DRS must be rotated to the TRS. Since the rounds studied belong to the main gallery and they are excavated in a straight line, the xy and xz planes for both TRS and DRS coincide (angles γ and ω are 0); only plane $y_z z_i$ is rotated in case the tunnel axis is uphill (positive θ angle) or downhill (negative θ angle), see Fig. 3. The rotation of the DRS contour blastholes coordinates (X_d, Y_d, Z_d) to the TRS (X_t, Y_t, Z_t) is obtained by introducing the three angles ($\theta, \gamma = 0, \omega = 0$) in a 3D rotation matrix. The translation of the Y_t and Z_t coordinates is carried out by adding the Y_{NTM} and the chainage values of the round studied, respectively. Fig. 4b sketches the overlapping of both excavated (red lines) and contour holes (blue lines) profiles for a round.

3.3. Excavated mean distance from the blasthole

The overbreak created around the blasthole by blasting is caused, among other factors, by the combination of the explosive and the rock mass condition around the blasthole (Hustrulid, 2010; Johnson, 2010). Considering the contour blastholes position per round, an Excavated Mean Distance (EMD) has been defined (Fig. 5). It corresponds to the area between the midpoint of the spacing on both sides of the hole and the excavated profile, normalized by the distance between the mid-points of the spacing on both sides of the blasthole. When two adjacent holes are on the same side of the excavated profile, the EMD is calculated by (Fig. 5, EMD 1):

$$EMD = \frac{A_{T1}}{\frac{S_1}{2} + \frac{S_2}{2}} \quad (4)$$

where S_1 and S_2 are the spacing between the current blasthole and the

Fig. 5. Calculation of the Excavated Mean Distance.

adjacent ones (they are generally around 0.7 m); A_{T1} is the area excavated by each blasthole defined by points 1, 2, 3 and 4 in Fig. 5, EMD 1; it is positive when there is over-excavation and negative for under-excavation.

In case two consecutive blastholes are located one inside and the other outside of the excavated profile, corresponding to over and under-excavation (Fig. 5, EMD 2), the total excavated area is obtained by adding both areas with their respective sign:

$$EMD = \frac{A_{T2}}{L_a} + \frac{A_{T3}}{L_b} \quad (5)$$

where A_{T2} is the area defined by 5, 6 and p_{int} (Fig. 5, EMD 2); A_{T3} is the area defined by 7, 8 and p_{int} (Fig. 5, EMD 2); p_{int} is the intersection point between the excavated profile and the line joining two adjacent blastholes; L_a and L_b are the distances between p_{int} and the mid-spacing point between the current blasthole and the adjacent one.

The scanner profiles and the EMD values per blasthole are evaluated at 0.2 m intervals. The MWD sample interval is 0.1 m and the collaring chainage of each contour blasthole differs due to irregularities of the free face. Thus, the actual chainage of the MWD logs of each hole varies so that the position of the measurements recorded by the MWD system does not coincide with the depth of the calculated EMD values. A piecewise cubic Hermite interpolating polynomial (Fritsch and Carlson, 1980) is used to interpolate the EMD values at the specific depths of the MWD logs.

As an example, Fig. 6a shows the area of influence of each blasthole (AT1 or AT2 + AT3 in Fig. 5), calculated every 0.2 m for a round between the chainages 398 and 406 m; over- and under-excavated areas are plotted in different colors and the contour blastholes are marked in black. Fig. 6b represents the EMD values for one of the blastholes in graph a (that includes the backwards and forwards extensions, as explained above) versus chainage. It shows the EMD calculated (black dots), the cubic interpolation of the EMD calculated (red line), and the estimated EMD values corresponding to the MWD depths logged (blue dots).

4. MWD data processing

The response of MWD parameters is often affected by external influences different than the rock mass, such as the calibration of the

monitoring sensors, the hole length and/or the drill rig performance (Schunnesson, 1998). The control system of the jumbo, during the adjustment of the parameters while drilling, induces systematic variations in the MWD data. All together, they add uncertainty to the data that must be previously normalized in order to highlight changes in the parameters by the rock effect. This process comprises: (i) filtering out of unrealistic data, (ii) removing of the ramp-up section of the logs, (iii) correction of systematic variations in the MWD parameters such as the effect of hole length and feed pressure influence and (iv) normalization with the standard deviation to account for fluctuations in the signals. Fig. 7 shows a flow chart of the filtering and corrections applied to the raw MWD data, with the acronym used after each step.

4.1. Filtering of unrealistic values

Production data often includes unrealistic high and low performance values of the jumbo, which may lead to a wrong interpretation of the MWD data (Ghosh et al., 2015). A new filter of the data is developed before any further analysis. For that, the empirical probability distribution function of each MWD parameter is built from the complete data set values (54 blasts, comprising more than 6500 blastholes). The 95% confidence interval of MWD parameters defines the interval to retain data for the analysis. Fig. 8 shows the cumulative distribution functions (CDF) of the eight parameters, where the black dashed horizontal lines mark the 95% coverage and the vertical lines mark the corresponding limits of the parameters, given in Table 1.

4.2. Removing of ramp-up operation mode

The control system is normally based on three main operational modes (Schunnesson, 2017; Navarro et al., 2018): (i) collaring, (ii) ramp-up, both of which control the increase of the drilling pressure to minimize hole deviations, and (iii) normal drilling, which controls the performance of the parameters to optimize the operation and minimize damages to the drill system. As can be seen in Fig. 9, the feed pressure (FP) shows, initially, a sharp rise (ramp-up mode) until it reaches a pre-set threshold at which it stabilizes (normal drilling mode). Only values included in the normal drilling mode are considered for the analysis, as data in the ramp-up mode are not representative of changes in the rock mass conditions. Signals of the 8 parameters have been divided

Fig. 6. (a) Area excavated along each blasthole; (b) EMD cubic interpolation and EMD value according to the MWD depth.

Fig. 7. MWD data processing flow chart.

Fig. 8. Cumulative distribution function of the MWD parameters. Units and acronyms are given in Table 1.

Table 1

Range of reasonable values of the MWD parameters as from their 95% coverage.

Parameter	Acronym	Units	Range
Penetration rate	PR	dm/min	1–38
Hammer pressure	HP	bar	130–235
Feed pressure	FP	bar	20–80
Damp pressure	DP	bar	45–100
Rotation speed	RS	rpm	170–310
Rotation pressure	RP	bar	35–80
Water pressure	WP	bar	12–22
Water flow	WF	l/min	54–175

considering the point at which the feed pressure stabilizes. Values before this point comprise the ramp-up data set and forward values the normal drilling data set. The latter will be referred as MWD_{c1} .

4.3. Hole length and feed pressure corrections

Systematic variations generated by the drilling system and other parameters can be estimated and removed by averaging, for a large amount of data, the response of the parameters (Schunnesson, 1998; Hjelme, 2010). Schunnesson (1998), Hjelme (2010) and Ghosh (2014) described a significant hole length dependence in some parameters for percussive, rotary-percussive and Wassara water-hydraulic DTH drilling modes, respectively. This phenomenon is related to the increase of the frictional resistance between the drill rods and the blasthole walls, the reduction of the available pressure over the hammer, the reduction of the flushing efficiency with depth and the bit wear (Schunnesson, 1998). The average value of the eight MWD_{c1} parameters at every 0.1 m hole length has been calculated for the entire data from the 54 blasts

Fig. 9. Drilling operation modes in MWD signal from blasthole 71 at chainage 349. Units and acronyms meaning are given in Table 1; HL is hole length.

analyzed.

According to incidence reports, the sensors that monitor the position of boom 3 were out of calibration in most of the rounds, and the blastholes drilled by this boom had a lower lookout and lookout direction angles than recorded by the MWD files. This resulted in a systematic under-excavation in the contour profiles drilled by boom 3. The reports indicated a more intensive scaling on this side to meet the contour requirement. This results in a significant distortion of the excavated profile, hence in the over-excavated values calculated. Since the precise blasts for which boom 3 was out of calibration were not available, all data from boom 3 has been discarded for the analysis.

The correction of the hole length influence (to obtain a signal MWD_{c2} , see Fig. 7) is done by:

$$MWD_{c2}^i = [MWD_{c1}^i - MWD_{fit,HL}^i] + MWD_{fit,HL}^1; \quad \text{with } i = 1, 2, \dots, N \quad (6)$$

where i indicates each measurement in a hole log, N being the number of these. $MWD_{fit,HL}^i$ is a polynomial regression with hole length, of the average value at every 0.1 m hole length for the entire data of each MWD_{c1} parameter. $MWD_{fit,HL}^1$ is the intercept of the fit, i.e. the value at depth zero.

Fig. 10 shows the average MWD_{c1} signal (blue lines), the polynomial regression ($MWD_{fit,HL}$, green lines) and the hole length normalized average signal (MWD_{c2} , red lines), at every 0.1 m hole length for boom 1. The hole length normalization is not applied for the hammer pressure and for the water flow because there is no noticeable effect of depth in the average signal (see $Av.HP_{c1/c2}$ and $Av.WP_{c1/c2}$ in Fig. 10).

Previous work by the authors (Navarro et al., 2018) analyzed the relationship between MWD parameters to limit the number of the MWD

Fig. 10. Correction of hole length influence in MWD parameters (pass from MWD_{c1} to MWD_{c2} , see Fig. 7); blue lines: average signals, MWD_{c1} (obtained after filtering unrealistic values and removing ramp-up section, see Fig. 7); green lines: polynomial regression; red lines: average normalized signals, MWD_{c2} . Units and acronyms of the parameters in the ordinates are given in Table 1. (For interpretation of the references to color in this figure legend, the reader is referred to the web version of this article.)

variables, and to select the more significant ones that are required for a sound rock mass characterization. They did a cross-correlation analysis between the signals of the eight parameters in order to find leadership behaviors. The results pointed to the feed pressure as the lead parameter that drives the adjustment of the other variables to optimize the drilling. They showed that penetration rate, hammer pressure, damp pressure and rotation pressure are influenced by the feed pressure. According to this, the feed pressure generates systematic variations in these parameters that may hide the rock dependence on them. On the contrary, rotation speed, water flow and water pressure are little influenced by the feed pressure, thus being considered independent.

The same methodology followed for the hole length influence is now used to correct the feed pressure influence. The average value of the seven MWD_{c2} parameters (feed pressure is not included) is calculated at steps of 1 bar feed pressure value for the 54 blasts for boom 1. Similar to Eq. (6), the correction of the feed pressure influence (to obtain a signal MWD_{c3} , see Fig. 7) is done by:

$$MWD_{c3}^i = [MWD_{c2}^i - MWD_{fit,FP}^i] + MWD_{fit,FP}^1; \text{ with } i = 1, 2, \dots, N \quad (7)$$

where i indicates each measurement in a hole log, N being the number of these. $MWD_{fit,FP}$ is a polynomial regression with the feed pressure, of the average value at every 1 bar feed pressure for the entire data of each MWD_{c2} parameter. $MWD_{fit,FP}^1$ is the intercept of the fit, i.e. the value at the minimum feed pressure.

Fig. 11 shows the average MWD_{c2} signal (blue lines), the polynomial regression ($MWD_{fit,FP}$, green lines) and the feed pressure corrected average signal (MWD_{c3} , red lines), as function of the feed pressure. In line with Navarro et al. (2018), it can be seen that penetration rate (PR), hammer pressure (HP), damp pressure (DP) and rotation pressure (RP) parameters have a strong dependence from the feed pressure (FP). For the case of rotation speed (RS), water pressure (WP) and water flow (WF), the influence of feed pressure is considerably less, and these data are not normalized for the subsequent analysis; Fig. 11 shows no large differences between MWD_{c2} and the resulting MWD_{c3} signals for these parameters.

Fig. 11. Correction of the feed pressure influence in MWD parameters (MWD_{c2} conversion to MWD_{c3} , see Fig. 7); blue lines: average signals, MWD_{c2} (corrected for hole length influence, see Fig. 7); green lines: polynomial regression; red lines: average normalized signals, MWD_{c3} . Units and acronyms of the parameters are given in Table 1. (For interpretation of the references to color in this figure legend, the reader is referred to the web version of this article.)

4.4. Analysis of fluctuations in the MWD signals

Schunnesson (1996, 1997) claimed that when discontinuities in rock are drilled, penetration rate and rotation pressure show significant fluctuation, resulting in a noisy signal. Following this reasoning, parameters involved in the rotational mechanism of the jumbo (rotation pressure and rotation speed) and the penetration rate have been processed. The procedure is carried out for the signal of each hole individually, where the MWD_{c3} values in a hole are divided by the standard deviation of the entire signal of that hole. The resulting signals for these three parameters are MWD_{c4} .

$$MWD_{c4}^i = \frac{MWD_{c3}^i}{std(MWD_{c3})}; \text{ with } i = 1, 2, \dots, N \quad (8)$$

where i indicates each measurement in a hole log, N being the number of samples per signal.

5. Detection of potential overbreak zones

The processed penetration rate (PR_{c4}), hammer pressure (HP_{c3}) and rotation pressure (RP_{c4}) result in rock dependent parameters. The normalized rotation speed (RS_{c4}) and water flow (WF_{c3}) are independent parameters sensitive to rock variations (Navarro et al., 2018). The response of these normalized parameters can detect variations in the rock and for equal blasting conditions they may explain variations in EMD data. The feed pressure is not considered as it has been used during the normalization.

