Chapter - J

COAL

KÖMÜR

Coking Coal Supply for Iron and Steel Industry and Advanced Carbonization Techniques

H. Ergun
Independent Consulting Engineer, Turkey

ABSTRACT People, who are involved in the Mining Industry and in the Steel Making Technology, know the importance of Coking Coal for Industrial Development. In our Days the wonder word "Clean Coal Technology" is a basic assumption in respect to CO_2 as well as climatic enviromental politics. Because of shortage of Coking Coal worldwide the possibility of utilizing of non-coking coals for metallurgical coke production especially by advanced carbonization technologies like "Formed Coke" processes are gaining on increasing importance.

1 COKING COAL AND INDUSTRIAL DEVELOPMENT

1.1 Carbonization Technologies

The institutions of the montan and steel industry are important germ cells for new technologies and new products in our industry.

The readiness and the courage for new ways will be now and in the futiure driving forces for the industry.

Only so we could secure the competitiveness and keep existing jobs and create new ones. Therefore it should be one of our most important aimes to strengthen the innovative dynamic of the economy. We need the innovative dynamic for the iron and steel industry and especially for the case of supply with coking coal for carbonization technologies world wide.

1.2 Conventional Coke Oven Technologies

With the Coke Oven Schwelgern, the Company ThyssenKrupp Stahl AG is operating the most modern and clean conventional Coke Oven of the World errected in the industrial Location Duisburg in Germany since March 2003.

However, the Construction of the Coke Oven Schwelgern at the turning point to the 3rd millennium is not one under many.

In the context with the worldwide industrial development of the hard coal mining and the iron and steel industry in the past decades, that led at later moments to well known world finance crises (last in September 2008), also the number of the coke ovens in the world were drastically decreased.

1.2.1 The Situation with the Coal Supply

The term coal comprises a very wide palette of carbon containing raw materials that can be classified after the most varied viewpoints.

If we base on the respective age as characteristic distinction features, we would designate the youngest coal as a turf coal and the oldest as an anthracite coal.

The coal can be classified after appearance, fields of application, chemical composition, carbonization technological characteristics, or also after coal microscopic viewpoints**.**

Corresponding to the age and the mineral deposit conditions, the process technological treatment of the coal is undertaken in order to be able to use it for the Coke Production in the iron and steel industry.

A mineral beneficiation processing of the coal is basically necessary in order to separate the raw material from mineral contaminations.

The classification of coal is generally based on the content of volatiles. However, the exact classification varies between countries. According to the German classification, the fat coal is the coking coal.

Already in the coal preparation or later in the coke oven the grinding of the coal is undertaken.

1.2.2 Preparation of Coal for Carbonization Technology

The grinding and comminution of the coarse particles of the coal should create the prerequisite out of carbonization technological view point for a good mixing ability of several coal components.

The comminution of the coal is however also therefore necessary in order to control special processes in the carbonization (like expansion pressure).

Based on the worldwide shortage of the good quality coking coal and the endeavoring emerging from that to expand the raw material basis for the carbonization, following procedure lines have been developed for the coal blend:

- Optimization of the coal blend,
- Conditioning through supplement of pitch and special grog,
- Intensification through Tramping,
- Mixture through Partial Briquetting,
- Formed Coke Production.

With the goal of increasing on the one hand the capacity of the coke ovens, on the other hand to reduce the harmful material emission, around through it to increase the environment contractual situation, the following ways have been followed in the course of the last decades:

- Reduction of the chamber width (1960-1970)
- Reduction of the Runner Stone Thickness
- Increase of the Flue Chamber Temperature
- Pre-drying of the coal blend
- Pre-heating of the coal blend
- Intensification through special binders
- Compacting through tramping
- Coke Dry Quenching
- Increasing of the chamber width (1980-1990)

2 COKING COAL PRODUCTON AND DEMAND

Coal is a very inconsistent material. This heterogenity is to be attributed to the inconsistent genetic material and in addition also on the different temporal and geothermal influences with the Coalification. Already early was recognized that all coals do not suit themselves to the Carbonization. Prerequisite for a Coking Coal is a sufficient baking capacity.

The special characteristic to form a firm baked residue, what is to be found only in certain bituminous coals, the so-called coking coals, are since beginning of the 20th century again and again object of numerous research and investigations.

Coal provides 29.6 % of the global primary energy and generates 42% of the world's electricity.

World Total Coal Production (including hard coal and lignite):

7 229 Mt (2010)

6 823 Mt (2009)

World Total Bituminous Coal Production

6 185 Mt (2010)

891 Mt coking coal + 5 294 Mt steam coal

5 789Mt (2009):

782Mt coking coal + 5 007 Mt steam coal

Approximately 13% of total hard coal production is currently used by the steel industry and over 60% of total global steel production is dependent on coal.

International Hard Coal Trade

Steam Coal: 676 Mt

Coking Coal: 262 Mt

Total Trade: 938 Mt

Worldwide Coking Coal Deficit is estimated to approximaetly 200 MT/a.

For the year 2015 it is estimated that the coking coal deficit of only China and India will be together around 200 MT/a.

Coking coal price surged to US $ 300 per tonne in the first quarter of the calendar year 2011.

Supply disruptions caused by flooding in Australia influenced also the coking coal price.

3 ADVANCED CARBONIZATION TECHNIQUES

Because of Shortage of Coking Coal worldwide the possibility of utilizing of non-coking coals for metallurgical coke production especially by advanced carbonization technologies like formed coke processes are gaining on increasing importance.

The sucessful application of the proposed advanced carbonization processes would result in considerable saving by the substitution of essential amounts of coking coals through non-coking coals.

3.1 Formed Coke Process

The briquetting of the coal blend under aid of pitch is the first level of the formed coke production. Herewith the components are usually finely ground up, mixed through well and brought by presses in close contact.

The briquettes should be carbonized at high temperatuıres over 1000 grad celcius, if formed coke is supposed to be produced.

The carbonization cannot take place in the Horizontal Chamber Coke Oven because the Coke Cake has no sufficient Stability.

For this reason new coke oven reactor had to be developed for the formed coke production that is based partially upon the old vertical type coke oven technology.

With Formed Coke, that was produced in a Vertical Chamber Type Coke Oven, industrial attempts were accomplished in a Blast Furnace in Japan whereby the Formed Coke ratio amounted to up to 25%.

Non-coking coal continuously to compact and to carbonize, busied the professions already since the '1950's.

3.2 Possibilities of Opening of New Coal Mines in order to cover the Requirement of the Steel Industry

Metallurgical coal is an essential raw material in the production of steel which is used in buildings, household items and other goods and services that modern societies need.

As an example for increasing Coking Coal production BHP Billiton Mitsubishi Alliance (BMA) owns and operates seven Bowen Basin mines – Goonyella Riverside, Broadmeadow, Peak Downs, Saraji, Norwich Park, Gregory Crinum, and Blackwater – and the Hay Point terminal near Mackay.

The mines are located in the coal-rich Bowen Basin of Central Queensland, which covers an area of 75,000 km² of Permian Age. BHP Billiton owns 50 per cent of BMA.

Further new coking coal reserves are going into operation in Mozambique and Colombia, which should cover the demad of the steel industry in the next decades.

The difficulties in Coal Mining and the increasing steel demand will led to development of special steel making technologies like stainles steel, forged steel by using electric - arc fuırnaces instaed of integrated iron and steel technologies using coke ovens, sinter plant, blast furnace and basic oxygen furnace.

The subjects of the future developments in Turkey will possibely be “stainless steel” and “forged steel”.

4 SYNOPSIS

As Synopsis we can make the Observation that worldwide there is a deficit of good quality coking coal for the iron and steel industry in a yearly amount of approximately 200 MT.

There are advanced technologies to produce metallurgical coke from non-coking coals, which however should be further developed.

New Mines for Coking Coal should be opened to cover the worldwide demand for good quality coking coal.

In future the stainless steel and forged steel production should be also supported by using sophisticated and high developed steel making technologies.

REFERENCES

Ergun, Hayri: January 25-26, 2012, Coking Coal Supply for Integrated Iron and Steel Works and Advanced Carbonization Techniques as well as Possibilities of Erection of New Mines for Producttion of Coking Coal., International Iron and Steel Summit, EBY Int.,, Sheraton Ataköy Hotel, Istanbul,Turkey.

Ergun, Hayri: September 12 - 17, 1999. An Improved Carbonization Process to produce Formed Coke. International Conference on Coal Science, Taiyuan – China.

Wagener , D. and H. Ergun : 9 - 13 October 1978. Modern Industrial Ways of Coke Making. South East Asia Iron and Steel Institute (S.E.A.I.S.I.) Symposium on Raw Materials , Feed and Energy Sources for the Iron and Steel Industry, Philippines.

Çayırhan Yeraltı Linyit İşletmesi B Sahasında Ayak Uzunluğunun Değişmesinin Üretim Çalışmalarına Etkisinin İncelenmesi

Investigation of the Face Length Changes Effects to Production Operations in Çayırhan Underground Lignite Mine-B Field

E. Kahraman
Park Termik A.Ş., Ankara

F. Ustabaş, Ş. Tarhan ve F. B. Taşkın
Park Termik A.Ş., Ankara

ÖZET Mekanize yöntemle çalışan yeraltı kömür işletmelerinde diğer yöntemlere göre üretim hızı ve miktarı oldukça fazladır. Ancak, diğer yöntemlerde olduğu gibi mekanize sistemlerde de üretim hızını ve miktarını etkileyen faktörler vardır. Çeşitli nedenlerden dolayı ayak uzunluğunun değişmesi de üretim miktarını ve sürecini etkileyen faktörlerden bir tanesidir.
Çalışmada, geri dönümlü göçertmeli tam mekanize uzun ayak üretim yöntemi kullanılarak tavan ayak ve taban ayak şeklinde dilimli üretim yapılan Çayırhan Yeraltı Linyit İşletmesi B sahasında B-07 panosunda ayak üretiminde karşılaşılan problemler ve çözümü için yapılan çalışmalar incelenmiştir. Elde edilen sonuçlar değerlendirilerek gelecekte karşılaşılabilecek bu tarz sorunlar için çözüm önerileri belirtilmiştir.

ABSTRACT Production rate of mechanized mining methods in underground coal mines are quite high compared to other mining methods. However, in mechanized systems, there are also certain factors that affect production rate. Changes in face length are one of the main factors affecting production quantity and process.
In this study, technical problems and their solutions are observed and recorded during coal production process in Çayırhan coal mine B field, B-07 longwall panel where retreat full mechanized longwall two-sliced coal production method is used.
Consequently, obtained results are evaluated and recommendations to their solutions related to such possible problems are indicated.

1 GİRİŞ

Ayakta üretim çalışmalarında kesici-yükleyici makinenin, ayak tahkimatında yürüyen tahkimatların ve ayakta kazı nakliyatında zincirli konveyörlerin kullanıldığı kazı çalışmaları tam mekanize kazı olarak adlandırılmaktadır. Tam mekanize kazı çalışmaları, klasik ve yarı mekanize kazı çalışmalarına göre oldukça avantajlı bir yöntemdir (Fauser, 1981).

Tam mekanize kazı ile uzun ayaklardan kömür üretimi çalışmalarında süreci olumsuz etkileyen faktörler bulunmaktadır.

Bu faktörler; kontrol edilemeyen faktörler ve kontrol edilebilen faktörler olarak iki başlıkta toplanabilir. Kontrol edilemeyen faktörler; damarın derinliği, damarın kalınlığı ve karakteristikleri, bölgesel jeoloji ve hidrojeolojik koşullardır. Kontrol edilebilen faktörler ise; pano geometrisi, nakliyat ekipmanlarının kapasitesi ve gücü, ekipmanların güvenilirliği ve kullanım oranı, ayak iklimi, damardan gelebilecek gaz emisyonu, kesici makinelerin gücü gibi faktörlerdir (Yavuz, vd., 2003).

Madencilik çalışmalarında, işin doğası gereği (kontrol edilemeyen faktörlerden dolayı) proje ile uygulama arasında farklılıklar olabilmektedir. Bu farklılıkları gidermek amacıyla, uygulamada kontrol edilebilen faktörler üzerinde ilave operasyonlar yapılması gerekebilmektedir. Genellikle, pano geometrisinde değişiklikler yapılarak, bu sorunlar aşılmaktadır. Pano geometrisinde değişiklikler, ayak uzunluğunun artırılması, ayak birleştirilmesi, ayak uzunluğunun ve pano uzunluğunun kısaltılması, vb. şekilde yapılmaktadır.

Gerçekleştirilen bu faaliyetler iş akış sürecine, dolayısı ile verimliliğe etki etmektedir. Bu yüzden, çalışma ortamına özgü bir iş olan bu faaliyetlerin iyi analiz edilmesi, genel madencilik sorunlarının etkin bir şekilde çözüme kavuşturulmasında etkili olacaktır.

Çalışma kapsamında, Park Termik A.Ş. Çayırhan Linyit İşletmesi B Sahası 07 panosunda sorunları çözmek amacıyla gerçekleştirilen faaliyetler incelenerek, üretime etkileri saptanmış, sürecin uygun koşullarda ve sıkıntısız atlatılabilmesi amacıyla yapılması gerekenler tespit edilmiş, madencilikte karşılaşılması mümkün olan bu durumun giderilmesi için yapılan işlemler sıralanmıştır.

2 ÇAYIRHAN LİNYİT İŞLETMESİ

2.1 İşletme Sahalarında Madencilik Faaliyetleri

Çayırhan Linyit İşletmesi, Ankara İlinin Nallıhan İlçesine bağlı Çayırhan Beldesi' nde faaliyet göstermektedir. İşletme, Ankara' ya 122 km mesafede olup Ankara-Nallıhan yolu üzerinde bulunmaktadır.

İşletmeye ait yeraltı ocaklarından tam mekanize kazı kullanılarak 5-5,5 milyon ton/yıl üretim yapılarak, 620 MW gücünden kurulu bulunan Çayırhan Termik Santrali' nin enerji ihtiyacı karşılanmaktadır.

Sahalarda gerçekleştirilen madencilik çalışmaları,

i. etüd-proje çalışmaları,
ii. hazırlık kazıları,
iii. üretim kazıları,
iv. söküm-montaj çalışmaları
v. diğer çalışmalar

olmak üzere sınıflandırılabilir.

İşletmede mevcut durumda madencilik faaliyetlerinin gerçekleştirildiği 6 adet sektör/saha (B, G1, G2, D, E ve H Sahası) bulunmaktadır. Bu sektörlerden mevcut durumda sadece B, G1 ve G2 sahalarında üretim çalışmaları yürütülmektedir. Diğer sektörlerde etüd-proje ve hazırlık çalışmaları yürütülmektedir.

Sahalarda yapılan çalışmaların ilki etüd-proje çalışmalarıdır. Bu aşama, kömür arama ve bulma, maden yatağının bilgisayar programı (Micromine) ile modellenmesi, rezerv hesaplamaları ve üretim yöntemi seçiminin yapıldığı süreçten oluşmaktadır.

Hazırlık çalışmalarında ocakların üretime hazırlanması amacıyla kazılar yapılmaktadır. Hazırlık çalışmaları, mekanize kazı çalışmaları (galeri açma makinesi ile kazı), tahkimat çalışmaları, hazırlık galerilerinin havalandırılması ve diğer işler (nakliyat ünitelerinin uzatımı, basınçlı sistem boru bağlantıları, vb.) gibi aşamalardan oluşmaktadır (Kahraman ve Erarslan, 2011).

Kömür üretimi, geri dönümlü göçertmeli tam mekanize uzun ayak yöntemi ile gerçekleştirilmektedir. Bu yöntem genel olarak; ayakta mekanize kazı çalışması, ayakbaşı ve kuyruk bölgesi tahkimatları, ayakiçi tahkimatları, ayak havalandırması ve diğer çalışmalardan (nakliyat ekipmanlarının ilerletilmesi, enerji ekipmanlarının, hidrolik ekipmanların ilerletilmesi, vb.) oluşmaktadır.

Söküm-montaj çalışmalarında, kömür üretimi tamamlanmış olan panodan ayak ekipmanları ve yardımcı donanımlarının sökülerek, hazırlanmış olan yeni üretim panosu ayağına taşınması işlemidir. Söküm-montaj çalışmaları mekanize kazının en önemli aşamalarından birisidir. Uzmanlık isteyen ve dikkatli yönetilmesi gereken bir süreçtir. Bu sürecin uzaması üretim çalışmalarını olumsuz etkilemektedir (Kahraman, 2012).

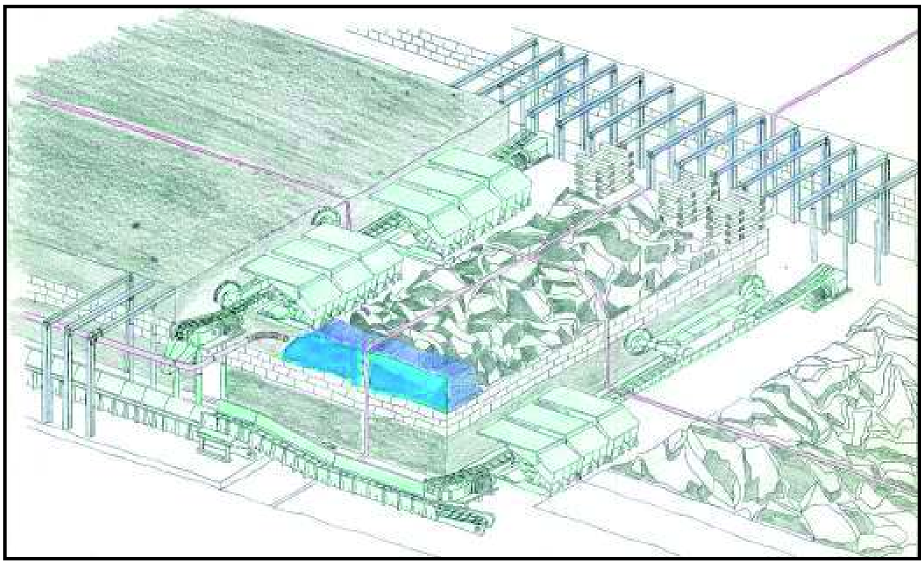
2.2 B Sahasında Üretim Çalışmaları

B sahasında ara kesme kalınlığının fazla (1,3-2m) olması nedeniyle sahada dilimli kazı (tavan ayak, taban ayak) uygulanmaktadır. Ocakta pano boyları 1.100-1.300 m, ayak uzunlukları 200-230 m olacak şekilde planlama yapılmaktadır. Ayak ortalama yüksekliği; tavan ayakta 1,60 m, taban ayakta 1,90 m' dir. Tavan ve taban ayağa gelen basıncı dengelemek ve yangın riskini en aza indirmek amacıyla tavan ve taban ayaklar arasında 15–30 m mesafe bırakılmaktadır (Şekil 1).

Hazırlık çalışmalarında, arakesme, tavan kömürü ve taban kömürünü kapsayacak şekilde, galeri açma makinesi ile taban yolları (bacalar) açılmaktadır. Alt taban ve üst taban yolları, ayak kılavuzları (başyukarılar) ile birbirlerine bağlanırlar. Daha sonra bu kılavuzlara ayak ekipmanları montajı yapılarak, üretim çalışmalarına başlanılır.

Tavan ve taban ayaklarda tam mekanize kazı sistemi ile kazı işlemi gerçekleştirilmektedir. Bu sistemde, çift tamburlu kesici yükleyici makine ile kazılan kömür ayak içinden alt taban yoluna zincirli konveyörler ile nakledilmektedir. Ayak tahkimatı kalkan tipi yürüyen tahkimatlarla (şild) sağlanmaktadır. Tavan ve taban ayakta kullanılan tahkimat ünitelerinin özellikleri Çizelge 1' de, kesici yükleyici makinelerin teknik özellikleri Çizelge 2' de ve zincirli konveyörlerin özellikleri Çizelge 3' de sunulmuştur (Kahraman, vd, 2012).

B sahasında, yeraltına insan ve malzeme nakliyatı tekkars ve yerkars taşıma araçlarıyla gerçekleştirilmektedir. Ocağın ana havalandırılması, emici tip cebri (vantilatörlerle) havalandırma sistemi ile sağlanmaktadır. Hazırlık galerilerinin havalandırması (tali havalandırma), tali vantilatörlerle üfleyici sistem şeklinde yapılmaktadır (Kahraman, 2012).



Şekil 1. B sahasında yapılan ayak üretim çalışması şematik gösterimi (Kahraman, 2012)

Çizelge 1. Tavan ve taban ayaktaki tahkimat ünitelerinin teknik özellikleri

Açıklamalar	B Tavan Ayak Şildi	B Taban Ayak Şildi
Toplam Adedi	149	151
Tipi	KB 8/20,5	KB 13/25,5
Minimum Yükseklik (mm)	800	1.300
Maksimum Yükseklik (mm)	2.050	2.550
Genişlik (mm)	1.500	1.500
Ağırlık (kg)	11.010	12.640
Tahkimat Direnci (kN/m^2)	382 (2m)	382 (2m)
Üretici Firma	Saar Tech (DBT)	Saar Tech (DBT)
İmalat Yılı	1996	1996
Yardımcı Valfler	Saar Tech (DBT)	Saar Tech (DBT)
Kontrol Sistemi	12 fonksiyonlu	16 fonksiyonlu

Çizelge 2. B sahasında kullanılan kesici yükleyici makinenin teknik özellikleri

No	Makine Özellikleri	Eickhoff SL 300
1	Kesici motor gücü (2 adet, kW)	300
2	Yürüyüş motoru (2 adet, kW)	35
3	Hidrolik motoru (1 adet)	7,5
4	Toplam kurulu güç (kW)	677,5
5	Ağırlık (Tamburlar hariç, ton)	30
6	Maksimum kesme yüksekliği (m)	2,66 (kol dahil)
7	Çalışma gerilimi (V)	3300
8	Tambur dönüş hızı (devir/dak)	50
9	Tambur çapı (m)	1,4
10	Soğutma suyu basıncı (bar)	10-25
11	Soğutma suyu debisi (lt/dak)	156
12	Maksimum kuvvet (kN)	600
13	Kontrol sistemi	Radyo dalgası ile uzaktan kumanda
14	Maksimum yürüyüş hızı (m/dak)	14

Çizelge 3. Ayakiçi ve tabanyolu zincirli konveyörlerinin teknik özellikleri

Özellik/Konveyör	Ayakiçi Konveyörü (AFC)	Aktarma Konveyörü (BSL)
Tipi	Halbach Braun	Long Airdox
Kapasite (t/h)	1800	2000
Kurulu Güç (kW)	Tavan ayak=(1×250)+(1×400) Taban ayak=(1×250)+(1×400)	400
Zincir Hızı (m/s)	1,12	1,34
Şanzıman tahvil oranı	39/1	33/1
Yürüyüş sistemi	Eicotrak	-

3 B-07 PANOSUNDA ÜRETİM SIRASINDA AYAK UZUNLUĞUNUN DEĞİŞTİRİLMESİ

Pano hazırlık çalışmaları kapsamında, alt taban yolunun galeri açma makinesi ile kazısı sırasında süreksizlikler ve bunlara bağlı olarak su geliri ile karşılaşılmıştır. Galeride, bu bölgede göçükler (tavan akmaları) meydana gelmiştir. Su geliri ve tavan akmalarından dolayı galeri açma çalışmaları 818. metrede durdurulmuş, bu bölgede yapılan sondaj çalışmaları neticesinde, galeri istikametinin değiştirilmesi (pano üretim planının değiştirilmesi) kararlaştırılmıştır. Galerinin 500.metresinde; panonun planlanan ayak uzunluğunda 40 m daralma olacak şekilde alt taban yolu ilerletilerek, hazırlık çalışmaları tamamlanmıştır. Dolayısıyla, B 07 panosu, üretim açısından B 07-1 ve B 07-2 olarak adlandırılan, ayak uzunlukları farklı iki ayrı pano haline gelmiştir (Şekil 2). Bu panoların boyutları Çizelge 4' de sunulmuştur. Hazırlık çalışmalarını takiben, B 07-1 panosunda (15.08.2010) üretim çalışmalarına başlanılmıştır.

Çizelge 4. B 07-1 ve B 07-2 panosu

Pano Adı	Pano Uzunluğu (m)	Ayak Uzunluğu (m)
B 07-1	500	190
B 07-2	420	150

B 07-1 panosunda üretim çalışmalarına devam edilirken aynı zamanda da, B 07-2 panosuna geçiş için ayak uzunluğunun artırılacağı bölgede hazırlık çalışmaları yapılmıştır. Bu çalışmalarda, 1 no' lu galeri (tavan ayak kılavuzu), 2 no' lu galeri (taban ayak kılavuzu), 3 no' lu galeri (ara nakliyat galerisi) açılmıştır (Şekil 3). 1 ve 2 no' lu galerilere yürüyen tahkimatlar ve nakliyat donanımları (zincirli konveyör) montajları yapılmıştır. 3 no' lu galeriye de zincirli konveyör montajı yapılmıştır.

B 07-1 panosunda üretim çalışmalarının tamamlanıp, B 07-2 panosuna geçiş noktasına gelindiğinde, ayağın uzatımı için gerekli işlemler yapılmıştır. Bu işlemlere, ait iş akış şeması Şekil 4' te sunulmuştur.

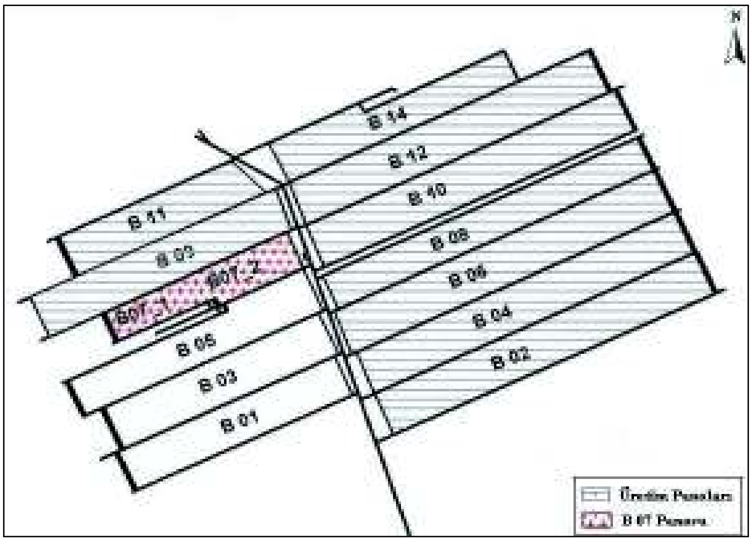
Ayak uzatım işlemi sonrası B 07-2 panosunda üretim çalışmalarına devam edilmiştir. Panoda üretim çalışmaları 30.06.2011' de tamamlanmıştır.

B 07 panosunda, hazırlık aşamasında karşılaşılan ve önceden öngörülemeyen sorunlar nedeniyle, panodan planlanan miktarda kömür üretilememiştir. Panodaki değişime bağlı olarak, rezerv ve kalori miktarlarındaki değişim Çizelge 5' te sunulmuştur.

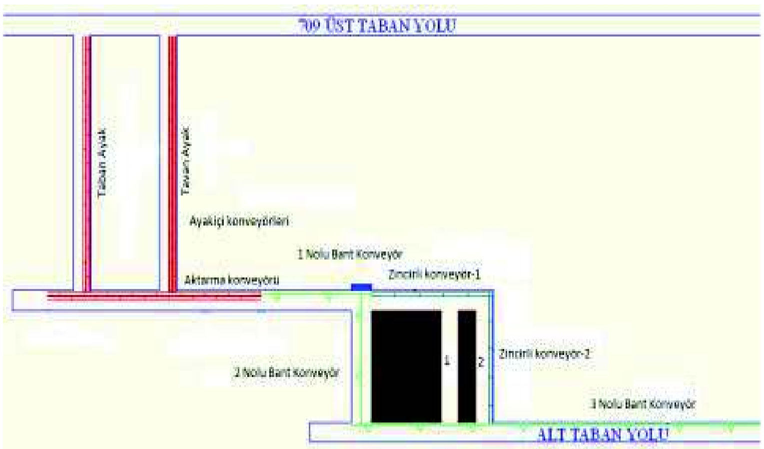
Ayak uzunluğunun artırılması sürecinin aşamaları, bu aşamalarda gerçekleştirilen faaliyetler ve faaliyetlerin süreleri Çizelge 6' da sunulmuştur.

Çizelge 5. Rezerv ve kalori değerleri

	B-07 Panosu	Tüvenan Kömür (ton)	Kalori (kcal/kg)	Kül (%)	Nem (%)	Uçucu Madde (%)	Sabit Karbon (%)	Toplam Kükürt (%)
Planlanan	Tavan Kömürü	519.879,69	2.518,00	35,18	19,95	24,83	20,58	4,23
	Taban Kömürü	555.148,61	2.249,00	40,38	18,16	23,17	18,44	3,80
	Toplam	**1.075.028,30**	**2.379,09**	**37,87**	**19,03**	**23,97**	**19,47**	**4,01**
Gerçekleşen	Tavan Kömürü	460.151,22	2.543,00	34,58	20,03	25,12	20,91	4,22
	Taban Kömürü	488.097,05	2.262,00	39,93	18,25	23,36	18,65	3,80
	Toplam	**948.248,27**	**2.398,36**	**37,33**	**19,11**	**24,21**	**19,75**	**4,00**

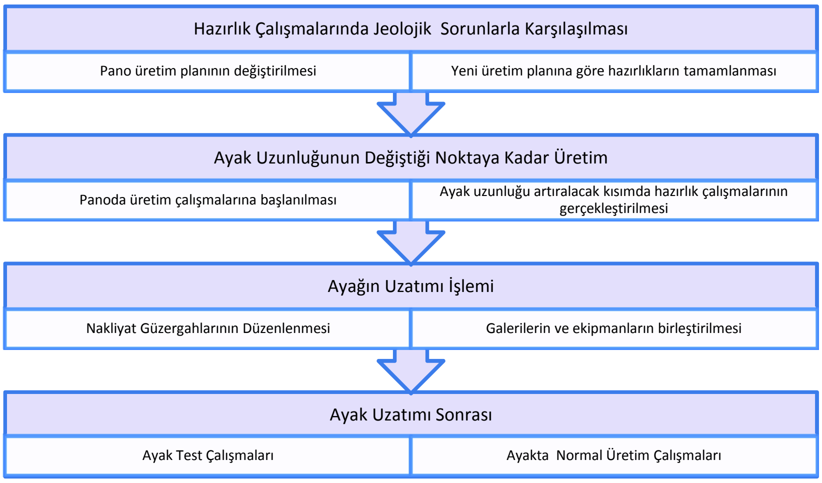


Şekil 2. B 07 panosunun konumsal görünümü



Ölçeksizdir.

Şekil 3. Mevcut durumda ayak üretiminde nakliyat ve ayağın uzatımı için hazırlıklar



Şekil 4. Ayak uzatım işlemi iş akış şeması

Çizelge 6. Ayak uzunluğunun artırılması sürecinde gerçekleştirilen faaliyetlerin süresi

Aşamalar	Gerçekleştirilen Faaliyetler	Süre (saat)	Açıklamalar
Pano üretim planının değiştirilmesi	20 m lik galeri açılması	120	Hazırlık süresini artırmıştır
	Taban yollarının tamamlanması	1200	Etkisi yok
Yeni üretim planına göre hazırlıkların tamamlanması	1 nolu galerinin açılması	144	Etkisi yok
	2 nolu galerinin açılması	144	Etkisi yok
	3 nolu galerinin açılması	288	Etkisi yok
	1 nolu galeriye şilt montajı (22 şilt)	96	Etkisi yok
	2 nolu galeriye şilt montajı	96	Etkisi yok
	Zincirli konveyör-1 in montajı	48	Etkisi yok
	Zincirli konveyör-2 nin montajı	48	Etkisi yok
	1 nolu galeriye zincirli konveyör montajı	48	Etkisi yok
Nakliyat sistemlerinin düzenlenmesi	3 nolu bant konveyörün kısaltılması	24	Üretim yapılmadı
	2 nolu bant konveyörün demontajı	24	Üretim yapılmadı
	Enerji treni çekimi	24	Üretim yapılmadı
	Aktarma konveyörü çekimi	48	Üretim yapılmadı
Ayağın uzatımı ve ekipmanların birleştirilmesi	Tavan ayağın uzatımı	112	Taban ayakta üretim yapılabilmiştir
	Aktarma konveyörünün sökümü	24	Üretim yapılmadı
	710 alt taban yoluna zincirli konveyör montajı	48	Üretim yapılmadı
	Taban Ayağın birleştirilmesi	96	Üretim yapılmadı

4 SONUÇ VE DEĞERLENDİRME

Bu çalışmada, pano hazırlık aşamasında karşılaşılan jeolojik sorunların pano üretim planına etkisi ve bu süreçte hem hazırlık hem de üretim aşamasında gerçekleştirilen faaliyetlerin sürece olan etkisinin belirlenmesi hedeflenmiştir. Yapılan çalışmada elde edilen sonuçlar aşağıda sunulmuştur:

i. Galeride karşılaşılan jeolojik olumsuzluklardan dolayı pano üretim planı değiştirilerek üretime devam edilebilmiştir. Bu durum, geri dönümlü çalışmanın bir avantajı olarak değerlendirilebilir.
ii. Dilimli kazı yapılan (tavan ayak, taban ayak) ocaklarda herhangi bir nedenle ayak uzunluğunun artırılması gerektiğinde, ayak üretimi devam ederken yapılan diğer hazırlık çalışmalarının önemi görülmüştür.
iii. Dilimli kazı yapılan ayaklarda, böyle bir sorunla karşılaşılması durumunda, öncelikli olarak tavan ayağın uzatılmasının zorunluluk olduğu ve planlamanın buna göre yapılmasının doğru bir uygulama olduğu görülmüştür.
iv. Hazırlık aşamasında karşılaşılan jeolojik süreksizliklerden dolayı, pano üretim planının değiştirilmesi neticesinde; panodan linyit kömürü üretiminde %11,79' luk rezerv kaybı olmuştur (Çizelge 5).
v. Gerçekleştirilen faaliyetlerin üretim sürecine zaman yönünden etkisi incelendiğinde; bu faaliyetler sırasında 400 saatlik (50 vardiya) bir çalışma süresinin kaybolduğu gözlemlenmiştir.

KAYNAKLAR

Fauser, H. (Çeviren: Eskikaya, Ş.), 1981. *Kesici-Yükleyiciler ve Galeri Açma Makineleri, İstanbul Teknik Üniversitesi*, Sayı:1186, İstanbul.

Kahraman, E. ve Erarslan, O., 2011. *Çayırhan Yer altı Linyit İşletmesi G Sahasında Hazırlık Çalışmalarının Değerlendirilmesi, 3.Maden Makinaları Sempozyumu*, İzmir.

Kahraman, E., 2012. *Çayırhan Yeraltı Linyit İşletmesinde Kollu Galeri Açma Makineleri İçin Performans Modeli Geliştirilmesi*, Yüksek Lisans Tezi, Niğde Üniversitesi.

Kahraman, E., Erdem, H.H. ve Sığırcı, C., 2012. *Park Termik A.Ş. Çayırhan Linyit İşletmesi Kazı Tahkimat Söküm İşleri Eğitim Kitabı*, Ankara.

Yavuz, M., Ankara, H. ve Erçelebi, S.G., 2003, *Tam Mekanize Uzunayaklarda Üretimi Etkileyen Faktörler*, Türkiye 18. Uluslararası Madencilik Kongresi ve Sergisi, Antalya.

Improvement of Octane Number of Naphtha by Urea Adduct

E. Nassef
Dr.,Petrochemical Departement,Faculty of Engineering,Pharos University,Alexandria,Egypt

H. Salah
Technology Development Unit, El-Amreya Petroleum Company, Alexandria, Egypt

ABSTRACT The present work deals with studying the upgrading of octane number n-parraffins from c6 upwards which can be obtained by means of urea adduct. The valuable iso paraffins do not form urea adduct. The octane number can be improved by the use of specially selected fractions of straight run of reformed gasoline in the laboratories was shown that the octane number of reformed gasoline could be raised from 95 to 102 .It can be assumed that the procedural possibilities are not yet ex-being developed .This study aims to study the effect of amount of urea added on the refractive index, specific gravity, density and other physical properties to the gasoline produced. It was found that the new technique enhance the octane number of gasoline 80 to 82,and for gasoline 90 to 92.

1 INTRODUCTION

Separation of branched chain and straight chain paraffins from petroleum products has a great importance in chemical industry; these parafins are used as an in industry mainly in alkylation process especially in detergent industry. Urea forms crystalline complexes with straight chain organic compounds, has led to intense investigations of the complex-forming properties of urea in laboratories throughout the world. The urea molecules form spirals with the hydrocarbon molecules situated at the center. The urea complex form excellent needle crystal type. Urea dewaxing which had a brief period of success in the 1950's to 1960′s and which removed normal paraffins using shape selective clath-rate In a process for dewaxing liquid mineral oils comprising reacting said liquid mineral oils with urea in the presence of an organic solvent to form solid, n-paraffin-urea adducts, separating said adducts from the solution of the dewaxed mineral oil, decomposing or extracting the separated adducts, recovering and recycling the urea and, optionally, recovering the n–paraffins. The object of this study is to improve the urea dewaxing process so that very small amounts of n-paraffins contained in mineral oil distillate and similar hydrocarbon mixtures may be removed there from in a simple relatively rapid manner, and that the removal is practically complete and is preferably performed in a continuous operation. Another object is to produce lubricating oil for refrigerating machines from a naphthenic mineral oil distillate. n-paraffin's from C5 upwards can be obtained by means of urea and suitable procedures (particularly SO_2 solvent as suggested by Fetterly from straight run or reformed gasoline by extraction . The valuable iso paraffins do not form urea adduct.The objective of this study is to improve the octane number of gasoline (80 & 90) by urea dewaxing

2 EXPERIMENTAL EQUIPMENT AND PROCEDURES

The experimental set-up used in this work as well as the materials utilized will be presented. Properties evaluation methods are also shown.

2.1 Experimental setup

The reaction vessel consists of a conical flask provided with a mechanical stirrer for mixing. A hot plate adjusted at the reaction temperature is connected to a distillation unit used to recover the solvent. The wax and oil weights are determined using digital balance. The experimental set up used in the present work is schematically shown in figure (1).

Figure 1. Experimental set-up

2.2 Materials

Gasoline (80 & 90) were obtained from gasoline station in Alexandria, Egypt. The characteristics of the of the two types of gasoline are shown in Table (1) and Table(2). The saturated urea solutions were prepared by using an analytical grade urea dissolved in distilled water, which needed for the experimental condition investigated. A toluene; nitrification grade, was used as a solvent for washing the oil product. A paraffinic wax was separated as wax urea adducted after each run and weighted.

2.3 Experimental Procedure:

Gasoline 80 and 90 were depentanized first before adduct formation using simple distillation. A solution mixture of urea and methanol are mixed together with depentanized gasoline (100ml) in a three neck round flask (1000 ml). The reaction mixture was heated and controlled at specific temperatures using hot plate and a thermometer, for specific time while stirring by mechanical stirrer as shown in figure 1. For each experiment the reaction mixture was prepared by weighting the required amount of a Gasoline 80 and 90. The saturated solution of urea was added to the gasoline to ensure the fluidity of the material. After complete adduction, shown by the separation of the mixture into two distinctive layers, oil layer and wax layer. The wax layer was washed by toluene and filtered. The oily layer with its solvent added in the very beginning was recovered in a distillation unit and pure dewaxed gasoline was obtained and weighted. The urea wax adduct mixture was washed with hot water and two layers were obtained and separated in a separating funnel; wax layer and urea solutions. After each run the densities, Refractive Index, % normal Paraffin, % Aromatics and finally Octane Number were Determined for the produced oil.

2.4 Characterization of gasoline (80&90)

Gasoline 80 and 90 used in this work were characterized by GCMS in Elamria petrochemical company located in Alexandria, Egypt. Tables (1&2) show the characteristics for both of them

Table 1. Chemical composition of Gasoline octane 90

	aromatics	Iso-paraffin	naphthen	olefins	paraffins
C3					0.1
C4					3.78
C5		18.99	0.41		5.67
C6	1.99	8.57	0.63	0.05	2.81
C87	13.21	2.98	0.32	0.01	1.39
C8	14.71	4.79	1.11	0.03	2.52
C9	7.60	3.41	0.10	0.02	0.9
C10	1.99	0.83	0.08		0.21
C11	0.37	0.04			0.16
C12	0.10	0.02			
total	40.00	39.65		0.11	17.59

Table 2. Chemical composition of Gasoline octane 80

	aromatics	Iso-paraffin	naphthen	olefins	paraffins
C3					0.19
C4					4.12
C5		13.93	0.73	0.02	5.65
C6	1.72	12.02	2.32	0.18	6.61
C7	9.75	8.01	2.77	0.02	4.22
C8	9.56	4.78	1.19	0.03	2.55
C9	3.57	2.54	0.52	0.03	1.03
C10	0.85	1.00			0.23
C11	0.08				0.09
C12	0.05				
total	25.58				24.69

3 RESULTS AND DISCUSSION

3.1 Effect of urea amount on refractive index of gasoline (80&90)

Fig. 2 shows the effect of amount of urea on refractive index for gasoline (80and90)from figure it was shown that as the amount of urea increases from 20 grams to 100 grams the refractive index increases from1.4154 to 1.4373 for gasoline 80 and from 1.4621 to 1.4794 for gasoline 90. This is due to the fact as the amount of urea increases, the adduct formation increases and hence increase in the removal of paraffin which leads increasing in refractive index.

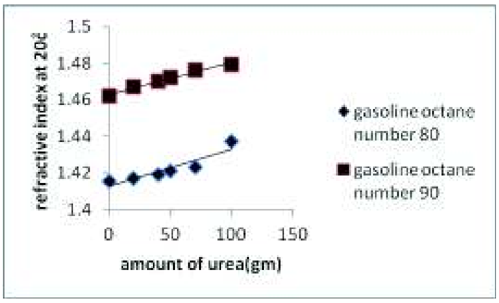


Figure 2. Effect of amount of urea on refractive index of gasoline (100 ml gasoline, 100 ml methanol T=70°C, t= 10 h)

3.2 Effect of urea amount on density of gasoline (80&90)

Fig. 3 shows the effect of amount of urea on density for gasoline (80and90) from figure it was shown that as the amount of urea increases from 20 grams to100 grams the density increases from0.7152to 0.7270 for gasoline 80 and from 0.7476 to 0.7624 for gasoline 90. This is due to the fact as the amount of urea increases, the adduct formation increases and hence increase in the removal of paraffin which leads to increasing in densities.

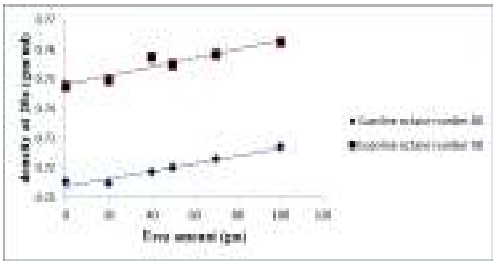


Figure 3. Effect of amount of urea on density of gasoline (100 ml gasoline, 100 ml methanol at 70°C for 10 hours)

3.3 Effect of urea amount on % n-paraffin of gasoline

Figure 4 shows the effect of amount of urea on n-paraffin's for gasoline (80 and 90) from figure it was shown that as the amount of urea increase from 20 grams to 100 grams the n-paraffin's decrease from 24.69 to 22.52 for gasoline 80 and from 17.59 to 15.82 for gasoline 90. This is due to the fact as the amount of urea increase, the adduct formation increases and hence decrease in the % weight of paraffin.

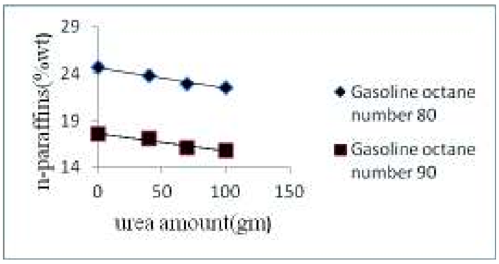


Figure 4. Effect of urea amount on % n-paraffin of gasoline (100 ml gasoline, 100 ml methanol at 70°C for 10 hours)

3.4 Effect of urea amount on %total aromatics

Fig. 5 shows the effect of amount of urea on total aromatics for gasoline (80 and 90) from figure it shown that as the amount of urea increase from20 gm to100 gm the total aromatics increase from28.58to 29.72 for gasoline 80 and from 40.0 to 43.21 for gasoline 90 This is due to the fact as the amount of urea increase, the adduct formation increase and hence increase in the % weight aromatics.

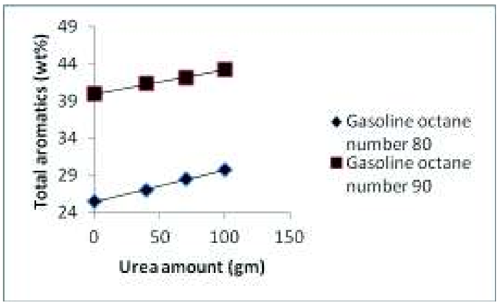


Figure 5. Effect of urea amount on %total aromatics of gasoline(100 ml gasoline, 100 ml methanol at 70°C for 10 hours)

3.5 Effect of urea amount on octane number

Fig. 6 shows the effect of amount of urea on octane number for gasoline (80and90) from figure it was shown that as the amount of urea increase from20 gm to100 gm the octane number increase from80to 82.8 for gasoline 80 and from 90.0 to 92.1 for gasoline 90 This is due to the fact as the amount of urea increase, the adducts formation increase and hence increase in the removal of paraffin which leads to increasing in octane number.

