Chapter - C

DRILLING & BLASTING

DELME & PATLATMA

Sıkılama Boyunun Patlatma Verimliliğine Etkisi
The Effect of Stemming Length to Rockpile Fragmentation

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ÖZET Bu çalışmada, sıkılama boyunun atım kütlesi parça boyut dağılımına etkisi incelenmiştir. Her biri, on delikten oluşan ondört atım irdelenmiştir. İlk gruptaki onbir atımda üç metre sıkılama boyu, ikince grupta ise 4,5 m sıkılama boyu uygulanmıştır. Çalışma Konya Çimento Fabrikası Kireçtaşı ocağında gerçekleştirilmiştir. Uygulamalarda, delik boyu 11 m delik çapı ise 89 mm'dir. Patlatma sonrası, atım kütleleri Split-Desktop yazılımı ile değerlendirilmiştir. Elde edilen verilere göre, patar miktarında büyük artış olmuştur. Örneğin, sıkılama boyunun 3 m'den 4,5 m'ye çıkmasıyla, +70 cm boyut %31'den %71'e çıkmıştır. Böylece, zaman kaybının yanı sıra, kırma, öğütme ve taşıma masrafları açısından önemli fark oluşacaktır. Elde edilen sonuçlar ekonomik açıdan çimento sektörü için önemli veriler sunmaktadır.
Anahtar Kelimeler: Sıkılama boyu, patlatma, sıkılama, parçalanma boyutu

ABSTRACT In this study, the effect of stemming length on rock pile size distribution is investigated. Fourteen blasting rounds with 10 holes each are test blasted. In the first group of test eleven blasting rounds were performed where stemming length was 3 m long, and three groups of blast rounds with stemming length of 4.5 m long were tested at the same limestone of quarry of Konya cement factory. In these tests, average length of blast holes was 11 m and diameter of holes was 89 mm. After the trial blasts, rockpile size distribution was measured with standard "compare photo" method and verified by Split Desktop software results. The analysis of measurements showed that large size boulder generation was increased with increase in stemming length. For instance, +70 cm size fragments were increased from 31% to 71% by increasing stemming length from 3 m to 4.5 m. It has been proved that the increase in high percentage of boulder formation means the cost of crushing, grinding and hauling will be increased as well as increased time loss. Evaluating blast efficiency resulted in important economical findings for cement sector.
Keywords: Length of stemming, blasting, stemming, fragmentation

1 GİRİŞ

Sıkılama, patlatma deliğinin serbest yüzeye sınır olan kısmının patlayıcı olmayan bir madde ile tıkanarak, patlayıcıdan elde edilen enerjinin, tıkama başarısına bağlı olarak, daha yüksek verimlilik oranıyla, kayacın kırılmasına yönlendirilmesi işlemidir. İyi sıkılamanın en önemli faydası kayacın daha iyi kırılmasıdır. Ancak iyi sıkılama, aynı zamanda daha iyi öteleme de sağlar. Dolaysıyla, yükleme ve taşıma daha kolay ve ekonomik olarak yapılabilir.

Ülkemizde, tüm dünyada olduğu gibi, sıkılama için, genellikle delik delinirken ortaya çıkan taş tozu ve toprak kullanılır. Bunun nedeni, ucuz ve temininin kolay olmasıdır. Bu yöntemde, ideal bir sıkılama yapılamadığı için patlatmadan elde edilen enerjinin, bir bölümü, atmosfere kaçarak, kaybolmaktadır. Cevizci (2010) patlatma deliklerinin sıkılanması için ilk kez alçı kullanmış ve daha iyi bir tıkama ile başarılı sonuçlar elde etmiştir.

Bilgin (1986) sıkılama için köşeli iri kayaç parçalarının kullanılabileceği hatta daha iyi olacağını belirtmektedir. Ancak bu yöntem, gelişmiş ve yaygın kullanılan Nonel ateşleme sistemde kablo kopmasına sebep olabilir. Böylece, patlamayan delikler kalabilir ve bu son derece tehlikeli bir durumdur. Bu yüzden asla Nonel sistem ateşleme sisteminde, iri taşlar, sıkılama amaçlı kullanılmamalıdır.

Çimento yapımında ve inşaat sektöründe büyük miktarlarda kırılmış kireçtaşı kullanılmaktadır (Cevizci ve Özkahraman, 2012). Parçalanmış kireçtaşı elde etmenin en ucuz yolu, patlatmadır. Patlatılmış kireçtaşı boyut küçültme amacıyla kırıcılara gönderilir. İyi parçalanmış bir atım kütlesi taşıma ve yükleme maliyetlerini düşürmenin ötesinde, kırma ve öğütme aşamasında da, büyük karlar sağlayabilir.

Dobrilović vd. (2005) en iyi sıkılama malzemesinin, 16/32 mm boyutlu kırılmış kireçtaşı olduğunu belirlemiştir. Boshoff and Webber-Youngman (2011) küçük çaplı deliklerin sıkılama performansı testi için yeni düzenek geliştirmiştir. Bu düzenekle farklı sıkılamanın kaya kırılmasında farklı etkisi olduğunu belirlemişlerdir.

Bu çalışmada taş tozu ile yapılan sıkılamanın boyuna göre atım kütlesinin fregmantasyonunun nasıl değiştiği irdelenmiştir.

2 METOT

Çalışma, Konya Çimento Fabrikası kireçtaşı ocağında gerçekleştirilmiştir. Şehrin büyümesiyle fabrika ve kireçtaşı ocağı yerleşim alanı içinde kalmıştır. Şekil 1'de Konya Çimento Fabrikası kireçtaşı ocağından bir görünüm sunulmuştur. Konya - Ankara anayolu üzerinde yer alan Konya Çimento Fabrikası kireçtaşı ocağının yakın çevresinde konutlar ve işyerleri bulunduğundan (Şekil 1) bir atımda 10 civarı delik patlatılmaktadır. Böylece patlayıcı miktarı ve patlatmanın çevresel etkilerinin azaltılması amaçlanmıştır.

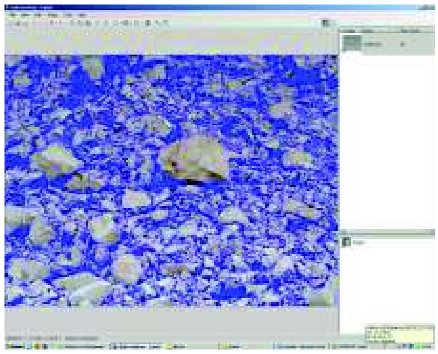
Şekil 1. Konya Çimento Fabrikası kireçtaşı ocağından bir görünüm

Çalışma kapsamında, her biri 10 delikten oluşan 14 atım gerçekleştirilmiştir. Bu atımların 11 adedinde sıkılama boyu 3 m seçilmiştir. Geri kalan 3 atımda ise sıkılama boyu 4,5 m seçilmiştir. Sıkılamada, delik delinirken ortaya çıkan taş tozu sıkılama malzemesi olarak kullanılmıştır. Ortalama delik boyu 11 m ve delik çapı 89 mm'dir. Ateşleme elemanı olarak, 25 ms'lik yüzey gecikme ve 500 ms'lik delik içi gecikmeli tümleşik Nonel kapsüller kullanılmıştır. Konya Çimento Fabrikası kireçtaşı ocağında kireçtaşı özellikleri ve patlatma düzeni Çizelge 1'de özet olarak verilmiştir.

3 ATIMLARIN DEĞERLENDİRİLMESİ

Atımların parça boyut dağılımına yönelik değerlendirilmesi, görüntü analiz yöntemlerinden, Split – Desktop programı ile yapılmıştır. Ayrıca, Standart sayısal fotoğraf yöntemi ile de karşılaştırılmış ve her iki yöntemin birbirine yakın sonuçlar verdiği görülmüştür. Sıkılama boyu 3 m olan atımların parça boyut dağılımı Çizelge 2'de ilk kolonda; 4,5 m olan ilk grup atımların parça boyut dağılımı ise ikinci kolonda belirtilmiştir. Örnek bir atım kütlesinin Split – Desktop programıyla değerlendirilmesi Şekil 2'de gösterilmiştir.

Sıkılama boyunun 3 m'den 4,5 m'ye çıkmasıyla parçalanma ve patar miktarı çok büyük oranda etkilenmektedir (Şekil 3). Sıkılama boyu 3 m olan 11 adet atımın ve sıkılama boyu 4,5 m olan 3 atımın ortalama parça boyut dağılımı ve standart sapmaları Çizelge 3'de sunulmuştur.



Şekil 2. Split – Desktop programıyla atım II'nin değerlendirilmesi

Çizelge 1. Konya Çimento Fabrikası kireçtaşı ocağında kayaç özellikleri ve patlatma patterni

Atımlar	*Süreksizlik yönü/sür.eğimi /atımla açısı*	*Blok boyut indeksi I_b,(cm)*	*RQD (%)*	*Sıkılama boyu (m)*	*Basam. yüks., (m)*	*Ort.dilim kalınlığı (m)*	*Ort. delikler arası mesafe,(m)*
Atım 1	160/30/150	55	68	3	10	2,5	2,5
Atım 2	120/40/60	62	62	3	10	2,5	2,5
Atım 3	120/40/60	55	68	3	10	2,5	2,5
Atım 4	120/40/60	57	62	3	10	2,5	2,5
Atım 5	110/35/50	85	65	3	13,5	2,5	2,5
Atım 6	160/30/150	65	59	3	10	2,5	2,5
Atım 7	130/10/115	80	77	3	10	2,5	2,5
Atım 8	130/10/115	62	70	3	10	2,5	2,5
Atım 9	130/10/115	60	70	3	10	2,5	2,5
Atım 10	130/15/155	58	67	3	10	2,5	2,5
Atım 11	250/20/195	58	72	3	10	2,5	2,5
Atım 12	170/5/90	85	62	4,5	11	2,5	2,5
Atım 13	170/5/90	65	67	4,5	11	2,5	2,5
Atım 14	170/5/90	60	62	4,5	11	2,5	2,5

Çizelge 2. Konya çimento fabrikası kireçtaşı ocağı atımları elek üstü (%) oranları

Tane Boyutu (cm)	Sıkılama boyu 3 m olan atımlar											Sıkılama boyu 4,5m olan atımlar		
	I	II	III	IV	V	VI	VII	VIII	IX	X	XI	XII	XIII	XIV
300	0,0	0,0	4,7	15,1	0,0	0,0	0,0	0,0	0,0	0,0	0,0	6,3	0,0	42,0
200	0,0	6,7	24,6	28,5	0,0	0,0	1,5	0,6	0,0	0,0	0,0	33,1	12,2	55,3
150	2,6	14,1	37,1	33,2	3,5	2,0	9,3	6,5	5,1	0,0	0,5	49,3	34,0	59,6
100	7,4	25,1	46,9	39,5	17,8	16,7	22,2	19,9	13,1	9,1	10,1	61,2	60,4	68,1
70	12,7	35,9	53,2	45,7	28,8	31,1	32,9	37,2	19,1	24,8	23,6	68,5	71,1	73,4
50	22,4	45,1	57,9	50,5	38,3	42,4	41,7	52,3	28,8	39,5	38,7	73,1	75,3	75,5
40	30,5	49,9	60,4	52,8	45,3	48,4	46,9	59,7	37,7	47,2	47,2	74,9	76,8	76,3
30	41,6	54,6	63,2	55,7	54,2	54,3	52,7	66,0	48,8	54,9	55,1	76,4	78,1	77,0
20	53,9	59,5	66,0	59,7	63,8	60,8	58,6	73,0	60,1	63,0	63,1	77,6	80,2	79,2
15	61,2	63,2	68,0	62,3	69,4	65,3	62,5	77,5	66,7	68,0	68,5	79,4	83,1	81,6
10	69,5	67,9	70,6	65,7	75,9	70,7	67,4	82,6	74,2	74,0	74,7	82,7	86,5	84,6
5	79,8	74,5	74,6	70,9	84,0	78,2	74,4	88,8	83,3	81,8	82,7	87,2	90,7	88,5

Şekil 3. Sıkılama boyu 4,5 m olan XIV no'lu atımdan elde edilen yığın görüntüsü

Çizelge 3. Konya atımlarında sıkılama boyuna göre elek üstü (%) oranları

Tane Boyutu (cm)	3 m sıkılama boylu 11 atım ortalaması(%)	4,5 m sıkılama boylu 3 atım ortalaması %)
300	1,8	16,1
200	5,6	33,5
150	10,4	47,6
100	20,7	63,2
70	31,4	71,0
50	41,6	74,6
40	47,8	76,0
30	54,6	77,2
20	62,0	79,0
15	66,6	81,4
10	72,1	84,6
5	71,7	88,8

Örneğin, +150 cm boyut için, sıkılama boyu 3 m olan 11 adet atım ortalaması % 10,4 olmasına rağmen sıkılama boyu 4,5 m olan 3 atımda % 47,6 olmuştur. Aynı şekilde, +70 cm boyut oranı ortalama 31,4'den 71,0' a çıkmıştır.

4 SONUÇLAR VE YORUM

Sıkılama boyu çok dikkatli bir şekilde seçilmelidir. Sıkılama boyu arttıkça, patar miktarı hızla artmakta ve patlatma başarısı düşmektedir. Bu durumda, kırma, öğütme, yükleme ve taşıma masraflarını çok ciddi bir şekilde etkilemektedir.

Aşırı sıkılama boyu seçilmesi durumunda oluşan büyük boyutlu patarların tekrar patlatılması zor tehlikeli olup ciddi maliyet oluşturur.

Cevizci ve Özkahraman (2012), Düzgünlük katsayısı (n) ile kırılma oranı (I_b/K_{50}) arasında anlamlı bir ilişki olduğunu belirlemişlerdir. Ayrıca sıkılama boyunun 4,5 m'den 3 m'ye düşmesiyle, kırılma oranının yaklaşık 1,5 kat arttığını belirlemişlerdir.

Patlatmanın, daha iyi bir patlatma düzeni seçilerek yapılması madencilik ve inşaat sektöründeki patlatmalarda çok büyük kazanç sağlayabilir. Daha iyi bir patlatma düzeni seçilerek yapılan bir patlatma, malzemenin daha iyi parçalanması ve özellikle çimento sektöründe olduğu gibi, ufalama işlemlerinde de büyük önem arz etmektedir. Çünkü daha ince boyutlu malzemenin kırılması ve öğütülmesi daha kolay ve ucuzdur. Kırıcı verimi, daha başarılı bir patlatma ile daha ince boyutlu malzeme üretilerek artar ve ayrıca çimento üretimindeki kırma ve öğütme maliyetleri azalır.

Çimento üretiminde kireçtaşı büyük miktarlarda kullanılmaktadır. Çimentonun ana girdisini % 60-70 oranında kireçtaşı teşkil etmektedir. Ülkemizin çimento üretimi yıllık yaklaşık 60 000 000 tondur (Türkiye Çimento Müstahsilleri Birliği, 2010). Bahsi edilen kireçtaşları da taş ocaklarından patlatma ile kazılmaktadır. Bu yüzden, iyi belirlenen sıkılama boyu ve iyi belirlenen patlatma düzeni, ekonomik açıdan çok büyük kar sağlayacaktır.