The lookout distance (i.e. distance from the collaring to the position of the hole in the XY plane at each depth, see Fig. 2) is also considered for the analysis as it may reflect the confinement effect by depth. Fig. 12a shows, as an example, variations of EMD with the hole length and lookout distance for the holes of the blast located at the chainage 500. As can be seen, there is a negative influence of the lookout in the EMD which means that the excavated area in relation with the blasthole position decreases with the lookout distance; this EMD relation with lookout distance is the same as with hole length since the lookout distance increases with depth. An increase of confinement with lookout

Fig. 12. Preselection of the data for blastholes of boom1 at the chainage 591; (a) EMD vs. hole length and lookout distance; (b) MWD signals for blastholes 6, 7 and 11. See Table 1 for the acronyms of MWD parameters; subscripts c_3 and c_4 refer to the corrections shown in Fig. 7.

distance and depth is associated with an increase of the difficulty in breaking the rock, resulting then in a decrease of the over-excavated area, hence the EMD, which may be even negative (under-excavation). Therefore, the larger the lookout distance and the deeper the drilling, the more under-excavation is created in relation with the position of the contour blastholes. In some cases (data for holes 6, 8 and 11, Fig. 12a), the general trend is not followed. Fig. 12b shows the respective MWD $_{c3}$ and MWD $_{c4}$ logging for holes 6, 8 and 11. One may think that local rock conditions may cause such results and that MWD parameters should show a peak or some kind of variation in the signal. This is effectively observed for hole 6, where MWD $_{c3-c4}$ parameters show a significant peak or signal fluctuation at depth 3–4 m, in line with the over-excavation peak represented in graph a for this hole. However, no distinctive variation appears in the MWD $_{c3-c4}$ records of holes 8 and 11. Hole 8 shows an over-excavated section at 2.5–3.5 m, whereas hole 11 indicates under-excavation at depth 1–2 m, though no significant changes can be observed in the MWD signal of these two holes at any depth. Other causes like drill deviations, malfunctioning of blasting, scaling, etc. may be behind such EMD outlier profiles, uncorrelated with the MWD logs; these outliers, that happen at most in one or two holes per blast (many blasts do not show any) are removed from the analysis.

Table 2 shows the statistics for the MWD $_{c3}$ and MWD $_{c4}$ parameters, the lookout distance, and the EMD values for boom 1.

A power function of the MWD parameters to predict the EMD is considered:

$$\text{EMD} = A_0 + \text{PR}_{c4}^{A1} \cdot \text{HP}_{c3}^{A2} \cdot \text{RS}_{c4}^{A3} \cdot \text{RP}_{c4}^{A4} \cdot \text{WF}_{c3}^{A5} \cdot \text{L}_{\text{dist}}^{A6} \quad (9)$$

where L_{dist} is the lookout distance.

Since EMD has positive and negative values, an additive constant A_0 has been included. The normalized damp pressure and water pressure have been removed since their contributions were found minimal. The non-linear regression model has been programmed with Matlab 2016b by a Levenberg-Marquardt non-linear ordinary least squares method (Matlab, 2016). The model coefficients are given in Table 3.

The determination coefficient of the fit R^2 is 0.74. Coefficient values

Table 2

Statistic of the MWD $_{c3}$ and MWD $_{c4}$ parameters for boom 1.

Parameter	Units	Mean	Std. ^a	Min.	Max.
Penetration rate $_{c4}$	m/min	13.01	9.40	0.04	57.10
Hammer pressure $_{c3}$	bar	211.81	15.04	63.00	230.0
Feed pressure $_{c3}$	bar	73.94	5.71	21.20	79.89
Damp pressure $_{c3}$	bar	65.31	12.76	48.23	86.63
Rotation speed $_{c4}$	rpm	34.73	30.91	0.18	98.58
Rotation pressure $_{c4}$	bar	26.55	12.99	0.09	128.05
Water flow $_{c3}$	l/min	81.99	28.20	53.70	145.56
Water pressure $_{c3}$	bar	16.43	4.87	13.57	21.84
Lookout	m	0.27	0.15	0.02	0.94
EMD	m	0.25 ^b	0.30	−0.63	1.09

^a Std. is standard deviation.

^b Mean absolute Value.

estimated for each parameter are significantly different from zero, i.e. their p-value is < 0.0001 for all cases, which means that the analysis is statistically significant. According to the value and sign of the coefficients, the MWD parameters affect EMD differently:

- The normalized penetration rate (PR_{c4}) and rotation pressure (RP_{c4}) show negative correlation (negative sign in A_1 and A_4) with EMD. According to Schunnesson (1996, 1997), high fluctuations in the PR and RP signals indicate soft/fractured rock. High fluctuations mean high standard deviation so that, according to Eq. (8), PR_{c4} and RP_{c4} show low values. This means that low PR_{c4} and RP_{c4} values indicate soft/fractured rock, prone to suffer over-excavation. The contrary is also true.
- The normalized hammer pressure (HP_{c3}) also shows a negative correlation with EMD (negative sign in A_2), which indicates that, when discontinuities or soft rock is drilled, the hammer of the boom needs a lower pressure to hit the rock. As explained by Peng et al. (2005) and Navarro et al. (2018), when open fissures or softer rock are found, the feed pressure shows a sharp drop to the level of drilling the air. This behavior is also noticed in the hammer

Table 3
Non-linear regression model coefficients.

	A_0	PR_{c4} A_1	HP_{c3} A_2	RS_{c4} A_3	RP_{c4} A_4	WF_{c3} A_5	L_{dist} A_6
Coefficient	−2.8432	−0.0341	−0.1188	0.0352	−0.0108	0.0834	−0.0600

pressure; in open fissures, there is no resistance for the hammer to hit the rock which results in a reduction of its pressure to avoid damage in the system. In this way, low hammer pressures values are related with over-excavated areas and vice versa.

- The normalized rotation speed (RS_{c4}) shows positive correlation with EMD (positive sign in A_3). Schunnesson (1998) claimed that at high feed pressure, the rotation pressure required for the rotation of the bit increases and sometimes a reduction in the rotation speed can be appreciated. The higher resistance to rotate may also translate into an increase in the signal fluctuation. According to Eq. (8), the higher the standard deviation of the RS parameter, the lower the RS_{c4} and the lower the excavated area is created, hence the positive sign of its exponent in Eq. (9). For the case of discontinuities, Hustrulid (1968) showed that at low feed pressure, the bit will not be in constant contact with the bottom of the hole, resulting in a free rotation of the bit that may show a lower fluctuation in the rotation speed. This way, the lower the standard deviation of the RS parameter, the higher the RS_{c4} and the greater the excavated area.
- The normalized water flow (WF_{c3}) has a positive correlation with EMD (positive sign in A_5). This can be explained by the fact that, when discontinuities or soft rock is drilled, the control system requires a higher water flow due to the higher amount of drill cuttings or the water leakage to the discontinuities.
- The lookout distance presents a negative correlation with EMD, which means, as discussed before, that the excavated area in relation with the blasthole position decreases with an increase of depth and lookout.

The results of the EMD predicted with the suggested model versus the EMD data are plotted in Fig. 13a. The linear regression obtained has a slope of one with a zero-constant term. Fig. 13a also shows the upper

and lower prediction band at a 95% confidence level. The residuals are normally distributed (see Fig. 13b).

Fig. 13c shows the distribution of the root mean square error (RMSE) for each of the 54 blasts. The median and the 25th and 75th percentiles of RMSE are 0.142 m, 0.102 m and 0.186 m, respectively. An illustration of the application of the model is shown in Fig. 14 for three blasts, representing the 75% (large error, Fig. 14a), 50% (expected error, Fig. 14b) and 25% (small error, Fig. 14c) of the RMSE values; they correspond to blasts located at the chainages 388, 500 and 591, respectively. In the three cases, the EMD predicted is compared with the EMD measured for blastholes of boom 1. Five different over-excavation ranges have been defined. For high RMSE values (Fig. 14a), visual differences between the predicted EMD and the EMD measured are apparent; these are considerably reduced when representing medium and low RMSE values (Fig. 14b and c). Fig. 14c shows a predicted EMD generally in line with the EMD measured though still some light differences exist. Considering the noise of the MWD data – unavoidable in the harsh conditions where such data are measured – and the additional uncertainty brought by drilling deviations, the scaling and primary support done before scanning the excavated section, sensors potentially out of calibration, possible variations in the explosive linear density, etc., the quality of the fit is outstanding.

6. Conclusions

The overbreak of the remaining rock mass in tunnel blasting has been analyzed in the light of MWD records, with the purpose of developing a prediction model of over- and under-excavation depths from blasting. Such predictive model may also be seen as a drill or rock index that could be used to identify zones of potentially high geotechnical risk (those for which the over-excavation prediction is high). By comparison

Fig. 13. Result of the EMD predicted model: (a) predicted versus measured EMD values (upper graph: residuals); (b) Box-plot of the RMSE values per blast.

Fig. 14. Representation of the predicted EMD model (right graph) and the EMD measured (left graph) for blastholes of the boom 1: (a) blast in chainage 388, 75th percentile of the RMSE; (b) blast in chainage 500, median value of the RMSE; (c) blast in chainage 591, 25th percentile of the RMSE.

of scanner profiles of the excavated sections with the blasthole positions, a methodology has been developed to obtain an Excavated Mean Distance (EMD) between the blasthole and the excavated profile, which

may be considered a damage measure. Sources of uncertainty such as drilling deviations, the scaling and primary support done before scanning the excavated section, possible variations (unrecorded) in the

explosive linear density, etc., are assumed to be of random nature, unavoidable in the harsh condition in which such data are measured. Quantitative predictions for different conditions would require a re-calibration of the model for the new site, following the methodology described here.

Given that blasting factors are constant (as from blast reports) the overbreak and underbreak are considered mainly influenced by the geotechnical condition of the rock mass. Such rock mass properties are assessed from MWD parameters that show the response of the jumbo to the rock before blasting. A thorough transformation of the MWD logs has been carried out to filter out systematic variations due to the nature of the drilling process and to highlight the dependence of the rock in the parameters. This transformation includes: (i) filtering out of unrealistic data, (ii) removal of the ramp-up section of the logs, (iii) normalization of systematic variations in the MWD parameters so that the influence of hole length and feed pressure are corrected and (iv) normalization with the standard deviation to account for fluctuations in the signals.

A non-linear power-form model has been developed that predicts the excavated mean distance as function of the normalized penetration rate, hammer pressure, rotation speed, rotation pressure and water flow parameters, and the lookout distance. They combine the rotational, hydraulic and percussive mechanisms of the drill, and the confinement of the rock mass by depth. The model has a determination coefficient of 0.74, with the coefficients of the model strongly significant. Residuals are essentially normally distributed. The signs of the exponents indicate that normalized penetration rate, hammer pressure, rotation pressure and the lookout distance inversely influence the excavated distance i.e. high values for them reflect hard, unaltered rock. On the other hand, normalized rotation speed and water flow are directly correlated with the excavated distance, so that high values for them indicate soft, fractured rock.

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Appendix A. Supplementary material

Supplementary data associated with this article can be found, in the online version, at <https://doi.org/10.1016/j.tust.2018.08.060>.

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APPENDIX E. Paper E

Assessment of drilling deviations in underground operations

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Assessment of drilling deviations in underground operations

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ABSTRACT

At present, modern jumbos equipped with MWD (Measure While Drilling) provide the position of the blastholes collar and, from the drill length and the azimuth and inclination angles (monitored outside the blasthole), the theoretical end position of the blasthole. Since the trajectory of the hole during the drill is not measured, deviations are not accounted with the result that the actual spatial position of the blasthole is unknown. This paper investigates the quality of the drilling in underground blasting operations with a view to quantify the distance of the position assessed by the MWD system with respect to the actual end position of the blasthole logged. For that, a Pulsar Micro Probe Mk3 has been used to measure the actual trajectory of several production blastholes, drilled in semi-automatic mode, by measuring inclination and azimuth values at 1 m intervals. The results indicate significant deviations between the actual end position of the blastholes and the end position given by the MWD system. Deviation measurements are compared with the MWD parameters. This points out that possible disturbance zones in the rock, indicated by peaks and drops in the variability of the rotation pressure and in the signals of feed and hammer pressure, are correlated with changes in the blasthole trajectory.

1. Introduction

Drilling and blasting is a well-known technique used for rock excavation in tunnelling or underground civil works. In it, drilling operation is one of the critical stages of the overall excavation process with major influence in the efficiency of the next stages, such as blasting, scaling, loading, hauling and support operations. In underground constructions, drilling is normally carried out by top hammer rotary-percussive jumbo drills. Generally, these jumbos bring the possibility to monitor the information of sensors installed along the mechanisms of the feeder and the boom to record digital signals of the parameters involved in the operation. The Measurement While Drilling (MWD) technique is a drill monitoring system that collects operational drilling data at predetermined length intervals along the blasthole (Schunnesson, 1997). This technology allows to log not only real time information of the rock mass condition while drilling, but it also brings the possibility to record the collaring position of the blasthole and, from the length of the drill rod and the azimuth and inclination angles of the boom, its theoretical end position inside the rock mass.

Nowadays, modern jumbos like those manufactured by Atlas Copco, Sandvik and AMV include drilling automation, e.g., ABC (Advanced Boom Control), iREDES and Bever Control, respectively, to optimize drilling. As an example, the drill rig of jumbos manufactured by Atlas Copco jumbos has three operational levels of automatization: basic,

regular and total (Nord and Appelgren, 2001; Atlas Copco, 2010; Östberg, 2013; Navarro et al., 2018a). The basic level authorizes only the manual performance of both positioning and drilling. The regular level allows to follow a predesigned drill plan with manual control of the boom whereas drilling is automatic. The total level enables the use of a predesigned drill plan and to switch collaring and drilling between manual (manual collaring and drilling), semi-automatic (manual collaring - automatic drilling) and fully-automatic (automatic collaring and drilling) mode.

