

**Chapter - G**

**UNDERGROUND MINING /  
YERALTI MADENCİLİĐİ**



# Bir ayaktaki Sıcaklık Değişimlerinin İncelenmesi

## *Investigation of Temperature Changings in Longwall*

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**ÖZET** Kömürler üretildikleri andan başlayarak, oksijenle reaksiyona girerler ve kendiliğinden yanmaya maruz kalırlar. Bu reaksiyon neticesi buldukları ortama ısı verirler. Bu ısı ortamdaki uzaklaştırılmaz ise, kendiliğinden yanma oluşarak, alevli yangına dönüşürler. Bu durum yeraltı kömür ocaklarında hem zehirleyici hem de yanıcı ve patlayıcı gazlar oluşturur. Gerekli ve güvenli şartlar sağlanamazsa, ocak içerisinde bu gazlar yayılarak zehirlenmelere, yangınlara ve patlamalara yol açmaktadır. Böyle durumlarda ısınmanın meydana geldiği bölge olan ayaklar barajlanarak, ayağın hava ile teması kesilir. Bu çalışmada, Ermenek bölgesinde bulunan bir yeraltı kömür ocağındaki bir ayakta, hava sıcaklık ölçümleri yapılmıştır. Ayakta zaman içerisinde meydana gelen kendiliğinden yanma nedeniyle, ortama yayılan sıcaklık değerleri izlenmiştir. Deney süresince her saat başı ayak içerisindeki sıcaklık değerleri ve ayrıca ocağı terk eden pisli havanın sıcaklık değeri de belirli aralıklarla ölçülmüştür. Ayak içinde sıcaklıklar tehlike oluşturacak seviyeye kadar yükselmiş ve ayak barajlanmıştır. Hava ile irtibatı kesilen ayakta, baraj gerisinden sıcaklık ölçümleri almaya devam edilmiştir. Elde edilen grafikler yardımıyla oksijensiz ortamda kalan ayağın, iç sıcaklıklarının zamanla nasıl düşüş gösterdiği tespit edilmiştir.

**Anahtar Kelimeler:** kendiliğinden yanma, kömür, ayak havası sıcaklıkları

**ABSTRACT** Coals react with oxygen from the moment they are produced and are subject to spontaneous combustion. This reaction is an exothermic reaction and coals give heat to the surrounding air. If the heat is not removed from the surrounding air, spontaneous combustion occurs and then flaming fire starts. In addition, in underground coal mine besides exothermic reactions, hazardous, explosive and flammable gases are released to the surrounding air. This situation causes interruptions in production, economical losses and environmental problems. In order to avoid these problems, production at the longwall should be closed and the contact of the coal with air should be prevented. In this study, the temperatures changes in the longwall in the spontaneous combustion condition were measured by temperature sensor placed inside the longwall. In addition, while the temperature recording was carried out for inner part of the longwall, the temperature of the exhaust air were measured. The temperatures of the longwall were increased in time and the longwall was closed. After closing the longwall, the temperature recording was continued. The temperate changings for the inner part of the longwall were observed continuously.

**Keywords:** spontaneous combustion, coal, longwall air temperatures

## 1 GİRİŞ

Kömür doğası gereği oksijenle reaksiyona girerek hem zehirleyici hem de yanıcı ve patlayıcı gazlar açığa çıkarırlar. Bu durum yeraltı kömür ocaklarında birçok problemler meydana getirmektedir. Kömürün havayla

meydana getirdiği bu ekzotermik reaksiyon sırasında ortama ısı verirler. Normal şartlar altında, kömürün oksidasyonu sonucunda açığa çıkan ısı, eğer ortamdaki uzaklaştırılmazsa ortam sıcaklığının, yaklaşık 70°C'den sonra CO ve CO<sub>2</sub>

gazlarının yayılımı artar ve 125°C civarında su buharı oluşur. Sıcaklık artışı devam ettiğinde ise, kömür tutuşma sıcaklığına ulaşır(Ökten, 1988). Bu durum kömürlerin üretildiği ayaklarda çoğu zaman karşılaşılan bir olaydır. Gerekli ve güvenli şartlar sağlanmazsa, ocak içerisinde zararlı ve tehlikeli gazlar yayılır (Şensöğüt, C., 1999). Kendiliğinden yanma olayı, yeraltı kömür ocaklarında karşılaşılan, hem emniyet hem de ekonomi açısından en önemli tehlikelerden birisi olarak kabul edilmektedir. Ne kadar küçük olursa olsun, her kendiliğinden kızışma olayına, zamanında müdahale edilmezse açık alevli yangına veya kömür tozu/gaz patlamasına sebebiyet verecektir. Sınırlı miktarda hava, bir yandan oksidasyon için gerekli oksijeni sağlarken, diğer yandan, oluşan ısıyı uzaklaştırmakta yetersiz kalarak ortamda sıcaklık artışının ve kendiliğinden yanmanın başlıca nedeni olmaktadır (Didari, V., 1986). Günümüzde kendiliğinden yanmada etkili olan kömür özellikleri ve parametreleri incelenmekte (Ren vd., 1999; Wang vd., 1999; Wang vd., 2003; Nugruho vd., 2000; Kadioğlu ve Varamaz, 2003; Kucuc vd., 2003) ve kendiliğinden yanma sonucu oluşan gaz ürünlerinin yardımıyla kendiliğinden yanmanın erken saptanması yönünde çalışmalar yapılmaktadır(Lu vd., 2004, Şahin ve Didari, 2002; Ören ve Şensöğüt, 2007; Saraç, 1993; Saraç ve Soytürk, 1992).

Ermenek kömürleri üzerinde laboratuvar ölçekli birçok çalışma yapılmış olmasına rağmen, doğrudan yeraltı işletmesinde kendiliğinden yanma ile ilgili çalışmalar çok azdır. Yapılan kendiliğinden ısınma ile ilgili laboratuvar ölçekli çalışmalar (Şensöğüt ve Çınar, 1998), gram mertebelerinde kömürler üzerinde gerçekleştirilmekte ve sınırlı sayıdaki parametrelerin olay üzerindeki etkisi, yalnızca deney şartlarında geçerli olmak üzere incelenebilmektedir. Bu nedenle laboratuvar ölçekli deneysel çalışmalar ile kömürlerin kendiliğinden yanma olayının bir bütün olarak incelenmesi güç olduğu gibi, elde edilen sonuçların pratiğe uyarlanması da oldukça zor olmaktadır(Akgün, 1994). Kendiliğinden yanma olayı üzerinde etkin olan parametrelerin büyük çoğunluğunu veya

tümünü kapsayan, özellikle kömürün kendiliğinden yanması esnasındaki genel davranışının belirlenmesi amacıyla yönelik endüstriyel bazı çalışmaların önemi büyüktür (Özdeniz, 2003; Özdeniz ve Şensöğüt, 2005; Fierro ve diğ., 1999 ve 2001). Ancak bu tür çalışmaların çok uzun süre gerektirmesi ve pahalı olması nedeniyle sayıları oldukça kısıtlıdır. Bu çalışmada, bir yeraltı kömür işletmesinde kömür üretim yeri olan ayaklarda kendiliğinden yanma nedeniyle meydana gelen sıcaklık artışları izlenmiştir. Yükselen sıcaklık değerlerinin ayağın kapatılması durumunda, hangi zaman aralıkları içerisinde düşüş eğilimine girdiği incelenmiştir.

## 2 DENEY SİSTEMİ VE DENEYİN YAPILIŞI

Ülkemizin Batı Toroslarda bulunan Ermenek linyit havzası yaklaşık 620 km<sup>2</sup> lik bir alanı kapsamaktadır. Ermenek havzasında farklı yörelerde, birbirinden bağımsız ve ekonomik olarak işletilen ocaklar dört bölgeye ayrılır (Şensöğüt ve Çınar 1998). Bu bölgeler, Ermenek havzasının orta kesiminde yer alan Asar dağı temel alınmak üzere, Asar dağının doğusundaki Çanakçı çukuru ve Keşirlik bölgesi ile Asar dağının kuzeybatısında yer alan Pamuklu-Tepebaşı yöresi ile Asar dağının batısındaki Boyalık mevkiidir (Arslansan, 1996).

Bölgede irili ufaklı birçok firma, yeraltı üretim yöntemiyle kömür üretimi yapmaktadır. Deneysel çalışmalar Tepebaşı yöresinde çalışmakta olan özel sektöre ait bir yeraltı ocağında yapılmıştır. Bu işletmede kısa ayak üretim yöntemi uygulanmaktadır. Damar derinliği 350-400 m olup, damar eğimleri 30-50° ve genellikle damar kalınlıkları 1,5-5 m arasında değişmektedir. Düzensiz damar kalınlığının yanında damarın çok ondülasyonlu olması, üretim sırasında kömür kayıplarının artmasına neden olmaktadır. Ayrıca üretim yapılan kömür panosunun tavanının göçmeye uygun özellikte olması nedeniyle, bol miktarda kırık ve çatlaklar içermesi kendiliğinden yanmaya çok uygun zemin hazırlamaktadır.

Kısa ayak yöntemiyle kömür üretimi yapılan ayağa, sıcaklık değerlerini hassas şekilde algılayabilecek 1 adet (Pt100) sıcaklık sensörü yerleştirilmiştir. Bu sıcaklık sensöründen saatte bir alınan, ortam sıcaklık değerleri, otomatik olarak bir datalogger da kayıt altına alınmıştır. Belirli zaman aralıklarında datalogger daki bilgiler bilgisayara aktarılmıştır. Ayrıca ölçüm alınan süre içerisinde, belirli aralıklarla ocak nefeslik çıkışındaki havanın sıcaklıkları ölçülmüştür. Ölçümler 18/03/2012 tarihi ile 10/07/2012 tarihleri arasında yapılmıştır. Çalışmada ayakların bulunduğu noktadaki kömürlerden örnek alınarak laboratuvar analizleri Çizelge 1’de verilmiştir.

Çizelge 1.Yığının ilk yapıldığı zamandaki kömür analiz sonuçları

Örnek adı	Alt Isıl Değer (kcal/kg)	Uçucu madde (%)	Kül (%)	Nem (%)	Kükürt (%)
A	4470	40,26	18,50	13,8	2,9

### 3 VERİLERİN DEĞERLENDİRİLMESİ

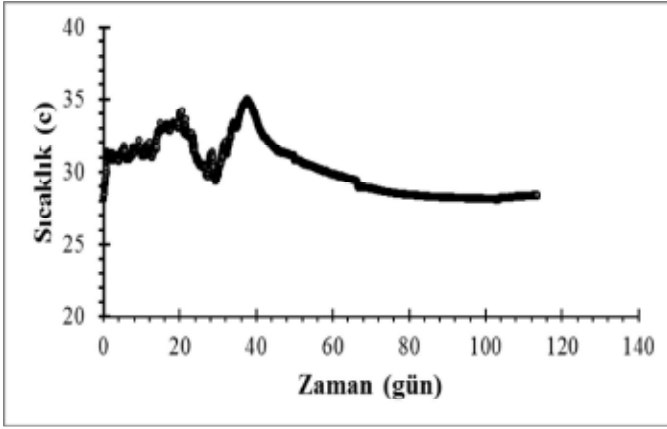
ErmeneK/Konya da bulunan bir yeraltı kömür işletmesinin kömür üretim panosundaki bir ayakta, kendiliğinden yanmanın tespitine yönelik olarak, ayağın uygun bir yerine ayak içi sıcaklıkları algılayacak olan bir sıcaklık sensörü yerleştirilerek, saatte bir sıcaklık değerleri ölçülmüştür. Veri alma işlemleri düzenli bir şekilde devam ederek, yaklaşık olarak 4 ay gibi bir süre almıştır. Bütün bu zaman içerisinde alınan sıcaklık değerlerini gösteren grafik Şekil 1’de verilmektedir. Verilerin alındığı ilk 12 saate ait değerler Çizelge 2’de verilmektedir. Ayağa ilk yerleştirildiği andan itibaren, ayaktaki havanın sıcaklıklarını algılayan T1 sıcaklık sensörü, 28°C’lerde değerler almaya başlamıştır. İlk günün sonunda 32°C’lere ulaşmıştır. 12. güne kadar

sıcaklıklar 30-32°C’lerde dalgalanmalar göstermiştir. 20. günde ise 35°C’lere ulaşmıştır. :alışmanın yapıldığı bu günde ocak içinde havanın hareket ettiği bazı yollarda durum değişiklikleri yapılarak, ayak içerisine başka noktalardan temiz hava gönderilmeye başlanmıştır.

Çizelge 2. Sıcaklık değerlerinin alındığı ilk 12 saate ait değerler

Tarih ve saat	Saat	T 1 (°C)
2012-03-18 12:19:05	1	28.12
2012-03-18 13:19:05	2	28.11
2012-03-18 14:19:05	3	28.26
2012-03-18 15:19:05	4	28.35
2012-03-18 16:19:05	5	28.46
2012-03-18 17:19:05	6	28.78
2012-03-18 18:19:05	7	28.81
2012-03-18 19:19:05	8	28.90
2012-03-18 20:19:05	9	29.00
2012-03-18 21:19:05	10	29.09
2012-03-18 22:19:05	11	29.16
2012-03-18 23:19:05	12	29.25

Bu takviye havanın etkisiyle 27. günde ayak içi sıcaklıklarda dalgalanmalar yaparak 29°C’lere düşmüştür. Fakat ilave havanın etkisiyle geçici sıcaklıklarda düşmeler olsa bile, 28. günden itibaren sıcaklıklar artmaya devam ederek, 37. günde 35°C’lere ulaşmıştır. Bu sıcaklıklarda ayaktaki ortam ikliminin özellikle sıcaklık ve nem yönünden oldukça kötüleşmesi neticesinde, 37. günde pano barajlanarak hava ile irtibatı kesilmiştir.



Şekil 1. Ayak içirisinde alınan sıcaklık ölçümleri

Ayağa temiz havanın kesilmesiyle, oksijeni azalan ayağın içerisindeki havanın sıcaklıkları, stabil bir şekilde düşmeye başlamıştır. Ayak havasında oksijenin zamanla azalmasıyla 45.günde sıcaklıklar 30°C'lere gelmiştir. 67.günde sıcaklık değeri 29°C'ler iken, ölçümleri sonlandırıldığı 113. güne kadar, geçen 46 gün içerisinde ayak sıcaklığı 1°C gibi değer kaybederek yatay durumda kalmıştır. Deneylerin sona erdirildiği gün olan 113.günde baraj gerisi sıcaklıklar 28°C'ye düşmüştür. Barajın yapıldığı 37.günde sıcaklık değeri 35°C'den, deneyin sona erdirildiği günde 28°C'ye düşmüştür. Yaklaşık ölçüm süresinin 2/3 de ayak sıcaklığı 7°C kadar azalmıştır.

Ayakta ölçümler alınırken belirli zaman aralıklarında ise ocaktan çıkan kirliliğin sıcaklık değerleri de ölçülmüştür. Bu sıcaklık değerlerini gösteren veriler Çizelge 3'de verilmektedir. Şekil 1 ve Çizelge 3 incelendiğinde, mart aylarında Ermenek bölgesinde havaların soğuk olduğu zamanlarda, ocak çıkışındaki havanın sıcaklık değeri 19,5°C'den, 22,5°C'ye yükseldiği görülmektedir. 37.günde barajlanan ayaktan ocak içerisine sıcaklık yayılımının durması nedeniyle, sıcaklıkların düştüğü açıkça görülmektedir.

Çizelge 3. Ocak dışına çıkan pis havanın sıcaklık değerleri

TARİH	Zaman (gün)	SICAKLIK (°C)
02.02.2012	-	19.5
18.03.2012	1	22.5
08.04.2012	21	23.7
01.05.2012	48	20.1
31.05.2012	78	20.2
05.07.2012	113	23.4

#### 4 SONUÇLAR

Ermenek/Konya'da bulunan bir yeraltı kömür işletmesinin kömür üretim panosundaki bir ayakta kendiliğinden yanmanın tespitine yönelik olarak çalışmalar yapılmıştır. Üretim yapılan bir ayağa, atmosfer sıcaklıklarını saatte bir algılayacak olan bir sıcaklık sensörü ve verilerin depolandığı bir datalogger sistemi yerleştirilmiştir. Datalogger'dan belirli zaman aralıklarında alınan veriler, bilgisayarda kayıt altına alınmıştır.

İlk gündeki sıcaklık değerleri 28°C lere olmuştur. Devamlı bir şekilde sıcaklık değerleri yükselerek 28°C'den 35°C'ye çıkmıştır. Ayak içi sıcaklıkların maksimum olduğu gün olan 37.günde kömür panosu barajlanmıştır. Geçen bu zaman içerisinde ayak içi sıcaklıklar 28°C'lerden yaklaşık 36°C'ye yükselmiştir. Ayak içerisine temiz hava girişinin kesildiği günden itibaren, ayak içi sıcaklıklar yavaş bir şekilde düşmeye başlamıştır. Ölçümlerin sonlandırıldığı gün olan 113.günden baraj gerisi sıcaklıklar 28°C lere düşmüştür.

Ayrıca yeraltı kömür işletmesinin ocak içerisinde kirlenerek ocağı terk eden havanın

sıcaklık değerleri alınmıştır. Bu değerler kış aylarında ocağa beslenen soğuk havaya rağmen yeraltı kömür ocak içerisinde meydana gelen kendiliğinden yanma nedeniyle ortama yayılan sıcaklık artışları, ocağı terk eden hava sıcaklık değerinde belirli bir miktar artışların olduğu görülmektedir. Yaklaşık 20°C'lerden 24°C'lere ulaşmaktadır. Barajın yapılmasıyla ocak içerisine kendiliğinden yanma nedeniyle ortama yayılan ısıda azalmalar meydana gelerek, kirli hava sıcaklığının 20°C'lere düştüğü gözlenmiştir. Yaz aylarında ocağa giren havanın belirgin artışı nedeniyle de sıcaklıklar tekrar 24°C'lere ulaşmıştır.

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# Ömerler Mekanize Yeraltı Kömür Ocağında Tahkimat Tasarımına Yönelik Kaya Kütle Sınıflama Sistemleri Uygulaması

## *Application of Rock Mass Classification Systems at Omerler Mechanized Underground Coal Mine for Support Design*

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**ÖZET** Yeraltı boşlukları açılırken yapılacak işlemlerin başında bu boşlukların duraylılığının sağlanması gerekmektedir. Duraylılığın sağlanması için gereken destek tipinin seçiminde gözlem ve tecrübeye dayalı olarak geliştirilen kaya kütle sınıflama sistemlerinden faydalanılmaktadır. T.K.İ G.L.İ. Ömerler Mekanize Yeraltı Kömür Ocağında kömür oluşumuna bağlı olarak üretim ve hazırlık çalışmaları daha derin seviyelerde yapılmaya başlanmıştır. Derin kotlarda yapılacak gerek hazırlık gerekse üretim çalışmalarında, kömür ve yan kayaç özelliklerinin bilinmesi, bu malzemeler içerisinde dizayn edilecek mühendislik yapılarının projelendirilmesi bakımından önemlidir. Bu çalışmada Ömerler Mekanize Yeraltı Kömür Ocağında derin sahalarda kullanılacak tahkimat tasarımının belirlenmesi amacıyla kaya kütle sınıflama sistemleri yardımıyla açıklıkların tahkimatsız kalma süreleri belirlenmiştir.

**ABSTRACT** The first step of design of underground excavations is to provide the stability of underground openings. Rock mass classification systems are one of the design method for providing the stability of underground openings. Rock mass classification systems are practically helpful for determining the support systems. The preparation and the production level at T.K.İ G.L.İ. Omerler Mechanized Underground Coal Mine have been going deeper thus the studies of mechanical, physical and rock mass properties of rock around the coal seam is more important for design of these kind of engineering structures. In this study, rock mass classification system was used for determining the standup time of underground openings at deeper sections of Omerler Mechanized Underground Coal Mine to help support design.

## 1 GİRİŞ

Yeraltında açılacak boşlukların kesitlerinin belirlenmesi ve tahkimatının tasarımında, kayacın fiziksel ve mekanik özelliklerinin yanında kaya kütlelerinin yapısal özelliklerinin de çok ayrıntılı bir şekilde tespit edilmesi gerekmektedir. Kaya kütle sınıflama sistemleri tünel, galeri vb. yeraltı açıklıklarının destek türlerinin seçimine

yönelik olarak gözlem ve tecrübeye dayalı olarak geliştirilmiş sistemlerdir. Kaya kütlelerinin sınıflandırılma amaçları; kaya kütleleri özelliklerini karakterize edecek başlıca parametreleri tanımlamak, kaya kütlelerini kaya birimlerine bölmek, her bir kaya sınıfının karakteristiğinin anlaşılması için bir temel hazırlamak, mühendislik tasarımları için sayısal veriler sağlamak,

maden galerileri ve tünellere ait tahkimat gereksinimlerinin belirlenmesinde kılavuzluk yapmak, bir bölgedeki kaya kütlelerini karakterize edecek parametrelerin belirlenmesi ile elde edilen tecrübelerin diğer kaya birimlerinde de uygulanabilmesidir (Bieniawski, 1993). Kaya kütle sınıflama sistemleri sayesinde jeolojik ve jeoteknik koşulların etkisi altındaki kaya kütlelerinin dayanımı hakkında fikir verecek olan Kaya Kütle Kalitesi belirlenmektedir. Kaya kütle sınıflama sistemleri ile kaya kütleleri çok zayıf kayadan çok iyi kayaya kadar sınıflandırılmakta ve her kaya sınıfı için desteksiz durma süreleri ve buna bağlı olarak gerekli destek türü önerilmektedir.

Kaya kütlelerinin davranışlarının kestirilmesinde kullanılan birçok kaya kütle sınıflama sistemi geliştirilmiştir. Günümüzde yer kazılarının tasarımında yaygın olarak kullanılan sınıflama sistemleri; Kaya Kütle Oranı (RMR) (Jeomekanik sınıflama), Kaya Kalite İndeksi (Q) ve Jeolojik Dayanım İndeksi (GSI) olarak sıralanabilir. Genel olarak tünel kazılarının tasarımı için geliştirilmiş olan bu sistemler daha sonra yeraltı maden kazılarında kullanılacak şekilde modifiye edilerek MRMR, Q' ve RMR' sınıflama sistemleri geliştirilmiştir.

Kaya kütlelerinin sınıflandırılmasında genel olarak, kayacın tek eksenli basınç dayanımı, süreksizlik özellikleri; süreksizlik aralıkları, süreksizlik yüzeylerinin durumu, konumu (eğimi, eğim yönü), yeraltı su durumu, yerinde gerilme durumu vs. gibi birçok parametre kullanılmaktadır. Bu çalışma kapsamında T.K.İ G.L.İ. Ömerler Mekanize Yeraltı Kömür Ocağı derin sahaları için kaya kütle sınıflama sistemleri kullanılarak tahkimat tasarımı yapılmıştır.

### 1.1 Jeomekanik Kaya Kütle Sınıflama Sistemi (RMR)

Bieniawski (1973) tarafından önerilen jeomekanik kaya kütle sınıflama sistemi

(RMR) daha sonra modifiye edilerek bugünkü halini almıştır. RMR kaya kütle sınıflama sisteminde aşağıda verilen altı farklı değişken kullanılmaktadır.

1. Kaya numunesinin tek eksenli basma dayanımı ( $\sigma_c$ )
2. Kaya kalite göstergesi (RQD)
3. Süreksizlik aralığı
4. Süreksizliklerin durumu
5. Yeraltı su durumu
6. Süreksizliklerin konumu (Eğim ve Doğrultuları)

RMR sınıflandırma sisteminde kullanılan tanımlamalar ve değerlendirmeler Çizelge 1'de verilmiştir. Şekil 1'de tünel ve galeriler için Jeomekanik Sınıflama Sisteminde farklı kaya kütleleri değerleri için "Kazı açıklığı – tahkimatsız durma zamanı" ilişkisi verilmektedir. Şekil 1' de verilen bu abak kullanılarak her kaya sınıfının kazıdan sonraki davranışı hakkında bilgi edinilerek kazı boşluğunun nasıl destekleneceğine karar verilebilmektedir. Çizelge 2'de ise RMR'ye göre her kaya sınıfı için kazı şekli ve tahkimat sistemleri önerilmektedir.

## Çizelge 1 RMR Jeomekanik Sınıflama Sistemi (Bieniawski,1989)

## Sınıflama Parametreleri Dereceleri

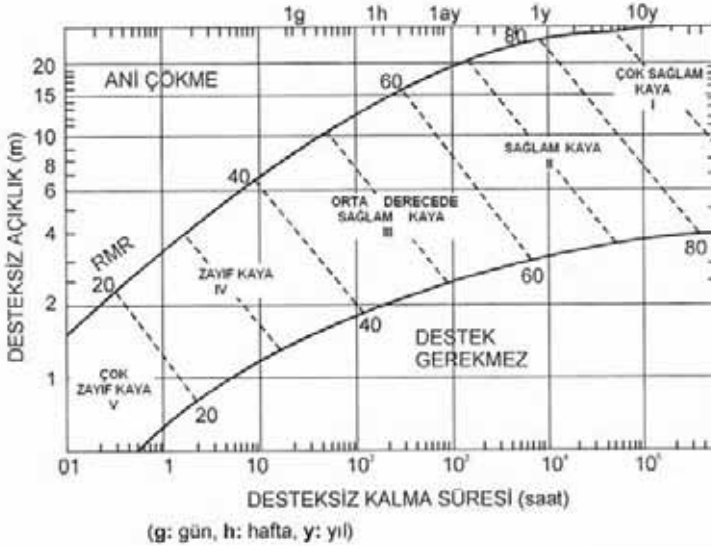
1	Sağlam Kayacın Dayanımı	Nokta Yüklü Dayanımı İndeksi (MPa)	>10	4 - 10	2 - 4	1-2	Düşük Aralıklar İçin Tek Eksenli Test	
		Tek Eksenli Basınç Dayanımı (MPa)	>250	100-250	50 - 100	25 - 50	5 - 25	1- 5
Derecelendirme			15	12	7	4	2	1 0
2	RQD (%)		90 - 100	75- 90	50 - 75	25 - 50	<25	
	Derecelendirme		20	17	13	8	3	
3	Süreksizlik Aralığı (mm)		>2000	600-2000	200-600	60 - 200	<60	
	Derecelendirme		20	15	10	8	5	
4	Süreksizlik Durumu		Çok kaba yüzeyler, süreklı değil, ayrılma yok. Sert eklem yüzeyleri	Az kaba yüzeyler, ayrılma <1 mm. Sert eklem yüzeyleri	Az kaba yüzeyler, ayrılma <1 mm. Yumuşak eklem yüzeyleri	Sürtünme izli veya fay dolgusu < 5 mm veya 1-5 mm.	Yumuşak fay dolgusu >5mm kalınlıkta veya açık eklem > 5 mm devamlı süreksizlikler	
		Derecelendirme		30	25	20	10	0
5	Yeraltı Suyu	Tünelin 10 m'lik kısmında gelen su	Yok	10 lt/dk	<25 lt/dk	25 - 125 lt/dk.	>125 lt/dk	
		Oran (Eklem su basıncının ana asal gerilmeye oranı)	0,0	0,0 - 0,1	0,1 - 0,2	0,2 - 0,5	>0,5	
		Genel Koşullar	Tamamen Kuru	Nemli	Islak	Damlama	Su akışı	
Derecelendirme			15	10	7	4	0	

## Eklem Yönelimine Göre Düzeltmeler

Eklemelerin doğrultu ve eğim yönelimi		Çok uygun	Uygun	Orta	Uygun değil	Hiç uygun değil
Derecelendirme	Tüneller	0	-2	-5	-10	-12
	Temeller	0	-2	-7	-15	-25
	Şevler	0	-5	-25	-50	-60

## Toplam sayısal değerlendirmeye göre (puanlama) kaya kütle sınıfları

Derecelendirme	100 - 81	80 - 61	60 - 41	40 - 21	<20
Sınıflama No.	I	II	III	IV	V
Tanımlama	Çok iyi kaya	İyi kaya	Orta kaya	Zayıf kaya	Çok zayıf kaya



Şekil 1. RMR kaya kütle sınıflarına göre desteksiz durma süresi ve desteksiz tavan açıklığı arasındaki İlişki (Bieniawski,1989)

Çizelge 2. RMR Sistemine Göre Destek Sistemleri Seçim Kılavuzu (Ulusay, 2001)

Birincil Tahkimat				
Kaya Kütle Sınıfı	Kazı şekli	Kaya Bulonları (Tünel Genişliği 10 m)	Püskürtme Beton	Çelik Takımlar
I	Tam kesit 3 m ilerleme	Gerekli belirli noktalar haricinde genellikle destek gerekmez		
II	Tam kesit 1-1,5 m ilerleme. Komple destek. Aynaya 20 m mesafe	Tavanda bölgesel kaya civatası, 3 m uzunlukta, 2,5 m aralıklı,gerektiğinde çelik hasırla birlikte	Gerektiğinde tavan kemerinde 50 mm	Gerekmez
III	Tavan kemeri ve tabandan ilerleme. Tavanda 1,5-3 m ilerleme, Komple destek. Aynaya 10 m kadar gerekli	3-4 m uzunlukta sistematik bulonlar, kemerde tel kafesli duvarlar ve kemerde 1,5-2 m aralıklı	Tavan kemerinde 50-100mm, yan duvarda 30 mm	Gerekmez
IV	Tavan kemeri ve tabandan ilerleme. Tavandan 1-1,5 m ilerleme, kazıyla uygun şekilde aynaya 10 m'ye kadar gerekli tahkimat tamamlama	Tel kafesli duvarlarda ve kemerde 1-1,5 m aralıklı, 4-5 m uzunluklu sistematik bulonlar	Tavan kemerinde 100-150 mm ve yan duvarlarda 100 mm	Gereken yerde 1,5 m aralıklı yer yer hafif bağlar
V	Tavan ve tabanda müşterek ilerleme. Tavandan 0,5-1 m ilerleme, kazıyla birlikte destek yerleştirilmeli.	Tel kafesli duvarlarda ve kemerde 1-1,5 m aralıklı, 5 m uzunlukta sistematik bulonlar	Tavan kemerinde 150-200 mm, yan duvarlarda 150 mm aynada 50 mm	Çelik iksal 0,75 m aralıklı orta ağır bağlar

## 1.2 Q Kaya Sınıflandırma Sistemi

Q veya NGI (Norwegian Geotechnical Institute) sistemi olarak bilinen bu sistem,

Barton ve ark. (1974) tarafından geliştirilmiştir. Sistem uzun yıllar kullanıldıktan sonra, sistemin destek seçimine yönelik bölümü Grimstad ve

Barton (1993) tarafından revize edilmiştir. Bu sınıflama sisteminde kullanılan veriler;

- RQD,
- Eklem takım sayısı,
- En kötü süreksizlik veya çatlakın pürüzlülüğü,
- Ayırışma derecesi veya en zayıf çatlaktaki dolgu,
- Tünel içine su akışı,
- Tünel derinliği

Kaya kütlelerinin değerini belirten Q sayısı (tünelticilik niteliği belirteci), aşağıda verilen eşitlik yardımıyla hesaplanır.

$$Q = \left( \frac{RQD}{J_n} \right) \times \left( \frac{J_r}{J_a} \right) \times \left( \frac{J_w}{SRF} \right)$$

Burada;

RQD : Kaya kalite tanımı

J<sub>n</sub>: Eklem takım sayısı

J<sub>r</sub>: Eklem pürüzlülüğü değeri

J<sub>a</sub>: Eklem alterasyon değeri

J<sub>w</sub>: Eklem suyu azaltma faktörü

SRF : Gerilim azaltma faktörüdür.

Q sınıflama sistemi çoğunlukla tünelticilik çalışmalarında kullanılmaktadır. Bu yüzden bu çalışma kapsamında Q sistemi kullanılmamış ve daha detaylı bilgi verilmemiştir.

### 1.3 Jeolojik Dayanım İndeksi (GSI)

Özgün Hoek-Brown yenilme ölçütü sert kaya kütlelerinin dayanımını belirlemek için geliştirilmiştir. Kaya kütleleri sabitlerinin Bieniawski'nin RMR (Kaya Kütle Oranı) kayaç + sınıflandırmasından yararlanılarak belirlenebileceği bir yöntem önerilmiştir. Bu yöntem, RMR değerleri 25'den büyük olan kaya kütleleri için geçerli olup, çok zayıf kaya kütleleri için kullanılamamaktadır. Bu sınırlamayı ortadan kaldırmak için Jeolojik Dayanım İndeksi (GSI) olarak adlandırılan yeni bir indeks tanımlanmıştır. GSI, çok zayıf kaya kütleleri için 10 değerini alırken, çok sağlam kayaç için 100 değerini almaktadır (Hoek ve ark., 1995). GSI, daha sonraki yıllarda yapılan değişikliklerle tek başına bir sınıflama sistemi olarak ölçütte yer

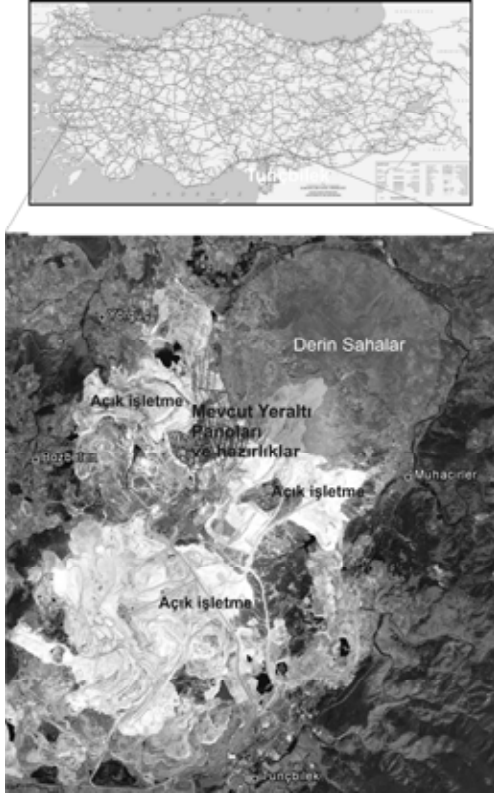
almıştır (Hoek ve Brown, 1997; Hoek ve ark., 1998).

GSI ölçütü üzerinde 2002 yılında Sönmez ve Ulusay tarafından sağlam kayalar da içine alacak şekilde değişiklikler önerilmiş ve günümüzde halen kullanılan son şekli verilmiştir (Ulusay ve Sönmez, 2007). Ölçütün Ulusay ve Sönmez (2002) tarafından önerilen son halinde, GSI değeri bulunabilmesi için kaya kütleleri içindeki süreksizliklerin niceliksel olarak değerlendirilmesini sağlayan Yapısal Özellik Puanı (SR) ve Süreksizlik Yüzey Koşulu Puanı (SCR) parametrelerinin kullanılması önerilmiştir.

## 2 FİZİKO-MEKANİK ÖZELLİKLER

GLİ Ömerler Mekanize Yeraltı Kömür Ocağı Kütahya İli Tavşanlı ilçesi sınırları içinde Tunçbilek kömür havzasında yer almaktadır. Havzada ana linyit damarı 4-12 m kalınlıkta ortalama 10° kuzeybatıya doğru dalımlı olarak uzanmaktadır. Bu nedenle açık işletme yöntemi ile üretim, örtü kazı oranına göre belirli bir derinliğe kadar gerçekleştirilmekte, belirli bir derinlikten sonra ise yeraltı üretim yöntemi ile kömür üretimi gerçekleştirilmektedir. Ayrıca havzadaki kömür damarının tektonizma sonucu büyük kütleli bloklar şeklinde bulunması üretim yönteminin belirlenmesinde etken olmaktadır. Mevcut duruma göre havzanın batı, güney, güney batı kısımlarında açık işletme yöntemi ile üretim çalışmaları belirli bir aşamaya gelmiştir. Geçmiş yıllarda yeraltı işletmesi olarak yaklaşık 150-250 m derinliklerde üretim çalışmaları gerçekleştirilmiştir. Devam eden süreçte sahanın kuzey batısında ortalama 350-500 m derinliklerde derin sahalar olarak tanımlanan bölgede yeraltı üretim yöntemi ile üretim yapılacaktır. Üretim yöntemi olarak üretim kapasitesi ve kömürün yapısı göz önünde bulundurularak tam mekanize sisteme karar verilmiştir. Hazırlık galerilerinin bu üretim sistemine izin verecek kesitlerde açılması gereklidir. Bu teknik şartlar altında artan derinlik ve açılan boşluğun kesit alanına göre tahkimat tasarımının gözden geçirilmesi gündeme

gelmiştir. Sahanın konumu Şekil 2'de verilmektedir.



