Investigation on Fires in Coal Mines and Approach for Prevention – Case-Study

A THESIS SUBMITTED IN PARTIAL FULFILLMENT OF THE REQUIREMENTS FOR
THE DEGREE OF

BACHELOR OF TECHNOLOGY
IN
MINING ENGINEERING

By

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CERTIFICATE

This is to certify that the thesis entitled **"Investigation on Fires in Coal Mines and Approach for Prevention – Case-Study"** submitted by **Sri Siddhartha Basudeb Parashar and Sri Sunil Kumar Bisoyi** in partial fulfillment of the requirements for the award of Bachelor of Technology degree in Mining Engineering at National Institute of Technology, Rourkela is an authentic work carried out by them under my supervision and guidance.

To the best of my knowledge, the matter embodied in the thesis has not been submitted to any other University/Institute for the award of any Degree or Diploma.

Date:

Prof. B. K. Pal

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Last but not least, our sincere thanks to our friends who have patiently extended their hands for help in accomplishing this undertaking.

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ABSTRACT

Coal absorbs oxygen even at lower temperatures than the ambient temperature and they start heating up. This heating without any external source of supply of heat generally leads to due to auto-oxidation, also called spontaneous heating of coal. There are numerous reasons for which generally this occurs and they are broadly classified as mining, seam and geological factors. The sub-classification of these factors is so intimately associated that exact reason is still unknown. However, this phenomenon is the cause of a numerous accidents and disasters in mining industry, not only in India but is the problems all over the world. Coal is considered to be the prime fuel of our country. The major portion of our power supply relies on coal. The power generated from thermal power station is the source of energy for many other activities/industries. The production and productivity further depend on the supply of energy generated from the burning of coal in the thermal power station. Due to auto-oxidation, we are faced with many dangerous problems. Therefore the assessment of this problem has become necessary and hence we are trying to calculate and predict how susceptible or prone any coal might be towards spontaneous heating before its use in any industry.

India is having a huge amount of coal resources spread all over the country and most of the coals are liable to auto-oxidation. Hence coal samples are collected from different subsidiaries of Indian Coal sectors, viz. Central Coalfields Limited (CCL), Mahanadi Coalfields Limited (MCL), Western Coalfield Limited (WCL) and Digwarhi Mines (TATA Steel) in order to assess the situation and to conclude the result found out of the critical analyses. In the laboratory, different analytical techniques, viz. proximate analysis and Crossing-Point Temperature study of the samples were carried out and their corresponding relationships are established. This study helped us to predict the liability of a coal sample towards spontaneous heating. These results are compared with the observations of the earlier investigators and it is observed that there are similarities with our findings. Various spontaneous heating cases were observed. As a Mining Engineer some steps towards the prevention were noted from the observations of earlier investigators and some suggestions are made from our findings which may reduce the possibilities of tendencies towards auto-oxidation and also to find some suitable way to minimize the threat caused due to this spontaneous heating.

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CHAPTER – 1

INTRODUCTION

1.1 Introduction:

Carbonaceous Organic Aromatic Lump which generally known as COAL is a formation due to the decaying of vegetable matters (Drift or Allochthonous theory) or submerging forest due to tectonic movements (In-situ or autochthonous theory) with carbon percentage in a wide range. The classification according to their carbon percentage would be Graphite (100%), Anthracite (91%), Bituminous (78.5%), Semi-bituminous (69.5%), Sub-bituminous (64.5%), Lignite (61.8%) and Peat (50.9%).

Spontaneous heating is auto-oxidation at coal sometimes leads to fire restricts the mining operation. This phenomenon was first observed by plot in 1686.

1.2 Definition of spontaneous heating:

Coal absorbs oxygen even at lower temperatures than the ambient temperature and they start heating up. First, the heating causes the water or moisture to evaporate and release from the coal, after which the rate of oxidation increases. But, every heating doesn't lead to ignition because of dissipation of heat during this process, when the dissipation of heat is greater than the accumulation then it cannot reach to "critical temperature of oxidation" and is generally called as weathering. But, when accumulation is greater than the oxidation it leads to fire or ignition.

The critical temperature of oxidation depends upon the mineral content, temperature, pressure, geographic location etc., few examples would be:

Indian coals -70-85°c

French coals – 80-85°c

US/Russian coals – 60-70°c

1.3 History of Spontaneous Heating in India:

Table 1.1 Trend of accidents due to spontaneous heating in India

Year	U/G working	Surface working	O/C working
1980	18	4	0
1981	8	3	1
1982	8	1	2
1983	19	5	2
1984	13	1	2
1985	19	1	3
1986	19	1	2
1987	19	0	2
1988	12	0	1
1989	9	3	2
1990	14	5	0
1991	11	10	2
1992	14	22	0
1993	9	11	3
1994	5	22	6
1995	10	26	2
1996	12	10	3
1997	8	19	7
1998	10	9	2
1999	10	14	4
2000	11	9	2
2001	10	14	1
2002	8	9	4
2003	6	13	1
2004	3	11	4
2005	6	16	3
2006	11	17	1
2007	11	20	1
2008	14	0	0

In Indian coal mines, 80% of the mine fires occur due to spontaneous combustion. The main aspect of a start of fire in India is because the coal seams are thicker and there is a tendency of spontaneous heating during the depillaring operation. The problem of extraction of thick seam and coal standing in pillars becomes a serious threat when coal is of high moisture, high volatile matter and low ash content which is more liable to spontaneous combustion. It is impractical to be able to extract all the coal by caving method. Pillars standing for long time are liable to deteriorate and spilling may occur. If coal is heated to about 93°C and O2 in the air is available, the oxidation will take place at an accelerated rate and if the heat of dissipation is less than the heat of accumulation then coal bed temperature increases, ultimately leading to fire.

1.4 Objective:

- The objective of this project is to find out the susceptibility of coal towards spontaneous combustion by proximate analysis and Crossing-Point Temperature with the help of a few samples collected.

- To observe the occurrences of spontaneous heating and the preventive measures that was taken to avoid the same.

CHAPTER-2

LITERATURE REVIEW

2.1 Mechanism of spontaneous heating:

The main mechanism of spontaneous heating is absorbing the oxygen from atmosphere and decomposing into various carbon and oxygen complexes. This process generally liberates heat. As coal is very diverse in its chemical composition, mineral matter etc. the procedure of decomposition becomes very complex during the oxidation. There will be so many concurrent and overlapping reactions taking place that it would be very difficult to separate out the reactions.

Coal first adsorbs the gaseous or liquid so as to oxidize which is otherwise known as sorption. When the accumulation of other solution restricts further penetration into the body, it is only known as adsorption. But, when the solution can enter the bulk it is called absorption.

Coal when absorbs oxygen it heats up because oxidation of coal in presence of air is exothermic. Up to $50^{o}c$ the oxidation is very slow, when it reaches up to $80^{o}c$ the oxidation rate is increased and when reaches near $120^{o}c$, the water evaporates and moisture of the coal gets released. Here the oxidation is faster and steady. The evolution of oxidation of carbon starts when the temperature is from 120 to $180^{o}c$. At this temperature up to $220^{o}c$, thermal decomposition of coal starts. When the temperature increases from $220^{o}c$ to $275^{o}c$ coal starts self-sustained process of combustion. When the temperature rises from $275^{o}c$, coal starts active combustion which sometimes can lead to ignition.

Fig 2.1 Stages of Spontaneous Combustion (Pal B. K., 2012)
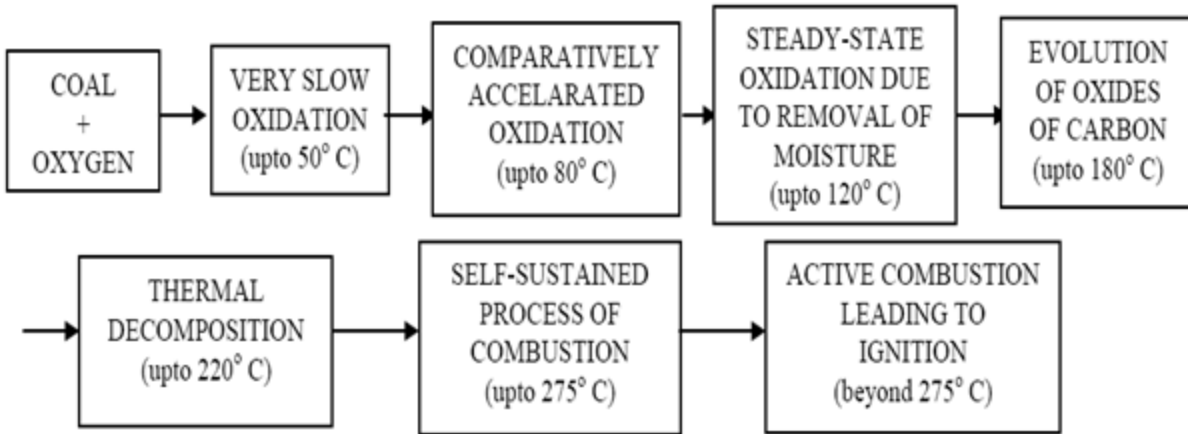
There are various theories of spontaneous heating because of the complex reaction occurring due to the diverse properties of coal.

2.1.1 Pyrite theory:

When there is pyrite content in coal and heating of coal starts then, the pyrite forms a complex acid solution of greater volume, this causes cracks and fissure development, although the effect is negligible when pyrite content is less than 5%. It follows the reactions written under in general:

$$2FeS_2 + 7O_2 + 16H_2O = 2H_2SO_4 + 2FeSO_4 + 2FeSO_4.7H_2O + 31Kcal$$

$$FeS_2 + 15/4O_2 + 5/2H_2O = FeO.OH + 2H_2SO_4$$
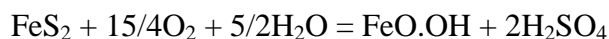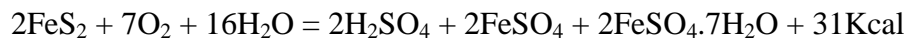
2.1.2 Bacterium theory:

Potter observed that there was a slight increase in heat build-up when bacteria were introduced.