Based on the MWD records, manufacturers have developed their own software (Tunnel Manager - Atlas Copco, iSURE - Sandvik and Bever Control in cooperation with AMV) as a tool for planning, management and evaluation of drill parameters. From the MWD records, blastholes can be represented in 3D maps and rock properties such as hardness and fracturing are inferred and displayed (Atlas Copco, 2017; Sandvik, 2017; Bever Control, 2017). Although these software packages show the position of the blastholes, this is only a theoretical representation since the actual position is not measured. Four variables mainly influence deviations during the drilling (Östberg, 2013): (i) setting out, (ii) collaring and alignment, (iii) drill rod deflection and (iv) rock structure.

Blasthole positioning is monitored from sensors (inclinometers-accelerometers) installed along the boom and thus, outside the blastholes. Since the boom remains still during the drill, it normally measures

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constant values of its direction. However, the actual path inside the rock is not measured, due to neither the drill rod nor the bit are equipped with any sensor. This suggests an unknown error of the bottom hole position in relation to the design position (given by the MWD system). Olsson (2010) analysed this error for the contour blastholes by measuring, with a total station, the end position of the half cast contour holes when they were visible. He determined a mean deviation value of 11.6 ± 6.8 cm (mean $\pm$ standard deviation). To the authors knowledge, no additional data on deviations of production blastholes in tunnelling has been published.

Drill deviations may generate a non-uniform explosive charge concentration: excessive proximity between two blastholes may increase the specific charge (or volume charge concentration) in this zone, whereas a higher distance between them may reduce it, resulting in problems with rock breakage, fragmentation and pull. They may also induce problems in the perimeter excavated. Outwards deviations create an excessive over-excavated zone generating short-term stability problems around the perimeter of the excavation. This also increases production costs due to the retrieval of the extra material blasted and the need of a sturdier primary supports. Inwards deviations cause under-excavation zones in the perimeter, requiring a more intensive scaling to fulfil the pay-line requirements.

There is a large number of studies focused on the geological and geo-mechanical interpretation of the MWD (Teale, 1965; Scoble et al., 1989; Schunnesson, 1996, 1997, 1998, 2011; Schunnesson and Kristoffersson, 2011; Liu and Karen Yin, 2001; Kahraman et al., 2016; Peng et al., 2005; Tang, 2006; Hjelme, 2010; Naeimipour et al., 2014; Hatherly et al., 2015; Leung and Scheding, 2015; Ghosh, 2017); however, the effect of the rock structure, determined through the MWD data, on the drilling deviation has not been studied.

A thorough description of the jumbo navigation and positioning is presented, in order to get a deeper understanding of the different stages required for an accurate drilling. In order to assess the hole deviation, a Pulsar Micro Probe Mk3 has been used to measure the actual end position of five production blastholes, drilled with an Atlas Copco jumbo, by measuring inclination and azimuth values at 1 m intervals of the trajectory followed by the blasthole. The results are compared with the end position provided by the MWD system and the error between this and the actual end position is estimated. Deviation measurements are also compared with MWD parameters in order to assess the influence of the rock structure in the drilling path.

2. Jumbo navigation and blasthole positioning

Jumbo navigation is the first operation before starting a new round to follow the correct tunnel layout. The operation is carried out in two steps: jumbo positioning and alignment with the tunnel axis.

The first operation consists of measuring the exact position of the jumbo inside the tunnel to create an absolute coordinate system. For this purpose, the jumbo has a laser scanner on its front side. The position of the jumbo is calculated by trilateration, measuring distances from the laser scanner to target points (with known coordinates) located along the wall side of the tunnel (Fig. 1a). The absolute coordinates X_{abs} , Z_{abs} obtained are given, in this case under study, in the EUREF 89 Norwegian Transverse Mercator (NTM) projection and the absolute coordinate Y_{abs} is measured as the height above sea level.

The second operation consists on aligning the drill rig with the tunnel axis (i.e. the perpendicular line to the face of a new round). The laser scanner points to the free face in the direction of the tunnel axis and two targets are aligned along one of the booms of the jumbo. For the alignment, the boom rotates until the laser beam passes through both targets (Fig. 1b). At this stage, the drill rig creates a tunnel reference system ($\vec{x}_t$, $\vec{y}_t$, $\vec{z}_t$) with one axis parallel to the tunnel axis and the other two in the plane of the tunnel free face (Fig. 1b). Finally, the chainage (ch) inside the tunnel of the new round is measured as the distance from the laser scanner to the free face. This ch is taken as

reference plane of the collaring depth position of the blastholes. Negative depth values are assigned to measures behind this plane and positive values, to measures ahead of this plane (Navarro et al., 2018b).

The former operations make the jumbo to be oriented by three angles (γ , θ , ω), according to the horizontal ($\vec{x}_t$, $\vec{z}_t$) and vertical ($\vec{z}_t$, $\vec{y}_t$) directions of the tunnel axis and the ($\vec{x}_t$, $\vec{y}_t$) rotation of the free face, respectively. To calculate the position of the blastholes, the drill rig rotates the planes formed in the tunnel reference system (X_tZ_t , Y_tZ_t , X_tY_t), to create a drilling reference system defined by two vertical planes Y_dZ_d , X_dZ_d and a horizontal X_dY_d plane, as correction of the jumbo orientation by angles θ , ω and γ , respectively. Fig. 2 shows the rotation carried out by the drill rig over axes $\vec{y}_t$ (left graph), $\vec{x}_t$ (centre graph) and $\vec{z}_t$ (right graph) to define the drilling reference system of axes ($\vec{x}_d$, $\vec{y}_d$, $\vec{z}_d$). Graphs in red correspond to horizontal (X_dZ_d) and vertical (Y_dZ_d , X_dY_d) planes in the drilling reference system and graphs in black to the rotated planes (X_tZ_t , Y_tZ_t , X_tY_t) according to the tunnel axis or free face orientation.

Blasthole position measured in the drilling reference system is defined by three spherical coordinates (see Fig. 3): blasthole length (l_b), azimuth (angle of the horizontal $\vec{x}_d$ axis and the hole projection in the X_dY_d plane) or lookout direction (L_D) and inclination or lookout angle (L_I). The two later are logged by sensors installed along the boom outside the blasthole and the blasthole length (l_b) corresponds to the drill rod length introduced in the hole. The inclination angle varies between 0 and 90° both for holes drilled upwards or downwards so that the azimuth is between 0 and 180° for holes drilled upwards and between 0 and -180° for holes drilled downwards. Fig. 3 shows the theoretical (X_F , Y_F , Z_F) projections of a blasthole (drilled downwards) in the drilling reference system; they are (Navarro et al., 2018b):

$$X_F = l_b \cdot \sin(L_I) \cos(L_D) \quad (1)$$

$$Y_F = l_b \cdot \sin(L_I) \sin(L_D) \quad (2)$$

$$Z_F = l_b \cdot \cos(L_I) \quad (3)$$

Once the boom is placed in the required position and before starting to drill, the drill rig registers, in the drilling reference system, the collaring position of the blasthole and the azimuth (L_D) and inclination (L_I) angles of the boom (see Fig. 3). The end coordinates of the blasthole are theoretically calculated by adding, to their collaring coordinates, the result from Eqs. (1)–(3).

3. Data overview

The study has been developed in the underground extension work of the municipal wastewater treatment plant in Oslo, Norway. The facility is composed of five caverns, a main access gallery of about 850 m length and other sections. The construction was excavated by drill and blasting in competent rock mass, composed by gneiss with small tonalite and quartzite intrusions. This study is focused in the fifth cavern of 460 m² section (20 m width $\times$ 23 m height) and around 180 m length. The excavation of the cavern was divided in two sections, being the upper one (20 m width $\times$ 10 m height) firstly developed by horizontal parallel holes with an advance of 4.5 m/round and secondly the bottom section (20 m width $\times$ 13 m height) by vertical holes through bench blasts. The excavation of the upper section was done by dividing the blasting front in two sections, right and left (10 m width $\times$ 10 m height, each), following the next drilling and blasting procedure: three blasts in the left side of the section and next, three blasts in the right side to start over the cycle.

Drilling was carried out by a three-boom jumbo XE3C, manufactured by Atlas Copco, with percussive-rotary top hammer drilling mechanism, working in semi-automatic ABC total system. Data comprises production face blastholes of short length (5–5.5 m), drilled by using only one rod of 5.5 m length and 38 mm diameter and a bit of 46 mm diameter. A 3.18 revision control system (RCS) was installed in

Fig. 1. Jumbo navigation; (a) Jumbo positioning; (b) Jumbo alignment with tunnel axis (tunnel reference system) and measurement of chainage (ch) of a new round.

Fig. 2. Transformation from drilling reference system ($\vec{x}_d, \vec{y}_d, \vec{z}_d$) to tunnel reference system ($\vec{x}_t, \vec{y}_t, \vec{z}_t$). Left: Rotation towards right and left, or bearing angle, over $\vec{y}_d$; center: uphill- downhill inclination, or elevation angle, rotation over $\vec{x}_d$; right: lateral inclination, roll or bank angle, rotation over $\vec{z}_d$ (Navarro et al., 2018a, 2018b).

Fig. 3. Drilling reference system for blasthole positioning (hole direction in green).

the jumbo. Eight parameters are monitored while drilling by the percussive-rotary Atlas Copco jumbos to gather information at a defined hole depth sample interval (m): penetration rate (m/min), hammer pressure (bar), feed pressure (bar), damp pressure (bar), rotation speed (rpm), rotation pressure (bar), water pressure (bar) and water flow (l/min). They have been described in detail by Peng et al. (2005), Beattie (2009), Hjelm (2010), Schunnesson et al. (2011) and Navarro et al.,

(2018b). Other parameters, such as collaring position, lookout angle (°) and lookout direction angle (°) are also measured. The two later show the inclination and azimuth angles, respectively, of the orientation of the boom before the drilling starts.

The assessment of the drilling quality has been done with a Pulsar Micro Probe Mk3, manufactured by geo-koncept, of 0.037 m diameter and 0.30 m length. The probe works as an inertial measure system that

Fig. 4. Measured blastholes (highlighted in red) in blasts of the upper right side of cavern 5: (a) ch 129.5; (b) ch 133. (For interpretation of the references to colour in this figure legend, the reader is referred to the web version of this article.)

logs data registered by an accelerometer, a compass and an inclinometer, each one oriented in one of the 3D axes. This allows to measure the actual path of pseudo sub-horizontal production blastholes.

For the measurement of drill deviations, the probe is introduced in the blasthole and it is pushed gradually inside the hole with a steel bar, with a rubber on its extreme. The probe records at every 1 m sample interval (there are marks in the wire at every 1 m length), in exception of the first measurement, that corresponds to a length of 0.865 m, included the length of the probe. The azimuth (horizontal angle) and the inclination (vertical angle) are also logged and they are measured with respect to the magnetic North and the horizontal plane, respectively. The accuracy of the azimuth and inclination is $\pm 1^\circ$ and $\pm 0.25^\circ$, respectively. The magnetic declination of the day when the measures were taken (22-12-2015) at Oslo, Norway, is $2^\circ 56.76'E$; this must be removed from the data gathered to pass from magnetic to geographical North.

Five blastholes were measured from two blasts located on the top right section of the fifth cavern at chainage (ch) 129.5 m and 133 m. The collaring position of these blastholes are shown in Fig. 4; all blastholes were drilled by boom 1 except blasthole 32 (ch 129.5), that was drilled by boom 3. Table 1 shows the nominal values of lookout angle and lookout direction from the MWD records.

4. Data analysis and results

4.1. Drilling deviations

The trajectory of the blasthole is calculated for both MWD data and the probe by using Eqs. (1)–(3). For the probe, L_i and L_D correspond to the inclination and azimuth angles, respectively, and l_b is the length of wire introduced inside the blasthole at each measure.

To transform data obtained with the probe into the drilling reference system, the complete trajectory logged by the probe is rotated to match the first measurement, 0.865 m, with the trajectory direction of the MWD data in the drilling reference system. For that, a Matlab code (Matlab, 2017) has been developed. This presupposes a correct collaring and alignment of the bit and the drill rod for the first measurement of the probe in the blasthole, which may likely occur due to the information provided by the MWD to the jumbo operator.

Table 1

Lookout angle and lookout direction from the MWD records for ch 129.5 and 133, cavern 5.

Chainage	129.5			133	
Blasthole	75	76	32	76	77
Lookout angle ($^\circ$)	7.4	5.6	4.8	1.2	3.2
Lookout direction ($^\circ$)	−157.9	166.6	−30.1	−159.4	−173.2

Figs. 5 and 6 show the comparison of the trajectory followed by the five blastholes as from the MWD data (i.e., theoretical) and the probe records for rounds located in the ch 129.5 (blastholes 75-graph a, 76-graph b and 32-graph c) and ch 133 (blastholes 76-graph a and 77-graph b). For each hole, three graphs are included, corresponding to the projection of the trajectories in the X_dY_d plane (upper-left side), X_dZ_d plane (right side) and Y_dZ_d plane (bottom-left side). The actual trajectory measured by the probe is coloured different for each hole (dots are the position of the measures taken); the black line represents the trajectory given by the MWD system and the black dot the collaring of the blasthole. As can be seen, the blasthole length in the probe measurements is always shorter than the maximum length given by the MWD. This variation may be caused by the difficulty to introduce the probe inside the blasthole, especially in the final meters, probably due to sharp changes in the actual hole path or to the blockage of the blasthole after drilling.