Teşekkür

Yazar, katkıları için Prof.Dr.H.Tarik ÖZKAHRAMAN'a ve Konya Çimento Fabrikası Taş Ocağı işletmecisi olan Çamtaş Madencilik'e ve Maden Mühendisi Sayın Fikret OLTULU'ya teşekkür eder.

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Conventional Heading by Blasting in Hard Coal – A Standard Operation with Room for Improvement demonstrated on a Roadway in Coal and Rock

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ABSTRACT Beyond the current technical procedures of deposit exploration by means of full or partial cut roadheading machines, blasting is still used for driving roadway in coal and rock.

When facing small roadway cross sections, rock subjected to high tectonic stress, simple, straightforward and cost effective equipment, limited personnel resources, and higher gas load, safe blasting agents are the first method of choice.

Successful blasting and fast advance essentially depend on the drilling work. The possible advance per round undisputedly also depends on the stability of the surrounding rock and especially on the geometric quality of the drilling within the cut. Simple, understandable cutting procedures, which can be followed by the drilling crew and which correspond to simple and variable templates, are needed.

In the presentation the example of a driveage in coal and rock will be used to demonstrate the room for improvement in the application of safe blasting agents in combination with high quality drilling. A wedge cut will be used to exemplify geometric correlations and the basics of blasting and ignition. Another central point is blasting sequence and timing to win the advance in one single stage of blasting.

1 INTRODUCTION

1.1 Significance of Blasting

The initial question for this presentation at the 23. IMCET was: How can the procedures for heading by blasting in regional hard coal mining in Turkey at the Back Sea be improved under given conditions?

Modern full or partial cut roadheading machines are often employed with the expectation of achieving daily heading rates of 15 m or more. In the real world highly developed technology can often not be operated cost effectively if unexpected factors such as changes in deposit exploration, geological changes, emissions from the deposit or simply insufficient technical equipment or a lack of qualified operators are encountered. When considering and evaluating all factors, conventional heading by blasting can be the method of choice.

In this presentation the room for improvement will be exemplified on a practical example. So the focus is on drilling and blasting. Tried and true methods in the very sense of “best practise” will be presented.

The presentation is based on collaboration between some regional Turkish hard coal mines and ISSA-MINING.

The Author is the director of the RAG Expert Body of Blasting, the special department of blasting in German hard coal mining.

1.2 Advantages of Heading by Blasting:

Some of the central advantages of heading by blasting are:

- Simple machinery (matured technology)

- Machinery is not prone to breakdowns and easily repaired
- Low investment costs (drilling equipment, possibly platform and loader/scraper)
- Fewer blockages by external factors (stoppages in conveyance)
- Reduced involvement of personnel
- Discontinuous operation (heading on demand)
- Available on a short term basis
- Small roadway cross sections
- Short headings
- Can be used in strong geological faults
- Roadway can follow the seam (inclined roadway)
- No requirements on rock solidity
- Can be used in higher CH_4 levels

Notice: Heading in geological layers with desorbable Methane concentrations of 10 to 20 $m^3 CH_4$, in individual cases also above 100 m^3 CH_4,, can only be done safely by conventional heading by blasting.

2 HEADING BY BLASTING

2.1 The Blasting Diagram

The room for improvement is exemplified on a driveage in rock, starting from the present blasting diagram.

Rigid arch supports will be used and drilling will be done manually with rock drills.

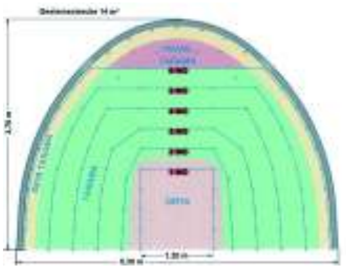

Figure 1: Current Blasting Diagram for Operation in Rock with 14 m²

What can be noticed?

- Number of blasting holes
- Space between the blasting holes

- 5-row inclined cut (wedge cut)
- no auxiliary lines within the cut
- in practice: 3 stage blast
- dangerous delays in the blasting sequence (detonation in unsupported rock) in German “Kantenschuss-Bedingung” (literally: condition of a shot at the edge): named after a testing/proofing setup from safety tests for explosives. 14 explosive cartridges are ignited in a steel shaft (Ø 300 mm) into which a 90° cut has been milled. This configuration simulates the partial or full exposure of an explosive charge due to a time step, e.g. blasting interval 5 next to interval 7. The blast discharge from the perforated blasting hole takes place through the small open angle, which may cause an ignition of CH_4.

2.2 Optimising the Blasting Diagram

Preliminary note: it is a company consulting policy of „ISSA-MINING“ to only make suggestions for changing procedures if a limited modification of existing procedures and structures will likely yield a successive, sustainable improvement. In the mines at the Black Sea we observed a very good practice, e.g. the high competence and danger awareness of the blasting staff.

In the blasting diagram the number of the charged holes resp. the space between them is remarkable.

Even when in the same blasting stage, charged holes must be at a certain minimal distance to avoid possible compression of the charge.

This is the first suggestion for optimising the mentioned blasting driveage:

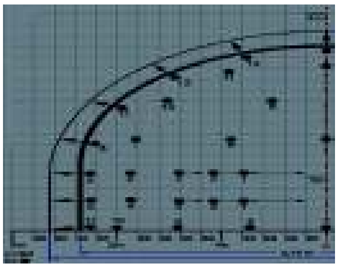

Figure 2: Blasting Diagram Roadway in Rock 14 m²

What can be noticed?

- Almost identical amount of explosives
- Drilling work reduced by 50 %
- Blasting columns are 50 % longer (rock fragmentation)
- One stage blast
- Bigger diameter

These advantages are combined with new blasting agents (see 2.3) to win the entire advance in one single blast.

Tabel 1: Standard Values

Parameter	as is	new
Number of Drilled Holes (BL)	115	50
Meters Drilled [m]	230	100
Drilled Holes/m² [BL/m²]	6,5	< 3
Number of Drilled Holes per m³ [BL/m³]	3,3	1,4
Ø Drilled Hole [mm]	~ 30	~ 40
Length of Blasting Column [m]	0,600	≥0,875
Explosives/Drilled Hole [kg/BL]	0,375	0,91
Amount of Explosives/ Advance [kg/m³]	1,2	1,3
Amount of Explosives [kg]	43,125	45,5
Time Intervals	4	12
Stage blast	3	1

For a roadway cross section of 18 m² the same approach is used.

2.2.1 Roadway in Coal

The optimised result for the driveage in coal at figure 3.

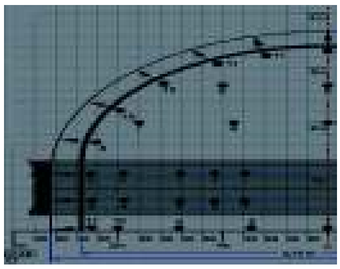

Figure 3: Blasting Diagram Roadway in Coal

2.3 Optimising the Blasting Agents

The present blasting agents had a higher performance than those employed in German hard coal mining.

The application of safe blasting agents is inseparably linked to the strict principles of sequencing blasting stages. Explosives in powder form which are proven to be safe with CH_4 have proven their worth in matters of safety in German hard coal mining for decades.

From **April 2013** on, all explosives in the European Union must be marked in such a way that the whereabouts and use of each piece can be traced. Therefore it is advisable to shift to blasting agents which are equipped with machine readable labels.

2.3.1 Non-Safety Explosives

Non Safety Explosives are only legal for operations purely in rock (Figure 4). If the smallest traces of carbon are present in the advance their application is forbidden and safety explosives are used.

Figure 4: Wasag Gelit 2

The gelatinous non-safety explosive can be used in up to 0.5 vol. % of CH_4, the cartridges are manufactured in 30 mm x 380 mm and 400 g.

2.3.2 Safety Explosives

Safety explosives are divided into the classes I, II and III. They are legally required for operations in coal and for winning. The cartridges are sealed in hoses.
Class I safety explosives (Figure 5) can be used in up to 0.3 vol. % of CH_4.

Cartridge size is 30 mm x 135 mm with 125 g respectively 40 mm x 175 mm with 250 g.

Figure 5: Riocoal A
(RCA, Class I)

Class II safety explosives (Figure 6) can be used in up to 0.5 vol. % CH_4, cartridge size is 30 m x 135 mm with 125 g.

Figure 6: Riocoal C
(RCC, Class II)

Class III safety explosives (Figure 7) can be used in under 1 vol. % CH_4, cartridge size is 30 mm x 135 mm with 125 g.

Figure 7: Riocoal D
(RCD, Class III)

2.3.3 Detonators

Non-fiery (SWS) class 1.4 B rapid detonators with a delay of 30 ms and time intervals of 1 to 16 are used.

Figure 8: SWS-U-Detonator
Time Interval 10

These detonators are already machine readable (Figure 8).

2.3.4 Safety Blasting Cord

To be able to blast precisely within profile in a manner which does not damage the outside rock, in the next stage of optimisation blasting cord is used in the contour row of holes (see Figure 14).

Figure 9:
Safety Blasting Cord 6 g/m

Safety blasting cord of a strength of 6 g/m and a connected blasting column are used.

3. DRILLING

Heading by blasting can be broken down to 4 main stages:
- Drilling
- Blasting
- Removal of blasted rock
- Supporting and backfilling

Drilling is done with rock drills equipped with support legs. It is time-consuming, expensive and causes high physical stresses on the drilling crew. To conserve company and ergonomic resources, the amount of drilling must be kept to a minimum under all circumstances.
To conserve company and ergonomic resources the amount of drilling must be minimised.

3.1 Optimising the Drilling

Each sort of rock has specific demands on drilling and corresponds with the characteristics of the deposit.
The drilling effort is layed out on the basis of empirical data (see table 2) and can be continously improved. From this, the layout of the drilled holes (see figure 10) on the face can be gathered. It is determined by the buttock and the lateral distance.

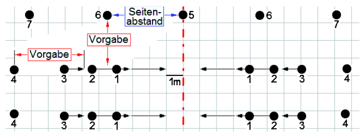

Figure 10: Buttock-lateral Distance
roadway in Coal

3.1.1 Guidelines for Drilling

Tabel 2: GuideValues for Drilling

Empirical Guide Values for Drilling			As-Is Value
Type of Explosive	WG 2 30 mm	WI 30 mm	Jelat Din. 25 mm
Drilling Effort: easy – difficult blasting [BL/m²]	1,8 bis 3,0	2,0 bis 3,4	**6,5**
Amount of Explosive: easy – difficult blasting [kg/m³]	0,8 bis 1,4	1,0 bis 1,5	1,2
Buttock: easy – difficult blasting [m]	1,0 bis 0,7	0,9 bis 0,5	**< 0,4**
easy blasting: e.g.: Coal, Slate (also with Falt Planes)			
difficult blasting: e.g.: Sandy Slate, Sand Stone			

Further practical measures are:
- Marking <u>all</u> starting points for drilling holes
- Drilling the wegde cut with a drilling template.

The result of the blasting depends on the quality of the drilling, the culminating point is how precise the cut comes out.

3.2. Pivoting Method

The wedge cut is the most common kind of cut in hard coal mining. This diagonal cut can be drilled non staggered or staggered in several stages. It is not easy to determine the angle of the first breaking holes with the naked eye.
This is aided by the slewing rate method and the pivoting method. The latter will be discussed here.

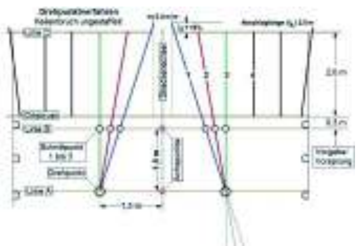

Figure 11: Pivoting Method

3.2.1 Templates for the Pivoting Method

Drawing the template demands only basic knowledge of geometry (see the sketch in figure 11). All angles or drilling layouts of the wedge cut can be drawn from three lines respectively cords. In practice this is drawn up in the rear part of the free roadway.
Line C is just an auxiliary line. It is drawn “Advance plus buttock“ (here 2.0 m + 0.3 m) away from line B. On this line the distances between the ends of the drilling holes are marked. The lines B and C in a distance of 1.5 m are permanently marked with the **central markers** of the center line of the roadway. On line A the **pivoting point** is permanently marked 1.5 m away from the center marking.
To mark the **points of intersection** on line B a straight line (e.g. by means of a cord) is drawn from the pivoting point (line A) to the corresponding end of a rilling hole (line C). The point of intersection is permanently marked, e.g. with coloured tape.
All marked points on lines A and B are in symmetry with the center line of the roadway.
For the practical use of this template only lines A and B are needed. They are drawn with a distance of 1.5 m between them in front of the face (plus additional **buttock**) with the markings for the centerline. The drill operator determines the correct position of the drill rod (length and angle) with plumb-bobs mounted at the pivoting poit and the point of intersection.

Especially with outward-sloping faces the correct depth of the drilled holes is important. It is mneasured from line B plus buttock plus 20 % (see figure 12) The percental addition only applies for the central holes, drilled opliquely.

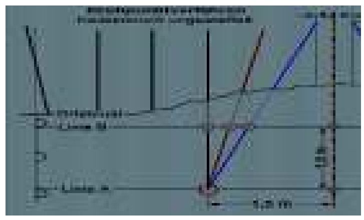

Figure 12: Pivoting Method
Face Sloping Outwards

4 BLASTING CORD

As mentioned above, in the next stage of optimisation blasting cord is used in the contour row of blasting holes.
Some of the central advantages of blasting cord are:
- Blasting precisely within profile with little damage to outside rock
- reduced overbreakage
- improved solidity and service life of the roadway
- increased rock pressure
- strong convergence
- repeated use of the roadway in winning
- etc.

With an amount of explosives of 6 g/m the blasting cord has only limited blasting power and serves as a "linear" means of ignition within the drilled hole.

A small number of explosive cartridges are placed in the drilled hole parallel to the blasting cord in distances of 30 cm or more. They are ignited by the exploding blasting cord (see Figure 13).

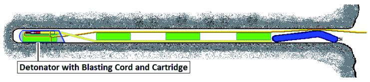

Figure 13: Blasting Cord with Distribution of Cartridges

4.1 Application of the Blasting Cord

The reduced damage to outside rock is due to the reduced amount of explosives in the contour holes, which do not fracture the neighbouring (roof) layers as much.
At the same time the distances between the contour holes are reduced, the "perforating" effect panders to a contour more true to the profile.
The combination of both effects reduces overbreakage, accelerates the settling on the support, slows the fracture of the roof layers and positively influences the rigidity of the roadway cross section.etting load-bearing rock bolts as soon as possible additionally supports these effects.

4.2 Blasting Diagrams with Blasting Cord

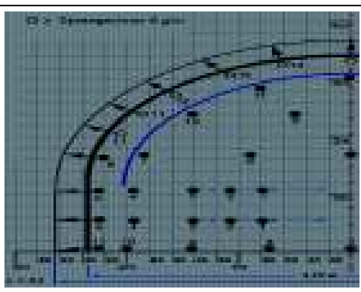

Figure 14: Roadway in Rock with Blasting Cord 14 m²

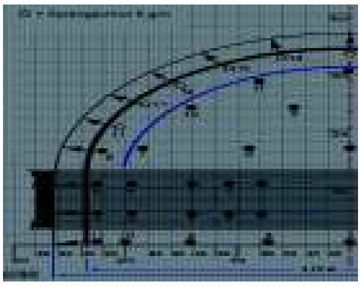

Figure 15: Roadway in Coal with Blasting Cord 14 m²

5 CONCLUSION

Heading by blasting is a standard operation in the preparation if winning. Beyond the highly advanced machinery underground blasting remains economiccaly effective and is the method of choice for many mining applications. Especially in higher concentrations of Methane the highly advanced safety explosives are an alternative. In connection with blasting cord the rock is disturbed as minimally as possible and expenses due to backfilling caused by overbreakage are reduced. Each blasting result is achieved by the quality of mining know-how in drilling. The use of simple tools reduces the anount of work and improves the blasting result.