Graham observed that the heat build-up in sterilized and unsterilized coal both were at the same rate. Therefore there is no certain proof of this theory being true.

2.1.3 Humidity theory:

Stott saw that the atmospheric oxidation of coal liberated less heat than the required heat for evaporating moisture from coal. Therefore the temperature of heating can be decreased by inducing evaporation of water.
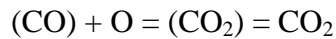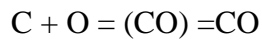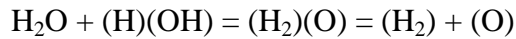
Mukherjee and Lahiri observed that the source of CO and CO_2 could be resulted due to the following at low temperatures.

$H_2O + (H)(OH) = (H_2)(O) = (H_2) + (O)$

$C + O = (CO) = CO$

$(CO) + O = (CO_2) = CO_2$

2.1.4 Coal-Oxygen Complex Theory:

Oxidation of coal is believed to be initiated at native radical sights. Formation of Peroxyl radicals and hydro peroxides is commonly to be thought to be they mechanism by which oxygen and moisture are initially in corporate into organic matrix. These species may react, rearrange or decomposed to form wide range of oxygen functionality in the matrix or gaseous product.

Kroger and Beier suggested that the coal oxygen interaction goes through the intermediate formation of peroxy complex. The progress of chain reaction was explained through the intermediate peroxy radical with water giving rise to OOH and OH radicals.

2.2 Prevention and Control

A number of methods and techniques have been developed throughout the world to extinguish coal fires and prevent their (re-)ignition. Though some of these are very efficient in dealing with the problem, they only apply to specific geo-mining conditions. Their performance rate also varies from case to case. The methods currently available may be summarized as follows:

Trench cutting and filling	Hydro-pneumatic/fly-ash stowing
Blanketing	Digging out
Sealing through isolation stopping	Application of sealant/surface coating material
Hard foam for isolation of fire area	Injection of cement/hydrogel in coal mass

Application of water mist   Dynamic pressure balancing

Chamber ventilation   Multi-zonal ventilation systems

                                                Application of fire-fighting chemicals

Inertisation processes:

- Liquid/gaseous nitrogen,

- High-pressure high-stability foam,

- Carbon dioxide

- halons, steam

- flue gas

Some of the few examples of successful implementation of some of the above methods in Indian coalmines are given in Table.

Table 2.1 Successful examples of coal fire-fighting in Indian coalmines

Name of Mine	Year	Working Method	Method Developed & Applied by CMRI for Fire Control
Kurasia Colliery	1960	Coal stack fire	Application of water and thermal monitoring
Kurasia Colliery	1961		Reopened sealed of fire area by monitoring gas analysis from fire areas
H Colliery	1966	-	Pressure balancing, Water infusion through boreholes, Surface was sealed by silicate and cement injection
R Colliery	1967-68		Sand blanketing
N Colliery	1967-68		Pressure balancing
K Colliery	1971-72		Water flooding

Name of Mine	Year	Working Method	Method Developed & Applied by CMRI for Fire Control
Jhingurda Colliery	1971-72	Opencast bench	Application of emulsion of tar & burnt mobile
Laikdih Deep Colliery	1971-72	Board & Pillar	Infusion of inert gas CO_2 and N_2, Cement grouting, Water infusion
Simlabahal colliery	1971-72	Board & Pillar	Reversal of fan, CO_2 infusion and Bentonite application
Sudamdih Colliery	1977-80	Board & Pillar	Application of CO_2 in combating mine fire
Bhulan Bararee Colliery	1980-81	Board & Pillar	Application of CO_2 in combating mine fire
Shankarpur Colliery	1980-81	Board & Pillar	
South Tisra Colliery	1980-81	Board & Pillar	Infusion of CO_2, Cement and Sodium silicate infusion
Bhojudih washery	1984-85	Coal stack	Using fly ash slurry with suitable fire retardant chemicals
Kenda Colliery	1984-85	Board & Pillar	Sand seals at regular interval
Lodna Colliery		Board & Pillar	Computer simulation study of pressure balancing arrangement and high-pressure High stability nitrogen foam injection
Madhujore colliery	1988-89	Board & Pillar	Pressure balancing to prevent mine fire

Name of Mine	Year	Working Method	Method Developed & Applied by CMRI for Fire Control
Jitpur Colliery	1988-89	Board & Pillar	Pressure balancing to prevent mine fire, infusion of silicic acid and Slurry of fly ash mixed with bentolite and diammonium phosphate
Jaganath Mine	1988-89	Opencast	Application of latex asphalt emulsion and Infusion of silicic acid
Dakra Open Cast	1988-89	Opencast	infuse silicic acid gel apply fire resistant bitumen based protective coating
Peal Kajora colliery	1990-91	Board & Pillar	Application of liquid nitrogen
west Bokaro Colliery	1990-91	Coal stack fire	Infusion slurry (water) solid:: W:1) added with sodium silicate and diamonium phosphate.
Balkudra Seam at Sayal "D" colliery	1990-91	Board & Pillar	gel infusion and other ancillary measures like reducing the fan pressure and strengthening the stoppings
Chirimiri colliery	1990	Board & Pillar	co grout mixture of cement and sand in a ratio of 1:1.5 was passed through the holes with requisite quantity of water

Name of Mine	Year	Working Method	Method Developed & Applied by CMRI for Fire Control
Peal Kajora Unit at Naba Kajora colliery	1990	Board & Pillar	liquid nitrogen was flushed through stopping, dynamic balancing of pressure
Sayal'D Colliery, CCL	1992	Board & Pillar	Dynamic balancing of pressure
Churcha East, SECL	1992	Board & Pillar	Dynamic balancing of pressure
Naba Kajora Colliery, ECL	1993	Board & Pillar	Dynamic balancing of pressure
North Searsole Colliery	1993	Board & Pillar	Dynamic balancing of pressure and injection of hydrogel
Sijua Colliery, TISCO	1994	Board & Pillar	Dynamic balancing of pressure and injection of nitrogen
New Kenda Colliery	1994	Board & Pillar	dynamic balancing of pressure
Chirimiri Colliery		Board & Pillar	remote filling method by pouring cement concrete through boreholes
South Tisra	1994		cooling of strata by providing water pools on the surface and re-circulation of chilled goaf gases
Ningah Colliery, ECL	1995	Board & Pillar	Dynamic balancing of pressure
Hingiram Pur Colliery	1997	Board & Pillar	Dynamic balancing of pressure

Name of Mine	Year	Working Method	Method Developed & Applied by CMRI for Fire Control
Kottadih Colliery, ECL	1997	Longwall	Dynamic balancing of pressure and injection of nitrogen
Jhingurdah, NCL	1998	Opencast	Application of sealant
Kachhi Balihari Colliery, BCCL	2000	Bord & Pillar	Dynamic balancing of pressure
Jhanjra Colliery, ECL	2000	Longwall	Chamber ventilation and injection of high-pressure high-stability nitrogen foam
VK-7 Incline (BG-1) panelMine, SCCL	2001	Blasting Gallery	Sealant; pressure control; injection of foam, N_2, and CO_2
Sudamdih Shaft Mine	2001	Horizon Mining	Dynamic balancing of pressure and application of foam
GDK-10 Incline Mine, SCCL	2002	Blasting Gallery	Sealant; pressure control; injection of foam, N_2, and CO_2
Sendra Bansjora, BCCL	2003	Board & Pillar	Dynamic Balancing of pressure and application of foam
Haripur, ECL	2003	Board & Pillar	Application of CO_2, detection of leakage paths, repair of stoppings
Lodna Colliery, BCCL	2004	Fire in Shaft Pillar	Dynamic balancing of pressure, injection of foam, stowing and strengthening of pillar
Shatabdi, BCCL	2004	Opencast	Application of chemicals
West Bokaro, TISCO	2005	Opencast	Application of hydrogel

Name of Mine	Year	Working Method	Method Developed & Applied by CMRI for Fire Control
Shatabdi OCP BCCL	2006	Opencast	Application of chemicals
Giddhi Quarry, CCL	2006	OB dump	Application of chemicals
Anjan Hill Mine, Chirimiri	2007	Blasting Gallery	Sealant; injection of CO_2
Shatabdi OCP	2008	Opencast	Application of chemicals
NCPH Colliery of Chirimiri Area	2009	Bord and Pillar	Sealant; injection of CO_2

2.2.1 Inertisation

Inert gases were used to fight mine fires during latter half of the nineteenth century. Subsequently, flue gases, carbon dioxide and nitrogen were used for combating mine fire with varied degree of successful attempts. Probably, the earliest report to extinguish mine fires using inert gases was in the year 1850, at the Clackmannan Mine, Scotland (Adamus, 2001, Amartin 2001). In India carbon dioxide was used for the first time in the year 1977 to smoother an underground fire at Sudamdih Shaft Mine, BCCL and first trial of nitrogen was done to control an old fire in Laikdih Deep Colliery, BCCL in March 1981 (Tripathi and Singh, 2002; Mohalik et. al. 2006).

2.2.2 Different Inertisation Methods

Application of inert gases in cutting of the source of oxygen and taking away heat from the reaction interfaces has gained importance in combating coal fires recently. The technology, efficacy and economics of operations, however, determine the applicability of the method in large scale fire combating practices. Application of suitable inert gases permits rapid sealing of fire zones and significantly reduces the hazard of gas explosion. However, the practice is generally less effective and more costly than conventional direct fire fighting methods. The technology of inert infusion for fire combat activities in underground mines may be divided broadly in four categories as follows:

- Replacement of air with inert gas in goaf vulnerable to heating leading to prevention of gas explosions by modifying the atmosphere;
- Replacement of air before sealing a district where heating has been detected by reducing the intensity or spread of secondary combustion and to cool the area surrounding the fire zone;
- Control and extinction of fire in sealed off area by inert gas infusion with pressure chambers for safety and recovery;
- Protecting shaft pillars and ribs adjoining main and haulage road from advancing fire by putting up an inert gas plug.