Şekil 2. Ömerler Mekanize Yeraltı Kömür Ocağı Konumu

Açılan galerilerin tahkimat tasarımı için yararlanılan ve yukarıda tanıtılan kaya kütle sınıflama sistemlerinde kullanılan en önemli verilerin başında kayaçların fiziko-mekanik özellikleri gelmektedir. Bu nedenle çalışma kapsamında kömür ve çevre kayaçların fiziko-mekanik özelliklerinin belirlenmesi için yeraltı ocağından numuneler alınmış ve bunların üzerinde kaya mekaniği deneyleri uygulanmıştır.

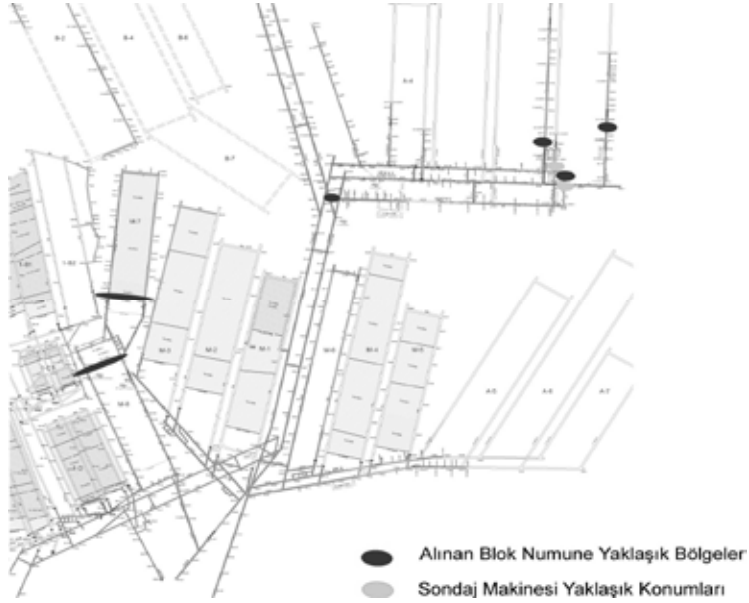
Numuneler ocakta yapılan gözlem ve inceleme çalışmaları süresince farklı

zamanlarda iki farklı yöntemle alınmıştır. Bunlardan birincisi üretim ayağı ve galeri açma işleri sırasında kazı aynalarından blok numune alınmasıdır. Bu yöntemde kazı aynalarından alınan taze bloklar ocak dışına çıkarılarak Ömerler Mekanize Yeraltı Ocağı karo sahasına kurulan laboratuvar tipi karot alma makinesi ile 54.4 mm çapında karot alınması şeklinde yapılmıştır (Şekil 3).



Şekil 3. Karotiyer ile Karot alınması

Numune alma çalışmalarında kullanılan ikinci yöntem ise, GLİ'ne ait morsetli delik delme makinesinin modifiye edilerek yeraltından karot numunesi alınmasıdır. Bu yöntemde yeraltı şartlarına göre imal edilmiş delik delme makinesine tüplü karotiyer modifiyesi yapılmış ayrıca karot alınmasını sağlayan özel elmas matkap temin edilmiştir. Derin sahalarda açılan hazırlık panolarından olan A1 ve A2 panolarının başlangıç kısmında konumlandırılan sondaj makinesi ile farklı yönlerde delik delinerek karotlar alınmıştır. Her iki yöntemle alınan karotlar ince naylon film ile kaplanarak laboratuvara ulaştırılmıştır. Alınan blok numunelerin ve sondaj makinesinin konumları Şekil 4'te verilmiştir (Yenice ve ark., 2012)



Şekil 4. Numune alınan bölgelerin yaklaşık konumları (Yenice ve ark., 2012)

## 2.1 Laboratuvar Deneyleri

Kömür ve çevre kayaçların kütle özelliklerinin belirlenebilmesi için yapılan testlerde numuneler belli bir sistematikte sınıflandırılmıştır. Yapılan sınıflandırmada kömürlü seviye üstü kayaç numuneleri **Tavan Taşı**, kömürlü seviye altı kayaç numuneleri **Taban Taşı** olarak sınıflandırılmış, deneyler bu sistematik ile gerçekleştirilmiştir.

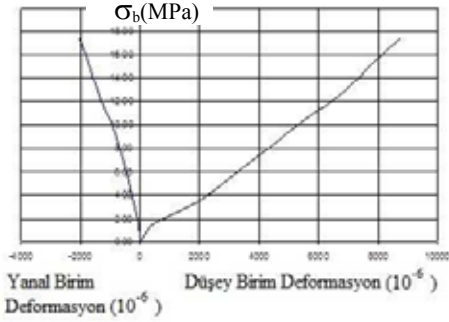
Yapılan çalışma kapsamında bölgeden getirilen numunelere birim hacim ağırlık, yoğunluk, tek eksenli basınç dayanımı deneyi, dolaylı çekme dayanımı deneyi, elastisite modülü ve poisson oranının tayini deneyi, üç eksenli basınç dayanımı ve içsel parametrelerin tayini deneyleri TS (1900-1, 1900-2), ASTM (1994) ve ISRM (1981) tanımlanan esaslar göz önünde bulundurularak gerçekleştirilmiştir. Çalışma alanından alınan numuneler üzerinde yapılan deneyler ve elde edilen sonuçlar Çizelge 3'te verilmiştir. Kil içerikli kayaçlarda bir diğer önemli husus da killerin şişme özellikleridir. Bu kapsamda kömür tavan kayaçlarından alınan numuneler üzerinde yapılan XRD çekimlerine bakıldığında, %38-43 aralığında

illit ve %59-64 aralığında da kaolinit kil mineralleri görülmektedir. Kömür üzeri örneklerde şişme özelliği yüksek olan simektit kil minerali bulunmadığı tespit edilmiştir.

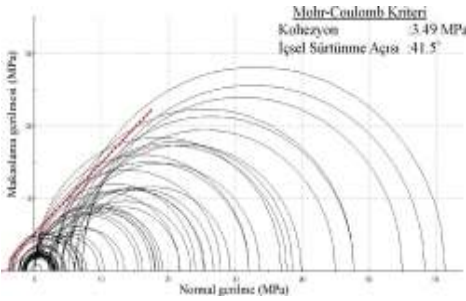
Çizelge 3. Tabantaşı, Tavantaşı ve Kömür Numunelerine Ait Toplu Deney Sonuçları

	Tabantaşı	Tavantaşı	Kömür
<b>Doğal Birim Hacim Ağırlık (gr/cm<sup>3</sup>)</b>	2,25±0,1	2,19±0,14	1,37 ± 0,19
<b>Tek eksenli basınç dayanımı (MPa)</b>	23,95±6,57	19,14±8,58	16,72 ± 2,71
<b>Dolaylı çekme dayanımı (MPa)</b>	2,44±0,68	2,02±0,79	1,64 ± 0,09
<b>Ortalama Elastisite Modülü (MPa)</b>	5878,75	4447,33	1870,00
<b>Poisson Oranı</b>	0,31	0,30	0,36
<b>Kohezyon (MPa)</b>	4,15	3,49	3,39
<b>İçsel Sürtünme Açısı (°)</b>	42,90	41,5	32,9

Yapılan deformasyon ve üç eksenli sıkışma deneyleri sonucunda çizilen gerilme-deformasyon eğrileri ve Mohr dairelerine ait grafikler Şekil 5a ve 5b'de verilmiştir.



Şekil 5 a) Tavan taşı blok numuneleri gerilme-deformasyon ilişkisi (Yenice ve ark., 2012)



Şekil 5 b) Tavantaşı numuneleri üç eksenli sıkışma Mohr daireleri (Yenice ve ark., 2012)

## 2.2 Jeoteknik Parametrelerin Belirlenmesine Yönelik Saha Ölçümleri

G.L.İ. linyit havzası açık ocaklarında ve Ömerler yeraltı ocağında yapılan gözlemler ve çalışma sahası ile ilgili eski çalışmalar incelenerek, öncelikle Ömerler Mekanize Yeraltı Ocağı A panoları olmak üzere kömür damarı yakın çevre kayaç birimleri ve jeoteknik özellikleri tespit edilmiştir. Kaya kütle sınıflamalarında kullanılan kayaç malzemesi özellikleri için yukarıda deney sonuçları baz alınmış, sınıflamalar tavan taşı, taban taşı ve kömür özelinde detaylandırılmıştır.

### 2.2.1 Süreksizlik Özelliklerinin Belirlenmesi

Süreksizlik ölçüm ve gözlemleri, ISRM (Uluslararası Kaya Mekaniği Derneği) (ISRM, 1978b) tarafından önerilen tanım ve yöntemlere uygun şekilde yürütülmüştür.

Arazi gözlemleri sonucu tavan taşı kütleleri “çok bloklu” olarak tanımlanmış ve aşağıda sıralanan süreksizlik özellikleri Şekil 4’de verilen Ömerler Mekanize Yeraltı Ocağı derin sahalarda panolarından A1 Panosu hazırlık çalışmalarının yeni A1 Kömür galerisi kısmının faylı bölgesinde gerçekleştirilmiştir.

### Görünür ortalama süreksizlik aralığı:

Yeraltı ocağı A panolarında tavan tabakasının gözlenebildiği kısıtlı yerlerden Yeni A1 Kömür galerisinde yaklaşık 10 metrelik faylı kısmında galeri yan duvarında yapılan gözlem ve ölçümlere göre, süreksizlik konumlarının yaklaşık aynı olup (K-B doğrultulu ve yaklaşık 80° ve üzeri eğimli) eklem takımı aralıklarının 0.03-0.43 m arasında değiştiği ve ortalama 0.12 m olduğu belirlenmiştir. Yeraltı koşullarında galeri yan duvarında süreksizliklerin veya tabakalanma düzlemlerinin düşey yöndeki değişimleri belirlenememiştir. Süreksizlik ölçümlerinin yapıldığı lokasyonda faylanma sonucu tavantaşı örselenmiş olduğundan burada yapılan ölçümlerin tüm sahayı temsil etmeyeceği düşünülmektedir.

Ömerler Yeraltı Ocağı derin sahalarda Beyhan (2008) tarafından yapılan çalışmada tabakalanma düzlemlerinin 0.20-1.20 m arasında değişmekle beraber, ortalama 0.45 m ve eklem takımı aralıklarının ise 0.35-1.10 m aralığında olup ortalama 0.51 m olduğu belirtilmiştir. Süreksizlik aralığı, tabakalanma düzlemleri ile eklem takımı aralıklarının ortalaması 0.48 m olarak saptanmıştır.

### Süreksizlik yüzeylerinin pürüzlülüğü:

Yeraltı ve açık ocaklarda süreksizlik yüzeyleri üzerinde yapılan gözlemler ve ölçümler sonucu, süreksizlik yüzeylerinin pürüzlülük durumu “dalgalı-düz” olarak tanımlanmıştır.

### Süreksizlik yüzeylerinin açıklığı ve dolgusu:

Yeraltı ocağında tavan taşında süreksizlik “yüzeylerinin açıklığının <1 mm” ve “dolgusuz” olduğu tespit edilmiştir.

### Süreksizlik yüzeylerinin bozunma derecesi:

Yeraltı ocağında tavan taşında yapılan gözlemlerde süreksizlik yüzeylerinde çok az yerde bozunmaya uğramış yüzey



tespit edilebilmiştir. Bundan dolayı, tavan taşı “az bozunmuş” olarak tanımlanmıştır.

**Süreksizlik yönelimleri:** Yeraltı ocaklarında üretim planlarına bağlı olarak süreksizliklerin yönelimleri galeri doğrultularına göre devamlı farklılık göstermektedir.

**Su Durumu:** GLİ Ömerler Yeraltı Ocağında yapılan gözlemler sonucunda yeraltı su durumu “nemli” olarak değerlendirilmiştir.

**RQD değeri:** Çalışma kapsamında RQD değerinin belirlenmesine yönelik herhangi bir sondaj çalışması gerçekleştirme imkânı olmamıştır. RQD değerinin belirlenmesi için Ömerler Ocağı derin sahalarında yapılmış önceki çalışmalar değerlendirilmiştir. Derin sahalar ve çalışma sahası içinde kalan ve MTA tarafından yapılan JT4 jeoteknik sondaj verileri çeşitli araştırmacılar tarafından çalışılmış ve sahaya ait ortalama RQD değeri % 42.2 olarak belirlenmiştir (Emir, 2000, Beyhan, 2008). Ancak yeraltı açıklıklarının tasarımı açısından açıklığın yakın çevre kayaların özelliklerinin bilinmesi daha anlamlı olduğundan tüm sondaja ait ortalama RQD değeri yerine yakın çevre kayalarına ait değerlerin kullanılması daha doğru bir seçim olacaktır.

JT4 sondajı jeoteknik etüdünde derinliğe bağlı RQD değerleri incelendiğinde proje konusu pano derinliklerine yakın ve kömür damarı üstünde kalan formasyonlara ait RQD değerleri Şekil 6’da verilmiştir. Buna göre kömür üstü 18.2 m yüksekliğin ağırlıklı RQD değeri % 47.4, kömür altı 13.7 m derinliğe kadar ağırlıklı RQD değeri % 63.8 olarak belirlenmiştir.

Ayrıca bölgeden gönderilen ve A Panolarının en kuzey kısmından itibaren yaklaşık 500 m kuzeyde bulunan 4291 nolu sondajın kömür üstü 11.4 m’lik kısmında RQD değeri % 41.6 olarak hesaplanmıştır (Şekil 7). Bu verilere göre RQD değerleri tavan taşı için % 47.4, taban taşı için % 63.8 olarak belirlenmiştir. Kömür için RQD değeri JT4 sondajında % 100 olarak verilmiş olsa da sahada yapılan gözlemlere göre en iyi şartlarda tavan taşının RQD değerine eşit olacağı, kömür damarı içerisindeki killi ara kesmeler nedeniyle bu değerden yüksek olmayacağı yorumu

yapılmış ve kömürün RQD değerinin % 45 alınmasının uygun olacağı sonucuna varılmıştır.

Diğer taraftan kömür içerisinde yapılan kazılarda kömürün hava ile teması sonrasında ayrışmalar sonucu dökülmeler meydana gelmesi nedeniyle, emniyetli olması açısından tahkimat tasarımında kömürün RQD değeri, JT4 sondajında tespit edilen RQD değerinden düşük alınmıştır. (Yenice ve ark., 2012)

Giriş-Çıkış (m)	Kalınlık	RQD
345.05-348.00	2.95 m	45
348.00-348.50	1.50 m	75
348.50-351.65	2.15 m	80
351.65-354.40	2.75 m	80
354.40-356.55	2.15 m	70
356.55-358.65	2.10 m	45
358.65-359.60	0.95 m	45
359.60-361.70	2.10 m	5
361.70-362.85	1.15 m	5
362.85-363.90	1.05 m	50
363.25-373.90	10.65 m	100
373.90-375.75	1.85 m	50
375.75-378.80	3.05 m	20
378.80-381.80	3.00 m	85
381.80-384.85	3.05 m	100
384.85-387.80	2.75 m	80

18.2 m  
AĞIRLIKLIL  
TAVAN TAŞI  
RQD : 47.4

13.7 m  
AĞIRLIKLIL  
TAVAN TAŞI  
RQD : 63.8

Şekil 6. JT4 sondajı kömür ve yakın kayaların RQD değerlendirmesi (Yenice ve ark., 2012)

Giriş-Çıkış (m)	Kalınlık	RQD
488.00-492.30	4.30 m	49
492.30-493.60	1.30 m	8
493.60-496.60	3.00 m	18
496.60-499.40	2.80 m	62

11.4 m  
AĞIRLIKLIL  
TAVAN TAŞI  
RQD : 41.6

Şekil 7. 4291 nolu sondajın kömür üstü kayaların RQD değerlendirmesi (Yenice ve ark., 2012)

### 3. KAYA KÜTLE SINIFLAMASINA YÖNELİK ÇALIŞMALAR

Yapılan jeoteknik saha ölçümleri ve laboratuvar çalışmaları sonucunda GLİ Ömerler Mekanize Yeraltı Kömür Ocağında derin panolarda galerilerin tahkimatına yönelik RMR sınıflama sistemine göre sınıflandırma işlemi yapılmıştır.

Sahada kaya kütle sınıflamasına yönelik ölçümler kullanılarak A panoları kömür üstü tavan kayaçları, kömür altı taban kayaçları ve kömür için RMR değerleri belirlenmiştir (Çizelge 4). Tahkimat tasarımında girdi parametresi olarak kullanılacak RMR değeri tavan kayaçları için 37, taban kayaçları için 46, kömür için 34 olarak tespit edilmiştir. (Yenice ve ark., 2012)

Çizelge 4. Ömerler Ocağı A Panoları Tavan, Taban Ve Kömür Kayaçları İçin RMR Değerinin Belirlenmesi

1	Sağlam Kayacın Dayanımı	Nokta Yüklü Dayanımı İndeksi (MPa)	>10	4 - 10	2 - 4	1-2	Düşük Aralıklar İçin Tek Eksenli Test		
	Tek Eksenli Basınç Dayanımı (MPa)	>250	100-250	50 - 100	25 - 50	5 - 25	1-5	<1	
	Derecelendirme	15	12	7	4	2	1	0	
	<b>TAVAN KAYACI</b>					<b>2</b>			
	<b>TABAN KAYACI</b>					<b>2</b>			
	<b>KÖMÜR</b>					<b>2</b>			
2	RQD (%)	90 - 100	75- 90	50 - 75	25 - 50	<25			
	Derecelendirme	20	17	13	8	3			
	<b>TAVAN KAYACI</b>				<b>8</b>				
	<b>TABAN KAYACI</b>			<b>13</b>					
	<b>KÖMÜR</b>				<b>8</b>				
3	Süreksizlik Aralığı (mm)	>2000	600-2000	200-600	60 - 200	<60			
	Derecelendirme	20	15	10	8	5			
	<b>TAVAN KAYACI</b>				<b>8</b>				
	<b>TABAN KAYACI</b>			<b>10</b>					
	<b>KÖMÜR</b>					<b>5</b>			
4	Süreksizlik Durumu	Çok kaba yüzeyler, sürekli değil, ayrılma yok. Sert eklem yüzeyleri	Az kaba yüzeyler, ayrılma <1 mm. Sert eklem yüzeyleri	Az kaba yüzeyler, ayrılma <1 mm. Yumuşak eklem yüzeyleri	Sürtünme izli veya fay dolgusu <5 mm veya 1-5 mm.	Yumuşak fay dolgusu >5mm kalınlıkta veya açık eklem > 5 mm devamlı süreksizlikler			
	Derecelendirme	30	25	20	10	0			
	<b>TAVAN KAYACI</b>			<b>20</b>					
	<b>TABAN KAYACI</b>			<b>20</b>					
	<b>KÖMÜR</b>			<b>20</b>					
5	Yer altı Suyu	Tünelin 10 m'lik kısmında gelen su	Yok	10 lt/dk	<25 lt/dk	25 - 125 lt/dk.	>125 lt/dk		
		Oran (Eklem su basıncının ana asal gerilmeye oranı)	0,0	0,0 - 0,1	0,1 - 0,2	0,2 - 0,5	>0,5		
		Genel Koşullar	Tamamen Kuru	Nemli	Islak	Damlama	Su akışı		
	Derecelendirme	15	10	7	4	0			
	<b>TAVAN KAYACI</b>				<b>4</b>				
	<b>TABAN KAYACI</b>				<b>4</b>				
	<b>KÖMÜR</b>				<b>4</b>				

**Eklem Yönelimine Göre Düzeltmeler**

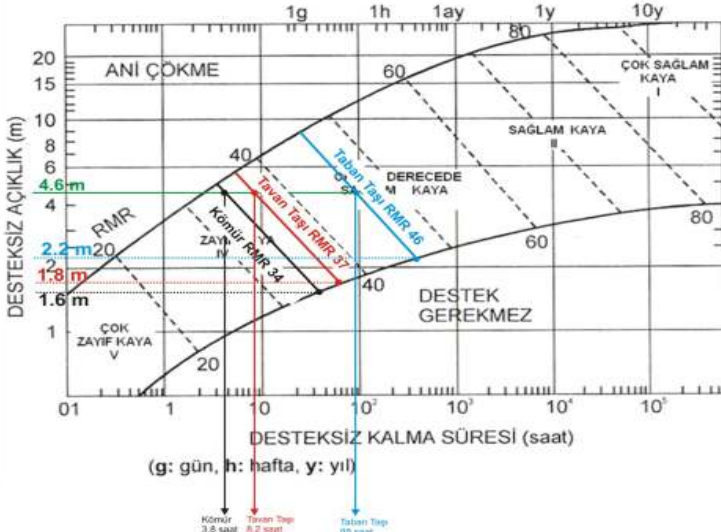
Eklemlerin doğrultu ve eğim yönelimi	Çok uygun	Uygun	Orta	Uygun değil	Hiç uygun değil
Tüneller	0	-2	-5	-10	-12
Derecelendirme	0	-2	-7	-15	-25
Şevler	0	-5	-25	-50	-60
<b>TAVAN KAYACI</b>			<b>-5</b>		
<b>TABAN KAYACI</b>			<b>-5</b>		
<b>KÖMÜR</b>			<b>-5</b>		

**Toplam sayısal değerlendirmeye göre (puanlama) Kayakütü sınıfları**

Derecelendirme	100 – 81	80 - 61	60 - 41	40 - 21	<20
Sınıflama No.	I	II	III	IV	V
Tanımlama	Çok iyi kaya	İyi kaya	Orta kaya	Zayıf kaya	Çok zayıf kaya
<b>TAVAN KAYACI RMR</b>				<b>37</b>	
<b>TABAN KAYACI RMR</b>			<b>46</b>		
<b>KÖMÜR</b>				<b>34</b>	
<i>Kaya kütlesi sınıflarının ifade ettiği anlamlar (Karpuz ve Hindistan,2006)</i>					
Ortalama tahkimatsız göçmeden durma zamanı	15m açıklık için 10 sene	8 m açıklık için 6 ay	5m açıklık için 1 hafta	2.5 m açıklık için 10 saat	0.1 m açıklık için 30 dakika
Kaya kütlesinin kohezyonu	>400 kPa	300-400 kPa	250-300 kPa	100-200 kPa	<100 kPa
Kaya kütlesinin içsel sürtünme açısı	>45°	35°-45°	25°-35°	15°-25°	<15°

Belirlenen RMR değeri için çalışma sahasında açılacak olan galerilerde desteksiz durma süresi Bieniawski 1989 tarafından önerilen ve Şekil 8’de verilen abak kullanılarak belirlenmiştir. Elde edilen sonuçlar Çizelge 5’de verilmektedir. Çizelge 5’de görüldüğü gibi derin sahalarda

açıklık miktarına göre desteksiz kalma süreleri en yüksek tabantaşı, daha sonra tavan taşı ve en düşük kömürde tespit edilmiştir. Bu veriler ışığında tabantaşı orta derecede sağlam kaya, tavantaşı zayıf kaya - orta derece kaya sınırında, kömür ise zayıf kaya sınıfında yer almaktadır.



Şekil 8. RMR'ye derin sahalarda tanımlanan kaya kütleleri için tahkimatsız durabilme süreleri (Yenice ve ark., 2012)

Çizelge 6. RMR Sistemine Göre Tahkimatsız Kalma Süreleri (Yenice ve ark., 2012)

Litoloji	RMR Değeri	RMR'a Göre Tahkimatsız Durma Açıklıkları		
		Açıklık (m)	Açıklık (m)	Süre (Saat)
Taban Taşı	46	2.2	4.6	88
Tavan Taşı	37	1.8	4.6	8.2
Kömür	34	1.6	4.6	3.8

#### 4 SONUÇLAR VE DEĞERLENDİRME

Kömür madenciliğinde yüzeye yakın yeraltı madencilik çalışmaları gün geçtikçe azalmakta daha derinlerdeki rezervlerin değerlendirilmesi yoğun olarak gündeme gelmektedir. Bunun doğal sonucu olarak önceleri yüzeye yakın şartlar için belirlenen yeraltı madenciliği tasarım parametreleri tekrar değerlendirilmektedir. Yeraltı madenciliği için bu tasarım parametrelerinden en önemlisi açılan açıklığın büyüklüğüne ve çevre kayaçların özelliğine göre belirlenen tahkimat sistemleridir. Derin sahalarda yapılacak üretimlerde bir diğer beklenti ise üretim kapasitelerindeki artıştır. Bu nedenle yüksek üretim kapasiteli makine ekipmanın yeraltında uygun çalışma şartlarının sağlanması ancak açılacak boşlukların genişlemesi ile mümkündür. Özellikle mekanize sistemlerde pano hazırlık galerileri yürüyen tahkimatların (şiltlerin) taşınmasını sağlayabilecek genişlik ve yükseklikte olmalıdır. Bu çalışmada T.K.İ. G.L.İ. Ömerler Mekanize Yeraltı Ocağı derin panolarında hazırlık galerileri tahkimat çalışmaları için kaya kütle sınıflamaları kullanılarak açıklık miktarına göre tahkimatsız kalma süreleri belirlenmiştir. Buna göre;

1. Yeraltında yapılacak üretim ve açılacak hazırlık galerilerinin tahkimat tasarımında kullanılmak üzere, sahadan alınan kayaç numuneleri üzerinde kaya mekaniği testleri (fiziksel ve mekanik özellik) yapılmıştır.

Deney sonuçlarına göre, taban taşı dayanım değerlerinin tavan taşı ve kömürden daha yüksek olduğu görülmektedir. Bu durumda işletmede halen uygulanmakta olduğu gibi derin panolarda da hazırlıkların genel olarak taban taşında sürdürülmesi uygundur.

2. Arazi çalışmaları, laboratuvar deneyleri ve daha önceki yıllarda saha ile ilgili yapılan diğer çalışmaların değerlendirilmesi sonucunda belirlenen kaya kütle kalite değerlerine göre, tavan taşı, kömür ve taban taşında açılacak olan galerilerde, açıklığa bağlı olarak tahkimatsız durma süreleri tespit edilmiştir. RMR puanı 37 olarak belirlenen tavan taşında açılacak olan galerilerde 4.6 m açıklık için tahkimatsız kalma süresi 8.2 saat olarak tespit edilmiştir. Benzer şekilde RMR puanı 34 olan kömür damarında açılacak olan galerilerde 4.6 m açıklık için tahkimatsız kalma süresi 3.8 saat olarak tespit edilmiştir. Sahada en yüksek dayanım değerlerine sahip olan taban taşında RMR puanı 46 olarak belirlenmiştir. Bu değerler için taban taşında açılacak olan galerilerde 4.6 m açıklık için tahkimatsız kalma süresi 88 saat olarak tespit edilmiştir.

Genelde tünellerde çalışmalarında kullanılan kaya kütle sınıflama sistemlerinin yeraltı galerileri tahkimat tasarımında kullanılması bir takım sınırlar ve kabüller içermektedir. Diğer taraftan derin sahalarda açılacak hazırlık galerilerinin tahkimatı için galerinin açıldığı kaya kütlelerinin sayılaşırılması tasarımın belirlenmesi aşamasında önemli girdi parametreleri ile katkı sağlamaktadır. Galerilerin açılması sırasında kullanılacak desteksiz açıklık süresinin kütle özellikleri kullanılarak belirlenmesi sonucunda elde edilen değerler pratikte uygulanabilir gerçekçi değerlerdir. Sonraki aşamada tahkimat tasarımına ve tahkimat sistemi seçimine yönelik problemlerin çözümü sayısal yaklaşımlar ve/veya matematiksel modeller (sonlu elemanlar, sonlu farklar) kullanılarak gerçekleştirilebilir.

## TEŞEKKÜR

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## Mechanical Properties of Shotcrete with Calcium Carbonate Admixture

### *Kalsiyum Karbonat Katkılı Püskürtme Betonun Mekanik Özellikleri*

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**ÖZET** Ülkemizde faaliyet gösteren bir altın madeninde açık işletme sınırı geçildikten sonra yeraltı işletme faaliyetleri başlamıştır. Cevher içeriğinin fazla olduğu bölgelerde burğu şeklinde ana nakliyat galerileri sürülerek belirli bir kata kadar inilmiştir. Bu galerilere bağlı üretim galerileri sürülerek cevher içeren kayaç çıkartılmaktadır. Ömürleri kısa süreli olan bu galerilerin sağlamlaştırılması için püskürtme beton kullanılmaktadır. Üretim galerileri kazıldıktan sonra dolgu ile doldurulur ve bir üst seviyedeki üretim galerileri açılmaya başlanır. Görüldüğü gibi bu türden galerilerin ayakta kalma zamanı kısa sürelidir. Çimento pahalı bir püskürtme beton elemanı olduğu için emniyet sınırları dâhilinde çimento içeriğinin azaltılarak yerine kullanılabilecek bir katkı malzemesi eklenmesiyle maliyet optimizasyonu yapmak mümkün olabilecektir. Bu çalışmada bu şekilde bir iyileştirme yapmak için püskürtme beton çimento içeriği yerine nispeten daha ucuz olan kalsiyum karbonat katkısı %18, 21, 25 ve 30 olacak şekilde içeriğe eklenmiş bunların mekanik özellikleri ortaya koyulmuştur. Çalışma göstermiştir ki belirli oranlardaki kalsiyum karbonat katkılı püskürtme beton güvenliği riske atmayacak şekilde kullanılabilir.

**Anahtar Kelimeler:** *Kalsiyum karbonat, püskürtme beton, üretim galerileri, yeraltı altın madeni.*

**ABSTRACT** Underground mining facilities have been started after exceeding the optimum open pit limit in a Turkish gold mine. Main haulage roadways are entirely excavated in non-valuable rock formation to reach a particular level which is an access to the ore body. Production galleries connected to the main haulage roadways, are excavated for extracting ore body. Shotcrete is used for reinforcing and supporting these production galleries. Production galleries have been filled just after the excavation is finished and the excavation of upper level production galleries are initiated. As shown in the working cycle, the life time of these galleries are not so long. Cement is a very expensive component of shotcrete. Cost optimization can be possible with conserving the safety factor of shotcrete by using admixtures instead of cement. The subject of this study is the mechanical analysis of the shotcrete by using calcium carbonate admixture which is cheaper relatively, instead of cement to optimize the cost of the shotcrete expense. Calcium carbonate admixture with 18%, 21%, 25%, and 30% is used instead of cement rate in total prescription. When results are analyzed, the current study indicates that shotcrete by using admixture of calcium carbonate may be used without risking the safety factor in the circumferences of production galleries.

**Keywords :** *Calcium carbonate, production galleries, shotcrete, underground gold mine*

## 1 INTRODUCTION

A gold mine, which is resisted to declaration its name by their commendatory proposal, operates a mechanized underground Au-Ag mine in the middle west of Turkey in a place one of the province of İzmir. The transportation for equipments is not a major problem for that gold mine because it is located between two developed county; Dikili and Bergama and next to the Izmir-Canakkale Highway.

The mine began operation as an open pit mine and moved to underground mining because of exceeding the optimum open pit limit. The mining method commonly used in this mine is the mechanized cut and fill method. It is possible to divide that mine into two main parts according to major ore bodies such as S and M veins. Sublevel caving method, the secondary mining method, is occasionally used because cut and fill method is not always feasible in the S vein. Production in the current mine is carried out with full mechanization.

Cut and fill method is applied to two parallel main haulage roadways which are completely excavated in non-valuable rock mass and production galleries are connected to them. There are 15 meter differences in elevation of two roadways. Roadways and production galleries are excavated by drill and blast method. Reinforcement and supporting systems of these galleries are carried out by shotcrete applications. Backfill process of the production galleries has been begun just after the production galleries' life is over.

Shotcrete is generally used as an integral part of a ground support system consisting of multiple components and also it is an unavoidable component of this mining method. One of the main expense item of the underground mine operation is shotcrete in which cement is the highest cost of the shotcrete mixture. A reduction of the shotcrete cost may be possible by using some additives instead of cement. Current paper focuses on investigating the mechanical properties of shotcrete with calcium carbonate addition which is relatively

cheaper than cement. Calcium carbonate admixture was added into shotcrete mixture in a share with 18%, 21%, 25% and 30% of total weight of the cement respectively. Results are analyzed with respect to the use of calcium carbonate addition into shotcrete mixture while preserving the safety factor.

It has been proven by the several researchers that calcium carbonate admixture has been a various advantages. Calcium carbonate addition significantly increased the early strength of shotcrete (Menendez et al, 2003, Ramachandran, 1986) when comparing with other fillers such as fly ash (Felekoğlu et al, 2006, Elkhadiri et al, 2002). Calcium carbonate admixture is not just a filler component of the shotcrete but also a reactive agent and it has a chemical effect on the hydration. It accelerates the hydration process of the shotcrete (Kakali et al, 2000, Ramachandran and Zhang, 1986, Gesoğlu et al, 2012). Moreover, calcium carbonate additive raises the viscosity of the shotcrete mixture (Petit and Wirquin, 2010, Gesoğlu et al, 2012) which is very useful for application of the shotcrete process.

## 2 SHOTCRETE WORKS

### 2.1 Shotcrete Process

Shotcrete, which consists of aggregate, cement, water and some additives, can be defined as pneumatic application of a mixture onto a surface at high pressure. Shotcrete application can be divided into two main processes according to the use of water and additives (Arioğlu et al, 2008): dry and wet shotcrete. Water is added into the shotcrete mixture during the application at dry shotcrete and it is added into the mixture before the application in a storage tank at wet shotcrete. Flexible use conditions, usage of without molding and improving features with additives are the advantages of shotcrete.

### 2.2 Shotcrete Applications in the Mine

Shotcrete is mainly used to provide surface support or skin control between the roof bolts which serve as the primary ground support elements in the relevant gold mine.



Shotcrete mixture, interested in this paper used as a reinforcement and supporting element at only production galleries. Usage time of these production galleries is relatively short. Shotcrete applied with steel mesh and some types of fibers for main galleries in long-term. Shotcrete is prepared in an integral concrete plant and transferred into the underground mine with trans-mixers which are drawn near the mobile sprayer and last controls of the machines are done by the operators for applying shotcrete properly and successfully. Application of shotcrete can be seen in the Figure 1.



Figure 1. Application of Shotcrete

### 3 EXPERIMENTAL STUDIES

The experimental program was planned for investigating on mechanical properties of shotcrete with calcium carbonate admixture in various rates with two different curing time 7-28 days.

#### 3.1 Materials

In this research, commercially available normal cement, which is produced by the Cimentas in Turkey, was used. CEM I 42.5 model Portland cement has C<sub>3</sub>S with 63.60%, C<sub>2</sub>S with 9.35%, C<sub>3</sub>A with 6.62% and C<sub>4</sub>AF with 10.99% as a mineralogical content.

Calcium carbonate admixture which is used in tests has at least CaCO<sub>3</sub> with 95% and 100% of its particles passing through a 2mm sieve mesh and at least 70% passing through a 63µm sieve mesh.

Granulometric distribution of aggregate is 100% passing through 7 mm sieve mesh and

the water in the shotcrete mixture is provided from the public water supply. Plasticizer used in the shotcrete mixture is produced and named as a brand with Rheobuild by BASF Chemical Company.

#### 3.2 Testing Methods

In order to find out mechanical properties of shotcrete mixtures, experiments carried out are listed below:

- Uniaxial Compressive Strength (UCS) Test
- Point Load Index (PLI) Test
- Splitting Tensile Strength (STS) Test
- Flexural Strength (FS) Test
- Elasticity Modulus

#### 3.3 Recipes of the Shotcrete Mixture

Four different types of recipe were prepared for investigating the mechanical properties of shotcrete with calcium carbonate admixture. The recipe having calcium carbonate admixture with 18% instead of total cement, is already used by the relevant gold mine operations and it was a reference for preparing the recipes. The calcium carbonate ratio of the recipe has been then increased to 21%, 25%, and 30% respectively when the other mixture contents were remained same. The ingredients of the recipe have been shown in the Table 1.