Table 2 shows the absolute distance between the end position given by the probe and the respective position obtained from the MWD data at the same hole depth, for the horizontal (X_dZ_d plane) and vertical (Y_dZ_d plane) projections, and for the spatial distance (distance in the 3D). These values are expressed as the distance with respect to the maximum probe length measurement in m/m. The mean value and the standard deviation are also given. In the vertical plane (Y_dZ_d , Figs. 5 and 6), trajectories for blastholes 75 and 76 (ch 129.5) and for blasthole 76 (ch 133) show upwards deviation, and trajectories for blasthole 32 (ch 129.5) and blasthole 77 (ch 133) indicate downwards deviation, with a mean absolute value of 0.07 ± 0.03 m/m (mean and standard deviation, Table 2). An interesting result occurs for blasthole 75 (Fig. 5), where the blasthole results in a final upwards lookout instead of the designed downwards. Considering the horizontal plane (X_dZ_d , Figs. 5 and 6), deviations always occur in the direction of the theoretical MWD orientation with a mean absolute deviation value of 0.05 ± 0.02 m/m (Table 2). The X_dY_d plane also shows differences in the actual blasthole trajectory direction in relation to the one given by the MWD system, with a mean absolute value of 0.08 ± 0.03 m/m. The mean absolute error in the 3-D is 0.11 ± 0.03 m/m.

4.2. Comparison of drilling deviations with MWD data

The comparison between MWD parameters and deviation measurements will improve the understanding of the drill/rock interaction, showing the impact of the rock structure in the drillhole path. For that, the magnitude and the fluctuation of the MWD signals are considered. Fig. 7 shows the eight MWD parameters for the five blastholes analysed; signals for each blasthole are coloured in line with their respective actual trajectory in Figs. 5 and 6. According to the MWD parameters description (Peng et al., 2005; Beattie, 2009; Hjelme, 2010; Schunnesson et al., 2011; Navarro et al., 2018a, 2018b), parameters mainly controlled by the drilling control system are (i) the feed

Fig. 5. Blastholes trajectory for round in ch 129.5 m, cavern 5. Projection in the X_dY_d plane (plots at the top-left side), X_dZ_d plane (plots at the right side) and Y_dZ_d plane (plots at the bottom-left side).

Fig. 6. Blastholes trajectory for round in ch 133 m, cavern 5. Projection in the X_dY_d plane (plots on the top-left side), X_dZ_d plane (plots on the right side) and Y_dZ_d plane (plots on the bottom-left side).

Table 2

Deviation measurements at the depth of the probe data.

Chainage	Blasthole	MWD length (m)	Max. probe measured length (m)	Absolute deviation in X_dZ_d plane (m/m)	Absolute deviation in Y_dZ_d plane (m/m)	Absolute deviation in X_dY_d plane (m/m)	Absolute deviation in 3-D (m/m)
129.5	32	5.50	4.00	0.04	0.05	0.06	0.10
	75	5.50	5.02	0.05	0.11	0.12	0.14
	76	5.50	4.40	0.04	0.09	0.10	0.13
133	76	5.50	4.87	0.07	0.06	0.09	0.12
	77	5.50	4.12	0.03	0.03	0.04	0.07
Mean $\pm$ std.				0.05 $\pm$ 0.02	0.07 $\pm$ 0.03	0.08 $\pm$ 0.03	0.11 $\pm$ 0.03

Fig. 7. Representation of the MWD parameters for the 5 blastholes analysed; where, BH is blasthole and ch is chainage, HD is hole depth, PR is penetration rate, HP is hammer pressure, FP is feed pressure, DP is damp pressure, RS is rotation speed, RP is rotation pressure, WF is water flow and WP is water pressure.

pressure, defined as the thrust to keep the bit in contact with the bottom of the hole, (ii) the hammer pressure or impact pressure of the bit against the rock and (iii) the rotation speed that quantifies the number of revolutions of the bit per minute. The rotation pressure or torque and penetration rate, on the other hand, are considered a drilling response to rock conditions. As can be seen, hammer pressure, rotation speed and rotation pressure signals of blasthole 32, ch 129.5 have a different magnitude range than signals for the other holes. These values might be related with a different set-up of the boom 3 sensors.

The behaviour of the MWD parameters while drilling through possible disturbance zones has been investigated: Peng et al. (2005) determined the feed pressure and thus hammer pressure as good indicators for void detection and Navarro et al. (2018a) identified the feed pressure as the parameter that controls the adjustment of the other parameters when limiting values while drilling are exceeded. In addition, Schunnesson (1996, 1997) and Ghosh (2017) claimed that when discontinuities are encountered during the drilling, the rotation pressure shows significant fluctuation, resulting in a noisy signal. These variations are highlighted and calculated as the sum of the residuals over a defined interval along the blasthole (Ghosh, 2017):

$$[RP_{var}]_i = \sum_{j=i}^{i+N} |\bar{RP} - RP_j| \text{ with } i = 1, 2, \dots, L \quad (4)$$

where RP_{var} is the variability of the rotation pressure; N is the window size and is considered 4 (Ghosh, 2017); i is the sample number; L is the length of the RP signal; RP_i is the monitored parameter at the i sample; and $\bar{RP}$ is the moving average of the RP values in the window between i and $i + N$ samples.

Based on Schunnesson (2017), Navarro et al. (2018a) claims that the control system has three distinct operational modes: (i) collaring, (ii) ramp-up, that controls the increase of the drilling pressure to obtain a correct collaring and alignment, and (iii) normal drilling, that adjusts the performance of the parameters during the operation. In this case, the feed pressure (FP) normally shows an initial sharp rise (ramp-up mode) until it reaches a pre-set threshold at which it stabilizes (normal drilling mode, Fig. 7), being data in the ramp-up mode not representative of the rock conditions (Navarro et al., 2018b). Fig. 8 represents the variability of the rotation pressure (Eq. (4)) for each one of the five blastholes analysed, considering only values of the normal drilling mode.

The analysis of Figs. 7 and 8 outlines a certain relation between the MWD parameters and the actual blastholes path. Blasthole 32, ch 129.5 (Fig. 5a) shows the second smaller deviation values (see Table 2). MWD parameters for this hole indicate a lower hammer pressure and rotation pressure and a higher rotation speed. These values might be related with low deviations at the beginning of the drilling due to a lower pressure over the drill rod during the operation. Variability in the

rotation pressure signal at 3.5–4 m corresponds to an abrupt change in the actual trajectory from a depth of 3 m.

Blasthole 75, ch 129.5 (Fig. 5b) has the highest deviations (see Table 2). In this case, the MWD parameters magnitude is similar to other blastholes, with high hammer pressure values (Fig. 7). The large deviation can be explained by fluctuations on its signal. In Fig. 7 (yellow signal), the feed and hammer pressure parameters show a drop between 0.5 and 1 m depth; this is also appreciated as a peak in the RP variability (Fig. 8). This indicates a possible discontinuity that may have changed the path of the blasthole at the beginning of the drilling. Another possible disturbance zone between 2.5 and 3.5 m depth may be the reason of a new sharp change in the blasthole path deviation at 3 m depth.

Blasthole 76, ch 129.5 (Fig. 5c) also shows large deviations (see Table 2). Variability in the rotation pressure signal at 3–3.75 m (Fig. 8) corresponds to an abrupt change in the actual trajectory of the blasthole between 3 and 4 m depth, similarly as in blasthole 32 ch 129.5.

Blasthole 76, ch 133 trajectory represents a good alignment with the MWD path until 2 m depth (Fig. 6a). This is related with the ramp-up drilling mode (low feed and hammer pressure values) until 1.2 m depth. The high rotation pressure magnitude during the normal drilling (purple line, second lower graph, Fig. 7) may suggest an excessive pressure over the drill rod that causes the deviation.

Blasthole 77, ch 133 has the lowest deviation values (Table 2). Changes in the path between 1–2 m and 3–4 m depth (Fig. 6b), are associated with peaks in the rotation pressure variability signal between 1.5–2 m and 3–3.5 m (Fig. 8).

The results confirm that drilling deviations are influenced by the rock structure. Peaks and drops detected in the variability of the rotation pressure and in the signals of feed and hammer pressure are correlated with changes in the blasthole trajectory; these anomalies in the signal are normally associated with disturbances zones in the rock. In addition, variations in the magnitude of the hammer, feed and rotation pressure correspond to changes in the pressure of the drill rod against the rock. The higher this measure, the higher the stress in the drill rod with increased probability of deviations.

5. Conclusions

The quality of the drilling in underground blasting operations has been investigated with a view to quantify the error of the monitoring system data of an Atlas Copco drill rig with respect to the actual end position of the blasthole logged. For that, a Pulsar Micro Probe Mk3 has been used to measure the actual trajectory of five production blastholes, by measuring inclination and azimuth values along its length.

Measurements analysed have been carried out in the underground extension work of the municipal wastewater treatment plant in Oslo,

Fig. 8. Variability index of the rotation pressure parameter for the 5 blastholes analysed; BH is blasthole and ch is chainage.

Norway. The results indicate deviations, in some cases significant, between the actual trajectory of the blastholes and the position estimated by the MWD system. A mean absolute deviation error in the vertical and horizontal planes with respect to the deepest location measured by the probe are estimated in 0.07 ± 0.03 m/m and 0.05 ± 0.02 m/m (mean and standard deviation), respectively. Deviations in the X_dY_d plane also show differences in the length and orientation of lookout distance with a mean absolute value of 0.08 ± 0.03 m/m. The mean 3D absolute error is 0.11 ± 0.03 m/m. These values correspond to the minimum expected errors since they assume a correct collaring and alignment of the bit and drill rod.

Drilling deviations may encompass undesired variations of charge concentration at depth resulting in poor or excessive rock breakage and under- or over-excavation. The shorter final length of the measurements done with the probe may also indicate a shorter effective drill length and thus, shorter rounds than expected.

The comparison between MWD parameters and deviation measurements point out that drilling deviations are highly influenced by the rock structure. Disturbance zones in the rock indicated by peaks and drops in some parameters, such as the variability of the rotation pressure and the feed and hammer pressure, are highly correlated with abrupt changes in the blasthole trajectory. In addition, different magnitude of these parameters corresponds to changes in the pressure of the drill rod against the rock. The higher this measure, the higher the stress in the drill rod with increased probability of deviations.

Although this study sheds some light on the drill deviation issue in tunnelling, a higher number of measurements is necessary to draw a general conclusion in relation to the quality of the drilling and to improve the assessment of the MWD to detect cases of deviations in the blastholes due to the rock structure. This would allow to build e.g. a probability distribution of drilling error, to be applied to the blasthole trajectory given by the MWD system.

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Appendix A. Supplementary material

Supplementary data to this article can be found online at <https://doi.org/10.1016/j.tust.2018.10.003>.

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APPENDIX F. Paper F

Application of an in-house MWD system for quarry blasting

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Application of an in-house MWD system for quarry blasting

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ABSTRACT

The high cost and the difficulty of installing the drill monitoring system commonly known as MWD (Measurement While Drilling), especially in older drills, restrain the use of this technology in numerous small quarries and mines. An in-house MWD system is presented here as a low-cost alternative that allows monitoring the information of any rig while drilling. The digitization and automatic sampling of the analogical signals of the sensors involved in the operation have been carried out for their logging and retrieval. The prototype has been installed in a top hammer rotary-percussive vertical rig and has been tested in the monitoring of six blastholes. The MWD parameters have been combined, considering their variation and magnitude, to obtain a fracturing index to be used as an engineering tool for geotechnical rock characterization. The index has been assessed against photographic records of the blastholes walls made with an optical televiewer.

1 INTRODUCTION

Automatization of different unit operations involved in mining or civil works brings new possibilities to optimize the operation. A detailed rock mass characterization is one of the most important requirements to control the ore to be mined, improve blasting results and optimize the mine to mill performance. Classical methodologies typically assess global rock mass properties (Deere et al., 1967; Barton 1974; Bieniawski 1995). However, during all operations, the occurrence of unexpected anomalies will influence the outcome of the next production stages. Monitoring of performance data of modern drill rigs through Measurement While Drilling (MWD) systems gathers real time information of the penetrated rock mass, providing a complementary tool for rock mass characterization.

Schunnesson (1997) described the MWD technique as a drill monitoring system that logs drilling data at predetermined length intervals providing information of the operational parameters involved in the drilling. Drill monitoring systems use sensors that record the response of different operational parameters of a drill system, such as thrust, rotary torque, penetration rate etc.

Modern drilling rigs, manufactured by Atlas Copco, Sandvik and AMV, includes drilling automation, e.g. ABC (Advanced Boom Control) and Bever Control, to optimize drilling. Most drill rig manufacturers have developed their own software (Tunnel Manager - Atlas Copco, iSURE - Sandvik and Bever Control in cooperation with AMV) to evaluate rock properties based on MWD, such as hardness and fracturing (Atlas Copco 2017; Bever Control 2017; Sandvik 2017;).