The main difficulty with gas injection methods has been the limited availability of plant to provide large volumes of gas on site at short notice. Moreover such methods are costly. Inert gases which have been employed with varied degrees of success are halons, steam, CO_2, N_2 and flue gas.

2.2.3 Analysis of coal mine gases:

Coalification under high pressure and temperature transforms vegetation materials to coal. In this process many organic gaseous hydrocarbons and carbon dioxide is produced.

Gas emissions are calculated by measuring gas emission rate in return airway. The gas emission rate can be determined by:

$$Q = (C \times 60 \times 24 \times q) / T \times K$$

Where: Q - methane emission rate (m^3/t)

q - Air flow rate in return airway (m^3/min)

c – Concentration of methane in return airway (indecimal)

T – Coal production (t/d)

K = (max. conc. Of methane on any day in a week) / (avg. conc. Methane over the week)

Indian coal seams are classified as degree of Gassiness I, II or III when methane rate emissions are <1, 1-10 or >10 m^3/t output respectively.

2.2.4 Explosibility of Gases:

Generally methane burns when comes in contact with ignition in presence of air. But, under favorable conditions it can explode when firedamp in air is present. The general combustion takes place in the following equation:

$$CH_4 + 2(O_2 + 4N_2) = CO_2 + 2H_2O + 8N_2$$
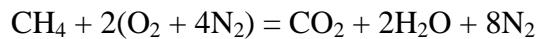
Therefore, one unit volume of methane requires two unit volumes of oxygen. Firedamp is most violent when the percentage is in 9.5% and it is explosive in nature within 5.4 to 14.5%.

Table 2.2 Flammable limits of combustible gases:

Gases	Lower flammable limit	Upper flammable limit
CH_4	5.0	14.0
CO	13.0	72.0
H_2	4.0	72.0
C_2H_2	2.5	80.0
C_2H_4	3.1	32.0
C_2H_6	3.2	12.5
H_2S	4.3	46.0

The lower flammable limit of firedamp even reduces to lower percentage when there is any presence of other combustible gases like C_2H_2, CO, etc. Le-Chatlier gave a relationship considering this properly as following:

$100/L = P_1/l_1 + P_2/l_2 + P_3/l_3 + \ldots$

Where 'P' is the percentage of corresponding gases and 'l' is lower flammable limits of those respective gases.

CHAPTER – 3

Experimental Techniques

3.1 SAMPLE COLLECTION AND PREPARATION

Sampling is the process of collecting a small portion of a whole such that the substance of that constituent represents that of the whole. It is the process by which physical and chemical features of a mineral or ore are determined with the desired accuracy. In the case for coal it covers the properties ascertained by proximate and ultimate analysis such as fixed carbon, volatile matter, ash, caking index and calorific value etc. the physical nature of the ore is sometimes requirement to be determined and the sampling process assumed should be able to give this information too. Samples are generally collected at regular interval. The interval of sampling point is governed by the regularity of the deposits as well as the accuracy for sampling projected. It should be coned and quartered on the sampling sheet, if the quantity is large before collecting it sampling bags. There is always change that missing of collected samples whereas the reduction in volume of samples has to be done individually. Different types of sampling are:

a) Bulk sampling

b) Chip sampling

c) Drill-hole sampling

d) Grab sampling

e) Channel sampling

3.1.1 Bulk sampling is the method of sampling and it is done where formal sampling methods do not give a instance scale, bulk sampling resorted to or large scale sampling. Bulk samples excrete the effect of irregular distribution of value.

3.1.2 Chip sampling can be taken in case of uniform ores and where the rock structures are independent of the values. It is done in hard ores where it is hard to cut the channels. It The sample is collected by breaking of small same sized chips from a face at points usually as spaced both horizontally and vertically.

3.1.3 Grab sampling is usually grab sampling of the ore broken in the stope is unreliable as accurate estimation of the volume of broken ore is unacceptable. In the stope or at the face which

carried ore where the broken ore are applied,. Grab sampling of tubs or ships are collected from units of regular volume, are however more representations since samples.

For the project, the samples were collected by Channel sampling method, which is the most common method followed throughout in sample collection of various mines.

3.1.4 Channel sampling (IS 436 Part I/Section I - 1964)

In channel sampling the section of seam to be sampled shall be exhibited from the roof to the floor. The particular seam sample shall, be taken in a channel representing the total cross-section of the seam having the dimensions of 30 x 10 cm, i.e. 30 cm in width and 10 cm in depth. For the channel sampling purpose there are two parallel lines, which are 30 cm obscure end at right angles to the bedding planes of the seam shall be labeled by a chalked string on the plane, freshly exposed surface of the seam. It is Obvious that, dirt bands exceeding 10 cm in thickness shall be omitted. The channel between the marked chalk lines in the seam shall be cut to a depth of 10 cm and the coal sample collected on a clean strong cloth or tarpaulin placed instantly at the bottom so that the chances of pieces flying off during excavation of coal are understated.

3.1.2 Sample preparation (IS 436 Part-I/Section 1-1964 and IS 436 Part II-1965)

The samples are taken from the different coalfield by channel sampling are broken in crusher or as per the experimental method necessities in the laboratory. The crushed samples are then sieved by the different sieve to required sizes and stored in air tight polythene packets. The packets are then stored in air tight containers for further use in the experiments.

3.2 Experimental techniques to assess spontaneous heating susceptibility of coal

In the past, a number of researchers have attempted to assess the spontaneous heating tendency of coal by carrying out different experiments in the laboratory and different countries of the world follow different methods for this purpose. It has also been observed that some of these methods are time consuming, tedious and do not give reproducible results. Various experimental techniques are described in the following sections:

3.2.1 PROXIMATE ANALYSIS (IS 1350 Part I -1984)

Proximate analysis was developed as a simple mean of determining the distribution of products found of coal. When the coal sample is heated under specified conditions, then it classifies the products into four groups: i) moisture; ii) volatile matter iii) fixed carbon, iv) ash, the inorganic residue remaining after combustion. For proximate analysis, i.e. for the determination of volatile matter, moisture, ash and fixed carbon, the method determined by IS (Indian standard) 1350 (Part-I) – 1984 was followed.

3.2.1.1 Determination of moisture content (M)

Coal is always associated with some amount of moisture, which is both physically and chemically bound, due to its nature, origin and occurrence. It is customary to differentiate between extraneous and inherent moisture. When a wet coal is exhibited to atmosphere, the external moisture evaporates, but the obviously dry coal still contains some moisture, which can be removed only on heating above 100 0C. External moisture is also called accidental or free moisture, whereas inherent moisture is named as equilibrium or air-dried or hygroscopic moisture. The quantity of external moisture counts mainly on the mode of occurrence and handling of coal, but the air-dried moisture is associated to the inherent hygroscopic nature of the coal.

Experimental Procedure:

- About 1g of finely pulverized -212 micron (BSS) air-dried coal sample is weighed in a silica crucible and then placed within an electric hot air oven.
- It is maintained at 1100C.
- The crucible with the coal sample is allowed to put in the oven for 1.5 hours and it is taken out with the help of tongues, then cooled in a desiccator for about 15 minutes.
- Then weighed. The loss in weight is reported as moisture (on percentage basis).

The calculation is done as per the following.

$$\% \text{ Moisture} = \frac{Y-Z}{Y-X} \times 100$$

Where,

X = weight of empty crucible, in grams (gm)

Y = weight of crucible + coal sample before heating, in grams (gm)

Z = weight of crucible + coal sample after heating, in grams (gm)

Y -X = weight of coal sample, in grams Y- Z = weight of moisture, in grams (gm)

3.2.1.2 Determination of volatile Matter (VM)

When coal is heated in defined equipment under appointed conditions, is concerned to as volatile matter, the loss of mass and corrected for moisture. The matter lost is composed of materials that form upon the thermal decomposition of the various constituents of coal. Some of the elements of coal volatile matter are hydrogen, carbon monoxide, methane and other hydrocarbons, tar vapors, ammonia, some organic sulphur and oxygen containing deepens and some incombustible gases, such as carbon dioxide and water vapour, all of which come from the decomposition of organic materials in coal. Inorganic materials in coal contribute the water of hydration of mineral matter, carbon dioxide from carbonates and hydrogen chloride from inorganic chlorides to the volatile matter.

Experimental Procedure:

- For determining the volatile matter a special volatile matter silica crucible (38mm height, 25mm external diameter and 22mm internal diameter) was used.
- First the empty silica crucible along with the lid uncovered was heated at 800 0C for an hour in the muffle furnace and then cooled to room temperature.
- The empty volatile matter crucible was then weighed again.
- Approximately 1gram of coal sample was weighed in the volatile matter crucible and it was placed inside the muffle furnace maintained at 925 0C with the lid covering the crucible.
- The heating was carried out exactly for 7 minutes, after which the crucible was removed, cooled in air and then in a desiccator and weighed again.