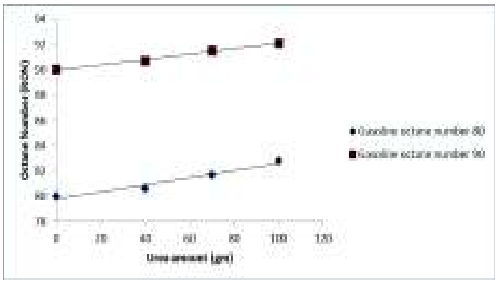


Figure 6. Effect of urea amount on octane number of gasoline (100 ml gasoline, 100 ml methanol at 70°C for 10 hours)

3.6 Effect of temperature on refractive index of gasoline

Fig. 7 shows the effect of the temperature on refractive index for gasoline (80and90)from figure it was shown that as the temperature increases from 20 to070c̊ the refractive index increases from1.4165to 1.4373 for gasoline 80 and from 1.4651 to 1.4794 for gasoline 90. This is due to the fact as the reaction mixture temperature increases, the adduct formation increases and hence increase in the removal of paraffin which leads increasing in refractive index.

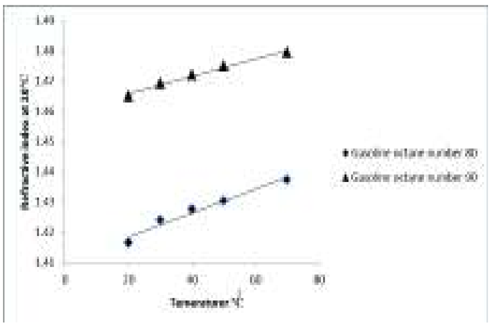


Figure 7. Effect of temperature on refractive index of gasoline (100 ml gasoline, 100 ml methanol and 100 gram urea for 10 hours)

3.7 Effect of temperature on density of gasoline

Fig. 8 shows the effect of the temperature on density for gasoline (80and90)from figure it was shown that as the temperature increases from 20°C to70°C the density increases from 0.7152to 0.7270 for gasoline 80 and from 0.7480 to 0.7624 for gasoline 90 This is due to the fact as the reaction mixture temperature increases, the adduct formation increases and hence increase in the removal of paraffin which leads increasing in densities .

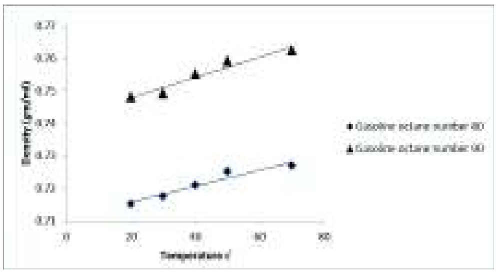


Figure 8. Effect of temperature on density of gasoline (100 ml gasoline, 100 ml methanol and 100 gram urea for 10 hours)

3.8 Effect of temperature on % total aromatics of gasoline

Fig. 9 shows the effect of temperature on total aromatics for gasoline (80and90) from figure it was shown that as the temperature increases from 20°C to 70 °C the total aromatics increases from26.21to 29.72 for gasoline 80 and from 40.0 to 43.21 for gasoline 90 This is due to the fact as the reaction mixture temperature increases, the adduct formation increases and hence increase in the % weight aromatics[10-11].

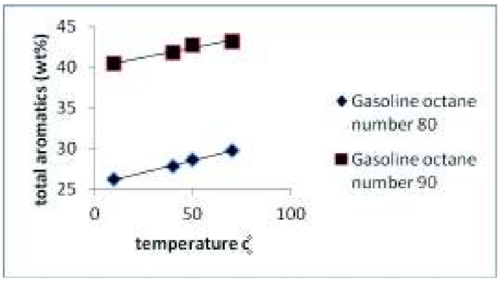


Figure 9. Effect of temperature on % total aromatics of gasoline (100 ml gasoline, 100 ml methanol and 100 gram urea for 10 hours)

3.9 Effect of time on refractive index of gasoline

Fig. 10 shows the effect of time on refractive index at 20c̊ for gasoline (80and90) from figure it was shown that as the time increases from 5 hr to 15 hr the refractive index increases from1.4154 to1.4405 for gasoline 80 and from 1.4621to 1.4812 for gasoline 90 This is due to the fact as the time of the reaction increases, the adduct formation increase and hence increases in the removal of paraffin which leads in increasing of refractive index of gasoline.

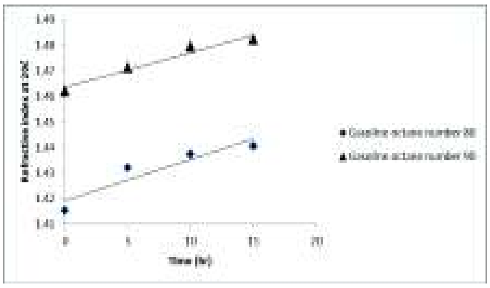


Figure 10. Effect of time on refractive index of gasoline (100 ml gasoline, 100 ml methanol and 100 gram urea at 70c̊)

4 CONCLUSIONS

In this study, the treated portion of gasoline was operated at different variables

a-Effect of urea amount.

b- Effect of temperature.

c-Effect of time.

And were investigated On Densities, Refractive Index, % n-Paraffin's, % total Aromatics and Octane number of oil produced it was found that by increasing the amount of urea,temperature, and the reaction time increases, the refractive index, density, total aromatics and octane number increase. While the n-paraffin's decrease.

REFERENCES

A.A.K. Abdul-Halim and N.S. Akram, "*The Effect of Operating Conditions of Urea Dewaxing on The Pour Point of Light Lubricating Oil"*, *Iraqi Journal of Chemical and Petroleum Engineering Journal*, 11, 2010.

A.K. Abdul-Halim and R.R. Yasin, "*Dewaxing of Distillate Oil Fraction 400-500°C Using Urea*", *Journal of Engineering*, Baghadad University, 13, (pp. 1268-1280), 2007.

A.K. Abdul-Halim, and R.R. Yasin ,"*Dewaxing of Distillate Oil Fraction 400-500° C Using Urea*", Journal of Engineering, Baghadad University, 13, (pp. 1268-1280), 2007.

B. Angla, Ann Chim., (12) 4, (p. 639) 1945.

É.L. Borovneva and A.G. Martynenko, "*Quality of Wax Obtained In Urea Dewaxing of Diesel Fuels In Relation To Choice of Solvent*", *Chemistry and Technology of Fuels and Oils*, 9(4), (pp. 262-263), 1973.

H. James Gary and E. Glenn Handwerk, "*Petroleum Refining Technology and Economics*", 4th edition, Marcel Dekker, Inc., New York, 2001.

I.C., Fetterly, Petroleum Refiner, 34 No. 4, (pp. 134-137) April 1955.

J.G. Speight," *The Chemistry and Technology of Petroleum*", 4th edition, CRC Press Taylor and Francis Group, LLC, 2007.

J.J. Connor and J. Boyd, "*Standard Handbook of Lubrication Engineering*", McGraw-Hill book Co., New York, 1968.

M. Ferund, R. Csikos, S. Keszthelyi and G.Y. Moze, "*Paraffin Products, Properities, Technologies and Applications*", Elsevier Scientific company, Amesterdam, The Netherlands, 1982.

N. Ya. Rudakova and N.I. Chernozhukov, "*Effect of Treatment of Dolinsk Petroleum With Urea on The Solidification Temperatures of Its Cuts*", *Chemistry and Technology of Fuels and Oils*, 8(1), (pp. 22-26), 1972.

N. Board, "*Modern Technology of Petroleum Greases, Lubricants and Petrochemicals*", NIIR Project Consultancy Services (NPCS), India, 2004.

Türkiye Taşkömürü Kurumunda Yeraltı Kömür Ocaklarının Üç Boyutlu Tasarımı Ve Maden Bilgi Sisteminin Kurulması

Three-Dimensional Design and Mining Information System in the Turkey Hard Coal Enterprises Underground Coal Mines

Y. Uzunkavaklı
Maden Yüksek Mühendisi, TTK İşletmeler Dairesi Başkanlığı, Zonguldak
T. Yiğit
Harita Teknikeri, TTK İşletmeler Dairesi Başkanlığı, Zonguldak

ÖZET Türkiye Taşkömürü Kurumunda, yeraltı imalat haritalarının bilgisayar ortamına aktarılması, bu yer altı yapılarının üç boyutlu (3D) görüntülenmesi ve her bir yer altı yapısına veri bağlanabilmesi amacıyla 4 yıl önce bir proje başlatılmıştır. Bir yazılım satın alınmıştır. Mevcut üretimlerin olduğu yeraltı imalat haritaları bilgisayar ortamına alınmış ve koordinatlarında sayısallaştırılmıştır. Yeraltının jeolojik yapısı, kömür damarları, üretim yerleri, galeri, kuyu gibi yeraltı yapıları ve tesisleri 3D oluşturulmuş ve üzerlerine verileri girilmiştir. Yapılan bu çalışma sonucu, Kurumun mühendis ve yöneticileri tarafından, Kurum içindeki intranet ağı ile bilgisayar ortamında, tüm yeraltı yapı ve tesislerini, üretim, iş güvenliği ve havalandırma bilgilerini 3D olarak takip edilebilmektedir. Her an üretim miktarı, çalışan işçi sayısı, sürülen galeri-taban miktarı, galeri ilerlemelerinde formasyonun özellikleri hakkında sorgular yapılabilmektedir.

ABSTRACT In the Turkish Hardcoal Enterprises (TTK), a project has been initiated to transfer these underground production maps into computer, to visualize the underground works in three dimension and to assign data to each of the underground works, started 4 years ago. Software was purchased. Firstly, the underground production maps related to existing production have been digitized. The geological structure of underground, coal seams, production panels, galleries, shafts underground structures and facilities have been three-dimensionally digitized and assigned data. This information can be accessed through the users TTK's intranet. As a result of this initiative; coal production location, all underground galleries and facilities, work safety application and ventilation information can be monitored online by the engineers and management of the TTK. Instantaneous queries can be performed regarding amount of production, number of workers, advances of galleries and gateroads and geological properties of the formations excavated.

1 ZONGULDAK TAŞKÖMÜRÜ HAVZASI

1.1 Tanım

Ülkemizde koklaşabilir taşkömürü üretimi sadece Zonguldak Kömür havzasında yapılmaktadır. Havzada 1848 yılından bugüne taşkömürü üretimi devam etmekte olup günümüze kadar yaklaşık 450 milyon ton taşkömürü üretimi yapılmıştır. Bugün ruhsat alanı 6885 km^2 dir (Şekil 1). Halen yeraltında kullanılan 400 km galeri ve en uzunu 750 m olmak üzere onlarca kuyu ve bür mevcuttur. Havzada yaklaşık 1310 milyon ton taşkömürü rezervi mevcuttur.

Şekil 1. Zonguldak Taşkömür Havzası sınırları

1.2 Kömür Damarlarının Jeolojik Tanımı

Havzada Türkiye Taşkömürü Kurumu (TTK) Armutçuk, Kozlu, Üzülmez, Karadon, Amasra müesseseleri olmak üzere 25 ocak ve ortalama 30-35 adet ayak (kömür üretim yeri) şeklinde üretim faaliyetini sürdürmektedir. Kömür üretim yerlerinin çalışma derinliği +100/-630 (metre) kotları arasıdır.

Havzada 47 adet işletilebilir kömür damarı mevcut olup kalınlıkları 0,8 m ile 10 m arasında değişmektedir. Eğimleri de 20-80 derece arasında değişerek kuzeye (Karadenize doğru) dalımlıdır.

1.3 Ocak Haritalarının Oluşturulması

1848 yılından bugüne yapılan üretim çalışmaları 1895 yılından itibaren ölçme ve haritalama çalışmalarıyla kayıt altına alınmıştır. 1908 yılında Fransızlar tarafından havzanın lokal koordinat sistemi oluşturulmuş ve tüm yer altı üretim faaliyetleri bu haritalara çizilmiştir. Halen üretim faaliyetlerinin planları da yine bu koordinat sistemi üzerine işlenmektedir.

Bu haritalar bez tabanlı kağıt, alüminyum tabanlı kağıt ve astragon haritalar şeklinde olup halen kullanılmakta ve oldukça yıpranmış durumdadır. Bugüne kadar 1/1000, 1/2000, 1/5000 ölçekli hazırlanan yaklaşık 1500 adet imalat haritası mevcuttur.

Bu haritalar üzerinde X,Y,Z koordinatlarını gösteren üretim alanlarıyla birlikte; yeraltı galerileri, Jeolojik - tektonik-hidrolojik bilgiler, kömür damarlarının isim-kalınlık-eğim-doğrultuları, ocağın havalandırma-gaz-yangın gibi iş güvenliği bilgileri, madencilik tesisleri- yeraltı yapıları, topoğrafik bilgiler, rezerv bilgileri, imar bilgilerini içermekte ve çok sayıda madencilik sembolleri bulundurmaktadır.

2 YERALTI HARİTALARININ BİLGİSAYAR ORTAMINA AKTARILMASI

2.1 Haritaların Bilgisayar Ortamına Aktarılması İhtiyacı

100 yıllık bilgi birikimini yansıtan ve oldukça yıpranmış olan bu haritaların, günümüz teknolojisinde bilgisayar ortamına alınarak kullanılması gerekli ve zorunlu hale gelmiştir. Bu haritalar madencilik faaliyetlerinin ve bu verilerin gelecek nesillere aktarılması açısından korunması gereken değerli belgeler niteliğindedir.

2.2 Harita Programdan Beklenenler ve Program Seçimi

Türkiye Taşkömürü Kurumunda tamamen yeraltı madenciliği yapılmaktadır. Kurum yöneticilerinin ve mühendislerin; ocak içinde başarılı bir yönetim ve çalışma uygulayabilmeleri için ocak içindeki teçhizatların, işçilerin ve çalışma yerlerinin bilinmesi gerekmektedir. Ocak büyüdükçe, hazırlık, iş güvenliği, havalandırma, makine tamir-bakımı, elektrik şebekesi kontrolü gibi konuların takip edilebilmesi ve ocağın tamamının denetimi de zorlaşmaktadır.

Bu tür gereksinimler sonucu; Kurumda, yeraltı imalat haritalarının bilgisayar ortamına aktarılması, yer altı yapılarının 3 boyutlu (3D) görüntülenmesi ve her bir yer altı yapısına veri bağlanabilmesi amacıyla 4 yıl önce bir proje başlatılmıştır. Bu projede;

-Yeraltının jeolojik yapısı (formasyonlar, kömür damarları, faylar, atımlar vb.)

-Yeraltında tesis edilen tüm yapıların (üretim panosu, üretim ayağı, galeri, taban, desandre, kuyu, su havuzu vb.)
-Bu yapılar içinde mevcut ve tesis edilecek tüm makine teçhizat ve donanımının (vinçler, nakliyat yolları, tulumba daireleri, elektrik şebekesi, basınçlı hava şebekesi, ayak içi nakliyat üniteleri) 3 boyutlu olarak oluşturulması, görüntülenmesi, değişimlerin aktif olarak güncellenmesi ve saklanması; 3D model üzerinde her türlü sorgunun yapılabilmesi; yerüstü tesislerinin de haritalar üzerine işlenerek yeraltı ve yerüstü tüm elemanların öznitelik bilgilerinin görülebilmesi amaçlanmıştır.

Bu çalışma kapsamında öncelikle madencilikte ve mekansal tasarımda kullanılan mevcut yazılımlar araştırılmış, Kurumun ihtiyacını karşılayan bir yazılım tespit edilerek alınmıştır. CAD tabanlı bu yazılım Kurum elemanlarının da çalışmalarıyla zaman içinde güncellenebilecek şekilde yeraltı haritalarına, yeraltı tesislerine ve yeraltı yapılarına uyarlanmıştır.

2.3 Haritaların Sayısallaştırılıp 3 Boyutlu Çizimi

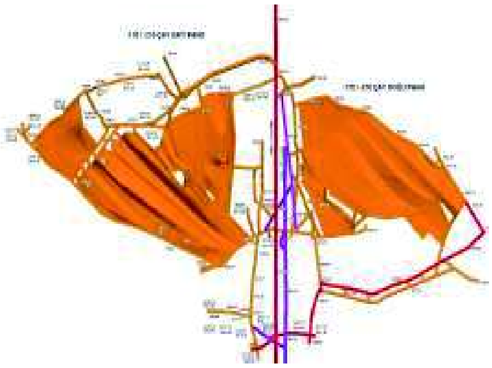
Öncelikle; mevcut imalat haritaları taranarak raster görüntü olarak bilgisayar ortamına aktarılmış ve bu raster görüntü ekran üzerinden haritalar üzerindeki herbir veri (galeri, pano, tabanyolu vb.) X,Y,Z koordinatlarında 3 boyutlu vektörize edilerek çizilmiştir (Şekil 2). Böylece her bir verinin tanımı ve boyutları tanımlanmış olup, yeraltının jeolojik yapısı, üretim yerleri, yer altı yapıları ve tesisleri 3 boyutlu oluşturulmuştur (Şekil 3).

Şekil 2. İmalat haritasının raster görüntüsü

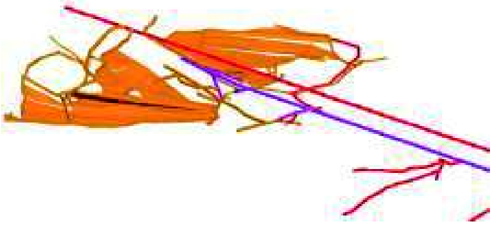
Şekil 3. Sayısallaştırılmış imalat haritası

2.3.1 Üretim panolarının ve galerilerin 3 boyutlu çizimi

Şekil 4 ve Şekil 5'te başka bir sayısallaştırılmış üretim panosunun 3 boyutlu plan ve perspektif görüntüsü verilmiştir.



Şekil 4. Sayısallaştırılmış imalat haritası

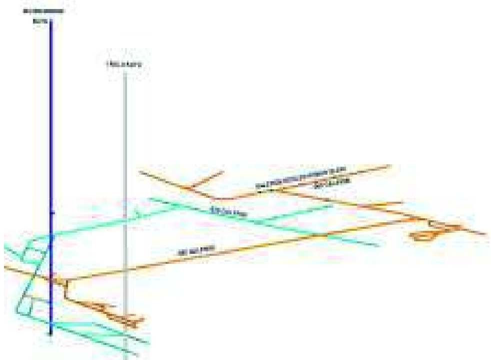


Şekil 5. Sayısallaştırılmış imalat haritasının perspektif görünümü

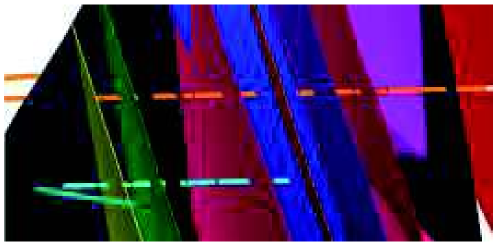
2.3.2 Jeolojik yapının, fay ve arızaların çizimi

Havzadaki kömür damarlarının her biri ayrı bir renkle tanımlanmaktadır. Kömürlerin K-G yönlü yatımı ve doğrultuları Şekil 6 daki kesitte verilmektedir. Ayrıca Şekil 6 da faylar tarafından kesilen kömür damarlarının 3 boyutlu görünümü verilmektedir. Şekil 7 de kuyu ve galerilerin 3 boyutlu görünümü verilmiş, Şekil 8 de de aynı galerilerin kestiği kömür damarları verilmiştir.

Şekil 6. Fay ve kömür damarlarının görünümü



Şekil 7. Galeri ve hazırlıkların görünümü



Şekil 8. Galerilerin kömür damarlarını kesimi

2.3.3 Sondajların ve kesilen formasyonların çizimi

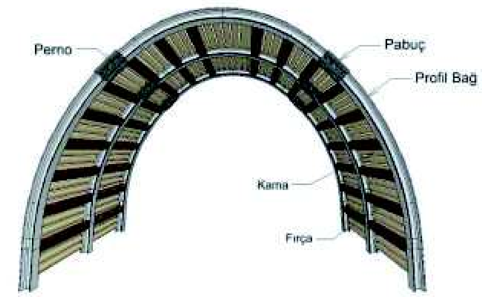
Galeri ilerlemelerinde kesilen formasyonlar karakteristik tanımları ile çizilebilmekte ve 3 boyutlu tanımlanabilmektedir (Şekil 9 ve Şekil 10).

Şekil 9. Galeride sondaj görünümü

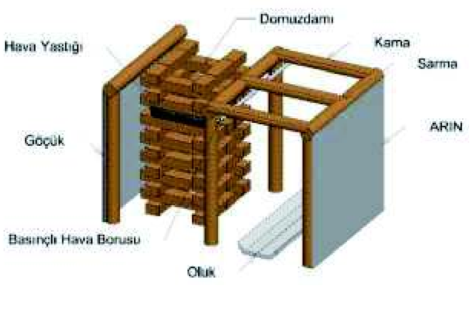
Şekil 10. Galeride kesilen formasyonların Görünüm

2.3.4 Ocak içi yapıların 3 boyutlu görünümü

Şekil 11 de galeri tahkimatının, Şekil 12 ayak tahkimatının ve Şekil 13 de galeride kesilen formasyonların 3 boyutlu görünümü çizilmiştir.



Şekil 11. Galeri tahkimatının görünümü



Şekil 12. Ayak tahkimatının görünümü

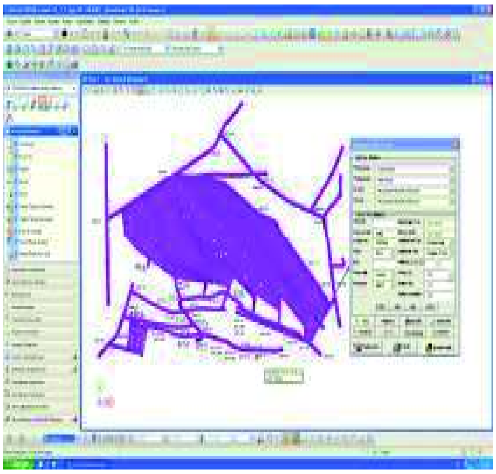
Şekil 13. Galeri tahkimatı ve formasyonların görünümü

3 MADEN BİLGİ SİSTEMİNİN KURULMASI

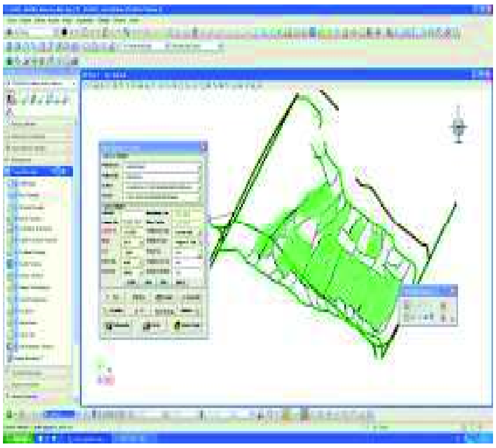
3.1 Yeraltı Üretim Panolarına, Galeri Ve Hazırlıklara Veri Bağlama

Yeraltı yapılarına sorgulama yapılabilmesi için, ayak, baca, pano, taban yolu, galeri, kuyu gibi her bir yapının üzerine verisi de tanımlanabilmektedir. Şekil 14 de üretim panosu ve taban yoluna veri bağlama, Şekil 15 de galeri ve hazırlıklara veri bağlama yöntemi verilmiştir.

Ayak, taban yolu, pano, galeri, kuyu, desandre gibi yeraltı yapılarının çizim esnasında bilgileri girilmekte, Kurumun mevcut veri tabanı ile ilişkilendirilmekte ve istendiğinde bu yapılar üzerinde sorgulama yapılabilmektedir.



Şekil 14. Pano ve taban yoluna veri bağlama



Şekil 15. Galerilere veri bağlama

3.2 Havalandırma Planlarının Oluşturulması ve Gaz İzleme

Ocak planlarının yanında; ocağın havalandırma, enerji dağılımı, nakliyat güzergahı ve basınçlı hava dağılımı planları da çizilmiş ve bilgisayar ortamında mühendislerin kullanımına sunulmuştur. Şekil 16'daki ocak planında temiz ve kirli hava yönleri çizilmiştir.

Şekil 16. Ocağın hava gidiş-dönüş yönleri

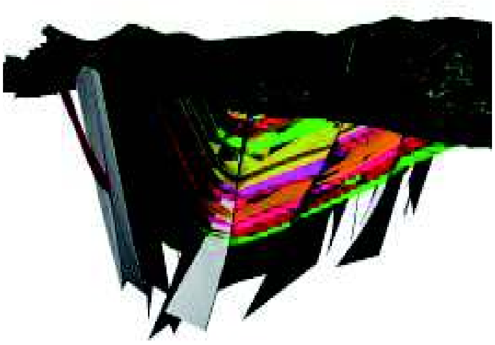
Şekil 17. Ocak planında iş güvenliği bilgileri

Şekil 17'deki ocak planında temiz ve kirli hava yönleri, galerilerden geçen hava miktarları ve ocak kaçış yönleri ve degaj yerleri de görülmektedir. Ayrıca bu planlara CO ve CH_4 sensörlerinin yerleri de konulmuş olduğundan sensörlerin izlenmesi ve takibi oldukça kolay (Şekil 18) yapılabilmekte ve herbir mühendisin bilgisayar ekranında görüntülenebilmektedir.

Şekil 18. Kozlu –560 katı sensör yerleri

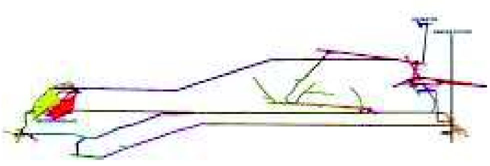
3.3 Yeraltı ve Yerüstünün Birlikte Görüntülenmesi

Havzada günümüze kadar yaklaşık 600 adet derin sondaj yapılmış olup, havzanın tüm yeraltı jeolojik yapısı belirlenmiştir. Bu sondajlardan alınan kömür damarlarının izleri de 3 boyutlu çizilerek, havzanın yer altı modeli oluşturulmuştur. Şekil 19 de küçük bir model verilmiştir.



Şekil 19. Kömür damarları ve fayların yeryüzü haritası ile görünümü

Kurumun herhangi bir Müessesesinin kuyuları, galeri ve üretim panoları olmak üzere tümü 3 boyutlu olarak koordinatlarında görüntülenebilmekte, çizim üzerinden koordinat, kot, uzunluk, hacim, rezerv bilgileri alınabilmekte ve istenilen ölçekte çıktı alınabilmektedir. Şekil 20 da Amasra Müessesesinin kuyu galerileri ve üretim ayakları 3 boyutlu verilmektedir.



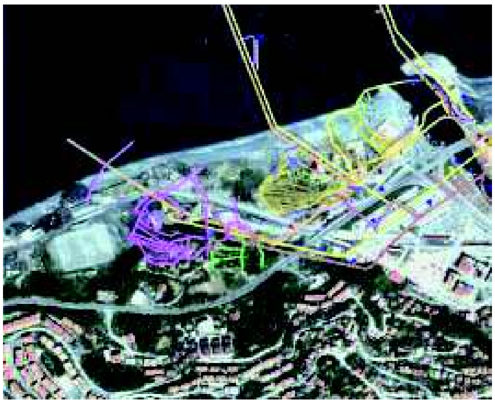
Şekil 20. Amasra Müessesesi ayakları ve galerilerin 3 boyutlu görünümü

Ayrıca, yer altı yapılarının üzerine yerüstü topoğrafik haritaları da koordinatlarında ilişkilendirilebilmekte ve yeraltı ve yerüstü haritaları 3 boyutlu bir ortamda değerlendirilebilmektedir

3.4 Yeraltı Üretimlerinin İmara Etkisinin Yansıtılması

Yer altı ve yerüstü yapılarının korunabilmesi, üretimlerin şehir altına giren kısımlarında madencilik tasmanından konutların etkilenip etkilenmeyeceği ve nerede üretim yapıldığı kolayca takip edilebilmektedir (Şekil 21 ve Şekil 22).

Şekil 21. Kozlu Müessesesi ocaklarının şehir altındaki görünümü



Şekil 22. Kozlu Müessesesi ocaklarının uydu haritasında görünümü

4 KURUMUN MÜHENDİSLERİNE SAĞLANAN HİZMETLER

4.1 Haritaların Kurum İçi İntranet Ağı İle Tüm Mühendislere Ulaştırılması

Yapılan bu çalışma sonucu, kömür üretim yerleri, tüm galeriler ve yeraltı tesisleri Kurumun mühendisleri ve yöneticileri tarafından bilgisayar ortamında 3 boyutlu olarak takip edilebilmekte ve her an üretim miktarı, çalışan işçi sayısı, sürülen galeri-taban miktarı hakkında sorgular yapılabilmektedir. Ayrıca galeri ilerlemelerinde formasyonun özellikleri hakkında da sorgular yapılabilmektedir

4.2 Yeraltı Proje Çalışmalarında Kullanımı

Ocak planlarının yanında; ocağın havalandırma, enerji dağılımı, nakliyat güzergahı ve basınçlı hava dağılımı planları da çizilmiş ve bilgisayar ortamında mühendislerin kullanımına sunulmuştur.

Yeraltında yeni üretim panolarının hazırlanması, hazırlıklar gibi ocak projelendirme çalışmaları 3 boyutlu yapılabilmekte, alınacak rezervin hesabı daha doğru ve kolay yapılabilmektedir.

5 SONUÇLAR

Türkiye Taşkömürü Kurumunda yeraltı imalat haritaları, yeryüzünden yapılan sondaj stamp verileri, yeraltı sondajları stampları, yeraltı damar izleri ve oluşturulan kesitler, yerüstü haritaları bilgisayar ortamında çizilmiştir. Bu haritalar Kurum içi intranet ağı ile mühendis ve yöneticilerin bilgisayarında görüntülenebilmekte ve yapılan çalışmalar takip edilebilmektedir. Yapılan çalışmalarla ilgili sorgulamalar yapılabilmektedir. Kurumda yeraltında yapılan her bir faaliyet mühendislerin bilgisine sunulmuştur.

Böylece Kurumda Maden Bilgi Sisteminin kurulması çalışmaları gerçekleştirilmeye başlanmıştır.

KAYNAKLAR

Kuşçu Ş., Şahin H., Akçın H. (2004) Maden İmalat Haritalarının Sayısallaştırılması ve Etkin Kullanım İmkanlarının Araştırılması. *10. Bilimsel ve Teknik Kurultayı. Ankara*

Gündoğdu i.B., Gökay M.K., (2002) Yeraltı Maden Ocaklarında Kullanılmak Üzere Geliştirilen Coğrafi Bilgi Sistemi. *Türkiye 13. Kömür Kongresi. Zonguldak*

Determination of Spontaneous Combustion Risk in The South African Coalfields

B. Genc

University of the Witwatersrand, Johannesburg, SA

A Cook

Latona Consulting Pty. Ltd., Johannesburg, SA

ABSTRACT Spontaneous combustion is a major challenge faced by the South African coal mining industry. Latona Consulting (Pty) Ltd. has obtained 86 coal samples between 2010 and 2012 from different coal mines in South Africa to determine the liability of spontaneous combustions in the majority of the South African coal mines. The database that Latona Consulting (Pty) Ltd. produced consists of information collected through several tests of coal samples from various mines over a three year period. These tests were conducted in order to determine both the Wits-EHAC index and the crossing point temperature which are combined to obtain the spontaneous combustion liabilities of the coal samples. This has provided a database of results, collated and maintained by Latona Consulting, to review and evaluate South African coal seams. Using this database the high-risk areas in terms of spontaneous combustion are identified and the results of the tests enable the South African coal mining industry to improve safety in coal mines.

1 INTRODUCTION

Spontaneous combustion is a major challenge faced by the South African coal mining industry. Latona Consulting (Pty) Ltd. has obtained 86 coal samples between 2010 and 2012 from different coal mines in South Africa to determine the liability of spontaneous combustions in the majority of the South African coal mines.

The spontaneous combustion tests were carried out at the University of the Witwatersrand (Wits University) to determine both the Wits-EHAC index and the crossing point temperature, which are combined to obtain the spontaneous combustion liabilities of the coal samples. This has provided a database of results, collated and maintained by Latona Consulting, to review and evaluate South African coal seams. Using this database the high-risk areas in terms of spontaneous combustion are identified and the results of the tests enable the South African coal mining industry to improve safety in coal mines.

2 SPONTANEOUS COMBUSTION TEST

At the School of Mining Engineering at Wits University, an apparatus was developed to measure the propensity of coal to spontaneously combust. This apparatus is used to test coals under predefined conditions and an index (Wits-EHAC) is obtained. Although the propensity of coal spontaneous combustion can be determined using various laboratory techniques, ignition temperature tests are used to study spontaneous combustion as they yield rapid results. Ignition temperature tests uses:

a) Crossing-point temperature (XPT), and
b) Differential thermal analysis (DTA)

In order to determine the XPT measurement, it is necessary to compare the relationship of how the temperature increases against time for coal and an inert material. XPT tests consist of samples of coal and an inert material both placed in similar sample holders and placed in an oil bath. Thereafter oil is heated at a constant rate so that the XPT can be measured. The XPT is defined as the temperature at which the coal sample equals that of an inert material sample (Gouws and Knoetze, 1995). When using DTA, the difference between the temperatures of the coal sample and an inert material sample is measured and plotted against the temperature of the inert material sample. It is important to understand that in DTA, three stages are realized. Initially, the temperature of an inert material sample is higher than the temperature of the coal sample (Stage l) and this is based on the cooling effect of the evaporation of moisture content in the coal. Next, evaporation of the moisture content; the coal sample starts to heat up at a faster rate than the heating rate of the inert material (Stage II) and this is based on the tendency of coal to self–heating and attempting to reach the temperature of the surrounding temperature (i.e. oil bath temperature). Finally the high exothermicity is reached at a point where the line crosses the zero base line and is referred to as the XPT (Gouws and Knoetze, 1995). Uludag et al, (2001) also confirmed the three stages and explained that stage I begins with minimal differential and gradually increases towards the XPT where the differential is zero. Stage II is a continuation from the XPT to the point called kick–point. Furthermore, Stage II is referred to as one of the best indicators of spontaneous combustion. Stage III is when the coal begins to burn beyond the kick–point. Figure 1 shows a typical differential analysis thermogram.

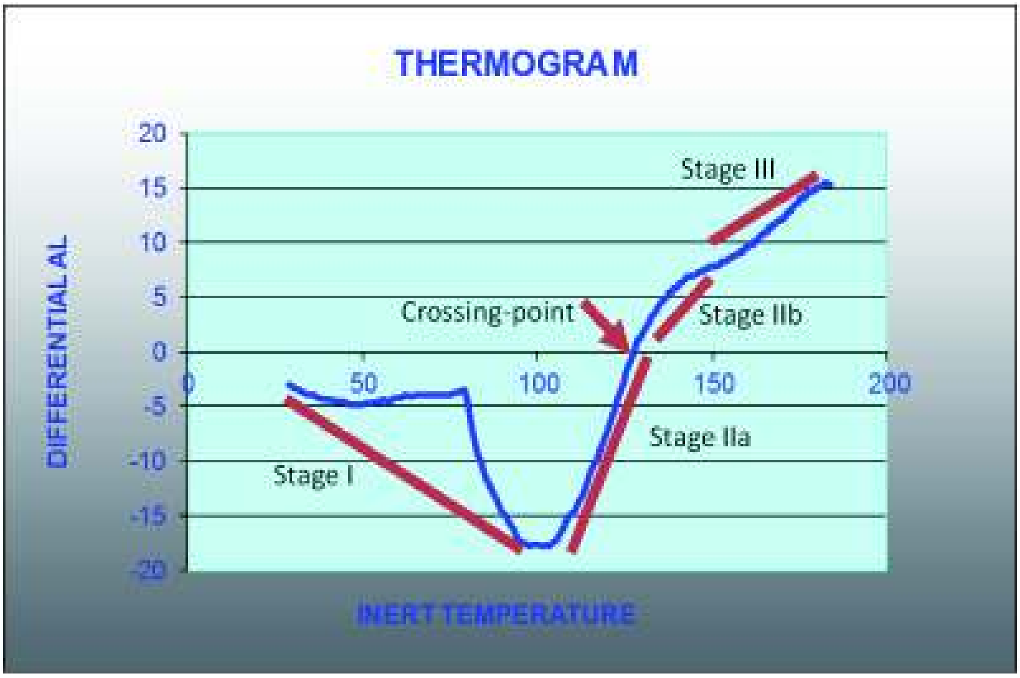


Figure 1. Typical differential thermogram

It is important to understand that the kick–point temperature means the temperature at which coal starts to burn. Stage II is a good indicator of spontaneous combustion liability and that the liability of spontaneous combustion increases with the increasing slope of Stage II. However, the most reliable indicator of spontaneous combustion is the Wits-EHAC Index. The Wits-EHAC Index is defined as:

Wits-EHAC Index = (Stage II slope/XPT) × 500 (Gouws, 1987)

According to Gouws (1987), the characteristics of the curves plotted using the obtained results (i.e. ignition temperature tests) are used to determine the propensity of coal to self-heat according to the Wits-EHAC liability index. It is important to understand that when an index value of coal is more than five, there is a high propensity to spontaneous combustion and when an index value is less than three, there is a low propensity to spontaneous combustion. An index value of between three and five indicates that the coal sample has a relatively medium risk to spontaneously combust. As indicated in Table 1, a higher index value represents a higher risk of a coal self-heating (Gouws, 1987).

Table 1. Spontaneous combustion liability index

Index	Spontaneous Combustion Liability
0-3	Low
3-5	Medium
> 5	High

The testing apparatus used for the Wits-EHAC Index consists of an oil bath, six coal and inert material cell assemblies, a circulator, a heater, a flow meter used for airflow monitoring, an air supply compressor and a computer. The oil bath is a 40L stainless-steel tank. The oil used has a low viscosity and high flash point and break down temperature. Generally, the oil used has a flash point of 210˚C and a maximum usable temperature of 320˚C. Figure 2 shows an apparatus setup of the Wits-EHAC testing apparatus.

The cell assembly consists of six cells whereby three of them are used for coal samples whilst the remaining three are filled with inert material such as calcined alumina. The cell assemblies have to be oil tight. It is important to understand that each cap used for closing the cells has a platinum resistance thermocouple (PRT) inserted through the middle. Fumes produced during heating of coal samples in the cells is released through the chimney built into the caps. Air supplied is directed to the bottom of the cells through a copper tube spiral. This spiral is long enough to allow air to reach the oil temperature before entering the cell. It is important to understand that the airflow rate is controlled using the flow meter located between the compressor and sample holder which are connected with a plastic tubing. Figure 3 shows the inserted platinum resistance thermocouple cell caps and the chimney. When preparing the coal samples, it is important to ensure that the coal remains as fresh as possible once taken from the mine. This is attained by storing coal in a sealed container until the preparation time of the cell assembly and testing arrives. Coal preparation is done using a glove box that is filled with nitrogen. The coal samples required size of (212 μm) is obtained through crushing and sieving. Thereafter, coal is weighed on an electronic scale. A 20g coal sample is used for each coal cell. When testing commences, the test environment must be initially at 30˚C temperature. This is achieved by sealing the chimneys on the caps until the temperature stabilizes. The test commences by switching on the heater, air compressor and then initiating the data capturing program on a personal computer (PC).

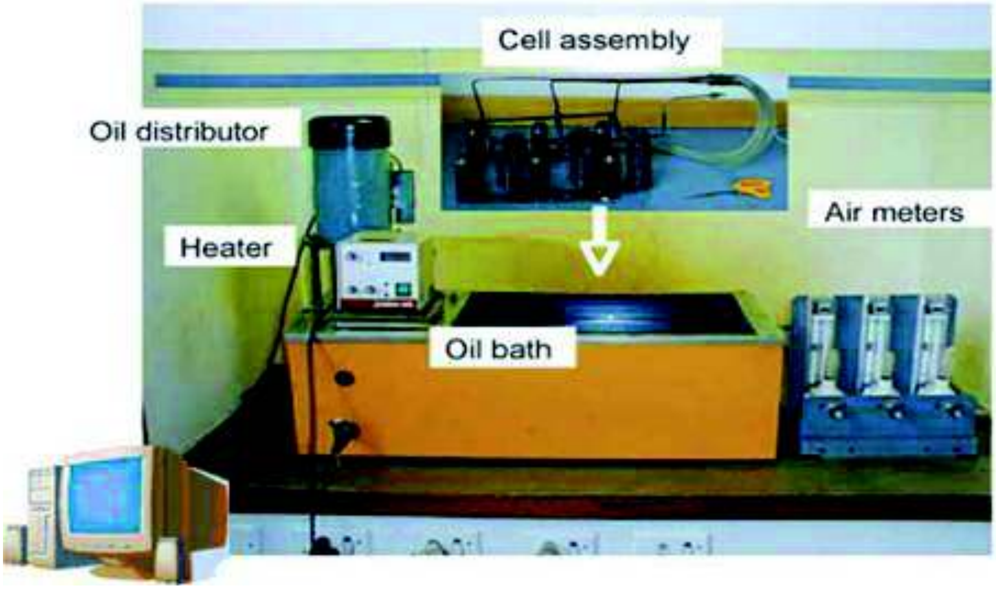


Figure 2. Wits–EHAC apparatus setup

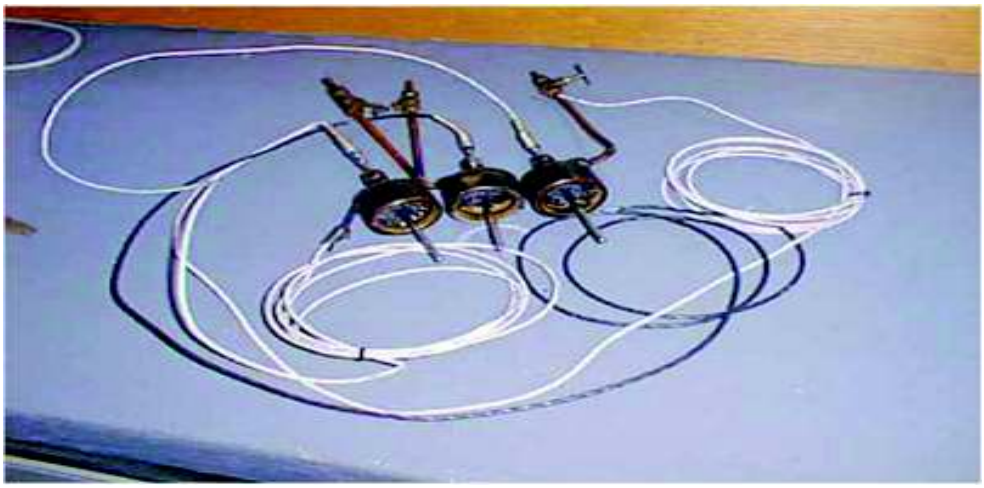


Figure 3. The cell caps with the inserted platinum resistance thermocouple (PRT) and the chimney

3 RESULTS

The test results entail the XPT in degrees Celsius and the Wits–EHAC index. The tests results are shown from 2010, 2011 and 2012 respectively. The names of the coal mines have been abbreviated.

Table 2. Spontaneous combustion test results (2010)

Name of the mine	Wits-EHAC index	Crossing point temperature (^{o}C)	Spontaneous combustion liability
Gr4	5.64	98.6	High
Gr5	5.44	102.6	High
Xs5	5.31	105.2	High
Ma	5.14	109.3	High
Ta	5.47	110.8	High
SW	5.26	117.3	High
Sl	5.58	104.8	High
D16	4.91	118.8	Medium
D15	5.51	102.9	High
Si	5.32	113.7	High
M1	4.64	117.8	Medium
NC	5.02	119.3	High
nK	5.38	108	High
Op	5.49	108.2	High
Op	5.23	120.9	High
Op	5.1	110.8	High
Op	5.52	113.6	High
Op	5.27	117	High
Wy	5.24	114.7	High
DE	5.6	121.2	High
VaA	5.58	95.2	High
KE	4.86	120.2	Medium
KW	4.71	113.9	Medium
Xs2	4.91	116.7	Medium
SW	4.92	121.9	Medium
VaG	4.86	108.5	Medium
Mo	5.33	124.7	High

Table 2 shows the 2010 tests results for spontaneous combustion where 27 tests were conducted. It is evident from the 2010 spontaneous combustion liability results that all of the tested coal samples from various South African collieries ranged from medium to high propensity of coal to spontaneously combust and most of the collieries have high propensity (20 out of 27). The minimum calculated Wits–EHAC index was 4.64 whilst the maximum index found was on 5.64 during 2010.