However, the high cost and the difficulty to retrofit drill monitoring system on old and existing drill rigs, restrain the use of this technology in numerous small quarries and mines. In addition, the existing interpretation software's are tailor-made for the supplier's drill rigs and do not reveals the details on how these softwares manage the MWD records. This complicates its application as a decision-making tool for other drilling systems.

This paper presents an alternative low-cost MWD system to monitor the information and performance of any drilling system. For that, the digitization and automatic sampling of the analog signals from the sensors involved in the rig control have been carried out. With the purpose

of developing an engineering tool for geotechnical rock recognition, a fracturing index has been developed by combining both the variation and the magnitude of several parameters. The results have been validated with photographic records from an optical televiewer. The development is framed within the EU SLIM project (SLIM, 2017).

2 DESCRIPTION OF THE IN-HOUSE PROTOTYPE

located at the South of the city of Toledo, in central Spain. The used drilling rig in El Aljibe is a Tamrock DX-800 from 2008 (manufactured by Sandvik), equipped with top hammer rotary-percussive rock drill. The rig uses 3.2 m rods with a diameter of 58 mm and 3.5 inch (89 mm) bits. Since the rig supplier did not provide any MWD system, an in-house MWD system was developed as a low-cost and efficient alternative to monitor the information from the rig while drilling.

Integra Automatización S.L.U. was retained to automatize the drill rig. The system records and digitizes the analogical signals of the existing sensors on the drill rig. PLC (Programmable Logic Controller) and HMI (Human Machine Interface) were developed to fulfil the following actions:

- Design of graphical user interface to represent live data of depth and pressure parameters on a 7" screen installed on the drill rig.
- Conversion from analogic to digital signals.
- Data logging and storing.

Drill parameters are digitized with a parallel connection of the sensors already installed on the rig for the analogical logging of the required parameters for controlling the drilling. The system monitors information at every sample interval value of depth. For this, a rotary encoder was used to detect an increase of depth during drilling by transforming the encoder pulses into linear displacement; at every pulse, the system registers information of the parameters when there is an increase in the hole depth count. Sample interval was preset at 0.01 m. Parameters monitored by the system are, with their acronyms and units:

- i. Percussive pressure (HP, bar)
- ii. Feed pressure (FP, bar)
- iii. Rotation pressure (RP, bar)
- iv. Air pressure (AP, bar)
- v. Damp pressure (DP, bar)
- vi. Hole inclination (°)
- vii. Hole depth (HD, m)

Two inclinometers were installed on the mast, to measure the vertical and horizontal angles of the feeder outside the blasthole. The rig system also registers values of the feeder inclination at 0.01 m sample interval, in line with the monitored parameters.

Technical specifications of the electronic sensors used to digitalize the analogic signal of the parameters and the inclination of the blasthole are:

- a. Percussive pressure: electronic pressure sensor with a range of 0 to 600 bar (maximum pressure 2500) and a resolution of 0.0183 bar.
- b. Rotation pressure, Feed pressure and Damp pressure: electronic pressure sensor with a range of 0 to 250 bar (maximum pressure 1200) and a resolution of 0.0066 bar.
- c. Air pressure: electronic pressure sensor with a range of 0 to 10 bar (maximum pressure 300) and a resolution of 0.0003 bar.
- d. Inclinometers: inclinometer of two axes to measure the horizontal and vertical angles of the feeder, with a resolution of 0.005° and a maximum angle measurement of +/-20°.

Other technical common characteristics of the sensors are: working temperature range (-40 to 90 °C), resistance to the vibrations (20 g, 10 to 2000 Hz), accuracy (<0.5%) and response time (1 ms).

Data logging and storage are carried out as follow: the PLC receives electrical signals of pressure, inclination and depth sensors which are handled by the program and they are registered every time the encoder increases a pulse and in addition the following boundary pressure conditions are fulfilled:

- Percussive pressure greater than 80 bar
- Feed pressure greater than 30 bar
- Rotation pressure greater than 25 bar

- Air pressure greater than 3 bar.

The visualization of the logging data is done through a 7" touch screen. Specific events, such as water in the hole, addition of a new rod or a new bit, drilling in manual or automatic mode, visible fracture, etc., can be manually entered into the system and visualized in the screen. This will help to understand the monitored data in a subsequent analysis. Figure 1a shows the screen of the system in the cabin of the Tamrock DX-800 and Figure 1b represents a view of the monitor screen, where all parameters explained above are represented. Once a complete blasthole is drilled, the system exports the data into a text file that can be transferred to a memory stick.

Development, installation and tuning cost of the prototype was around 20.000 euros.

3 DATA OVERVIEW

The developed drill monitoring prototype, installed on a Tamrock DX-800 rig was used for the monitoring of the first campaign of a large-scale tests (included in the SLIM project) in El Aljibe quarry. The rock quarried in El Aljibe is mylonite, which is a fine-grained metamorphic rock formed by ductile deformation under shear stress.

Drilling was done in automatic mode, i.e., parameters are automatically adjusted by the drill rig system during the operation. The analysis comprises six blastholes as an example of the results obtained by the prototype.

Figure 1 Installation of the in-house drill monitoring system. a) operator's cab; b) view of the monitor screen

3.1 Optical Televiewer

The condition of the blastholes was analyzed with an optical televiewer manufactured by Advance Logic Technology. The log provides a continuous unwrapped 360° oriented color image of the blasthole walls (Li et al. 2012). The televiewer components are:

- The logging tool quick link QL40 OBI-2G (Figure 2), of 1.47 m length, diameter of 0.40 m and mass of 5.3 kg has digital image sensor at the bottom with an active pixel array of 1.2 Mega Pixel and fisheye matching optics. The light source is provided by ten LEDs. This tool also includes hole deviation sensors in the central part, a connector

to the wireline in the top part and two centralizers.

- The data acquisition system (BBox).
- The mini-winch with 200 m of 1/8" wireline consisting in a steel armored cable with an insulated conductor at the center that gives power to the tool, allows data exchange with the surface, and moves the logging tool along the blasthole a constant velocity.
- A computer with the software ATL Logger Suite 11.2.

Figure 2 General aspect of the QL40 OBI-2G (User Guide OBI, 2017).

The comparison between the monitored MWD parameters and the televiewer records will improve the understanding of the drill/rock interaction, with a final purpose of developing a model to characterize the rock condition. Figure 3 shows an example of the monitoring data for blasthole 5 and the corresponding televiewer records. For the first 4.5 m, negative peaks in the percussive pressure (HP) and feed pressure (FP) and a significant variation in the rotation pressure (RP) can be seen, that may be related with a possible zone of disturbed rock (also indicated by a different color in the televiewer record). In the same way, fluctuations in the MWD signals at 11.2 m may show another possible disturbance zone. Schunnesson (1996, 1997) claimed that when discontinuities are encountered, the rotation pressure shows significant fluctuation, resulting in a noisy signal. In addition, Peng et al. (2005) determined the feed pressure and thus hammer pressure as good indicators for void detection and Navarro et al. (2018) identified the feed pressure as the master parameter that controls the adjustment of the other parameters

when variations while drilling occur. Damp pressure and air pressure, on the other hand, barely show changes found in any of the other parameters and thus, they will not be considered for the subsequent analysis.

For a proper measurement, the entire logging tool must be introduced in the hole to ensure that the two centralizers are in contact with the rock and the camera is located in the center of the hole, hence, the first 1.5 m cannot be measured. For blasthole no. 5, the last 1.4 m were not measured with the televiewer as the blasthole was blocked. This may be related with a zone of disturbances indicated by the noisy MWD signals at this depth. The rest of the parameters signal is mainly flat which indicate solid rock.

Once the drill reaches the end of the rod, a new one must be added to continue the drilling. At this point systematic variations in the parameters response (systematic peaks) are observed when a new rod is added, which occurs at intervals of 3.2 m. During this procedure, percussive pressure, feed pressure and rotation pressure parameters significantly drops since the drill rod is slightly pulled back to avoid the contact between the bit and the bottom of the hole, and the new rod is added to the end of the last one. The logging system starts to record values again when the drilling overcomes the last depth measure. Figure 3 also shows the points where a new rod is added. Systematic variations due to the addition of a new rod are filtered out in a post analysis, since these do not reflect any information of the rock.

Figure 3 Signal of the monitoring system for blasthole 5, blast 1 and televiewer records of the wall condition.

3.2 MWD fracture index

According to Schunnesson (1996, 1997), Peng et al. (2005) and Navarro et al. (2018), fluctuations in the signals of feed pressure, hammer pressure and rotation pressure parameters indicate changes in the geotechnical rock condition. These variations are highlighted and calculated as the sum of the residuals over a defined interval along the blasthole, see Equation 1 (Schunnesson, 1996 and Gosh, 2017).

$$[MWD_{var}]_i = \sum_{t=i}^{i+N} |MWD_t - \overline{MWD}| \text{ with } i=1,2,\dots,L \quad (1)$$

where, MWD_{var} is variability of the MWD parameter considered; N is window size (for the case under study, $N=4$); i is the sample number; L is the length of the MWD signal; MWD_i is the monitored parameter at the i sample; and $\overline{MWD}$ is the average of the MWD values in the window between i and $i+N$ samples.

The variability of the feed pressure, hammer pressure and rotation pressure (HP_{var} , FP_{var} ,

RP_{var}) at each i sample have been combined in a fracturing index. Since these parameters have different magnitude ranges, they are scaled by the mean value of each signal logs per hole to have equal impact in the combined index. They are also square-root transformed to limit the influence of the very high peak values, that would hide smaller fluctuation of interest

$$\text{Fract. Index} = \left(\frac{HP_{var}}{\overline{HP}} \right)^{\frac{1}{2}} + \left(\frac{FP_{var}}{\overline{FP}} \right)^{\frac{1}{2}} + \left(\frac{RP_{var}}{\overline{RP}} \right)^{\frac{1}{2}} \quad (2)$$

4 RESULT

The fracturing index has been compared with the televiewer records. As an example, Figure 4 represents the fracturing index and two pictures of the wall condition inside blasthole 5 (a 2D 360° scan of the wall and a 3D reconstruction). The trace of the different rock features is marked; seven different types of rock disturbances have been found: open fracture, continuous closed fracture, discontinuous fracture, filled fracture, weakness area, void and change of the lithology. The 3D representation includes the joint planes. The fracturing index has peaks and fluctuations when the bit drills through a rock disturbance. As can be seen in Figure 4, fracturing values greater than 1 normally indicate significant rock disturbances.

Table 1 describes the type of disturbance that creates each signal variation in the fracturing index of Figure 4. Different ranges of fracture values can be distinguished according the complexity of the rock disturbance. The more complex the disturbance, which normally includes voids, big fractures and weakness areas, the higher and wider the fracturing index peak.

Figure 4 Comparison between fracture index and televiewer records (section and 3D core reconstruction).

Although significant variations in the fracturing index are always related with rock disturbances, not all disturbances found in the televiewer records can be correlated to changes in the drilling

performance. This is the case of some filled fractures and isolated continuous closed fractures found at hole depths 6.2 m (filled fracture), 6.95 – 7.05 m (filled fracture and continuous close fracture), 8.8 m and 10.78 m (isolated continuous closed fracture). However, most of these isolated continuous closed fractures are almost horizontal with minimum influence during the drilling and the filled fractures may not have significantly different properties of rock to be noticed by the monitoring system

Table 1 Description of the rock disturbances that creates changes in the fracturing index

Hole depth (m)	Fracturing Index (range of values)	Rock disturbances
1.4 – 2.3	2 – 3.5	Open fracture Cont. closed fracture Weakness area Void
2.65 – 2.85	2	Open fracture Cont. closed fracture Weakness area
3.5 – 4	1.5 – 3	Open fracture Weakness area Void
4.3 – 4.6	1.5 – 2	Cont. closed fracture Weakness area
5 – 5.2	2.8	Cont. closed fracture Filled fracture Change of lithology
5.4 – 5.6	1.2	Cont. closed fracture Discont. fracture
7.6 – 7.7	1.5	Cont. closed fracture Discont. fracture
11.1 – 11.8	2 – 3	Filled fracture

Fracturing index results show a visual correlation between the geotechnical classification of the blasthole condition and the MWD parameters. However, the difficulty to assign values of fracturing index to a specific rock disturbance and the fluctuation of this index signal, complicates a quantitative interpretation of the results. To solve this, the fracturing index per hole has been divided in zones based on abrupt changes of variability in the signal. The procedure divides the signal into a preset number of sections and estimates the standard deviation for each one. These sections are varied iteratively to minimize the sum of the residual error of each region in the standard deviation by using Gaussian log-

likelihood (Lavielle et al., 2005; Lillick et al., 2012). Considering the long length of the signals (more than 1200 logs per blasthole) and the width of the fluctuations, the maximum number of sections is preset in 30 per blasthole. Figure 5 shows the division of the fracturing index signal (green dots lines) and the mean fracturing index value in each section (red lines) for blasthole 5. The analysis has been programmed in Matlab (Matlab, 2017).

Figure 5 Division of the fracturing index in zones based on abrupt changes of variability in the signal.

The mean fracturing index in each section is calculated and classified in five different range of values correlating them with a new definition of rock condition:

- Solid rock zone (SRZ): intact or massive rock.
- Slight perturbation zone (SPZ): rock with small filled, continuous or discontinuous fractures.
- Fractured zone (FZ): fracture zone composed by several fractures together (filled fractures, open fractures, continuous fractures or discontinuous fractures) or small voids.
- Blocky zone (BZ): zone composed by a weakness area, a big fracture (open fracture, filled fracture) and/or a medium side void.
- Low competence zone: zone of heavily broken rock mass, voids and/or big inclined fracture (open fracture, filled fracture).