In German hard coal mining preparational roadway is usually headed with excavated cross sections of 25 bis 28 m². Mainly pursuing bolted arch supports backfilled with mortar matter are used. The quality of the preparational roadway has significantly improved since the presented procedures are implemented. Nevertheless, daily heading rates are 7 to 9 m when four shifts are worked.

For the implementation of the measures mentioned above and to develop further, innovative concepts for drilling, blasting and occupational safety in Turkish hard coal mining, an established, experienced blasting staff with special knowledge is needed. ISSA-MINING has presented a stepped system for training, which starts from the blaster to the blasing overman and culminates with the blasting engineer with many years of experience innational and international mining.

ISSA-MINING is convinced that these technical, structural and safety measures will have a positive effect for the staff and will bring a sustainable reinforcement for hard coal mining in Turkey.

Glückauf

Development of Blast Vibration Predictor Using Statistical Regression Analysis – Some Important Aspects

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ABSTRACT Blasting is the most popular excavation technique in both surface and underground mines. The ground vibration generated due to the propagation of blast waves on the detonation of explosive during blasting is the principal cause for structural and rock damage. The ground vibration from a blast is, in general, measured using a seismograph placed at the point of interest (say the structure). The measured vibrations, in terms of peak particle velocity (PPV), are related to the maximum charge detonated at one delay and the distance of seismograph from the blast point. Eight to ten blast rounds of varying charge/delay and distances are, in general, chosen to monitor the ground vibrations in terms of PPV. A number of scaling factors of these dependencies (viz. Distance and maximum charge/delay) have been proposed by different researchers, namely, square root, cube root, CMRI, Langefors and Kihlstrom (1963), Ghosh – Daemon (1983), Indian standard (1973), etc. Scaling factors of desired type are computed for all the measured 8 – 10 blast rounds. Regression analysis is carried out between the scaling factors and peak particle velocities to establish the coefficients of the vibration predictor equation. In the regression analysis, if the index of determination is found to be accepted the predictor equation is declared established. However, from the statistical point of view, a regression analysis on a small sample population cannot be declared accepted without the testing of hypothesis.

In this paper, the different scaling factors proposed by different researcher have been reviewed. Both from mining and statistical point of view, the step by step approach to establish a vibration predictor equation is proposed.

Key Words Blast vibration, Peak Particle velocity, statistical analysis

1 INTRODUCTION

Drilling and blasting combination is an economical and viable method for rock excavation in mining and civil construction works. During blasting, generation of ground vibrations, air blasts, fly rocks, back breaks, noises, etc. is unavoidable. Amongst all these effects, ground vibration is major concern to the planners, designers and environmentalists. A number of researchers have suggested various methods to assess the ground vibration level during the blasting. Ground vibration is directly related to the quantity of explosive used and distance between blast face to monitoring point as well as with other geological and geotechnical, explosive and design parameters.

Blast induced ground vibration is resulted from the use of explosives, which generates shock waves on detonation. From the interaction of shock waves and rockmass a complex vibration waveform is generated.

Apart from the charge weight and distance between the blast site and monitoring point, the parameters that have the greatest effect on the composition of the ground vibration waveform are:

- Geology between the blast site and the monitoring location
- Accurate timing between blast holes in a detonation sequence

Geological and geotechnical conditions and distance between blast face to monitoring point cannot be altered but the only factor, i.e. quantity of explosive can be estimated based on certain empirical formulae proposed by the different researchers to make ground vibrations in a permissible limit.

2 REVIEW OF SCALING FACTORS

A number of investigators have studied ground vibrations from blasting and have developed theoretical analysis to explain the experimental data. The energy released is considered to be proportional to the square root of charge. Morris, 1950 showed that the amplitude of particle displacement can be given by

$A = K\frac{Q^{0.5}}{R}$ ……………. (1)

Where,

A is amplitude of particle displacement;

K is site constant;

R is the distance and

Q is the charge per delay.

Ground vibration can also be measured by the concept of peak particle velocity. Peak Particle Velocity is defined as the highest speed at which an individual earth particle moves or vibrates as the waves pass a particular site.

There are many predictive equations with different scaling factors to compute explosive weight per delay to attain a specific level of peak particle velocity (PPV). Some of the predictors are given below -

United State Bureau of Mines (USBM, (1980) proposed that PPV is proportional to scaled distance, which is ratio of the square root of the explosive charge weight detonated simultaneously and the distance and is given by,

$v = K[R/\sqrt{Q_{max}}]^{-B}$……………… (3)

Where, v = peak particle velocity (mm/s)

R = distance between blast face and monitoring point (m)

Q_{max} = maximum explosive charge used per delay (kg), and

K, B = site constants which can be determined by multiple regression analysis

Langefors et al (1958) proposed the following relationships for various charging levels to estimate peak particle velocity:

$v = K[\sqrt{Q_{max}}/R^{2/3}]^{B}$……………….. (4)

Where, v = peak particle velocity (mm/s)

R = distance between blast face and monitoring point (m)

Q_{max} = maximum explosive charge used per delay (kg), and

K, B = site constants which can be determined by multiple regression analysis

Ambraseys and Hendron (1968) suggested that any linear dimension should be scaled to the cube root of the explosive charge weight for spherical dispersion. "*An inverse power law was suggested to relate amplitude of seismic waves and scaled distance to obtain the following relationship*" (Pal Roy, 2005).

$v = K[R/(Q_{max})^{1/3}]^{-B}$ ………….. (5)

Where, v = peak particle velocity (mm/s)

R = distance between blast face and monitoring point (m)

Q_{max} = maximum explosive charge used per delay (kg), and

K, B = site constants which can be determined by multiple regression analysis

Indian standard (1973) suggested that the PPV can be related to the scaled distance as given below -

$v = K[{Q_{max}}^{2/3}/R]^{B}$ …………….. (6)

Where, v = peak particle velocity (mm/s)

R = distance between blast face and monitoring point (m)

Q_{max} = maximum explosive charge used per delay (kg), and

K, B = site constants which can be determined by multiple regression analysis

Holmberg and Persson (1979) considered independency of charge quantity and distance and proposed the generalized form of vibration predictor as,

$v = KR^{-B}(Q_{max})^{A}$ ………….. (7)

Where, v = peak particle velocity (mm/s),

R = distance between blast face and monitoring point (m),

Q_{max} = maximum explosive charge used per delay (kg), and

K, A, B = site constants which can be determined by multiple regression analysis.

Ghosh – Daemon (1983) proposed that various inelastic effects cause energy losses during wave propagation in various medium. This inelastic effect leads to a decrease in amplitude in addition to those due to geometrical spreading. They modified the propagation relations of USBM in terms of adding inelastic attenuation factor (α).

For surface mines,

$v = K[R/Q_{max}^{1/2}]^{-B}e^{-\alpha R}$ (8a)

For underground mines,

$v = K[R/(Q_{max})^{1/3}]^{-B}e^{-\alpha R}$ ………… (8b)

Where, v = peak particle velocity (mm/s),

R = distance between blast face and monitoring point (m),

Q_{max} = maximum explosive charge used per delay (kg), and

K, B and α = site constants which can be determined by multiple regression analysis.

Pal Roy (1993) proposed a new predictor equation based on the data collected from different Indian geo-mining conditions. This equation is only valid in the zone of disturbance, i.e. when Q_{max} >0 and v>0 and named as CMRI predictor equation.

$v = n + K[R/\sqrt{Q_{max}}]^{-1}$ …………… (9)

Where, v = peak particle velocity (mm/s)

R = distance between blast face and monitoring point (m),

Q_{max} = maximum explosive charge used per delay (kg),

n = site constants which is influenced by rock properties and geometrical discontinuities,

K = site constants which is related to design parameters.

A summary of the reviewed scaling factors and different predictor equation proposed are presented in Table 1.

3 GOODNESS OF FIT FOR REGRESSION AND CORRELATION ANALYSIS

Statistical investigations establish algebraic relationship between the independent variables and dependent variables. The established algebraic equation helps to predict one quantity exactly in terms of the others. However, the accuracy of this prediction rarely comes close to the actual value and, in most instances, only averages or expected values can be predicted. Pal Roy (2005) has given an account to investigate the statistical acceptance of predictor equation. A summary of the same is given in the following paragraphs.

The regression problem considers the frequency distributions of one variable when another is fixed at different levels, whereas, the correlation problem considers the joint variation of two measurements, when neither of which is restricted. A functional relation between dependent and independent variables does not guaranteed the actual cause and effect; nonetheless it enables prediction of the value of one variable on the condition that prior information about the other is available. If two variables are involved, the variable that is basic of estimation is conventionally called the independent variable and the variable whose value is to be estimated is called the dependent variable. The main use of regression equation is to predict the most likely measurement of one variable from the known measurement of another.

Correlation methods are considered appropriate when interest is centered on measuring the degree to which two variables are linearly related and when both variables are randomly sampled. The fact that two variables tend to increase or decrease together does not imply that the one has any direct or indirect effect on the other. Both may be influenced by the other variables. Such influence can be determined by partial correlation coefficients.

If there are *n* pairs of observations (x1, y1), (x2, y2) etc., the problem that frequently arises is the determination of whether an apparent relation between the two variables is significant, and if having shown it to be significant, to determine the best form of representation. In simple regression analysis the test is whether the data can be represented by the equation,

$y = \beta 0 + \beta 1x$ …………….. (10)

Using the method of least square, the coefficients, β_0 and β_1, are obtained as –

$$\beta_1 = \frac{n\sum x\,y - \sum x \sum y}{n\sum x^2 - (\sum x)^2} \text{ ……………. (11a)}$$

$$\beta_0 = \frac{\sum y}{n} - \beta_1 \frac{\sum x}{n} \text{ ……………… (11b)}$$

To test the significance of an apparently linear relation, the correlation coefficient is determined by,

$$r_{xy} = \frac{\sum x_i y_i - n\bar{x}\bar{y}}{(n-1)s_x s_y} = \frac{n\sum x_i y_i - \sum x_i \sum y_i}{\sqrt{(n\sum x_i^2 - (\sum x_i)^2)(n\sum y_i^2 - (\sum y_i)^2)}}$$

………….. (12)

The correlation coefficient is so characterized that if the relationship between the data can be represented by a straight line, then r = + 1; if, on the other hand, there is no relation at all between the variables, then, r = 0. However, a high correlation coefficient between two variables does not necessarily indicate a casual relationship. There may be a third which may cause the simultaneous change in the first two variables, and produce spuriously high correlation coefficient.

It is important to note that if no assumptions are made about the joint distribution of the concerned random variables, goodness of the prediction and goodness of the estimates, β_0 and β_1 given in equations 10 and 11 cannot be judged. If the data is considered to follow normal distribution, then only r^2 will determine strength of the regression line 10 in other cases not.

The percentage of the total variation of the y's accounted for the relationship with x is 100 r2, and it is in this sense that values of r between 0 and +1 or -1 are interpreted. Thus, when r = 0.9, then 81% of the y's is accounted for by the relationship with x.

Regression analysis of two independent variables – In the case of two independent variables associated in the model ine requires regression analysis of multiple variables. In vibration data analysis, the general empirical relationship between peak particle velocity (v), distance (R), and maximum charge per delay (Q) is,

$$v = KR^{-B}(Q_{max})^A$$

If the equation is expressed in a log-log scale, the equation becomes,

$y = \beta_0 + \beta_1 x1 + \beta_2 x2$…………….. (13)

The maximum likelihood estimates of β_0, β_1 and β_2 are,

$$\beta_0 = \frac{(A - b_1 B - b_2 C)}{n} \text{ ……… (14)}$$

$$\beta_1 = \frac{(c^2 - nH)b_2 - (AC - nG)}{(nF - CB)} \text{ …………… (15)}$$

$$\beta_2 = \frac{(AC - nG)(nD - B^2) - (AB - nE)(nF - BC)}{(nF - BC)^2 - (nH - C^2)(nD - B^2)} \text{……….(16)}$$

Where,

n is the number of data in the analysis and

$A = \sum y,\ B = \sum x_1,\ C = \sum x_2,\ D = \sum x_1^2, E = \sum x_1 y,\ F = \sum x_1 x_2,\ G = \sum x_2 y,\ H = \sum x_2^2$

The calculations for three independent variables are the same as those for two independent variables. The constants $\beta_0, \beta_1, \beta_2$ and $\beta_3$can be obtained in a similar manner.

3.1 Multiple Regression and Partial Correlation

The simple regression coefficients are given by

$\beta_{y1} = \frac{\Sigma' y x_1}{\Sigma x_1^{2'}}$ …………….. (17)

$\beta_{y2} = \frac{\Sigma' y x_2}{x_2^2}$ ……………..(18)

Where is$\sum'$ the summation of deviations from the means, i.e.,

$\Sigma' y x_1 = \Sigma(y - \bar{y})(x_1 - \overline{x_1})$………….. (19)

The correlation coefficient of x_1 and x_2 is

$r_{12} = \frac{\Sigma' x_1 x_2}{\sqrt{\Sigma' x_1^2 \Sigma' x_2^2}}$ …………….. (20)

Similarly,

$r_{y1} = \frac{\Sigma' y x_1}{\sqrt{\Sigma' y^2 \Sigma' x_1^2}}$ and $r_{y2} = \frac{\Sigma y x_2}{\sqrt{\Sigma' y^2 \Sigma' x_2^2}}$…… (21)

Using equation 17 ,18 and 20, when simple regression coefficients and correlation coefficient are calculated, the effect of the third variable is felt. If, for example, β_{yl} = 3.7, β_{y2} = 1.6, r_{12} = 0.75 then, since the higher values of x_1 will tend to be associated with the higher value of x_2 (as r_{12}> 0), the values of y for the longer values of x_1 will tend to be larger than they should be, because they are also associated with the larger values of x_2. The partial regression coefficient of y and x_1 when x_2 is kept fixed as

$r_{y1,2} = \frac{r_{y1} - r_{y2} r_{12}}{\sqrt{(1 - r_{y2}^2)(1 - r_{12}^2)}}$ …………….. (23)

The partial correlation coefficient of y and x_2 when x_1 is kept constant is

$r_{y2,1} = \frac{r_{y2} - r_{y1} r_{21}}{\sqrt{(1 - r_{y1}^2)(1 - r_{21}^2)}}$ ……………… (24)

The application and importance of equation 23 and 24 in vibration data analysis were given by Pal Roy and Gupta (1989c). The correlation coefficients were found to be markedly reduced in their transformation from simple to partial coefficients. It was also found that a positive simple correlation coefficient could be transformed into a negative partial correlation coefficient and vice versa.