% Volatile matter = $\frac{Y-Z}{Y-X}$ X 100 –M%

Where X = weight of empty crucible, in grams (gm)

Y = weight of crucible + coal sample before heating, in grams (gm)

Z = weight of crucible + coal sample after heating, in grams (gm)

Y -X = weight of coal sample, in grams (gm)

Y- Z = weight of volatile matter + moisture, in grams (gm)

3.2.1.3 Determination of Ash (A)

During the process, the coal ash is the residue left after the combustion of coal under defined conditions. It does not occur as such in the coal, but is formed as the result of chemical changes that take place in the mineral matter. Ash and mineral matter of coal are therefore are not identical. The extraneous and inherent mineral matters are the two types of ash forming materials in coal. The extraneous mineral matter consists of materials like calcium, magnesium and ferrous carbonates, pyrite, marcasite, clays, shales, sand and gypsum. The extraneous mineral matter owes its origin to i) The substances which got linked with the decaying vegetable material during its transition to coal, which is difficult to remove by mechanical methods, and ii) Rocks and dirt getting mixed up during mining and handling of coal. The representation of inherent mineral matter the inorganic elements combined with organic components of coal. The origin of such materials is likely the plant materials from which the coal is formed. Ash from inherent mineral

matter is unimportant as far as the total quantity of ash is pertained. But Indian coals suffer from the major disadvantage, that the mineral matter content is not only high, but of intimately associated type, due to its drift origin. The several changes that occur, such as loss of water from silicate minerals, loss of carbon dioxide from carbonate minerals, oxidation of iron pyrite to iron oxide, and fixation of oxides of sulphur by bases such as calcium and magnesium. Because ash is quantitatively and qualitatively different from the mineral matter originally present in coal. In fact, incineration conditions determine the extent to which the weight change takes place and it is essential that standardized operations should be closely followed to ensure reproducibility.

Experimental Procedure:

- weight of the empty crucible is taken.
- 1gm of desired coal sample is weighed in the crucible and is taken in a muffle furnace at 450 0C for 30 minutes and the temperature of the furnace is raised to 850 0C for 60 minutes.
- After that time interval, the crucible is taken out and placed in a desiccator and weighed.

% Ash= $\frac{Z-X}{Y-X}$ X 100

Where

X= weight of empty crucible in grams (gm)

Y= weight of coal sample + crucible in grams (gm) Before heating

Z= weight of coal sample + crucible in grams (gm) After heating

3.2.1.4 Determination of Fixed Carbon (FC)

The mathematical remaining after the determination of moisture, volatile matter and ash is fixed carbon by definition. In fact a measure of the solid combustible material in coal after the expulsion of volatile matter is fixed carbon. Fixed carbon plus ash present the approximate yield of coke from coal. The value of fixed carbon is determined by subtracting from 100 the resultant summation of moisture (M), volatile matter (VM) and ash (A). with all portion on the same moisture reference basis.

$FC = 100 - (M + A + VM)$

Where, moisture (M), volatile matter (VM), Ash (A) content of coal.

3.2.2 Ignition Point Temperature (Flammability Temperature) Method:

It is the minimum temperature at which the coal begins to ignite. Ignition point temperature method is based on the fact that the ignition temperature of a coal decreases with increasing oxidation of coal and the difference between the ignition temperature of coal before and after oxidation can be used as a measure of liability of the coal to spontaneous combustion. It depends on the rank of coal, nature and intensity of ignition source, particle size, moisture, ash content and oxygen concentration (Raju,1988).

The apparatus [Fig. 7] consists of a tube furnace which can be heated to the desired temperature. The coal sample of about 0.2 g of -200 mesh size is kept in a vessel and the pressurized air is circulated in the furnace. The pressurized air will form an intimate coal dust air mixture before entering the furnace. When the furnace temperature is equal to ignition temperature of coal, the coal catches fire and the temperature is noted. The air which is circulated is pre-dried to eliminate any errors due to humidity (Nimaje et al., 2010).

3.2.3 Crossing Point Temperature (CPT) Method:

It is the lowest temperature at which the temperature of coal coincides with that of the furnace. The coal sample of 4 gm having mesh size -100 +200 is heated in a reaction tube in furnace at a constant temperature of 1 ^{0}C/min with oxygen or airflow purging at 80 ml/min in an air bath medium (Mahadevan et al.,1985) as shown in Fig. 5. The proposed risk rating of coal based on crossing point temperature is shown in Table I.

They proposed a new liability index or MR index from the analysis of CPT curve is shown in Fig. 6. According to them the heating curve could be divided into three stages. The first stage was up to a point at which the rate of heating of coal rapidly increased and an inflexion occurred and this point was called the inflexion point. The second stage started from the inflexion point and extended up to the CPT. The third stage started from the crossing point to active combustion. The risk rating with respect to the liability index is shown in Table II. They have defined the new liability index as:

$$\text{Liability index} = \frac{\text{Heating rate at the crossing point}}{\text{Time to reach the crossing point}} \times \frac{\text{Time to reach the inflexion point}}{\text{Avg. heating rate between inflexion and crossing point}} \times 10$$

Where,   T- Temperature

t- Time

ΔT.CPT–The temperature difference between crossing point and ignition temperature.

Δt.CPT – The time difference between crossing point and ignition temperature.

ΔT.(CPT-IPT) - Temperature difference between crossing point and inflection point.

Δt.(CPT-IPT) - Time difference between crossing point and inflection point.

Feng et al.(1973) termed CPT as relative ignition temperature. They used the following index for coal categorization accommodating rate rise of exothermicity and classification of FCC index is shown in Table III:

$$\text{FCC index} = \frac{\text{Average heating rate between } (110 - 220)^0 \text{C}}{\text{Relative ignition temperature}} \times 100$$

WITS-EHAC index (Uludag, 2007) of self-heating liability of coal is calculated from the following formula:

$$\text{Area of the triangle} = 0.5 \times \text{Stage II slope} \times \frac{1}{\text{Crossing point}} \times 1000$$

Table 3.1 Classification of CPT (Mahadevan et al., 1985)

CPT(^{0}C)	Risk Rating
120-140	Highly susceptible
140-160	Moderately susceptible
>160	Poorly susceptible

Table 3.2 Liability index (Mahadevan et al., 1985)

Liability Index	Risk Rating
0-10	Low
10-20	Medium
>20	High

Table 3.3 Classification of FCC index (Feng et al., 1973)

FCC Index	Risk Rating
<5	Low
5-10	Medium
>10	High

3.2.4 Olpinski Index Method:

In this method, liquid quinoline is heated in an electric oven to boil gently at a temperature of 230^0C producing quinoline vapour as shown in Fig. 9. The coal samples are powdered and small pellets of 1g of -72 mesh are prepared out of these powdered samples. These pellets are heated indirectly by quinoline vapour in an atmosphere of oxygen which is made to flow over the coal pellet at predetermined rate. While heating the coal pellets, a thermocouple is inserted into it and output of the thermocouple is given to a temperature recording system. The temperature vs. time plot of the coal samples is directly obtained by this instrument. The rate of rise in temperature at the moment of equalization of supplied oxygen gas and coal pellet temperature is either graphically determined by drawing tangent to the curve at the point corresponding to the quinoline vapour temperature at 230^0C as shown in Fig. 8 or the first derivative of the curve at 230^0C represents the slope of the curve at the same temperature. The rate of rise of temperature determined in this way is expressed in ^{0}C/min is an indication of spontaneous heating susceptibility of the coal (Olpinski,1953 and Karmakar et al.,1989). This index is generally known as Olpinski index and is denoted by Sz_a. In this method Sz_a index is corrected for ash content of the coal and is expressed as Sz_b which is as given below:

$$Sz_b = \frac{Sz_a}{100-A} \times 100 \qquad\qquad [5]$$

Where, Sz_b – Spontaneous heating index free of ash

A – Ash content of coal expressed in %.

The increase of Sz_b index gives indication that the sample is more susceptible to spontaneous combustion.

3.2.5 Differential Thermal Analysis (DTA):

The rate rise of heat evolution of coal during aerial oxidation may be compared using DTA technique. DTA is a good tool to measure qualitative and quantitative heat changes of any physico-chemical transitions and can be used for comparing the spontaneous heating susceptibility of coals (Clemens et al.,1990).

In this method, according to Banerjee et al.(1967) recommended use of calcined alumina as the reference material. The amount of coal required is 0.6 g of size -72 meshes with a heating rate of 5^0C/min. It requires a horizontal and ceramic sample holder with oxidizing atmosphere maintained by keeping the coal sample exposed to air.

The criteria set for categorizing coals from DTA studies are:

1.   The sharpness of the slope of thermo-grams in second stage and
2.   The temperature of initiation of third stage.

The rate of rise of heat evolution in second stage is observed to be much lower for coals with lower susceptibility to spontaneous heating and delays the initiation of third stage. However, once initiated, the exothermicity in third stage for even the poorly susceptible coals may be as high as that of highly susceptible ones.

3.2.6 Differential Scanning Calorimetry (DSC):

Differential scanning calorimetry is a technique in which the difference in energy inputs into a substance and a reference material is measured as a function of temperature while the substance and reference materials are subjected to a controlled temperature program. By this technique, coal samples can be studied under experimental conditions that simulate spontaneous heating

process of materials (Mahajan et al., 1976, Panigrahi et al., 2001 and Mohalik et. al., 2009). DSC provides both qualitative and quantitative information about material transition during the spontaneous heating process. For some of these transitions, DSC can also provide not only the temperature at which the transition (reaction) occurs and how much heat is evolved, but it can also provide valuable information about the rate (kinetics) of reaction. There is no standard

About 10 mg coal sample of -212 meshes was taken in the sample pans and weighed accurately. The sample was put on the sample furnace while the reference furnace was kept empty. The DSC thermo-gram was now obtained up to 600°C at a heating rate of 30°C per minute with oxygen purging at a rate of 20 cc per minute.

It can be observed from the thermo-gram that initially the endothermic reaction dominates followed by the exothermic reactions. The temperature at which the initiation of the exothermic reaction can be considered as an indicator of spontaneous heating susceptibility of coal samples, which is known as the 'onset temperature'. The lower is this temperature, higher will be

 the spontaneous heating susceptibility. The procedure of determining onset temperature or characteristics temperature (T_o) of exothermic reaction is as given below:

- A tangent is to be drawn at the inflexion point of the pre-transition,
- A second tangent is to be drawn at the greatest slope of the first exothermic reaction and
- The intersection of the two tangents gives the characteristics or onset temperature.