Table 3. Spontaneous combustion test results (2011)

Name of the mine	Wits-EHAC index	Crossing point temperature (^{o}C)	Spontaneous combustion liability
COA	3.14	161.9	Medium
COA	3.46	150.5	Medium
COA	3.58	154.3	Medium
TshM	3.63	145.8	Medium
TshV	3.65	144.8	Medium
TshG	3.76	137.4	Medium
KD	4.21	133.2	Medium
M3	4.21	134.6	Medium
KG	4.3	130.3	Medium
Sl	4.41	129.4	Medium
Sp	4.56	121.2	medium
SW	4.62	126.9	Medium
GGV4	4.66	128.1	Medium
Kr	4.67	126.2	Medium
Ta	4.73	124.3	Medium
Tu	4.73	129.3	Medium
Gr4	4.74	118.7	Medium
Sp	4.79	117.3	medium
Kr	4.81	114.2	Medium
Kr	4.81	114.2	Medium
COA	3.86	131.3	Medium
Gr5	4.9	112.9	Medium
Dr	5.01	118.7	High
GGV2	5.12	123.2	High
M2	5.18	125.9	High
M1	5.33	123.4	High
M1	5.36	126.8	High
M1	5.91	127.8	High

Table 3 shows 2011 tests results for spontaneous combustion where 28 tests were conducted. Although the similar range (i.e. medium to high) is still found from the 2011 spontaneous combustion liability results, most of the collieries tested were having medium propensity (22 out of 28). The reason for the difference in the test results is because every coal seam has different properties and this impacts the propensity of the spontaneous combustion. The minimum calculated Wits-EHAC index was 3.1, which is still above the low range identified by Gouws (1987), while the maximum Wits-EHAC index was on 5.91.

Table 4. Spontaneous combustion test results (2012)

Name of the mine	Wits-EHAC index	Crossing point temperature (° C)	Spontaneous combustion liability
AA	4.23	139.6	Medium
AA	4.29	141	Medium
AA	4.15	142.9	Medium
G8	4.53	131.4	Medium
Gr4	4.73	132.5	Medium
Kr	5.5	128.2	High
Kh5	4.84	133.7	Medium
Ar8	4.78	132.1	Medium
Ar8	4.41	127.2	Medium
Ar8	4.06	138.3	Medium
Ar8	5.75	126	High
Ar10	5.7	127.9	High
KiD	5.05	109	High
KiD	4.25	137.8	Medium
KiD	4.8	125.1	Medium
KiD	4.3	139.6	Medium
VeD	4.43	117.9	Medium
VeC	4.49	128.2	Medium
VeT	4.62	119.3	Medium
VeR	4.19	134.8	Medium
MoW	4.75	119.9	Medium
Vu	5.26	124.8	High
Mo	3.93	138.3	Medium
TCM	5.11	128.1	High
DCM	4.56	125.1	Medium
FZN	4.78	127.8	Medium
FZS	4.7	124.9	Medium
WKC	3.71	153.8	Medium
KE	4.04	143.3	Medium
KW	4.83	120.4	Medium
Tu	4.15	142.9	Medium

Table 4 shows 2012 tests results of spontaneous combustion where 31 tests were conducted. 2012 results showed a very similar trend compared to 2011 as most of the collieries tested were having medium propensity (25 out of 31). The results indicate that each tested coal sample has a different property and this is depended on the coal seam it originated from. This is the reason why medium results are more dominant in 2012. The minimum calculated Wits-EHAC index was 3.71 while the maximum Wits-EHAC index was 5.75.

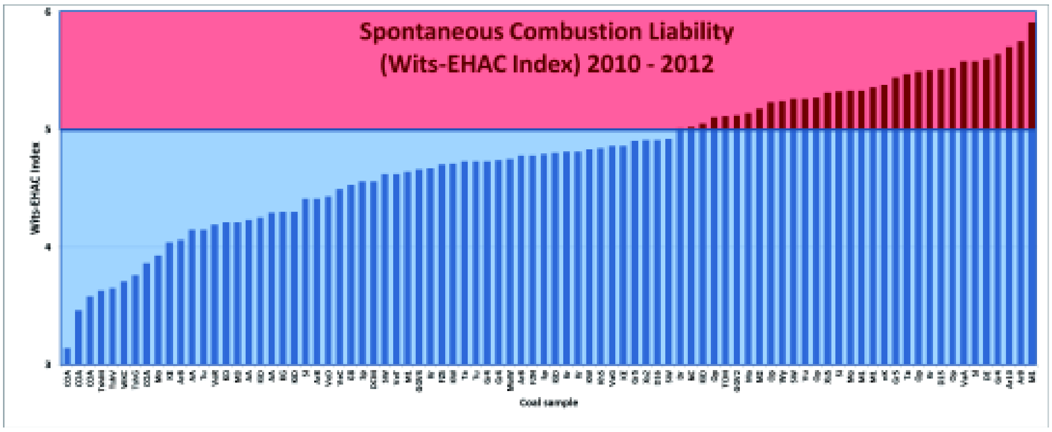


Figure 4. Spontaneous combustion liability (2010-2012)

The spontaneous combustion liability on Figure 4 shows the behaviour of the selected South African collieries with regards to spontaneous combustion. The Wits-EHAC index margins are indicated using colours. The blue part shows the coal samples that have a value of more than 3 but less than 5 which are known to possess a medium risk to spontaneous combustion. The red part indicates coal samples that have a value of more than 5 and are known to have a high risk to the propensity of spontaneous combustion. 54 out of 86 tested collieries possess a medium risk to propensity of spontaneous combustion (almost 63 per cent), whilst the remaining collieries (37 per cent) possess a high risk to propensity of spontaneous combustion.

4 DISCUSSION OF THE RESULTS

To address spontaneous combustion problems in the South African collieries, 86 tests were conducted and the results were rated according to the Wits-EHAC index. The results indicate that spontaneous combustion propensity is depended on the properties of each coal seam. Although in 2010, test results showed that more than 74 per cent of the mines were having high risk ratings, in 2011 the high risk ratings were just about 21 per cent, and in 2012 it was 19 per cent.

The results show that most of the collieries located in the Witbank and Highveld coalfields have a high risk to induce spontaneous combustion. The high risk areas also include the north eastern part of Ogies. The southern parts of Waterberg coalfields in the Ermelo area are also rated high in terms of risk. Although 34 out of 86 have a high risk to induce spontaneous combustion, most of the selected collieries possess medium risk ratings. Medium risk ratings can be seen around the northern parts of Ermelo as well as in the Kwa-Zulu Natal province where particularly anthracite coal is mined.

There were no low results recorded during the testing period from 2010 to 2012. These results have led us to re-visit the current definition of the spontaneous combustion liability index within the ranges of low, medium or high; but this requires a further study as to how Gouws (1987) defined these ranges, and if they need to be changed.

5 CONCLUSION

The state of spontaneous combustion of the selected South African collieries was analysed and classified through the series of laboratory tests. It is evident that South African collieries are predominantly challenged by spontaneous combustion. 86

tests were conducted between 2010 and 2012. Test results indicated that most of South African collieries have medium risk ratings and the propensity of spontaneous combustion of the collieries ranges from medium to high.

There is a significant conclusion made during the analysis of the spontaneous combustion test results which showed that there are no low results found, and this emphasizes the importance of monitoring spontaneous combustion in the collieries. However, there is a need to re-visit the spontaneous combustion liability index within the ranges of low, medium or high; but this requires a further study as to how Gouws (1987) defined these ranges, and if they need to be changed.

Based on the tests results, it was found that there are high spontaneous combustion risks in the Witbank and Highveld coalfields. Relatively low spontaneous combustion risk was also found in the Kwa-Zulu Natal province where particularly the anthracite coal is embedded as well as the northern parts of Ermelo.

REFERENCES

Gouws, M.J., Crossing point characteristics and Differential Thermal Analysis of South African Coals. M.Sc. Dissertation, University of the Witwatersrand, Johannesburg, 1987

Gouws M.J. and Knoetze T.P., Coal Self-heating and Explosibility. Journal of The South African Institute of Mining and Metallurgy, 1995, pp 39-41

Uludag, S., Phillips, H.R., Eroglu, N.H., Assessing Spontaneous Combustion Risk In South African Coal Mines Using A GIS Tool. Proceedings of the 17th International Mining Conference and Exhibition in Turkey, 2001.

South African Coal and Its Abrasiveness Index Determination: An Account of Challenges

N. Tshiongo* , A. Mulaba-Bafubiandi

University of Johannesburg, Faculty of Engineering and the Built Environment, School of Mining, Metallurgy and Chemical Engineering, Department of Metallurgy, Minerals Processing and Technology Research Centre, South Africa, POBOX 526, Wits 2050, South Africa

ABSTRACT Industry end users of coal like electricity generating stations have specifications on coal required in terms of reactive, chemical and physical properties; this includes the ash content, moisture, composition, hardgrove grindability index and abrasiveness index amongst many other properties. These properties affect each other including the overall coal properties and performance required during its specified usage. Some South African coals are known to be very abrasive, this causes operational challenges during the electricity generation combustion process as the coal abrades the plant equipment at a faster rate. Various South African coal samples were tested for abrasiveness index using the Yancey, Geer and Price (YGP) method. Results from these tests showed a lack of repeatability and reproducibility on the abrasiveness index values of coal samples. This lack of repeatability and reproducibility was observed in all coal samples tested. The same was found when either the same sample was tested in different laboratories or even when a mother sample was divided and tested repeatedly in one laboratory. Proximate and Ultimate analysis were conducted on the same South African coal samples for coal characterisation and classification. The size of the analysed sample; the size and shape, the degree of liberation of the abrasive coal component, and the interface between the abrasive component of coal and the blade surface are additional contributing factors. This study gives an account of challenges experienced and observed during the abrasiveness index determination of different South African coal samples. An attempt to holistically integrate the impact of main coal components contributing to the abrasiveness of coal will be presented.

Key words: Coal, abrasiveness index, Coal properties, holistic integration, coal composition

* Corresponding author. Tel.: +27(0) 11 559 6162 E-mail address: n.tshiongo@gmail.com

1 INTRODUCTION

Coal is composed of two main groups of materials, the organic matter and the inorganic matter. The organic matter also known as the maceral components are the part that defines the coal and its value in different utilization processes, and the inorganic matter also known as the mineral matter does not contribute anything to the value and utilization of coal; it is however the cause of unwanted abrasion, corrosive and erosive behaviour of coal (Ward, 2002).

South African coals are mostly of low quality with a significant amount of incombustible mineral matter. They are typically medium rank C bituminous coals rich in inertinite and contain high mineral matter content (Malumbazo, et al., 2012). General classification of coal samples can be done, but the actual quantitative proportions, modes of occurrence of mineral matter vary from one coal sample to the other. In South Africa 75% of electricity is generated from coal with ESKOM burning 110 million metric tons of coal in 2012. Power generation using coal is an expensive process, with the major share of cost being spent in mining and preparation stages (Papanicolaou, et al., 2004). The composition of the coal supplied to the power generation plant is reflected by the damage to coal handling machinery, weir of boilers, pulverising mills and other units of the plant (Bandopadhyay, 2010), (Wells, et al., 2005), (Wells, et al., 2004) & (Moumakwa & Marcus, 2005). This leaves the effectiveness of coal combustion as the remaining process of cost savings and this requires a thorough understanding of the effect of coal quality parameters (Ward, 2002), (Wells, et al., 2005), (Choudhury, et al., 2008), (Oman, et al., 2001) & (Van Dyk, et al., 2009) that affects electricity generation.

There are different criteria of classifying coal properties and quality depending on the influential factors like the coal rank, coal composition, coal mechanical and physical properties etc. (Oman, et al., 2001). Coal quality is specified in heating value, ash content, moisture and sulphur content, with additional properties like volatile matter, fixed carbon, ash fusion temperatures, grindability, abrasiveness index etc. The effects of the above coal quality properties includes fouling, slagging, abrasion, erosion and corrosion inside and on the parts of the electricity generation equipments which then affects the overall efficiency of the process.

Abrasiveness index of coal is affected by those minerals that are harder than steel like quartz and pyrite (Wells, et al., 2005)& (Wells, et al., 2004). Although nearly as hard as quartz, the abrasion- and erosive-weir damage caused by pyrite is significantly less than that caused by the same quantity of quartz (Bandopadhyay, 2010). Previous studies (Wells, et al., 2005) indicates that quartz is two to five times more abrasive than pyrite, and this was attributed to quartz being found as large and free particles whereas pyrite is often included in soft clays and coal matrix (Wells, et al., 2005)& (Wells, et al., 2004). But this was not proven to be true on worldwide coals and abrasive weir has also been linked to other variables like particle size and shape and also degree of inclusion (Wells, et al., 2005) as these can differ from one coal to the other (Wells, et al., 2004).The current work focuses on the investigations of abrasiveness index of South African coals, the effect that quartz and pyrite have on the abrasiveness index value and also give an account of challenges that were experienced in this investigation.

2 METHOD

2.1 Coal Characterisation and Classification

Different South African coals from various collieries around the country are used for this work. Proximate and ultimate analyses were done on all coal samples. The results were then used to classify and characterised coal samples from different areas into different groups according to their physical and chemical properties. These results were compared with results obtained from various international and national previously conducted researches on characterisation of various South African coals.

2.2 Abrasiveness index determination

Abrasiveness index of South African coal samples are identified using YGP method. This is done by placing 4kg of coal sample in a mill and mechanically rotating the sample with a steel blade of a certain mass and shape that stir up the sample during rotation. The weight of the blade is measured before and after stirring up the coal sample. Abrasiveness Index is the measure of how the coal sample abrades the equipment during its utilization in electricity generations, and this value is measured by the change in mass loss of the steel blade. Abrasiveness Index of coal is then measured in milligrams of weir blades per kilograms of coal processed.

Table 1: Coal characterisation as previously reported and according to the current study.

3 RESULTS

3.1 Characterisation of South African coals

South African coals are classified as "High Volatile Subbituminous C" using different properties in previously done researched both nationally and internationally, but they do not specify the list of important parameters that determines the actual group of characterisation. On this work some of the values identified for coal parameters are not on the general coal properties range for South African coal rank as averages on the values are used. All the coals listed on the table below are classified as high volatile bituminous coal, but in most of them the value ranges for parameters is stretched too wide.

Coal analysis (wt% dry basis)	Ranges for Sub-bituminous C	Coal1 (from Int)	Coal 2 (from Int)	Coal 3 (Witbank)	Coal4 (Natal)	Coal 5 (Ermerlo)	From this work (Ave)
Proximate analysis							
Ash		13.70	16.68	12.1	15.5	10.0	20.88
Volatile Matter	46 - 42%	34.94	22.85	33.6	12.2	33.8	31.86
Fixed Carbon		51.36	60.85	51.8	70.4	49.6	43.47
Ultimate analysis							
Carbon	76 - 78%	70.22	68.1	82.3	86.5	78.8	80.21
Hydrogen		4.90	3.49	5.3	4.1	5.0	5.52
Nitrogen		1.39	1.69	2.0	2.3	1.1	1.60
Total Sulphur		1.01	0.54	0.8	1.5	0.3	0.67
Oxygen	12%	8.78	7.47	9.6	5.6	14.8	11.99
Moisture	12 - 18%	-	2.44	2.5	1.9	6.6	3.79

[2] (I. Prieto-Fernandez, 2002), [2] (J. Barroso, 2006), [3] (Vassilev, et al., 2009) ,[4] (Vassilev, et al., 2009), [5] (Vassilev, et al., 2009)

3.2

Effect of quartz content on the abrasiveness index of South African coals

Quartz is the most common mineral in coal, and it occurs as angular to semi rounded grains that occasionally form clusters, making it more abrasive to electricity generating equipment components. Quartz fills cracks (Ward, 2002), forms lenses, encrustates or impregnates coal fragments, and even forms silicified pipes (Vassilev & Vassileva, 1996), giving clear access of quartz to equipment as it will be on the edges of coal particles. Quartz provides coal with an abrasive property due to its hardness (Bandopadhyay, 2010) and it contributes more as it is the most abundant mineral matter in coal which in this case, the degree of liberation would have less effect on the relationship with abrasiveness index.

Abrasiveness index of random quartz content coals was identifies and plotted on a graph to see how AI is affected by the quartz content. The figure above show the relationship between AI and quartz content of the Witbank South African coal samples. The AI shows a positive relationship with quartz with a very broad but positive trend in the plot. But the correlation between the two properties is very low at 0.37; this is an indication that there are other factors apart from quartz that affect the value of the AI for South African coals.

3.3 Effect of pyrite content on the abrasiveness index of South African coals

AI was measured at random pyrite content coal as it was done for quartz and the data is represented on the graph. The figure above show the relationship between AI and pyrite content of the Witbank South African coal samples. The two variables have a positive relationship as indicate by the positive slope, but their relationship trend is too broad.

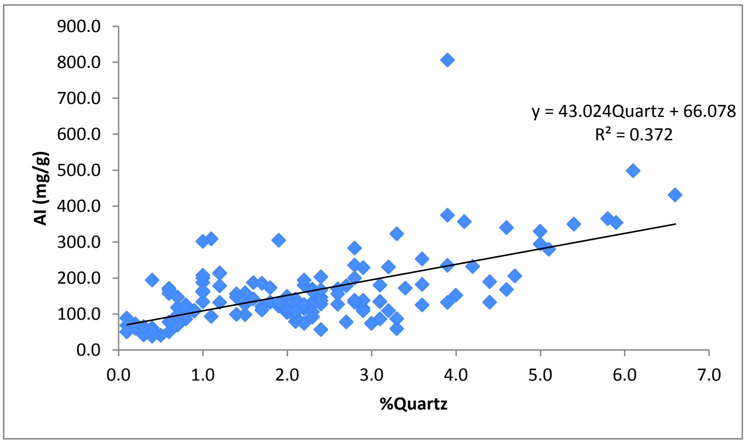


Figure 1. The effect of quartz content on the abrasiveness index (AI) values of South African coal.

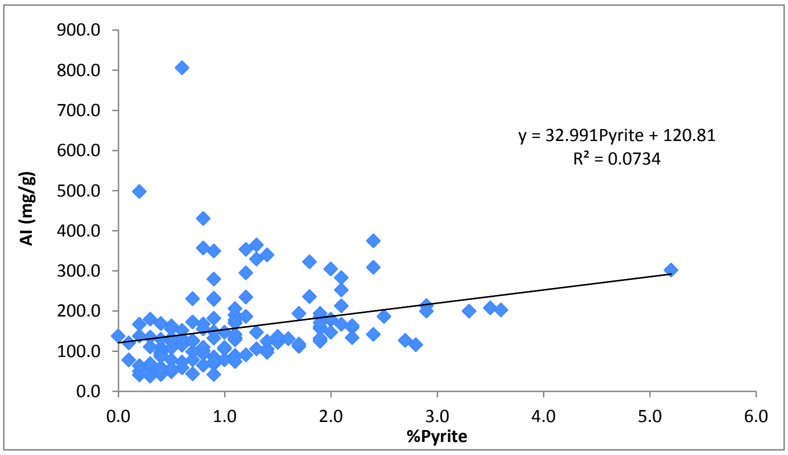


Figure 2. The effect of pyrite content on the abrasiveness index (AI) values of South African coal.

The correlation between AI and pyrite content is low at 0.07 and if it is compared with that of quartz at 0.37 it can be clearly be depicted that quartz has more effected on the abrasive behaviour of the Witbank coals that pyrite. This can be attributed to the differences in occurrences of pyrite, which is entrained in the coal particle and does not have direct contact with electricity generation equipment. The relationship with the AI would be changed depending on the degree of liberation as this would help expose pyrite; resulting in a direct contact with the metal blades and a high AI.

3.4 Relationship between ash content and the abrasiveness index of South African coals

The effect of ash content on the AI was investigated by determining the AI value at different and random ash content of South African coals. The results were plotted as shown on the figure above. The ash content has little effect on the AI of coal; AI values are on the same range despite the increase in ash content. The correlation of the two properties is closer to zero indicating the constant AI range on all ash content values.

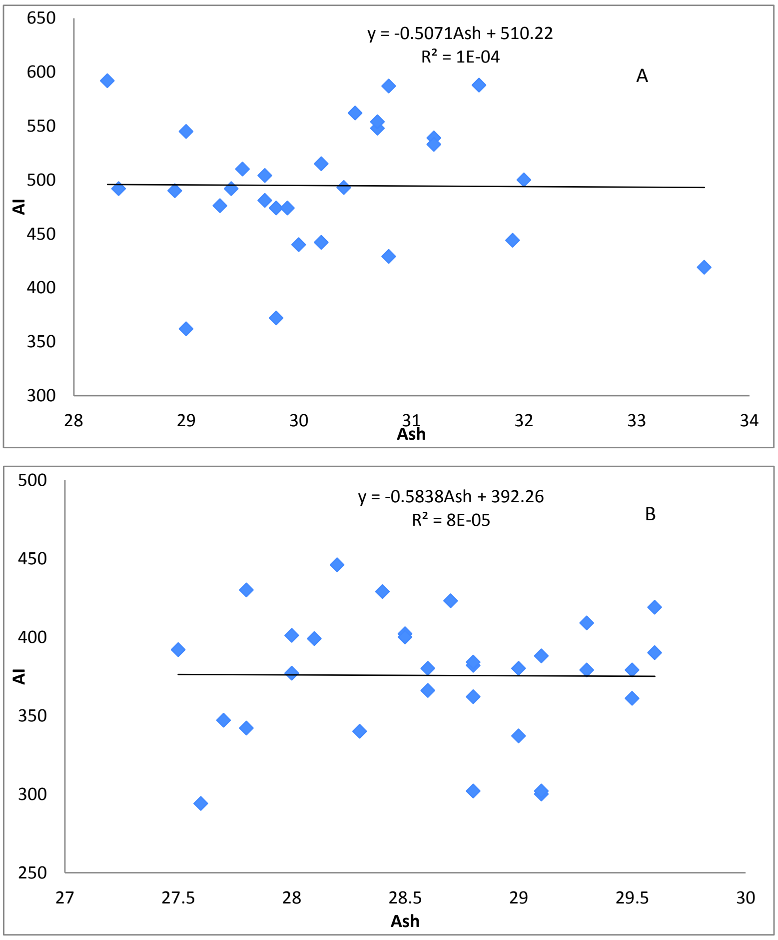


Figure 3. A&B The relationship between coal ash content and abrasiveness index (AI) of South African coals

3.5 Repeatability and reproducibility of abrasiveness index value

AI of the same coal was determined each day for a period of 30 days to assess the repeatability of the value. The figures above shows the lack of repeatability on the values of AI and the lack of reproducibility is presented by the R^2 value. The R^2 value for an ideal repeatability and reproducibility index should be zero which would represent the same AI value even when the measurements were done in different days. This behaviour can also be affected by the weir of the steel blade as it would be used repeatedly and the shape and contact with the coal sample will change and can result in changes on the milling and mechanism. The moisture content prior and after AI is not documented when there is also some moisture lost during transportation and handling of coal. Figure A and B are tests that were conducted in two laboratories. There has not been an indication of storage conditions (time and atmospheric conditions) that was employed before the AI test was done as oxidation can also affects coal properties and these properties affect the resulting AI value. This can also be the reason for lack of repeatability in the AI values. There are common variations that can be souses of lack of repeatability on the AI value when the tests are done in different laboratories. All experimental conditions are to be documented to make sure different laboratories are using the exact methods of determining the AI.

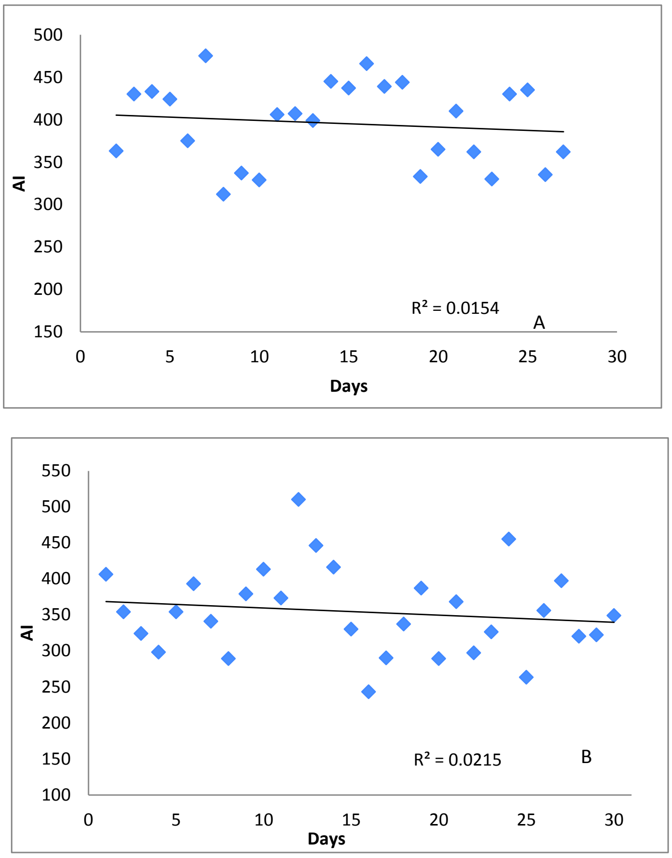


Figure 4. A&B Presentation of abrasiveness index (AI) repeatability and reproducibility

4 CONCLUSION

South African coals are subbituminous C coals with different properties as there are different coalfields in the country, and this provides different values for property analysis. There is a clear and visible indication of quartz having a more effect on the AI value than pyrite on South African coals but more investigation on the positioning of these phases in a coal particle can assist in understanding the relationship better. The results obtained shows that there is no relation between the ash content and AI on South African coal samples that were used for this study. There are challenges when it comes to AI determination and reasons behind the lack of repeatability and reproducibility of South African coals. A good correlation can be made comparing the same sample measurements that are done simultaneously using the same steel blades in terms of weir, size and shape. Clear documentation of experimental setup and methods are required to determine the difference in the AI values.

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REFERENCES

Bandopadhyay, A. K., 2010. A study on the abundance of quartz in thermal coals of India and its relation to abrasion index: Development of predictive model for abrasion. *International Journal of Coal Geology,* pp. 63-69.

Choudhury, N. et al., 2008. Influence of rank and macerals on the burnout behaviour of pulverized Indian coal. *International Journal of Coal Geology,* pp. 145-153.

I. Prieto-Fernandez, J.-C. L.-G. a. M. A., 2002. Determination of moisture in coal, in the case of discontinuous transport, using condensers. *Fuel Processing Technology,* Volume 75, pp. 129-140.

J. Barroso, J. B. L. M. F. a. S. J., 2006. Study of coa ash deposition in an entrained flow reactor: Influence of coal type, blend composition and operating conditions. *Fuel Processing Technology,* Volume 87, pp. 737-752.

Malumbazo, N., Wagner, N. J. & Bunt, J. R., 2012. The petrographic determination of reactivity differences of two South African inertinite-rich lump coals. *Journal of Analytical and Applied Pyrolysis,* pp. 139-146.

Moumakwa, D. O. & Marcus, K., 2005. Tribology in coal-fired power plants. *Tribology International,* pp. 805-811.

Oman, J., Senegacnik, A. & Dejanovic, B., 2001. Influence of lignite composition on thermal power plant performance Part 1: Theoretical survey. *Energy Conversion and Management,* pp. 251-263.

Papanicolaou, C., Kotis, T., Foscolos, A. & Goodarzi, F., 2004. Coals of Greece: a review of properties, uses and future perspectives. *International Journal of Coal Geology,* pp. 147-169.

Van Dyk, J. C., Benson, S. A., Laumb, M. L. & Waanders, B., 2009. Coal and coal ash characteristics to understand mineral transformations and slag formation. *Fuel,* pp. 1057-1063.

Vassilev, S. V. & Vassileva, C. G., 1996. Occurance,abundance and origin of minerals in coals and coal ashes. *Fuel Processing Technology,* pp. 85-106.

Vassilev, S. V., Vassileva, C. G., Baxter, D. & Andersen, L. K., 2009. A new approach for the combined chemical and mineral classification of the inorganic matter in coal.2.potential applications of the classification systems. *Fuel,* pp. 246-254.

Ward, C. R., 2002. Analysis and significance of mineral matter in coal. *international Journal of coal Geology,* Volume 50, pp. 135-168.

Ward, C. R., 2002. Analysis and significance of mineral matter in coal seams. *International Journal of Coal Geology,* pp. 135-168.

Wells, J. J. et al., 2004. The relationship between excluded mineral matter and the abrasion index of coal. *Fuel,* pp. 359-364.

Wells, J. J. et al., 2005. The nature of mineral matter in a coal and the effects on erosive and abrasive behaviour. *Fuel Processing Technology,* pp. 535-550.

Genetic Relationships between Oil, Gas and Coal Deposits as a Key to Discover New Hydrocarbon Deposits

V. Guliy
Lviv National University, Lviv, Ukraine

ABSTRACT Investigations of genetic relationships between oil, gas and coal deposits which are spatially associated together give chances for better understanding of their origin, development of exploration processes, possibilities of utilization of coupled methane, and ways to protect safety of underground coal mining. Nature of methane from coal-bearing sequences of the Donbass and Lviv-Volyn' coal basins (Ukraine) traditionally regards as local, and methane deposits here have been formed due to transformation of initial organic materials. Several important facts support a new point of view about necessity to accept a new way to determine an origin of methane. There are evidences on dept origin and permanent addition of the methane as consequence of unique gas deposits located under the coal-bearing sequences. Geological model on possible formation of the methane deposits from such sources is presented here. It will improve supplying by domestic fuels in regions with similar coal-bearing sequences.

1 GEOLOGICAL AND ENERGETIC BACKGROUND

Situation with own traditional fuel resources in Ukraine and estimated possible supplying periods of these sources according to native industrial demands has been analyzed. Beside of oil, gas, coal and uranium ores Ukraine has big additional fuel resources such as methane of coal-bearing suits, gas-hydrates, shale gas, and geothermal energy. Most realistic additional energetic resources at present time the authors connected with methane of the coal beds as well as using of gasification of coal by new proposed methods (Chepil' 2008, Guliy et al., 2007).

Ukraine produces a variety of types of coal. Coal reserves in Ukraine amount to around 47.1 billion tons (metric tons). The largest reserves of anthracitic coal (92%) are located in the Donets'k Coal Basin in the southeast of the country; most of the remainder is in the L'viv-Volyn Coal Basin in the West. A total of 244 operating mines were reported by IEA for the last time. Of these 239 were underground, and only five surface pits.

During last decades methane of coal deposits becomes more popular energetic source among other domestic fuels. For the Ukrainian coal basins methane resources have been estimated as 2.5Tm^3, according to relatively realistic calculations, and up to 25.0Tm^3 from very optimistic experts. Total recourses of the methane have similar scales if compare it to traditional natural gas deposits of Ukraine.

At the same time the methane is dangerous component during mining of coal at the dept levels. Very often it is difficult to ensure the fulfillment of necessary arrangements because of technical problems and geological factors. Solving these problems is very important for any country, which hopes to improve supply of energetic resources via domestic fuels. Very significant it is also for Ukraine as a result of methane and sands explosions in the coal mines, which happen very often.

To support safety of the underground mining there is a necessity to clean up and reduce contents of methane in the underground. The methane should be utilized during this process. On this way there is a big problem. Nature of methane from coal bearing sequences of the coal basins of Ukraine traditionally regards as local, dominantly organic in origin. According to this first idea methane deposits here have been formed due to transformation of initial organic materials and its modern resources are limited by total amount of organic sources. It is basement point of previous and modern strategy of providing with normal conditions of coal mining. The most common and effective methods of achieving this goal are ventilation with degassing and pre-gassing of separate blocks of the mine fields. But that previous idea about limited volumes of methane in coal bodies needs improving because there is evidence on deep origin of methane and its continuous addition. So, purposes of this study are:

1) Analysis of traditional hypotheses on methane nature and possible its volumes in coal seams.

2) Estimation of evidences on methane origin in coal-bearing sequences. It gives a chance to determine a valid strategy for safety of underground mining and methane utilization.

3) Provide facts on continuous addition of methane to underground mines and its depth nature.

4) Create a joint geological, geochemical and isotopic model of methane concentration in the Ukrainian coal basins.

5) Determine the most perspective areas for discovery of deep methane deposits, which are a source for methane within upper seams of coal.

2 MAIN INITIAL STATEMENTS

2.1 Spatial and Temporary Distribution of Methane and Its Isotopic Features

To create a new model of origin, migration and concentration of methane the author used such main initial statements. In the same area, oil and gas deposits of industrial scale as well as coal deposits are enlarged upon the same area. There is a coincidence in spatial distributions in the same area of oil, gas, and coal deposits, which have often industrial scales. In the underground coal mines, oil and gas are common. Appearance of oil and gas directly in the underground coal mines (Far East – (Miroshnikov 1968); Japan – (Kozlov 1960); etc.). Very often we can distinguish renewals oil and gas deposits with time or appearance of gas and oil in structures which suggested as having no prospects (Chepil’ 2008, Curliss 1999, Sokolov & Guseva). Very often we can see new portions of gas and oil in old deposits. Scale of hydrocarbons inflows is similar to the scale of industrial extractions. Practical consequence of this phenomenon is recalculation of the gas and oil resources estimated early (Chepil’ 2008). Isotopic compositions of the C from the methane support ideas about its deep source. Entering of methane and coal-dust in underground mines is repeated even after big scale ventilation with degassing and reducing of methane concentrations to safe levels.

2.2 Main Geological and Compositional Features of Coal-Bearing Sequences

Investigations of heat flows (Kutas 1978, Lepigov et al., 2008) show complicated pictures of isotherms location in different geological structures of Ukraine. The isotherm of 600^{o}C is indicator which suggests upper limits of methane generation.

Author revalued previous data on distribution of different sorts of coal in two main coal basins of Ukraine. Zoned localization of high and low quality coals is very clear for the Donbas region. Good example for illustration of spatial coincidence is distribution of oil and gas deposits in the Northern Donbas. Different in scale gas and oil deposits are disseminated among coal mines. In similar situation gas deposits (for example, Velyki Mosty and Lelykivs’ke gas deposits) and gas manifestations are distributed in the L’viv-Volyn’ Coal Basin.

3 DISPERSION AND SECONDARY HALOS AS MAIN INDICATORS OF METHANE MIGRATION

3.1 Main Features of Dispersion and Secondary Halos

Geological and geochemical investigations (Lepigov Orliv & Guliy, 2008; Lepigov Orliv & Guliy, 2008) show that oil and gas deposits and manifestations in coal-bearing sequences are accompanied by bitumen of different in compositions as well as numbers of secondary alterations. Author used these facts as a basement of geological model (Lepigov Orliv & Guliy, 2008; Lepigov Orliv & Guliy, 2008) of methane concentrations regarding ideas about primary and secondary halos.

Source of primary or dispersion halo is an independent gas deposit containing large resource. It is located in the low part of coal strata (Lepigov Orliv & Guliy, 2008). Three zones are determined in primary halo: 1) zone of gas (composition - mainly methane, with heavy isotopes of H, He, N, Hg), 2) zone of coal (^{12}C:^{13}C=99.0, 1H/D˜8300), 3) zone of bitumen (^{12}C:^{13}C=90.0 – 91.0, mainly light isotopes of N, S, Hg). Upper part of coal-bearing sequence will contain more disperse carbohydrates deposits or manifestations. Namely they can be the main source of gas components migration to coal mines.

Secondary halo or gas column has more wide distribution within rocks of coalbearing sequence. It is possible distinguish the gas columns by indicator characteristics, which include geochemical investigation of gas compositions, isotopic search of C and D from methane on separated depth levels sampling, etc. Most fine indicator which can mark limits of the gas column is Hg. In this sense there is a big similarity between secondary alteration of ore deposits and gas column for oil, gas, and coal deposits.

3.2 Isotopic Signatures of the Halos in Coal-Bearing Sequences

Generalizing of collected and published the $\delta^{13}C$ data from methane gives temperature limits for methane generation. Limits of data from 242 gas deposits of the former USSR and theoretical curve in connection with data from main gigantic gas deposits of the world determine field for $\delta^{13}C$ values, which are typical for mantle methane. Author founded more isotopically heavy composition of the methane carbon at the more depth levels. It is in accordance with tendency which determined for other regions (Curliss 1999, Kotarba 1988, Kotarba 1989, Rice 2000). General nature of oil and gas according these data is finding as deep originated under its migration along faults and crushed zones.

4 RESULTS AND DISCUSSION

4.1 Main Peculiarities of Methane Migration in the Ukraine Coal Basins

These noted indicators have been used to show possible mechanism of hydrocarbons migration and concentration in sedimentary sequences of the Donbas and L'viv-Volyn' Coal Basins. Author created a number of geological, geochemical, and isotopic models on possible origin and migration of methane to determine main zones of the Earth crust and local structures, which can indicate general source of the gas deposits within coal bearing sequences. Geochemical and isotopic peculiarities of evolution of hydrocarbon and accessory gases and some indicator chemical elements in different zones of the Earth for the Donbass region are established. Main temperature and pressure limits of hydrocarbon concentrations within these zones as well as inside primary and secondary dispersion halos are determined. As a result author suggested separate levels within coal-bearing sequences for studied regions. Possible stratigraphical levels of gas column locations have been found for two kinds of orogenies in the Donbass.

4.2 Gas Columns and Determining of Methane Origin in the Donbas and Lviv-Volyn' Coal Basins

We (Lepigov & Guliy 2008) examined the Donbas Coal Basin and discovered at least two areas with clear evidences on presence of the gas columns. They are located in the most dangerous parts of the Donbas, where methane explosions in coal mines happened very often. Due to more detailed studies on these areas we can determine relationships between structures, coal sorts, gas columns and find possible way to the primary gas deposits. For the western part of the Donets'k-Makiivka coal district we created a map to determine a main tendency of gas columns development.

Similar methods and indicators we used for analysis of the L'viv-Volyn' Coal Basin. In addition to the Velyki Mosty gas deposit clear gas column has been detected within Mezhyrichens'kyi coal district. From geological map we can see other possible gas columns. The northern-eastern end of these columns is located, probably, in the Lublin Hard Coal Basin, which is the Polish analogous of the L'viv-Volyn' Coal Basin.

5 CONCLUSIONS

Analysis of problems of methane origin in coal bearing sequences gives a chance to determine possibility of methane utilization and creation of safety conditions during underground mining of coal. It is shown that previous idea about limited volumes of methane in coal bodies needs improving because there is evidence on deep origin of methane and its continuous addition.

Several important geological, geochemical and isotopic facts support a nontraditional point of view for the Ukrainian Coal Basins about necessity to accept a new method to determine an origin of methane with receiving new precise data at local levels.

Searching gas deposits in the Donbas and L'viv-Volyn' Coal Basins has lately become the problem of current importance as a result of methane explosions in coal mines, which happen very often. It's used to be believed that methane accumulations were only because of coal strata degasification, but now this concept is inefficient to take preventive measures in underground mining. Other processes leading to high gas concentration and first of all its influx into mines from depths should be reviewed. One of the variants could be detection of an independent deposit containing large gas resource with big scale desorption from more disperse hydrocarbon deposits below the coal strata. Namely they can be the main source of gas migration to the coal mines. Discovery of main gas deposits, which are the main source for new portions of methane within coal-bearing sequences, will improve supplying by domestic fuels and safety of coal mining.

As we see, suggested way with exploring of gas columns is convenient instrument in struggle against the methane pollution out of mining areas as well as in exploration of new hydrocarbon deposits.

REFERENCES

Chepil', P. 2008. Second life of oil and gas deposits of Ukraine – miff or reality?, *Mineral Resources of Ukraine*, № 2, 37, pp. 37 - 38.

Curliss, W. 1999. The Mistery of Eugene Island. Science Frontiers. № 124, Jul. – Aug. 1999. Energy Information Administration Report DOE/EIA-0534 (U.S. Department of Energy. Washington, DC. 45p.

Guliy, V., Ozornoi, G. & Digonskiy, V. 2007. The role of own resources and new technologies to provide Ukrainian energy security and independent, *The Black Sea Security*, № 2 (6), 88, pp. 88 - 92.

Kotarba, M. 1988. Geochemical criteria for the origin of natural gases accumulated in the Upper Carboniferous coal-seam-bearing formations in Walbrzych Coal Basin, *Academy of Mining and Metallurgy Scientific Bull.*, 1199, 119p.

Kotarba, M. 1989. Isotopic geochemistry and habitat of the natural gases from the Upper Carboniferous Zacler coal-bearing formation in Nowa Ruda coal district, in *Advances in organic geochemistry*, edited by B. Durand and F. Behar (Oxford, Pergamon Press, Oxford), pp. 549 - 560.

Kozlov, V. 1960. Oil and gas deposits of Japan, *Geology of oil and gas*, № 5, 53, pp. 53 - 58.

Lepigov, G., Orliv, S. & Guliy, V. 2008. Gigantic gas deposit in Donbas (theoretical evidences), *Mineral Resources of Ukraine*, №3, 32, 32 - 33.

Lepigov, G., Orliv S. & Guliy, V. 2008.

Accumulation of hydrocarbons in the Donbas region in light of abiotic theory of their genesis, *Ukrainian Geologist*, №3, 73, pp. 73 - 79.

Lepigov, G., Orliv S. & Guliy, V. 2008. Geological model depth methane concentration in coal bearing sequences, *Geothechnical Mechanics, vol*. 80, 11, pp. 11 - 13.

Miroshnikov, L. 1968. Oil in coal mines of the Far East, *Transactions of USSR Academy of Sciences. Ser. Geology*. № 8, 125, pp. 125 - 127.

Rice, D. 2000. *Composition and Origins of Coalbed Gas*, U. S. Geological Survey, Search and Discovery Article # 40009, 35p.

Sokolov, B. & Guseva, A. 1993. On possibility of rapid modern generation of oil and gas, *Bull. Moscow University. Ser. 4. Geology*. № 3, 39, pp. 39 - 46.

Carbon Footprint of Coal Mining

A. K. Patra and B. K. Prusty
Department of Mining Engineering, Indian Institute of Technology, Kharagpur – 721302, India

ABSTRACT Carbon footprint is an indicator to assess the greenhouse gases (GHG) emitted from an activity or process. Both surface and underground mines, although contribute about 6% of global emission, has the potential to be significant contributors to overall GHG emission. In case of an underground mine, the operations associated the production of coal are the main causes of GHG emissions, primarily emission of methane. Inherent within the coal's structure, methane desorbs from the coal's internal surfaces during the mining process, and then moves to the atmosphere through a mine's ventilation system. The unit operations of opencast mining demands high energy and it directly and indirectly contribute GHG emission. Emissions from stock piles of coal/waste also contribute to carbon emission. In this paper different mining activities and their potential to contribute to GHG emission are discussed. Preliminary steps to assess carbon footprint of each of these mining activities are suggested.

1 INTRODUCTION

The impact of anthropogenic emission in causing the rise of earth surface temperature has been well accepted by the scientific community (Korre et al., 2010). Increased atmospheric concentration of greenhouse gas (GHG) is known to increase the global temperature by absorption of reflected infra-red radiations from earth surface. The prominent GHGs are carbon dioxide (CO_2), methane (CH_4), nitrous oxide (N_2O), chlorofluorcarbons (CFCs), Hydrochlorofluorcarbons (HCFCs), Perfluorcarbons (PFCs) and Sulfur hexafluoride (SF_6). It is estimated that, on an average, 43% of the total CO_2 emissions each year between 1959 and 2008 remained in the atmosphere. This fraction is expected to increase as the capability of absorption of natural carbon sinks (Ocean and land) is gradually decreasing (Le Quere et al., 2009). From a level 315 parts per million (ppm) in 1958 (when the scientific measurement of CO_2 level started), CO_2 in atmosphere has reached 392.81 ppm in November 2012 (NOAA, 2012). In the last 5 year atmospheric CO_2 level has increased from 383 ppm to 393 ppm, a rise of 10 ppm (Fig. 1).

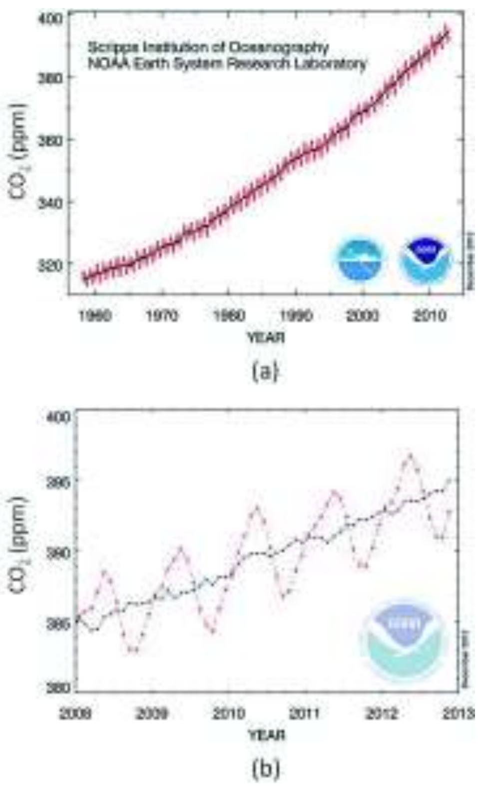


Figure 1. Atmospheric CO_2 level as measured at Mauna Loa observatory: (a) Since 1958, and (b) Last 5 years (Source: http://www.esrl.noaa.gov/gmd/ccgg/trends/)

The oceans play an important role in the Earth's carbon cycle. They are the largest long-term sink for carbon and have an enormous capacity to store and redistribute CO_2 within the system. Oceanographers estimate that about 48% of the CO_2 from fossil fuel burning has been absorbed by the ocean (Sabine et al., 2004). The dissolution of CO_2 in sea water shifts the balance of the ocean carbonate equilibrium towards a more acidic state (i.e., with a lower pH). This effect is already measurable (Caldeira and Wickett, 2003), and is expected to become an acute challenge to shell-forming organisms over the coming decades and centuries. Although the oceans as a whole have been a relatively steady net carbon sink, CO_2 can also come out of the oceans depending on local temperatures, biological activity, wind speeds, and ocean circulation.

