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Photogrammetric calculation of JRC for rock slope support design

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Abstract

For mining and civil engineering projects rock slope stability is an essential part of safety and financial considerations. While large-scale stability can be simulated using rock mass properties, at smaller scale local variations in rock properties become significant and failure purely along discontinuities is possible. Plane failures, wedging and rockfall are common occurrences at this scale. These are often prevented by bolting or shotcrete support. For temporary slopes such support measures can be costly, and an ability to simulate possible failures along discontinuity planes becomes useful for evaluation of support necessity. Conventional methods for evaluating rock mass properties require making assumptions for scale effect in upscaling for an effective slope scale. The scale effect is generally considered to be negative, while some studies show positive or no scale effect. Deriving parameters to elaborately represent mechanical properties of rock mass in a scale of a mining project would require conducting a large scale in-situ test. Representativeness of one test for a large region of interest is questionable. Photogrammetric recording of rock joint surfaces aims to automate the obtaining of rock joint roughness without the need for manual sampling and costly laboratory experiments. Measurements carried out manually using a profilometer are compared to results obtained with photogrammetry without human interaction. The results are compared to tilt table tests to obtain a ratio between the natural roughness to profiled roughness. This ratio may be interpreted as a measure of how well the joint surfaces match. The photogrammetric JRC values may be used to predict the strength of rock joints under low stress conditions, where dilatation is not prevented. *Finally discussion is given concerning the applicability of the method and the need for future research.*

1 Introduction

The idea of using photogrammetry to acquire JRC is not new. Recently, new numerical methods and development of customer level camera equipment have enabled the use of low-cost equipment to generate high resolution geometrical data. Nilsson et al. (2012) used off the shelf equipment and reached vertical accuracy of 0.5-1.0 mm compared to laser scanning. Uotinen et al. (2015) used a similar method and reached a point density of 16 pts/mm². Iakovlev et al. (2016) demonstrate how the digitally measured values compare to manually measured ones and attempted to convert the constant normal load test results to constant normal stiffness by removing the contribution of dilatation. In this paper, we continue the research to develop practical tools for rock slope support design. Our approach is suitable for plane failures, wedge failure and for rockfall. First we introduce the problem at hand, the photogrammetric approach used and the method to calculate the digital JRC. Next, the manual profilometer measurements are compared to photogrammetric results. Additionally the results are compared to tilt table tests. Finally, we discuss the results and give our recommendations on future research.

Stability of rock mass is an essential issue in many mining and civil engineering projects, as rock failures are a general safety issue. They are costly to clean up even when they do not happen to directly impact safety, and blocked passages incur losses due to the caused downtime. Often the rock is simulated as a single mass, only including jointing of the mass as a weakening parameter of the model. While this may work well

for large-scale evaluations, it is generally accepted that at a small scale there are issues of joint stability and rockfall, shown by the tendency to attempt to prevent such events by the use of shotcrete and bolting. In some cases, especially those of temporary slopes and tunnels, using such methods of strengthening the rock can become costly and certain risks are accepted instead. It is however possible to make a visual overview of an exposed rock surface to estimate the need for additional investigations, i.e. simulations of failure risks. Based on such simulations, risks can be reduced by either forcibly removing the risky area e.g. using blasting, or by using appropriate support methods. This study focuses on finding methods for obtaining parameters and simulating such small-scale failure risks. (lakovlev 2015)

Fractures or discontinuities in rock play a significant role on rock mass stability and this is taken into account in failure criteria as a parameter of how fractured the rock mass is, rather than behaviour of single discontinuities, as large-scale failures are almost invariably caused by a combined failure of the components of the blocky product of existing discontinuities. (Muralha et al. 2013, Bandis et al. 1981). This is, of course, an inevitable simplification, as it is nearly impossible to know the exact structure of discontinuities within a large rock mass. For blocky rock mass, the shear strength of single discontinuities may become critical.