3.1.1 Coefficient of Determination

The overall accuracy of any predictor equation is determined by the coefficient of determination (r^2). It is the variation in the dependent variable due to the combined linear influence of the independent variables divided by the total variation in the dependent variable. If r^2 has a value of 0.81, then the independent variables x_1, x_2, … in the regression equation explain 81% of the total variation of the dependent variable y. the formula of r^2 is

$r^2 = 1 - \frac{\Sigma(y_i - y')^2}{(y_i - \bar{y})^2}$ ………… (25)

Where $\bar{y}$ is the mean of observed values of y_i, and y' the value calculated from the regression equation.

3.1.2 Standard error of Estimate

The standard error of estimate (SEE) measure in absolute units the extent of accuracy of the regression equation in predicting the dependent variable y. the formula of (SEE) is

$SEE = \sqrt{\frac{\Sigma(y - y')^2}{(n - k - 1)}}$ ……………… (26)

Where n is the number of observations and k is the number of independent variables present in the model.

When y is normally distributed about the regression line, then 68% of the actual y values will fall within y + 1 (SEE), 95% of the actual values will within y + 2 (SEE) and 99.7 % will fall within y + 3 (SEE).

3.1.3 Adjusted Coefficient of Determination

The coefficient of determination does not account for the number of independent variables in the model. Seber (1977) has shown that the value of r^2 may increase due to the addition of more independent variables in the model. The formula of adjusted coefficient of determination r_a^2 as proposed by Seber (1977) is

$r_a^2 = 1 - \frac{(n-1)(1-r^2)}{(n-k-1)} = r^2 - \frac{k}{(n-k-1)} \frac{\Sigma(y - y')^2}{\Sigma(y - \bar{y})^2}$ .. (27)

The addition of more independent variables in the model decreases the sum of squares of errors which gives a smaller value of r_a^2.

3.2 Hypothesis Testing in Simple Linear Regression

An important part of assessing the adequacy of the simple linear regression model is testing statistical hypothesis about the model parameters and constructing certain confidence intervals. To test hypothesis about the slope and intercept of the regression model, it must be made the additional assumption that the error component εi is normally distributed. Thus, the complete assumptions are that the errors are normally and independently distributed, NID(0, σ^2).

To test the hypothesis that the slope equals a constant, say (β_1,0), the appropriate hypothesis is

H0 : β1 = β1,0

H1 : β1 ≠ β1,0 ……………….. (28)

where it is assumed a two-sided alternative. Now since the εi are NID(0, σ2), it follows directly that the observations yi are NID(β0 + β1xi, σ2). From equation, it is observed that $\hat{\beta}_1$ is a linear combination of the observations y_i. thus, $\hat{\beta}_1$ is a linear combination of independent normal random variables and, consequently, $\hat{\beta}_1$ is N($\hat{\beta}_1$, σ^2/S_{xx}). Furthermore, $\hat{\beta}_1$ is independent of MSE. then, as a result of the normality assumption, the statistic

$t_0 = \frac{\hat{\beta}_1 - \beta 1,0}{\sqrt{MS_E/S_{xx}}}$ …………… (29)

Follows the t distribution with n-2 degrees of freedom under H0 : β1 = β1,0. We would reject H0 : β1 = β1,0, if

|t0| > tα/2,n-2 …………….. (30)

Where t_0 is computed from equation 29.

A similar procedure can be used to test hypothesis about the intercept. To test

H0 : β0 = β0,0

H1 : β0 ≠ β0,0

We would use the statistic

$t_0 = \frac{\hat{\beta}_1 - \beta 1,0}{\sqrt{MS_E[\frac{1}{n} + \frac{\bar{x}^2}{S_{xx}}]}}$ ………………. (31)

And reject the null hypothesis if |t0| > tα/2,n-2.

A very important special case of the hypothesis of equation is

H0 : β1 = 0

H1 : β1 ≠ 0 …………..(32)

This hypothesis relates to the significance of regression. Failing to reject H0 : β1 = 0 is equivalent to concluding that there is no linear relationship between x and y. This may imply either that x is of little value in explaining the variation in y and the best estimator of y for any x is $\hat{y} = \bar{y}$ or that the true relationship between x and y is not linear. Alternatively, if H0 : β1 = 0 is rejected, this implies that x is of value in explaining the variability in y. however rejecting H0 : β1=0 could mean either that the straight –line model is adequate or that even though there is a linear effect of x, better result could be obtained with the addition higher order polynomial terms in x.

The test procedure for H0 : β1 = 0 may be developed from two approaches. The first approach starts with the following partitioning of the total corrected sum of squares for y:

$S_{yy} = \sum_{i=1}^{n}(y_i - \bar{y})^2 = \sum_{i=1}^{n}(\hat{y}_i - \bar{y})^2 + \sum_{i=1}^{n}(y_i - \hat{y}_i)^2$ ……………. (33)

The two components of Syy measure, respectively, the amount of variability in the yi accounted for by the regression line, and the residual variation left unexplained by the regression line. We usually call $SSE = \sum_{i=1}^{n}(y_i - \hat{y}_i)^2$ the error sum of squares and $SSR = \sum_{i=1}^{n}(\hat{y}_i - \bar{y}_i)^2$ the regression sum of squares. Thus, equation (33) may be written as

Syy = SSR + SSE ………….. (34)

Comparing equation

SSR = $\hat{\beta}_1 S_{xy}$ ……… (35)

Syy has n-1 degrees of freedom, and SSR and SSE have 1 and n-2 degrees of freedom, respectively.

We may show that E[SSE/(n-2)] = σ2 and $E[SSR] = \sigma^2 + \beta_1^2 S_{xx}$, and that SSE and SSR are independent. Thus, if H0 : β1 = 0 is true, the statistic

$$F_0 = \frac{SSR/1}{SSE/(n-2)} = \frac{MSR}{MSE} \ldots\ldots\ldots\ldots\ldots \text{(36)}$$

Follows the F1,n-2 distribution, and we would reject H0 if F0>Fα,1,n-2. The test procedure is usually arranged in an analysis of variance table (or ANOVA).

The test for significance of regression may also be developed from equation (29) with β1,0= 0, say

$$t_0 = -\frac{\widehat{\beta_1}}{\sqrt{MSE/S_{xx}}} \ldots\ldots\ldots\ldots\ldots \text{(37)}$$

Squaring both sides, we obtain

$${t_0}^2 = \frac{\widehat{\beta_1}^2 S_{xx}}{MSE} = \frac{\widehat{\beta_1} S_{xy}}{MSE} = \frac{MSR}{MSE} \ldots\ldots\ldots\ldots \text{(38)}$$

Note that,t_0^2 in equation (38) is identical to F0 in equation. It is true, in general, that the square of ai random variable with f degrees of freedom is an F random variable, with one and f degrees of freedom in the numerator and denominator, respectively. Thus the test using t0 is equivalent to the test based on F0.

F-Test for the Goodness of Fit – The F-test is based on the ratio between inter-groups variance and intra-groups variance. The F-distribution is not a single curve but a large family of curves, varying with the degrees of freedom. A null hypothesis is posed that there is no difference between inter-groups variance and intra-groups variance. Then the significance of the F-value is determined. The F-ratio is defined as

$$F = \frac{R^2/k}{(1-R^2)/(n-k-1)} \ldots\ldots\ldots\ldots\ldots\ldots\ldots \text{(39)}$$

The degrees of freedom for F are k and (n-k-1). When the calculated value of F exceeds the statistical table’s critical value for a given significance level, the null hypothesis will be rejected. Otherwise, the observed xi are not significant at the given significance level.

P Value – In statistical significance testing, the P-value is the probability of obtaining a test statistic at least as extreme as the one that was actually observed, assuming that the null hypothesis is true. In this context, value β0 is considered more "extreme" than β1 if, β0 is less likely to occur under the null hypothesis. One often "rejects the null hypothesis" when the P-value is less than the significance level α (Greek alpha), which is often 0.05 or 0.01. When the null hypothesis is rejected, the result is said to be statistically significant.

Traditionally, one rejects the null hypothesis if the P-value is less than or equal to the significance level, often represented by the Greek letter α (alpha). (Greek α is also used for Type I error; the connection is that a hypothesis test that rejects the null hypothesis for all samples that have a P-value less than α will have a Type I error of α.) A significance level of 0.05 would deem as extraordinary any result that is within the most extreme 5% of all possible results under the null hypothesis. In this case a P-value less than 0.05 would result in the rejection of the null hypothesis at the 5% (significance) level.

4 GROUND VIBRATION MEASUREMENT AND ANALYSIS

The experimented blasts were carried out in the drivages of an underground metalliferous mine comprising the host rock Quartz-Chlorite-Sericite-Schist along with some Magnetite to generate vibration data for deriving predictor equation.

The drilling operations are carried out by low profile jumbo drills of having fleet strength 7 units. Jumbo drills are used in stope and development faces. Standard drilling pattern for the face size is given in fig (1).

The explosive used for blasting operation is *Powergel of make Orica explosives* of 40mm cross section diameter and 300 mm length. Each cartridge has weight of 390 gm.

The blast vibrations were measured by using geophone based seismograph of *DS 077, MinimateBlasterTM*. The seismographs were fixed at the competent floor. The measured vibrations were analysed using the software (*Blastware. 10.0*). The details of the experimental data are presented in Table 3.

Statistical analysis of the experimental data is carried out using SPSS software. The vibration predictors proposed by USBM, Langefors and Kihlstrom, Ambrasyes-Hendron, Indian Standard, General, Ghosh-Daemon and CMRI are established and presented in the Table 4.

The regression analysis carried out for to establish the Langefors and Kihlstrom predictor was carried out on log-log scale and is statistically significant as the P-value is less than the significance level (α = 0.05). The F value is well above the statistical table's critical value for the significance level. Hence the null hypothesis (H0 : β1 = 0) is rejected. The correlation coefficient for the analysis is 0.851. It means the model is 85.1% explained by the independent variable ($\sqrt{Q_{max}}/R^{2/3}$) used. The coefficient and constant obtained in analysis are statistically significant (as the modulus of t value is greater than statistical table's critical value for the significance level).

The regression analysis carried out for to establish the General predictor was carried out on log-log scale and is statistically significant as the P-value is less than the significance level (α = 0.05). The F value is well above the statistical table's critical value for the significance level. Hence the null hypothesis (H0 : β1 = 0) is rejected. The correlation coefficient for the analysis is 0.853. It means the model is 85.3% explained by the independent variables (R and Q) used. The coefficient and constant obtained in analysis are statistically significant (as the modulus of t value is greater than statistical table's critical value for the significance level).

The regression analysis carried out for to establish the Ambraseys-Hendron predictor was carried out on log-log scale and is statistically significant as the P-value is less than the significance level (α = 0.05). The F value is well above the statistical table's critical value for the significance level. Hence the null hypothesis (H0 : β1 = 0) is rejected. The correlation coefficient for the analysis is 0.828. It means the model is 82.8% explained by the independent variable ($R/(Q_{max})^{1/3}$) used. The coefficient and constant obtained in analysis are statistically significant (as the modulus of 't' value is greater than statistical table's critical value for the significance level).

The regression analysis carried out for to establish the Indian Standard predictor was carried out on log-log scale and is statistically significant as the P-value is less than the significance level (α = 0.05). The F value is well above the statistical table's critical value for the significance level. Hence the null hypothesis (H0 : β1 = 0) is rejected. The correlation coefficient for the analysis is 0.771. It means the model is 77.1% explained by the independent variable ($Q_{max}/R^{2/3}$) used. The coefficient and constant obtained in analysis are statistically significant (as the modulus of 't' value is greater than statistical table's critical value for the significance level).

The regression analysis carried out for to establish the USBM predictor was carried out on log-log scale and is statistically significant as the P-value is less than the significance level (α = 0.05). The F value is well above the statistical table's critical value for the significance level. Hence the null hypothesis (H0 : β1 = 0) is rejected. The correlation coefficient for the analysis is 0.847. It means the model is 84.7% explained by the independent variable ($R/\sqrt{Q_{max}}$) used. The coefficient and constant obtained in analysis are statistically significant (as the modulus of t value is greater than statistical table's critical value for the significance level).

The regression analysis carried out for to establish the Ghosh-Daemon predictor was carried out on log-log scale and is statistically significant as the P-value is less than the significance level (α = 0.05). The F value is well above the statistical table's critical value for the significance level. Hence the null hypothesis (H0 : β1 = 0) is rejected. The correlation coefficient for the analysis is 0.852. It means the model is 85.2% explained by the independent variable (R and $R/(Q_{max})^{1/3}$) used. The coefficient of $ln(\frac{R}{(Q_{max})^{\frac{1}{3}}})$ and constant obtained in analysis are statistically significant (as the modulus of 't' value is greater than statistical table's critical value for the significance level) whereas the elastic damping coefficient α is not significant. Because of which it does not affect the prediction of PPV significantly in this case.

The simple linear regression analysis carried out for to establish the CMRI predictor was carried out and is statistically significant as the P-value is less than the significance level (α = 0.05). The F value is well above the statistical table's critical value for the significance level. Hence the null hypothesis (H0 : β1 = 0) is rejected. The correlation coefficient for the analysis is 0.925. It means the model is 92.5% explained by the independent variable ($\sqrt{Q_{max}}/R$) used. The coefficient obtained in analysis is statistically significant (as the modulus of t value is greater than statistical table's critical value for the significance level). The constant n has very less effect on defining the model in this case (as the t value obtained is less than the critical value in the statistical table). It is better defined by the variable $\sqrt{Q_{max}}/R$ only.

5 CONCLUSION

Blast vibration monitoring has been carried out in the drift of an underground metal mine. The measured vibration data were analyzed to arrive at the vibration predictors proposed by Langefors and Kihlstrom, Ambrasyes-Hendron,General, Indian Standard, USBM, Ghosh-Daemon and CMRI to establish the statistical significance of the vibration predictor equations, statistical testing, namely test, F test and P-value, were carried out. The t tests are carried out to test the significance of coefficient and constant. 'F' test has also been done to test whether the variances of regression and residuals are alike or not. P value is tested to check the significance of regression analysis.

From the above statistical analysis carried out on the vibration predictors it has been found that -

(I) The P value of regression analysis of Blast Vibration Predictors is less than the significance level 0.05, the null hypothesis (β1=0) is rejected. Their exists at least one β which is linearly related to the response variable ln(PPV) or PPV. The result of Regression analysis is statistically significant and hence is considered in our study.

(II) As the Coefficient of Determination indicates, the most accurate prediction of blast vibration in the Underground Uranium mine (in our case) is given by CMRI Predictor (1993).

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Table 1 – The summary of PPV predictors as proposed by different researchers

Sl. No.	Predictor Name	Predictor Equation
1	USBM Predictor	$v = K[R/\sqrt{Q_{max}}]^{-B}$
2	Langefors and Kihlstrom predictor (1963)	$v = K[\sqrt{Q_{max}}/R^{2/3}]^{B}$
3	Ambraseys-Hendron predictor (1968)	$v = K[R/(Q_{max})^{1/3}]^{-B}$
4	Indian standard predictor (1973)	$v = K[Q_{max}/R^{2/3}]^{B}$
5	General predictor (1964)	$v = KR^{-B}(Q_{max})^{A}$
6	Ghosh – Daemon predictor (1983)[Surface Mine]	$v = K[R/Q_{max}^{1/2}]^{-B}e^{-\alpha R}$
7	Ghosh – Daemon predictor (1983)[Underground Mine]	$v = K[R/(Q_{max})^{1/3}]^{-B}e^{-\alpha R}$
8	CMRI predictor (1993)	$v = n + K[R/\sqrt{Q_{max}}]^{-1}$

Where v = peak particle velocity (mm/s)
R = distance between blast face and monitoring point (m),
Q_{max} = maximum explosive charge used per delay (kg),
n = site constants which is influenced by rock properties and geometrical discontinuities,
K, A, B and α = site constants which are related to design parameters.