1.1 Carbon Footprint

Over the years carbon footprint has been defined in several ways. A general way of defining it is that the carbon footprint is a measure of impacts of human activities on environment. The activities here not only include industrial activities, but it also includes daily activities like travel, use of laundry, etc. (Global footprint network, 2007; Patel, 2006). A few definitions suggest that it must include all the direct and indirect emissions from an activity (Energetics, 2007; Grub and Ellis, 2007). On the other hand some suggest indirect emission should not be included as it might not be quantified properly (Carbon Trust, 2007). For example, the processes involved in manufacturing the product which can be directly associated with the product should be included; but not the indirect emissions from commuting of workers to the factory (Carbon Trust, 2007). From an individual process or product approach, it is suggested that the carbon footprint must include emissions across the life cycle of a process or product, i.e. emission from the process involved in conversion of raw materials to finished product and until the finished product reaching the market (Carbon Trust, 2007; Grub and Ellis, 2007; Haven, 2007; POST, 2006).

While one approach is to limit the carbon footprint estimate to the emission of CO_2 only (Energetics, 2007; Grub and Ellis, 2007), there is another view that all the greenhouse emission from anthropogenic activities in terms of CO_2 equivalent should be included so that global warming potential, the ultimate objective of knowing carbon footprint, is estimated (Patel, 2006; Carbon Trust, 2007; ETAP, 2007; POST, 2006). Whether some emissions, like carbon monoxide (CO), which do not have global warming potential at the point of emission, but may get converted to a greenhouse gas such as CO_2, will be included is still debated. Similarly, debate may be extended as to whether the natural emissions of greenhouse gases (e.g. CO_2 from soil, CH_4 from soil and marshy land) should be included (Wiedmann and Minx, 2008).

Wiedmann and Minx (2008) have proposed a comprehensive carbon footprint definition as a measure of the exclusive total amount of CO_2 emission that is directly and indirectly caused by an activity or is accumulated over the life stages of a product. This definition includes activities of an individual as well as group activities of organisation, industry sectors, communities, etc. and it includes all direct (on-site, internal) and indirect emissions (off-site, external, upstream/downstream).

1.1.1 Primary and secondary footprint

For any activity under consideration, the carbon footprint that can directly be associated with it is termed as primary footprint. Carbon release to the atmosphere from any upstream process/activity or downstream use of the product is termed as secondary footprint. For example, in case of mining, the CO_2 emitted from the activities like overburden removal, blasting of mineral, transport of mineral etc, can be included under primary carbon footprint. Carbon released due to the felling of trees prior to the start of mining and CO_2 released on the process of transport of mineral form the mine site to the downstream user can be put under secondary footprint (Cristobal el al., 2010). Therefore while the carbon emitting potential of activities associated with mining gives the primary footprint, the life cycle assessment approach of the mineral gives the secondary footprints, although it is more difficult to quantify. Cristobal el al. (2010) have shown that the secondary footprint can have no less relevance towards the final emission factor.

1.1.2 Life cycle assessment (LCA)

The life cycle approach includes all possible processes that give rise to carbon emission, i.e., direct (on-site, internal) and indirect emission (off-site, external, embodied, upstream, downstream etc.). Through this approach, the cumulative GHG emission of a product or process is estimated through all stages of its life. For example, in case of mining, it takes into account energy inputs and emission outputs throughout the whole production chain from exploration and extraction of mineral to processing, transport and final use.

1.1.3 Unit of measurement of carbon footprint

Several alternate units can be thought of such as tonnes of carbon emissions or tonnes of CO_2 emission or tonnes of CO_2 equivalent (CO_2-e) emission per unit time (e.g. day, year). Often it is measured as kilograms or tonnes of CO_2 emitted per person activity (Hammond, 2007). This is also expressed with reference to area unit such as kg CO_2-e $year^{-1}$ m^{-2} or simply kg CO_2-e m^{-2} (Day et al., 2010). The expression of footprint on the basis of area unit can introduce errors into the estimate because of number of assumptions to be made for the land use type. Therefore some researchers suggest that this may be avoided by expressing the carbon footprint in terms of mass units (kg or tonne) (Keuning, 1994; Stahmer, 2000; Wiedmann and Minx, 2008). However, it certain cases, such as emission from a spoil heap of a mine, expression of footprint in area units can provide more physical meaning than just expressing it in terms of mass unit (Day et al., 2010). Therefore the choice of unit of expression depends on the process that is being studied.

2 MINING AND CARBON FOOTPRINT

The production and combustion of fossil fuels as an energy source release large volumes of greenhouse gasses to the atmosphere. Specifically, CO_2 and CH_4, both well known greenhouse gasses, have direct ties to the coal industry. Carbon dioxide, produced from the combustion of carbon, represents the most prevalent anthropogenic greenhouse gas. Combustion of one ton of coal produces between one and three tons of CO_2, dependent upon the carbon content and heating value of the combusted coal (Kelm et al., 2010). In 2006, the United States released approximately 2,100 million metric tons (Mt) of CO_2 from combustion of coal (EIA, 2009). The EIA (Energy Information Agency) estimates that China's 2006 coal-related CO_2 emissions to be approximately 4,900 million metric tons (EIA, 2010). World total coal-related CO_2emissions are expected to increase at rate of approximately 1.7 % per year through 2030, with China accounting for 74 % of the increase (EPA, 2009).

An economic analysis indicates that the energy value of methane emitted from Australian coalmines is approximately 36 million GJ or enough energy to produce 467 MWh of power at 40% efficiency (modern gas turbine or reciprocating engine) (Wendt et al., 2000; Dave and Duffy, 2002). At 42 cents per kWh, this equates to $67.2 million worth of power per year. While tracking and reporting total energy usage has already been undertaken by some mining companies as a part of energy audit, the use GHG intensity as a key performance metric has yet to be practiced. Therefore estimate of GHG emission potential of a coal mine is required when it is estimated that coal mining contributes to 8% of global anthropogenic methane emissions, and CH_4 is 21 times more potent than CO_2 in trapping heat.

USEPA estimated in 1994 that global emissions from coal mining in 1990 were in the range of 24 - 40 Mt (USEPA, 1994). Globally, CMM constituted 6 percent of anthropogenic methane emissions in 2005 with 388.14 Mt CO_2-e. CMM emissions have dropped since 1990 but are expected to rise somewhat by 2020. The 2006 projections of global annual mine methane emission is presented in Figure 2.

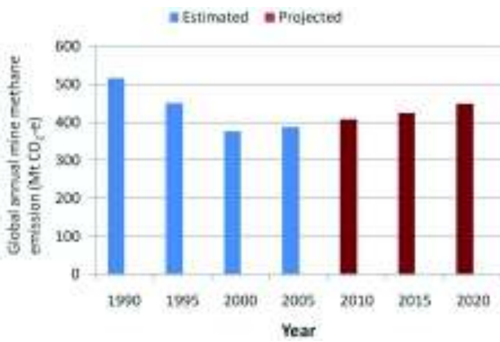


Figure 2. Global annual CMM emissions *(Source: USEPA, 2006)*

Estimates by an US study involving 30 mine sites showed that 4.669 Mt methane emissions took place from all major mining sources in US in 1995. The estimated emissions contributions from individual mining sources were as follows: underground mine shafts and portals (49%), methane drainage systems at underground mines (24%), coal handling of underground mined coal (15%), abandoned underground mines (4%), coal handling of surface mined coal (3%), and surface mines (2%) (Kirchgessner et al., 1997). Recent estimates show that active coal mines account for nearly 10% US anthropogenic methane emissions (Beedie, 2011).

2.1 Underground Coal Mine

In case of an underground mine, the operations associated the production of coal are the main causes of greenhouse gas emissions, primarily emission of methane. Inherent within the coal's structure, methane desorbs from the coal's internal surfaces during the mining process, and then moves to the atmosphere through a mine's ventilation system. Underground coal mining releases more methane than surface or open-pit mining because of the higher gas content of deeper seams. Among several factors on which methane emission from a coal mine depends, the important ones are coal rank, coal seam depth, and method of mining. Usually with increase in rank of coal, the CH_4 production increases as higher rank coal has gone through more transformation during the coalification process. Because pressure increases with the depth of the coal seam and the adsorption capacity of coal increases with pressure, deeper coal seams generally contain more methane than shallow seams of the same rank. Over time methane can be released to the atmosphere from near surface coal seams through natural fractures in overburden strata. Coal extraction tends to lead to the release of more methane than was originally trapped within the mined coal seam itself, because the drop in pressure draws in additional gas from surrounding strata. Also, the mining process tends to fracture the surrounding strata including neighbouring seams, particularly where extraction proceeds and aids in releasing more trapped methane (Irving and Tailakov, 2013).

An Australian study estimates about 64% of total CH_4 emission from an underground coal mine is associated with the ventilation air stream. Known as ventilation air methane (VAM), the concentration of methane in VAM varies from 0 to 2% (Wendt et al., 2000; Dave and Duffy, 2002). Remaining 36% of methane emissions are associated with the mine drainage gas, which has greater than 50% methane by volume. In India where methane drainage is not practiced, this could lead to addition to VAM or simple leakage to the surface through cracks. A study undergoing for CH_4 in the sealed off areas (CMM) also indicate that the concentration of methane in these voids could be as high as 100%. Methane released during coal mining operations is responsible for approximately 6.5% of Australia's greenhouse gas emissions and estimates indicate that 72% of the GHG emissions attributed to the coal mining industry are in the form of fugitive emissions from underground mining (Dave and Duffy, 2002). In Australia over 80% of the emissions come from just 14 gassy coalmines and the number of gassy mines is destined to increase as more mines develop into deeper, gassier coal seams (Wendt et al., 2000; Dave and Duffy, 2002).

Gassy coal seams are prevalent in the United States. Specifically, deep coal seams located throughout the Black Warrior and Appalachian Basins can possess gas contents greater than 300 cubic feet per ton. Comparatively, gas content's of deep coal seams located in China's Quinsui basin can exceed 500 cubic feet per ton. Additionally, these coals are thicker than those in the US, providing even greater gas content per unit area. Carbon dioxide equivalent methane emissions for the United States and China are respectively estimated to be 72,000 and 130,000 t (EPA, 2009).

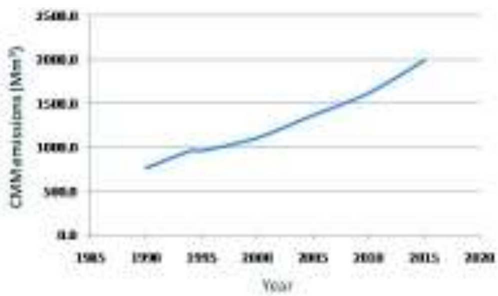


Figure 3. CMM emissions from operating underground coal mines of India - estimates and projections *(Source: USEPA, 2006)*

With more than 18 Degree III coal mines (Mines with rate/volume of emission of inflammable gas per tonne of coal produced is higher than 10 m^3) in India, methane from mines can be significant source of GHG. Figure 3 summarizes India's estimates and projections of CMM emissions.

2.2 Opencast Coal Mine

Opencast mining operation consists of process like removal of overburden to expose the mineral, extraction of mineral (includes blasting and excavation of mineral), transport of mineral to stock yard/processing plant within the mine lease hold and final despatch of the mineral to end user. Very few emissions measurements have been taken at surface mines. The limited data suggest that the primary sources of emissions include seam areas fractured by coal blasting, the lower most portions of surface mine pits (the pit floor), and inactive pits (Piccot et al., 1995). In general, the strata overlying the coal at surface mines do not appear to be a significant source of emissions but, as in underground mines, emissions may be contributed from underlying seams, faults or gas bearing reservoirs. Historically, methane emissions from surface mines have been thought to be much lower than from underground mines because of the lower gas contents associated with these relatively young and shallow coals. However, in absence of widespread measurements, the GHG emission potential of surface mine cannot be underestimated.

At surface mines, methane escapes from newly exposed coal faces and surfaces, as well as from areas of coal rubble created by blasting operations. Additional CH_4 may come from the overburden, which is broken up during the mining process, and the underlying strata, which may be fractured and distressed due to the removal of the overburden. Emissions per ton of coal are generally much lower from surface mining than from underground mining. Methane emissions also occur during post-mining handling, processing, and transportation. Some CH_4 is released from coal waste piles and abandoned mines. Emissions from these sources are believed to be low because much of the CH_4 would likely be emitted within the mine (Irving and Tailakov, 2013).

2.2.1 Energy use

Emission from the energy use in an opencast mine is one of the major sources of GHG emission. A typical estimate shows that one million tonne of base metal ore production involves the energy use that produces about 32,000 t of CO_2-e from diesel and electricity usage. In a coal mine, in addition to energy-use related GHG emission, emission of methane from coal seams constitutes a large share of total emission. It is estimated for one million tonne of coal production, about 74,000 t CO_2-e emissions takes place from diesel, electricity and seam gas emissions (Orica, 2013) (Fig. 4).

Several factors combine to increase total energy use and emissions, including increased stripping ratios, greater mining depths (requires longer hauls), lower coal grades and increased production. A key measure, therefore, should be the energy efficiency and targets set for the reduction of emissions per unit of coal mined. In many cases, especially where stripping ratio is high and quality of mined coal is low, meeting these efficiencies/targets will be a considerable challenge for mining companies.

2.2.2 Explosives and blasting

The GHGs emitted from blasting are primarily CO_2, CH_4 and N_2O (Sharma, 2008). The quantum and relative share of these emissions varies with the composition of the explosives. About 200 t of CO_2-e is produced from blasting required for production of one million tonnes of base metal ore. In case of coal, one million tonnes of coal produce about 400 t of CO_2-e from explosives detonation. It constitutes about 1 % of GHG emission from the entire mining activities (Orica, 2013) (Fig. 4).

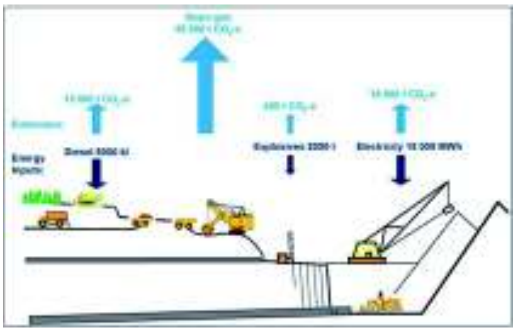


Figure 4. Coal mine energy and emissions per million tonnes of coal (ex-pit for a 7:1 stripping ration dragline operation) *(Source: Orica, 2013)*

2.2.3 Spontaneous combustion

CO_2 released from waste coal accumulated at the mine sites due to spontaneous combustion is another source of greenhouse emission both in underground as well as in surface mines (Wendt et al., 2000). Low-temperature oxidation as well as spontaneous combustion generates GHG. While low-temperature oxidation and spontaneous combustion arise from the same basic processes and represent different extremes of the same phenomenon, the greenhouse gas emissions from each are different. At ambient temperatures the major greenhouse gas emission from oxidation is CO_2. However, at higher temperatures present in material that are subjected to spontaneous combustion, parts of the coal can be starved of oxygen with the result that the chemical reactions change and appreciable quantities of CH_4 can be produced. This methane is produced as a result of the heating and it is addition to seam gas trapped in the coal.

2.2.4 Miscellaneous sources

Mining activities that may cause GHG emissions also include ground clearing, grading, trenching, and vehicular traffic. Activities conducted in locations other than the mine site include construction for transport systems that include access roads, rail lines and conveyor systems. Each of these process needs through observation for precise estimate of their GHG emission potential. Rehandling of pit head coal and overburden necessitate the use of energy by the machinery and therefore adds to the GHG emission.

Figure 5 shows the processes that leaves the carbon footprint from extraction of coal. The secondary carbon footprint includes the exhausts from vehicles used for transport of persons associated with mining. Use of air conditioners and other gadgets that produce greenhouse gas emissions are also contributing to secondary footprint.

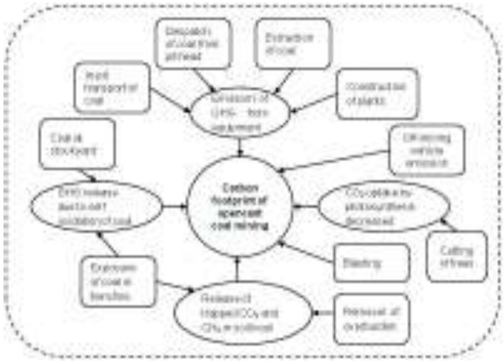


Figure 5. Components of opencast mining contributing on to carbon footprint of a mine

3 CONCLUSION

Regardless of where one stands politically on the subject of carbon regulation, a carbon constrained world is here, and most likely here to stay. Requirements to report the Greenhouse Gas (GHG) emissions are gradually coming into force. In an ever increasing "carbon" society the growth of an industry or a nation as a whole will be measured by its carbon footprint. The fundamental building-block for any business, strategic or management decision must be based on an understanding of one's footprint. Industries, especially mining, are under an ever increasing scrutiny of carbon, water and environmental mandates (Carpenter and Hairfield, 2010). Knowing the company's carbon emissions is the first step in creating any Inventory Reduction Plan (IRP).Making an inventory to determine how much carbon (e.g. GHG emissions) a company emits, is the first step in addressing any carbon management plan (Carpenter and Hairfield, 2010). And since Kyoto protocol, carbon credit came into existence and it economically now makes sense to reduce the carbon emission.

Although some earlier studies provide some indirect measurement of GHGs for some mining activities, comprehensive studies are needed to estimate GHG emissions from various activities of a mining operation in order calculate its carbon footprint. As of now no such study has been reported from any part of India. A detailed study will show GHG emissions from a mining activity in terms of fuel and non-fuel emissions, stationary and mobile source, point source and line/area source emission, emission on the basis of fuel category, emissions per person, emissions per tonne of output, emissions linked to rate of production, emission independent of rate of production, etc. This type of study will identify the carbon intensive activities in a surface/underground mine and therefore indicate the process that are to be given priority in order to reduce the carbon footprint from the mine.

The high-rank coal seams in deeper coalfields represent a significant target for future coal mine methane (CMM) and coalbed methane (CBM) development. In some of the Indian coalfields of the Damodar Valley, there are around 25 coal seams or even in excess of 40 in some areas, with cumulative thickness of more than 100 meters (M2M - India, 2005) which have the potential of high methane emission. Both surface mines and underground mines use high energy consuming equipment which leaves their carbon footprint. Emissions from stock yards and dumps are major source of carbon emission in opencast mines. It is proposed that in India carbon footprint of mine, especially of coal mines, should be developed so that hot-spots in terms of energy consumption and associated CO_2 emissions in mines is identified and emission factors and better generic model are developed to predict the GHG emission on the basis of mineral production/manpower involved/machinery used. By knowing their own carbon emissions and how they compare with other industries, the mining sector can formulate steps to reduce their carbon footprint.

REFERENCES

Beedie, M., 2008. A greener future for coal, http://www.mining-technology.com/features/feature1621/ (accessed on 24 June 2011).

Caldeira, K. and Wickett, M. E., 2003. Anthropogenic carbon and ocean pH, *Nature*, 425, 365 p.

Carbon Trust (2007). Carbon footprint measurement methodology, Version 1.1, 27 February 2007, The Carbon Trust, London, UK. http://www.carbontrust.co.uk (accessed on 29 June 2011).

Carpenter, S. M. and Hairfield, C. R., 2010. Fundamentals of carbon regulations and their impacts on the mining industry, *SME annual meeting and exhibit*, pp.403-406.

Cristobal, J., Albo, J. and Irabien, A., 2010. Environmental sustainability assessment of electricity from fossil fuel combustion: carbon footprint, *Low Carbon Economy*, 1, pp.86-89.

Dave, N. and Duffy, G., 2002. Technologies for mitigation of GHG emissions from coal industry. Technology assessment report 35, Cooperative Research Centre for Coal in Sustainable Development, QCAT Technology Transfer Centre, Pullenvale, Australia.

Day, S. J., Carass, J. N., Fry, R. and Williams, D. J., 2010. Greenhouse gas emission from Australian open-cut coal mines: contribution from spontaneous combustion and low-temperature oxidation, Environment Monitoring and Assessment, 166, pp.529-541.

Energetics, 2007. The reality of carbon neutrality, London, www.energetics.com.au/file?node_id=21228 (accessed on 29 June 2011).

EIA, 2009. Emissions of Greenhouse Gases Report, Energy Information Administration, http://www.eia.doe.gov/oiaf/1605/ggrpt/carbon.html (accessed on 25 June 2011).

EIA, 2010. International Energy Outlook 2009,US Energy Information Administration, http://www.eia.doe.gov/oiaf/ieo/emissions.html (accessed on 25 June 2011).

EPA, 2009. Global Anthropogenic Non-CO_2 Greenhouse Gas Emissions: 1990-2020, The United States Environmental Protection Agency. http://www.epa.gov/climatechange/economics/international.html. (accessed on 25 June 2011).

ETAP, 2007. Ecological and carbon footprints, Environmental Technologies Action Plan. http://ec.europa.eu/environment/etap/ (accessed on 29 June 2011).

Global footprint network, 2007. Global footprint network: advancing the science of sustainability, http://www.footprintnetwork.org/ (accessed on 29 June 2011).

Grubb and Ellis, 2007. Meeting the carbon challenge: the role of commercial real estate owners, users and managers, Chicago, USA.http://www.grubb-ellis.com/Data/Articles/CarbonWhitefnl.pdf (accessed on 29 June 2011).

Haven, J., 2007. *Environmental Business*, 129, 27 p.

Hammond, G., 2007. Time to give due weight to the 'carbon footprint' issue (Correspondence), *Nature,* 445, 256 p.

Irving, W and Tailakov, O., 2013. CH_4 Emissions: Coal Mining and Handling, Background paper on Good Practice Guidance and Uncertainty Management in National Greenhouse Gas Inventories,http://www.ipcc-nggip.iges.or.jp/public/gp/bgp/2_7_Coal_Mining_Handling.pdf (accessed on 02 January 2013).

Kelm, S., Luxbacher, K. and Karmis, M., 2010. Decrease carbon footprint through effective coal degasification, Prprint 10-090, SME annual meeting, Feb 28 – Mar 03, 2010, Pnoenix, USA.

Keuning, S. J., 1994. The SAM and beyond: open SESAME!, *Economic Systems Research,* 6, 1, pp.21-50.

Kirchgessner, D. A., Piccot, S. D. and Masemore, S. S., 1997. An improved inventory of methane emissions from coal mining in the United States, http://www.epa.gov/ttnchie1/ap42/ch14/related/mine.pdf (accessed on 02 January 2013)

Korre, A., Nie, Z. and Durucan, S., 2010. Life cycle modelling of fossil fuel power generation with post-combustion CO_2 capture, *International Journal of Greenhouse Gas Control*, 4, pp.289-300.

Le Quere, C., Raupach, M. R., Canadell, J. G., Marland, G., Bopp, L., Ciais, P., Conway, T. J., Doney, S. C., Feely, R., Foster, P., Friedlingstein, P., Gurney, K., Houghton, R. A., House, J. I., Huntingford, C., Levy, P. E., Lomas, M. R., Majkut, J., Metzl, N., Ometto, J. P., Peters, G. P., Prentice, I. C., Randerson, J. T., Running, S. W., Sarmiento, J. L., Schuster, U., Sitch, S., Takahashi, T., Viovy, N., van der Werf, G. R. and Woodward, F. I., 2009. Trends in the sources and sinks of carbon dioxide, *Nature Geosciences*, 2, pp.831-836.

M2M – India, 2005. Methane to Markets Partnership – CMM: India Profile, submitted to Methane to Markets International by the Government of India, 2005.

NOAA, 2012.Trends in atmospheric carbon dioxide, Earth System Research Laboratory, Global Monitoring Division, National Oceanic and Atmospheric Administration, http://www.esrl.noaa.gov/gmd/ccgg/trends/ (accessed on 31 December 2012).

Orica, 2013. Unearthing the Carbon Footprint, Technical Note, Orica's Blast Base Service. http://www.oricaminingservices.com/Section.aspx?SectionID=290&CultureID=3 (02 January 2013).

Patel, J., 2006. Green sky thinking, *Environmental Business*, 122, 32 p.

Piccot, S. D., Masemore, S. S., Ringler, E. S. and Kirchgessner, D.A., 1995. Developing Improved Methane Emission Estimates for Coal Mining Operations. In *Proceedings of the 1995 Symposium on Greenhouse Gas Emissions and Mitigation Research*; U.S. Environmental Protection Agency, Air Pollution Prevention and Control Division: Research Triangle Park, NC, 1995; EPA/600/R-96/072 (NTIS PB96-187752).

POST, 2006. Carbon footprint of electricity generation, POST note 268, October 2006, Parliamentary Office of Science and Technology, London, http://www.parliament.uk/documents/upload/postpn268.pdf (accessed on 22 June 2011).

Sabine, C. L., Feely, R. A., Gruber, N., Key, R. M., Lee, K., Bullister, J. L., Wanninkhof, R., Wong, C. S., Wallace, D. W. R., Tilbrook, B., Millero, F. J., Peng, T. H., Kozyr, A., Ono, T. and Rios, A. F., 2004. The oceanic sink for anthropogenic CO_2, *Science*, 305, pp.367-371.

Sharma, P. D., 2008. Carbon footprint reduction in mining and blasting operation – Control of greenhouse gas emission is essential to promote green environment, http://miningandblasting.wordpress.com/ (accessed on 24 June 2011).

Stahmer, C., 2000. The magic triangle of input-output, Paper presented at the *XIII International conference on input-output techniques*, 21-25 August, Macerata, Italy.

USEPA, 1994. International Anthropogenic Methane Emissions: Estimates for 1990, Office of Policy, Planning and Evaluation, Washington, DC.

USEPA, 2006. Global anthropogenic non-CO_2 greenhouse gas emissions: 1990–2020, EPA Report 430-R-06-003, United States Environmental Protection Agency, Office of Atmospheric Programs, Climate Change Division, June 2006, http://www.epa.gov/climatechange/economics/international.html (accessed on 24 June 2011).

Wendt, M. N., Mallett, C., Lapszewicz, J., Xue, S., Foulds, G., Mark, R., Sharma, S., Danell, R., Worrall, R. and Balusu, R., 2000. Methane Capture and Utilisation, Report No. C8058, Final Report, Australian Coal Association Research Programme (ACARP). http://www.acarp.com.au/abstracts.aspx?repId=C8058 (accessed on 24 June 2011).

Wiedmann, T. and Minx, J., (ed. Pertsove, C.C.), 2008. A definition of 'Carbon Footprint', *Ecological economics research trends*, Chapter 1, pp. 1-11. Nova Science Publishers, New York.https://www.novapublishers.com/catalog/product_infor.php?products_id=5999 (accessed on 22 May 2011).

On-Line Ash-Analyzers and Scales in Open Coal Mines. Coal Quality Management System

A. Artinyan
Institute for Nuclear Research and Nuclear Energy, Sofia, Bulgaria

L. Dimov
ANALiX Ltd., Sofia, Bulgaria

ABSTRACTA new generation of on-belt radioisotope ash-meters and scales was designed by Analix, Bulgaria. Accuracy, reliability, and stability are improved by a new high-tech electronics design and software.

A Coal Quality Management System (CQMS) based on seventeen on-belt ash-analyzers (and belt scales in same time) BCA-6 was built in the biggest Bulgarian coal open pits - Maritza East Mines. The on-belt instruments are mounted directly on the twelve coal excavators and at the end of the all five conveyor lines transporting coal to the three Thermal Power Plants working in the same area. The analyzers are connected in radio net and information is transferred on-line to the pit dispatchers and the staff in charge.

The benefits of using the CQMS come from meeting the requirements for a constant quality of the coal delivered to TPP and from making use of the high ash-content coal. Anticipated decrease of production costs for 1t of coal is 2-3%.

1 ASH ANALYZER BCA-6

1.1 Principles

Analix' Belt-Conveyor Ash-analyzers (BCA) use the well-known and common physical principle – attenuation of two gamma-energies (low and high), crosspassing the coal.

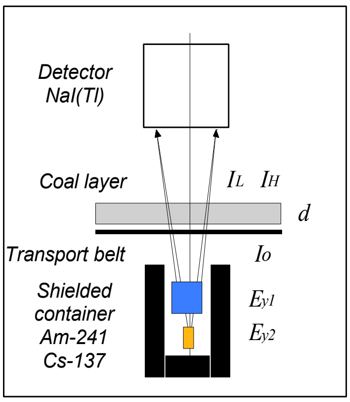


Figure 1. Principle diagram

The attenuation of the gamma-energy is calculating by the formula:

$$I = I_o . e^{-\mu(E\gamma, Z).\rho.d} \quad (1)$$

or:

$$\ln(I_o) - \ln(I) = \mu(E_\gamma, Z).\rho.d \quad (2)$$

where:
- I – intensity of the passed radiation, sec^{-1};
- Io – intensity at empty belt, sec^{-1};
- ρ – specific weight of the coal, g/cm3;
- d – thickness of the coal layer, cm;
- μ(Eγ,Z) – the factor characterizing the attenuation of a γ-radiation depending on its energy, cm^2/g.

When we have two different energies and divide their attenuations, we cancel the specific weight and the thickness:

$$R = \frac{\ln(I_{L_o}) - \ln(I_L)}{\ln(I_{H_o}) - \ln(I_H)} = \frac{\mu_L}{\mu_H} = f(Z_{eff}) \quad (3)$$

Z_{eff} is the "effective" atomic number of the coal, which is determined by the ratio between two components of the coal or lignite: organic ($Z_{eff} \cong 6$) and mineral ($Z_{eff} \cong 12$) – i.e. by the ash content.

But in theory as well as in the practice the things are not so simple as it could seems. There are a lot of factors, influencing this dependence:

- the variable chemical composition of the coal ash – we could obtain the same Z_{eff} when the ash content is different;
- the variable moisture and grain size could cause significant mistakes, if not accounted;
- climate changes;
- electrical and electro-magnetic interferences, etc.

What helped us to account all these factors and to measure coal ash-content directly at the Bucket Wheel and Chain Excavators? It was our high speed Gamma-spectrum Multichannel Analyzer that made possible on-line registration and developing of the full gamma-spectrum of the passed-trough radiation.

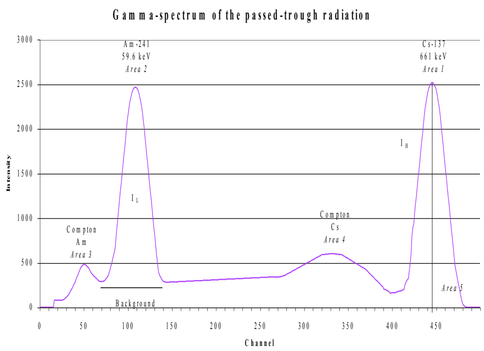


Figure 2. Gamma – Spectrum of the Ash Analyzer

We use five spectral areas for accounting the varying chemical composition of the coal mineral phase (so called “ash”) and the varying physical parameters of the coal. Trough an original algorithm and respective mathematical mode we achieve results with highest reliability regardless changes in the coal origin, chemical composition, humidity and size.

Our software for spectrum stabilization trough every second pick localization and respective HV correction was proved as the best by practice. Cs-137 (661 KeV) gamma spectral line was used for this spectrum stabilization 7 years ago in our first instrument applied in coal mines (BCA-05). Besides the good results during the first years of working in Maritza East Mines some problems occur, unknown in the routine spectral analysis in not so hard and so variable conditions.

In the generally applied way for gamma spectrum registration the intensities of the lines of Cs-137 and Am-241 are measured in fixed regions of interest. As rule this is a hardware adjustment.

We changed this routine approach and so significantly improved the sensitivity and the precision of the method.

Our on-belt ash-meters were the first instruments of this type using on-line full spectrum processing and respectively software determination of the line edges in the spectrum. The exact localization of these edges allows applying of the background subtraction and respective improving of the signal-noise ratio. But here is another problem following from the great variety of the spectrum depending on the quantity of the material over the belt. The spectrum line areas change up to 500 times from empty belt to fully loaded belt. Respectively the lines amplitude and wide changes all the time and the edges don’t stay at one place. Our software enables current fixing of the line edges and exact calculation of the background in every single second.

Together with changing the lines shape the great variety of the intensity during changing of the loaded coal causes a total change of the detector non-linearity. The dependence between the energy of the gamma-quanta and the height of scintillation detector outgoing pulses changes in different way for the low and for the high energy. That means the localization only of the high energy pick, which is enough in the routine spectral analyses is not enough for the ash-meters in the mines. So we develop an independent localization of the low-energy pick also. So we account the “breathing” of the spectrum in wide as well as in height.

These innovations introduced by us allow the Analix’ on-belt ash-analyzers to become the first successfully working directly at the excavators in the open pit coal mines. The large and very variable size of the coal, the hard meteorological conditions, the vibrations and the dust are not problem anymore due to our new design and software.

1.2 Construction and Installing of BCA-6

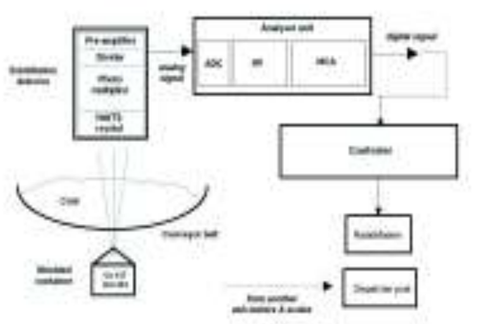


Figure 3. Block-Scheme of BCA-6

The Ash Analyzers are mounted directly on the Belt-Conveyors without changing the construction of the Conveyor.

Figure 4. On-belt Ash-analyzer on the outgoing belt conveyor

At the Bucket Wheel and Chain Excavators the on-belt weight and ash analyzers are installed on the discharge boom also without changing the belt construction.

The supporting structure of the on-belt ash analyzer is a rectangular frame vertically mounted and embracing the belt. On the top crossbeam is mounted the detector unit registering irradiation cross passing the coal.

Figure 5. On-belt Ash-analyzer on the discharge boom of a bucket chain excavator

Under the belt is mounted the shielded container with two sources of radiation. Due to a heavy lead protection covering sources no radiation could be obtained at any point around the instrument. There is only one narrow collimator letting a pencil type beam to go out vertically across the coal to the detector where this beam is totally absorbed. You can see the mechanics of this unit opening and closing the collimator. It could be made automatically, or by remote control, or locally during the service works.

In waterproof side mounted boxes are housed supplying units for low and high voltage, specialized electronics processing the detector signal, high speed spectral analyzer controlled by a local processor, and the computer. The computer controls the measurements, calculates the load and the ash content and supports the connection with the server.

The construction of the instrument is module-based and allows service and maintenance works in almost any conditions and usually without stopping the belt.

The computer of the instrument is installed in the control room.

In the operator cabin a terminal unit shows to the operator the current load and ash content as well as the respective values preset by the dispatcher. Also there are shown the position of the excavator, the disk thickness, the excavating regime, etc. A full history of the quantity and quality of the excavated coal is available. This on-line information helps much the operator to control the excavator work all the time and especially in the contact zones between the coal and overburden or between the coal and clay. Our instrument is user friendly so the excavator operators easy become familiar with the new features.

Trough a radiomodem on the top of the excavator the measurements are continuously transmitted to the server. From the data processed and stored there we have a full information and accounting about the excavated coal, overburden and clay, and respective ash content for all excavators.

1.3 Accuracy of BCA-6

The standard error for ash content:

- 0.5% for the range 5.0 ÷ 10.0%
- 1.0% for the range 10.0 ÷ 20.0%
- 1.5% for the range 20.0 ÷ 35.0%
- 2.0% for the range 35.0 ÷ 50.0%

For the measured weight the accuracy is about 2 % for quantities more than 500 t.

1.4 Radiation Safety of BCA-6

Our instruments are permanently checked and licenzed by the Bulgarian Agency for Nuclear Regulation.

All measurements done last 7 years by all kind of regulatory officials (Agency for Nuclear Regulation, Ministry of Health, Ministry of Internal Affairs, etc.) confirm the complete radiation safety of our radioisotope on-belt ash-analyzers.

The staff passing by, or working around as closed as possible, do not obtain any irradiation. Only on the way of the beam bellow the detector is a dangerous place when the collimator is open. But we can't be there when the belt is running. And when the belt is stopping, the collimator is closing. When collimator is open, the beam is 100% absorbed by the detector unit, so over it there is also no radiation.

There is no one case the radiation for people working there or passing by to be higher then the native radioactive background for that area.

2 COAL QUALITY MANAGEMENT SYSTEM (CQMS)

In 2005 in the biggest Bulgarian Lignite Open Pits - Maritza East Mines started building of a System for Coal Quality Management (CQMS).

2.1 The Goals of the System

The goals of the System were as follows:

- On-line Coal Quality Management and Control meeting both customer's requirements and own benefits;
- Optimization of the whole coal extraction, transportation and delivery process;
- Provision of reliable data for current state of mining excavators, diggers and transportation machinery;
- Administering of coal flow output;
- Central and local supervision and management of mining.

2.2 History of Building the System

In the beginning the on-line coal quality and quantity measurement was introduced only at the end of the all five belt conveyor transport systems in the three open pits of Maritza East Mines. In two of these pits coal is carried by trains to the TPP stations and in the third one the belt conveyor transports the coal directly to the TPP. The ash analyzers installed at these 5 points measure the belt load and the coal ash content, so we have on-line information about the coal quantity and quality in the loaded trains as well as at the belt loading coal in the TPP store.

Dispatcher has on-line information about the quality of outgoing coal so he can control it trough changing the output of each excavator to meet the desired value of the common coal quality. But if he had information for the ash content only at the end points of the 3-4 km long conveyor systems, than in case of big quality deviation and respective dispatcher decision to change the load of some excavators, the result would come with too big delay because there is a lot of coal already loaded on the conveyors.

It was the reason for us to design and built ash analyzers working directly at the Bucket Wheel and Chain Excavators. Nobody could do it before in any coal pit because of the serious fear that the extremely hard conditions as well as the large coal size right after excavating will obstruct the necessary accuracy and reliability. But we took the challenge and the practice of last 6 years showed that we successfully solved that seemed insoluble task. In 2006-2007 we successfully install on-belt ash-analyzers at all 12 coal excavators in Maritza East Mines.

Studding all the exploitation problems in next 2 years allow us to designing a next generation on-belt ash analyzers (BCA-06). Last two years all 17 on-belt ash-analyzers were renewed.

2.3 Structure and Functions of the Coal Quality Management System (CQMS)

2.3.1 Permanent on-line measurement of the coal quantity and ash content

The first and basic level of the SQMS is the on-line measurement of the coal quantity and quality at 17 points in the three open pits of Maritza East Mines. These points are all 12 coal excavators (4 in each pit), and the ends of the 5 conveyor transport lines carrying the coal out to the three TPP working in that region. The measurements are done at every second

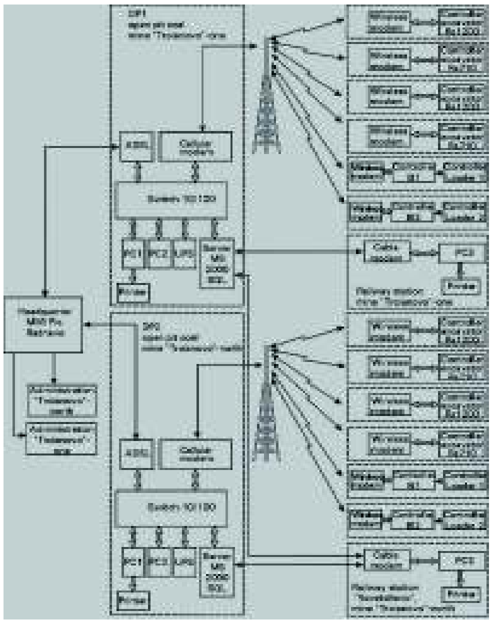


Figure 6. Block-scheme of the CQMS in Troyanovo-1 and Troyanovo-Nord Pits

Figure 7. On-belt Ash-analyzer on the discharge boom of a bucket wheel excavator

2.3.2 Transmitting data from the measurement points to the servers

Next level is the on-line transmitting of the measured data to the servers in the dispatcher posts (DP) of the three pits.

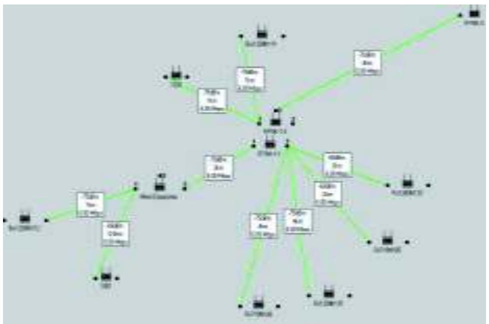


Figure 8. Block-scheme of the radio connections in one of the pits

All analyzers and the 3 servers are connected in a radio net which includes their own radio modems as well as some intermediate modems ensuring connections between the basic points in the hard for radio connection mine conditions.

2.3.3 Processing and storing collected information and using it by Pit Chief Dispatcher

All the information concerning coal excavation and transportation and the status of all excavators and conveyor belts for each of the three pits is processed and stored in the respective servers (third level of the System). These servers are installed in the Dispatcher posts of the three pits where trough their working stations the Pit Chief Dispatchers control all the excavation and transportation process and the coal quantity and quality. Besides the measurement data these servers store full data base for coal deposits including the general geological maps and all current data from geological samples. Special software helps the pit chief dispatcher to make the shift tasks for each excavator and generally for the pit. By means of the System during the shift he controls the realization of these tasks and could develop or change them.

At his display dispatcher can see all the time, or when necessary at several screens the next information:

- the current values of the coal quality and ash content at each working excavator and at the pit outgoing points;

Figure 9. First screen of the dispatcher's display

- location and status of the excavators, belt conveyor system and trains;
- current quantity and ash content of the coal loaded at every 50 meters of the each belt conveyor from the pit transportation systems;
- daily (shift) tasks for each excavator, for all the pit and for each client;

Figure 10. Assignment screen of the dispatcher's display

- detailed shift history of the excavating process as graphics or as table for each excavator, for each client and for all the pit;
- Geological map, graphically showing by color scale the vertical projection of technologically split mining layers to be excavated and location of the excavators.

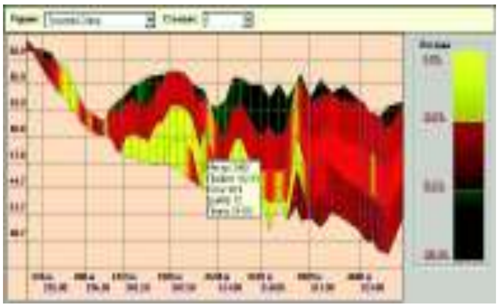


Figure11. Screen with the Geological map of the first mining layer in one of the pits

2.3.4 Delivery of information to the Mines Management

The highest level of the CQMS is the information delivery to the head office of Maritza East Mines. All kinds of reports could be taken of these servers trough the Maritza East Mines Internal Net. Depending on the level of access every specialist and manager can receive real time information concerning excavation process and coal deliveries to the TPP and reports for any period of time and by any criteria. So the System helps to the leading managers to make the best decisions for the current and for the general mines exploiting.

2.4 Benefits of Using Coal Quality Management System

2.4.1 Meeting customer's requirements

The on-line management of quality significantly improved the meeting customer's requirements. For example the biggest Bulgarian TPP "Maritza East-2" receiving coal from Maritza East Mines requires the mean coal ash content to be in 2% wide diapason and not more than 5% difference for a single shift. Before introducing CQMS the difference between the annual mean values of the ash content analyzed by coal lab in TPP "Maritza East-2" and given by Maritza East Mines certificates is 2.6%. This difference decreased to 1.2% after putting CQMS in operation. In same time the number of penalties because of exceeding 5% difference decreased from 90 to 10 annually.

2.4.2 Improving coal/overburden ratio

On-site blending of coal during the excavation trough the CQMS allows the usage of high-ash coal, which is up to 15% in some parts of Maritza East Mines. It brings additional annual income of millions euro to the Mines.

2.4.3 Optimization of the mining processes

The System provides reliable data for current state of mining excavators and transportation machinery which allows the staff in charge to have a full view over all the processes and a fast reaction when necessary. CQMS has become an indispensable device for the mining process management.

2.4.4 Supervision and management improvement

Benefits given by improving the control of processes, machinery and staff is difficult to be calculated. But they are easy visible - at the computer screens, during the production conferences, and in almost every information transfer - data, obtained by CQMS are in permanent use.

2.4.5 Decrease of production costs

The general estimation of the CQMS using in last years shows decrease of production costs for 1t of coal of 2-3%.