Evaluation of the strength of a single discontinuity builds on Mohr-Coulomb relationship between the peak shear strength and the normal stress (1), which represents a case of a planar surface. This relationship has been expanded by Patton (1966) to represent the relationship for a rough surface (2) in a low normal stress with shear displacement occurring as sliding along inclined surfaces. At higher strength the asperities will break and the true relationship takes a different form. These frictional characteristics were first studied by Barton and Choubey (1977), and then expanded by Barton and Bandis (1990) to the commonly recognized Barton-Bandis criterion for rock joint strength and deformability (3).

$$\tau = c + \sigma_n tan\varphi \tag{1}$$

$$\tau = c + \sigma_n \tan\left(\varphi_b + i\right) \tag{2}$$

$$\tau = \sigma_n \tan\left(\varphi_r + JRC \log_{10} \frac{JCS}{\sigma_n}\right) \tag{3}$$

Where:

au is the peak shear strength,

 σ_n is the normal stress,

c is the cohesion,

 φ is the angle of friction,

 φ_b is the basic friction angle,

i is the angle of asperity,

JRC is the joint roughness coefficient,

JCS is the joint wall compressive strength and

 φ_r is the residual friction angle.

The components for shear strength of discontinuities are illustrated in Figure 1. The base component is the basic frictional component, which represents a direct relationship between normal stress and shear strength based on frictional characteristics of the rock, weathered or not. On top of that any roughness, composed of geometrical variability of the joint and any asperities it has, creates extra shear resistance. It has however been shown that often the effect of the roughness component decreases as the joint length increases. This is partly due to the change of effective roughness with scale, and partly due to changes in

intact asperity strength. Thus, a rougher joint can be expected to be affected by scale more than a smoother one. On the other hand, behaviour of jointed rock mass cannot be directly deduced from joint behaviour studies alone. For instance, it has been shown that closely jointed rock mass with smaller blocks is likely to have higher peak shear strength than one with larger blocks. (Bandis et al. 1981)



Shear displacement

Figure 1 The two shear strength components of basic friction and joint roughness, and their length scale effect according to Bandis et al. 1981. In general, longer joint samples lead to reduced shear strength. There is, however, no universal relationship for all joint types

Though it is recognised that mechanical behaviour of rock joints vary as a function of scale, the size and direction of this effect is not consistent across studies. Many studies show negative scale effect, which means decreasing strength with increasing joint size, consistent with the idea of shear strength consisting residual friction plus a scale-dependent roughness component. Other studies show positive or no scale effect. Summary of previous studies that examined the scale dependence can be found in Tatone & Grasselli 2013.

Overall the jury is still out on the defining factors of scale effect, however at least lower roughness can be assumed to diminish it, while correlation of roughness and undulation can determine the direction of the scale effect. For example, smooth joints with high undulation could be expected to have a positive scale effect, while rough planar joints would have a negative scale effect. Studies comparing well matching joints (Kutter & Otto 1990; Leal-Gomes 2003) suggest a positive scale effect for fresh well-matched joints, possibly due to higher effective undulation in matching joints, as mismatched joints have virtually no effective undulation.

Both natural and man-made slopes of various sizes are a common occurrence in many civil and mining engineering projects, e.g. road embankments and open pit mine slopes. Slope design includes consideration of economic, environmental and safety factors, of which slope stability focuses heavily on the safety aspect. However, consideration of economic risk means that slope design is also affected by economic factors, i.e. costs of slope failure compared to costs and probability of stability, especially in mining projects. In general, slope stability design consists of determining acceptable safety factors or probabilities for larger failures as well as rockfall, obtaining necessary geological and geotechnical information and simulating slopes using some method of analysis. A safety factor is defined as the ratio of maximum forces resisting failure to forces driving failure. For more temporary slopes, using failure probabilities may be appropriate, as clean-up and other failure costs are considerably smaller than for e.g. a long-term civil engineering project, where reasonable certainty of long-term stability is preferred. (lakovlev 2015)

To simulate a slope the geotechnical properties of the slope material are necessary, as well as understanding of groundwater and climate conditions. Often literature values are used based on rock or soil type, as laboratory tests can be costly, time-consuming and inaccurate due to scale effects and other reasons. When past failure data is available, parameters can be back-calculated by simulating the failures. Geotechnical properties include material strength(s), weathering, and water permeability, as well as geological structure of the area, especially of discontinuity and weakness zones. Armed with the necessary parameters, which can be expressed as deterministic values or probabilistic distributions, a slope is simulated and analysed for failure possibilities. For failures beyond the height of several meters, simulation is done as rock mass analysis. For smaller failure possibilities, where failures might occur purely along single discontinuities, discontinuity stability analysis may be used.