#### 3.4 Schedule of Experiments

Mechanical properties of the shotcrete were investigated in a set of tests on samples from the mine concrete plant (MCP). Laboratory tests were performed on 312 samples having various sizes and shapes. All samples were cured in water for 7 and 28 days. From start to finish, all procedure for the laboratory experiments were suitable for the British Standards (BS EN 12390-1 and 2, 2000-2009). Totally, 48 samples for each one were collected from OCP for compressive strength and flexural strength tests. 80 and 136 samples were collected for point load strength and tensile splitting strength tests respectively. Shapes, sizes, curing time and number of samples are given in the Table 2.

Table 1. Shotcrete mixture rates

Ingredient	The Rate of the Calcium Carbonate Admixture*							
	%18		%21		%25		%30	
	kg/m <sup>3</sup>	%	kg/m <sup>3</sup>	%	kg/m <sup>3</sup>	%	kg/m <sup>3</sup>	%
Aggregate	1510.16	69.90	1510.83	69.97	1513.00	69.95	1505.17	69.80
Cement	368.83	17.07	354.00	16.39	337.00	15.58	316.33	14.67
Water	195.50	9.05	194.33	9.00	196.00	9.06	195.17	9.05
Admixture (Other)	4.515	0.21	4.52	0.21	4.49	0.21	4.53	0.21
Calcium Carbonate	81.50	3.77	95.67	4.43	112.60	5.21	135.17	6.27
Total		100		100		100		100

\* Represents the rates of the calcium carbonate admixture instead of cement

Table 2. Properties of the specimens

Name of Experiment	Shape of the Sample	Sizes of the Sample (mm)	Cure Time of the Sample (day)	Number of the Sample
TST	Cylinder	R:100 h:200	7-28	40
TST	Cylinder	R:100 h:100	7-28	96
PLI	Cylinder	R:100 h:100	7-28	80
FST	Rectangular Prism	150 *150*525	7-28	48
UCS	Cube	150*150*150	7-28	48
Total				312

### 3.5 Laboratory Experiments

#### 3.5.1 UCS test, elasticity modulus and density

UCS of the specimen is calculated to analyze performance of the shotcrete with calcium carbonate admixture according to Equation (1) (BS EN 12390-3, 2009);

$$UCS = \frac{F}{A_c} \tag{1}$$

where

UCS=the compressive strength, in MPa  
 F=the maximum load at failure, in N;  
 A<sub>c</sub>=the cross-sectional area of the specimen on which the compressive force acts, in mm<sup>2</sup>.

Elasticity moduli of the specimens are calculated due to Equations (2), (3) (Karpuz ve Hindistan, 2006);

$$\varepsilon = \frac{\Delta l}{l} \tag{2}$$

where

ε=unit reduction in length (dimensionless);  
 Δl=the amount by which the length of the specimen changes, in mm;  
 l=the original length of the specimen, in mm;

$$E_t = \frac{\Delta \sigma}{\Delta \varepsilon} \tag{3}$$

where

E<sub>t</sub>=Young's modulus, in GPa;  
 Δσ=unit stress, in MPa;  
 Δε=unit strain, (%).

Density of the specimen is calculated according to basic Equation (4);

$$p = \frac{m}{v} \tag{4}$$

where

p=density of the specimen in g/cm<sup>3</sup>;  
 m=mass of the specimen in g;  
 v=volume of the specimen in cm<sup>3</sup>.

### 3.5.2 TSS test

Tensile splitting test is applied into specimens to analyze performance of the shotcrete with calcium carbonate admixture. Equation (5) is used for getting the tensile splitting strength test result (BS EN 12390-6, 2009);

$$TSS = \frac{2 \times F}{\pi \times d \times l} \quad (5)$$

where

TSS=the tensile splitting strength, in MPa (N/mm<sup>2</sup>);

F=the maximum load, in N;

l=the length of the line of contact of the specimen, in mm;

d=the designated cross-sectional dimension, in mm.

### 3.5.3 Flexural Strength Test

Flexural Strength Test is applied to beam samples to analyze performance of the shotcrete with calcium carbonate admixture. To reach the flexural strength test results, Equation (6) is used (BS EN 12390-5, 2009);

$$FS = \frac{3 \times F \times S}{2 \times d_1 \times d_2^2} \quad (6)$$

where

FS=the flexural strength, in MPa (N/mm<sup>2</sup>);

F=the maximum load, in N;

S=the distance between the supporting rollers, in mm;

d<sub>1</sub>,d<sub>2</sub>=the lateral dimensions of the specimen, in mm.

### 3.5.4 PLI Test

Point Load Index Test applied to analyze performance of the shotcrete with calcium carbonate admixture. To calculate the point load index, Equations (7) and (8) are used and measurements are given in ISRM, 1985.

$$D_e = \sqrt{\frac{4F}{\pi}} \quad (7)$$

where

D<sub>e</sub>=equivalent radius in mm;

F=the cross-sectional area of the axis on which the force acts in mm<sup>2</sup>.

$$PLI = \frac{P}{D_e^2} \quad (8)$$

where

PLI=the point load index, in MPa;

P=the maximum load at failure, in N.

## 4 RESULTS

The average results of the experiments carried out in the Rock Mechanics and Natural Stones Laboratory of Istanbul Technical University are given in Table 3.

Some strength relationships may be seen according to results in Table 3 and they are illustrated between Figure 2 and 11. Regression correlations point out that they have meaningful regression coefficients, achieved according to the specimens which have 7 days curing time.

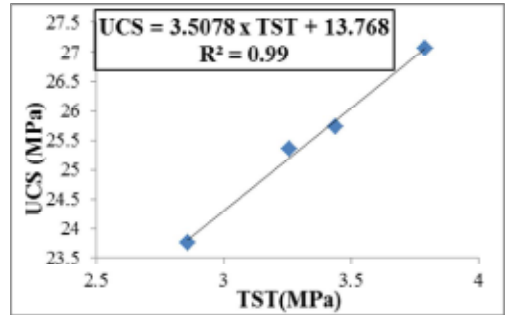


Figure 2. Relationship between TST and UCS (7 days curing time)

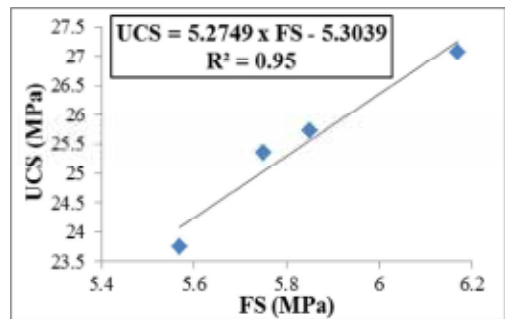


Figure 3. Relationship between FS and UCS (7 days curing time)

Table 3. Summary of the results

Specimen Identification	UCS (MPa)	SD**	PLI (MPa)	SD	FS (MPa)	SD	TST (MPa)	SD	p (g/cm <sup>3</sup> )	SD	E <sub>i</sub> (GPa)	SD
19-11-18-H	23.75	1.34	1.15	0.06	5.57	0.36	2.86	0.41	2.24	0.01	22.46	5.09
16-11-21-H	27.06	1.52	1.58	0.07	6.17	0.45	3.79	0.62	2.27	0.02	32.99	6.49
17-11-25-H	25.72	0.72	1.52	0.14	5.85	0.56	3.44	0.43	2.24	0.03	35.45	6.64
18-11-30-H	25.36	1.10	1.37	0.03	5.75	0.52	3.26	0.47	2.21	0.01	26.64	5.80
19-11-18-H-P	-	-	0.70	0.16	-	-	2.57	0.46	-	-	-	-
16-11-21-H-P	-	-	0.67	0.19	-	-	2.07	0.46	-	-	-	-
17-11-25-H-P	-	-	0.62	0.20	-	-	1.68	0.33	-	-	-	-
18-11-30-H-P	-	-	0.49	0.33	-	-	1.19	0.38	-	-	-	-
19-11-18-A	42.44	3.03	1.31	0.17	4.91	0.26	3.12	0.51	2.31	0.01	63.28	10.53
16-11-21-A	37.94	3.23	1.63	0.10	5.35	0.39	3.63	0.66	2.30	0.01	49.85	8.15
17-11-25-A	38.02	1.23	1.57	0.07	6.36	0.55	3.31	0.55	2.30	0.01	47.48	4.75
18-11-30-A	32.06	2.03	1.54	0.17	5.29	0.48	3.06	0.57	2.29	0.01	47.20	8.64
19-11-18-A-P	-	-	1.14	0.09	-	-	2.79	0.42	-	-	-	-
16-11-21-A-P	-	-	1.21	0.19	-	-	2.49	0.45	-	-	-	-
17-11-25-A-P	-	-	0.73	0.20	-	-	2.28	0.52	-	-	-	-
18-11-30-A-P	-	-	0.49	0.17	-	-	1.23	0.55	-	-	-	-

\* Specimen Identification: First two digits of the code is the date (day-month) when the specimen was prepared. Third digit of the code is amount of specimen's calcium carbonate admixture instead of cement quantity. Meaning of last letters of the code; H: 7 days cure time, A: 28 days cure time, P: Specimen is taken directly from the applied shotcrete. \*\*SD: Standard Deviation

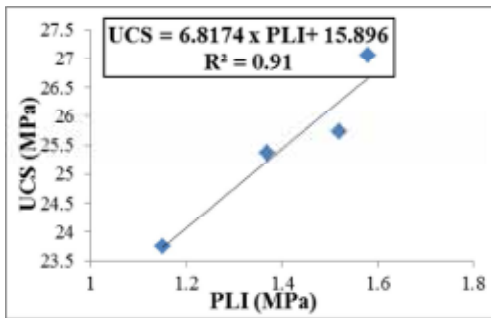


Figure 4. Relationship between PLI and UCS (7 days curing time)

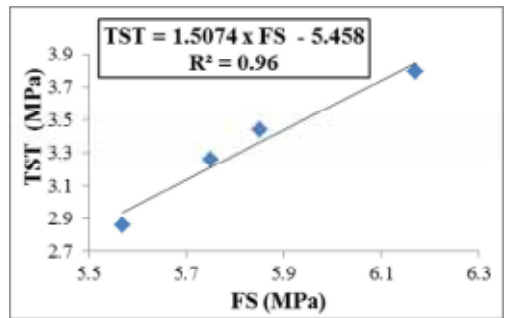


Figure 5. Relationship between FS and TST (7 days curing time)

Regression correlation on linear and logarithmic mode with a high correlation coefficient can be found with respect to analysis of the relationship between point load strength and others. These correlations can be found in Table 4. Moreover, interrelation between flexural strength, compressive strength and splitting strength can be found between Figure 2 and 5.

Regression correlation on linear and logarithmic mode with a high correlation coefficient had been found pursuant to relation between flexural strength, point load strength and tensile splitting strength. Estimate equations and correlation coefficients can be found in Table 4.

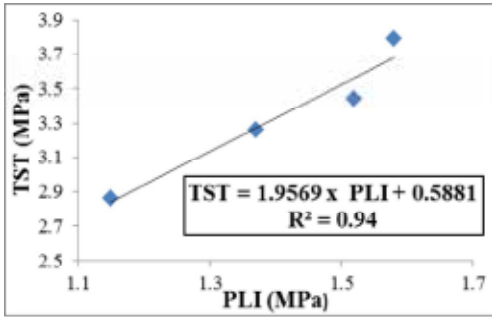


Figure 6. Relationship between PLI and TST (7 days curing time)

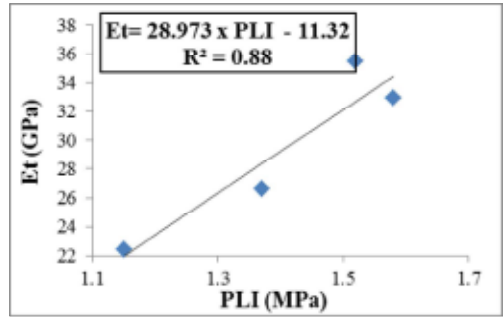


Figure 9. Relationship between PLI and Et (7 days curing time)

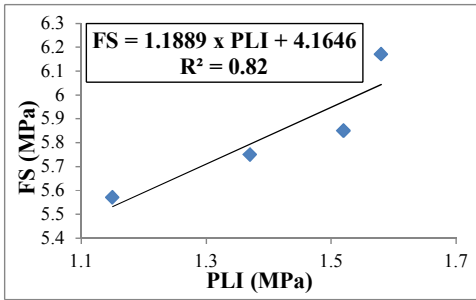


Figure 7. Relationship between FS and PLI (7 days curing time)

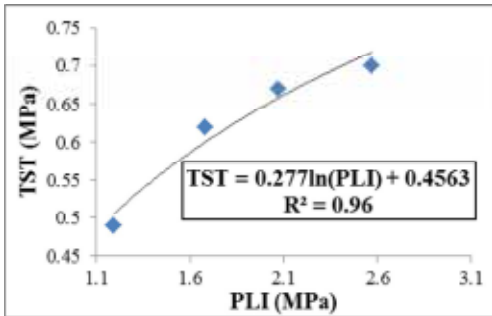


Figure 8. Relationship between PLI and TST (7 days curing time; specimens are directly taken from the shotcrete)

Table 4. Regression equations according to relationships of shotcrete strengths (7 days).

Compared Results	Regression Equation	Regression Coefficient
UCS-TST	UCS = 3.5078 x TST + 13.768	R <sup>2</sup> = 0.99
PLI-UCS	UCS = 6.8174 x PLI + 15.896	R <sup>2</sup> = 0.91
UCS-FS	UCS = 5.2749 x FS - 5.3039	R <sup>2</sup> = 0.95
TST-FS	TST = 1.5074 x FS - 5.458	R <sup>2</sup> = 0.96
PLI-TST	TST = 1.9569 x PLI + 0.5881	R <sup>2</sup> = 0.94
PLI-FS	FS = 1.1889 x PLI + 4.1646	R <sup>2</sup> = 0.82
PLI*-TST*	TST = 0.277ln(PLI) + 0.4563	R <sup>2</sup> = 0.96
Et-PLI	Et = 28.973 x PLI - 11.32	R <sup>2</sup> = 0.88

\*Specimens are taken from the shotcrete directly

## 5 CONCLUSIONS

Strength relationship on 7 days curing time specimens can be seen according to the results in Table 3 but similar relationships could not be observed on the specimens which have 28 days curing time because calcium carbonate participate in the cement hydration as a reactive, thus this process affects the final mechanical properties of the shotcrete negatively ( Matschei, 2007, Felekoğlu et al, 2006, Elkhadiri et al, 2002 ).

In Table 3, advantages of the calcium carbonate can be observed for early age strength; however these advantages cannot be seen for final strength. These effects on the final strength should be studied in future for much more than 28 days curing time also another study is necessary for determining the precise contribution of calcium carbonate on early age strength.

A lot of underground openings have to deal with water Influence of the water with calcium carbonate admixture on cement hydration should be also investigated.

According to the results, 21% ratio of the calcium carbonate admixture is more effective for the strength of shotcrete. The equations in Table 4 can also be used for estimating the strength parameters of the shotcrete which have %18-30 calcium carbonate admixture.

Strength variations of the specimens (up to 40%) which are taken directly from the shotcrete are caused by the segregation of the shotcrete mixture. Additionally, accelerator admixture effects on the shotcrete must be inquired for the reason of the segregation. In mining and tunneling, accelerator is an integral part of the shotcrete and its effects on the shotcrete performance is a considerable subject.

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## Ege Linyitleri İşletmesi (ELİ) Eynez Yeraltı Ocağı Mekanize Uzunayak Panosu Tavan Kömürü Üretiminin Sayısal Modellemesi

### *Numerical Modelling of Mechanized Longwall Mining with Top Coal Caving in Aegean Lignite Establishment*

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**ÖZET** Kalın kömür damarlarında, tavan kömürünün ayak arkasından göçertilerek üretilmesi ilkesine dayanan uzunayaklarda, üretim verimini etkileyen en önemli parametre, tavan kömürünün etkin bir şekilde göçertilmesidir. Tavan kömürü ayak arkasından akıtılırken, kömürün bir kısmı göçüğe karışma neticesinde üretilmemekte, aynı zamanda kazanılan kömür içerisine de tavan taşı karışarak kömürün seyrelmesine, rezerv kazanım oranının azalmasına ve dolayısıyla üretim veriminin düşmesine neden olmaktadır. Bu çalışma kapsamında, Ege Linyitleri İşletmesi (ELİ) Eynez Yeraltı Ocağı Mekanize Panosunun, sonlu farklar ilkesine göre çalışan FLAC<sup>3D</sup> programı kullanılarak üç boyutlu modeli oluşturulmuştur. İlk olarak kömür tabanındaki uzunayağın çevresinde oluşan ikincil gerilmelerin değerleri belirlenmiştir. Daha sonra, üretimin en önemli aşaması olan tavan kömürünün göçertilmesi ve kalkan tahkimat ünitesi olduğundan akması modellenmiştir. Çalışma sonucunda tavan kömürünün yeterince kırıldığı, fakat tavanda yer alan marnın yeterince kırılmadığı ve kömürün akıtıldığı bölgede büyük boşlukların oluştuğu hem yerinde çalışmalarda hem de modelleme çalışmalarında belirlenmiştir. Verimli ve hızlı bir şekilde üretim yapılabilmesi için Eynez Yeraltı Ocağında da marnlı bölgede çatlaklandırma patlatmasının yapılması öngörülmektedir.

**ABSTRACT** Effective caving of top coal is the most critical factor in maintaining an efficient production in longwall with top coal caving method. During production of the top coal by caving, considerable amount of coal may be left in the goaf and in addition, the produced coal may be diluted with rock, consequently, decreasing the extraction ratio and production efficiency. In this study, numerical studies associated with numerical modelling of Eynez Underground Mine have been carried out. Firstly, the longwall panel was modelled by using a software called FLAC<sup>3D</sup>, developed based on finite difference technique. Secondary, the induced stresses developing around the longwall face have been determined. Finally, the most critical step of production that is top coal caving through the frames of shields has been modelled. In Eynez Underground Mine, top coal was sufficiently fractured but the marl at the roof was not fractured and there occurred a big opening after caving of top coal. In order to achieve efficient and quick production, the marl must be cracked in as uniformly sized pieces as possible.

## 1 GİRİŞ

Enerji ihtiyacımız arttıkça, ülkemizde yaygın olan kömür rezervleri ön plana çıkmaktadır. Bu rezervlerin büyük kısmını kalın kömür damarları oluşturmaktadır. Ülke ekonomisi için bu kömürlerin düşük maliyetle ve verimli şekilde üretileceği en uygun üretim yöntemlerinin belirlenmesi gerekmektedir.

Kömür damarları ince, orta ve kalın damar olarak sınıflandırılmaktadır. Bu sınıflandırma, göreceli olmakla birlikte, istatistiksel olarak yapılan hesaplamalarla belirlenmektedir. Dünyada kalın kömür damarı tanımı, ülkelere ve yerel madencilik endüstrilerine göre değişmekle birlikte, 4,8 metre, kalın kömür damarları için alt sınır olarak kabul edilmektedir (Singh, 1999). Mekanize olarak yaklaşık 6 metre kalınlıktaki kömür damarından tek dilim uzunayak yöntemi ile üretim yapılabilir de pratik olarak ayak yüksekliği 4 metrenin üzerinde olmamakla birlikte, damarın eğimine ve düzenliliğine, kömürün ve yan kayacın mekanik özelliklerine, kömürün oksidasyon yeteneğine göre 2-2,5 metre ayak yüksekliği, oldukça yaygın olarak kullanılmaktadır (Yaşıtlı, 2005a; Yaşıtlı, 2005b).

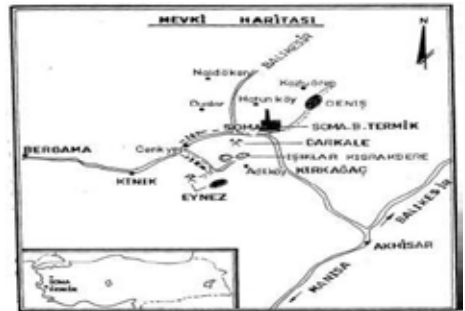
Bu çalışmada; ELİ Eynez Yeraltı Ocağı Mekanize Panosundaki kalın kömür damarlarında, tavan kömürü göçertmeli uzunayak yöntemi ile üretim yapılması sırasında, ayak çevresinde oluşan gerilmelerin ve tavan kömürü akmasının, FLAC<sup>3D</sup> programında sayısal olarak modellenmesi amaçlanmıştır. Çalışma sonunda, tavan kömürü akması sırasında kömür seyrelmesine neden olan etmenlerin belirlenmesi, kömür rezervi kazanım oranının ve üretim verimliliğinin artırılması hedeflenmiştir.

## 2 ELİ EYNEZ YERALTI OCAĞININ KISACA TANITILMASI

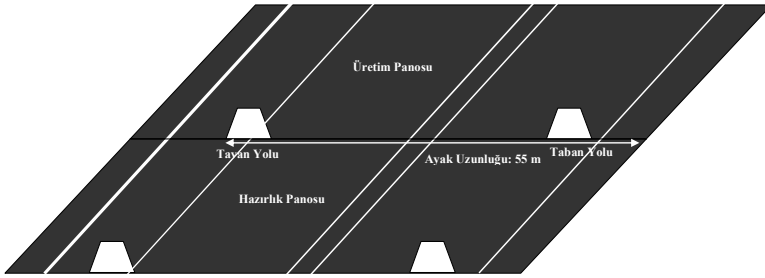
Eynez işletme Müdürlüğü Soma-Bergama yolu üzerindeki Cenk Yeri kasabasından 11 Km güney istikametinde Karanlık Dere

mevkiindedir. Somadan uzaklığı 25 km'dir. Açık ocak sahası ise Karanlık Dere yol ayrımından itibaren güney-doğu istikametinde 3 km uzaklıktadır (Şekil 1). Genel olarak Soma havzasında farklı zamanlarda oluşmuş üç adet linyit damarı mevcuttur. Eynez İşletme Müdürlüğü'ne ait ruhsatlı sahada KM1 ve KM2 olarak adlandırılan miosen yaşlı iki adet linyit damarı bulunmasına karşın bunlardan sadece KM2 ekonomik değerdedir. Ana damar kalınlığı 30 m'ye kadar çıkmaktadır. Kömürün tek eksenli basma dayanımı 23 MPa civarındadır. Damar eğimi 10°-25° arasında değişmektedir. Üretilen kömürün alt ısı değeri 5000 kcal/kg'dır (Ünver vd., 1991; Ünver, 1995).

Mekanize ayak hazırlık çalışmaları 1996 yılı sonlarında bitirilmiş ve 1997 yılında mekanize sistem ile üretim faaliyeti başlamıştır. Uygulanan işletme sistemi "pencereli (kalkan) tahkimat" sistemidir. Başka bir ifade ile "Yatay dilimli, tavandan (pencereli) göçertmeli, geri dönüşlü uzunayak" yöntemidir. Klasik uzunayakta 5 metre olan dilim kalınlığı, mekanize sistemde 7,5 - 9 metre arasında değişmektedir. Mekanize ayakta çalışılan dilimin 2,8 metresi ayandan kesici yükleyici ile kesilerek üretilmektedir. Dilimin kalan kısmı ise şilt tahkimatlarda bulunan pencerelerden akıtılarak üretilmektedir. Tavandan alınan kömürün akmaması durumunda delme-patlatma yapılmaktadır.



Şekil 1. ELİ üretim bölgesi coğrafi konumu



Şekil 2. Mekanize ayak kesiti

Mekanize ayak tavan yolu kömürün tavan taşı (Marn) kondağında sürülmektedir. Mekanize ayak taban yolu ise kömür damarının tabanına yakın kısımda sürülmektedir. Tavan ve taban yollarının birleştirilmesi ile oluşan ayak uzunluğu damar kalınlığına ve eğimine bağlı olmakla birlikte, 30-60 m arasında değişmektedir. Yapılan iki kesim ile şilt tahkimatın yürütülmesi sonunda şiltlerin pencereleri açılarak tavan kömürü alınmaktadır. Günlük iki vardiyada üretim, üçüncü vardiya da ayak bakımı yapılmaktadır. Günlük ortalama 2.5 metre ilerleme yapılmaktadır (Şekil 2).

### 3 EYNEZ YERALTI OCAĞI MEKANİZE PANOSUNUN FLAC<sup>3D</sup> PROGRAMINDA MODELİNİN OLUŞTURULMASI

Modelleme işlemi iki aşamada gerçekleştirilmiştir. Birinci aşamada ayak çevresinde meydana gelen gerilme dağılımlarının belirlenmesine yönelik bir model oluşturulmuştur. İkinci aşamada ise birinci aşamadan elde edilen gerilme dağılımları kullanılarak ayağın sadece küçük bir bölümü göz önüne alınarak akma modellemesinin yapıldığı ikinci bir model oluşturulmuştur. Bu modelde tavan kömürünün göçertilerek kalkan tahkimatların pencerelerinden üretilmeleri sırasında akma mekanizmaları belirlenmeye çalışılmıştır.

#### 3.1 Gerilme Dağılımlarının Belirlenmesine Yönelik Yapılan Modelin Geometrisi ve Ayağın Oluşturulması

FLAC<sup>3D</sup> programı kullanılarak mekanize panosunun 3 boyutlu modeli oluşturulurken izlenen yol, aşağıda verilmiştir:

1. Mekanize uzunayağının ayak uzunluğu 55 metredir. Modelde ayak uzunluğu "+x" yönünde 55 metre alınmıştır.
2. Mekanize uzunayağının pano boyu faylara göre sınırlandırılmaktadır. Bu proje çalışmasında pano boyu, modelde "+y" yönünde 250 metre alınmıştır.
3. Mekanize uzunayağın panosu, yüzeyden yaklaşık olarak 280 metre derindedir. Kömür damarı eğimli olduğundan "z" yönünde modelin tamamının oluşturulması mümkün olmamıştır. Bu sebeple modelde derinlik, "-z" yönünde 50 metre alınmış, örtü yükü modele dışarıdan uygulanmıştır.
4. Uzunayağın etrafında oluşan gerilmelerin daha ayrıntılı olarak belirlenebilmesi için model oluşturulurken, modelin üst kısmındaki ağlar (mesh) büyük, alt kısmındaki ağlar daha küçük seçilmiştir.
5. Modelde ayaktan kazılan kömür 6 ağa, tavadan göçertilten kömür ise 5 ağa bölünmüştür.
6. Model oluşturulurken dikdörtgen ve kare şekilli tuğla (brick) elemanlar kullanılmıştır. Modelde 81006 eleman ve 87467 düğüm noktası (grid) bulunmaktadır (Şekil 3).

Eynez Yeraltı Ocağı Mekanize panosundaki ayağın genişliği yaklaşık 4 metre, yüksekliği de 3,2 metredir. Şekil 3'de görüleceği gibi modelde tavan ve taban yolları açıldıktan sonra, bu iki yolu birbirine bağlayan ayak da modele yerleştirilmiştir.



#### 4 EYNEZ YERALTI OCAĞI MEKANİZE PANOSU ÇEVRESİNDE OLUŞAN GERİLMELERİN MODELLEME SONUÇLARI

Ayak yaklaşık olarak 170 metre ilerledikten sonra düşey ikincil gerilme değerindeki değişimlerin modellenmesi yapılmış ve Şekil 5’de, düşey yönde (“z” yönü) oluşan gerilmeler verilmiştir. Şeklin solunda renklerle ifade edilen gerilme değerleri MPa’dır. Burada  $1.39e+007$  değeri,  $13,9$  MPa’ı değerini göstermektedir.

Tavan kömürünün tek bir kalkan tahkimat ünitesi olduğundan akmasının modellenmesi yapılmış ve modelleme sonucunda, ayak ilerleme yönüne paralel (A-A’ kesiti) olarak meydana gelen akma, Şekil 6’da meydana gelen akma verilmektedir. Yukarıdaki şekiller üzerindeki her renk, malzemenin düşey yöndeki yer değiştirme değerini göstermektedir. Şeklin yanında verilen ölçekte, bu değerler sayısal olarak görülmektedir. Bu ölçekte birimler, metre cinsinden verilmektedir. Negatif değerler yer değiştirmenin aşağı yönde olduğunu göstermektedir. Örneğin “ $-2.00e0.01$ ” değeri, bu değere denk gelen her bölgede aşağı yönde 20 cm yer değiştirmenin olduğunu göstermektedir.

Yapılan modelleme çalışması sonucunda 130 cm yüksekliğinde ve 110 cm genişliğinde bir çekme elipsoidi

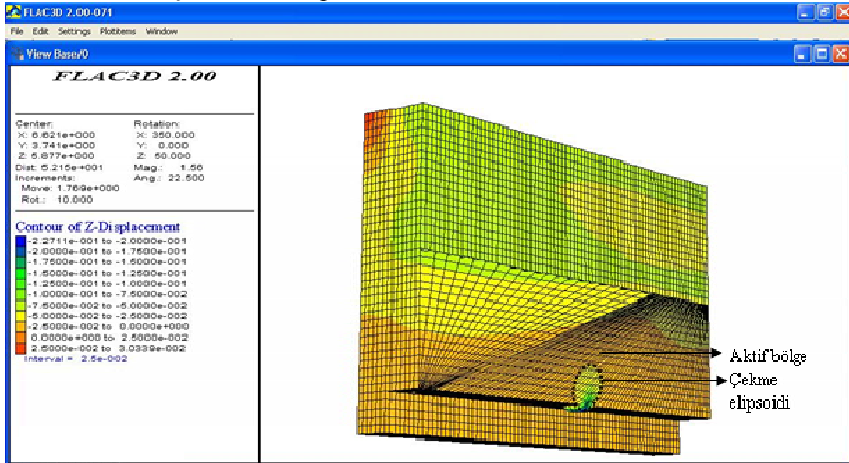
oluşmaktadır. Şekilde görüleceği üzere oluk ağzında yer değiştirme, yani kömür akma hızı fazla olurken, bu bölgenin çevresinde yer değiştirme, yani kömürün aşağı doğru hareketi, daha az olmaktadır. Yer değiştirmenin fazla olduğu bölge hareket elipsoidi, yer değiştirmenin az da olsa devam ettiği bölge ise aktif bölge olarak adlandırılmaktadır. Oluk ağzından kömür alındıkça aktif bölge içerisindeki kömür, hareket elipsoidinin içerisine girmekte ve tavan kömürü bitene kadar da akma işlemi bu şekilde devam etmektedir.

Tavan kömürünün akma sırasında nasıl bir yenilme gösterdiği, Şekil 7’de verilmektedir. Şekilde belirteçte verilen terimler ile ilgili açıklama aşağıda yapılmaktadır:

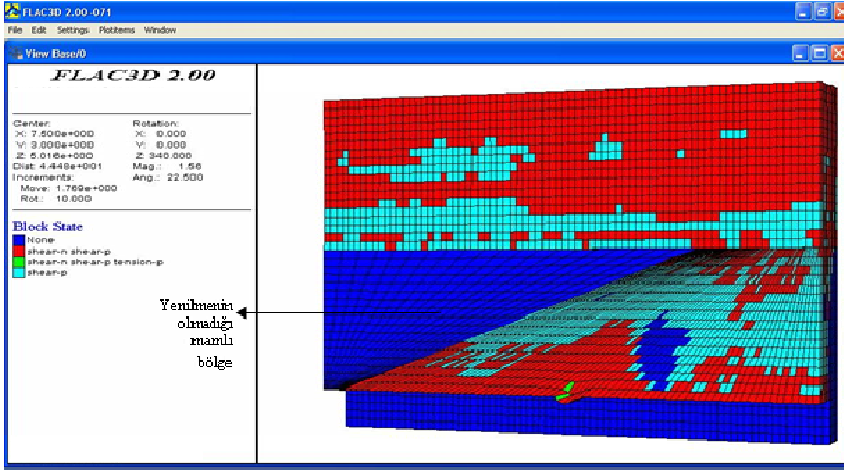
*None*: Herhangi bir yenilmenin olmadığını,  
*Shear n*: Makaslama kuvvetlerinin etkisi ile yenilmenin olduğunu ve yenilmenin hala devam ettiğini;

*Shear p*: Makaslama kuvvetinin etkisi ile yenilmenin olduğunu, fakat kuvvetlerin azalmasından dolayı yenilmenin durduğunu;

*Tension p*: Çekme kuvvetinin etkisi ile yenilmenin olduğunu, fakat kuvvetlerin azalması nedeniyle yenilmenin durduğunu göstermektedir.



Şekil 6. Tavan kömürünün tek bir kalkan tahkimat ünitesi olduğundan akmasının modellenmesi



Şekil 7. Tavan kömürünün tahkimattan akması sırasında tavan kömüründe meydana gelen yenilme

Modelleme sonucunda, tavan kömürünün, tek kalkan tahkimat ünitesi olduğundan akıtılmaya başlanmasıyla, kalkan tahkimat ünitesi olduğu üzerindeki tavan kömüründe yenilmenin tam olarak gerçekleştiği (*shear-n&tension-p*), yani tavan kömürünün düzgün tane boyu dağılımında kırıldığı ve kalkan tahkimat ünitesi olduğundan aktığı görülmektedir (Şekil 7). Fakat bu modelde marnlı bölgeye bakıldığı zaman, bu bölgede herhangi bir yenilmenin olmadığı görülmektedir. Bu bölgede kömür akıtılıp alınırken üstteki marnlı bölgede bir yenilme olmayacağı için marn kütle halinde yerinde kalacak veya bir müddet sonra çok iri bloklar halinde yenilerek ayağa ve dolayısıyla da ayak tahkimatlarına zarar verecektir.

## 5 SONUÇLAR VE ÖNERİLER

Eynez Yeraltı Ocağı Mekanize panosunda tavan kömürü akıtılırken, kömürün ayaktaki kalkan tahkimat ünitesi olduğundan (80 x 150 cm) akabilecek boyutta kırıldığı belirlenmiştir. Fakat marnlı bölgenin hem yerinde yapılan incelemelerde hem de modelleme çalışmalarında yenilmediği görülmüştür. Bu da bu bölgede çok geniş boşlukların oluşmasına ve göçmeme problemlerine neden olmaktadır. Ocakta böyle bir problemle karşılaşıldığında, göçmeyen bölgelerde patlatma yapılmaktadır. Patlatma işlemi hem ayağın durmasına neden olmakta, dolayısıyla da çalışma verimi azaltmakta, ayrıca patlatılan marnın çok ince taneli olmasına ve

kömürün içine karışarak kömürün seyrelmesine neden olmaktadır. Bu sebeple ayak tavan yolu açılırken aynı zamanda marnlı bölgede çatlaklandırma patlatması yapılması hem üretimin hızlanmasını sağlayacak hem de kömüre marnın karışmasını engelleyecektir. Bundan sonraki dönemde, bu bölge için ayrıık elemanlar prensibine göre çalışan PFC3D programında, tavan kömürünün akmasının modellenmesi yapılarak, tavan kömürünün seyrelmesi ve kömürün göçüğe karışma koşulları daha belirgin bir biçimde belirlenecektir.

## TEŞEKKÜR

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## Simulation Experiments based on U-type Mine Ventilation System of Sallent mine

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**ABSTRACT** It is quite difficult to research ventilation systems, and predict process behavior under different conditions and constraints in fieldwork. Therefore, a similarity ventilation model based on geometric and dynamic similarity principles is established in the laboratory. The area ratio  $K_s$  of roadway between the real mine and the model is 1:5000. By simulating a U-type ventilation network, the wind quantity distribution and speed in different sections were surveyed and compared with the data obtained from Sallent mine. Results show that the air in roadway is in turbulent flow state, which means the kinematical and dynamic similarity principles are achieved. The characteristics of wind quantity distribution and velocity in Sallent mine are basically simulated by the similarity model in the laboratory. It can provide experimental evidence on the selection of the ventilation system and the optimization of the existing ventilation systems.