Anti-Explosive Powders For Coal Mining Industry

B. Buczek, E. Vogt
AGH, University of Science and Technology, Faculty of Energy and Fuels, Cracow, Poland

ABSTRACT New methods of hydrophobization of limestone surfaces must be searched. In the work two methods hydrophobization: from the stearic acid vapour phase and from silicone solutions are proposed. The first technique of limestone powder hydrophobization was carried out in an installation of own design (Vogt, 2008), and it consisted in free sedimentation of the powder layer dispersed by stearic acid vapour in powder counter-current flow. The second way of modification consisted in mixing in the evaporating dish substrates: limestone powder and dope - silicone solution - Sarsil® H-15. Evaluation of properties so-obtained waterproof powders was carried out according to the Polish Standard, as well as using original powder determination ways, with the Powder Characteristic Tester. Moreover water vapour adsorption isotherms were obtained and the thermal decomposition of powder was made.

1 INTRODUCTION

As a precaution against coal dust explosions, stone powder is spread within mine barriers. During an explosion the stone powder disperses, mixes with the coal dust and prevents flame propagation, acting as an inhibitor. Stone powder reduces the flame temperature to a point where devolatilization of the coal dust can no longer occur; starved of fuel, the explosion is inhibited. The amount of stone powder required to inert an explosion depends on: particle size of the stone material, particle size and type of the coal dust, as well as atmosphere composition (humidity, content of air and methane) present in underground coal mines.

Two types of stone powder are produced (regular and water-proof) which are used for sprinkling and for constructing dust barriers. A regular limestone powder is most commonly used for these purposes. Its major defect is its tendency to lose volatility, because of agglomeration under humid conditions, often reaching 100 % water saturation in mine atmospheres. Using the waterproof powder may eliminate it. Such powder has been produced by coating regular powder with stearic acid during grinding in stone mills (PN-G-11020, 1994), (Lebecki, 1993). In modernized quarries and plants, modern mills of a complex construction are employed, in which contamination with hydrophobizing agents is practically avoided. For this reason, new methods of modifying the character of limestone surfaces are searched for.

2 EXPERIMENTAL PROCEDURES

2.1 Materials and Manufacturing Methods

2.1.1 Materials

2.1.1.1 Limestone powder

In this work limestone powder (meal) from the Czatkowice Lime Quarry (Buczek at al., 2007) with the grain diameter less than 80 µm was used as a raw material during researches. This powder is, among others, used in the coal mining industry as one of the elements of the anti-explosive safety system (PN-G-11020, 1994), (Lebecki, 1993), (Skalski, 2005), (Man at al., 2009).

The average chemical composition of meal in accordance with the manufacturer's data is presented in Table 1. The real density of the meal, marked with the method of helium picnometry with the use of the AccuPyc 1330 apparatus, amounts to 2.7642 g/cm^3.

Table 1. Average chemical composition of limestone meal – manufacturer's data

Component	$CaCO_3$	SiO_2+NR	$MgCO_3$	Fe_2O_3	Al_2O_3	Na_2O	K_2O	Heavy metals
(% w/w)	96.00	1.50	1.50	0.11	0.08	0.023	0.037	In trace amounts

2.1.1.2 Modifiers

Industrial modifiers were chosen for research as they guarantee good contacting of powder grains with a hydrophobizing preparation. Stearic acid is used in mining production plants for the production of water-resistant powder preventing the coal dust explosion in mines (PN-G-11020, 1994).

Another dope used in the research include existing on the market representatives of groups of compounds applied for the hydrophobization processes of mineral materials; these are: a silicone preparation with the marketing name Sarsil® H-15 produced by the Chemical Plant "Polish Silicones" Ltd in Nowa Sarzyna. The solution of methyl salicylate resin in organic solvent - has got the density of 0.78 kg/m^3.

2.1.2 Manufacturing methods

Two methods of manufacturing of hydrophobic material are proposed: hydrophobization from stearic acid vapour and from silicone solutions.

The commercially used in mining industry hydrophobic limestone powder has been produced by coating regular powder with stearic acid during grinding in stone mills.

During these researches stearic acid vapour is contacted with limestone meal in a counter current flow in an installation of own design (Vogt, 2008).

The second method of producing of hydrophobic limestone powder consists in mixing raw powder with commercial silicone solution in the evaporating dish. The initial research determined the liquid dope volume that should be added to the powder in order to obtain optimal conditions for the contact of the preparation with a solid.

3 INVESTIGATION OF MATERIALS PROPERTIES

3.1 Evaluation of the Hydrophobization Degree of Analyzed Materials

Materials obtained in this way, may be used as an anti-explosive agents in mining industry. This waterproof product protects human life so its properties are very important and should be well known. One of the most important issues is the determination of the index of hydrophobization of samples. It is easy to determine it when stearic acid is used as a modifier, because there is a standard, which defines this measurement (PN-G-11020, 1994). The manufactured, above written way, sample contains 0.18 % of stearic acid, being an acceptable level according to the Polish Standard (PN-G-11020, 1994).

In the case of sample modified with the use of silicone solution authors had to work out the method for determination of hydrophobization C coefficient. The film flotation method (Diao at al., 1991) was used for this purpose when the commercial material produced by coating regular powder with stearic acid during grinding in stone mills was used as a comparative sample. The C coefficient defined to what extent the hydrophobic properties of the sample obtained with the use of silicone solution as a modifier are different from the hydrophobic properties of the commercial sample on contact with a suitable (10, 20, and 60 % (w/w)) methanol solution. The average value of the C = 84 % coefficient shows that the sample doped by silicone solution obtained sufficient hydrophobic properties.

3.2 Water Adsorption

The hydrophobicity of obtained materials has been also studied, by the determination of water vapour adsorption isotherms, using the liquid micro burettes (Fig. 1).

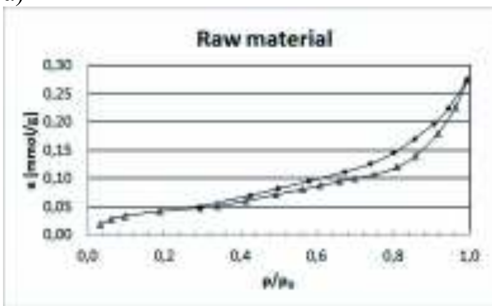
The greater water vapour adsorption for the hydrophobized powders (Fig. 1), especially for the sample modified with the use of stearic acid proves that products have achieved the water-resistance.

3.3 Method Measurement of Powder Properties

Obtained samples were analyzed with the use of the research methods originally applied in the powder technique due to the powder state of the material. It is interesting to know how the modification process influenced the change of typical powder properties, e.g. flowability or volatility. These properties are as well as hydrophobicity an important quality when we look at the possibility of using modified powders as an anti-explosive agent in mining industry. During researches Powder Characteristics Tester (PChT) – type PT-E, Ser. No. 90133 was used.

Carr (Carr, 1965) has tried to evaluate powder's flowability and floodability in a numerical manner with the combination of listed in Table 2 various physical characteristics. The tables for the conversion of the measured figures into a common index were published.

a)



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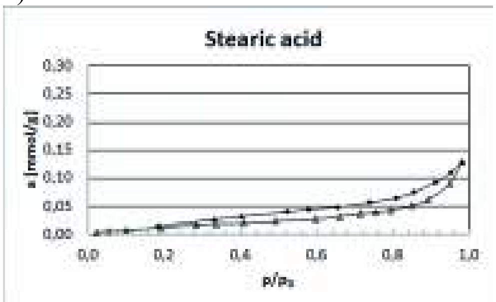

c)

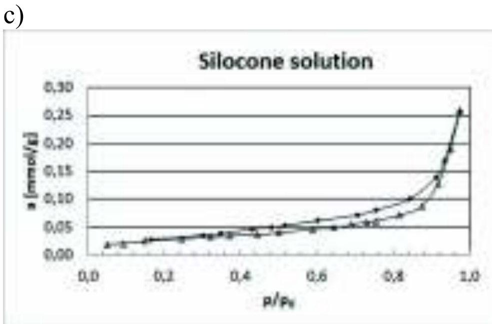

Figure 1.Water vapour adsorption isotherm for limestone powder a) raw, b) hydrophobized in stearic acid vapour, c) hydrophobized in silicone solution (Δ- sorption • - desorption curve).

Table 2.The characteristic of raw and hydrophobized limestone powders.

Characteristics	Material		
	Raw	Modified by stearic acid	Modified by silicone solution
Bulk density (kg/m^3)	724	798	790
Packed bulk density (kg/m^3)	1475	1377	1414
Repose angle (deg)	52	47	37
Fall angle (deg)	35	33	34
Difference angle (deg)	17	14	3
Dispersibility (%)	20	41	16
Carr's ratio (%)	50	42	44
Hausner's ratio	2.0	1.7	1.8

The parameters obtained with the use of PChT enable us to estimate the flowability and volatility of powders and give the possibility of determining the properties of limestone powder from the cohesion point of view. The obtained results may be used to assess the direction of changes of flow properties of limestone powders, which were caused by the hydrophobization process.

3.4 Thermal Decomposition

The thermal decomposition of the limestone powders hydrophobized by commercial modifiers was studied.

Generally the role of limestone powder in the system of protection against explosions brings to the increasing of content of non-combustible parts in coal dusts and physical prevention of the flame propagation. However, this role is much wider. Under the influence of the flames temperature comes to the thermal decomposition of limestone powder and both calcium oxide and carbon oxide (IV) are emitted. This endothermic process consuming some of the flame energy. The gases mixture is enriching in non-flammable CO_2. It causes the reduction of the system's explosion. So the limestone powder has a bigger efficacy than other anti-explosive agents. By implication, analysis of the thermal decomposition of modified limestone powder may be important when these one is characterized as the anti-explosive agent.

The thermo balance TA Instruments 2960 SDT was used during researches. DTA curves and the composition of obtained gasses (EGA) was measured. The results are showed at the Figure 3.The continuous line (Fig. 2a, 2b) represents the course of TG, DTG and DTA curves for the raw material. The dashed lines were obtained for limestone powder modified by stearic acid (Fig. 2a) and silicone solution (Fig. 2b).

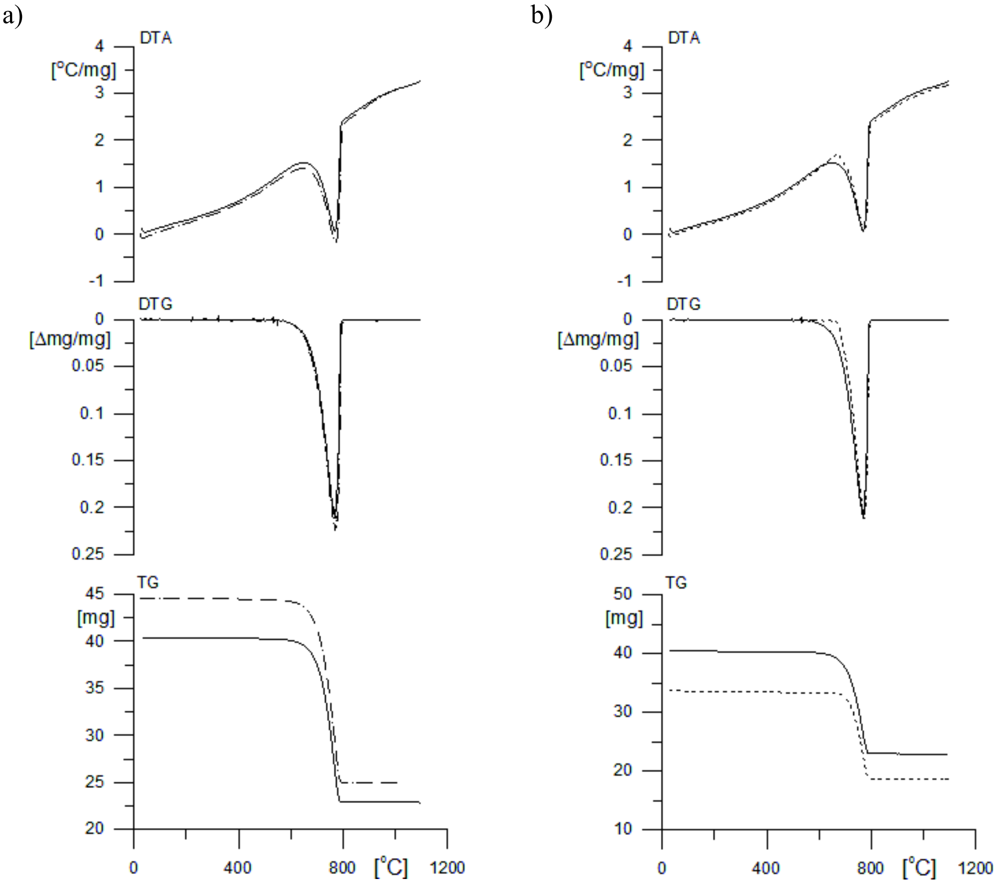

Figure 2. TG, DTG and DTA curves for limestone powder: raw – continuous line a) modified by stearic acid – dashed line, b) modified by silicone solution – dashed line.

The CO_2 contents in emitted gasses showed at the Figure 3 (EGA). The curves obtained for modified powders overlaps with this one obtained for the raw material.

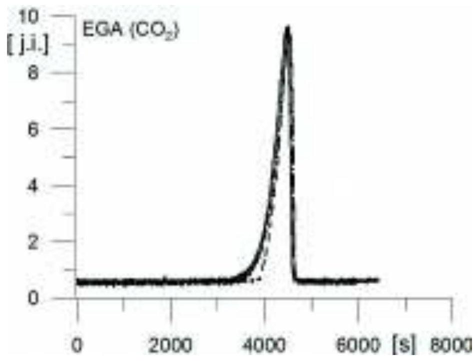

Figure 3. EGA curves for limestone powder: raw –continuous line a) modified by stearic acid – dashed line, b) modified by silicone solution – dashed line

The little differences between of courses of TG, DTG or DTA curves for all investigated materials was stated. The obtained results show that the character of the thermal decomposition of modified samples is the same as this one for raw powder, what is profitable for application of hydrophobized powders as an anti-explosive agent.

4. CONCLUSIONS

Both the sample modified by stearic acid and doped silicone solution acquired the hydrophobic character. The greater water vapour adsorption for the hydrophobized powders (Fig.1) especially for sample modified with the use of stearic acid proves that the product has achieved the water-resistance. Therefore we can state that the both proposed methods of hydrophobization of the limestone powder are useful.

It was interesting how the modification process influenced the change of typical limestone powder properties. The parameters obtained with the use of Powder Characteristics Tester enable us to make a characterization of limestone properties not only as a water resistant material bat also from the cohesion point of view.

On the base of TG, DTG or DTA and EGA curves for all investigated materials was stated that the character of the thermal decomposition of modified samples is the same as this one for raw powder, what is profitable for application of hydrophobized powders as an anti-explosive agent.

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REFERENCES

Buczek B., Vogt E., 2007, Twenty - Fourth Annual International Pittsburgh Coal Conference: Coal - Energy, Environment and Sustainable Development: September 10–14, Johannesburg, South Africa, PCC©2007. 51.

Carr R., 1965, Classifying Flow Properties of Solids, Chem. Eng. 72, pp. 163-168.

Diao I., Feuerstenau D.W. 1991, Characterization of the wettability of solid particles by film flotation. Part II Theorectical analysis. Colloids and Surfaces. 60, pp. 127-144.

Lebecki K. 1993, Neutralizacja osiadłego pyłu węglowego jako zabezpieczenie przed przenoszeniem wybuchu, Prace Naukowe Główn. Inst. Górn. 784, (In Polish).

Man C.K., Teacoach K.A., 2009, How does limestone rock dust prevent coal dust explosion in coal mines?, Mining Engineering, 61, 9, pp. 69-73.

Polish Standard PN-G-11020, 1994, Mining-Antiexplosive stone powder.

Skalski Z., 2005, Workowe zapory pyłowe dla podziemnych wyrobisk kopalń węgla kamiennego, Wiadomości Górnicze, 12, pp. 612-619 (In Polish).

Vogt E., 2008, Hydrophobization of fine solids presented on the example of limestone powder, Polish Journal of Chemical Technology, 10, 1, pp. 49-51.

Kömürün Yağ Aglomerasyonunda Ses Ötesi Dalgaların Emülsifikasyon için Kullanılması

Usage of Ultrasonic Waves in Oil Agglomeration of Coal for Emulsification

E. Şahinoğlu, T. Uslu
Karadeniz Teknik Üniversitesi, Maden Mühendisliği Bölümü,Trabzon

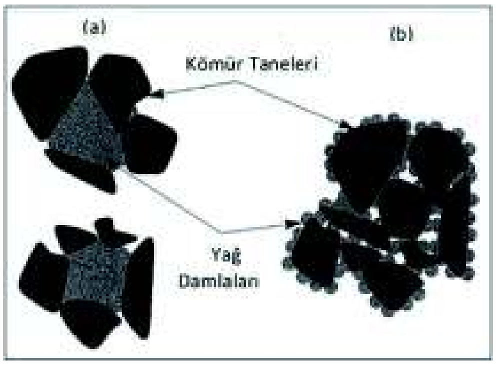
ÖZET Emülsifikasyon işlemi genelde mekanik karıştırma ile yapılan ve bir çok sektörde kullanımı olan bir işlemdir. Son yıllardaki bilimsel çalışmalar, ses ötesi dalgaların emülsifikasyon işleminde mekanik karıştırmaya göre daha başarılı olduğunu kanıtlamıştır. Emülsifikasyon, kömürün yağ aglomerasyonunda performansı etkileyen önemli aşamalardan biridir. Bu çalışmada, yüksek kül ve piritik kükürt içeriğine sahip toz kömür, ses ötesi dalgaların kullanıldığı emülsifikasyon işlemi sonrasında yağ aglomerasyonuna tabi tutularak aglomerasyon işleminin performansı belirlenmiştir. Ayrıca ses ötesi dalga işlemin gücü ve süresi değişken olarak incelenmiştir. Ses ötesi dalga bazlı emülsifikasyon işleminin kömürün yağ aglomerasyonu ile temizlenme performansını arttırdığı görülmüştür. Kül ve piritik kükürt uzaklaştırma oranlarının sırasıyla %53-57 ve %85-89 arasında değiştiği belirlenmiştir. Ses ötesi dalga işleminin süresi ve gücündeki artışın kül ve piritik kükürt uzaklaştırma oranlarını önemli düzeyde etkilemediği bulunmuştur.

ABSTRACT Emulsification is generally carried out by using mechanical stirring and has application in many sectors. Recent scientific researches have proved ultrasonic waves to be more successful than mechanical stirring in emulsification process. Emulsification is one of the most important stages affecting the performance of oil agglomeration of coal. In the present study, a fine coal with high ash and pyritic sulphur content was subjected to oil agglomeration process after emulsification with ultrasonic waves, and performance of the agglomeration process was determined. In addition, power and time of ultrasonic wave treatment were examined as variables. It was observed that ultrasonic wave-assisted emulsification increased the performance of coal cleaning by oil agglomeration. Ash and pyritic sulphur removals were determined to be 53-57% and 85-89%, respectively. It was found that increases in power and time of ultrasonic wave treatment had not considerable affects on ash and pyritic sulphur removals.

1 GİRİŞ

Aglomerasyonda kullanılan yağların, su içerisinde yeterince dağılamamaları ve oldukça yüksek viskozitelerinden dolayı kömür tanecikleri ile temasa geçmeleri zor olmaktadır (Kılınç, 2000). Ayrıca, yağın ortama kütle fazı şeklinde verilmesi durumunda oluşacak ara yüzey alanı küçük olmakta ve düşük verim elde edilmektedir. Bu nedenlerle, bu tür yağların çok küçük damlacıklar halinde dağıtılmaları gerekmektedir. Yağın su içinde dağıtılması (emülsifikasyonu) genelde mekanik karıştırma ile sağlanmaktadır. Ortamdaki damlacık sayısının artması, kömür taneleri ile yağ damlaları arasındaki çarpışmayı arttırmaktadır (Aksay vd., 2010).

Şekil 1'de emülsiye edilmemiş yağların ve emülsiye edilmiş yağların meydana getirdiği aglomeratlar görülmektedir. Emülsiye olmamış yağlar, sınırlı yüzey alanına sahip büyük damlalar şeklinde olup kömür yüzeylerini tam olarak kaplayamazlar (Şekil 1a). Emülsifikasyon sonucu toplam yüzey alanı fazla olan daha küçük boyutlu damlalar oluşarak daha iyi aglomerasyon sağlanır ve daha sıkı aglomeratlar oluşur (Şekil 1b), ayrıca yağ daha verimli kullanılmış olur (Kawatra ve Eisele, 2001).



Şekil 1. Emülsifikasyonun olmadığı (a) ve olduğu (b) durumda oluşan aglomeratlar (Kawatra ve Eisele, 2001)

Son yıllarda, yağ emülsifikasyonunda ses ötesi dalgalar kullanılmaya başlanmış ve bu konuda bir çok çalışma yapılmıştır (Abismail vd., 1999; Jafari vd., 2007; Cucheval ve Chow, 2008; Gaikwad ve Pandit, 2008; Sivakumar vd., 2008; Kentish vd., 2008; Djenouhat vd., 2008; Leong vd., 2009; Nii vd., 2009). Ses ötesi dalgalar ile yapılan emülsifikasyonda yağların daha küçük tanecikler halinde (Abismail vd., 1999; Sivakumar vd., 2008), ortamda homojen bir şekilde yayıldıkları (Sivakumar vd., 2008), daha duraylı oldukları (Abismail vd., 1999; Sivakumar vd., 2008) ortaya koyulmuştur. Bu çalışmalar genel nitelik taşımakta olup kömürün yağ aglomerasyonu öncesi yapılan emülsifikasyon ile ilişkili değildir. Kömür ve ses ötesi dalga ilişkili çalışmalar ise genellikle ses ötesi dalgaların kömürün yüzeyini değeştirmesinden hareketle yapılan flotasyon çalışmaları ve ses ötesi dalgaların kömürdeki kükürtü kolay uzaklaştırılabilen kükürt formlarına dönüştürmesi esasına dayalı kimyasal kömür temizleme işlemleridir. Kömürün yağ aglomerasyonunda ses ötesi dalga ile emülsifikasyon işlemine, 1980' lerde kısa süre uygulanan BHP (Broken Hill Proprietary) prosesi dışında literatürde rastlanmamıştır. Bu konuda yeni bilimsel çalışmalara ihtiyaç vardır. Bu çalışma, kömürün yağ aglomerasyonunda ses ötesi dalga esaslı emülsifikasyonu hakkında daha önce pek çalışma olamaması sebebiyle önemlidir ve yağ aglomerasyonu ile kömür temizleme konusuna önemli katkılar yapacaktır.

2 MALZEME VE METOT

2.1 Malzeme

Bu çalışmada, Müzret (Yusufeli-Artvin) havzasından getirilen kömür örneği kullanılmıştır. Kömür örneğinin kimyasal ve tane boyu analizleri sırasıyla Çizelge 1'de ve Çizelge 2'de verilmiştir. XRD analizinde, sülfür minerallerinden pirit, kil minerallerinden kaolinit, montmorillonit ve illit, karbonat minerallerinden kalsit, sülfat minerallerinden jips, silikat minerallerinden kuvars tespit edilmiştir.

Aglomerasyon işleminde kullanılan yağ, atık ayçiçeği yağı olup Karadeniz Teknik Üniversitesi İnşaat Mühendisliği Bölümü kantininden temin edilmiştir. Atık yağ fiziksel safsızlıkların giderilmesi amacıyla vakum filtre ile süzülmüştür. Atık ayçiçeği yağının yoğunluğu Alla France tipi hidrometre, viskozitesi Tanaka AKV-202 tipi viskometre ve yüzey gerilimi de CSC Dunouy tipi yüzey gerilim ölçer cihazı ile belirlenmiştir. Atık ayçiçeği yağının fiziksel özellikleri Çizelge 3'de gösterilmiştir. Ses ötesi dalga ile emülsifasyon işlemi için 750 watt gücünde ve 20 kHz frekansında ses ötesi cihaz (Cole-Parmer) kullanılmıştır (Şekil 2). Cihazın genlik değerine bağlı olarak ortama verdiği ses ötesi dalga gücü (şiddeti) Çizelge 4'de görülmektedir. Emülsiye edilmiş yağların görüntüleri Leica

DM4000-M tipi optik mikroskopda çekilmiştir. Aglomerasyon ürünlerinin parlak kesit incelemeleri Nikon Eclipse E400-POL mikroskopunda yapılmıştır.

Çizelge 1. Müzret kömürünün kimyasal analiz sonuçları

Bileşenler	Havada Kuru	Kuru
Nem (%)	2,25	-
Kül (%)	34,85	35,65
Uçucu Madde (%)	10,73	10,98
Sabit Karbon (%)	52,17	53,37
Sülfat Kükürt (%)	0,99	1,01
Piritik Kükürt (%)	5,44	5,57
Organik Kükürt (%)	1,3	1,33
Toplam Kükürt (%)	7,73	7,91
Üst Isıl Değer (kcal/kg)	4970	5084

Çizelge 2. Kömür tane boyutu analizi

Tane boyutu (mm)	Ağırlık(%)
-0,5+0,3	22,16
-0,3+0,212	17,54
-0,212+0,15	14,11
-0,15+0,106	10,10
-0,106+0,053	15,93
-0,053	20,16
Toplam	100

Çizelge 3. Atık ayçiçeği yağının fiziksel özellikleri

Yağ Türü	Yoğunluk (g/cm^3)	Viskozite (mm^2/s)	Yüzey Gerilimi (dyn/cm)
Atık Ayçiçeği Yağı	0,91	35,81	34,6

Çizelge 4. Ses ötesi cihazın genliğe (%) bağlı olarak hesaplanmış güç ($watt/cm^2$) değerleri (Yazıcı, 2005)

Genlik(%)	Güç ($watt/cm^2$)
20	9,5
40	28,5

2.2 Metot

Aglomerasyon deneyleri öncesi, büyük çoğunluğu toz boyuta sahip kömür örneği kontrollü olarak 0,5 mm'nin altına indirilmiştir. Deneyler -0,5 mm boyutundaki kömür örnekleri kullanılarak gerçekleştirilmiştir. Aglomerasyon deneyleri için, iç çapı 11,7 cm olan silindirik bir cam kap kullanılmış olup içerisine genişlikleri 1,1 cm olan 4 adet plastik levha (buffle) koyulmuştur. Deneyler, RZR 2021 tipi hızı ayarlanabilen bir mekanik karıştırıcı ile saf su kullanılarak yapılmıştır. Karıştırıcının pervane çapı 50 mm olup pervaneler yatayla 45° açı yapmaktadır. Karıştırma işlemi kap tabanından 8 mm yükseklikte yapılmıştır. İlk olarak emülsifikasyon yapılmaksızın aglomerasyon deneyi yapılmıştır. Kömür-su karışımı, 1000 dev/dak. karıştırma hızında 5 dakika şartlandırılmıştır. Şartlandırılmış kömür-su karışımına yağ ilave edilip karıştırma işlemi yapılmış ve kömürlerin aglomera olmaları sağlanmıştır. Daha sonra, pülp 0,5 mm boyutlu eleğe beslenerek aglomeratlar elek üstü olarak kazanılmıştır. Aglomeratlar, yüzeylerine yapışan mineral maddelerin uzaklaşması için yıkandıktan sonra, elekten uzaklaştırılmış, vakum filtre ile susuzlandırılmış, 200 ml aseton ile yıkanarak yağdan arındırılmıştır. Susuzlandırılmış ve yağından arındırılmış aglomeratlar (temiz kömür) $105^{\circ}C \pm 5^{\circ}C$'de kurutulmuş ve tartılmıştır. Deney parametreleri ve değişkenleri şu şekildedir: Katı oranı: %10, atık yağ oranı: %10, karıştırma hızı: 1400 dev/dak., aglomerasyon süresi: 10 dak., yıkama suyu miktarı: 1,5 lt, kömür tane boyutu: -0,5 mm, aglomerat kazanım eleği boyutu: 0,5 mm. Katı oranı, kömür ağırlığı / kömür+su ağırlığı'nı, atık yağ oranı ise atık yağ ağırlığı / kömür ağırlığı'nı ifade etmektedir. Daha sonra, ses ötesi dalgalar ile yağın emülsiye edildiği aglomerasyon deneylerine geçilmiştir. Önce, kömür-su karışımı 1000 dev/dak. hızda 5 dak. şartlandırılmıştır. Yağ, su içerisinde ses ötesi dalgalarla farklı güçlerde (9,5; 28,5 watt/cm^2) ve farklı sürelerde (0,5; 1; 1,5; 2 dak.) emülsiye edilerek kömür-su karışımına eklenmiş ve deneyler gerçekleştirilmiştir. Tüm aglomerasyon deneyleri, emülsifikasyon olmaksızın yapılan deneydeki sabit koşullarda yapılmıştır. Aşağıdaki eşitlikler kullanılarak, yanabilir verim (YV %), kül uzaklaştırma oranı (KUO %) ve piritik kükürt uzaklaştırma oranı (PKUO %) hesaplanmıştır.

$$YV\ (\%) = (W_{Ü}/W_{B}) \times 100 \quad (1)$$

$$KUO\ (\%) = [(A_{B}-A_{Ü})/A_{B}] \times 100 \quad (2)$$

$$PKUO\ (\%) = [(PK_{B}-PK_{Ü})/PK_{B}] \times 100 \quad (3)$$

Burada;

$W_{Ü}$: Aglomerasyon ürününün, kuru-külsüz-yağsız bazda ağırlığı (gr)

W_{B}: Aglomerasyon beslemesinin, kuru-külsüz bazda ağırlığı (gr)

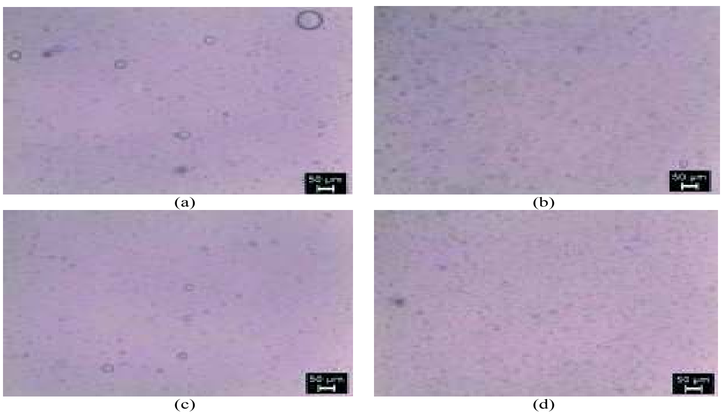
A_{B}: Kuru aglomerasyon beslemesinin kül oranı (%)

$A_{Ü}$: Kuru-yağsız aglomerasyon ürününün kül oranı (%)

PK_{B}: Kuru aglomerasyon beslemesinin piritik kükürt oranı (%)

$PK_{Ü}$: Kuru-yağsız aglomerasyon ürününün piritik kükürt oranı (%)

Şekil 2. Atık yağın ses ötesi dalgalar ile emülsiye edilmesi

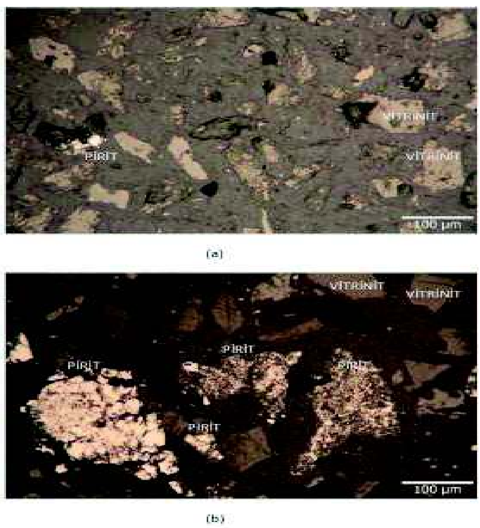


Şekil 3. Farklı güç ve sürelerde emülsiye edilen yağın mikroskop görüntüleri (a: 9,5 watt/cm^2 ; 30 sn; b: 9,5 watt/cm^2 : 2 dak.; c: 28,5 watt/cm^2 : 30 sn; d: 28,5 watt/cm^2 : 2 dak.)

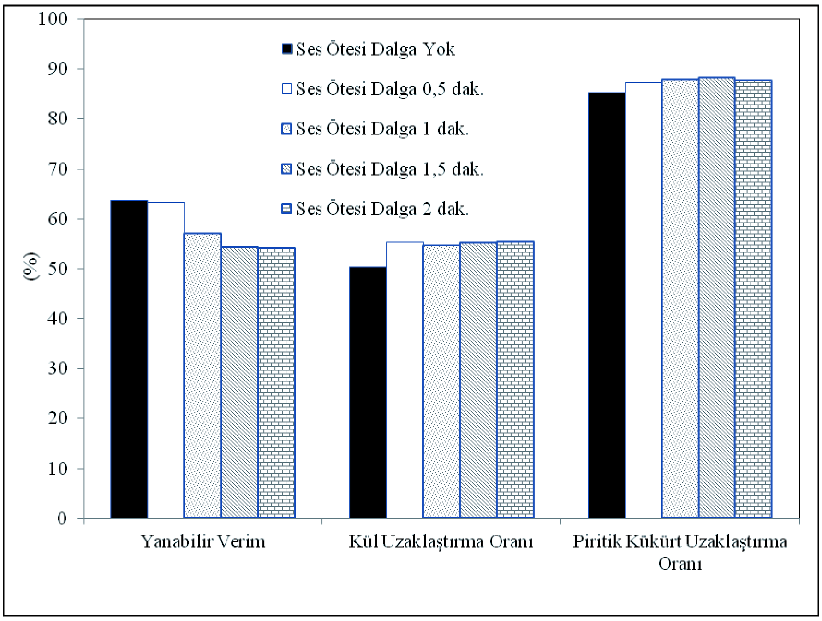
3 BULGULAR VE TARTIŞMA

Şekil 2'de ses ötesi dalgalarla emülsiye edilmiş yağın suyun içerisinde dağıldığı ve su-yağ karışımının renginin değiştiği görülmektedir. Farklı güç ve sürelerde emülsiye edilen yağın mikroskop görüntüleri Şekil 3'de görülmektedir. Şekilde görüleceği üzere, güç ve süre arttıkça yağların tane boyutu azalmaktadır.

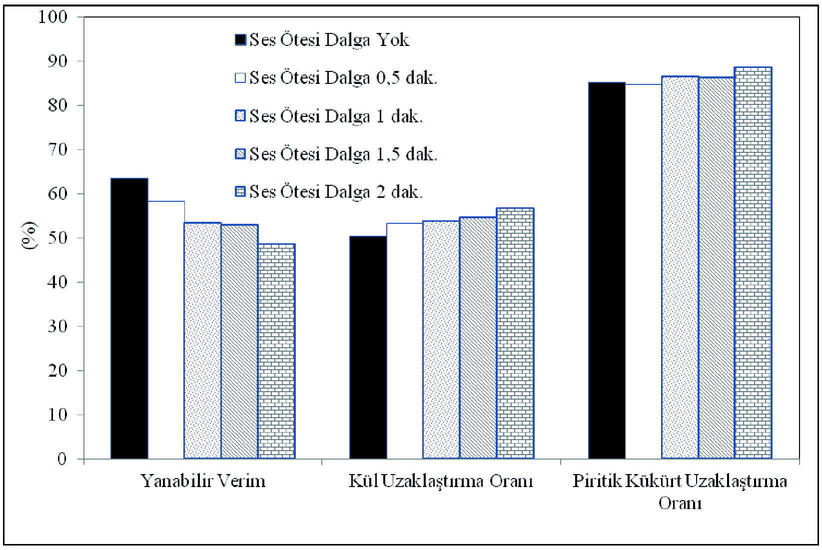
Aglomerasyon konsantresi ve atığının parlak kesit görüntülerinden (Şekil 4), kömürün, emülsifikasyon aşamasında ses ötesi dalganın kullanıldığı bir aglomerasyon işlemiyle başarılı bir şekilde temizlendiği açıkca görülebilir. Ses ötesi dalga gücü 9,5 watt/cm^2 olduğunda elde edilen aglomerasyon sonuçları Şekil 5'de görülmektedir. Yanabilir verim %54-63, kül ve piritik kükürt uzaklaştırma oranları ise sırasıyla %55-56 ve %87-88 arasında değişmektedir. Ses ötesi dalga gücü 28,5 watt/cm^2 olduğunda elde edilen aglomerasyon sonuçları ise Şekil 6'de görülmektedir. Yanabilir verim %49-58, kül ve piritik kükürt uzaklaştırma oranları ise sırasıyla %53-57 ve %85-89 arasında değişmektedir. Ses ötesi dalga gücü artınca kül ve pritik kükürt uzaklaştırma oranları fazla değişmemiştir. Ses ötesi dalgaların yağ emülsifikasyonu için kullanıldığı aglomerasyonda elde edilen sonuçlar emülsifikasyonun yapılmadığı aglomerasyon sonuçlarıyla karşılaştırıldığında, yanabilir verimin emülsifikasyonun yapılmadığı yağ aglomerasyonunda daha fazla olduğu görülmüştür. Ses ötesi dalgalar ile yağın emülsiye edildiği aglomerasyon deneylerinde güç ve süre artışına bağlı olarak yanabilir verimin biraz azaldığı görülmektedir. Bunun sebebi, güç ve süre artışıyla birlikte yağların tane boyutunun çok küçülmesidir. Dolayısıyla yağların tane boyutunun çok fazla küçülmesi fraksiyondaki iri boyutlu kömür tanelerini yüksek verimle aglomera edememektedir. Yağlar emülsiye edilince, kömür-yağ teması daha iyi oluşsa da kömür boyutu büyük olduğundan, kömürün ağırlığından kaynaklanan yerçekimi kuvveti, kömür-yağ arasındaki bağ kuvvetini yenerek fraksiyondaki iri boyutlu kömürlerin aglomeratlardan ayrılmasına yol açmaktadır. Emülsiye edilmiş yağlarla yapılan deneylerde daha fazla kül ve piritik kükürt uzaklaştırma oranları elde edilmiştir. Kül ve piritik kükürt uzaklaştırma oranlarının artması, kül yapıcı mineral maddelerin (kil mineralleri, pirit vb.), çok küçük boyuttaki yağ damlacıklarının oluşturduğu aglomeratlar arasına girememesinden kaynaklanmaktadır. Çünkü küçük yağ damlalarının oluşturduğu aglomeratlar içindeki boşluklar daha az olmaktadır. Ses ötesi dalga gücü 9,5 watt/cm^2 olduğunda sürenin artışıyla birlikte kül ve piritik kükürt uzaklaştırma oranlarının çok fazla değişmediği görülmüştür. Ses ötesi dalga gücü 28,5 watt/cm^2 olduğunda sürenin artışıyla birlikte kül ve piritik kükürt uzaklaştırma oranlarının az miktarda arttığı bulunmuştur.



Şekil 4. Aglomerasyon konsantresi (a) ve atığının (b) parlak kesit görüntüleri.



Şekil 5. Ses ötesi dalga ile emülsifikasyon işlemi (Güç: 9,5 watt/cm^2) sonrası yapılan aglomerasyon işlemi sonuçları



Şekil 6. Ses ötesi dalga ile emülsifikasyon işlemi (Güç: 28,5 watt/cm^2) sonrası yapılan aglomerasyon işlemi sonuçları

4 SONUÇLAR VE ÖNERİLER

Ses ötesi dalgalarla bağlayıcı yağ çok başarılı bir şekilde emülsiye edilmekte, çok küçük boyutlu yağ damlacıkları oluşmaktadır. Ses ötesi dalgalar kullanılarak yapılan emülsifikasyon işlemi sonrasındaki aglomerasyon deneylerinde, kömürden uzaklaştırılan kül ve piritik kükürt miktarlarında artış sağlanmıştır. Bununla birlikte, kömür ve yağ damlası arasındaki temas yüzey alanı artmasına rağmen, yanabilir verimde bir miktar düşme görülmüştür. Bunun sebebi, kömür tane boyutunun (-0,5 mm), oluşan küçük boyutlu yağ damlacıkları tarafından sağlam aglomeratların oluşturulabilmesi için biraz büyük kalmasıdır. Kömür daha düşük tane boyutlarına indirildikten sonra yapılacak deneylerde daha duraylı ve sağlam aglomeratların oluşacak olması, ayrıca serbestleşmenin de artacak olması sebebiyle verimdeki düşme eğilimi muhtemelen tersine dönecektir. Deneylerde yağ oranı %10 olarak sabit tutulmuştur. Yağ oranının arttırılmasının da aynı pozitif etkiyi yapacağı düşünülmektedir. Şöyle ki, Şahinoğlu (2012) ses ötesi dalgaların kullanılmadığı aglomerasyon testlerinde yanabilir verimin %20 yağ oranına kadar artarak %75'lere ulaştığını rapor etmiştir.

KAYNAKLAR

Abismail, B, Canselier, J.P, Wilhelm, A.M, Delmas, H. ve Gourdon, C, 1999. Emulsification by Ultrasound: Drop Size Distribution and Stability, Ultrasonics Sonochemistry, 6, (1-2), (pp.75-83).

Aksay, E. K, Arslan, V. ve Polat, H, 2010. Toz Kömürlerin Zenginleştirilmesinde Yağ Aglomerasyonu Yöntemi ve Yenilikler, İstanbul Yerbilimleri Dergisi, 23, (2), (s.97-108).

Cucheval, A. ve Chow, R.C.Y, 2008. A Study on the Emulsification of Oil by Power Ultrasound, Ultrasonics Sonochemistry, 15, (5), (pp.916-920).

Djenouhat, M, Hamdaoui, O, Chiha, M. ve Samar, M.H, 2008. Ultrasonication-Assisted Preparation of Water-in-oil Emulsions and Application to the Removal of Cationic Dyes from Water by Emulsion Liquid Membrane Part 2. Permeation and Stripping, Separation and Purification Technology, 63, (1), (pp.231-238).

Gaikwad, S.G. ve Pandit, A.B, 2008. Ultrasound Emulsification: Effect of Ultrasonic and Physicochemical Properties on Dispersed Phase Volume and Droplet Size, Ultrasonics Sonochemistry, 15, (4), (pp.554-563).

Jafari, S.M, He, Y. ve Bhandari, B, 2007. Production of Sub-micron Emulsions by Ultrasound and Microfluidization Techniques, Journal of Food Engineering, 82, (4), (pp.478-488).

Kawatra, S.K, ve Eisele, T.C, 2001. Coal Desulfurization High-Efficiency Preparation Methods, Department of Mining and Materials Processing Engineering Michigan Technological University.

Kentish, S, Wooster, T.J, Ashokkumar, M, Balachandran, S, Mawson, R. ve Simons, L, 2008. The Use of Ultrasonics for Nanoemulsion Preparation, Innovative Food Science and Emerging Technologies, 9, (2), (pp.170-175).

Kılınç, E., 2000. Toz Kömürlerin Yağ Aglomerasyonu ile Zenginleştirilmesi, Yüksek Lisans Tezi, Dokuz Eylül Üniversitesi, Fen Bilimleri Enstitüsü, İzmir.

Leong, T.S.H, Wooster, T.J, Kentish, S.E. ve Ashokkumar, M, 2009. Minimising Oil Droplet Size Using Ultrasonic Emulsification, Ultrasonics Sonochemistry, 16, (6), (pp.721-727).

Nii, S, Kikumoto, S. ve Tokuyama, H, 2009. Quantitative Approach to Ultrasonic Emulsion Separation, Ultrasonics Sonochemistry, 16, (1), (pp.145-149).

Sivakumar, V, Prakash, R.P, Rao, P.G, Ramabrahman, B.V, ve Swaminathan G, 2008. Power Ultrasound in Fatliquor Preparation Based on Vegetable Oil for Leather Application, Journal of Cleaner Prodution, 16, (4), (pp.549-553).

Şahinoglu, 2012. Kömürün Yağ Aglomerasyonu Yöntemi ile Temizlenmesinde Bitkisel Atık Yağların ve Ses Ötesi Dalgaların Kullanılabilirliği, Doktora Tezi, Karadeniz Teknik Üniversitesi, Fen Bilimleri Enstitüsü, Trabzon.

Yazıcı, E.Y, 2005. Atık Sulardaki Siyanürün Hidrojen Peroksit, Aktif Karbon Adsorpsiyonu ve Ses Ötesi Dalgalarla Uzaklaştırılması, Yüksek Lisans Tezi, Karadeniz Teknik Üniversitesi, Fen Bilimleri Enstitüsü, Trabzon.