Table 2 – Analysis of variance for testing significance of regression

Source of Variation	Sum of Squares	Degreed of Freedom	Mean Square	F_0
Regression	SSR = $\hat{\beta}_1 S_{xy}$	1	MSR	MSR/MSE
Error or Residual	SSE = $S_{yy} - \hat{\beta}_1 S_{xy}$	n-2	MSE	
Total	S_{yy}	n-1		

Figure 1a – Standard Burn Cut Drilling pattern for Development Face of the mine

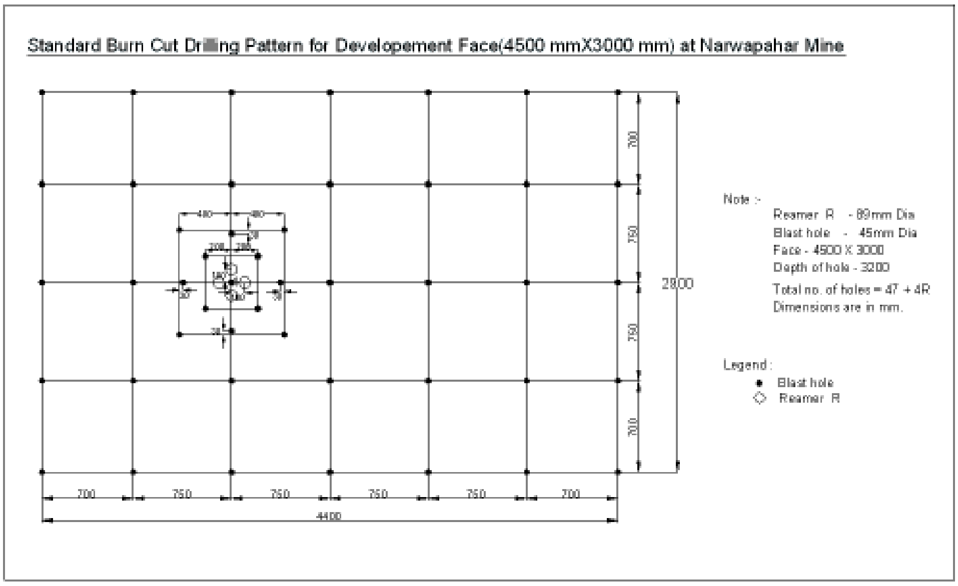

Table 3 – Experimental vibration data collected from blasting in the underground mine

Blast Number	Max. Charge per Delay (kg)	Distance (meter)	PPV (mm/s)
1	8	37	14.78
	8	93	1.96
2	15.4	13	48.13
	15.4	14	42.32
	15.4	63	9.00
3	18	20	32.1
4	15	54	13.9
5	9	98	1.37
6	34	98	8.48
	34	13	63.87
7	27	19.5	23.66
8	24.57	64.00	12.30
9	30.03	67.00	8.06
10	24.57	77.00	16.20

Table 4 – Established Predictor Equations and Correlation Coefficient of the Regression Analysis carried out to establish the same.

Predictor Name	Predictor Equation	Correlation Coefficient (R^2)
Langefors and Kihlstrom predictor (1963)	$v = 73.995[\sqrt{Q_{max}}/R^{2/3}]^{1.649}$	0.851
General predictor (1964)	$v = 103.544R^{-1.129}(Q_{max})^{0.746}$	0.853
Ambraseys-Hendron predictor (1968)	$v = 373.904[R/(Q_{max})^{1/3}]^{-1.209}$	0.828
Indian standard predictor (1973)	$v = 8.142[Q_{max}/R^{2/3}]^{1.238}$	0.771
USBM Predictor (1983)	$v = 194.028[R/\sqrt{Q_{max}}]^{-1.176}$	0.847
Ghosh – Daemon predictor (1983)[Underground Mine]	$v = 162.878[R/(Q_{max})^{1/3}]^{-0.949}e^{-0.007R}$	0.852
CMRI Predictor	$v = 0.725 + 138.564[R/\sqrt{Q_{max}}]^{-1}$	0.925

Table 5: Observed t value for different constants

Predictor Name	t (observed)					
	lnK	***K***	***A***	***B***	***a***	***N***
Langefors and Kihlstrom predictor (1963)	18.12	-	-	8.282	-	-
General predictor (1964)	4.454	-	2.897	-6.861	-	-
Ambraseys-Hendron predictor (1968)	13	-	-	-7.596	-	-
Indian standard predictor (1973)	12.495	-	-	6.361	-	-
USBM Predictor(1983)	15.114	-	-	-8.153	-	-
Ghosh – Daemon predictor (1983)[Underground Mine]	11.082	-	-	-2.357	-0.606 (sig.=0.557)	-
CMRI Predictor	-	12.138	-	-	-	0.328 (sig.=0.748)

The $t_{\alpha/2,n-2}$ value for α=0.05 and n=14 is 2.179.

Table 6: Observed F value

Predictor Name	F Value (observed)	F value (from table)	P Value (Sig.)
Langefors and Kihlstrom predictor (1963)	68.598	4.75	0.000
General predictor (1964)	31.811	3.98	0.000
Ambraseys-Hendron predictor (1968)	57.693	4.75	0.000
Indian standard predictor (1973)	40.468	4.75	0.000
USBM Predictor(1983)	66.468	4.75	0.000
Ghosh – Daemon predictor (1983)[Underground Mine]	31.667	3.98	0.000
CMRI Predictor	147.342	4.75	0.000

Assessment of Blast Effect of Shock Waves on Constructed Facilities and Environment

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ABSTRACT The blast effect problem of shock waves is growing in the area surrounding blasting activities. In addition to damage shock waves may cause on buildings and mining site faclities, they also impact badly human force there, namely the environment. Lately considerable research in the world has been dedicated to the examination and numeric modeling of this phenomenon. Specific standards have been established defining the blast effect margin level of shock waves on facilities and human force there. Numerous numerical and empirical models have been developed to predict and monitor them. In Serbia, there are no standards for the assessment of blast effect of shock waves. This paper deals with the assessment of blast effect of an open pit mine and specific conclusions that have been drawn.
Key words: blasting, shock wave, measurement, assessment, margin level, standard.

1.0. INTRODUCTION

Growing utilisation of blasting techniques in mining results from the fact that single blasting can replace the work of a number of workers and machines for the period of several months. The development in production capacity has caused the use of large amounts of explosive, which on the one hand results in the improvement of technico-economical indicators, and on the other hand in the increase of negative effects related to blast work.

By carrying out blast works, the potential energy of explosives converts into mechanical work. That energy, in the vicinity of the blasting area, breaks and crashes a rock mass further causing fractures and permanent deformations in the rock mass and even further it converts into elastic deformations. Seismic waves propagating through the rock mass induce the oscillation of soil and constructed facilities, impact the environment, etc.

Blasting effects, whether in minig or in any other sectors of economy, can be seen through two categories: *useful* and *useless* work.

* *Useful work* is shown in the form of the crumbling and crushing of the rock material in the limited area around the explosive matter.

* *Useless work* is a phenomenon known as the *seismic effect of explosion* and the work is related to elastic movement, i.e. a rock mass particle oscillation in a highly spacious place round a blast area, what is manifested and experienced as an earthquake. In addition to the seismic effect of an explosion, also, shock wave effect, sound effect, scattering of a blasted rock mass, the occurrence of harmful gasses, etc. are considered to be negative effects of blasting. For that reason increased attention is paid to the study of these occurrences tending to reduce them to tolerable limits.

Elastic deformations induced by explosive charges are an oscillation process, namely the seismic effect of blasting. Elastic deformations induced in that way propagate in the form of elastic waves in the radial direction from the point of explosion. As to the way of carrying of elastic deformations, seismic waves can be divided into two basic groups being: ***volume and surface elastic waves.*** The best known volume waves are longitudinal and transversal ones, while Rayleigh and Love elastic waves are the best known of surface ones. All kinds of elastic waves are induced simultanuously by the effect of explosion in the working environment, whereby their intensity is changed by varying the distance and the environment they propagate through.

The intensity of the seismic effect can be established if we measure one of basic dynamic parameters of the induced environment: *oscillation velocity (v), acceleration (a), or soil movement (x)*. It is possible to make a connection among these parameters if we establish one instrumentally measurable parameter, which enables other parameters to be determined by calculating. One of the most common parameters used for the assesment of seismic intensity is the oscillation velocity of induced soil (v).

2.0. EFFECTS OF BLASTING ON CONSTRUCTED FACILITIES

The intensity assessment of shock waves induced by blast work breaking a rock mass and its impact on construction facilities and an environment will be carried out on the basis of the following criteria:

A. Effects of blasting on constructed and mine facilities

a) Criterion according to the Institute of Physics of the Earth, Russian Academy of Sciences (IPERAS) scale

b) Criterion according to the standard DIN-4150

c) Criterion according to the Russian scale for mine facilities

B. Effects of blasting on environment

a) Criterion according to the standard DIN- 4150.

A. Effects of blasting on constructed and mine facilities

a) The criterion according to the IPERAS scale. One of the most commonly used criteria with us for the assessment of shock wave intensity induced by blasting has been established by the Institute of Physics of the Earth, Russian Academy of Sciences. The Russian scale (Table 2.1.) is of a descriptive type related to the ocsillation velocity of soil particles and the degree of seismic intensity and is given in the form of 12 seismic degrees.

Table 2.1.

Oscillation velocity, v (mm/s)	Level of seismic intensity	DESCRIPTION OF ACTIONS
To 2.0	I	Action is revealed only by instruments
2.0 – 4.0	II	Action is felt only in some cases when there is a complete silence
4.0 – 8.0	III	Action is felt by very few people or only those who are expecting it
8.0 – 15.0	IV	Action is felt by many people, the clink of the windowpane is heard
15.0 – 30.0	V	Plaster fall, damage on buildings in poor condition
15.0 – 30.0	V	Plaster fall, damage on buildings in poor condition
30.0 – 60.0	VI	Air cracks in plaster, damage, damage to buildings that already have developed deformations
60.0 – 120.0	VII	Damage to buildings in good condition, cracks in plaster, parts of the plaster fall down, air cracks in walls, cracks in tile stoves, chimney wrecking
120.0 – 40.0	VIII	Considerable deformations on buildings, cracks in bearing structure and walls, bigger cracks in partition walls, wrecking of factory chimneys, fall of the ceiling
240.0 – 480.0	IX	Wrecking of buildings, bigger cracks in walls, exfoliation of walls, collapse of some parts of the walls
Bigger than 480.0	X - XII	Bigger destruction, collapse of complete structures etc.

Deformations on the facilities, as it can be seen in Table 2.1., occur if oscillation velocity owing to blasting exceeds the fourth degree of the seismic scale. The state of the facilities, soil charscteristics, as well as the number and kinds of blasting activities should be taken into account for the assessment of blasting seismic effects on buildings and other constructed facilities.

b) Criterion according to standard DIN-4150 – In the Federal Republicof Germany, maximal tolerable limits for the values of soil oscillation velocity are regulated in dependence on the sig-nificance and the state of facilities for the frequency span from 5 to 100 Hz. Tolerable limits for the values of the soil oscillation velocity according to DIN- 4150 are presented in Table 2.2.

c) Criteria according to Russian scale for mine facilities - The level of rock mass deformation plays an important role in the protection of mine facilities constructed in a rock massif such as shafts, drifts, tunnels, rise headings, dip headings, chambers, stopes, sublevel posts, hydro-engineering tunnels, bench slopes, etc. Deformation characteristics of a rock massif have an essential impact while determining the threshold of deformations for facilities constructed in the rock masiff. On the basis of experimental measurements, there have been established oscillation velocities of the rock massif in varied mining-geological and mining-engineering conditions whose values (Russian standards) are presented in Table 2.3.

Table 2.2.

Row	**Type of the structure**	**Approximate values of vibration velocity, (v) mm/s**			
		Foundation			**Top floor ceilings**
		Frequency, HZ			All frequencies
		< 10	10-50	50-100	
1	Structures used for craftsmanship, industrial and similar structural structures	20	20 – 40	40 – 50	40
2	Dwelling buildings and structures similar in construction or function.	5	5 – 15	15 – 20	15
3	Structures that because of their particular sensitivity to vibrations do not fall into groups 1 and 2 and are essential for conservation (for inst. as cultural-historical monuments)	3	3 – 8	8 – 10	8

Table 2.3.

Description of occurrences in rock massif induced by seismic wave	**Oscillation velocity, cm/s**
There are no damages.	< 20
The occurrence of insignificant development of fissures induced by previous blasting; locally, falling out of single pieces along previously weakened surfaces.	20 – 50
Intensive development of existing fissures followed by minor caving of rock pieces with the dimensions to 0.2 x 0.2 x 0.2 m; the occurrence of cracks in tectonically weaker material filled fissures; the caving of bench slopes along tectonic deformations.	50 – 100
The development of tectonic fissures and the caving of rock pieces with the dimensions 0.5 x 0.5 x 0.5 m.	100 – 150
Caving from sides and roof of underground chambers along tectonic fissures, the formation of new fissures in undamaged part of the rock mass, collapse of safety pillars and benches.	150 – 300
Complete damage of sides and roof of chambers followed by large blocks with dimensions of 1 x 1 x 1 and filling up to the half of constructed surface; caving of hard rock slopes.	300 – 400
Complete demolition of rock mass, the caving of large blocks bigger than 1x1x1 m and covering up more than a half of the chamber.	> 400

B. Effects of blasting on environment

*** Effects on people in constructed facilities (buildings) according to DIN criteria –** data on vibration assessment in the frequency span from 1 to 80 Hz are given by this standard. It is possible to evaluate any periodical and aperiodical oscillations by the assessment procedure. In the standard, there are stated requirements and approximate stress values of people in flats and rooms used for similar purposes.

Jeopardizing of people by shock waves depends on the following factors: shock wave intensity (strength), frequency, duration of shock waves, frequent recurrence and the period of a day when they occur, the sort and way of work of a shock wave source, individual characteristics and situational circumstances, health state (physical psychical), activity during shock wave stress, the level of becoming used to them.

The assessment procedure of vibrations is taken on the basis of unweighted signals expressed by the vibration intensity KB_F. During assessment the maximal weighted vibration intensity KB_{Fmax}, is determined and if necessary the vibration intensity during assessment KB_{FTr} which are compared with approximate values.

An unweighted vibration signal is a signal limited by the span and proportional to the vibration velocity in the operating frequency range from 1 to 80 Hz.

A frequently weighted signal of vibrations is obtained from an unweighted vibration signal by filtration. The obtained signal is weighted by the calculating procedure according to the relation

$$|H_{KB}(f)| = \frac{1}{\sqrt{1+\left(\frac{f_0}{f}\right)^2}}, \qquad (2.1)$$

where there is: f – frequency in Hz; f_0 = 5.6 Hz (threshold frequency of high permeability filter).