Limit equilibrium analysis can be used when only a safety factor needs to be obtained, and it is computationally light, though it requires simplification of geometrical and geological data. Numerical methods, on the other hand, are used to more fully simulate the response of rock mass to various conditions, such as faults and groundwater conditions. Due to limits on computational resources, many discontinuity properties are input as weakening parameters of the rock mass, and only critical discontinuities and weakness zones are simulated as separate zones. However, it has been noted that slope failures advance through internal discontinuities and weakness zones of a rock mass, and lately work has been done to incorporate such a failure method into slope stability simulation. (Suikkanen 2014).

2 Methods

Eight rock joint surfaces were tested for evaluation of roughness characterization between traditional hand- measured approaches and photogrammetric roughness analysis in the rock mechanics laboratory of the department of civil engineering at Aalto University. The sample set consisted of three diorite samples and five glimmerite samples. The samples were collected by hand from post-blast and post-collapse sites by carefully removing loose surfaces with a hammer or by selecting visually undamaged joints from leftover rocks. The research extends studies conducted by lakovlev et al. (2016), by expanding the sample pool with five additional samples and implementing an image processing routine for enhancing the accuracy of the photogrammetric modelling process.

The joint roughness parameters were obtained with joint tilt tests, by hand measuring with profilometer and by hand-measuring surface roughness amplitudes from the sample surfaces. The hand measured length normalized profilometer JRC reading was used as the benchmark variable. The basic friction angle (ϕ_b) for the samples were determined by using a three-core tilt test. The joint compressive strength (JCS) of each sample was determined by using Schmidt Hammer tests. The JCS values did not differ significantly from UCS values determined in previous laboratory tests. The obtained parameters are provided in the results and discussion section. Finally, a photogrammetric procedure for roughness evaluation was conducted for inspection of applicability of photogrammetric calculation of JRC for estimating the friction angle of a joint surface.

2.1 Photogrammetry

The camera used for photographing of the samples was a Canon EOS 600D system camera with Canon EF 35mm f/2 IS USM objective lens, which has low optical distortion. The joint samples were photographed indoors in a consistent lighting environment with 4500 lx illuminance, measured at the center of the sample surface. ISO value was set to 100 to keep the noise in minimum. The shooting distance of 100 cm was selected to enable the sample surfaces to fit the images as whole. Aperture value of f16 was selected according to the shooting distance in order to fit the complete sample surfaces in the sharpness area of the photographs. Semi-automatic shooting mode was utilized for the exposure time to be selected automatically by camera enabling optimal exposure.

While photographing the whole sample surface to be photographed should fit in the picture. When optimal distance is being determined, it should be taken into account that the sample also fits the sharpness area,

which may be at an angle to the sample surface. The sharpness area is defined by the near distance of sharpness and the far distance of sharpness. The sharpness area is commonly referred to as depth of field (DOF). The principle of DOF is presented in Figure 2. The DOF can be calculated when the focal length, aperture value, subject distance and circle of confusion (CoC) are known. CoC for Canon 600D is 0.019 mm.



Figure 2 Example of front (193 mm) and back (315 mm) depth of fields with 1000 mm focus distance, 0.019 mm circle of confusion, and aperture f/16

For determination of the total DOF, the near limit for sharpness is calculated with Equation (4) and far limit for sharpness with Equation (5). For these equations, a hyperfocal distance is needed, and it can be calculated by using Equation (6). (Greenleaf, 1950). Hyperfocal distance considered to be the nearest distance in which objects at infinity are acceptably sharp when lens is focused to this distance. In that case it means that all objects from infinity to one-half of hyperfocal distance are acceptably sharp.

$$D_N = \frac{s(H-f)}{H+s-2f} \tag{4}$$

$$D_F = \frac{s(H-f)}{H-s} \tag{5}$$

$$H = \frac{f^2}{Fc} + f \tag{6}$$

Where:

 D_N stands for near distance of sharpness (mm),

 D_F stands for far distance of sharpness (mm),

H is the hyperfocal distance (mm),

s is the focus distance,

f is lens' focal length (mm),

F is the f-number and

c is the circle of confusion (mm)