Zonguldak – Türkiye Taşkömürlerinin Tutuşma Sıcaklıkları ile ilgili Özgün Kriterlerin ve Kendiliğinden Yanmaya Yatkınlıklarının Belirlenmesi

Determination of Specific Criteria about the Ignition Point Temperatures and Liability to Spontaneous Combustion of Zonguldak-Turkish Hardcoal Seams

F. Baş
Kimya Mühendisi, İş Güvenliği Birimi, Ar-Ge Laboratuarı, Türkiye Taşkömürü Kurumu, Zonguldak
İ. Erol
Bülent Ecevit Üniversitesi, Maden Mühendisliği Bölümü, Zonguldak
V. Didari
Bülent Ecevit Üniversitesi, Maden Mühendisliği Bölüm, Zonguldak
M. Öztürk
İş Güvenliği ve Eğitimi Daire Başkanı, Türkiye Taşkömürü Kurumu, Zonguldak
B. Salih
Hacettepe Üniversitesi, Kimya Bölümü, Ankara

ÖZET Zonguldak Kömür Havzası'nda bulunan kömür damarlarının pek çoğu kendiliğinden yanmaya oldukça yatkındır. Ocak yangınlarına karşı daha etkin mücadele için kömür damarlarının tutuşma sıcaklıklarının belirlenmesi ve kendiliğinden yanmaya yatkınlıklarının tespit edilmesi büyük öneme sahiptir. Bu nedenle Termogravimetrik Analiz (TGA) sistemi ve kesişme yöntemi ile taşkömürlerinin tutuşma sıcaklıkları tespit edilmiş, TG-DTG eğrilerinden yanma karakteristikleri belirlenmiştir. Kömürlerin bünyesel özellikleri ile kendiliğinden yanma deney sonuçları arasındaki ilişkiler incelenmiştir. Bu amaçla, Türkiye Taşkömürü Kurumu Kozlu (Çay, Hacımemiş, Kurul, Sulu), Üzülmez (Kurul, Piriç), Amasra (Kalın, Taşlı, Tavan) ve Karadon (Sulu, Hacımemiş) Müesseseleri damarlarından alınan örnekler üzerinde kömürlerin kendiliğinden yanmaya yatkınlıkları ayrıntılı bir şekilde araştırılmıştır.

ABSTRACT Most of coal seams are liable to spontaneous combustion in Zonguldak Hard Coal Basin. For more effective measures against fires in the mines, determination of spontaneous combustion characteristics and liability of coal seams to spontaneous combustion is very important. Therefore, ignition temperatures of hard coals have been determined by TGA-MS system and crossing point technique together with the related specific criteria. Also characteristics of combustion are calculated from TG-DTG thermograms. Also relationship between ignition temperatures and coal constituents are investigated. Tests are carried on the coal samples taken from Kozlu (Çay, Hacımemiş, Kurul, Sulu), Üzülmez (Kurul, Piriç, Acun, Taban Acılık), Amasra (Kalın, Taşlı, Tavan) and Karadon (Sulu, Hacımemiş) Collieries of Turkish Hardcoal Enterprises.

1 GİRİŞ

Kömür, havanın oksijeni ile doğrudan doğruya yanabilen % 55-95 arasında değişen oranlarda serbest veya bileşim halinde karbon içeren, yandığında değişik miktarda ve bileşimde kül bırakan organik kökenli tortul bir kayaçtır (Kaymakçı ve Didari, 2000).

Kömür oksitlenmeye eğilimli bir madde olup, yeni açılan yüzeyler hava ile temas eder etmez oksidasyon işlemi başlar. Kömürün kendiliğinden yanmasında etkili olan, kömür yüzeyinin oksijenle olan ilişkisidir. Oksijen molekülleri kömürün yüzeyine fiziksel olarak bağlanır (adsorpsiyon) ve difüzyon yoluyla mikro gözeneklere kadar ulaşarak kömürle oksijen arasında ekzotermik bir kimyasal reaksiyon oluşmasına neden olur (Ökten, 1988).

Ocakta, normal koşullar altında, dışa verilen ısı enerjisi alınmakta ve oksidasyon yavaş bir biçimde ve kızma tehlikesi doğurmaksızın sürmektedir. Ancak bazı durumlarda dışa verilen ısı enerjisi, ortamdan ayrılmamakta ve sıcaklık giderek artmaktadır. Sıcaklık arttıkça, ortamda yeterli oksijen varsa oksidasyon hızı artmakta ve buna bağlı olarak kömürün sıcaklığı yükselmektedir. Kömürün tutuşma sıcaklığına (kritik sıcaklık) ulaşıldığında ise yanma olayı baş göstermektedir (Didari, 1986).

Bu çalışmada Türkiye Taşkömürü Kurumu Kozlu (Çay, Hacımemiş, Kurul, Sulu), Üzülmez (Kurul, Piriç), Amasra (Kalın, Taşlı, Tavan) ve Karadon (Sulu, Hacımemiş) Müesseseleri damarlarından alınan örneklerin tutuşma sıcaklıkları Termogravimetrik Analiz (TGA) cihazı ve kendiliğinden yanma deney setinde belirlenmiştir.

Kömürün oksidasyonu esnasında karbon yanması ve dolayısı ile kömürün ısıl değerinde azalma ekonomik kayıplara neden olmaktadır. Yeraltında bu olaya bağlı olarak çıkan yangınlar, oldukça önemli insan ve ulusal servet kayıplarına yol açabilen kazaların kaynağı olabilmektedir (Didari, 1986). Bu nedenle kömürlerin kendiliğinden yanmaya yatkınlıklarının bilinmesi önem arz etmektedir.

Kendiliğinden yanma ile etkin mücadele edebilmek için, kendiliğinden yanmaya yatkın damarların belirli bir sınıflandırmaya tabi tutularak önlemlerin alınması oldukça önemlidir. Yanma yatkınlık sınıflandırılması yapabilmek için çeşitli indeksler geliştirilmiştir. Bu çalışmada damarların yanma riski sınıflandırılması Feng et al., (1973) tarafından geliştirilen indeks kullanılarak yapılmıştır.

2 LABORATUAR ÇALIŞMALARI

Çalışmalar TTK İş Güvenliği ve Eğitimi Daire Başkanlığı Ar-Ge Laboratuarında ve Bülent Ecevit Üniversitesi Mühendislik Fakültesi Maden Mühendisliği Bölümü'ndeki laboratuarlarda kurulu düzeneklerde gerçekleştirilmiştir.

Yeraltı kömür damarlarından alınan örneklerin tutuşma sıcaklıkları Termogravimetrik Analiz (TGA) sistemi ve kendiliğinden yanma deney setinde kesişme noktası sıcaklığı tekniği ile belirlenmiştir.

2.1 TGA Tekniği ile Yapılan Çalışmalar

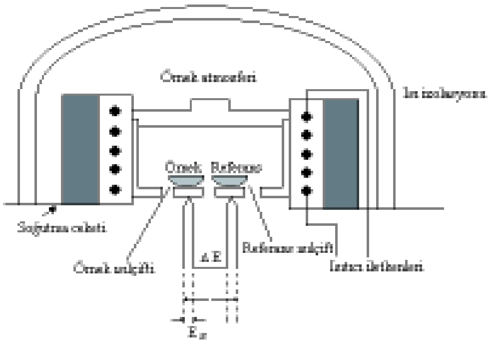
2.1.1 Termogravimetrik Yöntem

Maddenin bileşimi konusunda en yaygın olarak kullanılan yöntemler, TG (termogravimetri), DTA (diferansiyel termal analiz) ve DTG (diferansiyel termo gravimetrik analiz)'dir. TG eğrilerinde genellikle üç bölge vardır; ağırlık artış bölgesi, ağırlık azalma bölgesi ve ağırlığın sabit kaldığı yatay bölge. Termoanalitik uygulama açısından, özellikle de bir bileşiğin kararlılığı kontrol edilirken, TG eğrilerindeki yatay bölgeler en önemli kısımlardır.

DTA yöntemi pratikte, fırın sıcaklığı T ile analiz edilmesi istenen maddenin sıcaklığı arasındaki farkın (ΔT) kaydedilmesi şeklinde uygulanmaktadır. Örnek ile referans maddenin sıcaklıkları arasındaki fark en doğru şekilde, biri örnek diğeri ise referans madde içine daldırılmış bir türevsel ısıl çift yardımı ile kaydedilebilmektedir (Şek. 1).

Şekil 1. TGA/DTA cihazı

DTA yöntemi ile incelenebilenler endotermik reaksiyonlar, dehidratasyon, yapısal bozunma, erime, buharlaşma, süblimleşme ve ısı etkisi ile olan yapısal dönüşmelerdir. Ekzotermik reaksiyonlar ise oksidasyon (yanma), dondurma işlemleri, kristal yapının yeniden düzenlenmesi ve soğuma sırasında olan yapısal dönüşümlerdir (Yılmaz, 2000).



Şekil 2. TGA fırının içyapısının şematik gösterimi

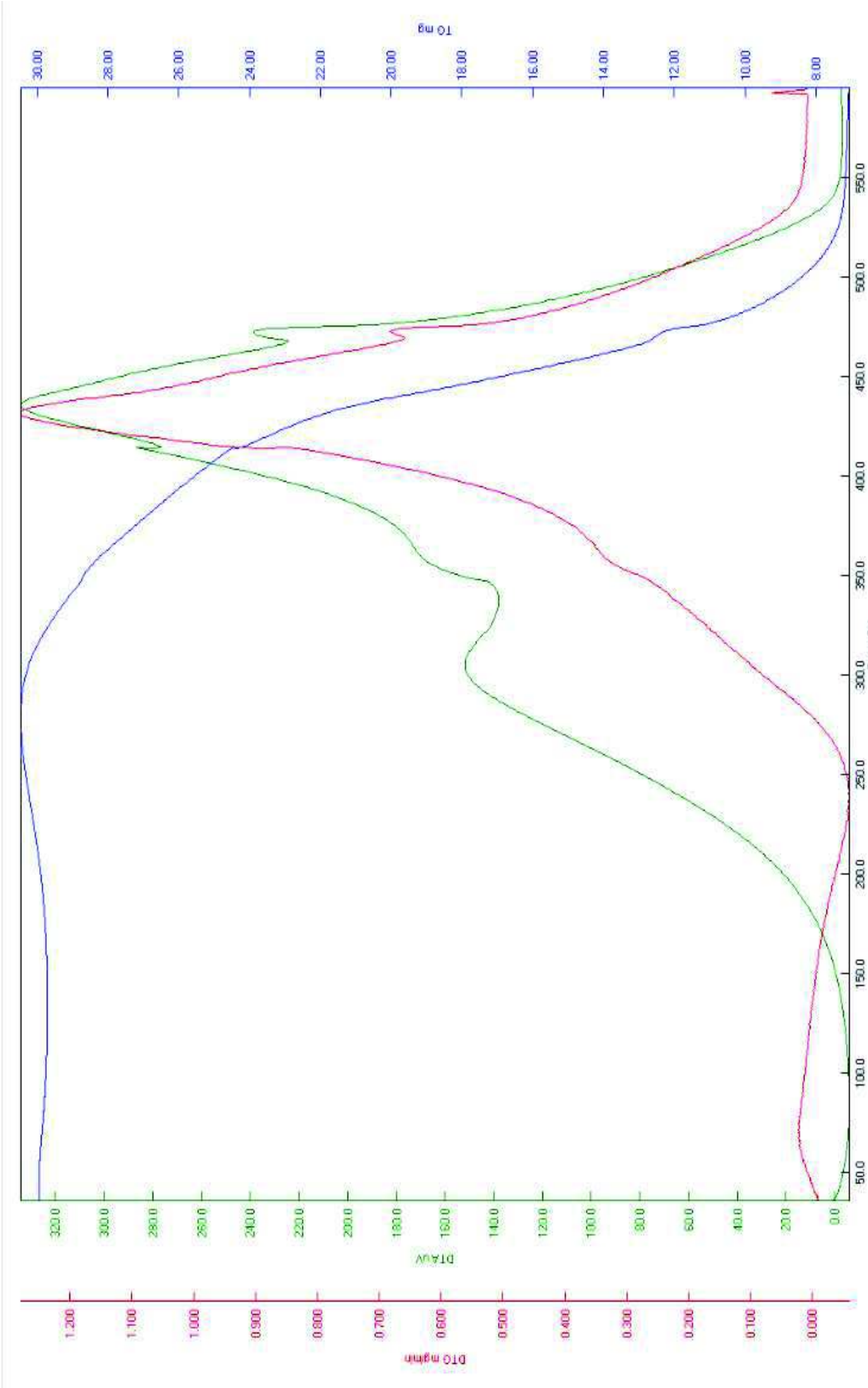
2.1.2 Deneyin Yapılışı

Bu çalışmada yeraltından alınan numuneler öğütülüp, elenerek 105 µm büyüklüğünde hazırlanmıştır. TGA fırınında bulunan terazinin sol koluna referans pota, sağ koluna ise analizi yapılacak örnek numunenin potası yerleştirilmiştir (Şek. 2). Yeraltından alınan kömür numunelerinden 30-35 mg kadar örnek numune potasına konulmuştur.

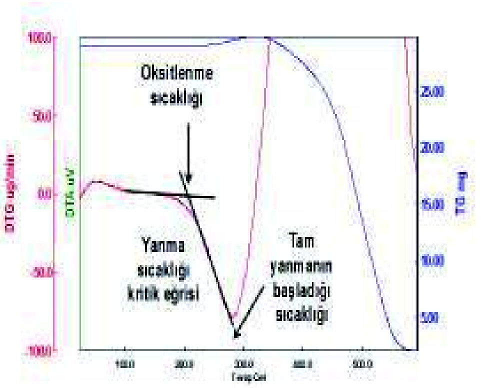
Sıcaklık artış hızı 5 ^{o}C/dak'ya ayarlanmıştır. 200 ml/dak kuru hava akışı sağlanarak, fırın sıcaklığı 600 ^{0}C'a ayarlanmıştır. Şekil 3'te deneyin sonucunda elde edilen TGA eğrisi verilmiştir.

Termogramdaki eğrilerden biri TGA eğrisi (yüzde ağırlık kaybı eğrisi-mavi ile belirtilmiş) diğeri ise DTG eğrisini (ağırlık kaybının türevsel gösterimi-kırmızı ile gösterilmiş) tanımlamaktadır. Yeşil renk ile gösterilen DTA eğrisinin altında kalan alan yardımı ile reaksiyon sıcaklığını veya sıcaklık değişikliğini saptamak olasıdır. Daha sonra bu konuyla ilgili çalışmalar-da yapılacaktır.

Şekil 3'teki TGA termogramda 100 ^{o}C' ın hemen üzerine kadar ağırlık kaybı görülmektedir (mavi renkle verilen eğri). Bunun nedeni, yapıdan nemin aynı zamanda adsorplanmış gazların ayrılmasıdır. Kütle artışı görülmeye başlanan noktada yapıya oksijenin adsorpsiyonu olmaktadır. Oksijenin karbonlu türlerle reaksiyona girerek oluşturduğu karbon-oksijen bileşiklerinin ve parçalanma sonucu oluşan hidrokarbonların yüzeyden ayrılması ile kütlede tekrar azalma görülür. İşte bu nokta yanma sıcaklığının başlangıcını gösterir. Bu kritik noktadaki değişim türevsel olarak elde edilen DTG eğrisinden daha net bir şekilde görülmektedir. Bu nedenle TGA çalışmalarında yanma sıcaklığı DTG eğrilerinden teğet eğriler çizilerek hesaplanmaktadır. Şekil 4'te DTG eğrisine gerekli teğetler çizilerek bu teğetlerin kesim noktalarından yanma sıcaklığının başladığı yerin tespiti gösterilmiştir (Salih ve ark., 2012).



Şekil 3. Amasra tavan damar TGA deney sonucu



Şekil 4. Yanmanın başladığı yerin DTG eğrisinden bulunması

Bu çalışmada Amasra kömürünün tutuşma sıcaklığı 153,9 ^{0} C olarak bulunmuş ve Şekil 5'te gösterilmiştir.

Numunelerin yanması sonucu kütle azalmalarını belirlemek ve yanmanın maksimum olduğu noktayı tespit etmek amacıyla TG-DTG (sıcaklık artışına karşı ağırlık kaybı ve dw/dt) eğrileri çizilmiştir. Bu eğrilerden pik sıcaklığı, son yanma sıcaklığı ve yandıktan sonra kalan kütle tespit edilmiştir. Şekil 6'da Amasra tavan damarın TG-DTG eğrileri verilmiştir.

Şekilde DTG eğrisinin pik yaptığı nokta yanmanın maksimum olduğu bir başka deyişle ağırlık kaybının birim zaman da en fazla olduğu noktadır. Bu noktadan sonra yapıdan yanma sonucu oluşan ürünlerin çıkmasından dolayı kütle de azalma görülmektedir.

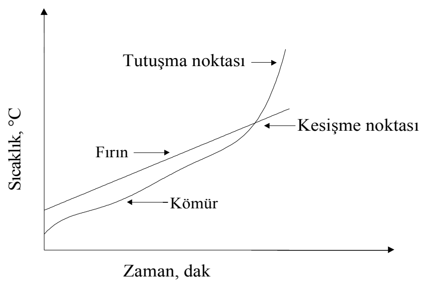
Diğer kömür numuneleri için bulunan sonuçlar (tutuşma sıcaklığı, pik sıcaklığı, son yanma sıcaklığı ve yandıktan sonra kalan kısım) ise Çizelge 1'de net olarak verilmiştir.

2.2 Kesişme Noktası Sıcaklığı Tekniği ile Yapılan Çalışmalar

2.2.1 Kesişme Noktası Sıcaklığı Tekniği

Kesişme noktası sıcaklılığı tekniği, kömürlerin kendiliğinden yanma eğilimlerinin belirlenmesinde kullanılan bir tekniktir. Kesişme noktası sıcaklığı; doğrusal olarak ısıtılan bir ortama (fırın) yerleştirilmiş bulunan küçük bir reaktör içindeki kömür örneğinin, hava akışına maruz bırakılması sonucunda kömürde meydana gelen ekzotermik reaksiyonun kendiliğinden devam edebilmesi için gerekli olan en düşük sıcaklıktır (örnek sıcaklığının, ortam sıcaklığına ulaştığı nokta) (Nandy et al., 1972;Kaymakçı, 1998).

Kesişme noktası esaslı deneylerde hem ortamın (fırın) hem de kömürün sıcaklıkları zamana bağlı olarak kaydedilmiştir. Ortam sıcaklığı eğrisinin, kömür ısınma eğrisi ile kesiştiği nokta kömür örneği için kesişme noktası sıcaklığı olarak saptanmıştır (Kaymakçı ve Didari, 1992). Şekil 7'de kesişme noktasını veren eğri gösterilmektedir.

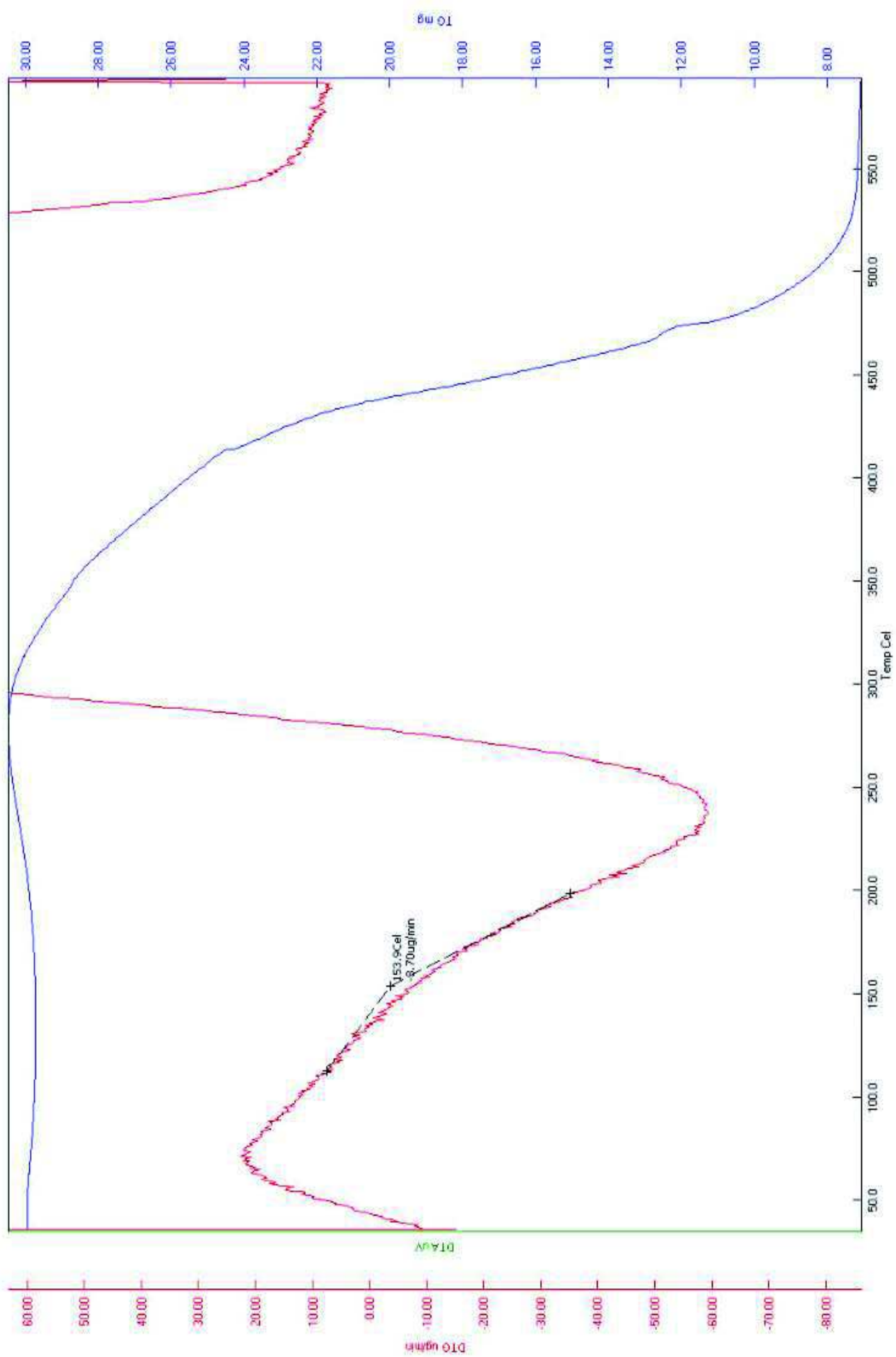


Şekil 7. Kesişme noktası sıcaklığını gösteren eğri (Kaymakçı,1998)

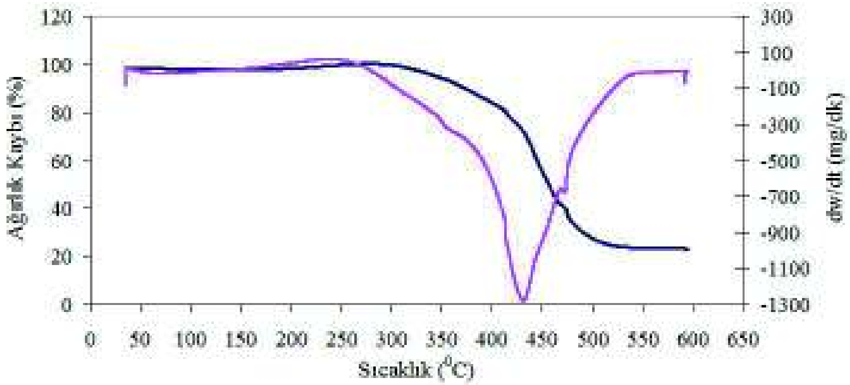
Bu eğriden; kömürün 110 °C'a ulaştığı zaman ile 220 °C'a ulaştığı zaman belirlenmiş ve 110-220 °C arasındaki ortalama sıcaklık artışı (OSA) aşağıdaki eşitlik ile hesaplanmıştır.

$$OSA = 110\ °C / (t_2 - t_1)$$

Burada;
OSA : ortalama sıcaklık artışı, °C/dak,
t_2 : 220 °C'ye karşılık gelen zaman, dak,
t_1 : 110 °C'ye karşılık gelen zaman, dak,
olmaktadır.



Şekil 5. Amasra tavan damar TGA deney sonucu



Şekil 6. Amasra tavan damar TG-DTG eğrileri

Çizelge 1. TGA ve TG-DTG deney sonuçları

Müessese	Kömür Numunesi Alınan Yer	Tutuşma Sıcaklığı (^{0}C)	Pik Sıcaklığı (^{0}C)	Son Yanma Sıcaklığı (^{0}C)	Yandıktan Sonra Kalan Kısım (%)
Amasra	Taşlı	142,4	431,4	546,1	22,5
	Tavan	153,8	436,1	560,6	23,2
	Kalın	144,2	433,6	553,6	36,5
Üzülmez	Kurul	197,8	483,7	592,8	21,6
	Piriç	196,7	476,7	593,3	20,1
Karadon	Hacımemiş	195,9	479,2	591,2	23,9
	Sulu	203	484,8	591,4	37,7
Kozlu	Hacımemiş	197	505,2	582,2	11,4
	Sulu	187	465,6	574,8	4,2
	Kurul	194,5	496,2	578,4	35,7
	Çay	196	471,5	572,6	12,1

Deney verilerinden elde edilen bu bilgiler ile yanabilirlik indeksi aşağıdaki eşitlik yardımıyla hesaplanmıştır.

$$\text{İndeks (FCC)} = \frac{110-220^{o}\text{C arasındaki ort. sıcaklık artışı}}{\text{Kesişme noktası sıcaklığı}} \times 1000$$

Eşitlikten elde edilen değerler kömürün risk sınıfının belirlenmesinde kullanılmaktadır. Çizelge 2'de farklı kömürlerin yanma yatkınlıklarının belirlenmesi amacıyla kullanılan risk sınıfları gösterilmektedir.

Çizelge 2. Kendiliğinden yanma indeksine (FCC) gore risk sınıflandırması (Feng et al., 1973)

Risk İndeksi	Risk Sınıfı
0 - 5	Düşük
5 - 10	Orta
>10	Yüksek

2.2.2 Deneysel Çalışma

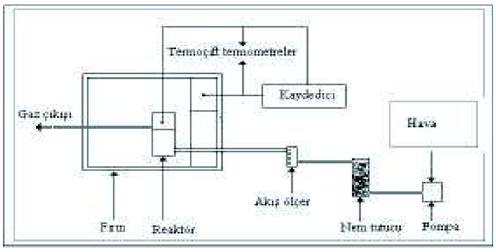
Bu araştırmada kendiliğinden yanma deneyleri, Bülent Ecevit Üniversitesi Mühendislik Fakültesi Maden Mühendisliği Bölümü'ndeki laboratuarlarda kurulu bulunan ve kesişme noktası sıcaklığı tekniğini esas alan deney setinde yürütülmüştür. Kendiliğinden yanma deneyleri için örnek alma işlemlerine uygun olarak kömür damarlarından alınan 11 örnek laboratuara getirildikten sonra sınıflandırılmış ve 200 mesh altına öğütülmüştür.

Deney setinin ana elemanları aşağıdaki gibidir:

- Kontrollü şartlar altında ısıtılan bir ortam (Fırın)
- İçine kömür konulan deney tüpü (Reaktör)
- Örnek ve ortamın sıcaklıklarının ölçülmesinde kullanılan sıcaklık ölçüm ve kayıt birimleri (ısıl çiftler ve kaydediciler)
- Deney setine hava verilmesini sağlayan mini kompresör (Kaymakçı, 1998).

Deney setini genel olarak gösteren bir resim ve akış şeması Şekil 8 ve Şekil 9'da verilmektedir.

Şekil 8. Kendiliğinden yanma deney seti



Şekil 9. Kendiliğinden yanma deney setinin akış şeması (Kaymakçı, 1998)

3 KENDİLİĞİNDEN YANMA PARAMETRELERİNİN DEĞERLENDİRİLMESİ

Bu çalışmada analiz edilmiş örneklerin kendiliğinden yanma deney sonuçları, kömür analiz sonuçları ve kendiliğinden yanma risk sınıfları topluca Çizelge 3'te verilmiştir.

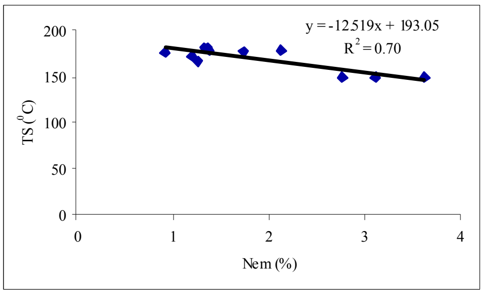
Türkiye Taşkömürü Müessesesi ocaklarından alınan kömür örneklerinin analiz sonuçlarına bakıldığında nem içeriğinin % 0,92 ile % 3,62, kül içeriğinin % 7,86 ile % 45,80, uçucu madde içeriğinin % 16,97 ile % 34,27 ve sabit karbon içeriğinin % 35,87 ile % 66,79 arasında değiştiği görülmektedir. Yanar kükürt içerikleri % 0,20-0,87, üst ısıl değerler 4071-7745 kcal/kg ve alt ısıl değerler 3922-7495 kcal/kg civarında olmaktadır.

Kesişme noktası tekniği deney sonuçlarına göre örneklerin tutuşma sıcaklıkları 149 ile 182 °C arasında değişmektedir. Bu değerler, taşkömürü için beklenen değerlerdir. Ortalama sıcaklık artışı ise 0,79-1,41 °C/dak'dır. Yatkınlık indeks değerleri 4,14 - 9,4 dak,$^{-1}$ olarak bulunmuştur. Bu değerlere göre kömürler, genel olarak, kendiliğinden yanma açısından "orta" risk grubundadır.

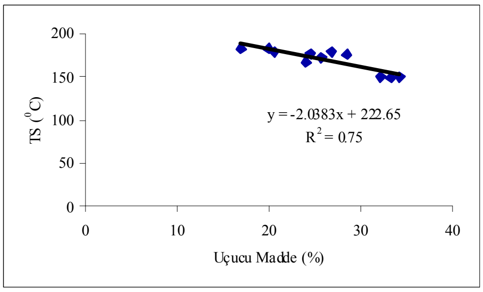
TGA deney sonuçlarına göre örneklerin tutuşma sıcaklıkları 142,4 ile 203 °C arasında değişmektedir. Buna göre tutuşma sıcaklığı en yüksek olan ve diğerlerine göre en düşük yanma riskine sahip olan damar Karadon Sulu damardır. Yanma sıcaklığı en düşük olan damar ise Amasra Taşlı damardır. Ağırlık kayıpları ise % 4,2 – 37,7 arasında değişmektedir. En fazla ağırlık kaybı Kozlu Sulu damarda olmuştur.

Bu bölümde ayrıca 11 deneyden elde edilen kendiliğinden yanma parametreleri; kesişme noktası sıcaklığı (KN), ortalama sıcaklık artışı (OSA) ve kendiliğinden yanma indeksi (FCC) ile kömürlerin bünyesel özellikleri arasındaki ilişkiler incelenmiştir.

Kendiliğinden yanma parametreleri bağımlı değişken olarak, kömür özellikleri kendiliğinden yanma olayını etkileyen bağımsız değişkenler olarak ele alınmıştır. Şekil 10'da Tutuşma noktası, Şekil 11'de FCC indeksi ve Şekil 12'de OSA ile kömür özellikleri arasındaki dağılım grafikleri gösterilmiştir.

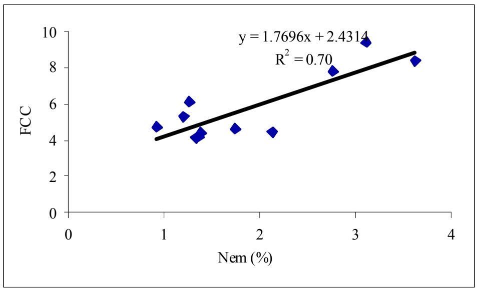


a. Tutuşma noktası sıcaklığı ve nem arasındaki dağılım grafiği

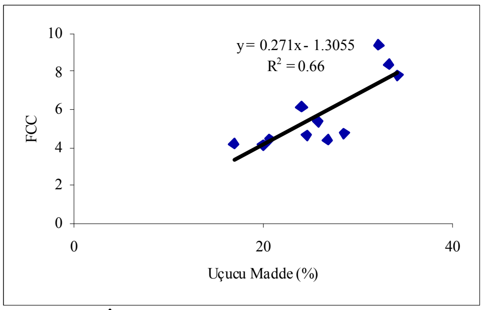


b. Tutuşma noktası sıcaklığı ve uçucu madde arasındaki dağılım grafiği

Şekil 10. Tutuşma noktası sıcaklığı ve kömür özellikleri arasındaki dağılım grafikleri



a. FCC İndeksi ve nem arasındaki dağılım grafiği



b. FCC İndeksi ve uçucu madde arasındaki dağılım grafiği

Şekil 11. FCC İndeksi ve kömür özellikleri arasındaki dağılım grafikleri

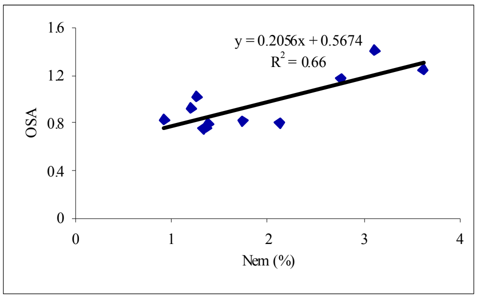
Çizelge 3. Kendiliğinden yanma deney sonuçları, kısa analizler (orijinal baz) ve risk sınıfları

Müessese	Örnek No	TS* (°C) (TGA)	TS (°C) (KN)	Risk Sınıfı	OSA** (°C/dak)	FCC**** (1/dak)	Nem (%)	Kül (%)	Uçucu Madde (%)	Sabit Karbon (%)	Yanar Kükürt (%)	Karbon	Üst Isıl Değer (Kcal/kg)	Alt Isıl Değer (Kcal/kg)
AMASRA	Taşlı	142,4	150	II	1,41	9,4	3,11	17,53	32,25	47,11	0,34	67,73	5851	5613
	Tavan	153,9	150	II	1,17	7,8	2,76	17,31	34,27	45,66	0,40	65,21	5937	5699
	Kalın	144,2	149	II	1,25	8,39	3,62	22,68	33,38	40,32	0,70	56,47	5320	5095
ÜZÜLMEZ	Piriç	196,7	179	I	0,79	4,39	1,38	17,53	26,91	54,18	0,87	78,92	6825	6600
	Kurul	197,8	177	I	0,82	4,64	1,74	27,55	24,59	46,12	0,77	69,61	5950	5750
KARADON	Hacımemiş	195,9	179	I	0,80	4,45	2,13	28,47	20,68	48,72	0,42	64,60	5946	5748
	Sulu	203	182	I	0,76	4,20	1,36	45,80	16,97	35,87	0,47	46,40	4071	3922
KOZLU	Hacımemiş	197	167	II	1,02	6,10	1,26	7,86	24,09	66,79	0,38	84,94	7745	7495
	Sulu	187	172	II	0,92	5,33	1,20	9,21	25,80	63,79	0,46	83,93	7489	7242
	Kurul	194,5	182	I	0,75	4,14	1,33	36,97	20,03	41,67	0,20	57,42	5126	4953
	Çay	196	176	I	0,83	4,73	0,92	9,14	28,59	61,35	0,25	87,68	7687	7441

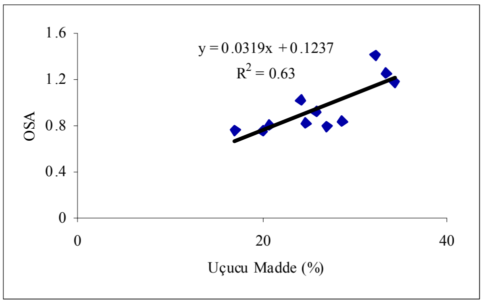
*TS:tutuşma sıcaklığı

**OSA:Ortalama sıcaklık artışı

***FCC:yanma yatkınlık indeksi



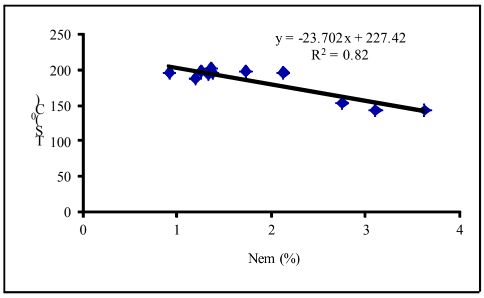
a. OSA ile nem arasındaki dağılım grafiği



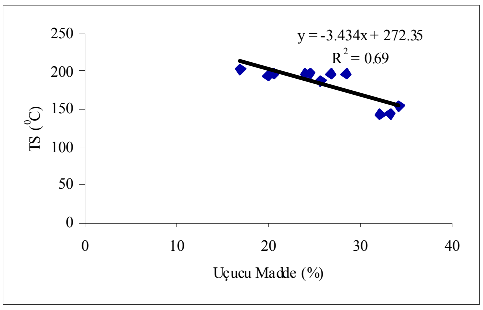
b. OSA ile uçucu madde arasındaki dağılım grafiği

Şekil 12. OSA ile kömür özellikleri arasındaki dağılım grafikleri

Şekil 13'de TGA yönteminden elde edilen Tutuşma noktası ile kömür özellikleri arasındaki dağılım grafikleri gösterilmiştir.



a. Tutuşma noktası sıcaklığı ve nem arasındaki dağılım grafiği



b. Tutuşma noktası sıcaklığı ve uçucu madde arasındaki dağılım grafiği

Şekil 13. TS ile kömür özellikleri arasındaki dağılım grafikleri

Şekillerden görüleceği üzere;

TS ile nem arasında yüksek derecede (r^2=0,70) doğrusal ilişki görülmektedir, TS ile uçucu madde arasındaki (r^2=0,75) doğrusal ilişki pratiğe uygundur. TS ile diğer kömür özellikleri arasındaki ilişkiler (r^2 değerleri çok düşük olduğundan) dikkate alınmamıştır.

FCC Risk İndeksi ile nem ve uçucu madde arasında yüksek sayılabilecek doğrusal ilişkiler (r^2 değerleri; sırasıyla 0,70, 0,66) mevcuttur.

OSA ile nem ve uçucu madde arasında da- doğrusal ilişkiler (r^2 değerleri sırasıyla; 0,66 ve 0,63) mevcuttur.

TGA yönteminden elde edilen tutuşma sıcaklığı ile nem arasında yüksek derecede (r^2=0,82); uçucu madde ile arasında (r^2=0,69) doğrusal ilişki görülmektedir.

Bu ilişkilere göre; nem ve uçucu maddenin kendiliğinden yanmayı etkileyen önemli kömür özellikleri olduğu görülmektedir, En iyi ilişki "nem içeriği" ile Tutuşma Sıcaklığı ve "uçucu madde" ile Tutuşma Sıcaklığı arasındadır.

4 SONUÇLAR

Bu çalışmadan elde edilen sonuçlar aşağıdaki gibi sıralanabilmektedir:

Genel olarak, tutuşma noktasının kömür bileşenlerinden nem ve uçucu madde ile anlamlı bir ilişkisinin görüldüğü söylenebilir.

TGA yönteminden elde edilen tutuşma sıcaklıkları, kesişme tekniğine göre bazı durumlarda yüksek çıkmıştır. Ancak hesaplamada oluşturulacak bir matematiksel yöntem ile daha kesin ve doğru sonuçlar elde edilebileceği düşünülmektedir. Ayrıca bu noktada her iki yöntem ile çalışma yapılan kömürlerin numune miktarı, nem ve diğer ortam şartlarının aynı olduğu durumlarda karşılaştırma yapılmalıdır.

Bunlarla birlikte, sadece TGA değerlerini yorumlayarak değil TGA-MS sonuçlarından hidrokarbon oranları (özellikle etilen/asetilen oranları), karbondioksit/karbon monoksit oranlarından daha ayrıntılı bilgilerin elde edilmesi için çalışmalar yapılmaktadır.

Farklı müesseselerde aynı isimle adlandırılan damarların tutuşma sıcaklıklarının farklı olduğu gözlemlenmiştir. Kendiliğinden yanma parametreleri ile kömür bileşenleri (nem, uçucu madde) arasında anlamlı ilişkiler saptanmıştır. Diğer kömür bileşenleri (kül, sabit karbon vb,) ile anlamlı ilişkiler bulunmamaktadır.

Kendiliğinden yanma üzerindeki en etkili kömür bileşenleri “nem ve uçucu madde” olarak belirlenmiştir.

TTK kömürlerinin kendiliğinden yanmaya yatkınlıkları “orta” derecededir, Bu ocaklarda madencilik ilkelerine uygun işletmecilik koşulları sağlandığı takdirde kendiliğinden yanma sorunlarının giderilebileceği düşünülmektedir.

KAYNAKLAR

Didari, V, 1986. Yeraltı ocaklarında kömürün kendiliğinden yanması ve risk indeksleri, *Madencilik*, s.29-34.

Feng, KK, Chakravorty, RN, Cochrane, TS, 1973. Spontaneous combustion-A Coal mining hazard, *The Canadian Mining and Metallurgical Journal*, October, pp. 75-84.

Kaymakçı, E, Didari, V, 1992. Kömürün kendiliğinden yanma yatkınlığının belirlenmesinde kullanılan indeksler, *Türkiye 8. Kömür Kongresi*, s.129-140.

Kaymakçı, E, 1998. Zonguldak havzası damarlarına uygulanabilecek bir kendiliğinden yanmaya doğal yatkınlığı değerlendirme tekniğinin geliştirilmesi, Doktora Tezi, Zonguldak Karaelmas Üniversitesi Fen Bilimleri Enstitüsü, s.148, Zonguldak.

Kaymakçı, E, Didari, V, 2000. Kömür özellikleri ile kendiliğinden yanma parametreleri arasındaki ilişkiler, *Türkiye 12. Kömür Kongresi*, s.147-156.

Nandy, DK, Banerjee, DD, and Chakravorty, RN, 1972. Application of crossing-point temperature for determining the spontaneous heating characteristics of coals, *Journal of Mines Metals and Fuels*, Vol. 20, No. 2, pp. 41-48.

Ökten, G, 1988. Kömürlerin kendiliğinden yanması ve önlenmesi için alınacak tedbirler, *Kömür Kimyası ve Teknolojisi Kitabı*, İstanbul, s, 102-103.

Salih, B, Öztürk, M, Sarıçam, F, 2012. Taşkömürü havzası kömürlerinin tutuşma sıcaklıkları ile ilgili özgün kriterlerin belirlenmesi, *Türkiye 18, Kömür Kongresi Bildiriler Kitabı*, s,471-485, Zonguldak.

Yılmaz, B, 2000. Termo-analitik yöntemler, Ders notları, Osmangazi Üniversitesi Mühendislik Fakültesi,Eskişehir.

Application of ARIMA Residuals Chart for Spiral at a Coal Preparation Plant

Adem Taşdemir
Eskişehir Osmangazi University, Mining Engineering Department, Mineral Processing Division, Eskişehir, Turkey

ABSTRACT In this research, the ash content of a fine coal product (0.1-0.5 mm) produced by a spiral at a coal preparation plant in Turkey was investigated by applying statistical process control (SPC). During the application of SPC method, the data investigated are assumed to be normally distributed and independent. Therefore, the data should be checked to verify these assumptions needed. Analyses of this study showed that the ash data of spiral product are non-normal and not independent i.e. auto-correlated. As a result, the standard Shewhart individual chart was found inadequate for controlling spiral ash data and more complicated technique, ARIMA residuals chart was applied. Many transformation methods was tried to achieve normality. Johnson transformation was found the best method for the ash data. Auto-correlation was removed successfully by the ARIMA (1, 0, 2) time series model. Finally, the correct control limits and unusual points of ash content were determined by applying ARIMA residuals chart.

1 INTRODUCTION

The importance of coal is well known in the world. It is necessary to remove the unwanted components of coal before its end usage. In Turkey, majority of coal resources are the lignites with low calorific value and high ash content. Therefore, a suitable coal preparation method is indispensible to improve its quality. In general coal is washed in a coal preparation plant and many different coal products with different coal quality and particle size are obtained. The quality of these products is affected by many factors such as coal quality feed to the plant, operating parameters of the plant etc. Quality of clean coal products produced at a coal preparation plant is very important and influences their final usage area. An efficient method is also essential to control the related coal characteristics.

In addition to calorific value, moisture content, fixed carbon, the ash content is one of the most important quality characteristics of clean coal. It is often selected as a product quality parameter for the evaluation of a coal beneficiation process.

Statistical process control (SPC) is widely used method in almost all industrial processes for many years. The quality of a produced coal at a coal preparation plant is mainly controlled by chemical analysis of the feed, concentrate and tailing products as in many mineral processing plants. The SPC charts are widely used to monitor manufacturing processes with the objective of detecting any change in a process that may affect the quality of the output (Stoumbos and Reynolds, 2000). It has been also used in mineral/mining applications to improve or control the process efficiency of different minerals (Ankara and Bilir, 1995; Ipek et al, 1999; Tütmez, 1999;

Bhattacherjee and Samanta, 2002; Bayat and Arslan, 2004; Aykul et al., 2005; Vapur et al, 2005; Elevli and Behdioğlu, 2006, Ankara and Yerel, 2008, Vapur, 2009; Elevli et al., 2009; Aykul et al., 2010, Deniz and Umucu, 2010; Yerel and Ankara, 2011; Taşdemir, 2012).

During the applications of SPC, the data tested are assumed to normally distributed and independent. Since the primary purpose of SPC is to detect quickly unusual sources of variability so that their root causes can be properly addressed, data auto-correlation and skewness have severe adverse impacts on the economic benefits of implementing SPC. (Castagliola and Tsung, 2005). However, many research studies have been showed that violation of these assumptions has resulted with many false alarms which cause wrong decisions about the process (Alwan and Roberts, 1988; Bisgard and Külahçı, 2005, Stoumbos and Reynolds, 2000 and the references cited therein). Therefore, both normality and independence assumptions should be satisfied since most industrial data are usually non-normal and also often auto-correlated.