The onset temperature is determined by selecting the temperature at which the tangents are to be drawn. The onset temperature is read out (displayed) directly from the intersection of the tangents as shown in Fig. 11. There is no general agreement and unanimity regarding the experimental standards to be adopted for studying the susceptibility of coal to spontaneous heating and fire in above techniques. Thermal behavior of coals depends on experimental conditions such as particle size, sample amount, heating rate, carrier gas and its flow rate (Mohalik et al., 2009).

3.2.7 Puff Temperature Method/Chemical Method:

The difference in the ignition temperature before and after the oxidation of coal is taken as a measure of the liability to spontaneous heating. Different coals show different reduction values in the ignition temperatures when treated with the oxidizing agents, higher susceptibility would entail lower value of ignition temperatures.

By using samples of the coal, pre-oxidised by hydrogen peroxide and reduced by benzedene, it is stated that the difference in explosion or 'Puff' temperatures of the samples can be taken as an index of self-ignitability. The relative degree of oxidation (RDOX) and susceptibility potential could be computed from the temperatures recorded (Pal,1995).

$$\text{Susceptibility potential}\,(T_R) = PT_R - PT_O$$

$$\text{Relative degree of oxidation} = \frac{PT_R - PT}{PT_R - PT_O} \times 100$$

Where,

PT_R = the ignition temperature of coal in fully reduced state (^{0}C)

PT = ignition temperature of coal in as-analyzed state (partially oxidized) (^{0}C)

PT_O = ignition temperature of coal in fully oxidized state.

It was observed by earlier researchers that the ignition temperature decreased with the degree of oxidation (when coal is continuously oxidized) insitu in the seams and the corresponding liability to combustion increases. This method is of great importance, in practice, in determining continuously the degree of oxidation of coals which is characterized by high T_R.

In general, coals not liable to spontaneous heating, give temperature lowering of only a few degrees while coals with high liability to spontaneous heating give a temperature lowering, anything between 25 and 50 ^{0}C. The liability index, thus suggested by Vaselovski et al.,(1951) is shown in Table IV. For Indian coals, Sankaran et al.(1980) reported the values of T_R as high as 50-80 ^{0}C.

In this method, a cylindrical copper block having 6 holes along the circumference and one at the centre of the block. These 6 holes are for holding tight-fitting reaction tubes while the central one is for the temperature probe for measuring the temperature of the copper block. This design ensures the same rate of heating for all the reaction tubes. The reaction tubes open at one end and

closed at the other end. The open end of the reaction tubes are connected to eudiometer tubes by flexible plastic tubes through 3 way cocks. The eudiometer tubes are placed in a trough filled with water, the upper end of which are connected to a common horizontal manifold. One end of the manifold is provided with an aspirator bulb.

The six reaction tubes get filled with three samples, in duplicate, treated in the following manner.
Sample 1 (Original): 1g of sample obtained from 750 mg of original coal sample (-40+60 mesh of 335 micron) with 250 mg of $NaNO_2$ mixed in a mortar.
Sample 2 (Fully oxidised): 1g of sample obtained from 750 mg of original coal sample (-40+60 mesh of 335 micron) with 0.5 cc perhydrol for 20 hours in darkness and then mixed with 250 mg of $NaNO_2$.
Sample 3 (Fully reduced): 1g of sample obtained from 750 mg of original coal sample (-40+60 mesh of 335 micron) mixed with 4% by weight of benzedene and 250 mg of $NaNO_2$.

These reaction tubes were placed into the holes of copper block. The water in the eudiometer tubes was raised up to a certain level by means of aspirator bulb and then the eudiometer tubes are put on-line with the reaction tubes. The furnace is switched on and the heating rate is adjusted to $4.5^0C/min$. as the heating continued, the gases starts evolving from coal samples. When the temperature of coal reaches a point, rapid evolution of gases occurs, which is indicated by the sudden fall of water columns in the eudiometer tubes. This temperature is noted which is known as puff temperature (relative ignition temperature) and the apparatus is shown in Fig. 12.. Depending upon the oxidation state of the coal samples, three different puff/ignition temperatures are recorded namely as-received (P_T), fully oxidized (P_{TO}) and fully reduced (P_{TR}) ones and calculated the susceptibility potential and the relative degree of oxidation. The repeatability of such method is a suspect and it is not that popular as a range of variation is small. They may, however, be used only as a support index.

Table 3.4 Classification of susceptibility potential (Vaselovski et al., 1951)

$T_R(^0C)$	Risk Rating
<10	No danger of self- heating
10-25	Less liable to self -heating
>25	Very liable to self -heating

3.2.8 Wet oxidation method:

An indirect method of measuring the oxidisability of coals, by interaction of coal with oxidants like H_2O_2, $KMnO_4$, Br_2 etc., have been tried out too in some countries. The oxidation of coal is a three stage process. Where they are stepwise oxidation of coal leading to immediate formation of surface oxides of easily oxidisable species, followed by the formation of colloidal humic acids and finally to small aromatic and aliphatic acids in an alkaline medium. It's been found that alkaline permanganate oxidation of different coals believed to produce carbonic, acetic, oxalic and many benzene carboxylic acids (Tarafdar & Guha, 1989 and Panigrahi et al., 1996). It is a very quick method of categorization of coal oxidation.

In this method, 0.5 gm of coal sample of -70 mesh is mixed with 100 ml mixture of 0.1 N solution of potassium permanganate and 1 N potassium hydroxide solution at ambient temperature (30 ± 1 ^0C) and the coal oxidant suspension is continuously stirred by using a magnetic stirrer [Fig. 3]. Millivoltmeter records the EMF (potential difference) between saturated calomel and carbon electrodes at an interval of 1 min until potential difference attains a constant value. The potential difference is recorded up to 30 min. (usually attain a constant value between 3 and 30 min, depending on the type of coal) is found out and then a graph between time and potential difference is plotted (Panigrahi et al., 1996 and Nimaje et al., 2005). Change in ΔE values is found to be directly proportional to spontaneous heating susceptibility.

Tarafdar & Guha (1989) suggested that differential temperature and potential difference measurements during wet oxidation of coal might be used as an alternative technique for the assessment of spontaneous heating tendency, the advantage of this method being the portable instrument and convenience of carrying out the study on site.

CHAPTER – 4

Results & Observations

Table 4.1 Proximate Analysis Results:

Name of the mine	Moisture content (%)	Ash content (%)	Volatile matter (%)	Fixed carbon (%)
Basundhara I, MCL	9.1	19.9	55.6	15.4
Basundhara II, MCL	11.3	23.28	54.55	10.87
Ray-Bachra I, CCL	2.24	42.2	22.31	33.25
Ray-Bachra II, CCL	2.3	42.9	23.01	31.79
Ray-Bachra III, CCL	2.21	43.13	21.5	33.16
Digwarhi I, TATA Steel	3.8	5.22	43.23	47.75
Digwarhi II, TATA Steel	3.64	5.31	42.39	48.66
Digwarhi III, TATA Steel	3.22	4.96	43.62	48.43
Pathakhera I, WCL	15.39	28.94	12.51	43.16

Name of the mine	Moisture content (%)	Ash content (%)	Volatile matter (%)	Fixed carbon (%)
Pathakhera II, WCL	14.96	29.61	12.01	44.42
Pathakhera III, WCL	15.26	29.12	12.26	43.36

Table 4.2 Crossing-Point Temperature Results:

Name of the mine	Corresponding CPT Values (Celcius)	Average
Basundhara I, MCL	140.12	
Basundhara II, MCL	152.31	149.24
Basundhara III, MCL	155.3	
Ray-Bachra I, CCL	145.7	
Ray-Bachra II, CCL	143.6	145.16
Ray-Bachra III, CCL	146.2	
Digwarhi I, TATA Steel	138.6	
Digwarhi II, TATA Steel	133.7	135.72
Digwarhi III, TATA Steel	134.88	
Pathakhera I, WCL	153.95	
Pathakhera II, WCL	154.91	155.31
Pathakhera III, WCL	157.08	