**Keywords:** u-type ventilation system, dynamic similarity; air quantity; similarity experiment

### 1 INTRODUCTION

Mine ventilation provides a flow of air to the underground workings of a mine of sufficient volume to dilute and remove noxious gases (Vergne & Jack, 2003). It plays a significant role in production systems, and a reasonable ventilation system exerts a long-term effect on mine safety and economic benefits (Izzet et al., 2011). However, it is quite difficult to fully observe mine ventilation systems and predict process behavior under different conditions and constraints in field work. Many scholars use different computer software to simulate mine ventilation systems, but these numerical simulations have many limitations (Shinsuke et al., 1992). Therefore, a U-sharp ventilation system model of Sallent mine based on geometric and dynamic similarity principles

is established, and the relevant experiments are carried out in the laboratory.

### 2 SIMILARITY THEORY

Simulation is the activity of producing conditions which are similar to real ones, especially in order to test something, or the conditions that are produced. It can be used when the real system cannot be engaged, because it may not be accessible, or it may be dangerous or unacceptable to engage. Simulation can also be used for ventilation system design and optimization in the area of mining industry.

#### 2.1 Flow similarity theory

One significant factor shows that two fluids are similar, so that their flow situations satisfy geometric and dynamic similarity.

Geometric similarity can be defined as two geometrical objects that have the same shape (Kundu & Cohen, 2008); therefore, their linear ratio is:

$$k_l = \frac{l_m}{l_p}$$

Where, m is the model, and p is the prototype.

Then, their circumference ratio, area ratio and volume ratio are:

$$k_U = \frac{U_m}{U_p} = k_l \quad k_A = \frac{A_m}{A_p} = k_l^2 \quad k_V = \frac{V_m}{V_p} = k_l^3$$

the corresponding relationship between their angles is:

$$\alpha_m = \alpha_p$$

and the speed ratio is:

$$k_v = \frac{v_m}{v_p} = C$$

Then,

$$v_m = \frac{l_m}{t_m}$$

$$v_p = \frac{l_p}{t_p}$$

and,

$$k_v = \frac{l_m/t_m}{l_p/t_p} = \frac{k_l}{k_t}$$

Finally, the time ratio and acceleration ratio are:

$$k_t = \frac{t_m}{t_p}$$

$$k_a = \frac{a_m}{a_p} = \frac{v_m/t_m}{v_p/t_p} = \frac{k_v}{k_t} = \frac{k_l}{k_t^2}$$

Dynamic similarity exists in geometrically similar systems, and it requires that the ratios of all forces acting on corresponding fluid particles and boundary surfaces in the model and prototype are constant (Konduri et al., 1997). Their force ratio is:

$$k_F = \frac{F_m}{F_p} = C$$

it can also be showed as:

$$k_F = k_m k_a = (k_p k_l^3) (k_l k_t^{-2}) = k_p k_l^2 k_t^{-2}$$

## 2.2 Flow similarity standard

The principles of simulation experiments are based on many similarity criterions, including Strouhal similarity criterion, Froude similarity criterion, Euler similarity criterion, Reynolds similarity criterion and Mach similarity criterion, etc.

In general, the relation between fluid motion and stress is described by fluid motion differential equations. Prototype flow and model flow are both incompressible flow, and their flow similarities are achieved on the condition that they satisfy the motion differential equation (Kundu & Cohen, 2008):

$$\frac{\beta v_{xm} + v_{xm}}{\beta t_m} \frac{\beta v_{ym} + v_{ym}}{\beta x_m} \frac{\beta v_{zm} + v_{zm}}{\beta y_m} \frac{\beta v_{xm} + v_{xm}}{\beta z_m} = f_{xm} - \frac{1}{\rho} \frac{\beta p_m + v_m \Delta v_{xm}}{\beta x_m} \quad (1)$$

$$\frac{\beta v_{xp} + v_{xp}}{\beta t_p} \frac{\beta v_{yp} + v_{yp}}{\beta x_p} \frac{\beta v_{zp} + v_{zp}}{\beta y_p} \frac{\beta v_{xp} + v_{xp}}{\beta z_p} = f_{xp} - \frac{1}{\rho} \frac{\beta p_p + v_p \Delta v_{xp}}{\beta x_p} \quad (2)$$

Put formula (1) into (2), we can get formula (3):

$$\frac{k_v}{k_t} = \frac{k_v^2}{k_l} = k_g = \frac{k_p}{k_p k_l} = \frac{k_l k_v}{k_l^2} \quad (3)$$

Formula (3) demonstrates Time-varying inertia force, Space-varying inertia force, Mass force, Pressure and Friction force, respectively. It also shows the force polygon of model flow is similar to prototype flow.

Divide formula (3) by space-varying inertia force, it can be obtained that:

$$\frac{k_l}{k_l k_v} = 1 = \frac{k_l k_g}{k_v^2} = \frac{k_p}{k_p k_v^2} = \frac{k_v}{k_l k_v} \quad (4)$$

Formula (4) manifests some constraints exists among different ratios when dynamic similarity is achieved between prototype flow and model flow. Then, further analysis indicates dynamic similarity can be described by similarity criterion.

As a matter of fact, equal flow and complete dynamic similarity cannot be met between the prototype flow and the model flow (Nicholas et al., 2001); thus, the key point in the simulation experiments is to ensure the main dynamic similarity. That is to say, if two different flow types have met dynamic similarity, their Reynolds numbers must be similar.

### 3 THE SIMILARITY EXPERIMENT PLATFORM

#### 3.1 Model design

The size of the ventilation model is based on the roadway size of Sallent mine. For the purpose of simplifying the model, the area of the model tunnels are based on the average of prototype roadway area (32m<sup>2</sup>), and the area ratio is 1:5000. Therefore, the model tunnel area is 0.0064m<sup>2</sup>.

For the model, it consists of five parts: model body (tunnel, ventilation networks and goaf), fan and its affiliated devices, noxious gases drainage system, collecting and testing system, respectively (Fig. 1). The air doors are controlled by the spherical valve, and the transformation among different types of ventilation system and the adjustment of the air quantity can be realized.

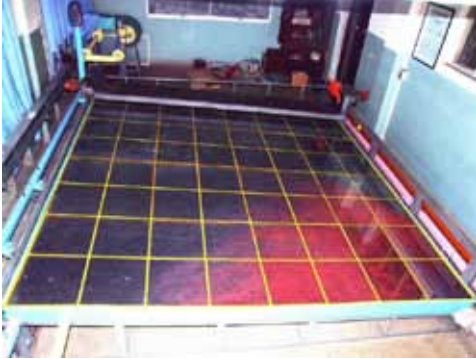


Figure 1. Air migration in similarity model of ventilation system

After several measurement and calculations, the ventilation resistance in the model is approximately 3300 Pa, and the power is 9KW. Thus, a centrifugal fan is selected as the main ventilator of the model, and it has many affiliated devices including inverted ventilation device, explosion-proof door, piping shock absorber and diffuser.

#### 3.2 Dynamic similarity and Reynolds numbers

These similarity experiments essentially depend on viscous force, pressure and inertia force, thus, Reynolds similarity criterion and

Euler similarity criterion should be satisfied as much as possible. Reynolds similarity criterion between model and prototype will be equal as long as the model's Reynolds number is in the resistance square region. That is to say, dynamic similarity could be automatically ensured when the relationship between the model and the prototype satisfies geometric similarity (Kundu & Cohen, 2008).

The critical velocity of the model in the turbulent state is as follows:

(1)The second critical value of Reynolds number in the general experience model is:  $1.0 \times 10^4 \sim 1.5 \times 10^5$ , here when  $Re = 1.0 \times 10^4$ ,

$$V = \frac{ReUv}{4S}$$

Where, V is the average wind speed in the tunnel, m/s; v is the fluid dynamic viscosity coefficient, which is related to temperature and pressure. In mine ventilation, it usually takes  $14.4 \times 10^{-6} \text{m}^2/\text{s}$ ; U is perimeter of tunnel, length and width are both 0.08m; S is tunnel's area, m<sup>2</sup>.

$$V = \frac{10000 \times 2 \times (0.08 + 0.08) \times 14.4 \times 10^{-6}}{4 \times 0.08 \times 0.08} = 1.8 \text{m/s}$$

(2)For the purpose of simplicity, Reynolds number is generally considered as a criterion of flow state in the tunnel in the practical engineering calculation, namely:

When  $Re \leq 2300$ , which indicates the fluid is in laminar flow condition; when  $Re > 2300$ , which demonstrates the fluid is in turbulent flow condition.

$$V = \frac{2300 \times 2 \times (0.08 + 0.08) \times 14.4 \times 10^{-6}}{4 \times 0.08 \times 0.08} = 0.414 \text{m/s}$$

The minimum wind speed meeting the turbulent flow condition in the similarity model must be higher than 0.414m/s, and its density scale  $K_\rho = 1$ , and the speed scale  $K_v = 1$  can be decided.

### 4 SIMILARITY EXPERIMENTS

#### 4.1 Overview of the prototype

The potassium deposit in the Catalan basin (Spain) is currently being exploited by the largest mining company (Iberpotash) in Catalunya. The company is situated in the comarca of Bages, in the municipalities of Balsareny, Sallent (Vilafrens) and Suria (Cabanasses). It is the only producer and supplier of potash fertilizers for agriculture and industry usage, and it produces over one million tons of red potash for fertilizers.

#### 4.2 Survey results in Sallent mine

The similarity experiments are based on the condition of Sallent mine, and use its ventilation data as the primary reference source. Firstly, 14 surveying ventilation points in the main intake and return entry of the ventilation system are selected through careful consideration and design (Fig. 2).

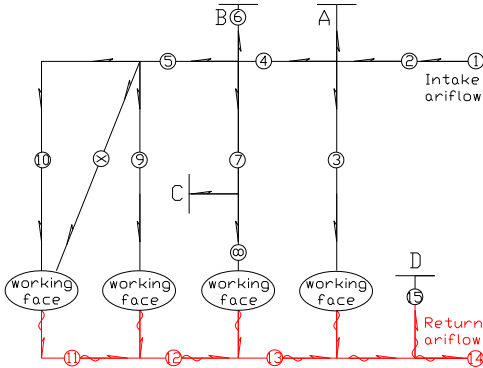


Figure 2. Simplified ventilation system in Vilafrens Mine

Data collection works are made twice as nature ventilation shows differences between summer and winter. Meanwhile, averages are adopted from multiple surveys while data is collected (Table 1 and Table 2).

Table 1. Survey results in Sallent mine, July 2011, summer

Points	Wind Speed (m/s)	Section (m <sup>2</sup> )	Flow rate (m <sup>3</sup> /s)
1	3.85	40.00	154.00
2	4.46	34.54	154.05
3	0.73	31.84	23.24
4	3.85	34.04	131.05
5	3.84	27.86	106.98
6	0.39	34.80	13.57
7	0.64	28.37	18.16
8	0.43	23.86	10.26
9	0.37	31.54	11.67
10	1.09	32.29	35.20
11	3.39	27.83	94.34
12	4.58	26.89	123.16
13	5.38	24.36	131.06
14	4.18	35.96	150.31
15	0.18	21.61	3.89

Table 2. Survey results in Results in Sallent mine, January 2012, winter

Points	Wind Speed (m/s)	Section (m <sup>2</sup> )	Flow rate (m <sup>3</sup> /s)
1	4.41	40.00	176.08
2	5.12	34.54	176.84
3	0.84	31.84	26.75
4	4.52	34.04	153.77
5	4.71	27.86	131.22
6	0.48	34.80	16.70
7	0.70	28.37	19.86
8	0.63	23.86	15.03
9	0.45	31.54	14.19
10	1.85	32.29	59.74
11	3.63	27.83	101.02
12	6.21	26.89	166.99
13	6.89	24.36	167.84
14	4.76	35.96	171.16
15	0.33	21.61	7.13

#### 4.3 Measurements results in the model

In the similarity experiments, the air is used as flow medium in the model, and its flow state is regarded as steady flow. Geometric similarity and dynamic similarity are achieved between the prototype and the model. Different valves in the model are opened and closed while the experiments are performed, which makes the model

ventilation system as much similar as that in Sallent mine (Fig. 3).

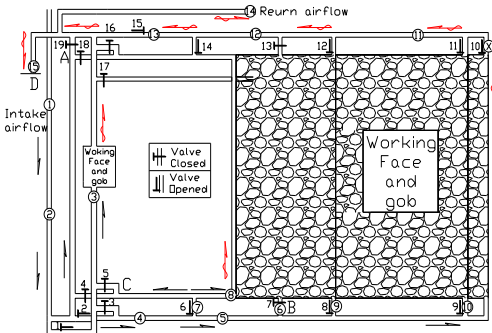


Figure 3. Ventilation model system and its valve control

Similarly, the surveys of wind quantity and wind speed are conducted twice, both in summer and winter (Table. 3 and Table 4).

Table 3. Measurement results in the model, summer, June 2011

Points	Section (m <sup>2</sup> )	Velocity (m/s)	Flow (m <sup>3</sup> /s)
1	0.0064	2.61	0.0167
2	0.0064	2.59	0.0166
3	0.0064	0.42	0.0027
4	0.0064	2.16	0.0138
5	0.0064	1.79	0.0114
6	0.0064	0.45	0.0029
7	0.0064	0.43	0.0028
8	0.0064	0.41	0.0026
9	0.0064	0.41	0.0025
10	0.0064	0.71	0.0045
11	0.0064	1.39	0.0089
12	0.0064	1.42	0.0091
13	0.0064	1.82	0.0116
14	0.0064	2.28	0.0146
15	0.0064	0.41	0.0026

Table 4. Measurement results in the model, winter, January 2012

Points	Section (m <sup>2</sup> )	Velocity (m/s)	Flow (m <sup>3</sup> /s)
1	0.0064	2.99	0.0192
2	0.0064	2.98	0.0191
3	0.0064	0.47	0.0031
4	0.0064	2.47	0.0158
5	0.0064	2.19	0.0141
6	0.0064	0.55	0.0035
7	0.0064	0.45	0.0029
8	0.0064	0.43	0.0028
9	0.0064	0.42	0.0027
10	0.0064	1.01	0.0065
11	0.0064	1.49	0.0096
12	0.0064	1.91	0.0123
13	0.0064	2.36	0.0151
14	0.0064	2.68	0.0172
15	0.0064	0.42	0.0027

#### 4.4 Comparisons between prototype and the similarity model

Fig. 4 demonstrates the measurement results of the wind quantity in Sallent mine and in the similarity model at different ventilation points in summer and winter.

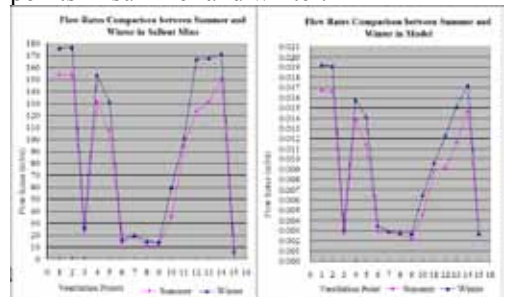


Figure 4. Comparison of air quantities between Sallent mine and the similarity model

It can be seen in fig. 4 that the air quantities in main intake and return tunnel are larger than that in other sections in Sallent Mine, being the maximum 175m<sup>3</sup>/s, and the same situation appears in the similarity model. This indicates the air distribution rule in the experiments is essentially the same as the one of Sallent mine. Besides, the air quantities in winter are higher than that in summer at the same

ventilation points both in Sallent mine and the similarity model because natural wind pressure in winter exerts a positive influence on ventilation network.

Fig. 5 demonstrates the measurement results of wind speed in Sallent mine and in the similarity model at different ventilation points in summer and winter.

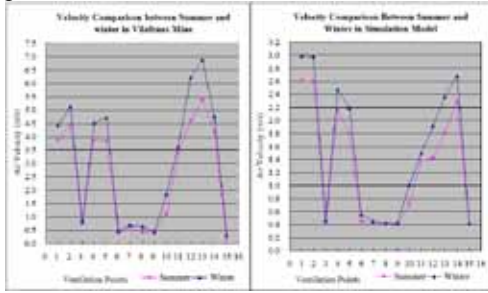


Figure 5. Comparison of wind speeds between Sallent mine and the similarity model

It can be seen in Fig. 5 that the wind speeds in main tunnels of intake and return is higher than that in other sections both in Sallent Mine and the similarity model. Similarly, the wind speeds in winter is higher than that in summer at the same ventilation points both in Sallent mine and the similarity model due to the effect of natural ventilation.

In addition, Fig.5 also shows that the minimum wind speed in the model is 0.41m/s, this indicates that the airflow in the model reaches turbulent flow state; thus, kinematics and dynamic similarity are achieved automatically.

However, under the circumstance where air quantity is identical, wind speeds in return tunnels are slightly higher than those in intake tunnels in Sallent mine while in return tunnels are slightly lower than those in intake tunnels in the model. There are two reasons for this. First, all the drift sectional areas are the same in the model. Second, many of the drift sections in the similarity model are right-angled, which results into excessive frictional restriction in the return tunnels.

## 5 CONCLUSION

For the purpose of conducting on-site research in mine ventilation systems, a similarity model based on the principles of geometric and dynamic similarity is established in the laboratory. By using a U-type ventilation system, the wind quantity and speed in different sections of the model are measured and compared with the practical data obtained from Sallent mine.

The results indicate that the wind quantities in the main tunnel of intake and return are larger than that in other sections, which is in accordance with the air distribution rule of Sallent mine.

Besides, the wind quantities and speeds in winter are higher than that in summer at the same ventilation points both in Sallent mine and the similarity model due to the effect of natural ventilation. Last but not least, with the minimum wind velocity being 0.41m/s, the airflow in the model is in turbulent state, and kinematics similarity and dynamic similarity are achieved.

Overall, the rules of the air quantity distribution and the characteristics of wind velocity are essentially the same as the actual situation in Sallent mine, which indicates the establishment of the similarity model is successful. The following experiments provide benefits for Sallent mine because the simulation model can be first adjusted before adjusting the real ventilation network. Then the decision of whether or not to apply it to the real ventilation system in Sallent mine can be made based on the experimental results.

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## *Implementation of Mine Control at Çayeli Underground Copper Mine (CBI)*

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**ABSTRACT** Mine Control was implemented at CBI in November 2010 to monitor underground activity in real-time to improve production efficiency and safety of the mining operation.

CBI is a complex underground mining operation targeting annual production of 1.2Mtonnes per annum from more than 160 stopes in challenging ground conditions. Achieving production consistency has always presented issues at CBI. With the introduction of this project, three of the critical factors for a successful mining operation, planning, communication and coordination are focused into the Mine Control office.

Mine Control fundamentally changes the mode of managing the mining activity underground. Some of the duties of the traditional Underground Shift Supervisor are reallocated to the new position of Shift Co-ordinator who operates Mine Control located on the surface. The traditional underground Shift Supervisors has more time to focus on quality, standards and safety of the personnel underground.

The Shift Co-ordinators are experienced mining supervisors empowered to make decisions and control the mine from the Mine Control office. PITRAM software is utilised to capture real-time information on equipment & personnel location, activity and status so that the Shift Co-ordinator can make informed decisions and allocate resources according to a prearranged plan and changing priorities.

PITRAM real-time data acquisition, reporting and management software can automatically generate shift, daily, weekly and monthly production and activity reports. It also captures equipment status including delays, breakdowns and maintenance, allowing automatic calculation of a range of maintenance KPI's (Key Performance Indicator) from availability to Mean Time Between Failure. This system is used at 36 mine sites throughout the world while CBI is the first application of this technology in Europe. This paper will describe the implementation, operation and benefits derived from the introduction of Mine Control at CBI.

### **1 INTRODUCTION**

CBI (Çayeli Bakır İşletmeleri) is an underground copper and zinc mine located in the Black Sea region of Turkey, 10km inland of the coastal city of Çayeli in the Rize district.

Production commenced in 1994 and has gradually increased 1.2M tonnes per annum budgeted for the past 2 years. A total of 16.5

million tonnes have been produced to the end of December 2012 at an average grade of 4.03% Cu and 6.01% Zn. Life of the mine is currently estimated to be ended in 2018. Development has been stable at around 3.5 to 4.0 km per year however the demand for rehabilitation ground support has been increasing as the mine expands and ages and as production rates have increased.

After 2014 the production rate begins to decline due to depletion of reserves and less independent mining areas being available, copper grade also declines therefore reducing future cash flow assuming constant metal prices.

The need to improve current mining consistency to ensure the 1.2M tpa budget mining rate is achieved and to improve future efficiency were 2 of the main justifications for the implementation of Mine Control at CBI.

## 2 BACKGROUND

### 2.1 Structure of the Mining Crews

Since 2005 CBI has divided the underground workforce into 3 mining crews (Table No.1) each performing separate but overlapping functions. This configuration had advantages while developing the footwall ramp and other priority waste headings but had inherent inefficiencies and restrictions on sharing equipment and manpower between crews. Each crew had a Mine Supervisor allocating tasks and priorities for his individual resources. Often the excuses for one crew not achieving its goals were the other crew did not provide the required service.

Table 1. Previous Mining Crews

<b>MINING CREW</b>	<b>Main Activity</b>	<b>Main Equipment</b>	<b>Area</b>
<b>Foot Wall Ramp (FWR)</b>	Development and Diamond Drilling	Jumbo, Blasting, Bolter, Shotcrete, Diamond Drill	Below 800 Lev
<b>Ground Support (Rehab)</b>	Rehab ground support, barricades and services	Bolter, Shotcrete Cable Bolt Drill, Platforms	Above 800 Level
<b>Production</b>	Development Stope drilling, blasting, muck and trucking	Jumbo Production Drills, Stope Blasting, LHD, Trucks	Dev above 800 Lev Production full mine

For example development might be delayed because Production had not mucked the FWR waste headings or the Rehab Crew have not shotcreted and bolted the Production Crew headings.

Combining the mining crews into 2 groups occurred in August 2010, Table No.2. This rationalisation of personnel and equipment brought all of the like equipment and skills into the 2 larger functionalised crews so that resources could be used more effectively. Now all of the ground support equipment and personnel are in 1 group and all production and blasting resources are in the other.

Table 2. Combined Mining Crews

<b>MINING CREW</b>	<b>Main Activity</b>	<b>Main Equipment</b>	<b>Area</b>
<b>Ground Support (Rehab)</b>	Rehab ground support, barricades face drilling and services	Bolter, Shotcrete Jumbo, Platforms	All Levels
<b>Production</b>	Stope drilling, All blasting, muck and trucking, Diamond Drill	Cable Bolt Drill Production n Drills, Blasting, LHD, Trucks, Diamond Drills	All Levels

Combining the mining crews had some disadvantages in that the single underground Supervisor had more men to supervise with typically 20 and 23 men per shift for the Ground Support and Production Crews respectively. Each crew is now distributed throughout the entire mine which encompasses more than 20km of access ramp and level development over 30 Levels of which more than 1/2 remain in active production.

An advantage of combining the crews was that experienced Mine Supervisors were liberated from underground duties and could be redeployed into Mine Control.

## 2.2 Mine Control

Mine Control is an office set up on the surface to monitor and control all the mining activity occurring underground. It is manned 24 hours per day by an experienced Mining Supervisor, called the Shift Coordinator.

The Shift Coordinator uses constant radio communication with the Shift Supervisors and the workforce underground to maintain a “real time” record of the location of personnel and mobile equipment plus the status of all activity in the mine. This status information is entered into a computerised software solution called Pitram to give the Shift Coordinator a visual representation of the status of all work locations, the status of equipment and the location of the personnel. The Mine Control office has monitoring and control of critical mine functions such as ventilation including blast fumes and gases, micro-seismic monitoring, pumping and access to hoisting, paste filling and other data. In addition, all mining forecasts, schedules, level plans, design information are readily accessible in the office. A weekly plan outlining the strategy and sequence for the critical activities is a key requirement for the Shift Coordinator. (Ballantyne R., 2010)

Planning Engineers, Mine Captains and other stakeholders gives the Shift Coordinator guidance on strategy and priority and assists in the tactical decision making during the shift.

The Shift Coordinator in Mine Control is empowered to make decisions and allocate all of the resources underground but can be overridden by the Underground Supervisor on safety consideration.

The Shift Coordinator can be expected to make better quality and more informed decisions than the traditional underground Shift Supervisor because he has:-

- been selected because he is an experienced Supervisor with knowledge of the mine
- up-to-date status information from underground available in Pitram
- all of the relevant plans and documents that he requires

- monitoring computers displaying critical mine functions
- schedules and priority lists
- radio communication access to all personnel underground
- phone and email communication

Accurate information available for better decision making will lead to improved efficiency of the underground resources due to reduced delay.

Real time data capture in Pitram allows for in shift validations and production adjustments in order to reach targets set for the shift, similarly the in-shift reporting will ensure the whole mining operation has access to live production data.

The establishment of Mine Control relieves the underground Shift Supervisor of many of the traditional tasks associated with tracking, allocating and reassigning work priorities. This assists the 2 remaining Shift Supervisors underground and provides them the time required to spend with the extra personnel in the larger work group and provide more focus on safety, standards and quality. Increased focus by the Shift Supervisors on standards and quality of workmanship result in less rework and productivity gains.

## 2.3 Pitram

Pitram is a software package developed by Micromine specifically to perform the function of monitoring and controlling the underground mining environment. Pitram, manages and coordinates all functions within a mine site; mine development, production, equipment and maintenance. It tracks and reconciles the delivery and quality of the ore. As the status of events are updated by the Shift Coordinator the software maintains the information in a database which is used to automatically generate shift, daily, weekly, monthly or ad-hock reports for the production physicals, key KPI's, monitored consumables and maintenance history of the equipment. Production and activity reports can be viewed in real time by key personnel to monitor the location and status of all

equipment and personnel. (Ballantyne R. and Ascott B, 2008)

### 3 IMPLEMENTATION

The implementation of Mine Control and Pitram at CBI was a major cultural change to the method of operating and managing the underground operation and the CBI Leadership Team recognised that effective management of change would be a crucial factor for the success of the project.

The project was implemented in stages with two working groups established to oversee the implementation of the project and change management.

#### 3.1 Phased Approach

The Mine Control project has been set up in a number of stages to reduce the impact of sudden change to ensure a level of stability so as not to adversely affect production or safety. The first move was to combine the mining crews to functionalise resources and liberate the experienced Supervisors from underground to man Mine Control.

Implementation of Pitram is being undertaken in 3 phases, they are:-

- Phase 1:
  - “Basic” Pitram Voice solution
  - Establish Control Room Environment
  - Turkish translation of solution
- Phase 2:
  - Pitram Grade Control
  - Pitram Shift Planner
- Phase 3:
  - SAP (Systems Analysis and Program Development) Integration - Maintenance
  - SAP Integration - Human Resources
  - SAP Integration - Accounting

At the time of writing this paper, September 2012, Pitram phase 2 is fully

implemented with all shift, daily, weekly and monthly reports being automatically generated by Pitram. In excess of 5 mining and maintenance spreadsheets and reports have been eliminated and the former MOP (Mine Operation Program) into which the Shift Supervisors used to enter the shift data has been made redundant except for time sheet preparation.

Phase 2 is partially implemented with additional customisation of the Shift Planner and Grade Control modules required to streamline their effectiveness.

Phase 3 interfaces are being developed and trialed.

#### 3.2 Steering Committee

A steering committee was formed comprising members of the CBI Lead Team to overview the implementation strategy, resourcing and to focus on the human or social side of the project. Obtaining buy in, ownership and overcoming resistance to change were identified as the greatest risks to the success of the project.

For the project to be effective a quick uptake by all underground mining and maintenance personnel would be required as every operator underground is required to radio through specific information accurately to Mine Control.

Several initiatives were implemented to inform the workforce of the importance of the project to the future of CBI and to provide information. These included:-

- Promoting a slogan “*Mine Control – Investing in our future*”
- Demonstrating management commitment to the project
- Dinner with workers at which the Managing Director & Inmet Vice President spoke about the expectations of the project
- A special edition of the site magazine dedicated to the Mine Control project

#### 3.3 Project Team

The Mine Control Project Team comprised a cross section of mining and technical staff

who met on a weekly basis and had a regular webcast conference with Micromine to track progress on the system development and project status against the project schedule. This team was mainly focused on the technical aspects of Mine Control office renovation, software development, hardware purchase and installation, training and project administration.

Once CBI's customised Pitram solution had been developed and translated to Turkish the team coordinated the onsite training and implementation. The first step was the preparation of "Call Cards" which specify the details and timing for each equipment operator to radio Mine Control. The next step involved intensive training for the Shift Coordinators who would man Mine Control and input the data into Pitram.

All mining and maintenance employees were trained in the information and timing that is required to be radioed to the Shift Coordinators in Mine Control.

Shaft, Paste Fill / Shotcrete Batch Plant operators were also trained to input specific data from their areas of responsibility into Pitram to ensure that all mining related activity was entered without overloading the Shift Coordinators and providing them with more time for planning and coordination of the underground activities.

Staff in Mining and Maintenance were trained how to monitor the live activity from their office PC's and how to generate reports from Pitram.

Training for Phase 1 commenced in late November 2010 and continued during the first week of December. An early objective of the project was to be collecting all data accurately by 31<sup>st</sup> December 2010 to ensure that the 2011 data set would be complete and accurate from the 1<sup>st</sup> of January, this was achieved.

#### 4 BENEFITS

One of the primary objectives of implementing Mine Control with Pitram was to improve production consistency and operational efficiency underground.

#### 4.1 Efficiency Gains

A measure that has been adopted at CBI to determine any increase in efficiency as a result of the implementation of Mine Control is described below.

Seven of the key activities undertaken underground have been monitored on a weekly basis and compared to standard targets as presented in Table No.3 below.

Table 3. Performance of key activities pre and post Mine Control implementation

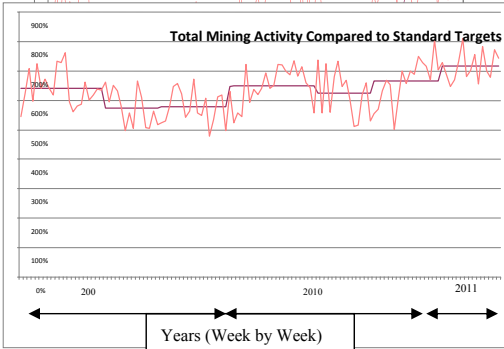
Activity	Units	Target	Average Pre Mine Control	Average Post Mine Control	% Increase
Development	m	77	90 %	99 %	10 %
Shotcrete	m <sup>3</sup>	700	82 %	98 %	19 %
Rock Bolting	each	1,450	95 %	120 %	26 %
Cable Bolting	each	2,480	102 %	114 %	12%
Production Drilling	m	150	88 %	98 %	11 %
Ore Production	tonnes	2,3077	96 %	97 %	1 %
Backfilling	m <sup>3</sup>	6,912	91 %	93 %	2%
Total			753 %	821 %	9 %

Performance of these 7 activities is plotted on a weekly basis from week No.1 2009 to week No.15 2011. During this period the resources available to perform the work has changed little with regard to personnel and equipment so any change in total activity is due to changing efficiency. Reduced total activity is evident during the summer periods due to high annual leave.

During the 119 week period reviewed in the graph the budget quantities varies slightly on an annual and monthly basis, however for this comparison the same average target has been used for each activity for every week. If all 7 activities are performing at 100% of their standard target rate then a score of 700% would be achieved for that week.

The trend in Figure 1 indicates that there has been an increase of 9% since the

implementation of Mine Control compared to the 48 weeks of 2010 immediately before implementation. During the first quarter of 2009 total activity was scoring high with development and cable bolting performing well above target, however ore production was

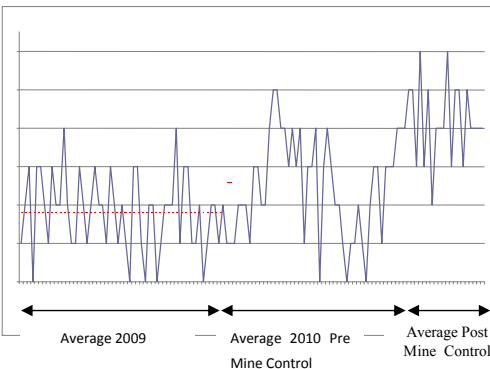


### 4.2 Improvement in Consistency

One of the difficulties at CBI is achieving consistency in all of the key activities. It is easy to focus attention in 1 or 2 activities and produce outstanding results in those activities but it often results in reduced focus or resources in another activity. The objective for sustainable long term efficient production is to maintain consistent target rates for all activities.

Figure 2 demonstrates that there has been an increase in the consistency of performance with an increasing number of activities achieving 100% of the weekly target rate.

Figure 2: Number of underground activities achieving above 100% of target each week



We are now achieving target more consistently for more activities but there is still scope for continued improvement on consistency as we are yet to achieve above 100% of target for all 7 monitored activities in a week.

### 4.3 Other Benefits

The establishment of mine control has created a focal point for the elements necessary for a safe and efficient operation that is planning, communication and coordination.

In addition improving efficiency and consistency of the mining operation there has been a number of other benefits including:-

- The Shift Coordinator provides a single point of contact for mine enquiries and decision making
- Complete set of accurate and up to date mining & equipment data in Pitram
- Reduced delays in the underground due to not deploying resources to locations that are not ready.
- Better utilisation of resources.
- More efficient transfer of information across shift change and crew rotation.
- Improved communication and in shift priority setting with maintenance
- Improved communication and coordination in the event of an underground emergency
- Automated report generation
  - Reduced turnaround time for reports
  - report generation less labour intensive
- Availability of real time (in shift) reporting on current shift data
- Monitoring of production status or equipment availability from each PC

In the 4 months since Pitram training has been completed the Mining and Maintenance Departments & crews have gained a much

better understanding of Mine Control and the Pitram system's capabilities. The more disciplined monitoring of equipment movements within mine has allowed improved management opportunities. The Pitram data base is expanding and is now beginning to be used for Continuous Improvement projects and other research to identify bottlenecks and inefficiencies in the operation.

## 5 FUTURE APPLICATIONS

Finalising customised solutions for CBI Shift Planner and Grade Control modules will deliver significant enhancements to the short interval planning and production monitoring by the Shift Coordinator and the ability to track CBI's 8 different ore types from stope to mill.

Implementation of phase 3 will automate data transfer between PITRAM and SAP and will streamline interfaces between mining and other departments such as HR (Human Resources) & Training, Accounting (mainly timesheet) and Maintenance.

Emergency response, including activation of stench gas, for the underground will be coordinated from Mine Control rather than surface security. With Mine Controls access to real time data, the location of all workers and equipment is known and can be rapidly coordinated.

The main focus over coming months will be to enhance the Shift Coordinators ability to better plan, prioritise and control the activities underground to further improve operational efficiency and consistency. Shift Planner will be a key element in this and is expected to produce another step change in efficiency and consistency of underground activity.

Future integration to 3<sup>rd</sup> party solutions for items such as personnel and equipment tracking system under investigation. With tracking system we are aiming to decrease radio traffic at Mine control room to focus on safety, automation and mine management issues.

## 6 CONCLUSIONS

Establishment of the 2 teams, the Steering Committee and Project Team, at the commencement of the project facilitated a smooth acceptance of the concept of Mine Control and implementation of Pitram.

The frequency of radio voice traffic coming into Mine Control for the Shift Coordinator to input is an emerging issue that will need to be addressed as we implement the short interval control Shift Planner module to ensure that the Shift Coordinator has sufficient time to plan and correctly prioritise activities.

Selecting experienced mining Supervisors who were computer literate with good knowledge of the mine and who were respected by the underground workforce accelerated the adoption of Mine Control and Pitram. This ensured that the shift Coordinators had the authority to control the underground mining activities from their surface office.

## ACKNOWLEDGEMENTS

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# Study of The Surveying Errors in Shaft Plumbing of a Potash Mine

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**ABSTRACT** The proper transmission of the orientation and cartographic system between the outer and inner part of an underground mining activity poses a difficult challenge if the shaft is over 200 meters in depth. The required accuracy in these situations depends on the requirements of the mining activity, but it is usual to require a precision of under a minute. If an appropriate procedure isn't followed then it is difficult to achieve targets of this magnitude. This paper provides an accurate description of the methodology used in the potash mine "Cabanasses" in Suria (Spain), property of Iberpotash SA. Here it is described the error factors that have affected the performed measurements. These have been deduced from prior studies and it is only explained those which have had a practical influence on the mine.