On the basis of the obtained weighted signal, the KB value with time constant τ = 125 ms is calculated based on the relation:

$$KB_\tau(t) = \sqrt{\frac{1}{\tau}\int_{\xi=0}^{\xi-t} e^{-\frac{t-\tau}{\tau}} KB^2(\xi)\cdot d\xi}\ , \qquad (2.2.)$$

While determining weighted KB values, as experience shows, the aberration of 15% occurs.

The measurement of oscillation values must be carried out in the vertical direction (z) with two horizontal directions being at the right angle (x and y).

The assessment of obtained results according to DIN 4150 is carried out on the basis of two KB values:

- KB_{Fmax} - maximal weighted vibration intensity (maximal KBt value),
- KB_{FTr} - maximal effective value in time interval.

The effective value of maximal values in time intervals KB_{FTr} is determined via the relation:

$$KB_{FTm} = \sqrt{\frac{1}{N}\sum_{i=1}^{N} KB^2{}_{FTr}}\ , \qquad (2.3.)$$

Both values (KB_{Fmax}, and KB_{FTr}) are determined separately for all three components in x, y (horizontal) and z (vertical) directions. The assessment is carried out on the basis of that component which is the highest.

Table 2.4.

Row	Workplace	Day			Night		
		A_u	A_o	A_r	A_u	A_o	A_r
1	A workplace where, in the vicinity, there are only industrial plants and possibly flats for owners, managers and monitorial staff and workers on duty (see industrial regions Article 9 Bau NVO, (Land Use Ordinance).	0,40	6,0	0,20	0,30	0,60	0,15
2	A workplace where, in the vicinity, there are predominantly located handicraft facilities (see craft fields Article 8. Bau NVO- (Land Use Ordinance).	0,30	6,0	0,15	0,20	0,40	0,10
3	A workplace where, in the vicinity, there are neither predominantly located industrial plants nor flats (see central areas Article 6. Bau NVO, rural areas Article 5. Bau NVO- Land Use Ordinance).	0,20	5,0	0,10	0,15	0,30	0,07
4	A workplace where, in the vicinity, there are predominantly or exclusively residential areas (see pure residential areas Article 3 Bau NVO, general residential areas Article 4. Bau NVO, small settlement areas Article.2. Bau NVO).	0,15	3,0	0,07	0,10	0,20	0,05
5	A workplace work requiring special protection, for example in hospitals, spa resorts, as well as special areas denoted for that purpose.	0,10	3,0	0,05	0,10	0,15	0,05

Values for assessment should be compared with approximate values: A_u- *lower margin,* A_o*-upper margin and* A_R*- resulting value,* in Table 2.4. under the following conditions:

* if KB_{Fmax} value is lower than (upper) approximate value A_o or the same, then requirements according to this standard are met.

* if KB_{Fmax} is higher than (upper) approximate value A_o then requirements according to this standard are not met.

* for momentary activities which rarely occur, the requirement according to the standard is met if KB_{Fmax} is lower than A_o.
* for more frequent activities, where KB_{Fmax} is higher than A_u but lower than A_o, another step of investigation is required in special cases, namely the determination of the vibration intensity for the assessment of KB_{FTr}. If KB_{FTr}. is not higher than the approximate value A_r, according to the Table 2.4, then the requirements according to the standard are also met.
* the criterion A_r serves for the assessment of highly variable or only momentarily acting variations whose value KB_{Fmax} is higher than A_u, but lower than A_o.

3.0. CONDITIONS OF BLASTING AND MEASUREMENT CONDUCTING

On the basis of laboratory analyses of dolomite from the open pit Zabrdica- in the vicinity of the town of Valjevo, the following values of the most essential physico-mechanical properties have been determined:

* Comprehensive strength (mean values)	
- in dry state - in water saturated state	156 MPa 135 MPa
* Volume mass with interstices	2.82 g/cm^3
* Volume mass without interstices	2.85 g/cm^3
* water suction	0.217 %
* Velocity of longitudinal waves	5651.0 m/s
* Velocity of transversal waves	2670.0 m/s

The measuring of seismic effects, namely the oscillation velocity of soil particles (v) induced by blasting was carried out by a measuring device of Vibralok type, a product of the Swedish Company ABEM. Basic characteristics of the seismograph Vibraloc are the following:

- Manufacturer	*ABEM, Sweden*
- Measurement possibilities	*velocity, acceleration, motion and air impacts*
- Number of components	*lateral, vertical, longitudinal*
- Frequency range	*2 - 250 Hz*
- Sampling	*1000; 2000 or 4000 Hz*
- Trigger levels	*0.1 – 200 mm/s*
- Trigger levels of the canal A (air)	*2 – 150 Pa*
- Recording length	*1 – 100 s or automatic length*
- Site location possibilities	*flat floors, plates, foundations, soil etc.*
- Data transfer and analysis	*UVSZ software; UVSZA software*

Measurement points were located at the following locations:

- Measurement point: MM-1; MM-2; MM-3; MM-4.	constructed facility – a house
- Measurement point: MM-5; MM-6.	mine plateau

3.1. DATA ON CONDUCTED BLASTING AND MEASURING NO. I

♣ **Data on blasting: -** The following means were used for this blasting:

- Overall number of boreholes	N_{uk}	= 13
- Overall depth of boreholes	L_{uk}	= 204.5 m
- Amount of explosive – Riogel 60/1785	Q_1	= 165.6 kg
- Amount of explosive Anfo-J	Q_2	= 392.0 kg
- Overall amount of explosive	Q_{uk}	= 557.6 kg
- Max. amount of explosive per one interval	Q_i	= 46.6 kg
- Length of intermediary stemming	L_{ms}	= 1.0 m
- Length of stemming	Ls	= 3.6 – 4.2 m.
- Rudnel detonators, 25/4500 ms	N_u	= 26 pieces
- Amount of slow-burning fuse	L_{sf}	= 1.0 m
- Delay action cap, DK-8	N_{DK}	= 1 piece

♣ **Instrumental observations** – The recording of seismic waves was carried out with four to five instruments. In Table 3.1 there are presented results of measuring for each measurement point.

Table 3.1.

Measu ring point M.P.	Dist. from blastin field to measuring point, *m*	Max. quantity per one inter. *kg*.	Overall quantity of exp. in kg.	Max. oscilation velocity per comp. *mm/s*			Max. oscilation velocity per comp. v_r, *mm/s*	Real result. max. oscillation velocity v_{st}, *mm/s*	Evaluation of measurement results *Hz*		
				V_V	V_T	V_L			*V*	*T*	*L*
MM-1	300,6	46,60	557,6	1,454	2,284	1,305	***3,005***	***2,330***	65,5	13,0	14,5
MM-2	309,5	46,60	557,6	1,108	2,231	1,323	***2,820***	***2,310***	32,2	22,7	14,3
MM-5	148,1	46,60	557,6	6,154	9,045	8,242	***13,697***	***9,120***	37,7	34,6	29,6

3.2. DATA ON CONDUCTED BLASTING AND MEASURING NO. II

♣ **Data on blasting: -** The following means were used for this blasting:

- Overall number of boreholes	N_{uk}	= 28
- Overall depth of boreholes	L_{uk}	= 448,0 m
- Amount of explosive – Riogel 60/1785	Q_1	= 242,48 kg
- Amount of explosive Anfo-J	Q_2	= 1.188,00 kg
- Overall amount of explosive	Q_{uk}	= 1.430,48 kg
- Max. amount of explosive per one interval	Q_i	= 50,14 kg
- Length of intermediary stemming	L_{ms}	= 1,0 m
- Length of stemming	L_s	= 3,5 – 4,0 m
- Rudnel detonators, 25/4500 ms	N_u	= 56 pieces
- Amount of slow-burning fuse	L_{sf}	= 1,0 m
- Delay action cap, DK-8	N_{DK}	= 1 pieces

♣ **Instrumental observations** –The recording of seismic waves was carried out with four to five instruments. In Table 3.2 there are presented results of measuring for each measurement point.

Table 3.2.

Measuring point M.P.	Dist. from blastin field to measuring point, *m*	Max. quantity per one inter. *kg.*	Overall quantity of exp. in kg.	Max. oscilation velocity per comp. *mm/s*			Max. oscilation velocity per comp. v_r, *mm/s*	Real result. max. oscillation velocity v_{st}, *mm/s*	Evaluation of measurement results *Hz*		
				V_V	V_T	V_L			*V*	*T*	*L*
MM-1	291,9	50,14	1.430,48	1,864	3,556	1,966	***4,470***	***3,950***	23,8	13,5	13,8
MM-2	319,0	50,14	1.430,48	2,214	2,282	1,637	***3,576***	***2,550***	30,2	12,9	20,9
MM-3	371,0	50,14	1.430,48	1,131	1,303	1,614	***2,363***	***1,910***	33,7	24,6	23,6
MM-4	195,6	50,14	1.430,48	2,335	2,951	2,719	***4,642***	***3,870***	25,1	33,8	29,3
MM-5	144,6	50,14	1.430,48	12,761	15,607	11,184	***23,054***	***17,390***	32,3	31,6	33,4

3.3. DATA ON CONDUCTED BLASTING AND MEASURING NO. III

♣ **Data on blasting: -** The following means were used for this blasting:

- Overall number of boreholes N_{uk} = 31
- Overall depth of boreholes L_{uk} = 479,5 m
- Amount of explosive – Riogel 60/1785 Q_1 = 410,4 kg
- Amount of explosive Anfo-J Q_2 = 958,0 kg
- Overall amount of explosive Q_{uk} = 1.368,4 kg
- Max. amount of explosive per one interval Q_i = 44,6 kg
- Length of intermediary stemming L_{ms} = 1,0 m
- Length of stemming L_s = 3,5 – 4,2 m.
- Rudnel detonators, 25/4500 ms N_u = 62 pieces
- Amount of slow-burning fuse L_{sf} = 1,0 m
- Delay action cap, DK-8 N_{DK} = 1 pieces

♣ **Instrumental observations** –The recording of seismic waves was carried out with four to five instruments. In Table 3.3 there are presented results of measuring for each measurement point.

Table 3.3.

Measuring point M.P.	Dist. from blastin field to measuring point, *m*	Max. quantity per one inter. *kg.*	Overall quantity of exp. in kg.	Max. oscilation velocity per comp. *mm/s*			Max. oscilation velocity per comp. v_r, *mm/s*	Real result. max. oscillation velocity v_{st}, *mm/s*	Evaluation of measurement results *Hz*		
				V_V	V_T	V_L			*V*	*T*	*L*
MM-1	286,9	44,6	1.368,4	1,681	3,392	1,935	***4.251***	***3,850***	26,6	14,2	14,6
MM-2	327,4	44,6	1.368,4	1,628	1,745	1,771	***2,972***	***1,950***	10,6	14,3	16,3
MM-5	150,8	44,6	1.368,4	8,590	8,122	8,794	***14,734***	***10,830***	29,7	37,0	29,9

3.4. DATA ON CONDUCTED BLASTING AND MEASURING NO. IV

♣ **Data on blasting: -** The following means were used for this blasting:

- Overall number of boreholes N_{uk} = 30
- Overall depth of boreholes L_{uk} = 475,0 m
- Amount of explosive – Riogel 60/1785 Q_1 = 204,06 kg
- Amount of explosive Anfo-J Q_2 = 1.188,00 kg
- Overall amount of explosive Q_{uk} = 1.392,06 kg
- Max. amount of explosive per one interval Q_i = 52,14 kg
- Length of intermediary stemming L_{ms} = 1,0 m
- Length of stemming L_s = 3,5 – 4,0 m
- Rudnel detonators, 25/4500 ms N_u = 60 pieces
- Amount of slow-burning fuse L_{sf} = 1,0 m
- Delay action cap, DK-8 N_{DK} = 1 pieces

♣ **Instrumental observations** – The recording of seismic waves was carried out with four to five instruments. In Table 3.4 there are presented results of measuring for each measurement point.

Table 3.4.

Measu ring point M.P.	Dist. from blastin field to measuring point, *m*	Max. quantity per one inter. *kg.*	Overall quantity of exp. in kg.	Max. oscilation velocity per comp. *mm/s*			Max. oscilation velocity per comp. v_r, *mm/s*	Real result. max. oscillation velocity v_{st}, *mm/s*	Evaluation of measurement results *Hz*		
				V_V	V_T	V_L			*V*	*T*	*L*
MM-1	291,8	52,14	1.392,06	1,955	1,693	1,452	***2,966***	***2,560***	31,5	11,5	11,9
MM-2	315,7	52,14	1.392,06	1,616	2,016	1,504	***2,990***	***2,090***	24,7	22,0	26,4
MM-3	369,0	52,14	1.392,06	0,978	1,038	1,323	***1,942***	***1,470***	34,4	23,0	15,4
MM-4	188,0	52,14	1.392,06	2,885	5,506	3,210	***6,996***	***5,690***	52,6	26,8	25,7
MM-5	129,7	52,14	1.392,06	9,653	13,712	9,567	***19,306***	***15,860***	39,5	37,2	47,2

3.5. DATA ON CONDUCTED BLASTING AND MEASURING NO. V

♣ **Data on blasting: -** The following means were used for this blasting:

- Overall number of boreholes	N_{uk}	= 25
- Overall depth of boreholes	L_{uk}	= 400,0 m
- Amount of explosive – Riogel 60/1785	Q_1	= 254,57 kg
-Amount of explosive Anfo-J	Q_2	= 1.150,00 kg
- Overall amount of explosive	Q_{uk}	= 1.404,57 kg
- Max. amount of explosive per one interval	Q_i	= 57,36 kg
- Length of intermediary stemming	L_{ms}	= 1,0 m
- Length of stemming	L_s	= 3,5 – 4,0 m
- Rudnel detonators, 25/4500 ms	N_u	= 50 pieces
- Amount of slow-burning fuse	L_{sf}	= 1,0 m
- Delay action cap, DK-8	N_{DK}	= 1 pieces

♣ **Instrumental observations** – The recording of seismic waves was carried out with four to five instruments. In Table 3.5 there are presented results of measuring for each measurement point.

Table 3.5.

Measu ring point M.P.	Dist. from blastin field to measuring point, *m*	Max. quantity per one inter. *kg.*	Overall quantity of exp. in kg.	Max. oscilation velocity per comp. *mm/s*			Max. oscilation velocity per comp. v_r, *mm/s*	Real result. max. oscillation velocity v_{st}, *mm/s*	Evaluation of measurement results *Hz*		
				V_V	V_T	V_L			*V*	*T*	*L*
MM-1	315,5	57,36	1.404,57	1,306	1,935	1,790	***2,941***	***2,220***	23,7	17,5	19,8
MM-3	394,3	57,36	1.404,57	1,954	4,036	1,102	***4,617***	***4,060***	22,2	17,6	17,1
MM-4	191,7	57,36	1.404,57	2,382	4,236	3,112	***5,773***	***4,430***	58,9	28,2	25,4
MM-5	178,0	57,36	1.404,57	12,776	20,504	10,718	***26,429***	***22,750***	35,0	33,5	31,3

3.6. DATA ON CONDUCTED BLASTING AND MEASURING NO. VI

♣ **Data on blasting: -** The following means were used for this blasting:

- Overall number of boreholes	N_{uk}	= 43
- Overall depth of boreholes	L_{uk}	= 344,0 m
- Amount of explosive – Riogel 60/1785	Q_1	= 94,33 kg
- Amount of explosive Anfo-J	Q_2	= 550,00 kg
- Overall amount of explosive	Q_{uk}	= 644,33 kg
- Max. amount of explosive per one interval	Q_i	= 15,27 kg
- Length of intermediary stemming	L_{ms}	= 1,0 m
- Length of stemming	L_s	= 3,5 – 4,0 m
- Rudnel detonators, 25/4500 ms	N_u	= 86 pieces
- Amount of slow-burning fuse	L_{sf}	= 1,0 m
- Delay action cap, DK-8	N_{DK}	= 1 pieces

♣ **Instrumental observations** – The recording of seismic waves was carried out with four to five instruments. In Table 3.6 there are presented results of measuring for each measurement point.