The symmetric control limits seen on charts results from the assumption normality. It has been shown that a typical X chart would provide very misleading results in the case of highly skewed distributions (Montgomery, 1997; Borror et al., 1999; Montgomery and Runger, 2011). Many researchers studied with control charts in the presence of auto-correlation (Alwan and Roberts, 1988; Lu and Reynolds, 1999; Bisgard and Külahçı, 2005, Smeti et al., 2006; Psarakis, S. and Papaleonida, 2007). Compared to other industries, significant effect of auto-correlation is often ignored in mining SPC studies. Limited examples of mineral/mining SPC applications considering auto-correlation can be given in literature (Samanta and Bhattacherjee, 2001; Bhattacherjee and Samanta, 2002; Samanta, 2002; Elevli et al., 2009; Tasdemir, 2012a; Tasdemir, 2013).

It is suggested that the application of traditional control charts to the residuals of an appropriate ARIMA(p,d,q) time series model fitting to the data investigated can be a suitable solution to overcome auto-correlation problem (Alwan and Roberts, 1988) Alwan and Roberts (1988) called these residual charts as special cause chart (SCC).

This study aims to present the SPC chart on ash data of a spiral product at Dereköy coal washing plant in Soma, Turkey. The one year as-received ash data of spiral product was used. The ash data was determined non-normally distributed and auto-correlated. Therefore, traditional Shewhart chart was found inadequate for controlling ash data. The Johnson transformation was found the best method to achieve a normal distribution of ash data. Since the transformed ash data also found auto-correlated, an appropriate ARIMA (p,d,q) time series model which is the same model for raw ash data was determined for transformed ash data to remove auto-correlation. Finally, Shewhart chart of residuals from ARIMA time series model were applied to ash data and compared with individual chart of raw ash data.

2 NORMALITY AND INDEPENDENCE IN CONTROL CHARTS

In industrial practice, very few quality characteristics of a process are actually normally distributed, especially in modern data rich manufacturing processes where large amount of data are routinely collected from individual units (Castagliola and Tsung, 2005).

A number of authors have point out that Shewhart charts for subgroup means work well irrespective whether the measurements ar normally distributed or not (Wheeler, 1991). However, the behavior of traditional control chart for individual measurements is seriously affected by departures from normality (the effects of skewness and kurtosis) (Borror et al., 1999; Stoumbos and Reynolds, 2000). The normality assumption is risky particularly *individual* measurements are used (Vermaat, 2006). Non-normal data are quite common in SPC applications. It is important to note that charts based on samples from non-normal distribution will have asymmetrical control limits.

Without a normal distribution, correct limits of control charts may fail. Montgomery and Runger (2011) studied the behavior of the Shewhart control chart for non-normal process data. They concluded that even if the process shows evidence of moderate departure from normality, the control limits given may be entirely inappropriate.

In situations involving a large number of measurements, it may be possible to subgroup the data and construct a mean ($\bar{X}$) chart instead of a X individual chart. However, the measurements should not be subgrouped arbitrarily for this purpose (Srinivasan, 2011). If subgrouping is not possible, two alternatives are to the data to normality.

One approach attempts to transform data. A number of mathematical transformations have been developed over the years. Transforming data means performing the same mathematical operation on each piece of original data, preferably with a transformation method like Box-Cox or Johnson transformation (Chou et al., 1998). Another approach is to modify the usual limits based on a suitable model for the data distribution (Castagliola and Tsung, 2005). Identifying a mathematical distribution can help to develop alternate control limits thus, it can be identified if the data belongs to a particular class of distribution.

In addition to normality, if the quality characteristics represent even low levels of positive correlation, control charts will give misleading results in the form of too many false alarms (Alwan and Roberts, 1988; Reynolds and Lu, 1997, 2001; Lu and Reynolds, 1999; Zhang, 1997; Testik, 2005).

The main effect of auto-correlation in the process data within SPC applications is that it produces control limits that are tighter than desired. This causes a substantial increase in the average false alarm rate and a decrease in the ability of detecting chances on the process (Psarakis, S. and Papaleonida, 2007). In the case of correlated processes the most common approach is to fit a time series model to process data so that forecasts of each observation can be made using previous observations and then to apply the residuals traditional control charts (Alwan and Roberts, 1988; Reynolds and Lu, 1997, Lu and Reynolds, 1999; Lu and Reynolds, 2001). In this approach, the data are modeled by an Autoregressive Integrated Moving Average Processes or briefly ARIMA(*p*, *q*, *d*) time series model. More information concerning the usage and the description of time series model can be found in Box and Jenkins (1976).

3 METHOD

The as-received ash data used in this work was obtained from a spiral product of Dereköy coal washing plant in Turkey. The fine size fraction of -0,5+0,1 mm coals are treated and separated from their schist by spiral at the plant. Statistical process control (SPC) method was applied for monitoring and quality control purposes of fine coals produced with spiral by using one year ash data from January 2010 to December 2010.

The data can be plotted individually or grouped and then plotted the mean on a control chart. Individual control charts are preferred over the mean chart since the mean chart obscures the vision of the practitioner by preventing them from seeing what the process actually looks like. In addition, usage of the mean to characterize the behavior of a process rather than actual data results in a loss of information. Since the available ash data concern in daily samples with size n=1, the type of control chart used in this study was regard to individual observations. The spiral was operated totally 353 days in 2010. Thus, X control chart for individuals (I chart) is used for the control of the process mean from 353 daily ash content data of spiral products.

Minitab 16 statistical software package was used whole of the analyses. In addition, Statgraphics Centurion XVI was also used for automatic selection of time series model which is the most suitable for the ash data evaluated. The normality and independence were checked in order to apply the SPC on raw ash data. Good of fit test for normality was evaluated out by Anderson Darling (AD) method. Auto-correlation for original and ARIMA residuals data was tested by applying auto-correlation function (ACF) and partial autocorrelation (PACF) plots.

According to the initial check of original data, both normality and independence assumptions were failed. The individuals control chart is very sensitive to non-normality. Therefore, firstly a suitable transformation method was applied to make the data normal and then the most suitable ARIMA time series model for the transformed ash values was determined to eliminate the auto-correlation.

Finally, the residuals of the ARIMA model (the difference between the actual values and the predicted from the model) were used for the control of the process mean of ash from spiral. During the evaluation of unusual data values on SPC charts, one or more points that fall outside the ±3σ control limits are considered as a run rule in this study.

4 RESULTS

4.1 Individual Chart of Ash Data

An individual control chart for daily ash values of spiral product when considering the entire years of data in 2010 as well as the determination of indices calculated based on the assumption of normality and independence is plotted in Fig. 1. The statistical parameters are presented in Table 1. According to these results, 19 data are found beyond the control limits (±3σ lines). 15 points of raw ash data are beyond the +3σ limit while 4 points are beyond the -3σ limit. These points are marked as "1" in Fig. 1 indicating that the spiral produced a fine coal concentrates with uncontrolled 19 times in terms of ash content in 2010.

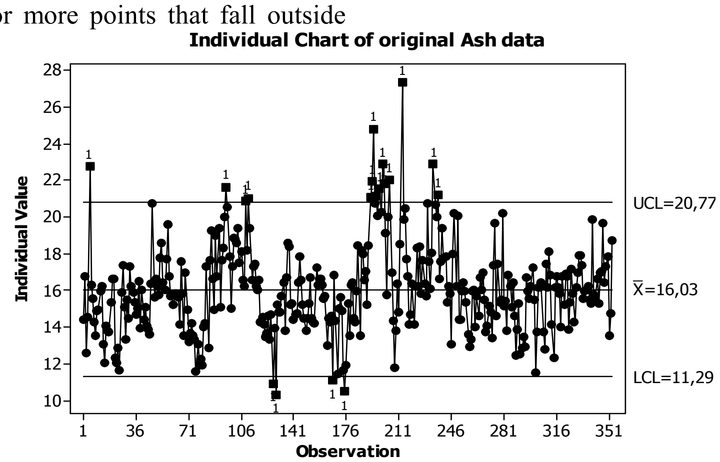


Figure 1. Control chart of raw ash data

Table 1. Individual Chart parameters

Parameter	*Values*
Process mean	16,0299
Process sigma*	1,58021
Mean MR(2)	1,78247
UCL: +3,0 sigma	20,7705
Centerline	16,0299
LCL: -3,0 sigma	11,2893

*: Sigma estimated from average moving range (MR)

Previous studies showed that violations of assumptions for the application of Shewharts charts in different industries resulted with many false alarms on the chart (Borror et al., 1999; Stoumbos and Reynolds, 2000). This causes wrong decisions about the process variable under consideration. Therefore, normality and independence controls were carried out on the ash data in the following section.

4.2 Normality Check of Ash Data

Since the control chart in Fig. 1 is based on a normality distribution, the ash data were tested for normality. According to initial normality check, the ash data found as highly skewed to right and did not obey the normal distribution. The apparent skewness in the data is troubling, since most statistical procedures assume that the data follow a normal distribution. Without a normal distribution SPC chart does not carry any meaning. When the process distribution is non-normal, traditional Shewhart charts may not be applicable. If normality is not tenable, one of the solutions is to find a transformation of the data that normality is a reasonable assumption in the transformed metric. A common practice in such cases is to normalize the process data by applying a suitable data transformation method. Another approach is to identify a statistical distribution which describes the data investigated.

Many statistical distributions such as normal, lognormal, exponential, Weibull, gamma, logistic, log-logistic were tested to fit the ash data to a particular distribution and some transformation methods such as Box-Cox transformation, Johnson transformation were applied to raw ash data. According to these tests log-logistic distribution and Johnson transformation were found statistically significant among the other distribution and transformation methods. Probability plots of normal, log-logistic, optimal Box-Cox transformation (lambda=0) and Johnson transformation with their Anderson Darling normality test results are presented in Fig. 2 to compare the results obtained. Though the histogram gives the impression that the ash data has a bell shape (histogram plot is not given here), the normal probability plot indicates that the data is not normal. Visually the data appears to be non-linear on the plot, and the *p*-value for the plot is less than 0.005 (Fig. 2a). The Anderson Darling (AD) test for normality indicates that ash values have an AD value of 2.831 and lie from normality (Fig. 2a). In order to achieve normality it is necessary to find a suitable distribution type or an effective transformation method to ash data. The optimal lambda value was determined as zero (0) suggesting that natural logarithmic transformation. However, this optimal Box-Cox transformation is not efficient for normality as in evidence of Fig 2c. Even the data show somewhat linearity on the probability plot, *p*-value from AD test is less than 0.05 (p=0.009) suggesting non-normality of Box-Cox transformation to ash data (Fig 2c).

Both log-logistic distribution and Johnson transformation of ash data passed the normality test since their *p*-values are higher than 0.05 (Figs.2b and 2d). However, Johnson transformed data produced higher performance than the log-logistic distribution for the normality. The log-logistic distribution has an AD value of 0.431 and *p*-value of 0.245 (Fig. 2b) while AD is 0.363 and *p*-value is 0.44 for Johnson transformation (Fig 2d). Therefore, Johnson transformation method was preferred for the normality solution for the ash data. The Johnson transformation function applied to ash data was equal to:

$$y = -0,635+1,719*Sinh^{-1}[(x-14,473)/3,332] \quad (1)$$

where;

y: transformed ash value

x: original ash value

These tests show that Johnson transformed ash data can be used to find the control limits of spiral ash data.

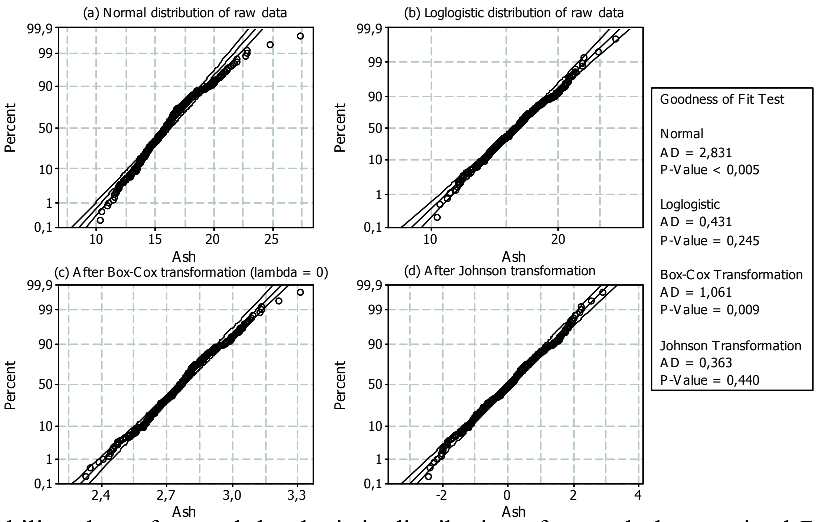


Figure 2. Probability plots of normal, log-logistic distribution of raw ash data, optimal Box-Cox transformed and Johnson transformed ash data with goodness of fit test results

4.3 Individual Chart and 3 Sigma Limits of Johnson Transformed Ash Data

Individual SPC chart of Johnson transformed ash data and its statistical parameters are given in Fig. 3 and Table 2 respectively.

The number of ash data beyond the ±3σ limit was determined as 16. The 7 points of Johnson transformed ash data (Fig. 3) are beyond +3σ which is less than 15 obtained for raw data (Fig. 1). The 9 points are below -3σ (Fig. 3) which is higher than 4 determined raw data (Fig. 1). This result shows us the effect of data normality on the right unusual points in the SPC. Montgomery and Runger (2011) concluded that if the process shows evidence of moderate departure from normality, the control limits may be entirely inappropriate.

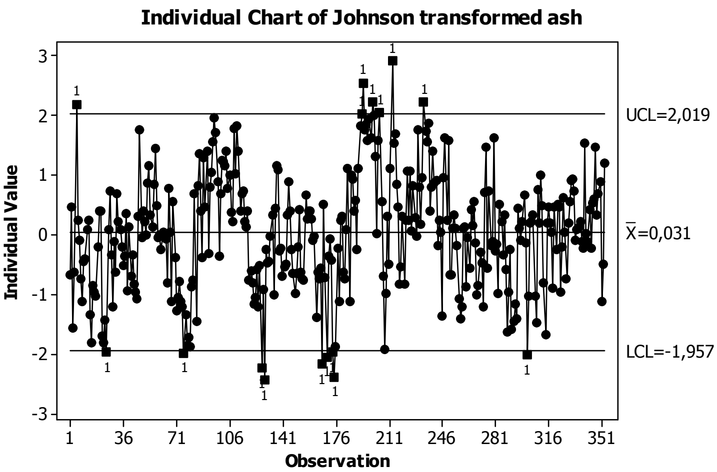


Figure 3. Control chart of Johnson transformed ash data

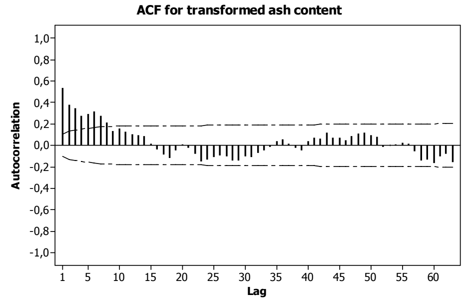
Table 2. Individual Chart parameters of Johnson transformed ash data

Parameter	*Values*
Process mean	0,0310892
Process sigma*	0,662617
Mean MR(2)	0,747432
UCL: +3,0 sigma	2,01894
Centerline	0,0310892
LCL: -3,0 sigma	-1,95676

*: Sigma estimated from average moving range (MR)

4.4 Independence Control of Johnson Transformed Ash Data

In addition to normality, correlation between consecutive observations can lead to a badly underestimated process sigma. To check the auto-correlation effect, correlogram graphs were generated for more than 60 lags. The auto-correlation Function (ACF) and the sample Partial Auto-correlation Function (PACF) plots which are representative of the auto-correlation structure are given in Fig. 4. Details of ACF and PACF methods can be found in Montgomery et al. (2008).



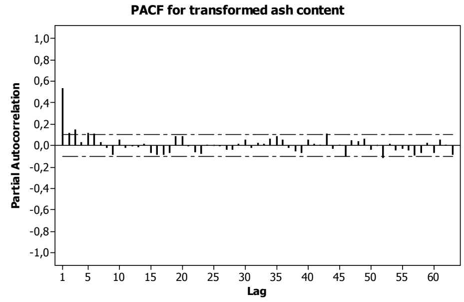


Figure 4. ACF and PACF plots of transformed ash data

According to these plots, the values decay linearly and approach to zero after 15 lags in ACF indicating the stationary of the ash data. Therefore, there is no need to difference the ash values. There is a significant auto-correlation at first lags of ACF and PACF plots (r = 0.539). There is an important spike at first lag in PACF plot.

4.5 ARIMA Model and ARIMA Residual Chart

To remove the auto-correlation, the best ARIMA (*p*, *d*, *q*) model of time series with lowest Akaike Information Criterion (AIC) value was determined by Statgraphics software. The details of this method can be found in Tasdemir (2012). The ARIMA (1,0,2) was found the best describing model for transformed spiral ash data. Initial check showed that the same model was also theh best for raw ash data (the results are not given here). The parameters of the model are presented in Table 3. For ARIMA (1,0,2) model, *p*-values of the all model terms for the AR(1), MA(1) and MA(2) are less than 0.05 thus, all of them are statistically significant. As the time series values of ACF plot lie down quickly (as seen in Fig. 4), there is no need to difference the transformed ash values i.e. the model does not contain difference term of *d*.

Table 3. ARIMA(1,0,2) model summary

Parameter	*Estimate*	*Stnd. Error*	*t*	*P-value*
AR(1)	0,89815	0,04164	21,568	0,0000
MA(1)	0,45297	0,06884	6,5799	0,0000
MA(2)	0,16187	0,06154	2,6303	0,0089

4.5.1 Normality and Auto-correlation of ARIMA Residuals

After fitting an appropriate ARIMA time series model to the data, the residuals (the difference between original data and model estimated data) are calculated. If the model is suitable, residuals should be independently and identically normally distributed and then control charts can be applied to residuals.

The residuals of ARIMA(1,0,2) model were also checked for normality and randomness since the residuals should obey normality and random in order to apply ARIMA residuals chart. The four in one plots of residuals are presented in Fig. 5. As seen in these plots, the residuals obey the normal distribution very well with a *p*-value of 0.609 based on the AD normality test.

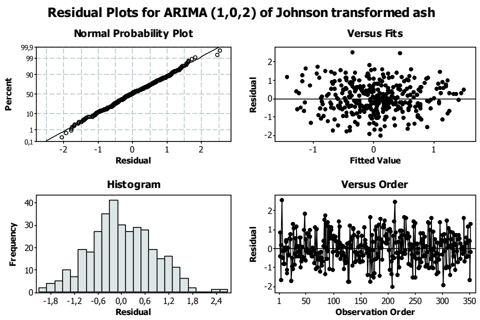


Figure 5. Residual plots of ARIMA(1,0,2) time series model for Johnson transformed spiral ash data

The residuals were also tested for the independence or randomness. The resulted ACF and PACF plots of residuals are given in Fig. 6.

As seen clearly, all of the auto-correlation and partial auto-correlation coefficients are within or very close to the 95.0% confidence limits. This results indicate that no significant auto-correlation remains in the residuals of ARIMA(1,0,2) model.

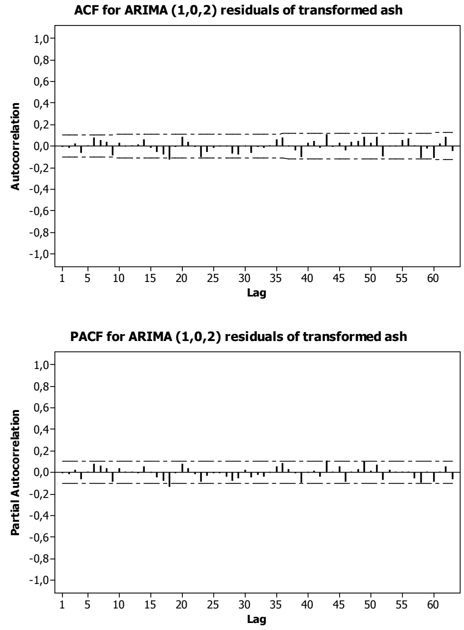


Figure 6. ACF and PACF plots for the residuals of ARIMA(1,0,2) model

4.5.2 ARIMA Residuals Chart for Johnson Transformed Ash Data

The ARIMA residual chart obtained is shown in Fig. 7. It is seen that only 2 of 353 observations presents uncontrolled data points. The remaining points seem within the ±3σ control limits.

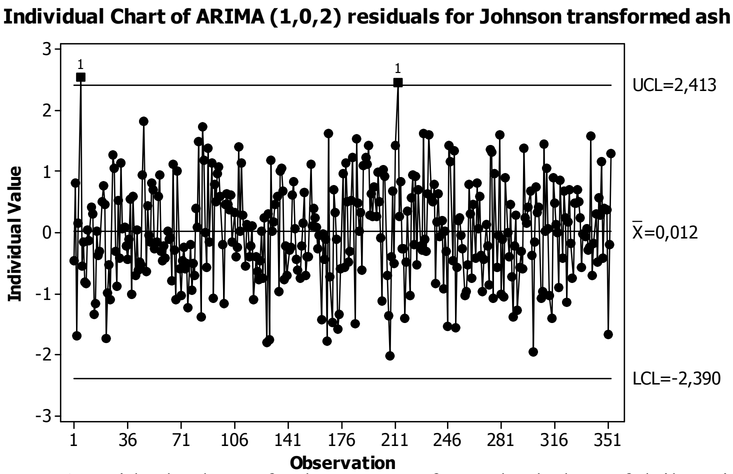


Figure 7. ARIMA residuals chart of Johnson transformed ash data of daily spiral products

5 CONCLUSION

It is important to know whether the coals processed are statistically in control or not and to detect the correct control limits in terms of ash content. Therefore, this study aims to determine the control limits and uncontrolled points of one year as-received ash data of fine sized coal concentrate (0.1-0.5 mm) obtained by spiral from a coal preparation plant.

It was shown that both normality and independence assumptions affect the performance of individual Shewhart chart reversely and should be satisfied in order to apply it. The suggested solution found for the ash content of spiral product was to apply a Johnson transformation on raw data for ensuring data normality, then to use ARIMA (1,0,2) time series model for removing auto-correlation and finally to construct a control chart by using the ARIMA residuals. The ARIMA residual charts can be useful tool for the improvement of the product quality produced at coal preparation plant since the time series model obtained for ash content captures the dynamic structure of ash data.

In addition, time series models are often preferred for the near future estimations by using past data since these models reflects the inherent characteristics of data investigated. Therefore the ARIMA(1,0,2) model obtained in this study for spiral concentrate may also be used to predict ash content of products' to be produced in near future since the ash content is measured after production process at the plant.

In order to use a control chart for future observations, all the points must be in control. After omitting the 2 out of control points determined in final ARIMA residual chart and plot a new control chart, the new control limits and mean of the ash content can be used to keep it in statistical control for future monitoring and coal product improvement. The SPC charts may be helpful for the mineral engineer to take preventive actions and may give him enough time to adjust spiral variables.

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REFERENCES

Alwan, L. C., and Roberts, H. V., 1988. Time series modeling for statistical process control, *Journal of Business and Economic Statistics*, 6, pp. 86-95.

Ankara, H. and Bilir, K., 1995. Kriblaj tesisinde kalite denetimi, *Madencilikte Bilgisayar Uygulamaları Sempozyumu*, pp. 235-240 (in Turkish).

Ankara, H., and Yerel, S., 2008. Use of range control charts in determination of variability in surface parallelisms of plates, *Journal of Scientific & Industrial Research*, 67, pp. 1078-1082.

Aykul, H., Akçakoca, H., Ediz, G. and Taksuk, M., 2005. Garp linyitleri işletmesi termik santral kömürleri için istatistiksel süreç kontrol analizi, *Türkiye 19. Uluslararası Madencilik Kongresi*, pp. 313-321 (in Turkish).

Aykul, H., Ediz, G., Akçakoca, H.,.Erarslani K., Taksuk, M., and Dixon-Hardy, D. W., 2010, Statistical process control for coal fired power plant system, *Journal of the Energy Institute*, 83, pp. 41-47.

Bayat, O. and Arslan, V., 2004. Statistical analysis in Turkish chromite mining, *Scandinavian Journal of Metallurgy*, 33, pp. 322-327.

Bhattacherjee, A. and Samanta, B., 2002. Practical issues in the consruction of control charts in mining applications, *The Journal of the South African Institute of Mining and Metallurgy*, pp. 173-180.

Bisgaard, S. and Külahçı, M. 2005. Quality quandaries: the effect of auto-correlation on statistical process control procedures, *Quality Engineering*, 17, pp. 481-489.

Borror, C. M., Montgomery, D. C. and Runger,, G. C., 1999. Robustness of EWMA control chart to non-normality, *Journal of Quality Technology*, 31(3), pp. 309-316.

Box, G. E. P. and Jenkins, G. M., 1976. *Time Series Analysis Forecasting and Control*, Revised Edition Ed. Enders Robinson, Holden-Day Series in Time Series Analysis and Digital Processing, , Oakland, California, USA, p. 571.

Castagliola, P. and Tsung, F., 2005. Auto-correlated SPC for non-normal situations, *Quality and Reliability Engineering International*, 21, pp. 131-161.

Chou, Y. M., Polannsky, A. M. and Mason, R. L., 1998. Transforming non-normal data to normality in statistical process control, *Journal of Quality Technology*, 30(2), pp. 133-141.

Deniz, V. and Umucu, Y., 2010. Particle size control with control charts in a vertical roller mill, *Proceedings of the XII^th International Mineral Processing Symposium*, pp. 109-116.

Elevli, S. and Behdioğlu, S., 2006. İstatistiksel proses kontrolü ile kömür kalitesindeki değişkenliğin belirlenmesi, *Madencilik*, 45(3), pp. 19-26 (in Turkish).

Elevli, S., 2006. Coal quality control with control charts, *Coal Preparation*, 26(4), pp. 181-199.

Elevli, S., Uzgören, N. and Savaş, M., 2009. Control charts for auto-correlated colemanite data, *Journal of Scientific & Industrial Research*, 68, pp. 11-17.

İpek, H., Ankara, H. and Özdağ, H., 1999. The application of statistical procss control, Minerals Engineering, 12(7), pp. 827-835.

Lu, C. W. and Reynolds, M.R. Jr., 1999. Control charts for monitoring the mean and variance of auto-correlated processes, *Journal of Quality Technology*, 31, pp. 259-274.

Lu, C. W. and Reynolds, M.R. Jr., 2001. CUSUM charts for monitoring an auto-correlated processes, *Journal of Quality Technology*, 33, pp. 316-334.

Montgomery, D. C., 1997. *Introduction to Statistical Process Control*, 3^rd edition, John Wiley & Sons, New York, NY.

Montgomery, D. C., and Runger, G. C., 2011. *Applied Probability for Engineers*, Fifth edition, John Wiley Sons, Inc. p. 792

Montgomery, D. C., Jennings, C. L., and Külahçı, M., 2008, *Introduction to Time Series Analysis and Forecasting*, Wiley Series in Probability and Statistics.p.469.

Psarakis, S. and Papaleonida, G. E. A., 2007. SPC Procedures for Monitoring Auto-correlated Processes, Quality Technology & Quantitative Management, 4(4), 501-540

Reynolds, M. R. Jr. and Lu, C. W. 1997. Control charts for monitoring processes with auto-correlated data, *Nonlinear Analysis, Theory, Methods & Applications*, 30(7), 4059-4067.

Samanta, B. and Bhattacherjee, A., 2001. An investigation of quality control charts for auto-correlated data, *Mineral Resources Engineering*, 10, pp. 53-69.

Samanta, B., 2002. Multivariate control charts for grade control using prncipal-component analysis and time series modeling, *Trans. Inst. Min. Metall. (Sect A: Min Technol),* 307, pp. 149-157.

Smeti, E. M., Kousouris, L. P. Tzoumerkas, P. C. and Golfinopoulos, S. K., 2006. Statistical process control techniques on auto-correlated turbidity data from finished water tank, *Proceedings of an International Conference on Water Science and Technology Integrated Management on Water Resources, AQUA 2006*, Athens, Greece.

Srinivasan, A., 2001. Application of information technology and statisitcal process control in pharmaceutical quality assurance & comliance, MSc Thesis in Electrical Engineering & Computer Science, Massachusettes Institute of Technology.

Stoumbos, Z. G. B. and Reynolds, Jr. M. R., 2000. Robustness to non-normality and auto-correlation of individuals control charts, *Journal of Statistical Computation and Simulation*, 66(2), pp. 145-187.

Taşdemir, A., 2012. Effect of auto-correlation on the process control charts in monitoring of a coal

washing plant, *Physicochemical Problems of Mineral Processing*, 48(2), pp. 495-512.
Taşdemir, A., 2013. Analysis of chromite processing plant data by first order autoregressive model, *Physicochemical Problems of Mineral Processing*, 49(1), pp. 157-174.
Testik, M. C., 2005. Model inadequacy and residuals control charts for auto-correlated processes, *Quality and Reliability Engineering International*, 21, 115-130.
Tütmez, B., 1999. *Cevher Hazırlamada İstatistiksel Process Kontrol ve Uygulaması*, Yüksek Lisans Tezi, Çukurova Üniversitesi Fen Bilimleri Enstitüsü, 92 p. (In Turkish).
Vapur, H., Bayat, O. and Akyol, F., 2005. Eti Gümüş A. Ş. liç prosesinde istatistiksel proses kontrolü uygulaması, *Türkiye 19. Uluslararası Madencilik Kongresi*, pp. 347-356 (in Turkish).
Vapur, H., 2009. Applicability of statistical process control for flotation of metals complex, *Asian Journal of Chemistry*, 21(1), pp. 655-665.
Vermat, M. B., 2006. *Statistical Process Control in Non-Standard Situations*, Instituut Voor Bedrijfs En Industriele Statistiek. p.99.
Wheeler, D. J., 1991. Shewhart's charts: myths, facts and competitors, *45th Annual Quality Congress Transactions ASQC*, pp. 533-538.
Yerel, S., Ankara, H., Konuk, A. and Özdağ, H., 2007. Preventive maintance policy supported by a quality control chart and Kolmogorov-Smirnov tests: Emet colemanite mineral processing plant, *Minerals & Metallurgical Process*, 24, pp. 152-156.
Yerel, S. and Anhara, H. 2011. Process control for a coal washing plant using a range control chart and multidimensional scaling analysis, *Energy Sources, Part A*, 33, pp. 1028-1034.
Zhang, N. F., 1997. Detection capability of residual control chart for stationary process data, *Journal of Applied Statistics*, 24(4), pp. 475-492.

Tekirdağ-Malkara-Ibrice Linyit Kömürünün Yıkanabilme Özelliklerinin Belirlenmesi

Determination of Washability of Tekirdağ-Malkara-Ibrice Lignite

Ö. Akyüz, F.Kahraman
Dicle Üniversitesi Mühendislik Fakültesi,Maden Mühendisliği Bölümü, Diyarbakır

ÖZET Linyitin bünyesinde bulunan organik ve mineral orjinli maddeler ile yan kayaçlardan dolayı külü artmakta ve buna bağlı olarak kalorisi düşmektedir. Ayrıca linyitteki fazla kül; hava kirliliğine sebep olmaktadır. Kömür temizleme teknolojilerinden en yaygın olanı; kömürün yıkanarak, mineral madde içeriğinin azaltılmasıdır.

Bu çalışmada; Tekirdağ Malkara-İbrice yöresinde bulunan linyit kömürünün yıkanabilme özellikleri belirlenmiştir. Deneysel çalışmalarda kullanılan kömür; havada kuru halde %2,95 nem, %20,23 kül, %36,39 uçucu madde, %3,52 kükürt ve 4754 kcal/kg'lık üst ısıl değerine sahiptir. Ykanabilme özelliklerinin tespiti için; -50+20 mm, -20+4,75 mm, -4,75+0,5 mm tane fraksiyonlarında yüzdürme-batırmadeneyleri yapılmıştır. Yıkama deneylerinde ağır ortam olarak $ZnCl_2$ kullanılmıştır. Yıkanabilme sonuçlarının değerlendirilmesinde; yoğunluk dağılımı, temiz kömür miktarı, yıkama kolaylığı ve yıkanabilme derecesi kullanılmıştır. Yıkama sonuçlarına göre; kömür numunesi en iyi -4,75+0,5 mm tane boyutunda ve 1,7 g/cm^3 yoğunluklu ağır ortamda yıkanabilmektedir.Tane boyutu azaldıkça yıkama kolaylığı artmaktadır.

ABSTRACT Ash measure of lignite increases because of including impurities and particles that organic and mineral origin correspondingly lignite's calories decreases. In addition, too much ash of lignite causes air pollution. The most common of coal cleaning technology is coal washing which diminish mineral particles contents.

In this study; investigation of washability of Tekirdağ-Malkara-İbrice lignite was carried out. The properties of the coal used in the studies were as follows; the upper coal seam contained %2,76 moisture, %20,23 ash, %36,39 volatile matter, %3,52 sulphur and 4754 kcal/kg gross calorific value. Float and sink experiments have been made to determine the washability properties of the -50+20 mm, -20+4,75 mm and -4,75+0,5 mm size fractions. $ZnCl_2$ was used in sink and float experiments. Density distribution, clean coal amount, washing convenience and washability degree have been used in utilizing the coal cleaning characteristics. According to washing results; optimum washing density and particle size of the coal sample are 1,7 g/cm^3 heavy media and -4,75+0,5 mm particle size. While the particle size is reducing, washability is increasing.

1 GİRİŞ

1.1 Kömür Hakkında Genel Bilgiler

Dünyanın enerji ihtiyacının %70'i fosil yakıtlardan kömür, petrol ve doğalgazdan karşılanmaktadır. Petrol ve doğalgaz kaynakları kısıtlı olan ülkemizde kömürün enerji üretiminde çok önemli bir yeri vardır. Türkiye'de Zonguldak ve çevresinde bulunan TTK tarafından işletilen taşkömürü dışında çeşitli bölgelere yayılmış değişik özelliklerde ve farklı kömürleşme derecesi gösteren linyit yatakları bulunmaktadır. Yer altı ve yerüstü üretim yöntemleriyle üretilen linyitler, genellikle termik santrallerde elektrik enerjisine dönüştürülmekte ve ev yakıtı olarak kullanılmaktadır.

Linyitin bünyesinde bulunan organik ve mineral orijinli maddeler nedeniyle ayrıca üretim aşamasında karışan yan kayaçlardan dolayı külü artmakta buna bağlı olarak kalorisi düşmektedir. Kömürdeki kül oranı arttıkça, kömürün yanması zorlanmakta ve belirli bir kül oranından sonra tamamen durmaktadır. Ayrıca fazla kül kömür nakliye masraflarını arttırmaktadır. Günümüzde, büyük şehirler başta olmak üzere birçok ilde yaşanılan hava kirliliğinin kaynağı olarak linyit kömürleri gösterilmekte ve bu linyit kömürlerinin kullanılmaması için kamuoyu oluşturulmaktadır (Kural, 1998).

Türkiye linyit rezervi yaklaşık 8.3 milyar ton civarındadır (Lynch, 2003). Ancak ülkemiz linyitleri çok miktarda kül ve kükürt içerdiğinden dolayı düşük kalorifik değerlere sahiptir. Düşük kalitedeki linyitlerin ev veya termik santral yakıtı olarak kullanılması ciddi hava kirliliğine ve sağlık problemlerine neden olmaktadır. Dolayısıyla yeterli kalitede linyit üretimi olamadığından yalnızca 2011 yılı kömür ithalatımız 24 milyon ton civarındadır(Kömür Sektör Raporu, 2012).

1.2 Yüzdürme-Batırma Deneyleri

Ülkemiz linyitleri, genel olarak kül içeriği yüksek, ısı değeri düşüktür. Bu özellikleri nedeniyle de öngörülen parametrelere uymamakta, kullanımı sınırlandırılmaktadır (Kömür Sektör Raporu, 2012).Fosil yakıtlar bakımından en fazla rezerve sahip olduğumuz linyit kömüründen daha fazla yararlanmak, ülke ekonomisi açısından kaçınılmazdır. Bu durum ise linyit kömürünün kullanıma sunulmadan önce zenginleştirilmesi ile mümkün olacaktır. Kömür temizleme teknolojilerinden en yaygın olanı kömürün yıkanarak mineral madde içeriğinin azaltılmasıdır. İnorganik maddeleri kömürden ayırmakla, kömürün kül yüzdesi düşürülür ve yanabilir kısım oranı yükseltilir. Böylece ısıl değer yükseldiği gibi, taşıma nakli, ısıtılması, ocak ve stok nemini boşuna işgal etmesi önlenir. Bu fiziksel ayırma işlemi ile kül içeriği tüvenan kömürden daha az olan yıkanmış kömür elde edilir.

Yüzdürme-batırma deneyleri laboratuarda kömürlerin değişik boyutlarda, istenilen özgül ağırlıklarda hazırlanmış inorganik sıvılarla yıkanmasıyla elde edilen ayrılmadır. Bu deney için kömür önce kırılır ve istenilen boyut aralığındaki kesim elenerek alınır. En düşük yoğunluktan başlaması durumunda, kömür numunesi önce en düşük yoğunluklu ağır sıvı içine konulur. Kömür numunesinde bulunan ve yoğunluğu bu sıvının yoğunluğundan daha az olan taneler yüzer, daha fazla olanlar batar. Böylece, sırayla bütün yoğunluklarda aynı işlem tekrarlanır. Deneyin yüksek yoğunluktan başlanması durumunda, kömür numunesi önce en yüksek yoğunluklu ağır sıvıya konulur. Bu defa her bir yoğunlukta batanlar bir kenara alınıp, yüzenler bir alt yoğunluktaki sıvıya beslenir. Genellikle, kömür numunesinin en düşük yoğunlukta yüzen oranı fazla olduğu zaman en düşük yoğunluktan, en yüksek yoğunlukta batan oranı fazla olduğu zaman ise en yüksek yoğunluktan başlanması işlemlerde kolaylık sağlar. Her iki durumda da aynı sonuçlar elde edilir(Abakay, 2007).

2 MALZEME VE YÖNTEM

2.1 Kullanılan Numune

Ülkemizin bilinen kömür yatakları içinde Trakya Bölgesi kömürleri önemli bir yer tutmaktadır.Trakya tersiyer havzası kömürleri araştırmacılar tarafından bulundukları yer dikkate alınarak, Istranca Masifi eteklerinde yer alan kömürler ile Keşan, Malkara, Uzunköprü ve Meriç

yöresinde yer alan kömürler şeklinde gruplandınlmıştır.
Deneylerde kullanılan numune Tekirdağ iline bağlı Malkara ilçesi-İbrice köyü kömür yatağından temin edilmiştir. 56/8 ruhsat sayılı bu yatak, Pullukçu Kömür İşletmesi tarafından işletilmektedir.

2.2 Yöntem

2.2.1 Kimyasal Analiz

Kömür numunesi üzerinde nem, kül, sabit karbon, uçucu madde ve ısıl değer analizi yapılmıştır. Analiz sonuçları tabloda verilmiştir.

Çizelge 1. Tekirdağ-Malkara-İbrice Tüvenan kömürün kimyasal analiz sonuçları

Analiz Edilen Bileşen	Havada Kuru Kömür	Tam Kuru Kömür
Nem %	Kaba = 2,95 Bünye = 5,43 Toplam = 8,38	-
Kül %	20,28	20,78
Uçucu Madde %	36,39	40,24
Sabit Karbon	34,95	35,65
Kükürt %	Piritik = 0,94 Sülfatik = 0,72 Organik = 1,86 Toplam = 3,52	-
Üst Isı Değeri kcal/kg	4754	4739

2.2.2 Petrografik Analiz

Kömür numunelerinin petrografik analizi Maden Tetkik ve Arama Genel Müdürlüğü, Maden Analizleri ve Teknoloji Dairesi' ne yaptırılmıştır. Kömür örneklerinin yansıma ölçümleri ve değerlendirmelerinde MPVSP Leitz marka cihaz kullanılmıştır. Örneğin Rmax değeri % 0,535 olarak ölçülmüş olup, örnek bileşenleri; hüminit %75, liptinit %5, inertinit %4, pirit %7, kil %9 şeklinde ölçülmüştür.

2.2.3 Kırma-Eleme

Kömür numunelerinin kırma işleminde laboratuar tipi Çeneli kırıcı (Retsch BB1/A) kullanılmıştır. Kömür örnekleri -50 mm altına indirilerek ve numune azaltma yöntemleri kullanılarak oluşturulan sınıflandırma numunesi 20 mm, 4,75 mm, 0,5 mm 'lik elekler kullanılarak eleme yapılmıştır.

2.2.4 Öğütme-Eleme

Kömür numuneleri kimyasal analiz için, halkalı değirmen kullanılarak 0,2 mm altına indirildi.

2.2.5 Yüzdürme Batırma

Kömür numunelerinin yüzdürme-batırma işlemleri, -50 + 20 mm, -20 + 4,75 mm, -4,75 + 0,5 mm boyut gruplarında 10 litrelik kovalarda $ZnCl_2$ çözeltilerinde yapılmıştır. $ZnCl_2$ belirli miktarlarda suda çözündürülerek 1.3, 1.4, 1.5, 1.6, 1.8 g/cm^3 yoğunluklu çözeltiler hazırlanmıştır. Numuneler öncelikle 1.8 g/cm^3 yoğunluktaki çözeltiye konup bu yoğunlukta yüzen malzeme süzgeçlerle alınarak 1.3 g/cm^3 yoğunluktaki çözeltiye kadar devam edilmiştir. Numuneler tekrar su ile yıkanarak kömür yüzeyindeki $ZnCl_2$ giderilmiştir.

3 ARAŞTIRMA BULGULARI

3.1 Elek Analizi Sonuçları

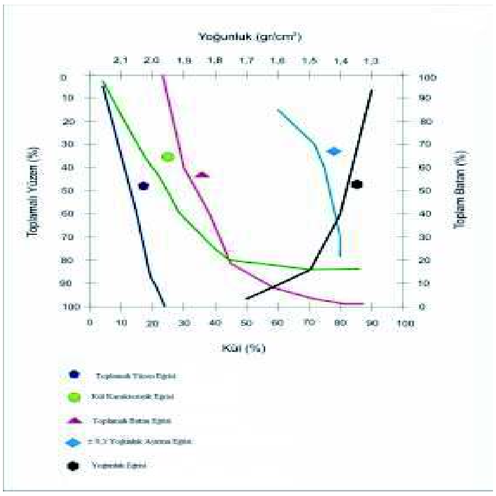
Kullanılan numuneye elek analizi uygulanarak tanelerin boyut dağılımı incelenmiştir. Boyut aralıklarında ki kül, kükürt ve kalori değerleri Çizelge 2' de verilmiştir.

Çizelge 2. Tekirdağ-Malkara Tüvenan kömürün boyut aralıklarında ki Kül, Kükürt ve Kalori Değerleri

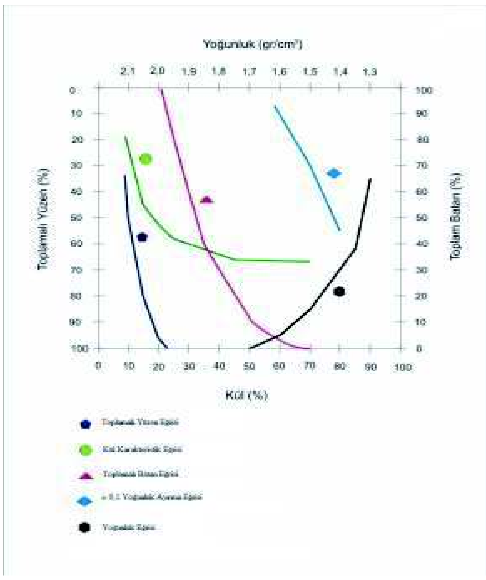
Tane Boyutu (mm)	Ağırlık %	Kül %	Toplam Kükürt %	Üst Isı Değeri Kcal/kg
-50 + 20	42,25	23,02	1,24	4908
20+4,75	35	19,75	1,16	4798
4,75+0,5	22,75	18,13	1,12	4292
Toplam	100	20,76	3,52	4739

3.2 Yüzdürme Batırma Deney Sonuçları

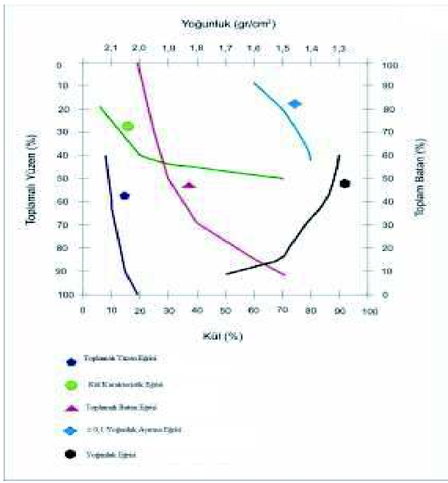
Elek analizleri esas alınarak her bir boyut aralığına ayrı ayrı yapılan yüzdürme batırma deney sonuçlarına göre yıkama eğrileri şekilde verildiği gibidir.