Determination of the depth of field yields the required f-number for fitting the samples in the DOF area. It should be noted that the ideal focus point may be closer than the center of the sample surface. Especially at very close distances, the required f-value can be quite high (f16 or higher), which may introduce diffraction on the photos. While a general rule of thumb suggests the optimal f-value to be two steps above the lowest f-value of the lens, a more elaborate analysis of lens performance is suggested. The chosen lens is considered to have virtually no distortions (-0.2 % barrel distortion) to worry about, and it maintains very good performance in terms of diffraction till f11, and at f16 the performance is still considered to be acceptable with approximately 2000 MTF50 LW/PH (line widths per picture height). For comparison, the maximum resolution for this lens camera combination is 2600 MTF50 LW/PH at f4.

Lightning conditions remained unchanged during the photoshoot, as photographing was carried out indoors in a room without windows. Lightning was created by utilizing room's 6 fluorescent lamps (3 on each side of the room), the nearest lamp at a distance of 2.2 m from the central point of the sample. Each lamp had two Philips TL-D 36W/830 fluorescent lights with length of 1.2 m. Also three fluorescent lamps were set above the samples in height of approximately 50 cm from the sample surface. These three lamps had each two pieces of Philips TL-D 58W/830 fluorescent lights, each with length of 1.5 m.

The imaging configuration is illustrated in Figure 3. The photoshoot was conducted by applying a camera stand for the camera. This configuration removes possible motion blur from the images, and enables imaging in low light conditions, as the possible movement of the camera during a shot is eliminated. The samples were placed on a rotational platform and rotated around the center point of the sample surface so that the angle of view differs 20 degrees between two consecutive photographs. This results in 18 images per one rotational round. The photography was conducted in three layers, corresponding to 37, 54 and 67 degrees from the horizontal plane defined by the sample surface. The photoshoot delivers 54 images from each sample. Remote control was applied for the photographing, so that no physical interaction with the camera is required during a rotational round. This assures that the camera position remains unchanged between shots. Changing from one layer to another on the other hand requires readjusting the camera position.



Figure 3 The imaging configuration viewed from side (on left) and from above (on right) with 54 camera locations in three rings at angles 37, 54 and 67 degrees containing 18 locations at 20 degree intervals as presented in lakovlev et al. (2016)

2.2 Image processing

The produced RAW images were converted to JPEG file format by using Canon Digital Photo Professional software version 3.14.47. The resulting images were post-processed by an image processing routine to segment potential error sources out from the images. The primary concerns were to eliminate the unchanged background in every image, and to segment out other areas outside of the DOF region. The routine implements an iterative image segmentation by Protiere and Sapiro (2006) that utilizes geodesic computation by generalizing weights for the geodesic distance. This routine is a semi-automated segmentation that requires user to provided seed regions describing the areas to be split. As the surface samples rotate between images, the seed regions were selected to separate the background, the sample surface and rotational platform from the images. Different shooting angles require the areas to be defined separately, but as the samples are almost of the same size, one setting is sufficient for one shooting angle. All images from one shooting angle were processed with the same regional definition, and minor changes to the regions was required when chancing to another shooting angle.

2.3 3D Model formation

The 3D point cloud formation was conducted with VisualSFM 0.5.25 software. The software utilizes SFM (structure from motion) technique that localizes features from 2D images and builds a 3D model reconstruction according to localized features (Ullman, 1979). Then images are matched with each other by SifGPU algorithm. The SiftGPU implemented in VisualSFM is modified version of Lowe's (1999) SIFT-algorithm, which converts the photographs to collection of local feature vectors to find and compare matching features between different photographs. SiftGPU distinguish special features, called DoG key points (Different of Gaussian), from the photographs. Found DoG key points are analyzed between photographs to find matching features. (Wu, 2007) The matching features are then combined to a sparse 3D point cloud by a multicore bundle adjustment routine (Wu et al. 2011).

The sparse reconstruction is then expanded to a dense reconstruction by PMVS/CMVS (Furukawa, 2010) routine. The CMVS (Clustering Views for Multi-view Stereo) creates clusters i.e. part models, which are then combined with PMVS (Patch-based Multi-view Stereo) function to one dense point cloud. The resulting point cloud is then saved as PLY (polygon file format). After point cloud construction, redundant parts are cropped off with Cloud Compare 2.6.0 software. Finally, the point cloud can be triangulated by applying a 2D-Delaunay triangulation (Delaunay, 1934) for the best fit plane of the surface, scaled to actual size with known dimensions, and saved in STL (Standard Tessellation Language).