Table 3.6.

Measuring point M.P.	Dist. from blastin field to measuring point, *m*	Max. quantity per one inter. *kg*.	Overall quantity of exp. in kg.	Max. oscilation velocity per comp. *mm/s*			Max. oscilation velocity per comp. v_r, *mm/s*	Real result. max. oscillation velocity v_{st}, *mm/s*	Evaluation of measurement results *Hz*		
				V_V	V_T	V_L			*V*	*T*	*L*
MM-1	367,6	15,27	644,33	0,250	1,042	0,675	***1,266***	***1,060***	9,68	13,6	11,1
MM-4	167,5	15,27	644,33	0,618	1,655	1,105	***2,084***	***1,720***	50,9	35,6	36,8
MM-5	228,3	15,27	644,33	3,423	3,251	4,556	***6,560***	***5,530***	29,8	29,3	37,1
MM-6	160,5	15,27	644,33	4,320	7,842	3,789	***9,721***	***8,140***	32,8	38,7	38,4

4.0. ASSESMENT OF MEASUREMENT RESULTS

The assesment of intensity of shock waves induced by blasting on breaking rock mass and its impact on surrounding facilities and environment, will be conducted on the basis of the following criteria:

A. Effects of blasting on constructed and mine facilities:

a) Criterion according to Institute of Physics of the Earth, Russian Academy of Sciences (IPERAS) scale.

b) Criterion according to the standard DIN-4150

c) Criteria according to the Russian scale for mine facilities

B. Effects of blasting on environment

a) Criterion according to the standard DIN-4150.

In order to conduct the assessment of induced shock waves by these three criteria, in Table 4.1, there have been given recorded values of velocity by components, resulting maximal oscillation velocity, frequency by components, as well as the KB calculated value whose values will be compared with the values presented in Tables 2.1. , 2.2. , 2.3. , 2.4.

To assess the shock wave intensity the following marks were used to fill in Table 4.1.

A. Effects of blasting on constructed and mine facilities

♦ The criterion according to the IPERAS scale (facilities of the third class according to Table 2.1 taken into account):

A – it meets requirements within thresholds of oscillation velocity

B – it does not meet requirements, above thresholds of oscillation velocity

♦ The criterion according to DIN 4150 standard (facilities of the second class according to Table 2.2. taken into account):

C – it meets requirements within thresholds of oscillation velocity

D – it does not meet requirements, above thresholds of oscillation velocity

♦ The criteria according to Russian scale for mine facilities (facilities of the first category according to Table 2.3. taken into account):

E – it meets requirements within threshold values

F – it does not meet requirements, above threshold values.

B. Effects of blasting on environment according to DIN standard (Table 2.4.)

G – it meets requirements within threshold values

H – it does not meet requirements, above threshold values.

In Figure 4.1 – 4.3. there is shown the value of wave component as well as the KB_{fm}. value.

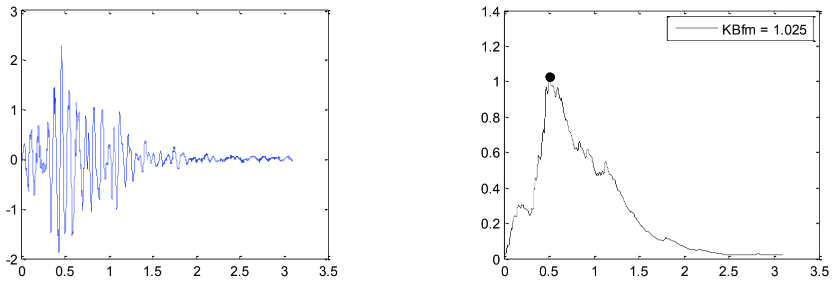

Figure 4.1. Value of V_T and KB_{fm} components. Blasting No.I, measurement point MM-1

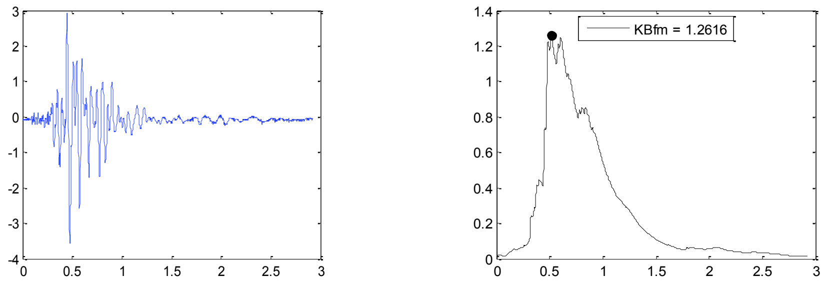

Figure 4.2. Value of V_T and KB_{fm} components. Blasting No.II, measurement point MM-1

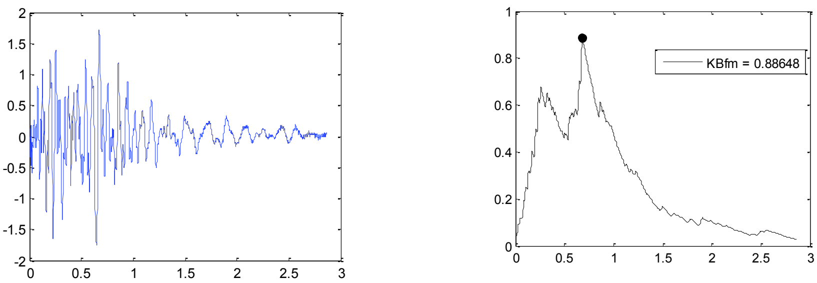

Figure 4.3. Value of V_L and KB_{fm} components. Blasting No.III, measurement point MM-2

Table 4.1 Results of blasting and measuring conducted at the open pit Zabrdica in the vicinity of the town of Valjevo.

Blast. No.	Meas uring point M.P.	Dist. from blastin field to meas. point	Max. quantity per one inter. kg.	Overall quantity of exp. in. kg.	Maximum oscilation velocity per comp. mm/s			Res. max. oscilation velocity mm/s	KB_{fm}	Frequency per components, Hz			Evaluation of measurement results			
					V_V	V_T	V_L			V	T	L	By IFZA	By DIN	Russian stand.	By DIN (KB_{fm})
I	MM-1	300,6	46,40	557,60	1,454	2,284	1,305	***3,005***	***1,025***	65,5	13,0	14,5	**A**	**C**	**-**	**G**
	MM-2	309,5	46,40	557,60	1,108	2,231	1,323	***2,820***	***0,793***	32,2	22,7	14,3	**A**	**C**	**-**	**G**
	MM-5	148,1	46,40	557,60	6,154	9,045	8,242	***13,697***	***3,380***	37,7	34,6	29,6	**-**	**-**	**E**	**G**
II	MM-1	291,9	50,14	1.430,48	1,864	3,556	1,966	***4,470***	***1,262***	23,8	13,5	13,8	**A**	**C**	**-**	**G**
	MM-2	319,0	50,14	1.430,48	2,214	2,282	1,637	***3,576***	***0,899***	30,2	12,9	20,9	**A**	**C**	**-**	**G**
	MM-3	371,0	50,14	1.430,48	1,131	1,303	1,614	***2,363***	***0,886***	33,7	24,6	23,6	**A**	**C**	**-**	**G**
	MM-4	195,6	50,14	1.430,48	2,335	2,951	2,719	***4,642***	***0,462***	25,1	33,8	29,3	**A**	**C**	**-**	**G**
	MM-5	144,6	50,14	1.430,48	12,761	15,607	11,184	***23,054***	***5,917***	32,3	31,6	33,4	**-**	**-**	**E**	**G**
III	MM-1	286,9	44,46	1.368,40	1,681	3,392	1,935	***4.251***	***1,261***	26,6	14,2	14,6	**A**	**C**	**-**	**G**
	MM-2	327,4	44,46	1.368,40	1,628	1,745	1,771	***2,972***	***0,886***	10,6	14,3	16,3	**A**	**C**	**-**	**G**
	MM-5	150,8	44,46	1.368,40	8,590	8,122	8,794	***14,734***	***0,301***	29,7	37,0	29,9	**-**	**-**	**E**	**G**
IV	MM-1	291,8	52,14	1.392,06	1,955	1,693	1,452	***2,966***	***0,762***	31,5	11,5	11,9	**A**	**C**	**-**	**G**
	MM-2	315,7	52,14	1.392,06	1,616	2,016	1,504	***2,990***	***0,752***	24,7	22,0	26,4	**A**	**C**	**-**	**G**
	MM-3	369,0	52,14	1.392,06	0,973	1,038	1,323	***1,942***	***0,462***	34,4	23,0	15,4	**A**	**C**	**-**	**G**
	MM-4	188,0	52,14	1.392,06	2,885	5,506	3,210	***6,996***	***2,191***	52,6	26,8	25,7	**A**	**C**	**E**	**G**
	MM-5	129,7	52,14	1.392,06	9,653	13,712	9,567	***19,306***	***5,871***	39,5	37,2	47,2	**-**	**-**	**-**	**G**
V	MM-1	313,5	57,36	1.404,57	1,306	1,935	1,790	***2,941***	***0,932***	23,7	17,5	19,8	**A**	**C**	**-**	**G**
	MM-3	394,3	57,36	1.404,57	1,954	4,036	1,102	***4,617***	***2,236***	22,2	17,6	17,1	**A**	**C**	**-**	**G**
	MM-4	191,7	57,36	1.404,57	2,382	4,236	3,112	***5,773***	***1,601***	58,9	28,2	25,4	**A**	**C**	**-**	**G**
	MM-5	178,0	57,36	1.404,57	12,776	20,504	10,718	***26,429***	***8,367***	35,0	33,5	31,3	**-**	**-**	**E**	**H**
VI	MM-1	367,6	15,27	644,33	0,250	1,042	0,675	***1,266***	***0,472***	9,68	13,6	11,1	**A**	**C**	**-**	**G**
	MM-4	167,5	15,27	644,33	0,618	1,655	1,105	***2,084***	***0,752***	50,9	35,6	36,8	**A**	**C**	**-**	**G**
	MM-5	228,3	15,27	644,33	3,423	3,251	4,556	***6,560***	***2,071***	29,8	29,3	37,1	**-**	**-**	**E**	**G**
	MM-6	160,5	15,27	644,33	4,320	7,842	3,789	***9,721***	***3,634***	32,8	38,7	38,4	**-**	**-**	**E**	**G**

5.0. CONCLUSION

The estimate of shock wave effects on constructed facilities and the environment, while carrying out blasting activities at the open pit Zabrdica, was made at surrounding constructed facilities according to the criteria of IPERAS, Russian standards for mine facilities and DIN-4150. On the basis of the carried out measurements it can be concluded:

- the recorded values of oscillation velocity at measurement points being within the quarry (the measurement point MM-5 and MM-6), are within threshold values, which according to scales 2.4. *do not affect facilities in the mine.*
- the recorded values of oscillation velocity in the vicinity of the mine (the measurement points: MM-1; MM-2; MM-3 and MM-4), meet requirements within threshold values, thus *do not affect constructed facilities.*
- predominant frequencies range from 15,5 –35,0 Hz, thus do not affect people in the surrounding facilities.
- for more detailed perception of blasting effects on constructed facilities, it is necessary to establish the state of constructed facilities (the way of constructing, the resistance of facilities, the age of facilities, etc.), as well as to monitor occasionally shock waves in the vicinity of the mine.
- in addition to determining of blasting effects on constructed facilities, the KB_{fm} values, namely the impact of rock mass oscillation velocity on environment, were also determined. The KB_{fm} values according to the results presented in Table 4.1 with constructed facilities where measurements were conducted do not exceed threshold values according to.

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Application of Explosives Combined at Kosovo's Limestone-Gremnik Area

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ABSTRACT Current rate of use of limestone in Kosovo has reached the same level with countries in the region, with a growing trend of requests for these construction materials.

Their use has found a wide range of applications in the construction industry and road infrastructure.

Having seen the need and benefits from the use of limestone as well as their specifications, the need of application blast- drilling methods and the use of explosive substances is presented.

This, for us in Kosovo, is very important considering the large number of quarries that we possess.

The reasoning for using explosives in the surface and underground Mining process lies in the achievement of very high productivity.

The purpose of this paper is to provide sufficient data for explosives and methods of application of blasting limestone in Kosovo.

1 INTRODUCTION

The area of Republic of Kosovo belong to central part of Balkan and her extension is between $41^{0}\ 50^{'}\ 58^{''}$ and $43^{0}\ 15^{'}\ 42^{''}$ northern latitude, $20^{o}\ 01^{'}\ 30^{''}$ and $21^{o}48^{'}02^{''}$ eastern latitude, with total area 10.887 km^2.

Because of quality determination of limestone Gremnik area and their application, are done physical-mechanical analysis. From this analysis, conclude that limestones from Gremnik area are usable for:

- Ballast and decorative stone,
- Road construction and tampon,
- Material for drainage,
- Material for hydrotechnic (revetment, plants, dike),
- Material for wall construction.

Figure 1. Morphologic chart of Kosovo 1:200000

Table 1. Physical-mechanical characteristics of limestone of Gremnik area

	Parameters		Value
1	Pressure Resistance σ_p [MPa]	Experiment in dry sampling	101.2
		Experiment in humid sampling	100.4
		Experiment in freezing sampling	90.3-108.2
2	Affinity resistance [MPa]		78.4
3	Los Angeles [%]		23.18
4	Water absorbing [%]		0.54
5	Specific mass [kg/m^3]		2695
6	Volume mass [kg/m^3]		2665
7	Compatibility [%]		97.3
8	Internal angle of friction [^{o}]		35^o
9	Cohesion [daN/cm^2]		90.5

2 APPLICATION OF EXPLOSIVES COMBINED

A matter to be explosive (CE-civilian explosives) must satisfy the condition:
During the process of the brake to release great energy that goes into mechanical work.
The products of the process of the blast (explosion) are fully or partially gases.
Breaking process (exploding) implemented as soon as possible, and for a shorter time released products resulting from the blast.
Fast ignition CE allows oxygen that is in chemical compounds within the mixture and therefore CE may explode in dark places without the presence of oxygen from the atmosphere.

Explosives are chemical preparations and the mixture, which under the influence of heat or mechanical impact quick pass in the gaseous phase, in which release energy and gas.

Explosion (Blast) - is rapid release of energy that follows the release of gases associated with rapid increase in temperature, which is related to the creation of pressure, which made mass crash that has been explosive.