**Keywords:** Shaft plumbing, Ventilation, Plumb oscillation, Gravity effect

## 1 INTRODUCTION

The mining company Iberpotash, SA has an underground potash mine called "Cabanasses" in Suria (Spain). The mine is located 700 meters below the surface and is accessed via two vertical shafts of 4.5 and 5.5 meters in diameter. The first shaft, Pou2, is used for staff access and ventilation input. The second shaft, Pou3, is used to extract the ore and for ventilation output. Both shafts have a separation of approximately 100 meters.

With the objective of increasing the productivity of the mine and improving the ventilation system, work commenced September 2012 to the dig a ramp between

the inner part of the mine with the surface. The features of the ramp are an 18% of slope and a section of 45m<sup>2</sup>.

During 2011 and 2012, before work commenced on the ramp, several surveys were conducted to connect the orientation and cartographic system from the surface to the underground point Q where the ramps will be started. Point Q is approximately 3 km away from the bottom of the access shafts, and will be connected by a ramp of 4,7 kilometres broken up by 4 parts to a I point located in the surface. Because of this, it is necessary a high precision in the surveying methods to guarantee that the offset at 3.6 km is within the fixed tolerance.

This paper focuses on describing the performed works in the orientation and cartographic transmission through the vertical shafts; and the error factors in this specific works.

## 2 METODOLOGY

Taking into consideration that the mines have 2 vertical shafts it is used the shaft plumbing method through two shafts. This method consists in lowering a plumb for each shaft (P1 for shaft Pou3 and P2 for shaft Pou2) and connecting them through a surveying traverse. Previously, the planimetric coordinates of the plumbs in the surface were deduced.

When the first station, station A, inside of the mine was done, there was not any known coordinates inside of the mine, so a random orientation was taken. All the calculations of the traverse were done by this false orientation, getting sight orientation (A'-B', B'-C' i C'-P2') and a coordinates points (B', C' and P2') that were not true. By difference of coordinates the false orientation between P1 and P2' was obtained.

The difference between this orientation and the true orientation (P1-P2) deduced from the plumb coordinates in the surface is called Disorientation and is used to compensate all the measurements done in the underground traverse between the two plumbs.

After this the true coordinates of the underground traverse (A, B, C i P2) was calculated. The true coordinates of P2, if any errors did not occur, would be equal to the computed coordinates in the surface.

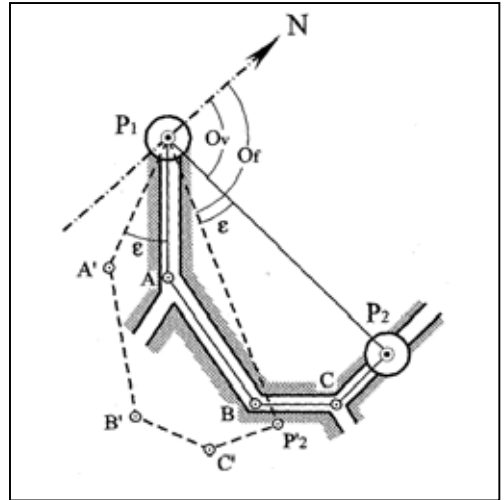


Figure 1. Outline of the surveying works of two shaft plumbing method.

The surveying works in the surface, performed before the underground works, were done with a closed traverse starting from a known coordinate point with a known orientation. With this traverse, the coordinates of each plumb (one for shaft) were read from two different stations. This allowed obtaining the plumb coordinates twice.

In the shaft Pou2 (entry of ventilation) two different measurements were taken: one with the top of the shaft opened and one with the top of shaft capped with a canvas with a hole in the middle to permit the entry of the plumb. This was done to minimize the effect of the ventilation and deduce the effect that it has in the vertical area of the plumbs. Therefore, there were double measurements taken in both underground and in the surface. In the surface it allowed to obtain the surface coordinates with the shaft opened and capped. The difference was only of one millimetre, demonstrating that there isn't a significant difference in the surface after capping the shaft.

At the bottom of the shaft Pou2 (700 m depth) the difference detected between the measurements was 1 cm after 8 hours of stopping the ventilation fans and having the

shaft capped. The last measurements were the ones used for the calculations.

The specified measurements of shaft Pou2 were accomplished on the 10th July, 2011. On November 27th, 2011, the measurements to plumb P1 located in the shaft Pou3 were taken. Similar to the acquired measurements in July it was deemed necessary to cap the entrance of shaft Pou2, and it was only possible to tap the 20% of the entrance of shaft Pou3.

In the inside of the mine it was done the part of the closed traverse planed between the shafts Pou2 and Pou3. With this part of the traverse it was done 2 sets of measurements towards the plumb P1 of the shaft Pou3. The first set consisted in 8 horizontal angle and distance measurements to the plumb P1 applying Bessel. These 8 measurements were done between 13:30 and 14:00 pm, 7 hours after stopping the ventilations fans and about 4 hours later the shafts were clapped.

The second set was identical to the first set but they were measured between 14:45 and 15:15 pm.

Once the data obtained was processed it was observed that the plumb P1 was displaced by 1.2cm from an average of the first and second measurement sets. Moreover, it was proved that this displacement could be attributing to the remainder of the mining ventilation.

The measurements picked for the definitive calculations were the average of the second set, because the ventilation slightly decreased and so the plumb was closer to his vertical position.

During the underground plumbs measurements the bottom shaft ventilation velocity was also measured. These measurements were done before and after clapping the shafts.

With the aim of increasing the stabilization of the plumbs an anti-rotation steel wire rope of 3 mm of diameter and a plumb weight of 65 kg were used. After descending the plumbs they were introduced in a viscous oil tank of 1 m<sup>3</sup>. The plumb weight was deduced from the equation

$10+0,08 \cdot L$  (Taton, 1966), where L is the depth of the shaft (680 in our both shafts).

In the following section errors obtained in this part of the works will be explained.

### 3 ERROR FACTORS IN THE SHAFTS PLUMBING METHOD

During the different operations to perform the surveying connection between the outer and inner part of a mine the following errors can be found:

- ms: errors in the exterior works produced during the obtaining of the orientation and coordinates of the plumb in the surface
- mp: errors in the verticality of the plumbs
- mb: errors in the underground traverse to connect both plumbs

Therefore, the error in the surveying connection can be expressed as follow (1):

$$m_{\alpha} = \sqrt{m_s^2 + m_p^2 + m_b^2} \quad (1)$$

Following we can see the characteristics of the outside closed traverse, the errors obtained in the closure of it and the maximum difference of the plumb coordinates obtained from two different points of the traverse.

- Total length: 436,374 m
- N° of legs= 5
- Angular error= 28°
- Error in X= 0,004 m
- Error in Y= -0,002 m
- Maximum error between the measurements of the plumbs through two different points = 1mm

In Figure 1 there is an outline of the outside traverse. It was started in the point Base C1 and orientated through the point Base C2, both points of known coordinates. Those coordinates were obtained through high precision GPS with dual frequency receiver, with differential and post processing system. The surface coordinates of P1 P2 in the system UTM-ED50 (31):

P1: X= 396550,982 Y= 4633143,203 Z= 389,561  
 P2: X= 396780,397 Y= 4632896,352 Z= 355,394

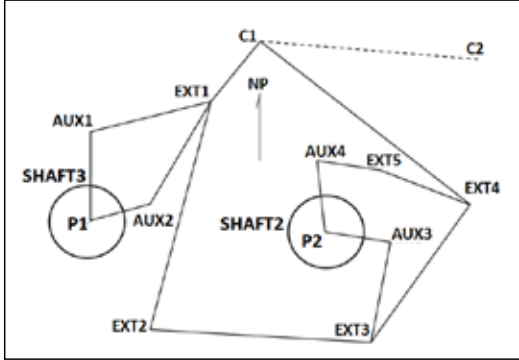


Figure 2. Closed surface traverse between plumb P1 of shaft Pou3 and plumb P of shaft Pou2.

The total error obtained in the surface traverse is (2):

$$T = \sqrt{0,004^2 + 0,002^2} = 0,0045 \text{ m} \quad (2)$$

Taking into consideration that each of the 5 sights of the traverse C1-EXT1-EXT2-EXT3-EXT4-C1 has the same displacement error “ed” caused by the combination of the angular and lineal error of the total station used, the error for each axis is (3):

$$0,0045^2 = 5 \cdot ed^2 \Rightarrow ed = 0,002 \text{ m} \quad (3)$$

It is assumed that in each sight of the traverse to arrive to the plumbs there is the same error for each sight. This implies that the total error of situation of plumb P1 (EsP1) and P2 (EsP2) taking in consideration 3 sights and 5 sights respectively from C1 is:

$$\text{For P1: } EsP1 = \sqrt{3 \cdot 0,002^2} = 0,0035 \text{ m}$$

$$\text{For P2: } EsP2 = \sqrt{5 \cdot 0,002^2} = 0,0045 \text{ m}$$

Therefore, the total error in the base P1-P2, due to the surface surveying works will be (4):

$$m_s = \sqrt{0,0035^2 + 0,0045^2} = 0,0057 \text{ m} \quad (4)$$

As previously mentioned, inside of the mine it a closed traverse between the plumb P2 and plumb P1 were measured. The errors and characteristics of this traverse are:

- Total length: 236,129 m
- N° of legs =4
- Angular error = 60s
- Error in X= 0,003 m
- Error in Y= -0,0015 m

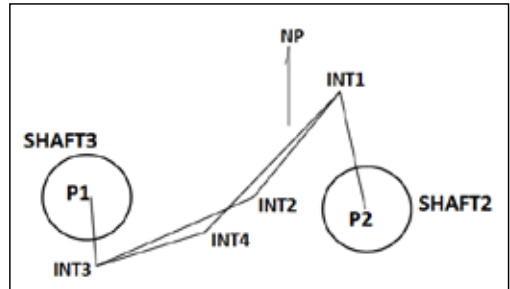


Figure 3. Interior traverse between plumb P2 and plumb P1

The total error in this underground traverse is (5):

$$T = \sqrt{0,003^2 + 0,0015^2} = 0,0034 \text{ m} \quad (5)$$

Taking into account that for each sight of the traverse INT1-INT2-INT3-INT4-INT1 there is the same displacement error “ed” due to the combination of the angular and lineal error of the total station used, then the error for each sight is:

$$0,0034^2 = 4 \cdot ed^2 \Rightarrow ed = 0,0017 \text{ m}$$

It is assumed that in each branch of the traverse to arrive to the plumbs there is the same error for each sight. This implies that the total error “mb” in the underground surveying works, considering P2 as the origin, is (6):

$$m_b = \sqrt{4 \cdot 0,0017^2} = 0,0034 \text{ m} \quad (6)$$

The sources of error of “mp” caused by the ventilation require a detailed study due to their complex deduction.

It is important to know that the errors “mp” of the plumbs have special importance in the transmission of the orientation ( $\alpha$ ) because it is an angular measurement and the final lineal error increases as long as the traverse is. In our case the farthest point of the traverse to guide the ramp is 3.9 km away.

With the objective of decreasing this error the plumbs should be as separate as possible. If there is only one shaft this poses difficulties because the maximum distance will be between 2 and 4 meters. If there is two shafts, as is present in this case, the separation of the plumbs is larger and because of it the angular error will be shorter. In the studied mine the separation between shafts is of 100 meters.

In the section following there is a detailed analysis of the different components of the error “mp” and the different ways to calculate and compensate it. To deduce the error “mp” the present paper is based in the studies done by Charzanowsky, A.; Robinson A. In these studies, Charzanowsky show that the verticality of a plumb hanged in a vertical shaft is effected by the following components: the ventilation, oscillation and vibration of the plumb, the spiral shape of the wire and the spherical shape of the earth.

### 3.1 Ventilation influence

In the undergrounds mines it is necessary ventilation to renew the stale air. Usually, the natural ventilation of the mine is not enough and large ventilation fans are required to renew the air. This ventilation produces a significant displacement on the verticality of the plumbs. In order to decrease the verticality error it's need to stop the artificial ventilation.

Once the ventilation's air flow is stopped, we need, in our case, between 12 and 24 hours to stop completely (deduced from previous experiences). Usually this time is too long because the mining activities have

to stop completely during this period. In our case, we stopped the ventilation 6 hours before the first underground plumb measurement. In the shaft Pou2, after the 6 hours without ventilation and the shafts capped, a reduction in the air speed from 1,7 m/s to 0,79 m/s was achieved.

In the shaft Pou3 (ventilation way out) the measures taken didn't have the same effect as the other shaft. The main reasons are that only 20 % of the top of the shaft could be capped and there is a big underground distance between the shafts, so the capping of shaft Pou2 didn't have much effect on Pou3. Moreover the plumb measurements were carried out only 5 hours later after capping the shafts.

To achieve a higher reduction in ventilation speed, similar to shaft Pou2, it would be necessary to perform the measures after a longer period of time, giving time for the capping of shaft Pou2 to have an effect to the shaft Pou3. The measured velocity in the bottom of shaft two during the first set of measurements was 1.16 m/s and in the beginning of the second set was 0.93 m/s.

Once all the data obtained was processed it was observed that the difference from the first set of measurements and the second set of plumb P1 was of 1.2 cm in the opposite direction were the ventilation pushed the plumb. As the ventilation decreased there was less force applied to the plumb. As it's logical it was taken the second set of measurements.

From the studies of Charzanowsky the displacement of the plumbs caused by the ventilation could be calculated. To do this it was took into consideration the air speed during the measurements in each shaft, the depth of the shafts, the plumb weights, the section of the shaft, galleries and other factors. The final result was of 3 mm in shaft Pou2 and 7 mm in shaft Pou3.

### 3.2 Influence of the plumb oscillation

The plumb is always in a pendulum movement, swinging around the suspension point. As long is the plumb wire the oscillation becomes broader. Unlike the error

caused for the ventilation, this is hardly predictable until the plumb is hanging. This is because the causal factors can't be modeled.

To compensate for this error the centre of the oscillation in the bottom of the plumb has to be defined. It has to be taken into account that the centre of the oscillation is not the exact point of verticality, because the ventilation, the spiral shape of the wire and the vibrations also apply a displacement on it.

The bigger the oscillation of the plumb, the larger the error in the deduction of the centre of the oscillation, decreasing the precision. For that reason the plumb should be submerged in a high density oil tank.



Figure 4. Plumb submerged in high density oil in a 1m<sup>3</sup> tank.

Formerly a broad variety of devices and difficult techniques to deduce the central oscillation point were used, but with the use of total stations the process has been substantially simplified. The EDM and the high precision total stations made the deduction of the central point an easy and accurate procedure.

With the explained procedure described to compensate for the ventilation effect, the error caused by the oscillation is automatically compensated. The taking of measurements of the ends of the oscillation amplitude and later averaging the lectures automatically determine the centre of the oscillation.

### 3.3 Influence of the vibrations and the wire spiral shape

Using a 2 mm anti-rotation wire rope and the method of submerging the finned plumb bob into a oil tank decreased most of the error caused for the wire spiral shape and the vibration. We consider this error so small that it can be discarded.

### 3.4 Error caused by the spherical shape of the earth and the gravity

When we are lowering two plumbs, they do not descend in a parallel way, and if we prolong them until the centre of the earth they would converge. This effect is caused by the influence of gravity. Therefore the distance between the plumbs to the surface is not the same at a certain depth, as the next scheme shows:

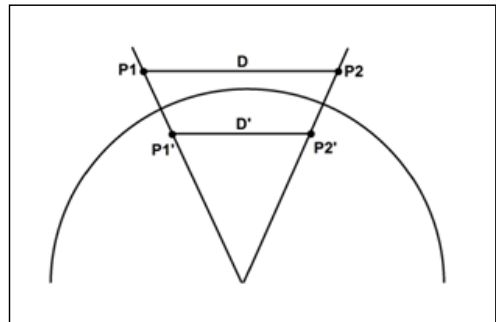


Figure 5. Scheme of the distance reduction because of the gravity effect.

The distance in a projected alignment to a certain deep can be deduced from the next formula (7) (Martín F., 1983):

$$D' = D - \left[ D \cdot \frac{H}{R} + D \cdot \frac{H^2}{R^2} \right] \quad (7)$$

Where:

- D : Distance of the alignment in the surface (m).
- D' : Distance of the alignment in a certain deep (m).
- H : Deep of projection of the surface alignment (m).
- R : Average Radio of the earth (6371000 m).

Taking into account that the distance “D” is 100,618 m and the deep “H” is 680,591m and applying the previous formula it can be deduced that the decrement of distance of the projected alignment is 1.1 cm. This error can be compensated through the lineal measurements of the underground traverse that join the two plumbs. Because of that, this error is eliminated when we apply this compensation.

### 3.5 Quality control

To do a quality control of the performed works it is assumed that the closing error of the underground traverse was a good indicator. This traverse was between plumb P2 and plumb P1. This lineal traverse closing error is an indicator of the global precision of the works, because it accumulates the errors committed on the surface works and the plumb verticality ones.

The lineal error of the underground traverse between the plumbs, after and before applying the compensation is shown in the next table:

	X Error (m)	Y Error (m)	Lineal Error(m)
Without ventilation or gravity compensation	-0,0096	0,0145	0,0174
With ventilation compensation	-0,0069	0,0103	0,0124
With ventilation and gravity compensation	-0,0020	0,0040	0,0045

Table 1. Error analysis table

### 3.6 Calculation of the errors $m_p$ i $m_\alpha$

As all the errors that compose the verticality error of the plumbs ( $m_p$ ) could be compensated with the studies of Charzanowsky and with the used methodology, the error “ $m_p$ ” can be considerate practically annulled. With the objective of being restrictive it has taken in consideration a 25% of the total indicated error. Because of this the considered error will be:

- Verticality error in the plumb P1 because of the ventilation= 7mm

- Verticality error in the plumb P2 because of the ventilation= 3mm
- Verticality error in the plumb caused by the gravity= 5.5 mm

$$m_p = \sqrt{7^2 + 3^2 + 5,5^2} * 0,25 = 2,7mm$$

The total error that affected the precision of the transmission of the orientation and cartographic system between the outer and the inner part of the Súría (Spain) potash mine was of:

$$m_\alpha = \sqrt{m_s^2 + m_p^2 + m_b^2}$$

$$= \sqrt{5,7^2 + 2,7^2 + 3,4^2} = 7,2 mm$$

With the indicated error and the separation of the plumbs it can be found the angular error:

$$Error = \frac{0,0072}{100,618} * r^g = 0,0046^g$$

Where “r” is the value in radians = 63,6620<sup>g</sup>. As the results show, the obtained error is lower than the maximum tolerance scheduled.

## 4 CONCLUSIONS

With this study I have demonstrated the large influence that the ventilation has in the verticality of the plumbs and an accurate work is required to minimize the effects.

The method used to compensate the effect of the ventilation has given excellent results in the studied case.

It has been demonstrated that it is necessary to compensate for the error caused by the reduction of the projected distances to a certain depth caused by the gravity. Once this error is compensated for, the closing error of the underground traverse has been reduced to the minimum value.

The angular error of the underground traverse between P1 and P2 only has an error of 0,0046<sup>g</sup> that is lower than the scheduled tolerance.

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## Technical-Economical Comparison Between Excavation by D&B and by Roadheader in Two Underground Gypsum Quarries

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**ABSTRACT** The goal of the research is to evaluate the most suitable exploitation technique in two underground gypsum quarries, both of them located in Piedmont (Italy). The methods currently adopted pertain to the “Rooms and Pillars” category, but the techniques employed in the two quarries are different, i.e. mechanical (by roadheader) and conventional (by Drill & Blast) respectively.

The choice of an excavation technique results from a study which takes into account a number of parameters, such as morphology of the orebody, productivity, depreciation, workforce, plant engineering and consumptions.

First of all, a thorough knowledge of the geological characteristics of the area is necessary, and geomechanical properties of gypsum must be known: a wide, bedded and regular orebody favors the use of the mechanical exploitation, whereas inhomogeneous, non-linear and small orebodies often require the great versatility of the explosive.

The technical comparison has highlighted that the productivities obtained by the two excavation techniques are almost equivalent: gypsum allows minimal tools wear, optimizing production times of the cutter head, whereas its plasticity can require specific consumptions of explosive higher than in rocks of comparable hardness.

As to the economical aspect, the total costs of excavation with Drill & Blast are 25% higher than those with roadheader: indeed, even if the mechanical excavation will lead to higher investment costs for machinery and to large consumptions of electricity, it involves a very important saving on diesel fuel and explosives, which has a fundamental impact, i.e. a third of the costs of consumption, more than one sixth of the total costs of extraction, as well as an increase in labor costs.

Another important aspect is the environmental sustainability of the mechanical excavation, with a diesel consumption much lower than the conventional method, and the possibility of an almost total abatement, if conveyor belts are employed instead of truck dumpers for muck transportation.

Based on these considerations, granted that the choice of the best technique is strongly linked to the orebody characteristics, the research focused that, in case of gypsum, the mechanical excavation is by far more suitable than the Drill & Blast, provided that the morphology is favorable.

## 1 GENERAL

Both quarries taken into account, which will be named in the following *A* and *B*, belong to the same set of hills, called *Monferrato*, an area with a peculiar history, located in the Tertiary Piedmont Basin (TPB), between the western sector of the Alps and the Apennines. The Tertiary Piedmont Basin was reliably created in the collision between the European plate and the Adriatic plate and placed on a set of other geological units, which can be attributed to the paleo-European margin.

The two quarries, although very close, have different geo-morphological characteristics, highlighted by recent geological studies carried out in the area: the *A* quarry is a set of Cenozoic sedimentary deposits (Eocene and Upper Messinian) of marine origin and covering a hilly area formed due to the back flow of metamorphic units of the Ligurian Alps, followed by a tectonic history very complex, which leads to a considerable stratigraphic variability across the hilly area.

The *A* quarry site is characterized by a gypsum structure in sub-horizontal benches in alternation to marly layers, only in some points partially deformed by phenomena of load, while the exploited area at *B* quarry is formed by gypsum blocks of spheroidal shape, probably originated following a paleo-landslide within the tectonic outcrop of the massif of Villadeati.

The area of *A* quarry has been characterized for years by so-called subjacent karst (Fioraso et al., 2004), in which the gypsum interval is only partially incised and drained by the hydrographic basin and the dissolution is well developed exclusively above the basic level, in eluvio-colluvial coverage and in the first meters of gypsum deposits, in case of fractures or cracks that allow the passage of water.

However, it has been recently identified a flow of underground water fed not only by

infiltrations from the surface, but also by the circulation of water deeper than the area of exploitation. It is an important water circuit that involves the whole underlying rock mass.

On February 2005, the excavation intercepted a karstic cavity, that flooded the adjacent area with a water volume of about 60,000 m<sup>3</sup>, a so-called inrush phenomenon, with extensive damage to the mining activity, which was interrupted for 4 months for technical and administrative reasons, and that led to substantial changes in the way of organizing the quarry cycle.

Following the inrush, it was decided to perform drilling along the axis of the tunnel before each advancement, in order to explore the possibilities to intercept karstic cavities; continuous monitoring and chemical analysis of water were carried out and the pumping station was potentiated.

The area of *B* quarry is, on the contrary, characterized by the presence of an entrenched karst (Fioraso et al. 2004), where the gypsum blocks are deeply dissected by the river network.

The gypsum lenses are broken in the first meters of depth and geomorphology of the orebody does not involve the formation of impressive karstic cavities, as rainwater flows through the rocks without any obstacles that could lead to inrush phenomena.

The deepest area is compact and waterproof, and does not involve geological phenomena of interest: any water ingress concern locally small areas of the rock mass as a whole waterproof.

### 1.1 Description of the site

The *A* orebody has been defined through a campaign in 1990 by 13 exploratory drills and geological surveys of the area, which continued in the following years until the achievement of 36 core samples, with depths between 100 and 220 m, which revealed five gypsum layers (only three are currently exploited – see Figure 1) interspersed with layers of marl with an average dip of about 10° in the direction N 60° E.

The gypsum is present in two facies, distinguishable through the average grain size, i.e. micro-crystalline gypsum ("micro"), constituting the first two layers (with a thickness from 10 to 30 m), and macro-crystalline gypsum ("macro"), which characterizes the three deeper layers: each of them is separated by a silty-clayey layer. From these layers, characterized by an uniform material having an average thickness of about 10 m, a gypsum of higher quality than the "micro" is present; anyway, the last horizon has not been exploited yet, being affected by phenomena of deep karstification.

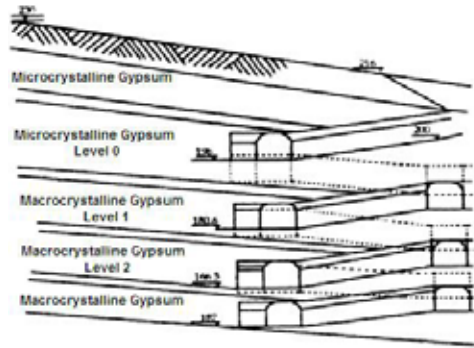


Figure 1. Scheme of the orebody exploited at *A* quarry, where it is possible to recognize: superficial bench, thickness 10-30 m, gypsum "micro", irregular and karst phenomena, impossible the exploitation; Level 0, thickness 10 m, gypsum "micro", still exploited; Level 1, thickness 10 m, gypsum "macro", exhausted, problems with the inrush of 2005; Level 2, thickness 10 m, gypsum "macro", still exploited, beginning of the new expansion area authorized; level 3, not yet exploited, gypsum "macro", with important karst phenomena and possible geo-structural problems, it has to be determined whether the exploitation can be start or not.

The scheme of exploitation is given in Figure 2. Gypsum is characterized by a strong inhomogeneity at all levels of exploitation: in part it's micro-crystalline (fine-grained), in part macro-crystalline (coarse-grained), often separated by the previously mentioned marly-clayey insertions.

The *B* quarry is characterized by a gypseous lens with thickness between 30 and 40 m and extension of about 100 m, irregular and elongated in the EW direction, with scarce outcrops at the surface, affected by marly intercalations crossing the structure, without however jeopardize the continuity and regularity of the orebody.

## 2 EXPLOITATION METHOD AT A QUARRY

The gypsum characteristics were detected through geo-gnostic campaigns made over the years in the area of the quarry. Data collected in Table 1 give an idea of the differences observed between the macro-and micro-crystalline gypsum. The exploitation method used at A quarry is by Room-and-pillar, which is specifically designed for mining of flat, bedded deposits of limited thickness.

By room and pillar methods, the ore is recovered in open stopes, leaving pillars to support the hanging wall; the roof must remain intact.

The method consists in the excavation of tunnels separated by diaphragms of variable size as a function of the geo-mechanical rock properties; it's the most advantageous system to employ the mechanical excavation by roadheader, that is the selected machine (a *Sandvik MR 380*, whose main characteristics are given in Table 2, is employed).

In addition, the method reduces the number of entrances, and it is easier to confine the dust due to the excavation.

Table 1. Main data pertaining to gypsum exploited at A quarry, used to perform the stability analysis aimed to define the exploitation method. Symbology:  $\gamma$ : specific gravity;  $C_0$ : Uniaxial compression strength;  $E_t$ : elastic modulus;  $T_0$ : tensile strength;  $C_p$ : peak cohesion;  $\phi_p$ : peak friction angle.

Characteristics	Macro-cryst. Gypsum	Micro-cryst. Gypsum
$\gamma$ [kg/dm <sup>3</sup> ]	2.18	2.12
$C_0$ [MPa]	8.40	14.51
$E_t$ [GPa]	14.77	20.38
$T_0$ [MPa]	0.56	2.27
$C_p$ [MPa]	1.60	3.87
$\phi_p$ [°]	55	58.1

The three levels are similar in structure and have been exploited with the same pattern,

allowing greater ease of extraction: two pairs of tunnels are driven parallel to the gypsum layers, from SW to NE, connected by two pairs of tunnels perpendicular (Figure 2).

Table 2. Description of the main characteristics of *Sandvik MR 380* roadheader.

Length	15200 mm
Height	4523 mm
Width	2600 mm
Total weight	65 t
Max cutting height	6300 mm
Cutting width	7600 mm
Pull (per track)	350 kN
Hydraulic pressure (max)	225 bar
Load capacity (max)	15 t/min
Conveyor capacity (max)	350 m <sup>3</sup> /h
Displacement speed (max)	5 m/min
Belt speed	1.88 m/s
Total powder	350 kW
Head power	200 kW

The main tunnels have the function of cardinal axis for the main facilities necessary for the progress of work, in particular the electrical and ventilation systems, and represent a starting point for the excavation of the exploitation tunnels, inclined between 115° and 150° with respect to them, optimal for the excavation by roadheader.

Also, they have been built downhill (10% slope) in order to set up the natural water catchment areas, which inevitably flow by gravity from the secondary tunnels, realized uphill, piling up in basins from which water will be transported through a pumping system; the main water supply basin is located in the deepest part of the quarry.

The excavation cycle is continuous, being realized thanks to a transverse head roadheader; mucking is removed by a dumper.

In Figure 3, the mean geometry of tunnels is given, referring to the cross sections employed at Levels 1 and 2 (De Cassai, 2007).

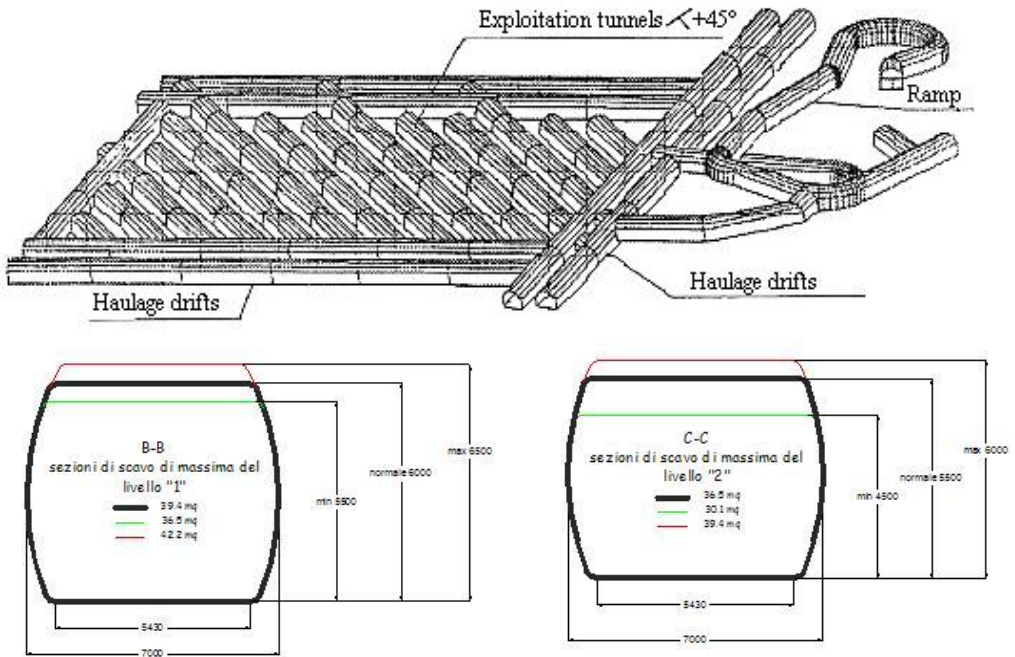


Figure 3. Minimum, mean and maximum cross section of the tunnels at Levels 1 and 2.

### 3 EXPLOITATION METHOD AT B QUARRY

The exploited orebody is a sedimentary deposit, approximately 100 meters wide and 40 meters deep, dipping approximately 50 degrees to the SW. The waste cover is about 20 m deep, consisting of loose colluvial debris, laying on a bed of marl and calcareous sandstone. The orebody develops in E-W direction, with interlayers of marl in the same direction.

Two main discontinuities, consisting of marl layers up to some decimetres thick, dip parallel to the orebody, with the joints filled with silt and clay. With room and pillar mining (Figure 4), the orebody is excavated as completely as possible, leaving sections of ore as pillars to support the hanging wall. Room and pillar is the most common method of mining flat deposits of limited thickness, and it is to a great extent used for rocks of sedimentary origin, such as gypsum. The

dimensioning of stopes and pillars depends on the stability of the hanging wall and the ore itself, the thickness of the deposit and the rock pressure. Pillars are arranged after a regular pattern, shaped as elongated walls, separating the stopes: the usual thickness of the pillar is the same as the width of the stope. The drilling (Figure 5) and blasting operations are organized with the objective to perform up to three blasts per day, generally at the same time, with short intervals (max. five minutes) amongst them.

The characteristics of gypsum at *B* quarry (Table 3) were detected through geo-gnostic campaigns made over the years in the area of the quarry.

The gypsum has a different structure compared to the *A* quarry and its characteristics pertain both to the micro-crystalline and to the macro-crystalline type (Massobrio and Gennaro, 2006).

The opening cut is performed with horizontal holes, according to different

geometries, V-cut and fan cut respectively. Three schemes are adopted for blasts, as shown in Figure 6. Scheme n.1 consists in a

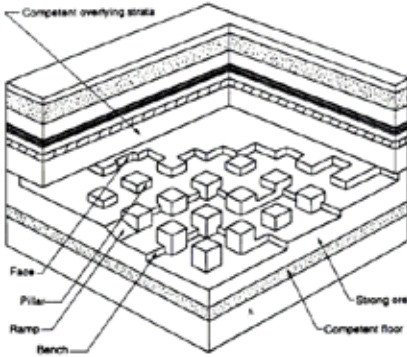


Figure 4. Schematic view of the “Room and pillars” exploitation method (Camm, 1991).



Figure 5. detail of a Tamrock jumbo while performing the drill-holes.

Table 3. Main characteristics of gypsum at *B* quarry. Symbology:  $\gamma$ : specific gravity;  $C_0$ : Uniaxial compression strength;  $E_t$ : elastic modulus;  $C_p$ : peak cohesion;  $\phi_p$ : peak friction angle.

Characteristics	
$\gamma$ [kg/dm <sup>3</sup> ]	2.2
$C_0$ [MPa]	7.66
$E_t$ [GPa]	13
$C_p$ [MPa]	1.38
$\phi_p$ [°]	50°

V-cut placed in the middle of the cross section and quite low down; each V in the cut, consisting of 6 holes, is fired with the same interval number, using 25 ms delay electric detonators, to ensure a good coordination between the blast-holes with respect to breakage.

The charge concentration in the cut holes is 2.4 kg/hole, resulting in a CPD (charge per delay) of 14.5 kg. Total consumption of explosive per blast is 210 kg, divided into 18 delays. 1 cartridge of dynamite is placed in each hole as bottom charge, while the column charge consists in 3 cartridges of emulsion.

Scheme 2 adopts a fan cut (Figure 6); total consumption of explosive per blast is 90 kg, divided in 10 delays, with a CPD of 14.1 kg.

Scheme 3 is quite uncommon, and is just adopted to rearrange the floor of the quarry; also in this case a fan cut is adopted, while all the stopping holes are drilled following an almost squared pattern with the same inclination (Figure 6).

Each row consists in 8 blast-holes, simultaneously fired. Total consumption of explosive per blast is approximately 120 kg, divided into 10 delays, with a CPD of 11,3 kg.