Figure 6 Comparison between of rock condition predictor televiwer records for blastholes of blast 1.

Figure 6 compares the rock condition classes estimated from the processed fracturing index with the analyzed televiwer records for the six blastholes under study. As can be seen, most of the important rock conditions (low competence zone, blocky zone and fractured zone) are predicted; however, as with Figure 4, not all disturbances found in the televiwer records are noticed during the drilling performance. Sometimes, isolated or small fractures (filled, open, continuous or discontinuous) are difficult to be made apparent while drilling. The same situation occurs when a change in lithology does not include a significant variation in the rock properties.

The results point out a good visual correlation between the fracturing index and the geotechnical features found in the rock; variations in the fracturing index are always related with rock disturbances. It is observed that the larger and the more complex the disturbance, the higher and wider the fracturing index variation. However, since it is difficult to assign values of fracturing

index to a specific rock disturbance, the index signal per blasthole has been divided in zones based on abrupt changes of variability of its values. This simplifies the fracturing index logs and helps their interpretation as to rock characterization. Preliminary results so far are promising and will be validated with more extensive test program.

5 ACKNOWLEDGEMENT

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APPENDIX G. Paper G

Application of drill-monitoring for chargeability assessment in sublevel caving

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Application of drill-monitoring for chargeability assessment in sublevel caving

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ABSTRACT

The use of drill-monitoring parameters in drilling operation for the large scale in sublevel caving mining may assess geotechnical issues in the holes before charging. Nowadays, the charging procedure is carried out with no prior information of the rock mass condition, which limits the reaction to unexpected rock issues that may collapse the blasthole. This results on charging problems and, in consequence, bad fragmentation of the rock after blasting that difficult ore loading and transportation and reduce the gravity flow of the rock.

This paper builds up the work done by [8], to classify the geotechnical rock condition in five classes (solid rock, fractured rock, cave-in, minor and major cavity). From it, two applications have been developed: one for geotechnical rock condition of orebodies and the other for predicting the risk of collapse in boreholes. [8]'s work has been improved into a rock condition block model to simplify the quantitative assessment and automatic recognition of rock trends. A thorough correction of the MWD parameters has been also applied to minimize external influences other than the rock mass. From it, the risk of borehole collapse model has been developed by comparing different combinations of the geotechnical rock condition block-model with the charging length of 102 production fan-holes. The assessment of the number of collapsed and non-collapsed blastholes and the charging length/blasthole length ratio has been used to assign high, medium or low risk of collapse to each combination. The results predict collapses in the first half of the fan-holes for the high risk, collapses in the second half of the fan-hole for the medium risk and no collapses along the hole for the non-risk holes. The two models have been applied in large scale for two orebodies in the Malmberget mine, Sweden, which comprises 20 drifts and 5060 fan shape long-holes.

Keywords: Rock mass condition, Underground blasting, Measurement While Drilling (MWD), Block model, Explosives.

1. Introduction

Sublevel caving is an underground mining method widely used to mine out steeply dipping ore bodies surrounded by weaker overlying rock mass. The procedure is based on the utilization of gravity flow of the blasted ore and the barren rock [27]. For that, the orebody is divided in levels on which cross cuts are excavated through the ore body from the foot wall to the hanging wall. From the cross cuts, fan-shape long-holes are drilled upwards to extract the ore above the drift [4, 12]. To get a proper gravity flow, the fragmentation of the ore plays a key role and will also have influence on the loading and hauling operations. Excessive coarse material may cause gravity flow problems and disturb ore extraction [10].

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Borehole collapses are closely related to geotechnical issues in the rock. The long time between drilling and blasting further exposes the fan-holes to external impacts. Furthermore, since a hole is an aperture in the rock, the original properties of the ground condition around the hole are influenced, inter alia, by the stress redistribution [16] and vibrations from adjacent blast-rounds that may induce stability problems around the fan-hole wall. When the surrounding stress exceeds the tensile, the compressive, or the shear strengths of the rock formation, failures in the hole wall can be generated [39]. As a result, rock detachments may be generated with possible collapses inside the hole, which later may result in charging problems.

Apart from drilling and blasting, fragmentation problems are also closely related with charging issues. One root cause of charging problems is bad rock mass conditions causing borehole instabilities, such as failures on the wall and collapses in the hole. This may hold back the charging at some depth and block the charging process beyond this point. [13] determined, with a mini-video camera, that boreholes problems normally are caused by deformation and boreholes jammed by stones. Since re-drilling is time consuming and costly and it is not allowed for blastholes close to pre-charged rings, collapsed holes will sometimes remain uncharged, generating coarser fragmentation and boulders that may disturb the ore flow. The long time period between drilling and charging (sometimes several months) and the absence of a detailed borehole condition knowledge may lead to unexpected charging issues that limits the reaction time for adaptation of the charging operation and in the planning of the following loading operation.

The performance data of a drill rig, through the MWD system, is a well establish technique for identifying the geotechnical condition of the penetrated rock mass and may be used to identify charging problems.

The Measure While Drilling is a technique that logs drilling data at predetermined length intervals, providing information of the operational parameters involved in the drilling operation [29]. Drill monitoring has been widely studied for geotechnical rock mass recognition. For rotary drilling, [34] defined different geological formations from the analysis of variations in the logged parameters. [25] found a correlation between the monitoring parameters, the rock compressive strength and the shear strength. [36] introduced the concept of specific energy (SE) as the energy required to excavate a unit volume of rock. This was later modified by [18]. [9] demonstrated that MWD logs from rotary drilling can be used to recognize properties of the rock mass if the wear of the bit is maintained constant. [19] developed a coal-seam detection model called SEM (modulated specific energy).

In relation to studies developed for percussive and rotary-percussive drilling, [29] proposed a methodology to estimate Rock Quality Designation (RQD, [5]) index using percussive drilling. The method was based, not only on the penetration rate and torque pressure, but also on the variation of both parameters, which showed a close correlation with the presence of discontinuities. [6] determined a relationship between the drill parameters and the geo-mechanical rock properties in tunnel roofs such as discontinuities and changes in the rock strength. [26] and [35] developed a method to measure void/fractures in the roof using bolt drilling. For the analysis, they used a normalized feed pressure parameter, claiming that it is a good detector of significant discontinuities in the rock. [31] suggested a model for the hydraulic properties of the rock mass, based on the monitored water flow and water pressure during rotary-percussive drilling. [32] explained a methodology to assess rock strength ranges based on an MWD hardness index provided by Atlas Copco software [2]. For that, they used Schmidt hammer to correlate MWD values with empirical rock strength measurements.

[14] studied the correlations between penetration rate, uniaxial compressive strength (UCS), Brazilian tensile strength, point load strength and Schmidt hammer tests for percussive drilling.

For MWD data, [30], for percussive drilling, [11] and [22], for rotary-percussive drilling, and [7] for Wassara water-hydraulic ITH (in-the-hole) drilling, have described procedures to normalize parameters in order to reduce external influences in their response.

Epiroc AB [6], Sandvik [28] and Bever Control, in collaboration with AMV [1, 3], have developed software for their own drilling equipment (Tunnel Manager MWD, iSURE and Bever Control, respectively) as a tool for planning and assessment of drill parameters. From collected MWD files, blastholes can be represented in 3D and hardness and fracturing indexes are provided. However, the existing interpretation software's are tailor-made for the supplier's drill rigs and do not reveal the details on how these softwares manage and refine the MWD data.

In previous work, [8] developed a probability geotechnical rock classification model for fan shape long-holes drilling. The model is based on the combination of MWD parameters and a calculated fracturing index, using principal components analysis. The model can be used to characterize and predict geotechnical issues in the rock condition around the holes, which are closely related to borehole collapses and charging problems.

This paper aims at demonstrating the use of MWD technique in the Malmberget mine, Sweden, based on the work done by [8], to classify the geotechnical rock condition in five classes (solid rock, fractured rock, cave-in, minor cavity and major cavity). From it, two applications have been developed: one for geotechnical rock condition of orebodies and the other for predicting the risk of collapse in boreholes. [8]'s work has been improved into a rock condition block model to simplify the quantitative assessment and automatic recognition of the rock trends. From it, the risk of borehole collapse model has been developed by comparing different combinations of the geotechnical rock classes with the charging length of 102 production fan-holes. The assessment of the number of blocked and non-blocked fan-holes and the ratio charging length/blasthole length has been used to assign high, medium or low risk of collapse to each combination. The two models have been applied to the large scale for two orebodies in the Malmberget mine, Sweden, which comprises 20 drifts and 5060 fan shape long-holes.

2. Data overview

2.1. Test site

The Luossavaara-Kiirunavaara AB's (LKAB) Malmberget underground iron ore mine is located close to the municipality of Gällivare, Sweden. The mine contains twenty orebodies spread over an underground area of about 5 by 2.5 km. Twelve of those orebodies are currently being mined. The orebodies are mainly composed of magnetite with hematite intrusions in some areas, mostly in the western field. The dip of the ore bodies varies between 15° and 75°, with an average dip of 45° - 50° [23]. The host rock is composed by andesite rock, skarn, granite and biotite schist [23, 27, 37, 38].

2.2. Drill system

Malmberget mine uses transversal and longitudinal sublevel caving mining as the method for ore extraction. Six Atlas Copco SIMBA W6C drill rig equipped with hydraulic Wassara In-The Hole (ITH) hammer are used for production drilling in mine. The production holes are drilled in a fan shaped pattern with holes between 10 and 50 m length. Drilling is done using 102 mm extension tubes with 2.3 m length and 4,5" (115 mm) bits diameter. A 4.5 revision control system (RCS) was installed in the rigs. In this drilling technique, water at high pressure (up to 180 bar) is used to power the impact mechanism of the hammer. From surface, the energy is transmitted through the drill rod in form of pressurized water, mechanical torque and mechanical feed force [7]. Once the bit strikes the rock, water is released through the bit, removing drill cuttings from the borehole [20].

Drilling in Malmberget is fully automated and remotely controlled i.e., logging of drill parameters and positioning of the holes are done automatically. The drill parameters registered during drilling are described next; the acronym and the units for each parameter are given in brackets.

- Feed pressure (FP, bar): this parameter measures the hydraulic pressure inside the feed cylinders, that generates the feed force necessary to keep the bit in contact with the bottom of the drill hole throughout the impact of the piston.
- Percussive pressure (water pressure, PP, bar): it is the water pressure used to force the piston to impact against the bit [7].
- Rotation pressure (RP, bar): pressure to generate the drill string torque to maintain the required rotation speed of the bit.
- Penetration rate (PR, m/min): is the speed at which the drill bit advances through the rock mass.
- Hole depth (HD, m): length of the drilling at each sample interval.

Two sets of MWD data have been gathered for the analyses. Table 1 shows available data, gathered from five different orebodies, used for the development of the geotechnical rock condition block model and, from it, the risk of collapse model. For that, the underground charging operation at the mine was followed for 102 blastholes (11 rings). The purpose is to compare the predicted risk of collapse in the blasthole with actual problems encountered by the charging personnel. The maximum depth of the charging hose introduced in the blasthole, in relation to the respective blasthole length, gives a measure of whether the charging hose was obstructed in the blasthole or not and, in case of obstruction, the depth at which it occurs.

The application of the two models in large scale is carried out for data from two orebodies (orebody 1 and orebody 4) located at different depths. Table 2 describes this second data set for the representation in the large scale of both geotechnical block model and risk of collapse models, which comprises 20 drifts and 5060 fan-holes.

Table 1. MWD data for the development of the risk of collapse model

Orebody	No. rings	No. Holes
1.	6	63
2.	1	7
3.	2	18
4.	1	7
5.	1	7

Table 2. Data for the large scale model

Orebody	No. drifts	No. Holes	Level (m)
1.	9	2662	1052
4.	11	2398	1031 & 1056

3. Drill Rig Navigation

Navigation in the Atlas Copco SIMBA W6C drill rigs is similar to jumbos for tunneling constructions [22]; they use three reference systems to locate the drilling machine inside the drift to carry out the drilling of the rings:

- An Absolute Coordinate System (ACS) that references the position of the drill rig by trilateration, i.e, the drill rig uses a laser scanner to measure the distance from the scanner to target points with known coordinates located along the wall of galleries and drifts (see Figure 1a). For this case, the ACS is measured in the EUREF 89 Norwegian Transverse Mercator (NTM) projection.
- A Total Reference System (TRS) composed of one axis parallel to the direction of the drift (Z_t) and the other two (X_t and Y_t) in the cross section of the drift (see Figure 1b). To obtain the TRS, the feeder is aligned with the direction of the drift; the laser beam of the scanner points in the direction of the drift and the feeder is rotated until the laser beam passes through two targets mounted on it.
- A Drilling Reference System (DRS) defined by two vertical planes X_dY_d , Y_dZ_d and a horizontal X_dZ_d plane. The drill rig monitors, in the Drilling Reference System, the collaring and end position of the fan-shape holes before drilling starts; the final position is automatically calculated by means of three spherical coordinates: the hole length, the azimuth and the inclination angles of the feeder [22].