2.1 The Calculation of Drill-Blast Parameters

Determination of breaking drilling parameters based on the distribution of the energy burst CE in rock formation, which thus is caused by specific consumption CE. Transformation of energy is a function of the characteristics of CE releases and features rock formation that receives this energy. The Energy that is transmitted depending on the acoustic impedance factor η1 (Impedance Factor), which is expressed by the equation.

$$\eta_1 = 1 - \frac{(I_{CE} - Ish)^2}{(I_{CE} + Ish)^2}$$

Where:
$\eta 1$ - impedance factor
I_{CE} - acoustic impedance of CE
Ish - rock acoustic impedance
Acoustic Impedance CE, I_{CE} "is defined as the product of CE density “ρ" and speed of detonation “D".

$I_{CE} = \rho * D$

Rock acoustic impedance is defined as the product of volume measure “γ" rock and spreading speed of waves in the rock “V".

$I_{sh} = V * \gamma$

Using energies during mining explosion depends on the right choice of CE.

There are several ways of determining the type of CE, but we will use the way of balancing the acoustic impedance of the CE and rock.

By balancing the acoustic impedance condition, explosive material selection will be: from CE ammonium nitrate in the form of dust, the "Strong Ammonal" and by ammonium nitrate, explosive type "ANFO”. Mixtures of these L.P. during mining in the area create and use a better combination of energy which is be released and this consequently have positive effects on energy rationalization, crushing desired fragmentation, economical cost and high security.

2.2 Explosives type "Strong Ammonal"

Are explosives in powder forms, which are used normally for civilian blasting? Their main composition of ammonium nitrate, trotyl and nitroglycerin, and additional materials are organic fuel for protection from moisture. These explosives find use in all types of mining, especially in combination with other explosives in mining massive. These explosives are not resistant to moisture, so packed in patrons of PVC. Characteristic of these explosives is that they are activated under the influence of mechanical action (shock, friction) or thermal effects (sparks, flames).
Following are the technical indicators LP able dust

Table 2. Technical parameters for some explosives from Ammonium Nitrate

	STRONG AMONAL
Density [kg/m^3]	1.05-1.1
Detonating velocity [m/s]	4100-4300
Gas volume [l/kg]	963
Detonating energy [kJ/kg]	4292
Detonating Transmission [cm]	4-8
Trauzl experiment [cm^3]	380-390
Temperature Of blast [^{o}C]	2567
Specific pressure [kbar]	>10
O_2 balance [%]	+0.09

2. 3 Explosives Type "AN-FO"

ANFO is a mixture of Nitrate Ammonium (Ammonium nitrate) with a Diesel-oil quantity. For using Ammonium nitrate as part of explosive are needed some special features such as a special form and The an attenuation processing as well, trying to keep fuels in liquid state to keep them constantly distributed and to take care that does not come up to unusual mixtures.

Ideal mixing content is 94,3 % Ammonium nitrate and 5,7 % Fuel Oil.

Table 3. The general drill-blast parameters

I	Rock characteristics	Symbol	Value
1	Velocity of longitudinal wave [m/s]	V	2700
2	Rock volume [kg/m^3]	γ	2650
3	Surface specific energy [MJ/m^2]	ε_{sh}	0.00147
II	Explosives characteristics		
1	Velocity of detonating [m/s]	D	4300
2	Density [kg/dm^3]	ρ	1.05
3	Specific energy [MJ/kg]	ε_{lp}	4.24
III	Charge characteristics		
1	Diameter of hole [m]	D_{bir}	0.08
2	Diameter of explosives charge [m]	D_{ng}	0.07
IV	Other characteristics		
1	Diameter of max block [m]	D_m	0.7
2	Bench height [m]	H	10
3	Angle of hole 1 [˚]	α	70˚

Table 4. Technical parameters of AN-FO explosives

	AN-FO (petrammonitis)
Density [kg/m^3]	0.8
Detonating velocity [m/s]	3000
Gas volume [l/kg]	970
Detonating energy [kJ/kg]	920
Transmission of detonation [cm]	4-8
Trauzl experiment [cm^3]	315
Temperature of detonation [^{o}C]	2600
Specific pressure [kbar]	>10
O_2 balance [%]	+0.09

Table 5. Additional drill- blast parameters

V	Other characteristics	symbol	value
1	Acoustic Impedance of rock [kg/sm^2]	I_{sh}	4957
2	Acoustic Impedance of explosives [kg/sm^2]	I_{lp}	4945
3	Diameter of charge in holl [m']	D_{np}	0.07
4	Density of charge in holl [kg/m^3]	Δ=	902.7
5	Relation between hole diameter and charge D_{br} D_{np}	D_{br}/D_{ng}	1.1
6	Fragmentation grade of material	$S=64/D_{max}$	107
7	Impedance factory	η_1	0.84
8	Conjunctive factor between explosives and rock	η_2	0.78
9	Destruction Factor	η_3	0.15
10	Specific charge of explosives [kg/m^3]	q	00
11	Max resistance line (Langford)	W_1	3.4
12	Min resistance line approved [m]	W	3.2
13	Min resistance line in floor [m	W_{dsh}	3.1
14	Over drill of the hole [m]	$l_{tç}$	0.5
15	Completely hole length [m]	L_b	11.17
16	Hole distance in row [m]	a	3
17	Hole row distance [m]	b	3.1
18	Length of bunk [m]	l_t	3.0
19	Charged length of hole [m]	L_{np}	8.17
20	Required explosives per 1m' [kg/m]	P_e	5.47
21	Required explosives quantity per hole [kg]	Q_{lpb}	44.69
22	Optimal retardment interval [ms]	T	16
23	Optimal retardment interval (appropriate) [ms]	T_p	20

3 SCHEMATIC PRESENTATIONS AND CALCULATION OF A BLAST

Table 6. The calculator of explosives for blast

$V = H_{sh} \times a \times b$						
$H_{sh} = L_{sh} \times \sin\alpha$						
$L_f = (L_{sh} + 3) \times N_b$						
$q_s = PI \times (d/2)^2 \times l$						
		Row 1	Row 2	Row 3		
Total length of drill	m	580.84	591.24	601.64	1773.72	m
Number of holes	$= N_b$	52	52	52	156	holes
Hole spacing	= a	3	3	3		
Row spacing	= b	3.1	3.1	3.1		
Diameter of hole	= Ø	89	89	89		
Angle of drill	= α	70	70	70		
Bung		3	3	3		
Over drilling according to orders		0.5	0.7	0.9		
Diameter of cartridge	= d	89.0	89.0	89.0		
Length of cartridge	l=cm	50.00	50.00	50.00		
Compression	%	5%	5%	5%		
Density of explosives	d	0.8	0.8	0.8		g/cm^3
Average hole length	$= L_{sh}$	11.17	11.37	11.57		m
Vol. of mass per hole	= V	92.97	92.85	92.74		m^3
Bench height	$= H_{sh}$	10.00	9.98	9.97		m
Weight of cartridge	$= q_s$	2.49	2.49	2.49		kg
No. of cartridge in hole		16.00	17.00	17.00		
Length of charge		8.00	8.50	8.50		m
Charge per meter of hole		4.98	4.98	4.98		kg/m
Charge per hole	$= q_{sh}$	39.82	42.30	42.30		kg
Specific charge of explos.	$= q_{sp}$	0.43	0.46	0.46	0.42	kg/m^3
Total charge of explosive.	$= Q_A$	2,070.40	2,199.79	2,199.79	6,469.98	kg
Vol. of blast quantity	$= V_A$	4,834.24	4,828.41	4,822.58	14,485.23	m^3
Detonating Cord		762.84	773.24	783.64	2,319.72	m

Table 7. The cost of blast with explosives: Ammonite 100%; ANFO 0%

		Ammonite 00%; ANFO 0%					
nr.	Type of explosive material and Services	Unit	Quantity	Price per unit €	Total €	Volume of blast m3	Price €/m3
1	Nonel detonator	pcs	0	5.5	0		
2	Nonel Connectors	pcs	2	4.25	8.50		
3	Nonel Dynoline	m	300	0.3	90		
4	Ammonite	kg	4700	1.35	6345		
5	ANFO	kg	0	1	0		
6	Detonating Cord	m	2500	0.5	1250		
7	Delays	pcs	4	4.25	17		
8	Drilling	m	1774	5	8870		
					16580.5	14485	1.15

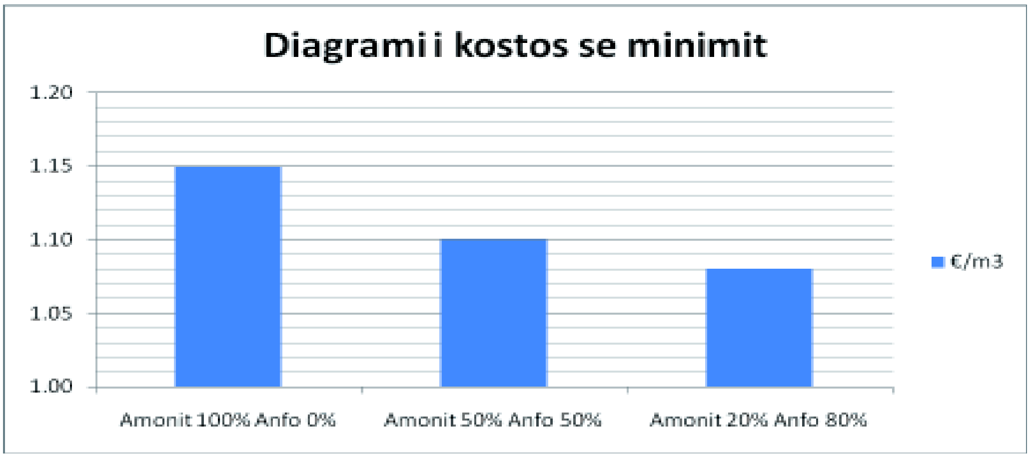
Table 8. The cost of blast with explosives: Ammonite 50%; ANFO 50%

		Ammonite 50%; ANFO 50%					
nr.	Type of explosive material and Services	Unit	Quantity	Price per unit €	Total €	Volume of blast m3	Price €/m3
1	Nonel detonator	pcs	0	5.5	0		
2	Nonel Connectors	pcs	2	4.25	8.5		
3	Nonel Dynoline	m	300	0.3	90		
4	Ammonite	kg	2350	1.35	3172.5		
5	ANFO	kg	2350	0.8	2600		
6	Detonating Cord	m	2500	0.5	1250		
7	Delays	pcs	4	4.25	17		
8	Drilling	m	1774	5	8870		
					16008	14485	1.10

Table 9. The cost of blast with explosives: Ammonite 20%; ANFO 80%

		Ammonite 20%; ANFO 80%					
nr.	Type of explosive material and Services	Unit	Quantity	Price per unit €	Total €	Volume of blast m3	Price €/m3
1	Nonel detonator	pcs	0	5.5	0		
2	Nonel Connectors	pcs	2	4.25	8.5		
3	Nonel Dynoline	m	300	0.3	90		
4	Ammonite	kg	940	1.35	1269		
5	ANFO	kg	5200	0.8	4160		
6	Detonating Cord	m	2500	0.5	1250		
7	Delays	pcs	4	4.25	17		
8	Drilling	m	1774	5	8870		
					15664.5	14485	1.08

Table 10. The Diagram of blast cost with combined explosives



4 CONCLUSION

In the paper "Application of combined CE Kosovo limestone – Gremnik area" previously treated as a feasibility type CE combined these limestone deposits, which are calculated all travel costs that enter into the process of preparation to final product.

Through this paper are obtained realistic results, especially with the use of combined (Mix) Ammonite and ANFO in three cases of the experiment with different percentages as follows:

1. Combined explosives: Ammonite 100% + Anfo 0%, acquired costs 1, 15 Euro/m^3
2. Combined explosives: Ammonite 50% + Anfo 50%, acquired costs 1, 10 Euro/m^3
3. Combined explosives: Ammonite 20% + Anfo 80%, acquired costs 1, 08 Euro/m^3.

As can be seen from the calculations obtained, the case 3 is the preferred option, because of the difference in price, which is 0.07 EUR/m^3, with which are obtained great savings during the destruction of limestone rock masses of Kosovo.

The combination of strong CE with weaknesses in massive blasting gives the following effects:

- Required fragmentation,
- Maximal energy optimalisation,
- Low costs of ANFO explosives as local product,
- Practical, fasted, and easy used of this explosive for blasting.

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Trio Chronos - A Portable, Modular Digital Chronometric Device for Mining and Explosives Industry

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ABSTRACT Trio Chronos is a high precision, portable digital chronometric device, used for determining the delay accuracy of electric and non–electric detonators, fuse–heads, impact fuses, detonating relays and trunk–line delay connectors. Trio Chronos also measures the detonation velocity of explosives, detonating cords and shock–tubes.

1 USAGE

The device is suitable for use in the production and testing of explosives and initiating means in the explosives and military industry and in mining. The electronic unit of Trio Chronos is placed in rugged *Peli*® case, thus making it suitable for both indoor and field use.

Picture 1. The electronic unit of Trio Chronos

2 FEATURES

Trio Chronos features an extremely high precision timer with microseconds accuracy and input channels that can be triggered by a piezoelectric sensor, photo–sensor, wire burst and optical fiber signals. All the necessary sensors with associated cable and transducers are included.

To initiate electric blasting caps (EBCs) and fuse–heads, the device features a built–in generator of a continuous current (firing impulse) that can be adjusted both in terms of duration and intensity.

To check the resistance of EBCs and fuse–heads, the device has a built–in ohmmeter that is capable of zeroing the resistance of connecting cables.

Although the device works as a standalone system, the output connectors that forward or amplify signals from the sensors and firing impulse generator, allow the system to be connected to an oscilloscope.

The device has four methods used for measurement, the option depending on the object of measurement and combination of sensors used.

The methods are: the Electric System Method – for measuring the time and amperage of the bridge wire burst, time of fuse–head initiation and overall delay accuracy of the electric detonators, the Non–electric System Method – for measuring of delay accuracy of non–electric detonators, the Impact Fuze Method – for measuring of delay accuracy of impact fuze igniter of hand grenades and projectiles and the Wire–burst and Optical Fiber Method – for measuring the detonation velocity of explosive and detonating cord, burning duration of safety fuses, shock–wave velocity of shock–tubes, detonating relay and trunk line delays accuracy and delay accuracy of initiating caps.

When measuring the characteristics of EBC, Chronos not only gives us a total period of delay, but also a current and the moment of burn–out of the fuse–head bridge, time of fuse–head initiation and a total delay time of EBC.

Picture 2. The EBC test results display

System setup and display of measurement results is provided over the LCD display.

In addition to this, after each measurement is carried out, the results are automatically sent to the built–in serial port. The simple format of the data allows logging to a computer for further statistical analysis and data storage.

Picture 3. The RS232 DB9 connector for PC communication and/or direct printing

3 SAFETY

Trio Chronos has numerous built–in safety features to prevent an accidental activation of the EBC.

To mention a few: The firing circuit is internally disconnected from the EBC and can get connected only if the fire button is held down for more than one second.

Even then, one has to hold the fire button one second further in order to fire. In the case of the method of measurement not requiring the firing ability, the firing function is fully disabled.

The ohmmeter circuitry is constructed in such a manner that even in a possible worst case scenario of a malfunction occurring, the maximum ohmmeter measuring current can not exceed 1,8 mA, thus making it absolutely safe for any type of blasting cap.

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