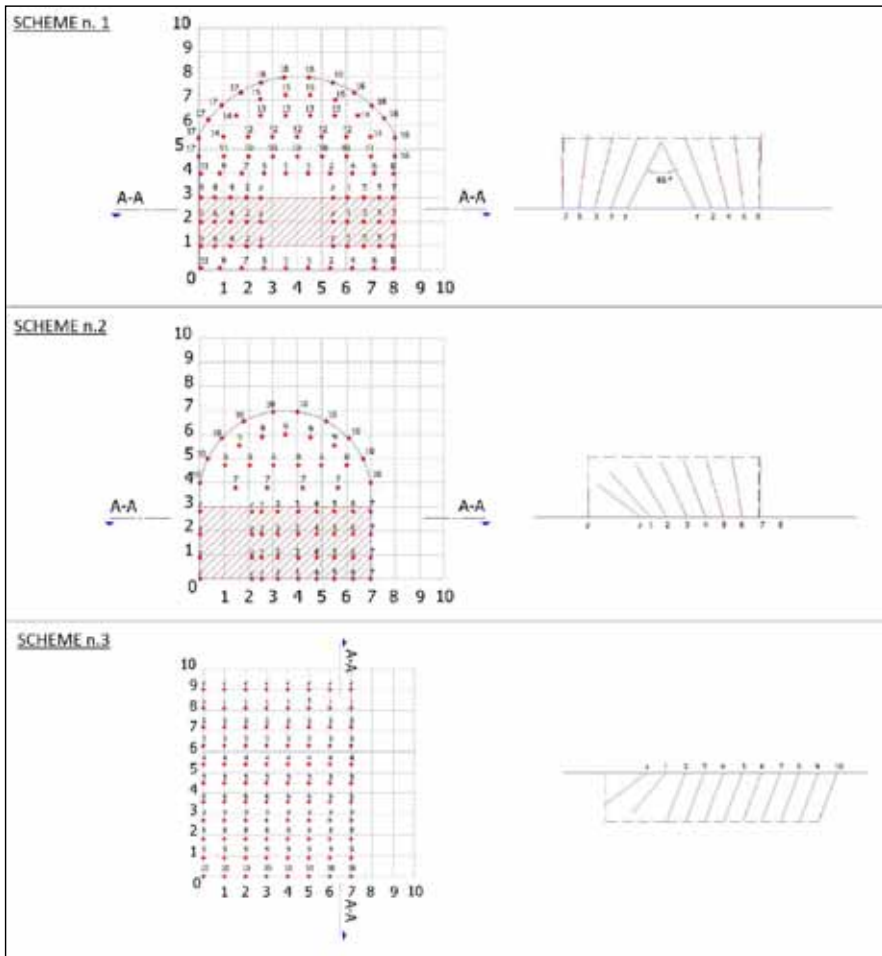


Figure 6. Schemes of the blasts adopted in the quarry (Cardu et al., 2011).

Charging geometry, both for scheme 2 and 3, consists of 2.5 cartridges of dynamite in each hole as a bottom charge, and of 2 cartridges of emulsion as a column charge.

Stemming is not adopted and the firing pattern is designed according to the employment of 25 ms detonators.

#### 4 TECHNICAL – ECONOMICAL COMPARISON

At the *A* quarry, the choice of a roadheader is favored by the presence of an

orebody characterized by parallel layers with slopes less than 20%, very extended at least in one direction.

This allows very long tunnels being performed (cross section  $\approx 40 \text{ m}^2$ ) with few entrances. Moreover, the rock is scarcely abrasive, favoring the mechanical excavation.

At *B* quarry, on the contrary, the ideal conditions for the use of explosive are the excellent self-supporting capacity of the rock, and the medium-large section of excavation. Furthermore, since the gypsum

is rather inhomogeneous, it is necessary to employ a flexible technique.

Productivity at *A* quarry is guaranteed by two *Sandvik MR 380* roadheader, working on 2/3 shift per day. The data relevant to the last three years show an increase of the volume of gypsum exploited, from 57,000 m<sup>3</sup> (2009) to 72,400 m<sup>3</sup> (2011), corresponding to an average advancement of about 4.8 m/shift. This fulfils a productivity of 165.87 m<sup>3</sup>/shift.

At *B* quarry, only one daily blast is scheduled at present. The data collected have provided a mean value (extrapolated taking into account that the three blast schemes correspond to different pulls and different cross-sections) of 39,723 m<sup>3</sup>/year exploited; this corresponds to an annual average productivity of 164.81 m<sup>3</sup>/shift.

From the above quoted results, it follows that the productivity is comparable, being 99.36% the ratio between the two units (*B/A*), with reference to a working shift.

The economic comparison is discussed in relative terms, using a parameter for each cost item. The basic parameters are: depreciation costs; investment costs (plant); labour costs; production (consumption) costs.

Each of these parameters will be subjected to comparison (in percentage terms) to verify what is actually the more convenient scenario.

The equipment at *A* quarry consists of: 2 *Sandvik MR380* roadheaders; 4 Volvo

dump trucks; 1 Volvo wheel loader; 1 Merlo lifter.

The equipment at *B* quarry consists of: 1 Tamrock jumbo HS 105 with generator annexed; 2 dump trucks (DAF, Astra); 1 CAT excavator for scaling; 1 CAT crawler loader; 1 *Liebherr* wheel loader; 1 *Dieci* lifter.

The depreciation costs are shown in the histogram of Figure 7, with reference to a base 100 of the fleet available at A/m<sup>3</sup> in place produced. Depreciation of mechanical excavation is more than a third greater, compared to the excavation with explosives; the roadheader is the most important item, so that alone exceeds the whole cost of the fleet/m<sup>3</sup> of the D & B.

Plants at *A* quarry consist of: ventilation: 2 fans + 2 filters; electricity: 2 cabs + wires; dewatering: pumps, pipes, connections, valves, pumps plant.

At *B* quarry, plants consist of: crushing: 1 crusher; ventilation: 3 fans; electricity; dewatering: pumps. The comparison through histograms (Figure 8) is carried out with reference to a base 100 of the investments for plants at A/m<sup>3</sup> in place produced. The item that makes the plants employed at *B* quarry much more expensive is the primary crushing. The cost of electricity and dewatering at *A* quarry is comparable to D & B. The ventilation system has a very high cost with regard to the excavation with roadheader, due to the dust problem. The pumping plants have cost three times higher at *A* quarry, because of greater hydro-geological problems.



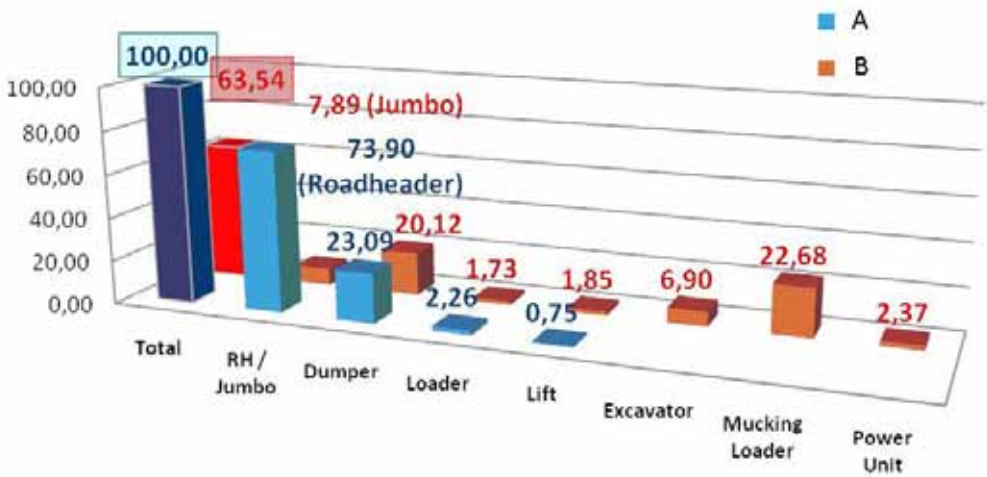


Figure 7. Comparison between the fleet of the two units with regard to depreciation.

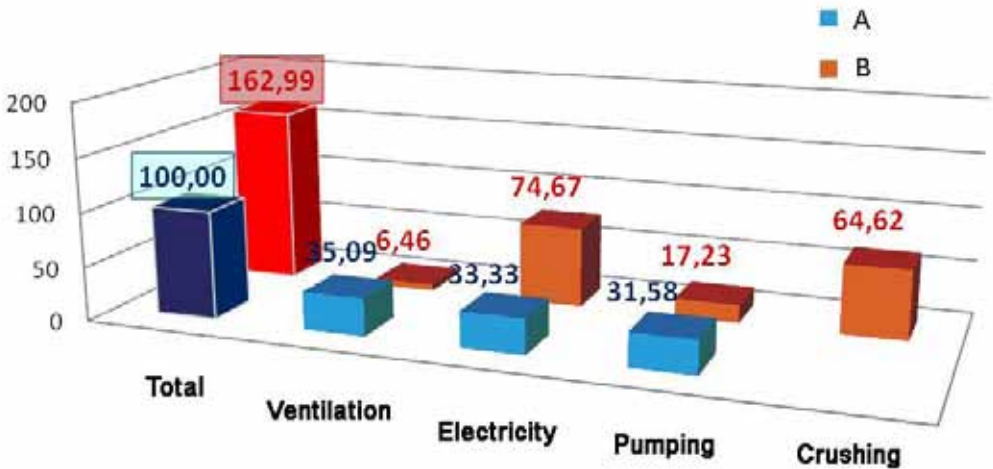


Figure 8. Comparison between the investments in the two units with regard to plants.

As to the labour, costs/m<sup>3</sup> are comparable, slightly in favour of mechanical excavation.

At *A* quarry there are 9 workers for three shifts, while at *B* quarry there are 6 workers, of which only 4 or 5 are involved in the daily shift. The comparison through histograms,

reported to a base 100 with respect to the cost of labour at A/m<sup>3</sup> in place, is given in Figure 9.

The items of consumption at *A* quarry include: electricity, influenced by the roadheader and by the dewatering plant; diesel fuel, used by trucks; tools (the

consumption of peaks and sleeves is very low, about 0.015 pieces/m<sup>3</sup>, due to the low abrasiveness of gypsum); oil, grease consumption and standard maintenance. The items of consumption at B quarry include: explosives and detonators; electricity, influenced by the crushing plant; diesel oil used for trucks, shovel and jumbo; tools, rods and tips (although the consumption is minimum); oil, grease and standard maintenance. The comparison between the consumptions in the two units is shown in Figure 10. The cost of explosives is the fundamental basis for discrimination between the two methods of excavation: not considering the net cost/m<sup>3</sup>, consumption

obtained between D & B and roadheader are roughly the same.

The consumption of electricity due to mechanical excavation is much higher (2.5 times greater than the D & B), since the roadheader is electrically powered, and vice versa as regards the diesel fuel. Maintenance is an item of constant weight.

The economical evaluation ends with the comparison between the general costs of exploitation of the two excavation techniques, defined on a base 100 of the costs linked to the consumption at A /m<sup>3</sup> in place (Figure 11).



Figure 9. Comparison between the costs of labour at the two units.

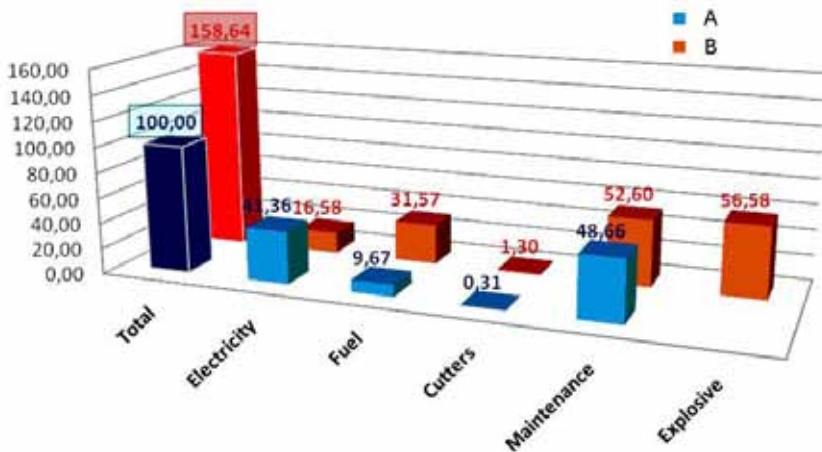


Figure 10. Comparison between the consumptions at the two units.

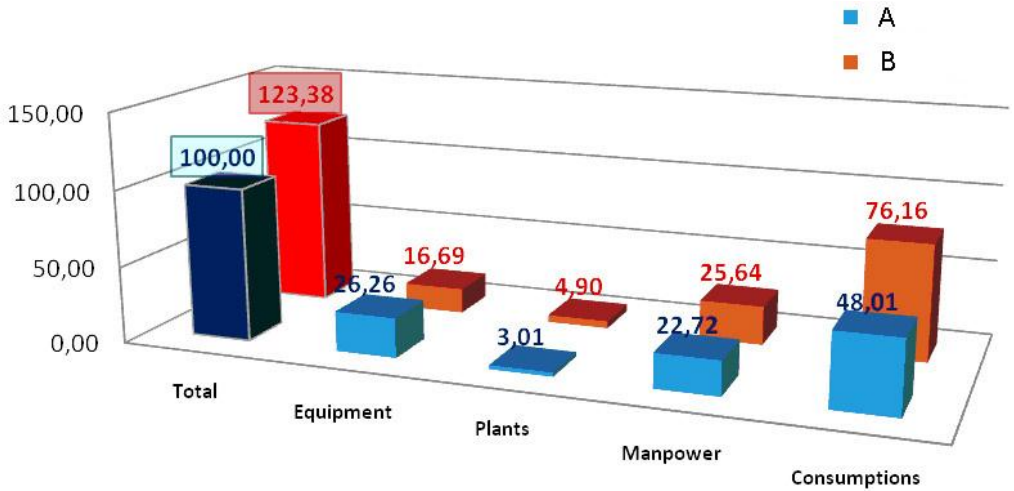


Figure 11. Comparison between the overall exploitation costs at the two units.

The cost of extraction of gypsum by D&B is 23.38% higher than that resulting from mechanical excavation, due to a greater cost/m<sup>3</sup> linked to the consumptions and especially to the explosive, despite the higher costs associated with depreciation.

## 5 CONCLUSIONS

The choice between the excavation with explosive and by roadheader for a gypsum quarry is the result of a scrupulous study, which must take account of a very large number of parameters.

First of all, a thorough knowledge of the geology of the area, together with the morphology of the orebody and the geomechanical properties of gypsum is necessary; gypsum shows a tendency for the mechanical excavation: a stratified orebody, wide-ranging, with linear development and modest slope are the ideal conditions for the roadheader, while in an inhomogeneous, non-linear, or small orebody the great ductility of the explosive leads to the choice of D & B method.

In this case, it is very important the history of the company and its market, because while the *A* quarry has an European market,

*B* quarry is limited to a regional demand and therefore the excavation with roadheader, having higher costs and important difficulties from the point of view of plants, can be hardly faced at *B* quarry.

The technical comparison highlights that the productivity obtained by the two methods of excavation are almost the same (slightly favourable for D & B, but with the same cross section the roadheader could be more productive than explosives): gypsum involves minimal wear of the tools, optimizing the roadheader's productivity, and is excellently self-supporting. The dead time caused by drilling before each advancement to prevent inrush phenomena in *A* quarry represents a loss of productivity that does not find similar evidence at *B* quarry.

At *B* quarry the hydro-geological problem is much less important, leading to a considerable reduction of the management costs and to an increase of productivity but, nevertheless, the total costs of extraction with D & B are 25% higher than those of the excavation with roadheader.

The mechanical excavation involves an investment cost almost double, a triple consumption of electrical energy (that is, however, partly due to the consumption of

the pumps for dewatering at *A* quarry, therefore the consumption under the same conditions would be even smaller), but a very important saving regarding the diesel fuel and the explosive, which has a fundamental impact, about one third of the cost of consumption, more than one sixth of the total cost of extraction, as well as an increase in labour costs, which are still comparable.

Another important aspect is the environmental sustainability of the mechanical excavation, with a diesel consumption much lower than the traditional method and the possibility of almost total excavation with the use, for example, of a conveyor belt for mucking (instead of dump trucks).

On the basis of these considerations, the decision to use a method of excavation rather than another is strongly linked to the characteristics of the orebody but, where the morphology is favourable, the use of mechanical excavation for gypsum is to be considered much more appropriate than the traditional method.

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# Numerical Analysis of Active Earth Pressure in Cohesionless Soils

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**ABSTRACT** This study examines the active earth pressure of cohesionless soil, acting against a vertical retaining wall by using a two dimensional finite difference analyses, implemented in the Fast Lagrangian Analysis of Continua code (FLAC). The active earth pressure coefficient is estimated, for various mechanical and geometrical parameters. The effect of backfill inclination, soil–wall interface friction angle and proximity of a slope are investigated. The results of the present study are compared with those available in the literature.

**Keywords:** Retaining wall, active earth pressure, interface, sloping backfill, numerical modelling

## 1 INTRODUCTION

The problem of active earth pressures acting against rigid retaining walls has been studied by different methods. Coulomb (1776) and Rankine (1857) proposed the theories that remain the fundamental approaches to analyse the active earth pressures. Coulomb (1776) studied the earth pressures problem by using the limit equilibrium method, with his hypothesis of the planer failure surface (Greco, 2005; Motta, 1994). The Rankine theory is rigorous but limited to simple situations. The two theories still occupy a dominant place in geotechnical engineering practice, though more elaborated methods have been developed using curved failure surfaces (Terzaghi, 1943). Also, the limit equilibrium method of slices has been used to calculate lateral earth forces (Janbu 1957; Shields and Tolunay 1973). The slip line method (Sokolovskii, 1960; Cheng et al, 2007) and the limit analysis method (Chen, 1975; Chen and Liu, 1990; Soubra and Macuh, 2002) are also used to estimate the earth pressure.

Numerical methods have been widely used as a powerful tool to study the behaviour of the wall and soil mass retained by taking into account their deformations. Potts and Fourie (1986) and Hazarika and Matsuzawa (1996) used the finite element

method to investigate the effect of the mode of the wall movement on the generation of earth pressure. Day and Potts (1998) examined the effects of the interface properties on the behaviour of the retained soil around a vertical wall subjected to uniform translation. Fenton et al. (2005) combined a nonlinear finite element analysis with a random field simulation to investigate the effect of spatial variability of soil properties on active earth pressure.

The aim of this study is, firstly to estimate the active pressure coefficient for rigid rough and smooth walls located at the toe of slopes, and located at a distance  $s$  of the toe of slopes. The numerical results are compared with the formulations presented in the literature.

## 2 PROBLEM PRESENTATION

The numerical analyses presented in this study involve two cases to evaluate the active earth pressure coefficients. The first considers a vertical rigid wall displaced horizontally on an inclined frictional soil; the top surface of ground is inclined with an angle  $\beta$ , which has the following values: 0,  $\varphi/3$ ,  $\varphi/2$ ,  $2\varphi/3$ , as shown in Figure 1. The

second analysis concerns the influence of the proximity of slope on the earth pressure coefficients; this case treats the same problem, with the slope toe located at a variable distance  $s$ ; the ratio  $s/h$  varies from 0 to 2, as illustrated in Figure 1. The problem studied considers a rigid vertical wall, of height  $h = 1$  m, subjected to a horizontal load, which mobilizes the active earth pressures in a purely frictional soil, with no surcharge. Various soil-wall interface angles  $\delta$  were considered.

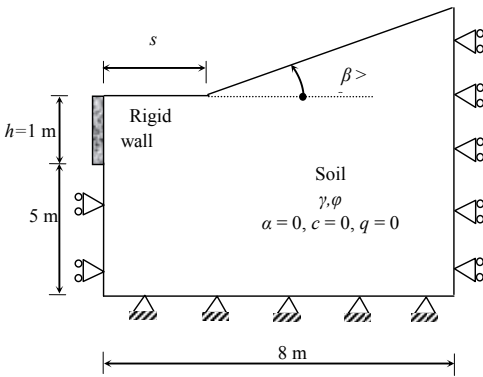


Figure. 1 Chosen domain and problem notation, slope toe located at a variable distance  $s$

### 3 MODELING PROCEDURE

The finite difference code FLAC (2005) is used to evaluate the influence of the sloping backfill and the proximity of slope on the active earth pressures against the rigid wall. FLAC is a two-dimensional explicit finite-difference code for engineering mechanics computations; it simulates the behavior of structures built of soil, rock or other materials that undergo plastic flow when their yield limits are reached.

In order to induce active failure, the rigid retaining wall is displaced horizontally away from the homogeneous soil. The dimensions of the models in the  $x$  and  $y$  directions are chosen to be eight and six times the wall height; they were chosen such that the failure surfaces developed in the ground do not cross the vertical boundaries on the

right-hand side. Calculations for a larger size model indicated that extending the boundaries further away from the wall does not influence the computed active earth pressure. The displacement of the left and right vertical sides is constrained in the horizontal direction only. The base of the model is constrained in all directions (Figure 1).

The adopted soil model is a linearly elastic-perfectly plastic model, obeying Mohr-Coulomb criterion with the associative flow rule. The elastic modulus used are the shear modulus  $G = 10$  MPa and the bulk modulus  $K = 30$  MPa; it should be noted that the values of the elastic parameters had a small effect on the limit active earth pressures. The values of  $K_{ay}$  were calculated in this numerical study for a purely frictional soil by considering a cohesionless soil with no surcharge ( $\gamma = 20$  kN/m<sup>3</sup>,  $c = 0, q = 0$ ); The angle of internal friction  $\phi$  was varied from  $20^\circ$  to  $40^\circ$  in  $5^\circ$  increments; for each value of the friction angle  $\phi$ , the dilation angle  $\psi = \phi$ . The rigid wall is connected to the soil via interface elements defined by Coulomb shear-strength criterion.

The loading of the rigid wall is simulated by imposing equal horizontal velocities at the nodes represented the wall. The progressive movement of the rigid wall induced by the horizontal velocity applied at the wall nodes is accompanied by decrease of the pressure in the soil. Finally, the resultant of the earth pressures stabilizes for a value that indicates a limit load. This load corresponds to the normal resultant of active  $P_{ax}$  earth pressures. The equivalent-earth pressures coefficients  $K_{ay}$  is calculated individually using equations 1 with  $q = 0$  and  $c = 0$ :

$$K_{ay} = 2P_{ax} / \gamma \cdot h^2 \cos \delta \tag{1}$$

Several preliminary numerical tests have been carried out to study the effect of the mesh size and the horizontal velocity applied at the wall nodes, on the limit active earth pressures. In all cases, the mesh in the wall neighborhood is refined to capture significant displacement gradients. The

refinement of the mesh and the choice of a small horizontal velocity do produce slightly better results, especially for rough interfaces. Figure 2 shows a typical mesh; it is the one used for the wall near the toe of the slope ( $s = 0.5$  m).

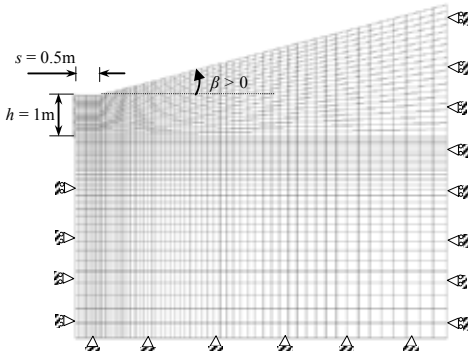


Figure. 2 Typical computational mesh and boundary conditions

## 4 RESULTS AND DISCUSSION

### 4.1 Comparison of the results for a horizontal backfill surface ( $\beta = 0$ )

To examine the accuracy of the present results, Figure 3 shows a comparison of the values of  $K_{ay}$  obtained with the results given by Kérisel and Absi (1990) (Eurocode 7, charts, (2004)) and Lancellotta (2002), for  $\delta=0$ ,  $\varphi/3$ , and  $2\varphi/3$  and  $\varphi$  varying from  $20^\circ$  to  $40^\circ$ .

Figure 3(a) presents the results of the active earth pressures coefficient  $K_{ay}$  in the case of a smooth interface ( $\delta=0$ ). It is clear that the values of  $K_{ay}$ , calculated by various authors, are very close and the difference is not significant.

The present results are slightly smaller than those of Kérisel and Absi (1990), and the maximum difference does not exceed 1.9%. As shown above, the Lancellotta's method gives the same values for  $K_{ay}$  as the Eurocode 7 expressions (2004). The difference between the present results and those of Lancellotta (2002) is very small in the case of a soil-wall friction angle  $\delta = 0$ . However, for high values of  $\varphi$  and  $\delta$ , the

present results are greater than those of Lancellotta (2002) (Figure 3(c)); the maximum difference is 9.2 % for  $\varphi = 40^\circ$  and  $\delta/\varphi = 2/3$ .

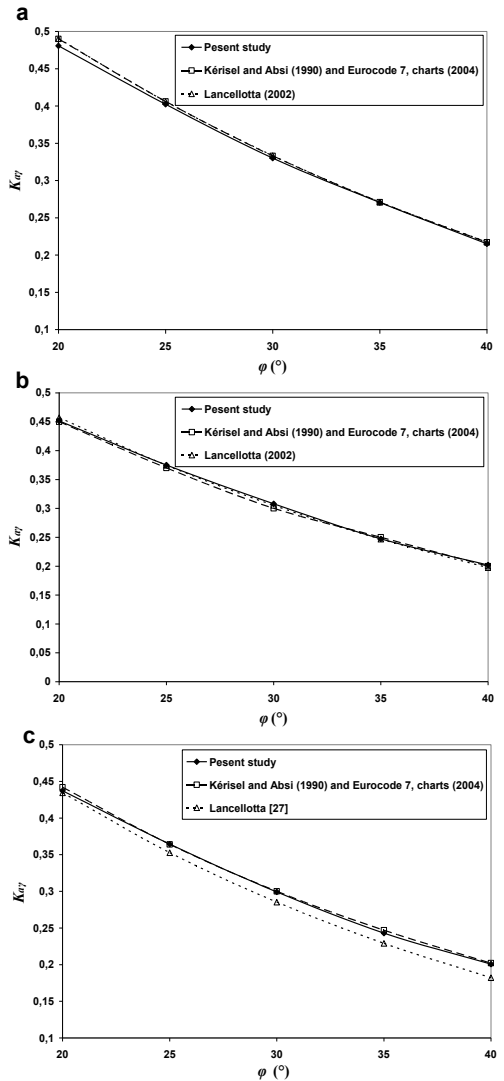


Figure. 3 Comparison of obtained  $K_{ay}$  with the various authors, for  $\beta/\varphi = 0$ , (a)  $\delta/\varphi = 0$ , (b)  $\delta/\varphi = 1/3$ , (c)  $\delta/\varphi = 2/3$

### 4.2 Comparison with published results for an inclined backfill surface ( $\beta > 0$ , $s = 0$ )

Table 3 presents the results of  $K_{ay}$  obtained for  $\varphi$  varying from  $20^\circ$  to  $40^\circ$ ,  $\beta/\varphi$

= 0, 1/3, 1/2, 2/3 and  $\delta/\varphi = 0, 1/3, 1/2, 2/3$ . According to this table,  $K_{ay}$  decreases with increasing  $\delta/\varphi$  and  $\varphi$ ; also, the slope of the backfill has an obvious influence. For a larger sloping angle, the active earth pressure coefficient is higher. The numerical predictions obtained for a smooth and rough interface, are in excellent agreement with those given by Soubra and Macuh (2002); in general, the present results are slightly smaller.

Table. 1 Results of  $K_{ay}$  obtained for  $\varphi$  varied from  $20^\circ$  to  $40^\circ$ ,  $\beta/\varphi = 0, 1/3, 1/2, 2/3$ ,  $\delta/\varphi = 0, 1/3, 1/2, 2/3$ , and  $s = 0$

$\varphi$	$\beta/\varphi$	$K_{ay}$			
		$\delta/\varphi$			
		0	1/3	1/2	2/3
20	0	Present study FLAC	Present study FLAC	Present study FLAC	Present study FLAC
	1/3	0.481	0.451	0.443	0.437
	1/2	0.524	0.497	0.490	0.484
	2/3	0.560	0.531	0.524	0.518
25	0	0.402	0.375	0.366	0.364
	1/3	0.447	0.420	0.415	0.409
	1/2	0.477	0.454	0.444	0.442
	2/3	0.511	0.488	0.482	0.481
30	0	0.330	0.308	0.302	0.299
	1/3	0.373	0.349	0.344	0.344
	1/2	0.407	0.387	0.379	0.377
	2/3	0.440	0.415	0.404	0.416
35	0	0.271	0.247	0.244	0.243
	1/3	0.295	0.287	0.283	0.276
	1/2	0.320	0.309	0.304	0.307
	2/3	0.355	0.335	0.334	0.333
40	0	0.215	0.202	0.200	0.201
	1/3	0.246	0.232	0.231	0.233
	1/2	0.262	0.246	0.248	0.248
	2/3	0.291	0.276	0.273	0.270

### 4.3 Effect of proximity of slope on the active earth pressure coefficient ( $\beta > 0, s > 0$ )

Figure 4 shows the results for  $K_{ay}$  coefficient obtained in the case of a rough interface, for an internal friction angle  $\varphi = 20^\circ, 30^\circ, 40^\circ$ ,  $\beta/\varphi = 1/3, 1/2, 2/3$  and various  $s/h$  ratios. The  $K_{ay}$  coefficient decreases with increases of  $s/h$  ratio and reaches the minimum value which corresponds to that of the horizontal backfill

surface for  $s/h$  about 0.5 and 0.7 for  $\varphi = 40^\circ$  and  $20^\circ$  respectively. Nevertheless, the reductions of the  $K_{ay}$  coefficient do not exceed 26% in all cases (obtained for  $\varphi = 40^\circ$ ,  $\beta/\varphi = 2/3$  and  $\delta = 0$ ). Also, it is clear that the effect of the slope of the backfill  $\beta$ , with regard to the ratio  $s/h$ , is the same for different internal friction angles  $\varphi$ .

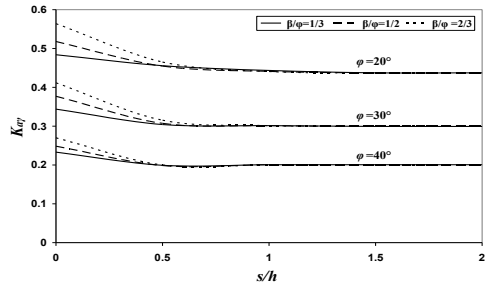


Figure. 4 The effect of  $s/h$  ratio on  $K_{ay}$  coefficients, for Rough interface

## 5 CONCLUSION

Numerical procedures were developed to evaluate the active earth pressure coefficient, using the finite difference method implemented in the FLAC code (2005). For the horizontal backfill surface, the active earth pressures coefficients of the present study are in excellent agreement with those of Kérisel and Absi (1990) which are also the base of the charts in Eurocode 7 (2004). Furthermore, the analytical results of Lansellotta (2002) are almost identical to those given by the expressions of Eurocode 7 (2004); their comparison with the present computations has shown that for high values of  $\delta$  they are the lowest.

In the case of an inclined backfill surface, the present results for  $K_{ay}$  coefficient are in good agreement with those obtained by Soubra and Macuh (2002) using the upper bound of the limit analysis. The effect of the proximity of the slope and its inclination are also examined. The magnitudes of  $K_{ay}$  decreases significantly to reach its minimum with an increase in the  $s/h$  ratio. The minimum of the active earth pressure coefficient, correspond to the values obtained for the horizontal ground surface.



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## Root Causes of Groundfall Related Incidents in Pongkor Underground Gold Mines Pt.Antam (Persero) TBK Indonesia

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**ABSTRACT** One of main objective of geotechnical assessment in underground mine is to minimize or eliminate the technical rock mass events that may cause fatal or nonfatal injuries to human workers and also equipment from rock fall causes. A commonly used technique is to devise an “incident prevention plan”, which is more often the product of thorough investigations of past reported incident events. Incident documentation, technical and methodical reporting and systematic and thorough data collection, often help identify the root cause of the incident – in other words, the etiology of the incident. These retrospective analyses of the incidents can efficiently identify the future steps to take to achieve the objective of minimizing occurrence of “events of interest”.

This paper introduces a “Taxonomic Analysis”, which identifies the root causes of an event (an incident in this case) and can provide future direction for corrective measures to reduce the probability of occurrence of the event. A taxonomic analysis involves systematic and organization of data based on observation, description, and classification. This paper describe statistically number of ground fall accident was occurred in 2003 – 2012 10 year period in **Pongkor Gold Mine. PT.Antam (Persero).Tbk**. Pongkor gold mine has implemented underground mining which has 4 (four) actively mine vein; there are Ciguha, Ciurug, Gudang Handak, and Kubang Cicaui since 1992. So these potential rock fall injuries should be manage to reduce the risk through find the root causes of rock fall. The method used in the case of rock fall injuries is the method of taxonomic analysis. This method is intended to classify the rock fall injuries that occurred in the mining thus earned a tendency pattern of accidents and mitigation.

The study found that all of groundfall incidents occurred in supported areas. 54% of Injuries Caused by un perfect support, 22 % Injuries Caused by Rock Fall initiated by Human Activities factors such as scaling or barring down, drilling or bolting, and setting h-beam,timbers or cribbing, 20% Injuries Caused by Rock Fall Initiated by Rock Mass Failure, Finally 4% Injuries Caused by Support Failure.

**Key words:** *rockfall, incident, accident, injury, underground mining, taxonomic*

## 1 INTRODUCTION

Underground mining is one of the types of activities in the mining industry that has a level of risk of injury by a falling rock is higher than the surface mining activities. Start from affected brittle to huge rock falling can occur. Commonly, one of mine accidents in the underground mining injuries and accidents caused by falling rocks, because of closely with perimeter working area. It can collapse at any time.

Groundfalls are a major source of fatal and non-fatal injuries in underground mines in every mine based on U.S. Coal mines, compared to metal or non-metal mines are more prone to groundfall-type incidents due to the nature of the rock mass involved and fundamental differences in mining practices. According to MSHA statistics during the 10-year period 1986 – 1995, groundfall was responsible for the largest proportion (41.7%) of fatal incidents in the coal mining industry and about 50% of fatal incidents in underground metal mining. Though new developments in ground control techniques helped to improve the ground stability problem in underground mines, it remains a substantial source of human fatalities.

The groundfall was classified in underground mines commonly divided into two categories:

- Induced or intentional, meaning rock falls are purposefully caused by the mining method such as caving rock behind a longwall face, collapsing roof in a retreat room-and-pillar mine or caving rock in a block-caving hardrock mine.
- Unplanned or unintentional, meaning any rock fall in mine workings where humans could be present.

In this case, the groundfalls of interest are the “so called” unintentional falls. Incidents due to unintentional groundfalls involve fatal or non-fatal injuries to a miner or a group of miners resulting from direct impacts of dislodged rock from the surfaces of mine openings. Groundfalls in underground mines take place in innumerable geologic and operational situations. In practice, the rock mass surrounding mine openings can collapse in different forms even though primary and secondary support systems reinforce it.

A groundfall originates from two locations that have the potential to endanger miners:

- Collapse or bursting of roof rock layers
- Spalling, slabbing or bursting of rock from the sides/ribs of openings



Figure 1 Rockfall cause form slabbing with equipment damage (Geotech: 2007)

The occurrence of a ground-fall-related incident requires fulfillment of three conditions:

- The groundfall itself (ground element domain).

- Some deficiency in the support system (support element domain).
- The presence of a miner or a group of miners (human factor element domain).



Figure 2. Fatal accident cause from Rockfall (Geotech: 2007)

To devise and implement a “groundfall incident prevention plan”, it is very important that each groundfall related incident involving a human fatality or injuries receive a thorough investigation in order to understand the exact contributions of the three conditions mentioned above. Investigating the patterns and the root causes of groundfall related incidents, i.e., the etiology of the incident (fatal or nonfatal), is essential for improving risk management plans and establishing key focus areas for future control.

This paper introduces a novel technique – “Taxonomic Analysis”, which identifies root causes of an event (an incident in this case) in Pongkor Underground Gold Mine PT.Antam (Persero) Tbk., and in turn can provide future direction of corrective measures to reduce the probability of occurrence of the event. A taxonomic analysis involves systematic and scientific organization of data based on observation,

description, and classification. This paper utilizes the rock fall related incident narratives, available from the Pongkor Safety Management Database (2003-2012), to pinpoint focus areas in which preventive measures and future research activities could concentrate.



Figure 3. Rockfall cause form sliding plane (Geotech: 2007)

To maintain in order rock fall injuries do not occur again, there should be a way of finding the root cause. The first thing to do is categorize each of the injury first. First of all injuries caused by falling rocks were divided into 2 groups, there are:

- Rockfall Injuries that occur in the area have been supported
- Rockfall Injuries that occur in the area has not supported

From the group in above majority of injuries caused by falling rocks that have occurred in Pongkor Gold Mining Unit is in the region that has been supported. Then the injury is subdivided based on the cause includes:

- Rock mass failure
- Support system is not perfect
- Human activity

- The collapse of support systems

Rock mass failures are the subdivided into four types based on the geological structure of the rock mass as

- Bedding plane
- Sliding plane & infill soft material
- Hydrostatic pressure
- Excessive span

On human activities are subdivided into three sections based on the activities performed include:

- Drilling & Bolting
- Mucking
- Barring Down
- Reinforcement supporting system

In designing and finding the roots from injuries that caused by falling rocks, it is important to investigate the causes of Rockfall itself. By linking each incident cause, it will be easier in finding the root of the problem. Therefore doing classification and grouping like the above. This method is called taxonomic analysis.