Figure 1. Drill rig navigation; a) Absolute Coordinate System (ACS); b) Alignment with drift line to measure the Total Reference System (TRS)

During the navigation operation, the drill rig transforms the DRS coordinates of a point in the fan-hole to the TRS by means of three rotation angles (γ , θ , ω). For Atlas Copco SIMBA W6C drill rigs, the directional coordinate vectors of the TRS ($\vec{x}_t$, $\vec{y}_t$, $\vec{z}_t$) and the NTM coordinates of the rig (X_{NTM} , Y_{NTM} , Z_{NTM}) are presented at the beginning of each MWD file. Figure 2 describes the rotation movements of the drilling reference system of vectors ($\vec{x}_d$, $\vec{y}_d$, $\vec{z}_d$, graphs in blue)

to the total reference system of vectors ($\vec{x}_t$, $\vec{y}_t$, $\vec{z}_t$, graphs in black). In order to simplify the explanation, the transformation system is represented in a drift section.

Figure 2. Trasformation from drilling reference system ($\vec{x}_d$, $\vec{y}_d$, $\vec{z}_d$) to total reference system ($\vec{x}_t$, $\vec{y}_t$, $\vec{z}_t$). Left: Rotation towards right and left, or bearing angle, over $\vec{y}_t$; center: uphill- downhill inclination, or elevation angle, rotation over $\vec{x}_t$; right: lateral inclination, roll or bank angle, rotation over $\vec{z}_t$.

4. Data analysis

4.1. MWD data processing

One general problem with MWD data analysis is that the logged response is a mixture of the response from varying rock mass conditions, from the drill rig control system and from the operator's intervention [30]. All together add uncertainty to the data that must be normalized to highlight changes in the parameters depending on the rock properties. This process is described next and comprises: (i) filtering of unrealistic values, (ii) removing of systematic peaks due to the addition of a new rod, and (iii) correction of the hole depth influence.

4.1.1. Filtering of unrealistic values

Production data often includes unrealistic high and low performance values of the drill rig, which may lead to a wrong interpretation of the MWD data. Unrealistic values found in the MWD raw data are filtered out according to the criteria established by [8], based on both frequency and practical experience in the Wassara water-hydraulic ITH technology. Table 3 describes the threshold values used for the analysis:

Table 3. Selection of intervals for filtering raw data [8].

Parameter	Units	Thresholds
Penetration Rate (PR)	m/min	$0.1 < PR \leq 4$
Percussive pressure (PP)	bar	$5 < PP \leq 200$
Feed pressure (FP)	bar	$35 < FP \leq 100$
Rotation pressure (RP)	bar	$25 < RP \leq 125$

4.1.2. Removing of systematic drops due to the addition of a new rod

For long holes, systematic variations are related to the parameters response when a new rod is added; when the rock drill reaches the end of the feeder, a new rod must be added to continue the drilling. During this procedure, percussive

pressure, feed pressure and rotation pressure parameters are shut down, the drill rod is pulled back and the new rod is added to the end of the last one. The logging system starts recording values again when the bit overcome the last depth measure. Figure 3a represents the raw signal values of MWD parameters for a single hole, where it can be seen systematic drops at every 2.3 m, matching with the addition of a new rod. Such systematic variations, that do not reflect any information of the rock conditions, are automatically filtered out here in a post analysis in Matlab [24]. Figure 3b shows the filtered signal from Figure 3a, in which all peaks associated to rod changes have been removed.

Figure 3. Filtering to remove variations in the signal: a) Raw signals; b) Filtered signals.

4.1.3. Correction of hole depth influence

Systematic variations generated by the drilling system and external influences other than the rock can be recognized and removed by averaging, for a large amount of data, the logged response [30]. [8] described a significant hole depth dependence in some parameters for percussive and Wassara water-hydraulic ITH drilling, respectively.

This phenomenon is partly related to the increasing energy losses when holes are getting longer, both mechanically through the reduced stress wave energy and the pressure drops for ITH drilling. It can also be related to the decreasing flushing efficiency with depth and the bit wear [30]. Increasing frictional resistance between the drill rod and the blasthole walls, can also cause this phenomenon.

Figure 4 shows the average value of penetration rate (Av.PR), percussive pressure (Av.PP), feed pressure (Av.FP) and rotation pressure (Av.RP) at every 1 m hole length for the 5162 fan-holes analyzed. The calculation is shown for four of the six drill rigs separately (represented with different color), to see whether there is any other systematic variation caused by the calibration of the rig sensors. From the four graphs, feed pressure (Av.FP) shows a well-defined increase with hole depth (HD) for all drill rigs. The normalization of the hole length influence is calculated, for this parameter, for the four rigs together, since all of them follow the same trend (Figure 4, Av.FP). Blastholes are normally drilled between 10 and 50 m length, corresponding the latter to blastholes in the middle of the fan. Since measures over 40 m are limited, they may produce deviations in the trend of the average signal at high hole depth values. These values are removed statistically from the analysis. For that, the empirical probability distribution function of the hole depth (HD) is built with the complete data set values and data above the percentile 97.5 (upper tail) are discarded.

The control system is normally based on three main operational modes [22, 33]: (i) collaring, (ii) ramp-up, both of which control the increase of the drilling pressure to minimize hole deviations, and (iii) normal drilling, which controls the performance of the parameters to optimize the operation and minimize damage to the drill system. As data in the

ramp-up mode are not representative of changes in the rock mass conditions, the first drilling meter is excluded from the analysis to retain only values from the normal drilling.

The average signal of the feed pressure (Av.FP) is calculated between 1 m and 36 m hole depth range for the hole depth normalization analysis to exclude data from ramp-up operation and the upper tail of the 95 % confidence interval (see black dashed lines in Figure 4, Av.FP). Penetration rate, percussive pressure and rotation pressure show no influence by the hole depth and they are not normalized.

The correction of the hole length influence (to obtain a signal FP_n) is done by:

$$FP_n^i = [FP^i - FP_{fit}^i] + FP_{fit}^1; \text{ with } i = 1, 2, \dots, N \quad (2)$$

where i indicates each measurement in a hole log, N being the number of these. FP_{fit} is a linear regression with hole depth of the average value at every 0.1 m hole length for the entire data of each FP^i value. The determination coefficient of the fit R^2 is 0.92. FP_{fit}^1 is the intercept of the fit, i.e. the value at depth zero.

Figure 4. Average variation for penetration rate (Av.PR), percussive pressure (Av.PP), feed pressure (Av.FP) and rotation pressure (Av.RP) with depth; data are measured at every meter depth for the 5162 data gathered (each color corresponds to a different rig). HD is hole depth.

4.2. Fracturing parameter

[29] claimed that when discontinuities or changes in the geotechnical rock condition are found, penetration rate and rotation pressure parameters show significant fluctuations, resulting in a noisy signal. [8] demonstrated for Wassara ITH hammer, an increase in the fluctuation of the penetration rate and rotation pressure when drilling through a fractured zone. These variations are highlighted and calculated as the sum of the residuals over a defined interval along the borehole:

$$[PR_{var}]_i = \sum_{t=i}^{i+N} |\overline{PR} - PR_t|, \text{ with } i = 1, 2, \dots, L \quad (3)$$

$$[RP_{var}]_i = \sum_{t=i}^{i+N} |\overline{RP} - RP_t|, \text{ with } i = 1, 2, \dots, L \quad (4)$$

where, PR_{var} is penetration rate variability, RP_{var} is rotation pressure variability, N is the window size and is considered $N = 4$ (Ghosh, 2017), i is the sample number, L is the length of the MWD signal, PR_i and RP_i are the monitored penetration rate and rotation pressure, respectively, at the i sample.

The variability of both penetration rate and rotation pressure (PR_{var} and RP_{var}) at each i sample have been combined, with a 50 % influence each, in a ‘fracturing’ parameter to make a more robust index; for that, the magnitudes of both parameters have been scaled with the standard deviation of the raw parameter. As shown in Figure 4a (Av.PR and Av.RP), the range of values for the penetration rate and rotation pressure is different between the four jumbos, thus their values are scaled with the standard deviation of the respective jumbo:

$$Fracturing_i = \frac{1}{5} \left[0.5 \cdot \left(\frac{PR_{var,i}}{std(PR)} \right) + 0.5 \cdot \left(\frac{RP_{var,i}}{std(RP)} \right) \right], \text{ with } i = 1, 2, \dots, L \quad (5)$$

where, $PR_{var,i}$ is penetration rate variability, $RP_{var,i}$ is rotation pressure variability, i is the sample number and L is the length of the MWD signal.

4.3. Geotechnical model

[8] used Principal Component Analysis (PCA) to correlate penetration rate, percussive pressure, normalized feed pressure and rotation pressure parameters, plus the calculated penetration rate variability, rotation pressure variability and fracturing index. Based on the results from filming a number of test holes, they assessed first principal component (PC1) values to different rock properties and ranked these values into four different rock classes (solid rock, fractured rock, potential cave-in and cavity). To obtain these ranks, they represented, as probability distribution functions, the first principal component values for these four studied geotechnical categories, obtaining 4 curves, see Figure 5. Based on the shape of these curves and the relation between them, [8] estimated five different rock classes ranks from the values of the 1st principal components (C1 is solid rock, C2 is fractured rock, C3 is cave-in, C4 is minor cavity, C5 is major cavity) and described their geotechnical features and their possible effect on borehole chargeability.

Figure 5. Proposed geotechnical model based on the probability density function of the 1st principal component values of each rock class [8]. C1 is solid rock, C2 is fractured rock, C3 is cave-in, C4 is minor cavity, C5 is major cavity.

Figures 6a and 6b represent the geotechnical model for two rings with expected bad and good rock condition, respectively. The rock classification is given in different colors as can be seen from the legend. The black line at the contour of the fan-holes represents the measured charging length. Charging lengths shorter than the total hole length point out a collapse in the hole at the end of this line, hence charging problems. No charging length drawn indicates a collapse at the collar position, remaining this hole uncharged.

Figure 6. Classification based on a ranking from first principal components interpretation ([8]’s, 2018 methodology). a) Analysis for an expected problematic ring (Orebody 1, ring 4); b) Analysis for an expected good ring (Orebody 3, ring 2).

5. Results

5.1. Geotechnical rock condition block model

In the geotechnical model developed by [8] (Figures 6a and 6b), it can sometimes be difficult to interpret the graphs due to the large number of different rock classes that results in many small intervals (mix of colors along the fan-hole), that complicates the quantitative assessment or the automatic recognition of rock condition trends and chargeability. To solve this, the first principal component results have been divided in zones based on abrupt changes of the mean value in the signal. The procedure divides the signal in a pre-set number of sections and estimates the mean value for each one. These sections are varied iteratively until the total squared error attains a minimum [15, 17]. Considering that the first principal component is a signal x of length N , the iteration finds k (for a two-section case) such that J , defined in

Eq. 6, is smallest. Eq. 6 can be easily generalized to any number of sections. The minimum of the residual squared error is obtained by using Gaussian log-likelihood.

$$J = \sum_{i=1}^{k-1} \left(x_i - \frac{1}{k-1} \sum_{r=1}^{k-1} x_r \right)^2 + \sum_{i=k}^N \left(x_i - \frac{1}{N-k+1} \sum_{r=k}^N x_r \right)^2 \quad (6)$$

The analysis has been programmed in Matlab [24]. For the test fan-holes, the maximum number of sections is pre-set to six. Figure 7 shows an example of this procedure, for fan-hole 9 from drift 8, Orebody 1 (Table 2); the green dot line indicates the edges of the sections that match abrupt changes in the mean value of the signal and the red dot line represents the average of the values within these sections.

Figure 7. División of the 1st PC in six zones based on abrupt changes in the signal.

A new set of limits has been defined from the average of the first principal component in each section for the test fan-holes (Table 1), to match the five rock classes by [8]; this ranking is given in Table 4. Figures 8a and 8b show the geotechnical block model application to the first principal components results from Figures 6a and 6b.

Table 4. New rank to assess the five rock classes in the geotechnical block model

Rock class	Rank
Solid rock	$x^1 \leq 0.44$
Fractured rock	$0.44 < x \leq 2.5$
Cave-in rock	$2.5 < x \leq 5$
Minor cavity	$5 < x \leq 8$
Major cavity	$8 < x$

¹x is the average value of the first principal component in each section.

Figure 8. Rock condition block model. a) Analysis for an expected problematic ring (Orebody 1, ring 4); b) Analysis for an expected good ring (Orebody 3, ring 2).

5.2. Large scale application of the block model

The application of the rock condition block model analysis on larger mining areas (Table 2) is done following the above procedure. The representation of the corresponding rings in a 3D plot is carried out by considering both collaring and end coordinates of the hole in the DRS, the alignment of the drill rigs (TRS) and its absolute coordinates when starting to drill a new ring. The rotation from the DRS coordinates to the TRS is also considered. For orebody 1, the fan-hole coordinates in the local DRS are rotated 212.5° around axis Y (i.e., rotation of X_dZ_d plane, $\gamma = 212.5^\circ$) with respect the TRS, being $\theta = 0$ and $\omega = 0$. For orebody 4, DRS and TRS reference system match, so θ , γ and ω angles are 0° . The translation to the absolute coordinates is obtained by adding the NTM coordinates of the drilling machine to the oriented coordinates (TRS) of the fan-holes. Figures 9 and 10 show the large-scale representation in 3D of the geotechnical block model for orebody 1 and 4, respectively. They represent the parallel drifts layout and the rings. The rock classifications (solid rock, fractured rock, cave-in rock, minor cavity and mayor cavity) are colored in dark blue, green, red, cyan and yellow, respectively; the drift layout of each orebody level is represented in black.