Şekil 1. -50 +20 mm Tane Boyutu Yıkanabilirlik Eğrileri



Şekil 2. Tekirdağ-Malkara Kömürü -20 +4,75 mm Tane Boyutu Yıkanabilirlik Eğrileri



Şekil 3. -4,75 +0,5 mm Tane Boyutu Yıkanabilirlik Eğrileri

4 DEĞERLENDİRME VE SONUÇ

Tekirdağ-Malkara-İbrice linyit kömür numunesi, kuru baza göre % 20,28 kül içermekte olup, kalorifik değeri 4754 kcal/kg' dır. Kimyasal analizler, tuvenan kömürün elek fraksiyonlarında, kül içeriklerinin, tane boyutunun küçülmesi ile azaldığını göstermiştir(Çizelge3.2).Yıkama çalışmasında kömür numuneleri suda ıslatıldıklarında çok az miktarda ufalanmışlardır. Bu gözlemden kömürün yıkama için yeterli parça sağlamlığına sahip olduğu sonucuna varılabilir.

-50 +20 mm tane boyutunda yapılan yüzdürme batırma testlerinde ; 1,6-1,7 g/cm^{3}yoğunluk değerlerinden daha düşük yoğunluklarda ±0.1 yoğunluk değerleri oldukça yüksektir. Bu durum, söz konusu yoğunluk değerinde ve tane boyutunda herhangi bir ayırım yapmanın çok zor olduğunu göstermektedir. -20+4,75 mm tane boyutunda ±0,1 yoğunluk değerlerinin düşük yoğunluk değerlerinde yüksek olmasına karşılık,1,6 g/cm^3'te 10,77 olup,bu yoğunlukta yapılacak bir yıkama işleminin kolay-zor arası olacağı görülmektedir. 4,75+0,5 mm boyutunda ise düşük yoğunluk değerlerinde ±0.1 yoğunluk değerleri yüksek olduğundan yıkamanın zor olacağı ancak 1,6 g/cm^3 yoğunluklu ortamda yıkama yapıldığında, ±0,1 yoğunluk değeri 8,53 olup, bu boyutta ve bu yoğunluklu ortamda yapılacak yıkamanın kolay olduğunu göstermektedir. Ayrıca Şekil 3.3.3'de görüldüğü gibi -4,75 +0,5 mm boyutunda parça külü eğrisi yataya yakın bir görüntü çizmektedir ve bu durum yıkama işleminin kolay olacağını göstermektedir. Tane boyutu azaldıkça yıkama kolaylığı artmaktadır.

KAYNAKLAR

Abakay, H., 2007, *Adıyaman-Gölbaşı Linyitinin Değerlendirilme Olanaklarının Araştırılması*, Osmangazi Üniversitesi, Maden Mühendisliği Bölümü, Doktora Tezi, (18-20 s)

Lynch,R., 2003, *An Energy Overview of Republic of Turkey* U.S. Department of Energy http://www.fe.doe.dov/international/turkey.hmtl

Kural, O, 1998. *Kömür Özellikleri Teknolojisi ve Çevre İlişkileri,* Özgün Ofsat Matbaası, İstanbul, (120-132s.)

Türkiye Kömür İşletmeleri, Stratejik Planlama Birimi,
Kömür Sektör Raporu, 2012,

Optimum location selection for coal preparation plants by using analytic hierarchy process method

S. Çolpan
Mining Engineer, MSc. Turkish Coal Enterprises, TURKEY

M. Yavuz
Associate Professor, Ph.D., Eskişehir Osmangazi University, TURKEY

ABSTRACT This paper presents an Analytic Hierarchy Process (AHP) model, which is developed for selecting the optimum plant location for coal preparation plant in coal mining industry. The every criteria which affect the decision making process in coal industry were determined to solve plant location problem in the AHP model. To select optimum plant location for a new coal preparation plant which is planned to install by Turkish Coal Enterprises located in the Soma region in Turkey, an analysis was carried out by applying the AHP method according to expert opinion. After solving the problem, sensitivity analysis was performed on the results. AHP model outputs were evaluated and the most suitable location was proposed for coal preparation plant. This analysis shows that the AHP method can successfully be applied for the selection of plant location as well as any decision making process in coal mining industry.

INTRODUCTION

Determining the most convenient plant location is one of the commonly encountered problems in engineering applications. Selecting a plant location is very important for all companies in minimizing cost and maximizing the use of resources. The new plant location should be evaluated carefully for the company's competitiveness. To achieve this goal, every potential criterion must be considered in selecting a particular plant location, including investment cost, human resources, availability of acquirement material, climate, etc.

Plant location selection can be described as Multiple Criteria Decision Making (MCDM) problem. MCDM is one of the most considerable branches of Operation Research. MCDM refers to making decisions in the presence of multiple, usually conflicting, criteria. The problems in MCDM are classified into two categories: Multiple Attribute Decision Making (MADM) and Multiple Objective Decision Making (MODM). However, very often the terms MADM and MCDM are used to mean the same class of models and mostly confused in practice. Usually, MADM is used when the model cannot state in mathematical equations and otherwise MODM is used (Hwang and Yoon, 1981). Therefore, determining the plant location is a MCDM problem but this kind of problem is mostly categorized in MADM so that the decision maker can evaluate the subjective criteria in the problem.

The decision maker wants to maximize more than one objective criterion for selection of plant location stage. Among the plant location alternatives, the most suitable plant location must be selected according to objectives and alternatives. MADM applications can help the decision maker to reach the optimal solution for the selection of plant location. In coal mining industry, MADM methods can be applied for the selection of coal preparation plant location because it is a process that includes subjective and objective criteria affecting the selection of plant location among the alternatives.

Analytic Hierarchy Process (AHP) is one of the well-known classical Multiple Attribute Decision Making (MADM) methods. The AHP method was developed by Saaty gives an opportunity to represent the interaction of multiple factors in complex unstructured situations. The method is based on the pair-wise comparison of components with respect to attributes and alternatives. The AHP, since its invention, has been a tool at the hands of decision makers and researchers; and it is one of the most prevalent used the MADM tools. The AHP method have been increasingly incorporated in mining applications such as drilling technology investment analysis, ground support design, tunneling systems design, mine planning risk assessment, underground mining method selection, plant location selection and open cast/pit mining equipment selection (Ataei, 2005; Kazakidis et. al, 2004; Karadogan et. al, 2001, Bitarafan and Ataei, 2004, Sammanta et. al, 2002; Ozer, 2005; Bascetin, 2004).

Turkey's indigenous energy resources consist almost exclusively of lignite and small amounts of hard coal. Turkey has around 1.3 billion tonnes of hard coal and 11.5 billion tonnes of lignite resources, of which 0.5 billion tonnes and 9.8 billion tonnes respectively are proven reserves. The Turkish coal sector produces both hard coal (2.8 million tonnes in 2010) and lignite (69.0 million tonnes), mainly used for power generation. Turkish coal-fired plants have a capacity of approximately 10.6 GW. Coal production in Turkey has increased approximately by 10 million tonnes in the last ten years and reached 71.8 million tonnes in 2010. Almost all of the coal produced is lignite while hard coal's share is only 3.9 %. Over 90 % of total coal production was from three state-owned enterprises in 2010: Turkish Coal Enterprises (TKI), Electricity Generation Company (EUAS) and Turkish Hard Coal Enterprises (TTK). The private sector's share was only around 8%. However, about 35% of coal production reported by the state enterprises is mined by private companies under subcontract. In the future, it is expected that the Turkish coal mining industry's production rate will increase and new coal mines and plants will be constructed in different region of Turkey (Anon, 2012).

The most suitable location must be selected to achieve planned production target in coal industry. To solve this problem, AHP, that is MADM technique, was used in this study and an AHP model was developed to help the decision maker working in coal process. In the developed model, an expert team determined all the criteria affect the plant location of coal industry and weighted the pair-wise comparison matrices at the first stage of solving process of the problem.

1 ANALYTIC HIERARCHY PROCESS (AHP) METHODOLOGY

The AHP method developed by Saaty gives an opportunity to represent the interaction of multiple factors in complex unstructured situations (Saaty, 1980; Saaty, 2000). The method is based on the pair-wise comparison of components with respect to attributes and alternatives. A pair-wise comparison matrix n×n is constructed, where n is the number of elements to be compared. The method is applied for the hierarchy problem structuring. The problem is divided in to three levels: problem statement, object identification to solve the problem and selection of evaluation criteria for each object.

After the hierarchy structuring, the pair-wise comparison matrix is constructed for each level where a nominal discrete scale from 1 to 9 is used for the evaluation (Table 1) (Saaty, 1980; Saaty, 2000).

Table 1. Scale for pair-wise comparisons

Relative Intensity	Definition
1	Of equal value
3	Slightly more value
5	Essential or strong value
7	Very strong value
9	Extreme value
2,4,6,8	Intermediate values between two adjacent judgments

The next step is to find the relative priorities of criteria or alternatives implied by this comparison. The relative priorities are worked out using the theory of eigenvector. For example, if the pair comparison matrix is A, then,

$$(A - \lambda_{max} \times I) \times w = 0 \quad \text{..............................1}$$

To calculate the eigenvalue "λ_{max}" and eigenvector $w=(w_1, w_2,..., w_n)$, weights can be estimated as relative priorities of criteria or alternatives (Saaty, 2000).

Since the comparison is based on the subjective evaluation, a consistency ratio is required to ensure the selection accuracy. The Consistency Index (CI) of the comparison matrix is computed as follows:

$$CI = \frac{\lambda_{max} - n}{n - 1} \quad \text{...2}$$

where λ_{max} is maximal or principal eigenvalue, and n is the matrix size. The consistency Ratio (CR) is calculated as:

$$CR = \frac{CI}{RI} \quad \text{..3}$$

where "RI" Random Consistency Index . Random consistency indices are given in Table 2.

Table 2. The consistency indices of randomly generated reciprocal matrices

	Order of the matrix														
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
R	0.0	0.0	0.5	0.9	1.1	1.2	1.3	1.4	1.4	1.4	1.5	1.4	1.5	1.5	1.5
I	0	0	8	0	2	4	2	1	5	9	1	8	6	7	9

As a general rule, a consistency ratio of 0.10 or less is considered acceptable. In practice, however, consistency ratios exceeding 0.10 occur frequently (Saaty, 1980; Saaty, 2000).

2 GENERAL INFORMATION ABOUT THE TURKISH COAL ENTERPRISES (TKI)

The coal production units that were under the control of Etibank Company, had been transformed into TKI in 1957. In accordance with the Government's general energy policy, TKI has been assigned to produce lignite and some other types of coal and to meet the country's coal demand, contribute to the economy of the country, prepare and execute plans and programs, determine application strategies and to realize them. Lignite reserves of our country are 8.3 billion tons and 2.5 billion tons of these belong to TKI. 46 percent of lignite production of our country is performed by TKI. The Company's production depends on the requirements of the power generation companies and the demand of heating and industry. It produced 26.2 million tons of marketable coal in 2007 and produced 3.6 million tons of coal through license. In 2007, it made a pickling of 268 million m3. 79 percent of the coal sold in 2007 (31.6 million ton) was supplied to thermal plants of EUAS and affiliates of EUAS and 21 percent of it was supplied to industry and domestic market with the purpose of heating. 8,017 MW of total installed power (40,700 MW) of electricity energy of Turkey belongs to thermal plants based on lignite and capacity of current thermal plants fed by TKI is 4,203 MW. 22 percent of total EUAS's electricity production in 2006 was performed by the input that TKI has provided. Market share is 48 percent in domestic market excluding imports and 37 percent including imports.

3 INFORMATION ABOUT THE DIRECTORATE ESTABLISHMENT OF AEGEAN LIGNITE'S IN SOMA

Mining operations begin in the region since 1913. This region depends on to the lignite of Garp from 1939 until 1978, and then left from Lignite's of Garp in 1978 and continued the activities with the status of establishment until 1995. Then TKI got given respectively, regional directorate, operation directorate and finally the re-establishment of a legal entity in April 2004 and then the establishment ongoing activities of the center of the largest entity which is in Soma and 90 miles away to Manisa
(Figure 1).

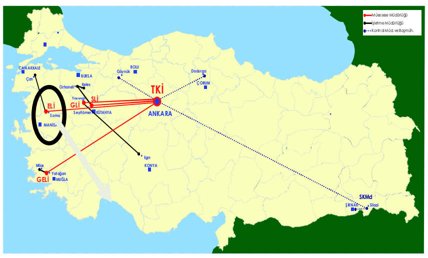


Figure 1. The location of Soma

Establishment which is located in the township of Manisa Soma has 610 million tons of lignite reserves with lower heating between the value of 2080-3150 kcal / kg. TKI have this reserves which are spread over 24.4 thousand hectares and 71% of this reserves licenses may be taken with underground operations by TKI.

This establishment supply fuel to Soma Thermal Power with 1034 MW (2x22, 6x165) power and meet the demand for heating and industry. 64% of sales were to Thermal power plants in 2011. TKI's approximately half of the total sales of heating and industries are met by this establishment. In addition, there are coal preparation and coal extraction plants in Soma to improve the quality of coal. In open cast mining uses large-capacity excavators and heavy trucks. Underground mining is done for the tender and the period of royalty with this method, the amount of the annual productions are 5.2 million tons. The total productions of establishment reached 9.6 million tons in 2011. Aegean Region Chamber of Industry (EBSO), the region's largest industrial corporations ranked by sales and productions are always top-ranked (Anon, 2012).

4 THE APPLICATION OF AHP MODEL FOR A COAL PREPARATION IN SOMA

The criteria and sub-criteria assessed by an expert team consisting of one mining engineer who is the head of operation department and 30 years of experience in coal industry, one mining engineer who have 20 years of experience in coal industry and one manager of production who have 30 years of experience. All decisions have a common hierarchical structure whereby options are evaluated against the various criteria that promote the ultimate decision objective. The problem of the new coal preparation plant location was structured in a hierarchy of different levels constituting goal, criteria and alternatives as shown in Figure 2.

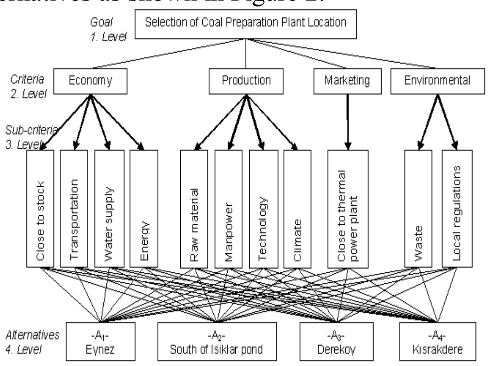


Figure 2. Hierarchy structure for coal preparation plant location

After structuring a hierarchy, the pair-wise comparison matrix for each level is constructed. During the pair-wise consideration, a nominal scale is used for the evaluation given in Table 1. As shown in Table 3, each main criterion affecting plant location selection was compared with the others and the pair-wise comparison matrix was constructed. The expert team carried out these comparisons. It is apparent that the economy criterion is the most important factor (priority 0.581) and it is followed by the production criterion (Table 3).

Table 3. Pair-wise comparison matrix of main criteria

Criteria	Economy	Production	Marketing	Environmental	W	λ_{max}	CR
Economy	1	3	7	5	0.581	4.043	0.016
Production	1/3	1	4	3	0.257	4.014	0.005
Marketing	1/7	1/4	1	2	0.094	4.023	0.009
Environmental	1/5	1/3	1/2	1	0.068	4.243	0.090
					Mean	4.081	0.030

After constructing the pair-wise comparison matrix of main criteria, all the subgroup of each main criterion should be compared with the others as shown in Table 4, 5 and 6.

Table 4. Pair-wise comparison matrix of the economy main criterion with sub-criteria of it

Economy	Close	Transportation	Water supply	Energy	W	λ_{max}	CR
Close to stock	1	2	1/5	1/3	0.112	4.097	0.036
Transportation	1/2	1	1/9	1/2	0.075	4.190	0.070
Water supply	5	9	1	2	0.569	4.028	0.011
Energy	3	2	1/2	1	0.243	4.168	0.062
					Mean	4.121	0.045

Table 5. Pair-wise comparison matrix of the production criterion with sub-criteria

Production	Raw	Manpower	Technology	Climate	W	λ_{max}	CR
Raw material	1	2	3	4	0.467	4.036	0.013
Manpower	1/2	1	2	3	0.278	4.025	0.009
Technology	1/3	1/2	1	2	0.160	4.026	0.010
Climate	1/4	1/3	1/2	1	0.095	4.037	0.014
					Mean	4.031	0.011

Table 6. Pair-wise comparison matrix of the environmental criterion with sub-criterion

Environmental	Waste	Local regulations	W	λ_{max}	CR
Waste	1	2	0.667	2.000	0.000
Local regulations	1/2	1	0.333	2.000	0.000
			Mean	2.000	0.000

The pair-wise comparison matrices are constructed by comparing the each plant location area with each sub-criterion of all main criteria. The comparison matrix for “Close to stock” sub-criterion of economy main criteria is given in Table 7 as an example, and general evaluation of economy criterion is given in Table 8. The same procedure was applied for the other main criteria and sub-criterion of them.

Table 7. Pair-wise comparison matrix of the "Close to stock" sub-criterion of Economy criterion

Land/Economy	Isiklar	Derekoy	Eynez	Kisrakdere	W	λ_{max}	CR
A_1-Eynez	1	1/2	1/3	1	0.145	4.184	0.068
A_2-Isiklar	2	1	3	2	0.423	4.363	0.135
A_3-Derekoy	3	1/3	1	2	0.270	4.323	0.120
A_4-Kisrakdere	1	1/2	1/2	1	0.161	4.061	0.022
					Mean Value	4.233	0.086

Table 8. Total priorities of Economy criterion

Alternatives	Sub-criteria of economy main criterion				Main Criterion	Total
	Stock	Transportation	Water supply	Energy	Priorities	Priorities
A_1-Eynez	0.145	0.231	0.173	0.070	0.112	0.149
A_2-Isiklar	0.423	0.475	0.291	0.519	0.075	0.375
A_3-Derekoy	0.270	0.225	0.245	0.323	0.569	0.265
A_4-Kisrakdere	0.161	0.069	0.291	0.088	0.243	0.210

The overall rating for each alternative is calculated by summing the product of the relative priority of each criterion and the alternatives considering the corresponding main criteria. For example, the overall rating of alternative "Eynez (A_1)" is calculated as; $= (0.145\times0.112) + (0.231\times0.075) + (0.173\times0.569) + (0.070\times0.243) = 0.149$. Similarly, the final matrix is constructed as shown in Table 9.

Table 9. Overall result/final matrix

Alternatives	Main criteria				Criterion	Overall
	Economy	Production	Marketing	Environmental	Priority	Result
A_1-Eynez	0.149	0.130	0.229	0.183	0.581	0.154
A_2-Isiklar	0.375	0.421	0.174	0.267	0.257	0.361
A_3-Derekoy	0.265	0.254	0.324	0.444	0.094	0.280
A_4-Kisrakdere	0.210	0.195	0.273	0.106	0.068	0.205

Since the comparison is based on the subjective evaluation, a Consistency Ratio (CR) should be calculated from Equation [3] to ensure the selection accuracy. It can be seen from Table 3 to 7, the maximum Eigen values (λmax) are near to the size of the matrix (In Table 3, the size of matrix is 4×4 and mean value of λmax is 4.081) and CR values are less than 0.1 (In Table 3, the mean CR value is 0.030), so these values are in the desired range. Therefore, the decision is taken without repeating the procedure.

Considering the overall results in Table 9, the alternative "South of Isiklar pond (A_2)" must be selected as the optimum plant location site to satisfy the goals and objectives of the ELI management because the priority of this alternative (0.361) is the highest value than that of the others.

5 SENSITIVITY ANALYSIS

Sensitivity analysis allowed us to verify the results of the decision. A sensitivity analysis can be performed to see how sensitive the alternatives will change with the importance of the criteria. As the priority of one of the criteria increases, the priorities of the remaining criteria must decrease proportionately, and the global priorities of the alternatives must be recalculated. Sensitivity analysis can also be used to determine the most important or critical criterion by computing the absolute or percentage amount by which the weight of any criterion must be changed in order to cause a switch in the ranking of either the top alternative or in any pair of alternatives (Triantaphyllou, 2000).

Increasing or decreasing the values of the Eigenvector of the main criterion in pair-wise comparison matrix simulated several scenarios. There was no change in the judgement evaluations in the final priority ranking when Eigenvector value of each criterion increased/decreased up to 35%. That is; the alternative "A_2" can always be selected and alternative "A_2", "A_3" and "A_1" considering the final priorities sequentially followed this model. Final decision found by the proposed AHP model was compared with the simulated scenarios in Figure 3.

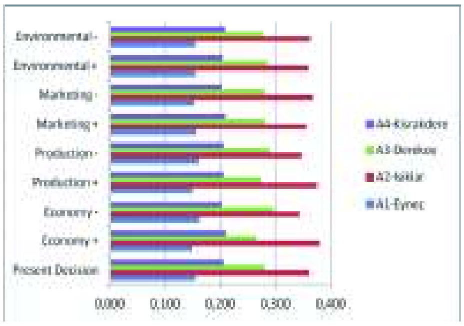


Figure 3. Sensitivity analysis results

6 CONCLUSIONS

Plant location involves the interaction of several subjective and objective factors or criteria. Decisions are often complicated and many even embody contradiction. In this study, the AHP model was developed which contains four main criteria and eleven sub-criteria. Among four alternatives under consideration, variant “South of Isiklar pond” (A_2) is the most acceptable one with allowance for all main and sub-criteria comprised in the analysis. Unlike the traditional approach to the plant location selection, AHP is more scientific method providing the integrity and objectivity of estimation process. The model is transparent and easy to comprehend and apply by the decision maker. For selection of plant location, the proposed AHP model is unique in its identification of multiple attributes, minimal data requirement and minimal time consumption.

Coal processing is one of the most important parts of the coal industry. Because of the size of the investment, it is very important to determine the optimum coal preparation plant location. The developed AHP model from this study can be used for all coal mines, and can generate an analysis of worldwide coal industry.

The AHP based decision making applications can be applied for different parts of the natural stone industry such as selection of best suitable cutting and polishing equipment, abrasives, diamond saw, etc. among the alternatives for a natural stone factory. The number of criteria and alternatives should be paid attention by the decision maker in the AHP applications because of the consistency and validity of the decision making process (Saaty and Ozdemir, 2003; Ozdemir, 2005). The number of alternatives should be 7±2, otherwise the grouping method should be applied following the same way presented in this study.

REFERENCES

Hwang, C.L. and Yoon, K., 1981. Multi Attiribute Decision Making Methods and Applications. Springer-Verlag.

Ataei, M., 2005. Multicriteria selection for alumina-cement plant location in East-Azerbaijan province of Iran. The Journal of the South African Institute of Mining and Metallurgy. 105 (7). pp. 507-514.

Kazakidis, V.N., Mayer, Z. and Scoble, M.J., 2004. Decision making using the analytic hierarchy process in mining engineering. Mining Technology (Trans. Inst. Min. Metall. A). 113. pp. A30-A42.

Karadogan, A, Bascetin, A, Kahriman, A. and Gorgun, S., 2001. A New Approach in Selection of Underground Mining Method, International Conference-Modern Management of Mine Producing, Geology and Environment Protection, Varna, pp. 171-183.

Bitarafan, M.R. and Ataei, M., 2004. Mining method selection by multiple criteria decision making tools. The Journal of the South African Institute of Mining and Metallurgy. 104, (9), pp. 493-498.

Samanta, B., Sarkar, B. and Mukherjee, S.K., 2002, Selection of opencast mining equipment by a multi-criteria decision-making process. Mining Technology (Trans. Inst. Min. Metall. A), 111, pp. A136-A142.

Ozer, S., 2005. Determination of optimum plant location for marble factories' Master of Science thesis, Eskisehir Osmangazi University, Graduate School.

Bascetin, A., 2004. An application of the analytic hierarchy process in equipment selection at Orhaneli open pit coal mine. Mining Technology (Trans. Inst. Min. Metall. A), 113, pp. A192-A199.

Anon, 2012. www.tki.gov.tr

Saaty, T.L., 1980. The Analytic Hierarchy Process, McGraw-Hill Publications.

Saaty, T.L., 2000. Fundamentals of Decision Making and Priority Theory with the Analytic Hierarchy Process. RWS Publications.

Triantaphyllou, E., 2000. Multi Criteria Decision Making Methods: A Comparative Study. Kluwer Academic Publishers.

Saaty, T.L. and Ozdemir, M.S., 2003. Why the Magic Number Seven Plus or Minus Two. Mathematical and Computer Modeling. 38, pp. 233-244.

Ozdemir, M.S., 2005. Validity and inconsistency in the analytic hierarchy process. Applied Mathematics and Computation. 161, pp. 707-720.

Determination of Optimum Coal Panel Dimensions for Fully Mechanized Underground Coal Mine in Western Lignite Colliery

I. U. Kombe
Mining Engineer, MSc., Turkish Coal Enterprises, TURKEY
H. Ankara
Associate Professor, Ph.D., Eskisehir Osmangazi University, TURKEY
M. Yavuz
Associate Professor, Ph.D.,Eskisehir Osmangazi University, TURKEY

ABSRACT The main purpose of this study is to produce maximum amount of coal with the lowest cost by determining the optimum dimensions of fully mechanized longwall panels of Western Lignite Colliery owned by Turkish Coal Enterprises in Turkey. The optimization has been done by using Operational Research techniques. A nonlinear programming model developed by Yavuz was used in this paper. The parameters that have great impact on fully mechanized longwall panel are determined and categorized according to original model. Then, a decision model was generated by using these economic and engineering parameters. This model was solved by using software called as LINGO and the results were analyzed. After the analysis of the results, sensitivity analysis was performed to determine which parameters have great effect on the results. According to data from the WLC "Omerler A" sector, optimum face length was found as 129 m. According to the model solution and sensitivity analysis results, it can be concluded that cost per ton of coal produced can be lowered or how a change in one of the any model parameter affect the cost per ton of coal produced can be easily monitored.

INTRODUCTION To meet mankind's increasing demand for raw material, today, mining has to produce more and more minerals at a suitable cost. This can best be achieved by mechanization of certain steps of mining. Mechanization allows miners both to increase the production capacity and to decrease the mining costs. Mechanization can be described as the future of mining sector. Especially, it is extremely important for both open pit and underground coal mining industry. In underground coal mines, Longwall mining method is the most preferred underground production method in coal mining industry.

Like every underground mining method, longwall mining has also its special conditions, limits and problems. These problems are related to rock mechanics, safety of operation, ventilation, transportation, output capacity and mechanization possibilities. All these factors are strongly combined with coal panel dimension that is panel length and width. Whereas the panel length is chosen in most cases depending upon geological conditions, Optimum face length is bed-specific. This means that an optimum face length determined for a certain colliery is not valid for another region, colliery or country, and data collected to calculate this length for a certain case cannot be generalized.

In this study, producing maximum amount of coal with the lowest cost by determining the optimum dimensions of fully mechanized longwall panels of Western Lignite Colliery (WLC) owned by Turkish Coal Enterprises in Turkey was analyzed.

1 WESTERN LIGNITE COLLIERY

WLC is located in the western part of Turkey in Tunçbilek as shown in Figure 1 and an important lignite producing company with an annual production rate of 6 million tons from several open pits and two underground mines. The calorific value of the coal is about 4000 kcal/kg in the district. Lignite production began in 1938 and has currently been consumed for electricity at the power plant and for domestic usage in several industries. Due to the increasing demand, the company plans on redounding the production rate (Taskin, 1999). However, it is predicted that open pit reserves will come to an end in about 10 year's period. Therefore, the mine management has also planned an annual production rate of 6 million tons from the deep coal seam reserves by using the fully mechanized longwall panels. Coal reserves of Tuncbilek district are given in the Table 1, below.

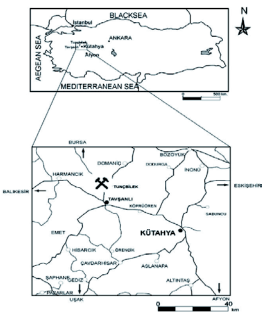


Figure 1. Location map of WLC region (Tuncbilek)

Table 1. Coal reserves of Tuncbilek district (Destanoglu et al., 2000)

	Proven Reserve (Ton)	Prepared (Ton)	Total (Ton)
Open Pit	37.301.000	1.011.000	38.312.000
Underground	235.107.000	3.176.000	238.283.000
Total	272.408.000	4.187.000	276.595.000

Mechanization applications in WLC have been continuing since 1984. The first application was unsuccessful due to the selected wrong method. The second application began in 1996 and is still in practice in "Omerler A" sector. In this study, an optimization study was conducted for "Omerler A" sector. As seen in Figure 2, six panels were planned for extraction by means of fully mechanized face in sector A. At the time of this study, two adjacent longwall panels, namely M1 and M2, had been completed and the production was carried out at the M8 panel. Coal has been produced by means of longwall retreat with the top-coal-caving production method where a 2.8m high longwall face was operated at the floor of the coal seam (Figure 3). Top-slice coal having a thickness of 5.2m was caved and produced through windows located at the top of the shields (Unver and Yasitli, 2005).

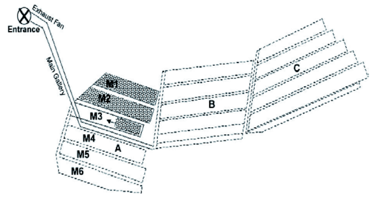


Figure 2. A simplified plan view of Omerler underground Mine (Destanoglu et al., 2000)

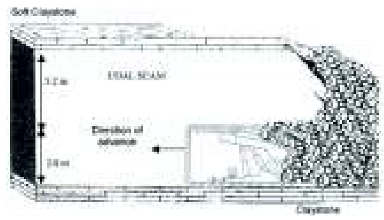


Figure 3. Longwall with top-coal-caving method as applied at Omerler underground mine

2 A NONLINEAR PROGRAMMING MODEL

A nonlinear programming model developed by Yavuz (2002) given in below. The parameters that have great impact on fully mechanized longwall panel are determined and categorized according to original model. Decision model was generated by using economic and engineering parameters. This model was solved by using software called as LINGO and the results were analyzed.

$$Min \frac{\sum_{i=1}^{4}\sum_{j=1}^{n} M_{k,i,l}}{\sum U_{i}} + \sum_{k=1}^{2}\sum_{l=1}^{6} M_{u,k,l} + \sum_{k=1}^{2}\sum_{m=1}^{2} M_{t,k,m} \text{ ....}$$

Subject to

$$Ge_{ZO}\times Y_{ZOT}\times(H_{ZO}\pm H_{KYK})\times Z_{AT}\times Rd_{Y}\geq 2\times U_{A}\times K_{D}\times D_{K} \quad \text{.......2}$$

$$Gu_{TZO}\geq\frac{H_{ZO}}{76,1\times Rd_{MK}}(Ag_{ZP}\times 2\times U_{A}\times Gu_{UKI}\times k_{CCDS}+k_{KCDS}\times Ag_{K}\times U_{ZOKY}) \quad \text{.......3}$$

$$\frac{Dy_{ZK}\times S_{Y}}{2,1175}\geq\left\{\begin{matrix}\left[(Ag_{K}\times k_{KCS}+Ag_{ZP}\times k_{CCS})\times U_{A}\times Cos\alpha\pm(Ag_{K}+Ag_{ZP})\times U_{A}+Sin\alpha\right]\\+\left[Ag_{ZP}\times U_{A}\times(k_{KCS}\times Cos\alpha+Sin\alpha)\right]\end{matrix}\right\} \quad \text{.......4}$$

$$H_{AMH}\times 857\times Ac_{UA}\times K_{D}\geq U_{A}\times Em_{CH_4}\times I_{V}\times Y_{K}\times\gamma_{k} \quad \text{.......5}$$

$$\left(2\times\frac{U_{P}}{I_{TG}}+\frac{U_{A}}{I_{KG}}\right)\times Z_{GC}\times 60\geq\frac{Z_{T}\times(U_{P}-Ge_{ET})}{D_{K}} \quad \text{.......6}$$

$$U_{P}\leq U_{JP} \quad \text{.......7}$$

$$U_{P}\geq 0 \quad \text{.......8}$$

$$U_{A}\geq 0 \quad \text{.......9}$$

In this model, the objective function is given by Equation 1. The first part of the objective function is contains development costs, the second part is contains longwall costs and the last part is contains face move costs. The goal of this model is to minimize total production costs per ton. The first constraint is capacity of Armored Face Conveyor (AFC) and given in Equation 2. The second constraint is engine power of AFC (Equation 3). The third constraint is tensile strength of AFC given in Equation 4 for double chain in the middle. The other one is ventilation constraint and given in Equation 5. In fully mechanized longwall panels, development activities must be compatible with production activities. So, Equation 6 is development constraint. The seventh constraint is geological and legal constraint and given in Equation 7. Last two constraints are being positive constraints for panel and face length (Equation 8 and 9).

3 DETERMINATION OF OPTIMUM DIMENSIONS FOR WLC

3.1 The Parameters Using in the Model

The parameters using in the nonlinear programming model are given in Table 2, below. In this model, three shifts per day working period were accepted in the mine. Production time is taken as 6 hours per day and effective rate for daily time is taken 80%.

Table 2. The parameters using in the nonlinear programming model (Taskin, 2003)

Symbol	Explanation	Value
K_D	Thickness of coal seam (m)	8
D_K	Excavation depth of shearer (m)	0,6
$?_K$	Density of coal (kg/m³)	1440
H_{KYK}	Cutting speed of shearer (m/sec.)	0,94
Ge_{ZO}	Width of the AFC (mm)	730
Y_{ZOT}	Height of protecting material of AFC (mm)	222
Y_K	Height of excavation (m)	3,2
U_{KYM}	Movement length of shearer (m)	20
U_{KYTA}	Length of between two drum of shearer (m)	5,4
H_{KYT}	Cleaning speed of shearer (m/min.)	12
Z_{TD}	Time of position changing of drum (min.)	1
Ag_{ZP}	Weight of chain and palette of AFC (kg/m)	68,55
Ko_E	Effective used rate (%)	80
k_{CCS}	Coefficient of friction between steel and steel	0,3
k_{CCDS}	Coefficient of static friction between steel and steel	0,5
Gu_{UKI}	Additional power which is required for curves in longwall (%)	3
Rd_{MK}	Mechanic efficiency for AFC (%)	75
Rd_Y	Loading efficiency for AFC (%)	85
Yz_{KZ}	Acquired rate of caving coal (%)	95
k_{KCS}	Coefficient of friction between coal and steel	0,5
k_{KCDS}	Coefficient of static friction between coal and steel	0,6
α	Inclination of longwall (°)	6
Dy_{ZK}	Strength of break of chain (kN)	850
S_T	Loading constant of AFC	0,85
H_{AMH}	Max air speed in longwall (m/sn)	0,5
?	Constant of ventilation	0,95
Mk_{HDM}	Legal max CH_4 quantity in air circulation (%)	1
Em_{CH4}	Emission of CH_4 (m³/ton)	0,4
S_V	Efficient constant for ventilation (%)	75
I_{VU}	Advance per shift of longwall (m)	1,8
Ge_{ET}	Thickness of pillar (m)	15
G_Y	Annual average working days (day/year)	250
M_P	Cost of personal ($/wage)	58-63
M_E	Energy cost (kw/hour)	0,071
M_{KU}	Cost of cutter bit ($/number)	11,53
S_{VK}	Number of cutter bit in per day (number/day)	3
I_G	Average Advance of preparation for longwall in per day (m)	3
A_{ANY}	Sectional area of main road (m²)	14
$?_{BHV}$	Density of compressed air (kg/m³)	3,24
U_{ZOKY}	The length of loaded AFC (m)	$0,9xU_A$
Ac_{UA}	Space of longwall (m)	3
Z_{VC}	Working time per shift (hour)	6

Symbol	Explanation	Value
A_{ANY}	Sectional area of main road (m²)	14
$?_{BHV}$	Density of compressed air (kg/m³)	3,24
U_{ZOKY}	The length of loaded AFC (m)	$0,9xU_A$
Ac_{UA}	Space of longwall (m)	3
Z_{VC}	Working time per shift (hour)	6

The following calculations were made to determine some of the parameters contained in the objective function and constraints (Taskin, 2003).

1. Calculation of longwall production:

$$Ur_{UP}=\left[Y_{K}+(K_{D}-Y_{K})\times Yz_{KZ}\right]\times\left[U_{A}\times(U_{P}-Ge_{ET})\times\gamma_{K}\right] \quad \text{.......10}$$

$$Ur_{UP}=\left[3,2+(8-3,2)\times 0,95\right]\times\left[U_{A}\times(750-15)\times 1,44\right]$$

$$Ur_{UP}=8213,184\times U_{A}\ \text{tone.}$$

2. Calculation of the cycle time according to face end method.

$$Z_{AT}=\left[\frac{2\times U_{KYM}}{H_{KYK}}+\frac{U_{KYM}}{H_{KYT}}+\frac{U_{A}+U_{KYM}+U_{KYTA}}{H_{KYK}}+3\times Z_{TD}\right]\times\frac{1}{Ko_{E}} \quad \text{.......11}$$

$$Z_{AT}=\left[\frac{2\times 20}{2,7}+\frac{20}{12}+\frac{U_{A}+20+5,4}{2,7}+3\times 1\right]\times\frac{1}{0,7}$$

$$Z_{AT}=41,27+0,53\times U_{A}\ \text{min.}$$

3. Calculation of daily coal production

$$Ur_{GU}=\frac{Z_{VC}\times Ko_{E}\times 60}{Z_{T}}\times U_{A}\times D_{K}\times\gamma_{K}\times\left[(Y_{K}+(K_{D}-Y_{K})\times Yz_{KZ})\right] \quad \text{.......12}$$

$$Ur_{GU}=\frac{6\times 0,80\times 60}{41,28+0,53\times U_{A}}\times U_{A}\times 0,6\times 1,44\times\left[(3,2+(8-3,2)\times 0,95)\right]$$

4. Calculating the weight of the coal on the AFC

$$Ag_K = \frac{Ge_{ZO} \times Y_{ZOT}}{2} \times \gamma_K \times Rd_y \quad \text{.......... 13}$$

$$Ag_K = \frac{0,730 \times 0,222}{2} \times 1440 \times 0,85 = 99,18 \text{ kg/m.}$$

5. Determination of total time for face move

According to data from WLC, face move time was chosen as 53 days.

3.2 The Objective Function of the Model

In this study, all the costs of fully mechanized longwall were examined and divided into three main groups. These are; development costs, longwall costs and face move cost. These three main costs were divided into subgroups as investment and operating costs (Taskin, 2003).

Development costs are calculated as:

$$M_k = \frac{0,517 \times UA + 0,000044 \times U_A^2 + 363,1}{U_A} \text{ \$/tone.}$$

Longwall costs are calculated as:

$$M_u = \frac{217,91 + 0,48 \times U_A + 0,000258 \times U_A^2}{U_A} \text{ \$/tone.}$$

Face move costs are calculated as:

$$M_t = \frac{1,58 + 0,00201 \times U_A}{U_A} + \frac{3,094}{U_A} = \frac{4,67 + 0,00203 \times U_A}{U_A} \text{ \$/tone.}$$

Total production costs per ton are calculated as:

$$?? ??? \; \frac{0,9997 \times ??_{??} + 0,000302 \times ??_{??}^2 + 585,6787}{??_{??}} \text{ \$/?????}$$

3.3 The Constraints of the Model

By using both the parameters in the nonlinear programming model given in Table 1 and Longwall production, cycle time, daily coal production, weight of the coal on the AFC and total time for face move, the constraints in the model were calculated in the below. These are (Taskin, 2003);

1. Capacity of AFC constraint;

$$0,73 \times 0,222 \times (56,4 + 2,7) \times (41,27 + 0,529 \times U_A) \times 0,85\,'\,2 \times U_A \times [(8 - 3,2) \times 0.95] \times 0,6$$

2. Engine power of AFC constraint;

$$2 \times 132 \geq \frac{0,94}{76,1 \times 0,75}(68,55 \times 2 \times U_A \times 1,03 \times 0,5 + 0,6 \times 99,18 \times 0,9 \times U_A)$$

3. Breaking strength of AFC constraint;

$$\frac{850000 \times 0,85}{2,1175}, \left\{ \begin{array}{l} [(99,18 \times 0,5 + 68,55 \times 0,3) \times U_A \times Cos6 \pm (99,18 + 68,55) \times U_A \times Sin6] \\ + [68,55 \times U_A \times (0,5 \times Cos6 + Sin6)] \end{array} \right\}$$

4. Ventilation constraint;

$$0,5 \times 857 \times 3 \times 3,2 \geq U_A \times 0,4 \times 1,8 \times 3,2 \times 1,44$$

5. Development constraint;

$$\left(2 \times \frac{750}{3} + \frac{U_A}{3}\right) 4,8 \times 60 \geq \frac{(41,27 + 0,529 \times U_A) \times (750 - 15)}{0,6}$$

6. Geological and legal constraint;

$$U_A \leq 400$$

7. Being positive constraints for face length

$$U_A \geq 0$$

3.4 Nonlinear Decision Model for "Omerler A" Sector

By using nonlinear programming model developed by Yavuz, the calculated objective function and constraints were simplified. The final version of the model is given below (Taskin, 2003).

$$\text{Min} \; \frac{0,9997 \times ??_{??} + 0,000302 \times ??_{??}^2 + 585,6787}{??_{??}}$$

Subject to

$$0,73 \times 0,222 \times (56,4 + 2,7) \times (41,27 + 0,529 \times U_A) \times 0,85\,'\,2 \times U_A \times [(8 - 3,2) \times 0.95] \times 0,6;$$

$$2 \times 132 \geq \frac{0,94}{76,1 \times 0,75}(68,55 \times 2 \times U_A \times 1,03 \times 0,5 + 0,6 \times 99,18 \times 0,9 \times U_A);$$

$$\frac{850000 \times 0,85}{2,1175}, \left\{ \begin{array}{l} [(99,18 \times 0,5 + 68,55 \times 0,3) \times U_A \times Cos6 \pm (99,18 + 68,55) \times U_A \times Sin6] \\ + [68,55 \times U_A \times (0,5 \times Cos6 + Sin6)] \end{array} \right\};$$

$$0,5 \times 857 \times 3 \times 3,2 \geq U_A \times 0,4 \times 1,8 \times 3,2 \times 1,44;$$

$$\left(2 \times \frac{750}{3} + \frac{U_A}{3}\right) 4,8 \times 60 \geq \frac{(41,27 + 0,529 \times U_A) \times (750 - 15)}{0,6};$$

$$U_A \leq 400;$$

$$U_A \geq 0;$$

3.5 Solution of the Model and Sensitivity Analysis

Non-linear programming model was solved by using software called as LINGO and the results were analyzed. In this study, two different models were solved. In the first model, the decision model was solved and optimum face length and minimum coal cost per ton were found to be 129 m and 5.58 $, respectively. In the second model, 90 m face length which applied in the mine was entered in the model and coal cost per ton was calculated in this case. The coal cost per tone was found as 7.53 $.

After the analysis of the results, sensitivity analysis was performed to determine which parameters have great effect on the results. After the sensitive analysis, coal cost per ton was found sensitive to engine power of AFC constraint. In the other sensitive analysis, coal cost per ton was found sensitive to development constraint and capacity of AFC constraint, respectively.

CONCLUSION

In this study, an optimization study was conducted to determine optimum dimensions of fully mechanized underground coal panels for Omerler sector in WLC. A nonlinear model proposed by Yavuz (2002) was used in this study for minimization for cost of total production. Application of this decision model was carried out to find the optimum face length of WLC Omerler fully mechanized underground coal panels. The length and the number of the panel variables were not taken into account in decision model. Optimum face length was investigated for 750 m coal panel length which is applied in the "Omerler A" sector. A cost function was obtained according to longwall length (UA) by using all cost functions in the sector.

According to data from the WLC "Omerler A" sector, optimum face length was found as 129 m. The cost per ton of coal for 129 m face length was calculated to be $ 5.58. For the 90 m face length which is still being applied in the WLC, the cost of coal per ton was calculated to be $ 7.53. This cost is in agreement with WLC data. The reserve of analyzed coal panel’s was approximately 1 million tons, so the economic value of the difference between costs of $ 1.95 per ton can be better understood.

An optimization study should be applied for constructing new coal panels in the WLC. Examination of constraint equations of the model and improvement of the factors restricting the face length, better production values could be obtained from the fully mechanized coal panels in WLC.

REFERENCES

Destanoglu N, Taskin F.B, Tastepe M., Ogretmen S., 2000. Application of GLI Tuncbilek–Omerler underground mechanization. Ankara: Kozan Publishing, Ankara. p. 211 (in Turkish).

Unver, B. and Yasitli, E., 2005. 3D numerical modeling of longwall mining with top-coal caving. International Journal of Rock Mechanics & Mining Sciences. 42. 219–235.

Taskin, F.B., 1999. Optimum dimensioning of the abutment pillars applied in the longwall panels of GLI Tuncbilek underground mine. Ph.D. Thesis. Osmangazi University, Eskisehir/Turkey (in Turkish).

Taskin, I.U., 2003, Determination of Optimum Length of Mechanized Longwall Face in Omerler Coal Colliery, M.Sc. Thesis, Osmangazi University, Eskisehir/Turkey (in Turkish).

Yavuz, M., 2002, Determination of Fully Mechanized Underground Coal Panels’ Optimum Dimensions. Ph.D. Thesis. Osmangazi University, Eskisehir/Turkey (in Turkish).