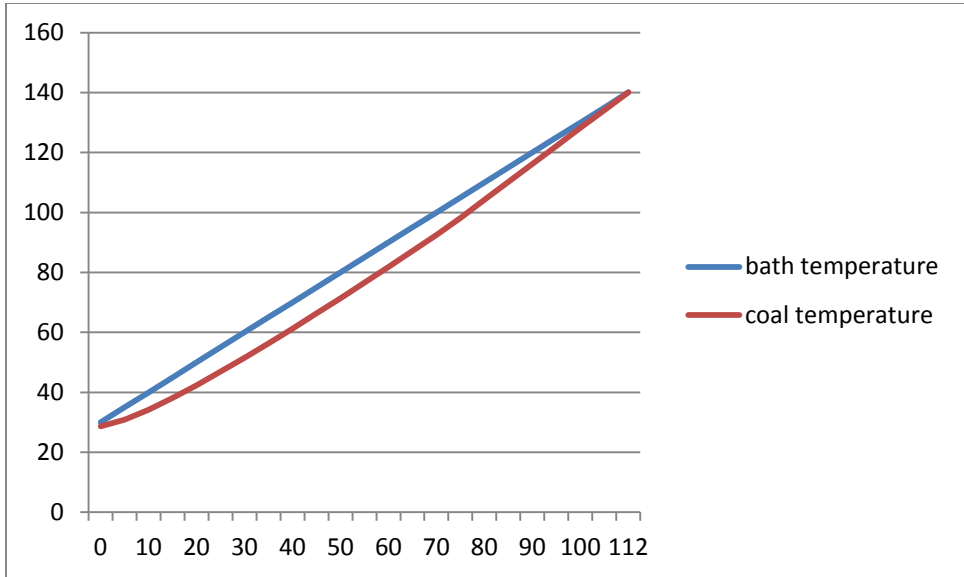**Fig 4.1 CPT Graph of Basundhara, MCL**

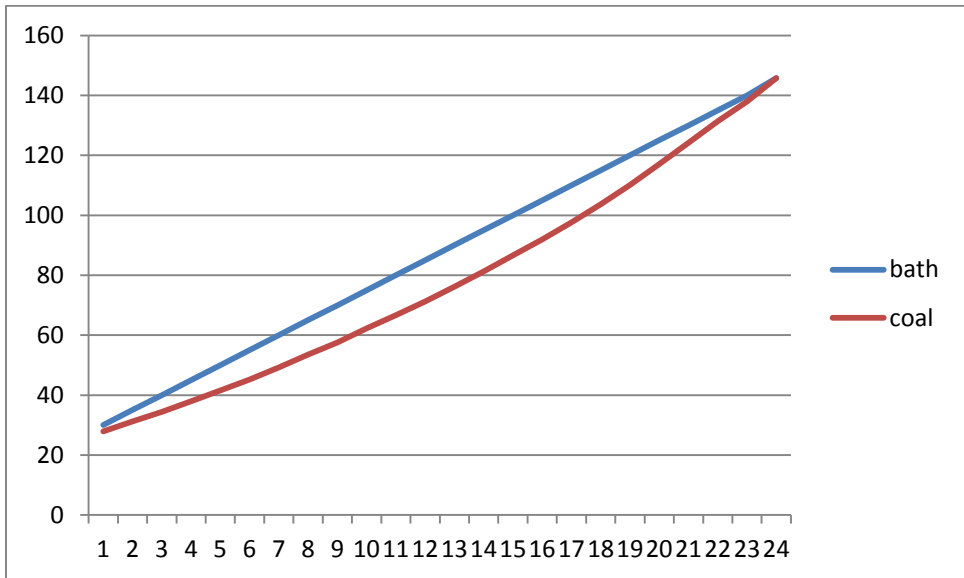**Fig 4.2 CPT Graph of Ray-Bachra Colliery, CCL**

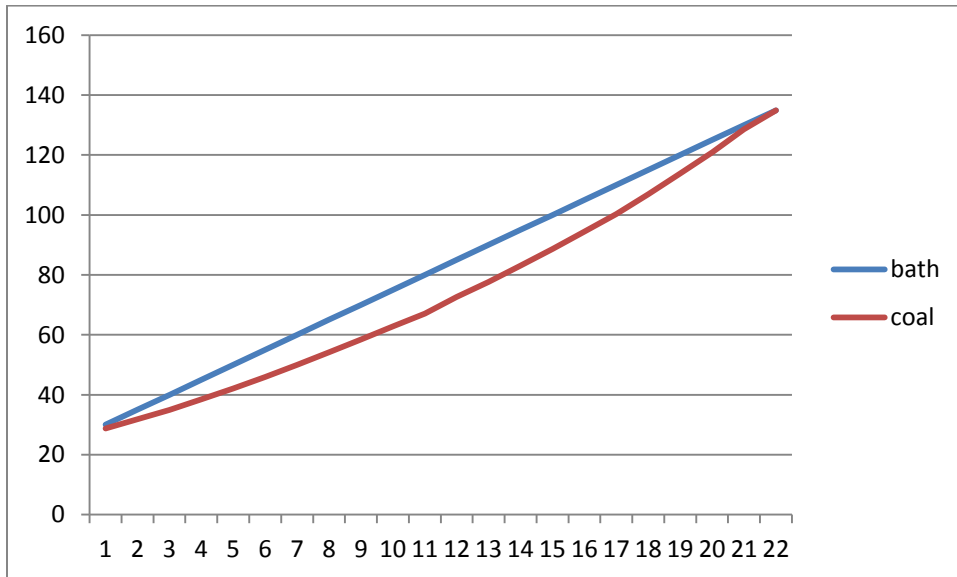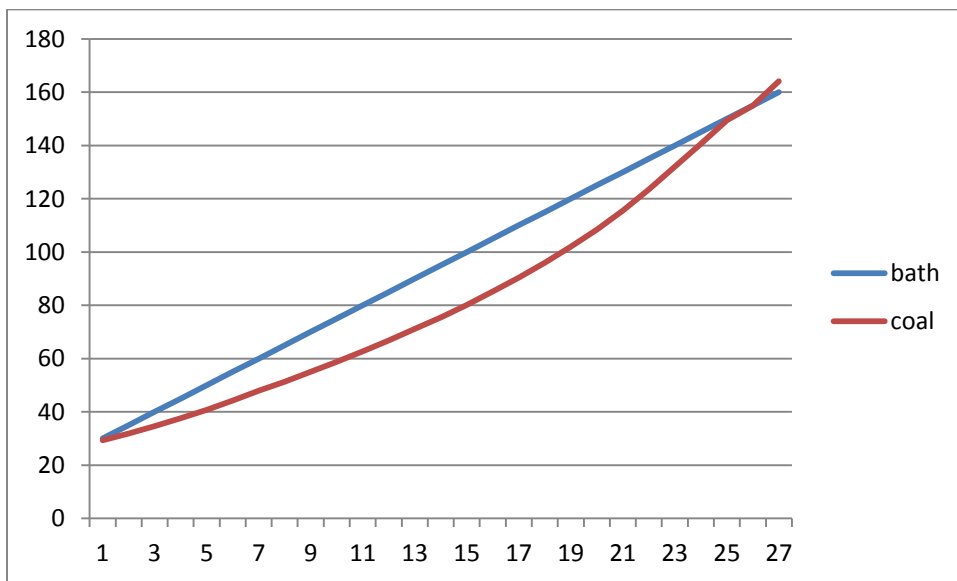**Fig 4.3 CPT Graph of Digwarhi Mine (TATA Steel)**

Fig 4.4 CPT Graph of Pathakhera Mine, WCL

4.2 CASE STUDY

4.2.1 GDK-10 Incline Mine

Spontaneous heating in Blasting Gallery (BG) panels during extraction is a major threat to safety and productivity in different mines of SCCL. In GDK-10 Incline Mine Ramagundam, the seam-3 (10.5 - 11.0m thick) was developed along bottom section leaving 1.5m coals at floor. So, an experimental work was started on December 2001, at Seam-3 of BG-2A Block–C panel (Area – 17200m^2) GDK-10 incline mine to prevent spontaneous heating /fire in goaf area of working panel (Fig.-1).
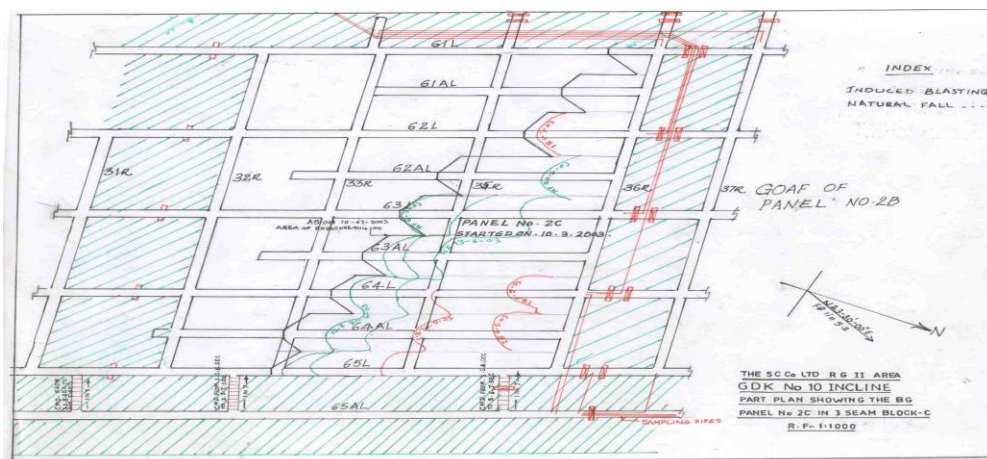

Fig 4.5 Blasting Gallery panel 2A of block – C, GDK-10 Incline Mine

(Singh V. K. et. al., 2006)

Regular thermo-compositional monitoring and application of preventive measures i.e. continuous flushing of nitrogen gas inside the goaf, sealant application over loose coal left inside the goaf as well as on barrier pillars has been carried out. A probe gallery had been developed first time in BG– 2A, C-block panel along the barrier for taking gas samples as well as for application of fire-fighting measures. A total of 5.80 lakh m^3 gaseous nitrogen was infused in the goaf to dissipate the accumulated heat and to maintain inert atmosphere for prevention of spontaneous heating at the panel. In May, 2002; CO was found in the probe gallery sample pipe and high pressure foaming material and carbon dioxide was injected in this panel. As an experimental trial basis, CO_2 has been tried in this panel and found effective to decrease the temperature inside the goaf.

4.2.2 New Kenda Colliery

Date of the Accident – 25.1.1994

Owner- Eastern Coalfields Ltd.

Number of persons killed – 55

Raniganj Coalfield

On 25.1.1994 at about 3.30 p.m. a fire broke out in the workings of Dobrana seam at New Kenda Colliery. The fire occurred in the main intake airway close to the downcast shaft. Smoke and noxious gases from the fire spread to the working places and caused the death of 55 persons. This has been the worst fire disaster of this century in Indian mines.

Two seams, namely, Kenda (8.31 m thick) and Dobrana (5.4 m thick) were being worked through shafts. The upper Kenda seam had been opened sometime in 1907 and had been developed extensively. At the time of the accident, it was being depillared in conjunction with hydraulic sand-stowing. The Dobrana seam, lying 55 m below Kenda seam, was started in the year 1962 and had been extensively developed through Pit nos. 2 and 3. The area on the rise side of the west shaft-levels had been depillared mostly by caving more than 10 years ago. The rise-side goaves were isolated by fire-stoppings with provision for water drainage where necessary. At the time of the accident, the seam was being developed on the south-west side of Pit nos. 2 and 3.