The rock condition block model for orebody 1 (Figure 9) is developed for one level, with nine drifts drilled at 1052 m level; a clear trend of fracture and cave-in rock can be identified (see red circle, Figure 9), which might be related with geotechnical problems found, as from internal reports from LKAB [21]. Some cave-in and cavity rock classes are found at the end of the fan-holes located in the middle of each ring, which may be related to the drilling of already fractured

rock in upper levels. The upper part of these fan-holes often reached rock mass affected by blasts from upper levels matching these features (see yellow circles, Figure 9b).

Figure 9. Rock condition block model in 3D for orebody 1 orebody: a) top-isometric view; b) isometric view.

The large-scale geotechnical block model for orebody 4 (Figure 10a) includes two levels; the lower level comprises nine drifts drilled at 1056 m and the upper level encompasses two drifts at 1031 m level. On the left side of the plot (see red circle), there is an area with a high density of green and red color that corresponds to a fracture and cave-in zone; this belongs to a steeply zone with softer or more fractured rock going through the two levels, as from internal reports [21]. Figure 10b shows a zoom-in representation of the problematic area represented in Figure 10a (see red circle).

Figure 10. Rock condition block model in 3D for orebody 4 orebody: a) top-isometric view; b) Zoom in of the rock condition block model in 3D for orebody 4 orebody: Front view (left), Top view (centre), isometric view (right).

5.3. Risk of borehole collapse planning model

As explained in section 5.1, the rock condition block model can characterize geotechnical issues around the boreholes. Figure 8a shows that problems in the measured charging length (black line drawn at the contour of the hole) are often generated when a big fractured zone is found (see green zones in fan-holes 6 and 7 starting from left) or when a fractured zone follows a cave-in zone or vice versa (fan-holes 3 and 8 starting from left), even if there is a short solid rock zone between them (fan-holes 5 and 9 starting from left). Figure 8b demonstrates, on the other hand, that when solid rock condition is dominant, no charging issues are expected.

The prediction model for the probability of risk of collapse in the blasthole is carried out for the test fan-holes described in Table 1. Fan-holes have been grouped based on the existence and length of zones where solid rock, fractured

rock, cave-in and cavities along the hole are defined. In this case, minor and major cavities are considered as one class, hence, the analysis is done only for four different rock classes. The charging length has been used to estimate a mean charging length/blasthole length (L_c/L_b) ratio and to assess hole chargeability. Two different types of classifications have been carried out: Table 5 (cases 1 to 4) shows the combination when only solid rock or when solid rock plus fracture, cave-in or cavity rock classes appear in the fan-hole and Table 6 (cases 5 to 8) describes the combination when there are more than two rock classes. Cells highlighted in the left part indicate the rock class and its length interval considered. The number of blocked and non-blocked blastholes, the mean charging length/blasthole length (L_c/L_b) ratio and the estimated risk of collapse are also shown for each combination. The latter is assessed from the value of the L_c/L_b ratio for each combination considering the amount of blocked or non-blocked blastholes. In this way, the smaller the L_c/L_b ratio and the higher the number of blocked blastholes, the higher the risk of collapse for each combination and vice versa.

According to Table 5, the existence of a large fractured zone (length, $L > 15$ m) or a large cave-in zone ($L > 2$ m) are always related with blocked blastholes. The mean L_c/L_b ratio in these combinations is low (0.24 and 0.45, respectively), which indicates that there is a high risk that the collapse occurs in the first half of the hole. The presence of one of these two combinations in a hole will indicate high risk of collapse. On the other hand, only solid rock or the existence of short fractured zones ($L < 3$ m) or cave-in zones ($L < 2$ m) normally show no collapse in the blasthole, with a mean L_c/L_b ratio above 0.90. These combinations thus predict no risk of collapse in the hole. An intermediate result is found for fractured zones between 3 and 15 m. Although the number of blocked blastholes for this combination exceed the non-blocked holes, there are also many cases where the blasthole was not blocked. In this way, a medium risk of collapse has been considered. No combination for solid rock and cavity combination has been found in the test holes analyzed. A medium risk of collapse has been considered when there is a large cavity zone ($L > 2$ m) and no risk of collapse for small cavities ($L < 2$ m). This is assumed because the existence of a big cavity may increase the difficulty of charging the hole, since the hose used to charge the hole may get stuck or cannot cross the cavity. Notwithstanding, further measures for these two combinations should be necessary to validate the assumption.

Table 5. Combination when only solid rock or when solid rock plus either fractured, cave-in or cavity rock classes appear in the fan-hole. L is length (m). Grids for solid rock, fractured rock, cave-in and cavities are highlighted in blue, green, red and yellow, respectively.

Case	Solid rock	Fractured rock			Cave-in		Cavity		No. Blasthole		Ratio L_c/L_b^1 (mean $\pm$ std)	Risk of collapse
		$L < 3$	$3 < L < 15$	$L > 15$	$L < 2$	$L > 2$	$L < 2$	$L > 2$	Blocked	Non-blocked		
1		✓							0	7	0.98 ± 0.02	No risk
2	a	✓		✓					10	0	0.24 ± 0.27	High
	b	✓	✓						12	9	0.7 ± 0.4	Medium
	c	✓	✓						2	23	0.96 ± 0.2	No risk
3	a	✓				✓			2	0	0.45 ± 0.35	High
	b	✓			✓				0	3	0.95 ± 0.03	No risk
4	a	✓						✓	-	-	-	Medium
	b	✓					✓		-	-	-	No risk

¹Mean value of charging length/ blasthole length ratios for each fan-hole.

The risk of collapse for the conditions in Table 6 has the same rock conditions and rock lengths as in Table 5. For cases 6, 7 and 8, the number of test holes is limited thus, the risk of collapse assumption is assessed based on the results from Table 5. Any combination with the existence of a large fractured zone (length, $L > 15$ m) and/or a large cave-in zone ($L > 2$ m) is correlated with high risk of collapse; since any of these situations separately normally block the blasthole, their combination with another rock class type will increase this risk of collapse. This occurs for cases 5a, 5b, 5c, 5e, 6a, 6b, 7a, 7b, 8a, 8b, 8c, 8d, 8e, 8f, 8i and 8j. From them, only cases 5a, 5b, 5c, 5e, 6b, 7a, 8d, 8e have been found in the test holes. These combinations show a mean L_c/L_b ratio lower than 0.5 and the majority of blastholes collapsed. An exception occurs when fractured zone (medium or large) is followed by a large cave-in zone or when just a large cave-in zone appears. In this case, the blasthole is collapsed at the end or along the cave-in zone, independently of its location. Large cavities and/or cave-in zones at the collar or large fractured zones along the hole normally collapse the blasthole from the collar giving a zero L_c/L_b ratio. Combinations for cases 6a, 7b, 8a, 8b, 8c, 8f, 8i and 8j have not been found in the test fan-holes and their risk of collapse classification has been assumed based on the results from Table 5.

On the other hand, the existence of any combination based on short fractured zones ($L < 3$ m), cave-in zones ($L < 2$ m) and/or cavity ($L < 2$ m), normally show no blasthole blocked, with a mean L_c/L_b ratio above 0.90. These combinations indicate no risk of collapse and comprise cases 5f, 6f, 7d, 8k, 8l. Medium risk of collapse is assumed for combinations with fractured zones between 3 and 15 m, short cave-in zones ($L < 2$ m) and/or both large or short cavities (cases 5d, 6c, 6d, 6e, 7c, 8g and 8h). L_c/L_b ratio for these combinations are measured between 0.7 and 0.9.

An example of the risk of collapse results from the rock condition block model are plotted in Figures 11a and 11b, with the respective combination case number added to each fan-hole.

Table 6. Combination when more than two rock classes appear in the fan-hole. L is length (m). Grids for solid rock, fractured rock, cave-in and cavities are highlighted in blue, green, red and yellow, respectively.

Case	Solid rock	Fractured rock			Cave-in		Cavity		No. Blasthole		Mean ratio L_c/L_b^1	Risk of collapse
		$L < 3$	$3 < L < 15$	$L > 15$	$L < 2$	$L > 2$	$L < 2$	$L > 2$	Blocked	Non-blocked		
5	a	✓		✓		✓			2	0	0.15 ± 0.21	High
	b	✓		✓	✓				4	0	0.35 ± 0.4	High
	c	✓	✓			✓			6	0	0.65 ± 0.08	High
	d	✓	✓		✓				5	4	0.79 ± 0.20	Medium
	e	✓	✓			✓			2	0	0.8 ± 0.2	High
	f	✓	✓		✓				0	6	0.98 ± 0.02	No risk
6	a	✓		✓				✓	-	-	-	High
	b	✓		✓			✓		1	0	0.52	High
	c	✓	✓					✓	-	-	-	Medium
	d	✓	✓				✓		-	-	-	Medium
	e	✓	✓					✓	-	-	-	Medium
	f	✓	✓				✓		-	-	-	No risk
7	a	✓				✓		✓	1	0	0	High
	b	✓				✓	✓		-	-	-	High
	c	✓			✓			✓	-	-	-	Medium
	d	✓			✓		✓		0	1	1	No risk
8	a	✓		✓		✓		✓	-	-	-	High
	b	✓		✓		✓	✓		-	-	-	High
	c	✓		✓	✓			✓	-	-	-	High
	d	✓		✓	✓		✓		1	0	0.25	High
	e	✓	✓			✓		✓	1	0	0.1	High
	f	✓	✓			✓	✓		-	-	-	High
	g	✓	✓		✓			✓	1	0	0.8	Medium
	h	✓	✓		✓		✓		-	-	-	Medium
	i	✓	✓			✓		✓	-	-	-	High
	j	✓	✓			✓	✓		-	-	-	High
	k	✓	✓		✓			✓	-	-	-	No risk
	l	✓	✓		✓		✓		0	1	1	No risk

Figure 11. Risk of collapse planning model. a) Analysis for an expected problematic ring (Orebody 1, ring 4); b) Analysis for an expected good ring (Orebody 3, ring 2).

5.4. Large scale application of the risk of borehole collapse model

In line with the procedure followed in section 5.2, the risk of collapse model can also be used for evaluation of larger mining areas. Since the model indicates the risk of collapse along the whole fan-hole it can be represented in a 2D map with the collaring coordinates of the horizontal XZ plane plotted in the absolute projection. Figures 12a and 12b show the full-scale representation of the risk of collapse model for orebodies 1 and 4 respectively. The prediction model is represented over the mining layout for each level, allowing to determine problematic areas at the operational level. Figure 12a represents the mining layout at level 1052 in orebody 1 and Figure 12b shows the mining plan design at level 1056 m in orebody 4. The later also shows two drifts represented outside the layout which corresponds, as commented above, to data of the upper level at 1031 m. Figures 12a and 12b show the collar position of the fan holes in orebodies 1 and 4, respectively. The color of each collar position indicates their risk prediction of hole collapse. Similar to Figure 9, orebody 1 (Figure 12a) shows a problematic zone on the left side of the plot with high and medium risk of collapse in the fan-holes. This matches internal reports of the mine [21] where this zone has stopped production due to the impossibility to charge the holes (most of them blocked) and due to stability problems in the roof and walls of the drifts. For the case of orebody 4 (Figure 12b), the problematic zone described in Figures 10a and 10b indicate a zone with medium and high risk of collapse in the holes, where charging problems are expected.

Figure 12. Risk of collapse planning model: a) orebody 1; b) orebody 4.

6. Conclusions and discussion

The development of a geotechnical rock condition block model and a risk of borehole collapse model has been carried out, with reference to MWD records, aiming at minimizing problems during the charging of holes. Such problems are closely related with non-optimal rock fragmentation after blasting. The study has been applied for individual blasts and for the large-scale, comprising 11 drifts with 102 fan-holes from five orebodies, in the first case, and 20 drifts with 5060 fan-holes from two orebodies in the second, located in the Luossavaara-Kiirunavaara AB's (LKAB) Malmberget underground iron ore mine, Sweden.

[8]'s work has been improved into a geotechnical block model to simplify the quantitative assessment and automatic recognition of the rock trends. For that, a thorough correction of the MWD parameters has been carried out in order to minimize external influences different than the rock mass. This comprises: (i) filtering of unrealistic values, (ii) removing of systematic peaks due to the addition of a new rod, and (iii) correction of the hole depth influence.

The risk of borehole collapse model has been developed by comparing different combinations of rock classes from the geotechnical block model with the charging length of 102 production fan-holes. The model considers the existence and length of solid rock, fractured rock, cave-in and cavities along the hole. The assessment of the number of blocked and non-blocked fan-holes and the charging length/blasthole length ratio has been used to assign high, medium or low risk of collapse to each combination. The results indicate collapses in the first half of the fan-holes for the high risk, collapses in the second half of the fan-hole for the medium risk and no collapses along the hole for the non-risk holes.

The two models have been applied to the full scale for two orebodies in the Malmberget mine, Sweden, which comprises 20 drifts and 5060 fan shape long-holes. In them, zones with fractured rock, cave-in and cavities have been

identified, and according to internal reports from LKAB, these zones are related with geotechnical problems that coincide with the impossibility to charge most of the holes. Considering that, presently, the charging procedure is carried out with no prior information of the rock mass condition, the two models presented here provide an early information of the rock mass condition around the boreholes and the prediction of expected charging problems before starting the operation. In addition, the risk of borehole collapse model can be used as a decision-making tool in the daily routine for production planning; re-drilling can be ordered in advance which makes the charging more effective since it minimizes unexpected problems during the procedure. These features have the potential of improving rock fragmentation, with the final purpose of optimizing production.

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