## 2. METHODOLOGY

As explained above the method used is taxonomic analysis. Taxonomy is the observation and grouping of data sets in order to get a hierarchy which describes the relationship of a case. For the case this time, discuss about the injured by falling rock that occurs in Pongkor gold mine. The first taxon is based on the distribution of Rock fall injuries where occur. This division produces two branches. The first branch of the injuries caused by falling rocks occurring in the area have been supported, the second branch of the injuries that occur in areas that have not been supported. In areas that have not supported this is a reasonable case of rock collapse, therefore branches stop there. Second taxon is based on the cause of injured by falling rock that produces four

branches. The first branch is due to rock mass failure, the result of support systems are not perfect, the third is due to human activity, and the fourth is caused by collapse of the support system. After that taxon in the next row is the type of activities of the cause of the injury.

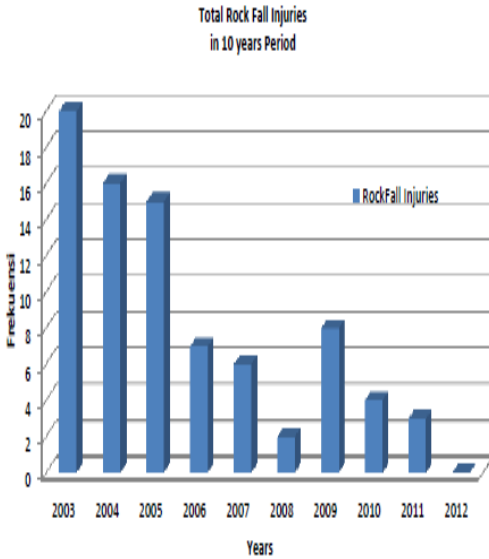
On human activities are divided into four classes, namely due to drilling, mucking, barring down and reinforcement activity.. Collapse of rock mass is divided into 4 is due to bedding plane, sliding plane, there is a weakness cause hydrostatic pressure and excessive span. Supporting system is not perfect and debris on the supporting system is divided into two parts. Inadequate supporting system and the space of installation is the second division of the buffering buffering is not perfect. While the collapse of the supporting is divided into 2 causes include inadequate in doing maintenance which includes the bolt length and fall between supporting.

## 3. DISCUSSION AND RESULT

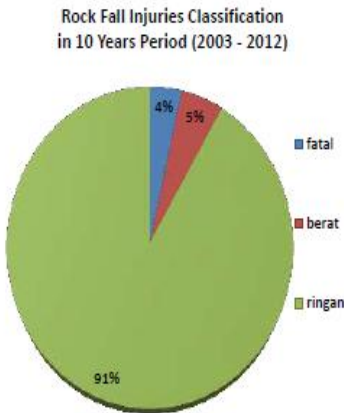
In the first graph was found that the injury caused by falling rocks experienced lowering in the period 2003 to May 2012. However in 2009 rose slightly to 8 cases, but then declined again. In 2003 injured by a falling rock reaches the 20 cases which is the highest in the period. Until May 2012 injuries caused by the fall of rocks have not been there. Total injuries caused by falling rocks during the years 2003 to May 2012 was 81 cases.

In the second graph shows that the injury caused by falling rocks in the gold mines Pongkor 91% were minor accidents, 5% severe accidents, and 4% fatality. However this can't be underestimated. In the presence of 4% which could result in fatality this needs to be a serious concern, as this effect is also in production activities. In the event of fatality in the mining area of mining activity can be stopped for later

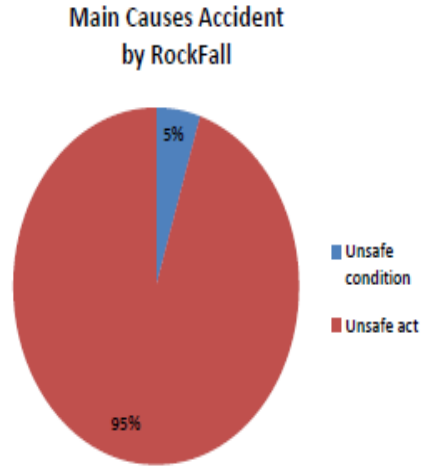
investigation. So that could interfere with the smooth production.



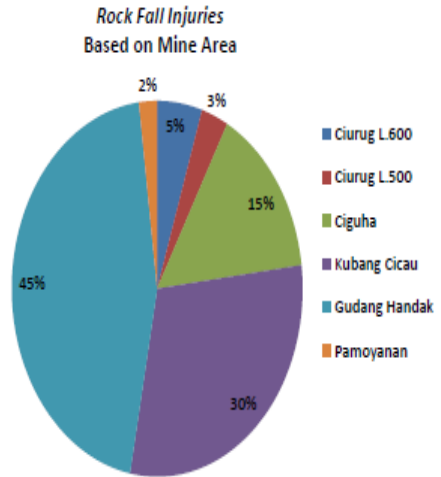
Graph 1 Rockfall injuries the period 2003 to 2012 in Pongkor Gold Mines



Graph 2 Rockfall injuries classification period from 2003 to 2012 in Pongkor Gold Mines

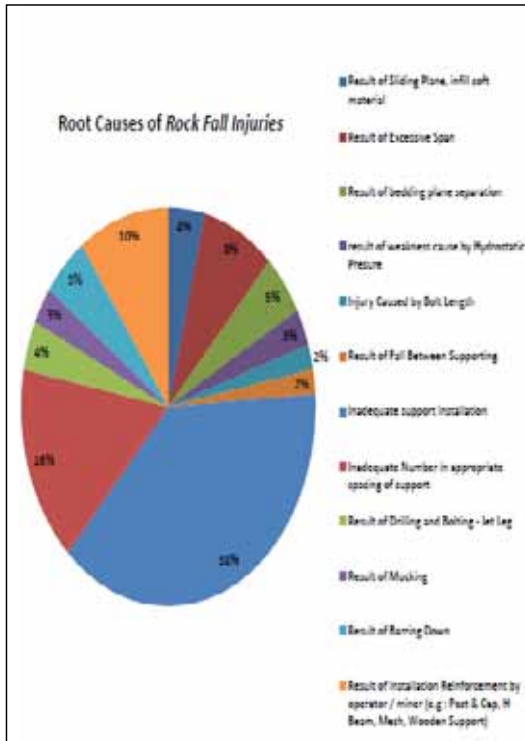
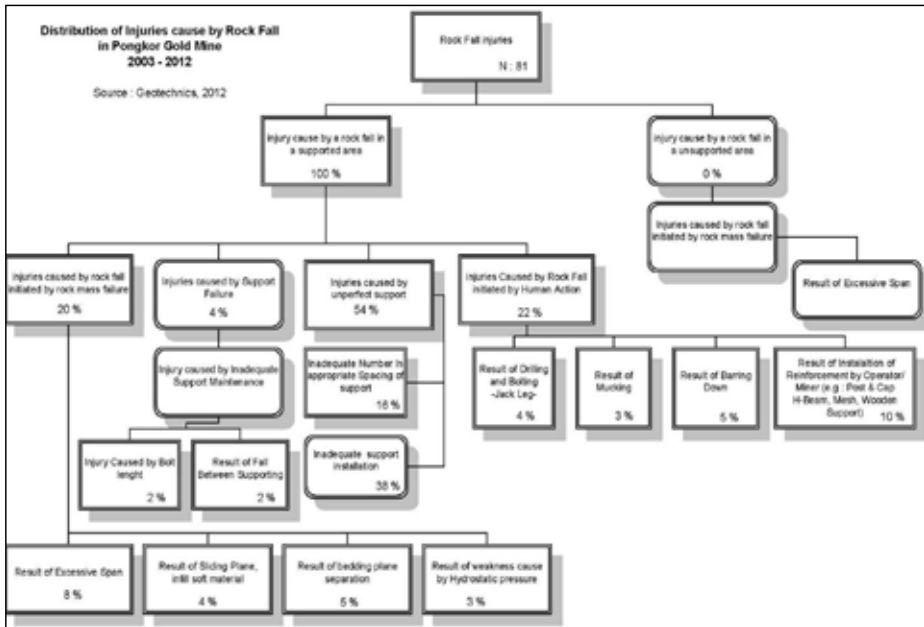


Graph 3. Main Causes of Accidents in Mining Activities in Pongkor Gold Mines



Graph. 4 Causes of rock fall injuries in Pongkor Gold Mines based on Mine Area

At the time of the analysis of the resulting taxonomy chart like Figure 1 below:



Graph. 5 Causes of rock fall injuries in UBPE Pongkor period 2003 to 2012



Clearly visible from the the chart-taxon taxon resulting in injuries due to rock fall. The first line describes the division based on place of occurrence. The second line is the cause of falling rocks and the third taxon that is the types of each taxon in the second line. From the data have been obtained, it is known that all of rock fall injuries occur in areas that the area has been supported.

In the second taxon is based on the cause of injury from falling rocks. At this stage, grouping is only focused on the areas that had been supported. This is because the majority of cases occur in areas that have been supported. In the division is divided into four classes, namely: 1) rock mass failure, 2) supporting system is not perfect, 3) human activity, and 4) supporting system failure. 54% rock fall injuries occurring in the Pongkor gold mines due to the support system is not perfect. For other factors that have 22% due to human activity, 20% due to the rock mass failure, and 4 % caused by support system failure

In the fifth graph is explained that the majority of cases of injuries caused by falling rocks in Pongkor Gold Mine occur in the rock mass grade 4 and 5.

With almost the same percentage that is 45% for rock mass class 4 and 40% in class 5 rock mass. From these data shows that the support system on the rocks grade 4 and 5 should be reviewed further to reduce the incidence of this in the future. In the next chart described the area in which Pongkor Gold mine frequently injured by a falling rock. Gudang handak and kubang cicau into place the most frequently injured. Around 40% of injuries occurred in gudang handak and 30% in kubang cicau. From this graph it is clear that gudang handak and kubang cicau into areas that are highly vulnerable to injury from falling rocks.

#### 4. CONCLUSION

Rock fall injuries are major causes for minor accident or serious injuries that occur in Pongkor Gold Mine during the period 2003 to 2012. Total cases that occurred during the period there are four times serious injury, 3 fatalities, and 74 minor accidents. However injured by rock falling continued to decline from the year 2003 reached 20 cases, and until now there is no case.

54% of the causes of the falling rock are from roof / front that are not propped up perfectly. This indicates that the monitoring of the supporting system that is necessary in order to review the injury from falling rocks do not occur again. Areas that need more attention are the areas that have rock mass class 4 and 5. From the results it is clear that 85% of injuries are from falling rocks occur in the rock mass class 4 and 5. Working area of concern is Gudang Handak and Kubang cicau. Gudang handak be the region most frequently injured by a falling rock with the percentage of 40% of all events in Pongkor Gold Mine the period 2003 to 2012

To prevent the above case does not happen again there are some actions to take are:

- Improve the competency through Training on safety, Work Location and Proper use of tools
- Ground Control Monitoring System ; Updated Mapping areas often exposed to rock fall
- Safety and UG. Geotechnis Representative New development of the supporting system in site characteristic area.
- SOP - WI Awareness enforcement,
- Structural enhancement ; safety signs in prone areas.
- Increasing coordination among the miners (feedback system)
- Reward and Punishment enforcement.

## 5. ACKNOWLEDGEMENT

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# Simulation of Underground Mine Ventilation Based on Lamé's Curves

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**ABSTRACT** The main purpose of ventilation in underground mines is to fit and control the quality and amount of air circulating there. The failure of such a system can cause irreversible damage to the health of workers and jeopardize the operation of the mine, even giving rise to great risk of explosion. Several underground mines will be opened or reopened in the coming years, stressing this issue's importance. After all, a well-designed ventilation system should address the correct calculation of air velocity profile either to the laminar or transitional regime, or even turbulent one. We present in this paper a simulation of the air movement profile employing a simplified gallery model with air profile description based in evolution of a Lamé's curves, varying their exponent  $n$  and parameter  $a$  and  $b$  (section semi-major axis and the semi-minor axis, respectively). The results show that this method can adequately predict the air volumetric flowrate in the gallery.

## 1 INTRODUCTION

Ventilation in an underground mine requires constant attention from mine operators. The main purpose of ventilation in underground mines is to fit and control the quality and amount of air circulating there. The failure of such a system can cause irreversible damage to workers' health and jeopardize the operation of the mine, even giving rise to great risk of explosion. The ventilation in underground mine should supply a flow of fresh air to all the work places in underground, in enough amounts to maintain the necessary conditions of hygiene and the workers' safety (Pinto et al., 2008). Uninterrupted air exchange in underground mine network is indispensable for safe and optimal ore exploitation.

Since definition of flow velocity profile provides important pieces of information such as pressure drop and flowrate, effort for

getting equations in order to describe fluid flow patterns is very relevant.

Equations that describe ideal profiles for laminar and turbulent ultimate flow ranges are fairly common, but the lack of equations for the profiles in transitional flow regimen hinders a significant approach of the actual air behavior in underground environment. A better definition of these profiles enables a better simulation of its behavior.

The velocity profile under laminar range in an idealized cylindrical mine opening is a paraboloid of revolution, according to Poiseuille's law. On the other hand, in fully turbulent flow regimen the fluid velocity is the same in any point inside cross section of the air pass. So, in transitional regimen (Reynolds number between 0.2 and about 1,000) velocity profile displays an intermediate behavior.

The aim of this paper is to describe the air flow velocity profile inside the transition range between the lamellar turbulent

regimens in the context of underground mine ventilation. For that it was sought appropriated parameter for a family of Lamé's curves as generatrices of velocity vectors profile.

## 2 MINING VENTILATION AT A GLANCE

### 2.1 Prolegomena

It is necessary the effective management of mine ventilation systems in the modern context, where economic, environmental and health and safety pressures are becoming ever more cost intensive. According Mine Ventilation Society of South Africa (2012), with depth increase of underground mines efficient ventilation planning and design of ventilation system are imperative. These need knowledge.

According University of Utah (2012), some topics had been covered extensively in ventilation: diesel emissions control, ventilation monitoring and control, and numerical modeling using computational fluid dynamics. Extended coverage of the first topic is the result of the increased use of diesel units in underground mines and the need to meet new regulations. For the second topic, the high level of interest results from the persistence of environmental hazards such as mine gases and dust in underground mines—particularly in coal mines. Finally, the rapid development of personal computers that can process complex fluid dynamic models quickly and efficiently has led to a rapid increase in research in this area. Currently, problems such as spontaneous combustion and air leakage through caved areas are being investigated using this tool.

Flow-through ventilation is the main ventilation circuit for the mine, according De la Vergne (2003). Air enters the mine from surface via a shaft, ventilation raise or adit. The air is distributed through the mine via internal ventilation raises and ramps, and flows are controlled by regulators and permanently mounted ventilation fans.

Ancillary ventilation system takes air from the flow-through system and distributes it to

the mine workings via temporarily mounted ventilation fans, Venturis and disposable fabric or steel ducting.

### 2.2 Ventilation Planning

The volume required to ventilate an underground mine is determined by mining engineers. It may also be regulated by the local governing body. However in some developing countries the mandated ventilation requirement may be insufficient and the mining company may have to increase the ventilation flow, in particular where ventilation may be required to cool the ambient temperature in a deep hot mine.

According De la Vergne (2003), one method to arrange ventilation is to place huge burner on the bottom of the vertical hole. Heated air rises up and the replacing air brings fresh air to the mine. In temperate climates ventilation air may need to be heated during winter months. This will make the working environment more hospitable for miners, and prevent freezing of workings, in particular water pipes. In Arctic mines where the mining horizon is above the permafrost heating may not take place to prevent melting the permafrost.

The analysis and optimization of the ventilation systems involve the use of computational simulation techniques (Pinto et al., 2008).

The estimation of airflows required within the work areas of a mine ventilation network is the most empirical aspect of modern ventilation planning. The majority of such assessments remains based on local experience of gas emissions, dust, or heat load and is still often quoted in the somewhat irrational terms of m<sup>3</sup>/s per ton of mineral output, particularly for non-coal mines (Malcom, sd).

Corrections can be applied for variations in the age of the mine, the extent of old workings, distances from shaft bottoms, depth and rates of production. However, as in all empirical techniques, the method remains valid only whilst the proposed mining methods, machinery, and geological

conditions remain similar to those from which the empirical data were evolved. Attempts to extrapolate beyond those circumstances may lead to serious errors in determining required airflows. Fortunately, simulation techniques are available to assist in assessing airflow requirements for both gassy and hot mines.

Underground mine ventilation provides a flow of air to the underground workings of a mine of sufficient volume to dilute and remove noxious gases. According De la Vergne (2003), the source of these gases is equipment that runs on diesel engines, blasting with explosives and the orebody itself. The largest component of the operating cost for mine ventilation is electricity to power the ventilation fans, which may account for one third of a typical underground mine's entire electrical power cost.

In underground mines, a ventilation system is basically a combination of pipes and ducts, fans, cooling and heating systems, and possibly air cleaning equipment. Air introduced into the mine workings via the pipes and ducts, and from natural openings and shafts into the mine workings flows through the workings, is affected by dust, gas, heat, or other constituents and is then removed (or exists naturally) from the workings (Tumbde et al, 2011).

### 2.3 Some Case Studies

Some mining operations, such as potassium in Taquari-Vassouras Mine (Brazil) require refrigeration of mine workings. Recent expansions of mines, as the Cuiabá Mine (gold, Brazil) had a share of the deployment of mine cooling stations. This question should adorn more frequent with the deepening of existing mines and the transition from opencast mining to underground. By other hand, mines such as Raglan Mine and Nanisivik Mine are designed to operate below zero degrees Celsius.

Also in Brazil, according Leite (2010), in Ferbasa, Mina Ipueira (chromite) has scaled its ventilation system in the installed power

equipment. The system works by exhausting stale air of fronts and software for circuit simulation it is used. It consists of excavations upward (raises) for air outlet, mostly open with drill raise borer, diameter 3.0 m, although it is still part of the excavations carried out with conventional system (drill and blast), with 2.10 to 2.40 m in diameter.

The total flow rate is 390 m<sup>3</sup>/s. Computing the monthly consumption of 273.6 MWh in these units, ventilation represents 40 % of total consumption of the underground mine, and energy costs represent approximately 10 % of material costs.

The work consisted of opening 230 m of adit (3 months). First dug up a pilot hole, diameter 12 ½ ", with extension of 190 m with a raise borer (1 month), followed by the extension to 3.0 m diameter (2 months); closure of access (1 month); manufacture of ventilation ports.

The reopening of the mine chimney Ipueira was considered a viable project in the views of technical and financial, with the following additional benefits: improved air quality in underground mine; increase the capacity of mine ventilation, improvement in productivity; possibility implementation of the control system of ventilation depending on demand in real time.

According Tumbde et al (2011), in Canadian mines the most common considerations are the control of diesel engine exhaust emissions and mineral dust concentrations, to within prescribed levels for humans to work in that environment. A lesser concern is the clearance of blast fumes which is normally when the mine is not occupied. In a smaller number of mine radiation and other gases, such as methane may also need consideration. A new and increasing concern as Canadian mines go deeper is the control of heat exposure to the degree that it requires more air to control the temperature of the working environment than is needed for other concerns.

Other examples are, according Mine Ventilation Society of South Africa (2012), deep gold mines at South Africa with high rock temperatures to view innovative

cooling systems such as Mponeng mine, where mining is taking place 3500 meters below surface with cooling provided by slurry ice from the surface; platinum mines such as Northam, which operates solely on hydropower and does not use any compressed air; coal mines such as the Syferfontein continuous haulage operation with 5 meters high seams; and Kloppersbos, the fire and explosion test facility operated by South Africa’s Council for Scientific and Industrial Research (CSIR).

### 3 DEVELOPING THE MODEL

#### 3.1 Theoretical Background

Fluid flow in duct is a key issue in engineering practice. As mining operation is concerned air flow throughput and

For circular ducts with diameter D (or openings of hydraulic equivalent diameter), Swamee (1993) has developed the following

$$f = \left\{ \left[ \frac{64}{Re} \right]^8 + 9.5 \times \left[ \ln \left( \frac{\varepsilon}{3.7 \times D} + \frac{5.74}{Re^{0.9}} \right) - \left( \frac{2,500}{Re} \right)^6 \right]^{-16} \right\}^{0.125} \quad (2)$$

Where:

- $\varepsilon$  – asperity or absolute wall roughness [m];
- D – diameter or twice the hydraulic radius [m];
- Re – Reynolds number ( $Re = \rho \cdot v_m \cdot D / \eta$ ) [-].

The hydraulic radius is calculated by:

$$R_h = \frac{D_h}{2} = \frac{Area}{Perimeter} = \frac{A}{P} \quad (3)$$

The average velocity (also known as spatial velocity) is obtained by:

$$v_m = \frac{Q_v}{A} \quad (4)$$

Where:

$Q_v$  – volumetric flowrate [ $m^3/s$ ].

In turn, the dynamic viscosity of air can be estimated with reasonable accuracy using the Sutherland equation (Hirschel, 2005), which is given by:

distribution in mine openings is a subset of this kind of feature.

Head loss in airways can be treated as conventional fluid resistance in pipes and ducts that is usually calculated by D’Arcy-Weisbach equation, which is done by:

$$\Delta p = \left[ \sum K_{loc} + f \times \frac{L \times P}{A} \right] \times \frac{\rho \times v_m^2}{2} \quad (1)$$

Where:  $\Delta p$  – pressure drop (head) [Pa];

$K_{loc}$  – local head losses [Pa];

f – friction factor [-];

L – airway or duct length [m];

P – “wetted” perimeter [m];

A – cross sectional area  $m^2$ ;

$\rho$  – fluid density [ $kg/m^3$ ];

$v_m$  – mean velocity of fluid [m/s].

explicit equation for calculation of friction factor given as function of Reynolds number:

$$\eta = \eta_{air} = \frac{1.458 \times 10^{-6}}{T + 110.4} \times T^{3/2} \quad (5)$$

In the preceding equation the viscosity is obtained in Pascal x seconds [Pa.s], while the temperature is given in kelvins [K].

Atkinson has based on D’Arcy equation to shape his correlated equation, quite often employed in calculation of mine ventilation systems (Montecinos & Wallace, 2010):

$$\Delta p = \left( k_{At} \times \frac{\rho}{\rho_{ref}} \right) \times \frac{L \times P}{A} \times v_m^2 \quad (6)$$

Where:

$k_{At}$  – Atkinson friction factor [ $kg/m^3$ ];

$\rho_{ref}$  – reference air density [ $kg/m^3$ ].

Reference value for air density of 1.2  $kg/m^3$  (corresponding to 15° C and 60 % of relative humidity) is used for determining standard Atkinson friction factor.

Actually fluid velocity ( $v$ ) in a duct is not a constant, but displays a vector field, with a distribution depending on flow regimen (laminar, transitional, and turbulent, in accordance with Reynolds number values). Under subcritical conditions there is a stagnant layer at wall duct (boundary layer “theory”), rendering velocity vector null. In an axisymmetric opening or duct, like one with circular cross section, the maximum velocity value ( $v_{max}$ ) occurs at center line.

So, there is a velocity profile which depends on the Reynolds number. Adopting an orthogonal Cartesian coordinate system, centered on the longitudinal axis of the cylindrical duct or airway, usually such a profile can be described by an equation like this one:

$$\frac{v}{v_{max}} = \left[ 1 - \left( \frac{x}{R} \right)^q \right]^p \quad (7)$$

Where:

- x – radial and horizontal axis[m];
- R – radius of duct or airway [m];
- q and p – exponential parameters [-].

For laminar range,  $q = 2$  and  $p = 1$  (resulting a parabola). On the other hand, for transition and turbulent range  $q = 1$  and  $1/p$  is a monotonically increasing function of Re, usually ranging from 6 to 10. The value  $p = 1/7 = 0,1428$  is quite often found in literature.

Figure 1 shows some of possible profiles in circular duct or mine opening.

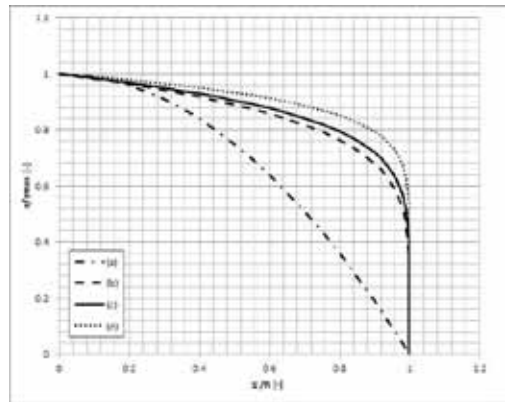


Figure 1. Relative velocity profile of Newtonian fluid flowing in axisymmetric duct: (a) – laminar; with  $q = 2$  and  $p = 1$ ; (b) – power law, with  $q = 1$  and  $p = 1/6$ ; (c) – power law, with  $q = 1$  and  $p = 1/7$  (solid line); (d) – power law,  $q = 1$  and with  $p = 1/10$ .

Figure 2 shows 3D velocity profile in laminar regimen (paraboloid of revolution, where  $y = v/v_{max}$ ,  $z/R = x/R =$  distance from the center divided by the radius).

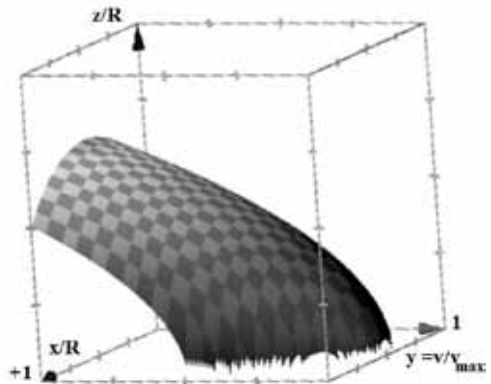


Figure 2. Relative velocity profile of Newtonian fluid flowing in cylindrical duct at low Reynolds number (oy – longitudinal axis; cross section in xoz plane).

Naturally, the relationship between the average velocity and a radial distance from the center line is expressed by an integral, which is:

$$v_m = \frac{1}{A} \int_A v(x) dx \tag{8}$$

For axisymmetric systems like circular cylindrical duct, one has:

$$v_m = \frac{1}{R} \int_0^R v(x) \times dx \cong \frac{1}{R} \int_0^R \left\{ v_{max} \times \left[ 1 - \left( \frac{x}{R} \right)^q \right]^p \right\} \times dx \text{ (assigning: } a = R \text{ for cylindrical gallery).} \tag{9}$$

It should be pointed out that in the case of turbulent flow velocity values  $v(x)$  are not stationary in time and thus represent an average (integral) in time domain.

Integration of the equation (9) for laminar flow ( $q = 2, p = 1$ ) gives a velocity ratio  $v_m/v_{max} = 0.5$ . In turn, for fluid flow with  $q = 1$  and  $p = 1/7$  the integration results the velocity ratio equal to 0.8167. Generically speaking, for turbulent flow ranging from  $5 < (I/p) < 10$  it results (Rodrigues, 2012):

$$v_m \frac{1}{R} \int_0^R v(x) \times dx \cong \frac{2 \times \left( \frac{1}{p} \right)^2}{\left( \left( \frac{1}{p} \right) + 1 \right) \times \left( 2 \times \left( \frac{1}{p} \right) + 1 \right)} \tag{10}$$

### 3.2 Lamé Curve as Velocity Profile Descriptor

Lamé curves are a geometric locus family named after the French mathematician Gabriel Léon Jean Baptiste Lamé (1795 – 1870). They can be expressed by:

$$\left( \frac{x}{a} \right)^n + \left( \frac{y}{b} \right)^n = 1 \tag{11}$$

Where  $n$  is a positive real number. This kind of curve is referred commonly as super-ellipse, since it is a generalization of ellipse equation ( $n = 2$ ).

In 3 dimensions a superellipsoid centered at the origin is described by (Weisstein, 2013):

$$\left( \frac{x}{a} \right)^n + \left( \frac{y}{b} \right)^n + \left( \frac{z}{c} \right)^n = 1 \tag{12}$$

The higher the exponent  $n$  the greater is the corner angularity. So if  $n$  is infinity a rectangle results (or a parallelepiped in 3 D instance). Within this context therefore, it is possible to describe the velocity profiles in turbulent range using Lamé curves

$$y = \sqrt[n]{1 - \left( \frac{x}{R} \right)^n} \tag{13}$$

Besides serving to describe the velocity profile in pipes and openings of circular section, a generalized Lamé curve can also be applied to describe the velocity profile of a rectangular duct or opening, especially suitable for galleries with rounded corners.

The axis  $oy$  stand for velocity vector direction and  $x$  and  $z$  for the shape of vertical cuts (curves of isovalues of  $v$ ) as far as the 3D velocity profile is concerned

A formula generalized to take into account an axisymmetric airway is this one:

$$\left( \frac{x}{a} \right)^m + \left( \frac{y}{b} \right)^m + \left( \frac{z}{a} \right)^m = 1 \tag{14}$$

The parameter is constant and represents the flow distribution aspect influencing the morphological feature of the superellipsoid (since:  $y = v$ ).

In fact the parameter  $m$  is a function of velocity (ie: ultimately depends on Reynolds number). It is linked to shape evolution of the isovalues curves, starting from the value used to describe the airway or gallery cross section ( $m = m_{gall}$ ) to  $m = 2$  (circle of radius approximately null at apical point of the 3D velocity profile, because semi-axis is  $a$ , the same for both  $x$  and  $z$  coordinates).



## 4 RESULTS

Adopting  $p = 1/7$  and considering that the velocity ratio is 0.81667,  $n$  could be estimated. The value  $n = 2.23$  does result in the same ratio:  $v/v_{max} = 0.8167$ .

For a airway with circular cross section the geometric locus becomes a superellipsoid of revolution. Putting the  $y$  isovalues curves into equation (14) finally results the following expression for mathematical model.

$$\left( \frac{x}{a \times \sqrt[m]{\left( \frac{v_{max}^n - v^n}{v_{max}^n} \right)}} \right)^m + \left( \frac{z}{a \times \sqrt[m]{\left( \frac{v_{max}^n - v^n}{v_{max}^n} \right)}} \right)^m = 1 \quad (15)$$

Considering a gallery of approximately square section  $m_{gall} = 4$  seems a good estimate for the usual mine opening silhouette.

In a first approach we can adopt a linear morphological evolution with respect to the distance from the walls. So:

$$m(y) = m(v) = m_{gall} - \frac{(m_{gall} - 2.0) \times v}{v_{max}} \quad (16)$$

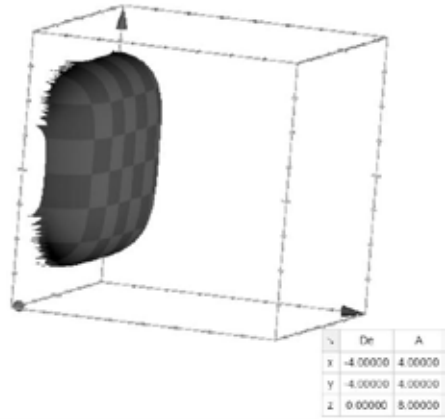


Figure 3. Velocity profile of air flowing in rectangular duct (airway, drift or gallery) at high Reynolds numbers.

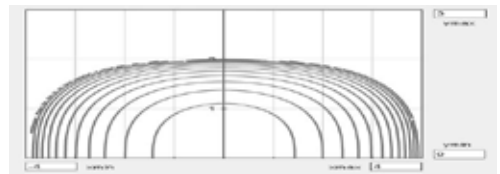


Figure 4. Contour map (top vision) of the velocity profile respective to previous figure.

## 5 CONCLUSIONS

It is important to promote engineering interest and technological progress in the ventilation of mines, tunnels and other subsurface openings. Mines that will operate for the next few decades are being designed today and with the current pace of technological change, significant new developments in mine ventilation can be expected.

The mining industry is under unprecedented technical, commercial and social pressures to continuously improve its performance at all. It is an industry that, globally, has always been cyclical: within

the global economy there are short good times followed by extended periods of low commodity prices; there is increasing shareholder demand for performance in all parts of the cycle; and evolution and operating excellence including outstanding technological endeavor have become essential for survival.

The construction of models is one form of developments to study and increase knowledge about this theme. It is seen, from the above assumption and construct that Lamé curves can help the modeling effort of mine ventilation systems.

Finally it should be borne in mind that, in possession of velocity profile, the mining engineer can easily calculate the volumetric flowrate and head losses inherent to the ventilation process.

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## Yeraltı Madenciliğinde Bilgisayar Kontrollü Ayarlı Hava Kapıları ile Hava Akımının Düzenlenmesi

### *Regulation of Underground Mining Through Computer Controlled Air Regulators*

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**ÖZET** Yeraltı madenciliğinde havanın kontrollü dağılımı, iş güvenliği ve işçi sağlığı açısından oldukça önemlidir. Bu amaçla uygulamada kullanılan en yaygın yöntem ayarlı hava kapılarıdır. Son yıllarda, yeraltı ocak havalandırmasında bilgisayar destekli sistemler ve sensörler yaygın olarak kullanılmaya başlanmıştır. Genellikle uzaktan gözlem veya erken uyarı adı verilen bu sistemler ile, yeraltı ocaklarında gerçek zamanlı hava akımı ve diğer çevresel parametreler hassas olarak izlenebilmektedir.

Bu çalışmada, bilgisayar kontrollü ayarlı hava kapılarının kullanımı ile ocak içindeki havanın kontrollü olarak dağılımı amaçlanmıştır. Bu amaçla, bilgisayar kontrollü ayarlı hava kapısı sisteminin tasarımı, imalatı ve yazılımı gerçekleştirilmiş ve daha sonra sistemin testleri yapılmıştır. Yapılan testlerden sonra, sistemin yeraltı madenciliğinde havanın kontrollü olarak dağılımını başarı ile gerçekleştirebileceği anlaşılmıştır. Türkiye’de yeraltı madenciliğinde henüz uygulaması bulunmayan bilgisayar kontrollü ayarlı hava kapılarının geliştirilmesiyle, yeraltı ocaklarında hava akımının hassas olarak düzenlenmesi sağlanabilecek ve böylece işyeri emniyeti artırılacaktır.

**ABSTRACT** Regulation of air flow in underground mining is very important for both health and safety in mines. In practice, the most common method employed for this purpose is the use of air regulators. In recent years, computer controlled systems and sensors have extensively been used in mine ventilation. The real time air flow and the other environmental parameters can precisely be monitored in underground mines through these systems which are usually called remote sensing or early warning systems.

In this research, it was aimed to obtain controlled distribution of air flow in underground mines by the use of computer controlled air regulators which can be integrated to a remote monitoring system of underground mines. For this purpose, the design, manufacture and the software of the computer controlled air regulator system was accomplished and then the tests of the system were carried out. After the tests, it was realized that the system can successfully be used for the controlled distribution of air flow in underground mining. By the development of computer controlled air regulators, which are not currently used in Turkey, the desired regulation of air flow can accurately be achieved and hence safety of workplaces be increased.

## 1 GİRİŞ

Günümüzün gelişen bilgisayar teknolojisi ile birlikte yeraltı ocaklarında bilgisayar kullanımı gittikçe artmaktadır. Üretim ve verimliliği artırmak, daha güvenli ve sağlıklı bir çalışma ortamı sağlamak amacıyla mekanizasyon, otomasyon ve bilgisayar kontrollü sistemler daha yaygın kullanılmaya başlanmıştır (Elevli, 1994). Özellikle yeraltı ocaklarında çalışanların sağlığını ve güvenliğini tehdit eden, zararlı ve patlayıcı özellikteki gaz ve toz birikimlerinin sürekli olarak izlenmesi büyük önem taşımaktadır. Ülkemiz kömür madenciliğinde bilgisayar kontrollü uzaktan gözlem sistemlerinin yaygınlaşmaya başlaması 80'li yıllarda olmuştur. Günümüzde ise, teknolojideki gelişime bağlı olarak, gözlenen veri çeşitliliği, veri hassasiyeti, veri iletimi ve veri değerlendirilmesindeki artışlara paralel olarak, otomatik kontrol sistemleri de gelişerek iş güvenliğinin sağlanmasını kolaylaştırmıştır (Koçal,1996).