The Dobrana seam has been classified as a gassy seam of the second degree but no occurrence of gas had ever been reported. The coal is very cleaty and its crossing point temperature varies between 145°C and 155°C. Although the seam had been extensively depillared with caving in the past 20 years, there had been no case of fire in the seam. The seam is therefore considered only moderately susceptible to spontaneous combustion. However, the Kenda seam at this colliery had several occurrences of heating and fire.

The fire in the 'zero' west level of Dobrana seam on 25th January was apparently caused by spontaneous heating of the roof coal although there was no indication of self-heating before the occurrence. The large quantity of air flowing in the roadway might have diluted the symptoms of

heating like smoke, stink, CO and heat to innocuous levels. Hot and burning coal appears to have fallen down from the roof and on being exposed to the large quantity of flowing air; it produced smoke and noxious gases profusely. Although normally it was a busy area, nobody was present near the spot where the burning coal fell down and the fire could not be tackled immediately. Water was not available to quench the fire. Pumps in the main sump could not be operated due to a cable fault. Attempts were made to quench the fire by throwing some stone-dust but were not successful. Subsequently Nitrogen was used in both liquid and gaseous form but it failed to extinguish the fire which kept burning like a furnace. Finally the fire area was sealed off in the middle of February enclosing 'zero' and 1 west levels. The plan shows the fire seals numbered 1 to 27.

As the fire area had already been sealed, the Court of Enquiry constituted to find out the causes and circumstances of the fire, could not make any progress for quite some time for lack of any direct evidence. After about one and a half year when the fire had died down, the area was reopened and examined by experts. Heavy roof falls were encountered in the sealed off galleries. The fallen mass contained unburnt coal at the bottom with stones on top mixed with coal or coal ash. The stones consisted of shale and sandstone which were subjected to various degrees of heating or burning.

Different parties had attributed the following three reasons for the fire-

1. Short circuiting or bursting of electrical equipment of the underground sub-station.
2. Fire travelling from the rise-side goaf by breaking open No.8 isolation stopping.
3. Spontaneous heating of the roof coal.

All electrical equipment and cables were examined by experts after re-opening of the area. The equipments were found in their respective positions and were in tact. There was no sign of any explosion.

The possibility of fire travelling from the adjoining goaf was ruled out as three witnesses who were standing near the said stopping at the time of the accident had heard a loud sound and seen smoke coming from the eastern side. Moreover, the rescue team which had entered the goaf after

breaking No.7 stopping had not found any evidence of fire inside the goaf. In fact they had found in tact wood and coal just below No.7 stopping. The wood and coal showed no sign of burning.

Cause of the fire

The fire was caused by spontaneous heating of roof coal in 'zero' west level. Spontaneous heating started in the top 25 cm of the roof coal. Because of the cooling effect of 2800 m^3/minute of intake air, the heating could not have started anywhere lower. The 25 cm coal band had shale both above and below and therefore the fire remained confined within the 25 cm coal band. This fire had spread towards the west and north sides, helped by partial crush of the coal and air supply. The fire had a much larger horizontal spread than the usual roof coal fires. This large horizontal spread acted as a bed-separation for the overhanging shale and coal below. To this was added the dead load of the heat-affected shale above. The roof thus gave way before the fire could surface along the ledge of top coal. When this fall occurred at about 3.35 p.m., a large area of burning coal was exposed to a large volume of air resulting in rapid spread of fire.
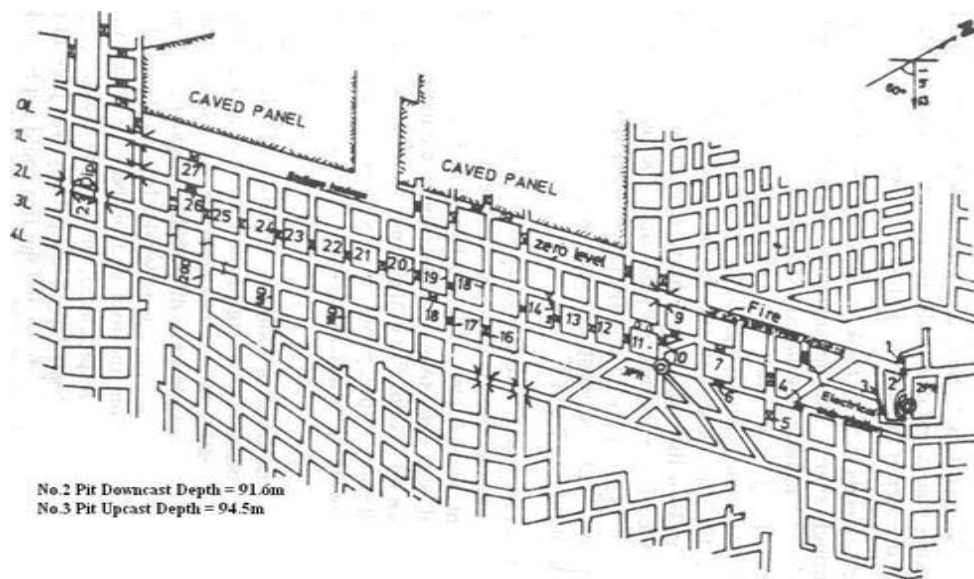

Fig 4.6 Plan of New Kenda Colliery Showing Sites of Fire

Responsibility

The fire was caused by spontaneous heating at a place where sufficient volume of air was passing through. There was no indication of self-heating. Workers were travelling through the 'zero' level all the time. Therefore, the Court held that nobody had defaulted in his duty in preventing or detecting the fire.

Observations

1. No telephone communication was in operation between the end of the haulage system and the pit-bottom and pit-top as required by CMR 87(4) (b). Had such communication been available, the workers could have been told to come out through the West side and a good number of lives could have been saved.
2. If water had been used initially on the hot mass that fell from the roof, possibly the fire would have been quenched. But there was no arrangement for water supply in the gallery. This was in gross violation of CMR 120(1) (a).
3. As the fire had occurred close to the downcast shaft, the fan should have been stopped or reversed promptly. The management would then have got sufficient time to rescue persons working inbye. It appears that nobody was ready to take upon himself the responsibility of this decision. They only tried to fight the fire through small attempts.
4. Workers had not been provided with self-rescuers in contravention of CMR-191D. If workmen had been provided with self-rescuers, they would have got at least half-an-hour or more time to escape.
5. The rescue plan showed an escape route but this was neither well-marked nor kept clear and secured. The workmen had no idea of this escape route. Another route to No.2 Pit was also available through the eastern side. The workmen could have used this route and saved themselves, but it appears they were not aware of this route either. Twenty six dead bodies in sitting posture were found at a place within 350 m of Pit No.3. Had these persons taken courage to move forward towards Pit No.3, they had a chance of saving themselves. But without doing so, they sat together and probably were conferring on their own fate. Obviously, the workmen were not made aware of the mine layout.

6. Large areas belowground in this mine have been developed unsystematically. This is undesirable. The government should frame regulations to ensure systematic development of underground mines.

7. Risk of fires in mines has not drawn serious attention of the management. Effective arrangements to deal with the fire immediately are lacking. A comprehensive regulation defining the role of the controlling authority and specifying the emergency response mechanism and method of fire-fighting is needed.

4.2.3 Jagannath Opencast Project

Date of the accident – 24.6.1981

Owner – Central Coalfields Ltd.

Number of persons killed – 10

Place – Talcher Coalfield, Orissa

At about 1 p.m. on 24.6.1981 a very unique kind of accident occurred at Jagannath opencast mine when a large quantity of hot ash and dust was ejected from the southern side of the quarry and was spread over a wide area. Fourteen workers, then present at a distance of about 60 m from the south quarry face, were engulfed in a cloud of ash and were severely burnt. Ten of them succumbed to their injuries within a few days while the remaining four survived with serious injuries.

This incident is the only one of its kind in this country and, as far as is known, it has no parallel anywhere else in the world. Because of the uniqueness of this accident, the Court of Inquiry decided to have the matter investigated thoroughly by scientists from CMRS and CFRI. Unfortunately, however, the Court and the scientists could make the first inspection of the site only after about 6 months of the occurrence when most of the field evidence was lost. The court's analysis was therefore based on facts brought out in the reports of the DGMS and the management and the evidence of the witnesses.

The mine

Jagannath Colliery had two quarries, namely, the Pilot Quarry and the Main Quarry. The accident occurred in Pilot quarry which was located in the trough of two strike faults running east-west. The strata dipped at 40 to 50 in northerly direction. No.3 seam was 9 m thick and occurred at a depth of about 12 m. No.2 seam (also called Jagannath seam) was about 35 m thick and occurred 6 m below No.3 seam. The quarry was worked with draglines, shovels, pay loaders and dumpers and was 52 m deep.

The coal seams were of poor quality and non-gassy, but highly susceptible to spontaneous heating. Coal dumped in the stockyard was reported to catch fire within 2-3 months. The coal benches in the south-side of the Pilot Quarry (which was close to a major fault) almost always had problems of fire since 1977. Attempts to dig out the fire and quench it rarely proved successful as the fire used to flare up again after some time. At the time of the accident the south-side face was on fire in patches over a length of 270m. In the opinion of the CFRI scientists the fire could have travelled to a depth of 15 m from the face. According to DGMS the fire could have penetrated to about 6 to 7 m inbye in the bottom bench. As a result of the burning away of the coal seams, there had been a number of side falls and large cracks had developed on the surface.

The accident

Prior to the accident there was heavy rainfall for 4 days. Between 20th and 23rd June a total of 162 mm of rain had been recorded. A pump was being installed on the quarry floor and the foundation job for this pump was awarded to a contractor who had engaged 3 workers including 5 females for the work. A civil supervisor, a pump operator, a mechanical fitter and the contractor himself were also present around the place. At about 1 p.m. when some of them were coming out of the pit for lunch, they heard a noise of rolling stones coming from the south side. When they looked towards the south, they saw a stream of stones rolling down the south side from a height of about 5 m above the quarry floor. As they started running towards the haul road, they felt that the sound of the sliding material was increasing and then they noticed a dense cloud of smoke and dust over their heads. The workers started shouting for help and water. Some of

them reported seeing stones falling around them. The noise was described as "whoa-whoa" as if some gas was being ejected through a small opening.