2.4 Photogrammetric JRC calculation

The photogrammetric JRC calculation is conducted by following a procedure presented in lakovlev et al. (2016). The reference coordinate system for the digital surface models is established by SVD (Singular Value Decomposition) routine, where the orthogonal base vectors are set as the new coordinate system, as illustrated in Figure 4a. After establishing the coordinate system, a sectioning plane is defined by taking a dot product of the base vector in the shearing direction and the plane normal. The sectioning routine is illustrated in Figure 4b. Then a search routine is applied to derive the 2D roughness profile in the shearing direction. The roughness profile is established by calculating the intersections of the shearing plane and the surface triangles that are hit by the plane from the corresponding line and plane equations.



Figure 4 Illustration of sectioning routine for digital roughness characterization: the reference coordinate system definition (a) and the sectioning plane in shearing direction (b), presented according to lakovlev et al. (2016)

The roughness characterization is performed by digital JRC calculation with slope length method (Tse & Cruden, 1979) by applying a normalization of sectioning plane by 0.5 mm sampling interval. The sampling interval was selected according to the sample window used for originally deriving the applied function. The normalization procedure is conducted by taking the mean value for height in a sampling interval, as this sampling resulted in the best match for studies conducted in Sirkiä (2015). The slope length method applies the root mean square (RMS) value from the local slopes of the profile with intervals between measured data points. The relationship with JRC and RMS presented as,

$$JRC = 32.2 + 32.47\log(Z_2)$$
(7)

$$Z_2 = \sqrt{\frac{\sum_{i=1}^{N-1} (z_i - z_{i+1})^2}{(N-1)ds^2}}$$
(8)

where:

- Z_2 stands for the RMS,
- z is the height of the profile above reference line,
- N the quantity of measures and
- ds the distance between measures.

3 Results

The sample surfaces were analysed for definition of a reference measure to be used in validation of digital JRC analysis. The JRC measures for the surfaces were evaluated by measuring three profiles for each sample, and recording the amplitude, profile length, and the Barton comb profile for each profile line. The profilometer measurements were compared against the roughness profiles corresponding to JRC values according to Barton and Choubey (1977). The amplitude measurements were converted to JRC values. Finally the derived JRC values were scaled to effective JRC measures with scale transformation proposed by Barton and Bandis (1982),

$$JRC_n = JRC_0 (L_n/L_0)^{-0.02JRC_0}$$
(9)

where:

JRC_0 and L_0 refer to 100 mm laboratory scale samples and

 JRC_n and L_n refer to sample size.

The JRC values derived by digital roughness characterization routine and by the traditional measures are presented in the Table 1. The effective JRC values are presented next to the derived parameters. Finally the average and the highest values for derived JRC parameters are presented for a comparison of effective roughness. A comparison of JRC values from different evaluation methods are presented in Figure 5, with manual profilometer measurements and digital predictions presented on the left side, and a comparison of JRC derived from tilt table testing with hand measured JRC presented on the right side.

Sample	Length (mm)	JRC(0) Highest	JRC(N) Highest	JRC(0) Mean	JRC(N) Mean	JRC(0) hand	JRC(N) hand
Diorite-1	113.6	15.3	14.7	12.8	12.3	10-12	9.7-11.6
Diorite-2	112.4	16.0	15.4	14.4	13.9	14-16	13.5-15.4
Diorite-3	92.1	15.6	16.0	14.3	14.7	12-14	12.2-14.3
Glimmerite-1	111.5	9.5	9.3	8.1	8.0	10-12	9.8-11.7
Glimmerite-2	124.6	12.2	11.5	10.4	9.9	14-16	13.2-14.9
Glimmerite-3	80.2	14.1	15.0	13.0	13.7	4-6	4.1-6.2
Glimmerite-4	93.3	12.2	12.4	10.3	10.5	4-6	4.0-6.1
Glimmerite-5	81.3	19.6	21.3	18.5	20.0	16-18	17.1-19.4

Table 1	Results of JRC digital and hand measurements. 0 denotes raw measurement and N denotes
	normalized measurement using Equation 6





Some characteristics are not fully covered by the roughness profiles. lakovlev et al. (2016) presents that the diorite samples have clear undulation, steppedness or extreme asperities that a two-dimensional profile is unable to describe. Additionally, the glimmerite surfaces are more uniform, but the Glimmerite-3 and Glimmerite-4 samples seem to be part of a large undulation, and the sample Glimmerite-5 has large diagonal fibre structure that may contribute significantly to mechanical behaviour of the surface.