Bilgisayar kontrollü uzaktan gözlem sistemlerinin çoğu üretim ile ilgili olmasına rağmen, bunlar üretimi sağlayan makineleri, aletleri ve üretim ortamını izleyerek gerekli verileri toplar ve analiz için bir merkeze iletirler. Merkeze iletilen bilgiler derlenerek analiz edilir ve bu bilgiler ışığında üretim sistemi veya ortamının verimini artırıp güvenli ve sağlıklı bir çalışma için gerekli kararlar alınır. Çoğu durumlarda elde edilen veriler kazaların önceden bilinerek, tedbirlerin alınması için uzmana zaman kazandırması nedeniyle, madencilikte bu sistemler "erken uyarı sistemleri" olarak da isimlendirilmektedir (Koçal,1996). Madencilikte uzaktan gözlem ve kontrol sistemleri, ilk zamanlar sadece ocak yangınları için geliştirilen basit uyarı ve ölçme sistemleri olarak başlamış, ancak çeşitli aşamalardan geçerek günümüzde madenciliğin vazgeçilemez bir parçası haline gelmiştir. Madencilikteki gözlem sistemlerinin gelişimi aşağıdaki aşamalarla değerlendirilebilir (Elevli, 1994; Darron et al., 1995):

1. Değişik yeraltı parametrelerinin elde taşınan aletlerle ölçülmesi (örneğin metan ve hava hızının ölçülmesi gibi)
2. Elektromekanik izleme sistemleridir. Buna örnek olarak vantilatör performansının ve kömür kazı makinelerinin izlenmesi
3. Her türlü çevresel parametrenin hassas sensörler ile ölçülüp, veri iletim hatları ile merkezi bilgisayara iletilip, analiz edildiği sistemler

Yeraltı madenciliğinin en önemli unsurlarından birisi işçi sağlığı ve iş güvenliğini doğrudan ilgilendiren ocak havası veya ocak havalandırmasıdır. Havalandırmanın başlıca amacı; yeraltında çalışan işçilere yeterli ve temiz hava temin etmek, patlayıcı/zararlı gazların ve tozların ocak havasındaki konsantrasyonunu düşürmek ve bunları ocak dışına çıkarmak, derin ocaklarda oluşan ısıyı düşürerek uygun ocak iklimini sağlamaktır. Özellikle yeraltı kömür madenciliğinde büyük can ve mal kayıplarına neden olan metan ve kömür tozu patlamaları ile ocak yangınları konusunda önceden gerekli tedbirleri almak ve bu patlama ve yangınların oluşmasını önlemek son derece önemlidir. Bu ise ancak ocak ortamının sürekli olarak izlenmesi ve ocak verilerine göre havalandırmanın doğru planlanması ile sağlanabilir.

Bir yeraltı ocak havalandırma planlaması yapılırken, öncelikle çalışan işçi sayısı, olası zararlı gaz ve toz birikimleri gibi farklı durumlar göz önüne alınarak ihtiyaç duyulan hava miktarı belirlenir. Belirlenen bu hava ihtiyacı için, ocak direnci de göz önüne alınarak uygun vantilatör seçimi gerçekleştirilir. Ancak ocağa gönderilen havanın ocak içindeki dağılımı, hava yollarının dirençlerine göre gerçekleşmektedir (normal dağılım). Havanın normal dağılımı ile üretim bölgelerinde arzu edilen hava miktarları her zaman gerçekleşemez. Bu durumda yetersiz havanın neden olacağı işçi sağlığı ve iş güvenliği ile ilgili sorunlarla karşılaşılabilir. Dolayısıyla, bu durumlarda havanın ocak için kontrollü olarak dağıtılması gereklidir.

Havanın kontrollü dağıtımı için en çok tercih edilen yöntem, havanın gereğinden fazla gittiği düşünülen hava yollarına tesis edilen "ayarlı hava kapısı/regülatör" kullanımudur (negatif ayarlama). Böylelikle, havanın daha az gittiği bölgelere daha fazla hava gönderilebilmektedir. Ancak ayarlı hava kapılarının arzu edilen hava dağılımını gerçekleştirecek şekilde tasarlanması, inşa edilmesi ve değişen koşullara göre yeniden hızlı ve güvenilir olarak düzenlenmesi kolay değildir. Bu nedenle ayarlı hava kapılarının hızlı ve güvenli olarak inşa edilmesi ve değişen şartlara göre yeniden ayarlanmasında otomatik bir sistemin kullanılabilmesi önem arz etmektedir.

Bu çalışmada, yeraltı madenciliğinde paralel hava yollarında havanın arzu edildiği şekilde kontrollü olarak dağılımının gerçekleştirilmesi hedeflenmiştir. Bu amaçla, yeraltı şartlarına uyum sağlayacak ve havanın kontrollü olarak dağılımını gerçekleştirecek otomatik bir kapının tasarlanması/imalatı sağlanmış ve sensörlerden gelen verilere göre kapıyı hareket ettirecek bir bilgisayar yazılımı geliştirilmiştir.

## 2 HAVA AKIMININ DÜZENLENMESİ

Yeraltı madencilik çalışmalarında, iş güvenliğini sağlayabilmek için, ocağa gönderilen havanın yeraltında belirli bir düzen içerisinde dolaştırılması, hem hava miktarı hem de hava kalitesi açısından gerekli kontrollerin yapılması gerekmektedir. Ocağa giren hava, yeraltında çeşitli yollara dağılır. Bu dağılımının arzu edildiği şekilde gerçekleştirilebilmesi ve istenilen yere istenilen miktarda havayı gönderebilmek için yeraltı madenciliğinde farklı düzeneklere ihtiyaç duyulmaktadır. Bu düzeneklerin başlıcaları şunlardır (Yalçın vd., 1999);

- Hava kapıları,
- Hava perdeleri,
- Hava köprüleri,
- Ayarlı hava kapıları

### 2.1 Hava Kapıları

Havalandırmada temel prensip, mümkün olduğu kadar bağımsız ve paralel devreler

oluşturmak ve bu devrelere gerektiği kadar hava göndermektir. Ocak yollarının uzunluk ve kesitleri ayarlanarak, istenilen miktardaki hava dağıtımı teorik olarak yapılabilir. Uygulamada ise; bu iş çok güç ve hatta olanaksızdır. Bağımsız yolları ayırmak, hava kaçaklarını ve kontrolsüz kısa devreleri önlemek, bir yolun direncini ve dolayısıyla bu yola paralel yoldaki hava akımını arttırmak için hava kapılarına gereksinim duyulmaktadır. (Yalçın vd., 1999; Ayvazoğlu, 1986).

### 2.2 Hava Perdeleri

Hava perdeleri, hava kapılarının kullanımının uygun olmadığı yerlerde kullanılır. Ayrıca bir galeri içinde hava giriş ve çıkışlarını ayırmak içinde kullanılırlar. Ancak, verimsiz bir yöntemdir. Bunun için hava boruları tercih edilmektedir. (Ayvazoğlu, 1986).

### 2.3 Hava Köprüleri

Hava köprüleri, giriş (temiz) ve dönüş (kirli) havayolları kesiştiğinde, hava akımına engel olmadan bu iki yolu birbirinden ayırmak için kullanılan yapılara denir (Ayvazoğlu, 1986; Thorp, 1989; Stanley et al. 1982). Hava köprüleri, genellikle kömür ocakları havalandırmasında kullanılır. Bu ocaklarda giriş ve dönüş havasının birbirleri ile karşılaşmaktadır. Hava köprülerinde düzgün inşa edilmedikleri takdirde hava kaçağı söz konusudur. Hava kaçağını azaltmak için hava köprüleri beton, tuğla veya galveniz ile kaplanmış demirden inşa edilirler (Thorp, 1989). Hava köprüleri dayanıklı olmalıdır. Hava akımına karşı direnci az olmalı, giriş ve dönüş havası birbirini etkilememelidir (Ayvazoğlu, 1986).

### 2.4 Ayarlı Hava Kapıları

Paralel yollardaki farklı hava gereksinimlerinden dolayı hava miktarının ayarlanması ayarlı hava kapıları ile yapılır (Yalçın vd., 1999; Ayvazoğlu, 1986). Sabit hava kapılarından temel farkı, üzerlerinde kesiti ayarlanan ve hava geçişine olanak sağlayan pencereler bulunmasıdır. Burada önemli olan gerekli hava dağılımını gerçekleştirebilmek üzere ayarlı hava kapıları üzerindeki pencerenin açıklığının

doğru hesaplanabilmesi ve daha sonraki olası değişikliklere de uyarlanabilmesidir. Konu daha detaylı olarak aşağıda irdelenecektir.

### 3 HAVANIN KONTROLLÜ OLARAK DAĞITILMASI

Ocağa gönderilen hava, yolların dirençlerine bağlı olarak doğal bir dağılım gösterir. Bu işlem “serbest dağılım” olarak bilinmektedir. Ancak bu dağılım, genellikle istenilen hava dağılımını yansıtmaz. Başka bir deyişle gerekli yerlerde yeterli miktardaki havayı sağlamaktan uzaktır. Bu nedenle havayı yeraltında değişik yerlere kontrollü olarak göndermek gerekmektedir. Hava miktarını ayarlama işlemini aşağıdaki yöntemlerle yapılır (Güyağüler, 1991).

- Hava miktarı artırılması istenen yolun direncini azaltarak (pozitif ayarlama),
- Hava miktarının artırılması istenen yola yardımcı vantilatör koyarak,
- Hava miktarının azaltılması istenen yola ayarlı hava kapısı (regülatör) koyarak (negatif ayarlama)

### 3.1 Hava Yolu Direncinin Değiştirilmesi (Pozitif Ayarlama)

Hava yolunun direncinin azaltılması, ya var olan havayolunun kesit alanının artırılması veya paralel bir hava yolunun açılmasıyla ya da sürtünme katsayısının azaltılmasıyla sağlanabilir. Galeri kesit alanının artırılması başlı başına bir kazı işidir. Havayolunun direncini azaltmak için sürtünme katsayısının azaltılması ise, tahkimat gibi hava akışını engelleyen unsurların hava yolu içinden kaldırılması, galeri yüzeyinin pürüzlerinin sıvanarak giderilmesi ya da galeri içinin temizlenmesi yoluyla gerçekleştirilebilir. Görüldüğü gibi havayolunun direncinin azaltılabilmesi için kullanılan yöntemlerin çoğu masraflı ve zahmetli bir iş olup, kontrollü hava dağılımında ender olarak uygulanmaktadır. Ancak, bazen ocaktan geçen toplam hava miktarını arttırmak için ana girişlerde, örneğin kuyularda bu yöntem uygulanmaktadır (Güyağüler, 1991).

### 3.2 Yardımcı Vantilatör Kullanımı

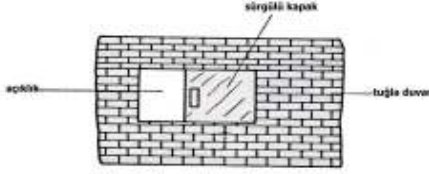
Bir havayolundan geçen hava miktarını arttırmak için, bu havayolu üzerine yardımcı bir vantilatör yerleştirmek de mümkündür. Bu durumda, direnci belli olan bir havayolu üzerinden, istenen hava miktarını geçirebilmek için gerekli basınç düşümünü karşılayacak bir yardımcı vantilatör seçilerek sorun çözülür (Yalçın vd., 1999). Genellikle yardımcı vantilatör, yüksek dirençli havayoluna konmalıdır (Güyağüler, 1991). Yardımcı vantilatörlerin yeraltında ocak içine konması özellikle kömür madenciliğinde iyi bir uygulama değildir. Ancak zorunlu durumlarda gerekli önlemler alınarak ocak içinde kullanılabilir. Pratik olarak 250-1000 Pa depresyonda dakikada 1000 m<sup>3</sup>'e kadar hava akımı bu tür vantilatörler ile sağlanabilmektedir. Bu işlem için basınç eğrisi oldukça dik olan vantilatörlerin kullanılması, değişik dirençlerde birbirine yakın hava miktarı sağlaması açısından avantajlıdır. Ancak genelde bu vantilatörler çok düşük verimde çalışmak zorunda kalmaktadır. Bu nedenler ayarlanabilir kanatlı aksiyal tip vantilatörlerin kullanılması problemi kısmen çözecektir. Yardımcı vantilatör, ana vantilatör üzerindeki yükü azaltmakta ve az bir enerji tüketimi ile istenen hava dağılımı sağlanmaktadır (Güyağüler, 1991).

### 3.3 Ayarlı Hava Kapılarının Kullanılması (Negatif Ayarlama)

Ayarlı hava kapısı, üzerinde belirli kesitte bir pencere açıklığı olan bir hava kapısı olup, “regülatör” olarak da isimlendirilir (Önce, 1986). Regülatör, gerekli hava miktarının geçişine izin veren hava kapısı üzerinde oluşturulan ve kesitleri değişen açıklıklardır (Thorp, 1989). Regülatör, pencere açıklığına bağlı olarak yerleştirildiği kolun direncini artırır, dolayısıyla koldan geçen hava miktarının artmasına neden olur (Önce, 1986)

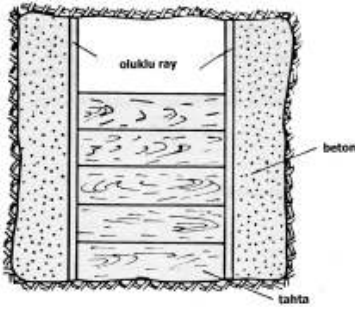
Regülatörler, sabit veya ayarlanabilir tiptedirler (Thorp, 1989; Stanley et al., 1982). Sürgülü regülatör, hareket eden tip regülatördür (Şekil 3.1). Gerekli açıklık ayarlanabilir. Sürgülü regülatör hem metal

madenciliğinde hem de kömür madenciliğinde kullanılır (Thorp, 1989).



Şekil 3.1. Sürgülü regülatör

Ağaçtan yapılmış regülatör, (Şekil 3.2) yüksek basınç farkının olduğunda kullanılan bir regülatör tipidir. Gerekli açıklık oluklu raydan parça tahta çıkartılıp eklenmesi ile ayarlanır (Thorp, 1989).



Şekil 3.2. Tahta tipi regülatör

Tüm regülatör tiplerinde gerekli ayarlamalar sağlam imal edilmeli, ayarlanabilir regülatör tipi kilitlenebilmelidir. Regülatör boyutlarını belirlemek için regülatörün konulacağı yolda oluşturacağı şok kaybının önceden belirlenmesi gerekir (Thorp, 1989).

#### 4 LABORATUAR ÖLÇEKLİ SİSTEM TASARIMI VE DETAYLARI

Bu çalışmada, laboratuvar ölçekli bilgisayar kontrollü bir ayarlı hava kapısının mekanik-elektronik tasarımı gerçekleştirilmiştir. Sistemin gerçekleştirilme aşamaları aşağıda özetlenmiştir:

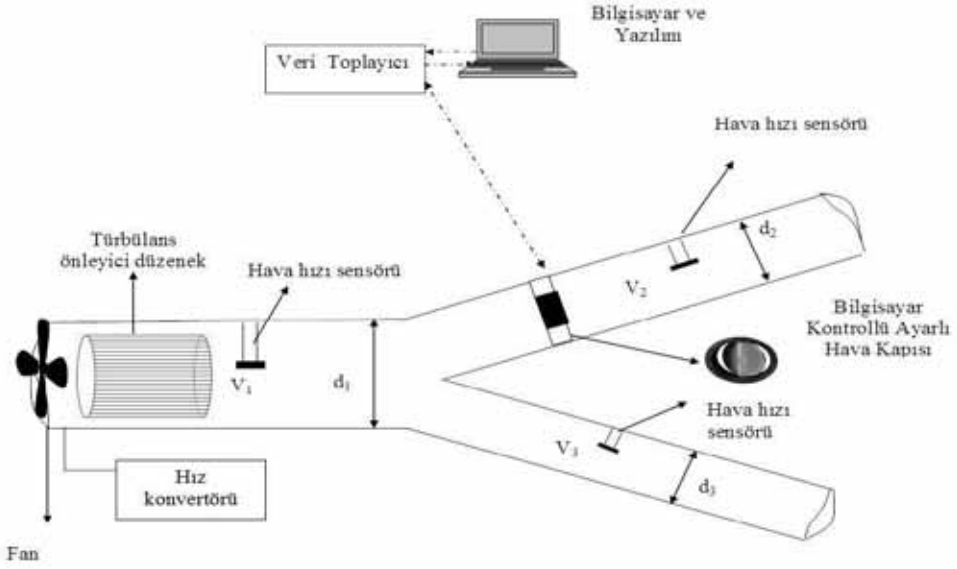
1. Öncelikle yeraltı şartlarını modelleyen bir hava yolu sistemi tasarlanarak, imal edilmiştir.
2. Elektronik olarak kontrol edilebilen bir ayarlı hava kapısı, bu hava yolu sistemine monte edilmiştir.
3. Hava hızını ölçerek, dijital olarak ortama aktarabilecek sensörler belirlenmiş ve hava yolu sistemine monte edilmiştir.
4. İstenen hava akışını sağlayacak bir üfleyci vantilatör sisteme monte edilmiştir.
5. Sistemden gelen verileri değerlendirerek arzu edilen havanın dağılımını gerçekleştirecek bir yazılım geliştirilmiştir.
6. Yazılım ve donanımdan oluşan sistemin testleri yapılmıştır.

#### 5 BİLGİSAYAR DESTEKLİ AYARLI HAVA KAPISI YAZILIMI

Bilgisayar destekli ayarlı hava kapısı sisteminin genel görünüşü, Şekil 5.1.'de verilmiştir. Bilgisayar destekli ayarlı hava kapısı ile kontrollü hava dağılımı için veri toplama cihazı ile uyumluluğu göz önüne alınarak "Visual C#" programlama dili kullanılmıştır. Ayrıca verilerin veri tabanına kaydedilmesi için "postgresql" programı kullanılmıştır. Bilgisayar destekli ayarlı hava kapısı ile kontrollü hava dağılımı için yazılan programın kullanılması ile ilgili bilgiler aşağıda verilmiştir.

Programın ilk çalışması ile kullanıcı ekranına "Açılış Formu" gelmektedir. Şekil 5.2'de bu form verilmektedir. Bu formda kullanıcıya 3 adet seçenek (senaryo) sunulmaktadır.

1. Tüm Yollardan Geçen Hava Miktarını Ayarlama,
2. Sadece 2. Yoldan (regülatör yolu) Geçen Hava Miktarını Ayarlama,
3. Sadece 3. Yoldan (regülatör olmayan yol) Geçen Hava Miktarını Ayarlama,

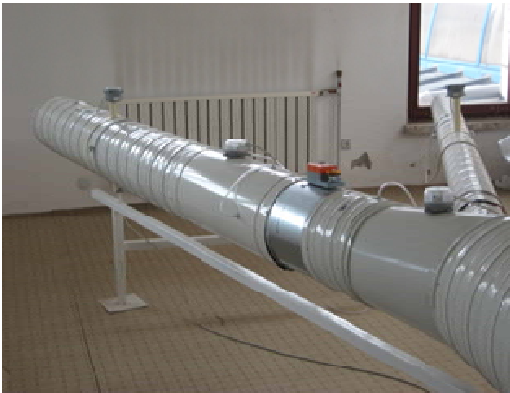


Şekil 4.1. Bilgisayar destekli ayarlı hava kapısı deney düzeneği

Kullanıcı tarafından eğer “Tüm Yollardan Geçen Hava Miktarlarını Ayarlama” seçilirse kullanıcı ekranına Şekil 5.3.’te verilen Form 2 açılmaktadır.

Bu formda hava hızı ve debi ölçümleri, 1. yoldan geçecek hava miktarı, 2. yoldan geçecek hava miktarı, 3. yoldan geçecek hava miktarı, tarih, zaman, kapı konumunun açıklığının grafiksel görünümü ve kapı açıklık derecesi gösterilmektedir.

Bu forma kullanıcı tarafından 2. ve 3. yoldan geçecek hava miktarları girilir. Sistem 2. yoldan ve 3. yoldan geçen hava miktarlarını ayarlamak için ayarlı hava kapısını kapatmaya başlayacaktır. Aynı zamanda 2. ve 3. yoldan geçen hava miktarlarının toplamı 1. yoldan geçen hava miktarına eşit olmasından dolayı vantilatörü de 1. yoldan geçecek olan miktara göre çalıştırmaya devam edecektir.



Şekil 5.1. Bilgisayar destekli ayarlı hava kapısı sistemi genel görünüşü



Form1

REGULATION OF AIR FLOW IN UNDERGROUND MINING THROUGH  
COMPUTER CONTROLLED AIR REGULATORS

Adjusting the amount of air passing through all pipes

Adjusting the amount of air passing through 2nd pipe only ( the pipe with regulator)

Adjusting the amount of air passing through 3 rd pipe only ( the pipe without regulator)

OK CLEAR EXIT

Şekil 5.2. Açılış formu

Form2

Air Velocity and Air Flow Measurement

Sensor-1  
2.26989 m/sec  
17 m3/min

Sensor-2  
2.39705 m/sec  
10 m3/min

Sensor-3  
5.81895 m/sec  
7 m3/min

Amount of Air Required in Pipes (m3/min)

Pipe 1 17  
Pipe 2 10 10  
Pipe 3 7 7

ENTER

Date and Time  
Date : 18.10.2012  
Time : 13:48:40

Regulator Position

48 °

DATABASE

Start Back Exit

Şekil 5.3. Form 2 (Tüm yollardan geçen hava miktarlarını ayarlama)

Kullanıcı tarafından eğer “Sadece 2. Yoldan Geçen Hava Miktarını Ayarlama” seçilirse kullanıcı ekranına Şekil 5.4’te verilen Form 3 açılmaktadır. Bu formda hava hızı ve debi ölçümleri, 2. yoldan geçecek hava miktarı, tarih, zaman, kapı konumunun

açıklığının grafiksel görünümü ve kapı açıklık derecesi gösterilmektedir. Kullanıcı 2. yoldan geçecek hava miktarını girdiğinde kapı bu yoldan geçecek hava miktarını ayarlayıncaya kadar ayarlı hava kapısını açık kapatmaktadır.

The screenshot shows a software window titled "Form3" with a blue background. It is divided into several sections:

- Air Velocity and Air Flow Measurement:** Contains three sensor data boxes. Sensor-1 shows 1.91903 m/sec and 14 m3/min. Sensor-2 shows 2.44324 m/sec and 10 m3/min. Sensor-3 shows 2.96330 m/sec and 4 m3/min.
- Amount of Air Required in Pipe (m3/min):** A section with a "Pipe 2" label and two input fields, both containing the value "10". Below them is an "ENTER" button.
- Date and Time:** A section with "Date : 18.10.2012" and "Time : 14:44:36" displayed.
- Regulator Position:** A circular gauge with a vertical line and a needle pointing to the value "73". Below the gauge is a "DATABASE" button.
- Control Buttons:** At the bottom, there are three buttons: "Start", "Back", and "Exit".

Şekil 5.4. Form 3 (Sadece 2. yoldan (regülatör yolu) geçen hava miktarını ayarlama)

Kullanıcı tarafından eğer “Sadece 3. Yoldan Geçen Hava Miktarını Ayarlama” seçilirse kullanıcı ekranına Şekil 5.5’te verilen Form 4 açılmaktadır. Bu formda hava hızı ve debi ölçümleri, 3. yoldan geçecek hava miktarı, tarih, zaman, kapı konumunun açıklığının grafiksel görünümü ve kapı açıklık derecesi gösterilmektedir. Kullanıcı 3. yoldan geçecek hava miktarını girdiğinde kapı bu yoldan geçecek hava miktarını ayarlayıncaya kadar ayarlı hava kapısını açıp kapatmaktadır.

Elde edilen veriler “postgresql” veri tabanı programı sayesinde veri tabanına kaydedilmektedir. Hava debisi, kapı

konumu, istenilen hava miktarları, tarih, zaman verileri veri tabanına kaydedilmektedir.

Formlardaki veri tabanı butonuna basıldığında ekrana Şekil 5.6.’da verilen Form 5 gelmektedir. Bu formda kullanıcı her yoldaki hava debisini, sisteme girilen istenilen hava miktarını, kapı konumunu, tarih, zaman gibi veri tabanına kaydedilen verileri ekrana getirebilmektedir.

**Form4**

**Air Velocity and Air Flow Measurement**

Sensor-1  
 3,20153 m/sec  
 24 m<sup>3</sup>/min

Sensor-2  
 2,80603 m/sec  
 12 m<sup>3</sup>/min

Sensor-3  
 10,39257 m/sec  
 12 m<sup>3</sup>/min

**Amount of Air Required in Pipe (m<sup>3</sup>/min)**

Pipe 3 12 12

ENTER

**Date and Time**

Date : 18.10.2012  
 Time : 11:46:31

**Regulator Position**

41 °

DATABASE

Start Back Exit

Şekil 5.5. Form 4 (Sadece 3. Yoldan (regülötör olmayan yol) Geçen Hava Miktarını Ayarlama)

**Form5**

01.08.2012 01.11.2012

Air Velocity Data Exit

	Number of Line	Pipe 1	Pipe 2	Pipe 3	Amount of Air Required in Pipe-1 (m <sup>3</sup> /min)	Amount of Air Required in Pipe-2 (m <sup>3</sup> /min)	Amount of Air Required in Pipe-3 (m <sup>3</sup> /min)	Regulator Position (°)	date	time
	13	27	21	6	27	18	9	90	17.10.2012	14:33:24
	14	27	21	6	27	18	9	90	17.10.2012	14:33:25
	15	27	21	6	27	18	9	90	17.10.2012	14:33:26
	16	27	21	6	27	18	9	90	17.10.2012	14:33:27
	17	28	22	6	27	18	9	89	17.10.2012	14:33:28
	18	28	22	6	27	18	9	89	17.10.2012	14:33:29
	19	28	22	6	27	18	9	89	17.10.2012	14:33:30
	20	28	22	6	27	18	9	89	17.10.2012	14:33:31
	21	28	22	6	27	18	9	87	17.10.2012	14:33:32
	22	28	22	6	27	18	9	87	17.10.2012	14:33:33
	23	28	22	6	27	18	9	87	17.10.2012	14:33:34
	24	28	22	6	27	18	9	87	17.10.2012	14:33:35
	25	28	22	6	27	18	9	87	17.10.2012	14:33:36

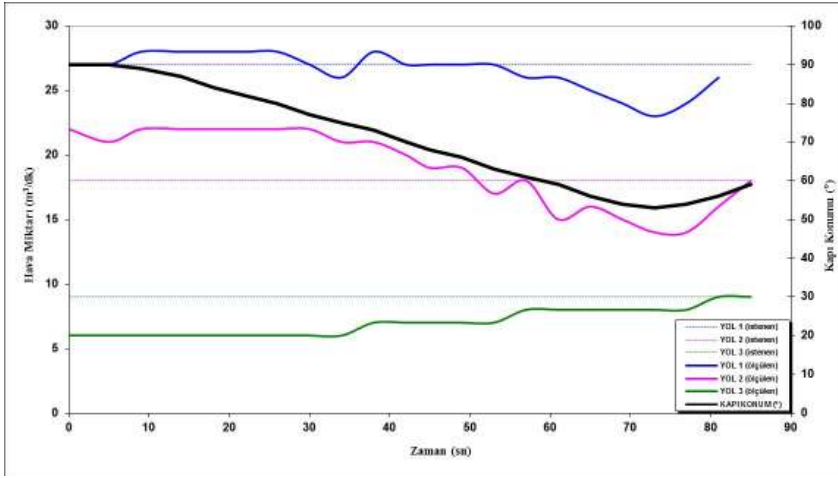
Şekil 5.6. Form 5 (veritabanı ile ilgili form)

## 6 DENEYSEL SONUÇLAR

### 6.1 Birinci Senaryo Uygulamasına Ait Sonuçlar

Sisteme tüm yollardan geçecek hava miktarları girilmiştir. 1. yol için 27 m<sup>3</sup>/dak, 2. yol için 18 m<sup>3</sup>/dak, 3. yol için ise 9 m<sup>3</sup>/dak hava miktarı girildiğinde ayarlı hava kapısı 1 dakika 26 saniye süre içinde kapıyı 31 °

kapatarak istenen 1. yol için 27 m<sup>3</sup>/dak, 2. yol için 18 m<sup>3</sup>/dak, 3. yol için ise 9 m<sup>3</sup>/dak hava miktarlarına ulaşmıştır. Şekil 6.1'de, bu uygulamaya ait grafik verilmiştir. Grafikte zamana bağlı olarak hava miktarı ve kapı konumunun değişimleri verilmiştir.



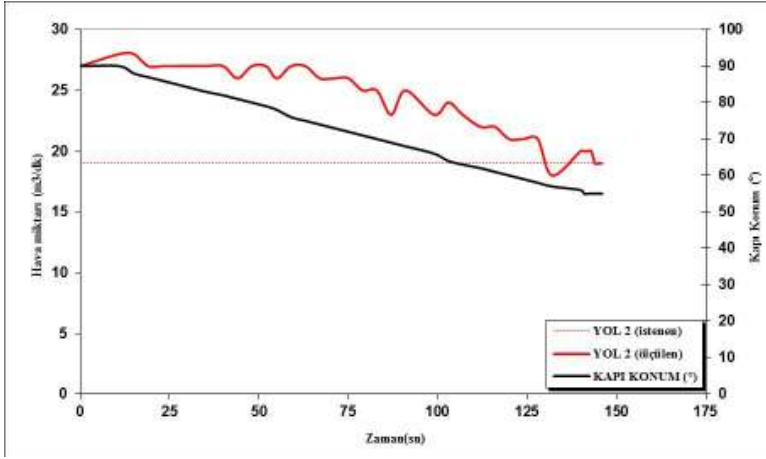
Şekil 6.1. Tüm yollardan geçen hava miktarını ayarlamının zamana bağlı olarak hava miktarı ve kapı konumunun değişimi

### 6.2 İkinci Senaryo Uygulamasına Ait Sonuçlar

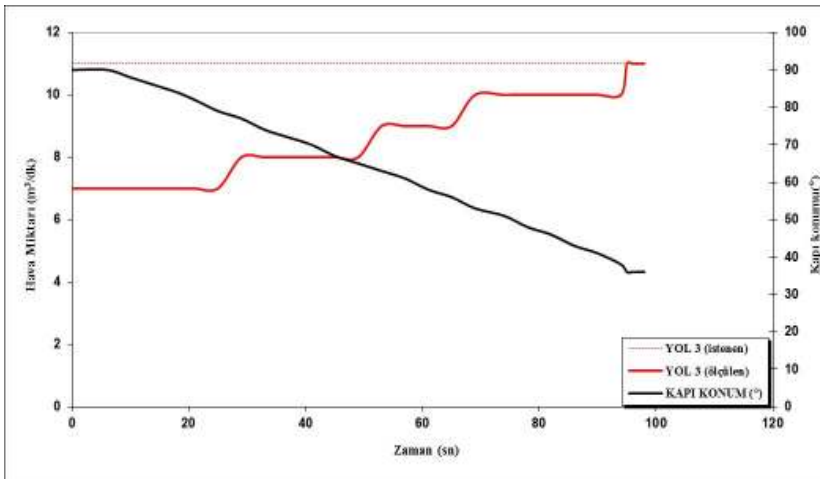
Sisteme sadece 2. yol (regülatör yolu) için 19 m<sup>3</sup>/dak hava miktarı girildiğinde ayarlı hava kapısı 2 dakika 26 saniye süre içinde kapıyı 35° kapatılarak, istenen 19 m<sup>3</sup>/dak hava miktarını ulaşmıştır. Şekil 6.2'de bu uygulamaya ait grafik verilmiştir. Grafikte zamana bağlı olarak hava miktarı ve kapı konumunun değişimleri verilmiştir.

### 6.3 Üçüncü Senaryo Uygulamasına Ait Sonuçlar

Sisteme sadece 3. yol (regülatör olmayan yol) için 11 m<sup>3</sup>/dak hava miktarı girildiğinde ayarlı hava kapısı 1 dakika 40 saniye süre içinde kapıyı 56° kapatılarak istenen 11 m<sup>3</sup>/dak hava miktarını ulaşmıştır. Şekil 6.3'te, bu uygulamaya ait grafik verilmiştir. Grafikte zamana bağlı olarak hava miktarı ve kapı konumunun değişimleri verilmiştir.



Şekil 6.2. Sadece 2. yoldan (regülatör yolu) geçen hava miktarını ayarlamının zamana bağlı olarak hava miktarı ve kapı konumunun değişimi



Şekil 6.3. Sadece 3. Yoldan (regülatör olmayan yol) geçen hava miktarını ayarlamının zamana bağlı olarak hava miktarı ve kapı konumunun değişimi

## 7 SONUÇ VE ÖNERİLER

Yeraltı madenciliğinde havanın kontrollü dağıtımı için en çok tercih edilen yöntem, havanın gereğinden fazla ulaştığı düşünülen hava yollarına tesis edilen “ayarlı hava kapısı/regülatör” kullanımındır. Böylelikle, havanın daha az ulaştığı bölgelere daha fazla hava gönderilebilmektedir. Ancak bu ayarlı hava kapılarının arzu edilen hava dağılımını gerçekleştirecek şekilde tasarlanması, tesis

edilmesi ve değişen koşullara göre yeniden hızlı ve güvenilir olarak düzenlenmesi kolay değildir. Bu nedenle ayarlı hava kapılarının hızlı ve güvenli olarak inşa edilmesi ve değişen şartlara göre yeniden ayarlanmasında otomatik bir sistemin kullanılabilmesi önem arz etmektedir.

Bu çalışmada, yeraltı madenciliğinde paralel hava yollarında havanın arzu edildiği şekilde kontrollü olarak dağılımının

gerçekleştirilmesi hedeflenmiştir. Bu amaçla, yeraltı şartlarına uyum sağlayacak ve havanın kontrollü olarak dağılımını gerçekleştirecek otomatik bir kapının tasarlanması/imalatı sağlanmış ve sensörlerden gelen verilere göre kapıyı hareket ettirecek bir bilgisayar yazılımı geliştirilmiştir.

Yapılan çalışmalar ve deneyler sonucunda tasarım ve imalatı gerçekleştirilen “bilgisayar kontrollü ayarlı hava kapısının” farklı madencilik şart ve gereksinimlerini yansıtan 3 farklı senaryoya göre, arzu edilen hava dağılımını kısa sürede ve hassas olarak gerçekleştirdiği görülmüştür. Bu sistem zaman içerisinde ocak havalandırmasında olabilecek değişiklikler veya aniden ortaya çıkabilecek gereksinimlere göre yine ayarlandığı değerlerdeki hava dağılımını sürekli yapabileceği gibi, ani değişikliklere göre de istenen hava dağılımını gerçekleştirebilecektir. Bu ise madencilik açısından önemli bir kolaylık ve avantaj sağlamaktadır.

Sistemin sadece paralel hava yollarında havanın kontrollü dağılımını sağlama görevi yanı sıra, ocak havalandırmasında kullanılan ve “şebeke analizi” yapan ticari yazılımlarla da bütünleştirilerek, ocak havalandırma verimliliğini artırmak üzere kullanımı da mümkündür. Bu durumda ocağa daha fazla hava göndermek yerine, bilgisayar kontrollü ayarlı hava kapıları kullanılarak ocak içerisindeki havanın en iyi dağılımının gerçekleştirilmesi, böylece havalandırma veriminin artırılması veya havalandırma maliyetlerinin azaltılması söz konusu olabilecektir.

Laboratuvar ölçekli olarak imal edilen böyle bir sistemin madencilik şartlarında da kolaylıkla imal edilmesi ve uygulanması mümkün olacaktır. Ancak, sistemin öncelikle bir pilot ocakta denenerek, gerçek şartlardaki başarısı ve elektro-mekanik sistemin performansı belirlenmelidir. Yine, laboratuvar boyutta ve paralel 2 hava yolunda arzu edilen hava dağılımının gerçekleştirilmesinde kullanılan sistem 2’den fazla havayolunda arzu edilen hava

dağılımının gerçekleştirilmesi için de kullanılabilir. Bu amaçla yazılımın geliştirilmesi gerekecektir.

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