The digital roughness parameters are generally in the same order of magnitude than the traditional parameters, but overestimates hugely the roughness of samples Glimmerite-3 and Glimmerite-4. This overestimation seems to be generated from small scale variations, as the curve comparison presented in Figure 6 matches better with the traditional measurements. Excluding these samples as outliers, the overall performance of the digital JRC calculation seems to be following the nominal line in Figure 5 (left). The JRC values derived from tilt table testing are clearly lower than the manual measured values (Figure 5, right). Iakovlev et al. (2016) presents that this effect might be resulting from poor matedness (Figure 6) in the tilt testing. The tilt testing results are presented in more detail in Table 2. The table also lists an estimated correlation with the hand measured JRC values and the JRC values derived from the tilt table testing. This correlation is expressed as tilt JRC per profile JRC, and can be considered to represent the matedness of the tilt sample. Dividing the tilt test JRC with the profilometer JRC (with implied JMC = 100 %) gives an estimate of the Joint Matching Coefficient, which is a percentage of the matedness.



Figure 6 Perfectly matching joint with high matedness (JMC = 100 %) on left, poorly matching joint with low matedness (JMC = 0 %) on right. Modified from Zhao (1997)

Table 2	Barton-Bandis parameters obtained for tilt table testing with profile JRC and interpolation of
	JMC with tilt JRC per profile JRC

Sample	фь	φ _r	JRC	JRC	JCS	JMC
	(°)	(°)	(tilt)	(profile)	(MPa)	(tilt/profile)
Diorite-1	31	31	6.0	12	216	0.50
Diorite-2	29	29	5.3	8	207	0.66
Diorite-3	31	29	6.1	10.5	194	0.58
Glimmerite-1	19	17	6.2	9.0	25	0.69
Glimmerite-3	19	17	4.2	5.0	25	0.84
Glimmerite-5	19	15	11.4	13.3	23	0.86

			0 mm		50 mm		100 mm
		Diorite-1 Measurement-1					4 - 6
		Diorite-1 Measurement-2					6 - 8
	<i>JRC</i> = 0 - 2	Diorite-1 Measurement-3		~~~~			6 - 8
	JRC = 2 - 4	Diorite-2 Measurement-1	\sim	~~~~			12 - 14
		Diorite-2 Measurement-2	~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~	~~~~~			16 - 18
	<i>JRC</i> = 4 - 6	Diorite-2 Measurement-3	\sim				
		Diorite-3 measurement-1	~~~~	\sim			
	JRC = 6 - 8	Diorite-3 Measurement-2		~~~~			8 - 10
	<i>JRC</i> = 8 - 10	Diorite-3 Measurement-3	~~~				12 - 14
		Glimmerite-1 Measurement-1					6 - 8
	JRC = 10 - 12	Glimmerite-1 Measurement-2	~~~~				
		Glimmerite-1 Measurement-3					4 - 6
	JRC = 12 - 14	Glimmerite-2 Measurement-1				<u></u>	× 8 - 10
		Glimmerite-2 Measurement-2					8 - 10
	JRC = 14 - 16	Glimmerite-2 Measurement-3					- 10 - 12
	JRC = 16 - 18	Glimmerite-3 Measurement-1		~~~~		~~~~	4 - 6
		Glimmerite-3 Measurement-2	\	~~~~			6 - 8
	<i>JRC</i> = 18 - 20	Glimmerite-4 Measurement-1	<u>~~~</u>				<u> </u>
		Glimmerite-4 Measurement-2		~~~~~		~	6 - 8
0 5 cm 10		Glimmerite-4 Measurement-3				~~~~~	6 - 8
		Glimmerite-5 Measurement-1	-2	~	\sim		14 - 16
		Glimmerite-5 Measurement-2		~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~~			14 - 16

Figure 7 Roughness profiles and corresponding JRC values according to Barton and Choubey (1977) (on left), and photogrammetrically derived JRC profiles (on right with red color)

Additionally, the digital roughness profiles calculated with the photogrammetric routine are compared with JRC curves, digitalized from curves presented by Barton & Bandis (1977). The results of this comparison are presented in Figure 7. The original roughness curves were measured with a profilometer and drawn by hand. As discussed by Tatone & Grasselli (2013), the authors wish to remind that JRC began as a curve fitting parameter. Additionally the subjectivity of JRC determination by profile comparison should be recognized, and it should be noted that the definition would benefit from expertise and experience in analysis of rock surfaces. Matching the photogrammetric JRC calculation with slope length method. These measures, while more subjective to the computerized calculation, seem to be matching more accurately the traditional parameters derived by profilometer measurements.

4 Discussion

In this study we used the camera on a stationary tripod and broad array of lights to emulate the evenness of cloudy day lighting. We could record the rock joint in continuous circles of camera locations at three different angles. These requirements cannot be met when performing in-situ photogrammetry in open-pit or underground mines. Technical requirements cannot be set for every foreseeable combination of conditions and equipment. However, a method to establish the acceptable limit should be possible to describe explicitly. Such a description should include how to determine the geometry for the photogrammetry, how to determine the resolution requirement, how many pictures are needed, how to measure the required amount of light. The post-processing of the images differs very little between laboratory and in-situ photogrammetry.

Poturovic et al. (2015) carried out two series for replicas in constant normal load and constant normal stiffness conditions. They showed that the friction angle and dilatation are key factors that determine the shear strength of rock joints. For underground, the CNS condition is more realistic and dilatation is suppressed. Both CNS and CNL produce the same peak strength. It may be possible to numerically remove the dilatation to simulate CNS condition response from CNL test data. One approach could be to use the Patton's (1966) interpretation of dilatation as a tangent between load and displacement. In CNS the suppressed dilatation is then translated into additional normal force.

To test the method towards larger scales, an affordable option would be to construct a large 2 m x 1 m tilt table and do blind predictions and then perform tilt tests. If these results are correct, the next step would be to construct a static load CNL shear test. This is the laboratory scale limit, but open pit mines provide opportunities for back-calculation of wedge failures. To overcome the effect the unknown variables (weathering, blasting induced damage, exfoliation, thermal effects e.g.), large quantities of failures should be analyzed simultaneously.

KTH Royal Institute of Technology and Aalto University have carried out small scale shear testing and scanned the surfaces using photogrammetry. These samples may be used as a benchmark for numerical simulations. If the numerical results can be calibrated to match the experiments, testable predictions can be given both for smaller samples (subsampling) and for larger samples (upscaling, e.g. large scale tilt table test). It would be beneficial to use several numerical codes to validate the codes to experimental results.

Finally, the numerical modelling of jointed rock mass should be further developed. This includes a more realistic fracture network with fracture set sequencing and termination. This enables the photogrammetry to record a fracture network intersecting the open surfaces and then generating the corresponding minimum energy extension to inside the rock mass. Ultimately, this results in a rigid synthetic rock mass. If particle models are added the model becomes full synthetic rock mass: crystalline material, fracture network and mechanical properties of rock joints. This may be considered as the long-term goal.

5 Conclusions

In this paper, we describe a method suitable for the digitization of rock joint surfaces. We describe a method to obtain a digital measure of joint roughness coefficient JRC. We have compared the results to hand measurements using a profilometer and using the amplitude method. Excluding two worst samples (Glimmerite-3 and -4), the digital JRC estimation works acceptably well and typically the roughness is digitally overestimated only by 1 JRC unit over the trend line. However, individual measurements rarely match the manually obtained values and the difference typically is 2-3 JRC units. The worst measurement (Glimmerite-3) has error of +10 units (+190 % overestimation). The best measurement (Diorite-2) just fits inside the upper limit of manual reading accuracy. We conclude that the method shows promise in ease of use and in reducing the subjectivity of JRC measurements. However, the deviation is currently too large and digital curve fitting appears to produce much better results. Further research is needed before the described method can be used in field measurements.

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