An Executive Engineer and an Overman, present at different places on the surface at the time of the accident, saw a big cloud of smoke and dust coming up from the Pilot Quarry but neither of them heard any noise. The Engineer suspected that the dragline had caught fire. He immediately rushed to the Time-office in his jeep to collect some engineering personnel and with them he went into the quarry. According to the Engineer, the cloud of dust persisted for about 10 minutes. On the way he met some, workers staggering out of the quarry while others were lying down. Their bodies were covered with dust. All these workers had burn injuries with the skin hanging loose at many places. The clothes they were wearing were partly burnt. The Overman in the meantime came in a truck. They immediately arranged to lift the injured in the jeep and the truck and sent them to the Regional Hospital at Talcher Colliery, about 8 km away. At the, hospital they were given proper treatment. Help of specialists from other nearby hospitals and from the Medical College at Cuttack was also procured.

Four of the injured persons expired on the same day. One died on the 25th and another on the 26th. Three more persons expired on 29th, 30th June and 1st July bringing the total to 9. The 10th person expired on 4th July. All these persons had 80% to 100% bum injuries. The remaining four had less burn injuries and they survived.

After the accident the quarry bed was found covered with ash and clinker, which had spread in a fan-shaped area with an arc length of 210m and a radius of 90 m. The thickness of the ash varied from 0.5 cm to as much as 25 cm and the clinkers, 6 to 8 cm in size, were scattered upto a distance of about 60 m. There was a 4.5 m high coal heap 18 m away from the toe of the bottom-most bench. The valley-like portion between the coal heap and the bottom bench was found filled with a good thickness of ash, large-sized burnt stone pieces and boulders thrown down from the top of the quarry.

Conclusions from the DGMS inquiry

There was a slide in which about 800 tonnes of material came down along the south side over a width of about 20 m. This slide might have been caused by fire, heavy rain, presence of fault-plane, water entering through wide cracks on the surface, etc. or more probably by a combination of these. But the slide alone could not have thrown 6 to 8 cm pieces upto a distance of 60 m or so. Moreover, the scattered material was thrown from places which were on fire. Therefore there must have been an outburst also. As to the cause of the outburst, the most probable appeared to be an explosion of water gas (a mixture consisting mostly of H_2 and CO).

The Management's view

The management's findings were more or less similar to that of the DGMS. They concluded that there was a slide on the southern side as well as an outburst and that the slide occurred prior to the outburst. However, they claimed to have seen at least one piece of stone weighing about 1.5 kg thrown to a distance of nearly 130 m from the site of the slide. As regards the cause of the outburst also their conclusion was the same; that is, the outburst could have occurred due to water gas explosion only. The possibility of high pressure steam being generated was ruled out because the pressure developed could not have been high enough to throw clinker and stones over such larger distances. The possibility of coal dust explosion was also ruled out because the heavy rain in the previous four days had completely saturated all the coal dust with water.

The water gas must have formed in the void created by the burning of the coal seam, and air entering the fire area must have rendered the mixture explosive which was ignited by the burning coal. Only the explosion of water gas could explain the degree of violence witnessed in this case.

The case put forward by CMRS

Like one of the Trade Unions, CMRS first considered the possibility of the incident having been caused by a simple slide. They assumed 24 m thickness of coal to have burnt completely reducing the original volume to one-third (as the ash content in coal varied form 30 to 35%). Assuming all favourable conditions, the velocity of the sliding mass could not have exceeded 18 m/s. Then assuming that any clinker might have attained this initial velocity, the maximum throw of a projectile at an angle of 450 would come to 33 m only. Therefore, the scattering of stone pieces beyond 33 m could not be justified by simple sliding and projectile ejection. They, therefore, concluded that there was an outburst also.

In trying to find out the cause of the outburst they considered the following possibilities:-

1. Accumulation of methane along the fault plane and its explosion.
2. Production of inflammable gases and methane by distillation of coal and their explosion.
3. Coal dust explosion
4. Hydraulic stimulation of crushed and permeable coal seam which may increase gas outflow upto 20 times.
5. Generation of steam which could mechanically eject the ash and clinkers.

Scientific analysis of all factors eliminated these possibilities. Last of all, they discussed the possibility of water gas explosion and considered it probable. According to them, hot water and steam may pass over burning coal and change into water gas. Small amount of this gas with wide explosive limit (5-70%) may explode deep in the cut created by the fire and initiate scattering of the burnt ashes and clinkers over a fan-shaped area as reported by witnesses.

The colour of the smoke in this case will be grey (like that of ash), violence will be less severe but overall temperature of the bed of already heated ash will then rise because of its exothermic reaction. This seems to justify the burning of victims even at 90 m distance without excessive violence, smoke or sound.

The case put forward by CFRI

CFRI did not take into consideration any possibility other than explosion of gases. Their contention was that the smouldering fire in the seam might have been gradually covered up by self-generated ash and cinders and subsequently further blanketed by a huge amount of rock debris falling from the top. Heavy blanketing of the fire zone might have led to a number of chemical reactions in the coal-bed, namely, combustion, gasification and carbonisation, all of which culminated in the continuous generation of gases which might have built up a tremendous pressure deep within the fire zone. Eventually, when a critical pressure was reached, the cover of rock debris was thrown out creating an opening through which air rushed into the gas zone and caused an explosion of inflammable mixtures of CH_4 air, CO-air and H_2-air. Such an explosion, deep within the fire zone, had perhaps led to the sliding of enormous quantity of rock debris and the shooting of very hot spent gases accompanied by hot ash and cinders.

Commenting on the CFRI report, the management argued that if the gases produced by pyrolisis of coal not escape, it follows that no air could enter such a zone. Therefore the coal burning would have stopped and the fire would have died out in course of time.

The Court did not consider the case put up by CFRI as a probability.

The Court's analysis

The Court based its analysis on the following three facts:

1. A clinker weighing 1.5 kg was thrown over a distance of 130 m. The pressure required to do this would be about 350 p.s.i. Such a high pressure could not have been developed by a gas explosion.
2. About 250 tonnes of material was thrown over a distance of 250 ft. The kinetic energy required to do this would be 17.9 mil Btu.
3. "Whoa-whoa" sound was heard immediately after the incident.

The Court contended that in the circumstances prevailing at the time of the incident, there was a possibility of generation of superheated steam at high pressure. The heavy rain on four days prior

to the incident could have clogged the cleavage planes and other natural fissures in the coal and associated rocks and steam superheated to about 800°F could be entrapped at pressures of 350 p.s.i. or more in the interstices of ash and strata. CMRS had not considered superheated steam and its entrapment in interstices of ash. Superheated steam can explain the high pressure required in this case and confinement in interstices explains how a large volume of steam could be stored. The "whoa-whoa" sound is explained by the escape of steam after the initial outburst.

It appears that a slight movement in the coal face was the fore-runner of the outburst and large scale slide of the side occurred only after the ejection of material from within the fire zone.

Conclusion

The incident could have occurred by a combination of circumstances. Probably high-pressure steam confined within the fire zone played the main role in the ejection of hot ash and cinders. But the contribution of other factors, such as the explosion of water-gas, cannot be ruled out.

This incident is a pointer to the grave danger that a blazing fire in a coal bench of an opencast mine can bring about. It is therefore necessary that such fires must be quenched as soon as noticed.

Recommendations

1. The Regulations only specify that the occurrence of a fire in an opencast mine should be reported to the DGMS but do not say what the DGMS should do thereafter or the mine management should do on its own. The Regulations should be suitably amended to specify clearly the steps to be taken by the DGMS and the management.
2. In an opencast mine as soon as a fire is noticed, it should be dug out and quenched. Alternatively, measures should be taken to prevent its spread by not allowing access of air to the fire; for example, by blanketing. In no case should mining operations proceed to such a depth, as in the case of Jagannath Colliery, that the management can claim that the fire cannot be tackled. Such a situation should not be allowed to develop. If the fire is not controlled, further mining operations in that quarry must stop.

3. The phenomenon of spontaneous heating and the measures to deal with fires in opencast mines need to be investigated thoroughly by a High Power Body which should evolve guidelines for the industry to deal with the problem.

CHAPTET 5

Discussion and Conclusion:

5.1 Discussion:

The proximate analysis and the CPT values have been found out for the samples obtained. Hence, the experimental results and comparison indicates the following:

- The volatile matter of Basundhara mines, MCL is very high reaching up to 54.55 % and has a very low fixed carbon with an average value of 13.14 %.

- Ray-Bachra Colliery of Piparwar Area, CCL has ash content of 42.74 % but with a fixed carbon of 32.73 %.

- Digwarhi mine, TATA Steel has coal of very high ash content with averaging about 43.08 % and average fixed carbon of 48.28 %.

- Pathakhera mine of WCL contains significant moisture content of 15.20 %, considerable ash content with 29.22 % of an average and it also contains on an average 43.65 % of fixed carbon content.

- The average CPT values of Basundhara (MCL), Ray-Bachra (CCL), Digwarhi (TATA Steel) and Pathakhera (WCL) mines are $149.24^{o}C$, $145.16^{o}C$, $135.72^{o}C$ and $155.31^{o}C$ respectively.

5.2 Conclusion:

The proximate analysis of the coal samples brought tells us that Digwarhi mines (TATA Steel) has the highest volatile matter content and Pathakhera very near to it. That means there is a high possibility of mine fire as higher percentage of combustible gases will be present in the air too.

The CPT value for the Digwarhi mines (TATA Steel) is the lowest and it is also below 140°C. Which indicates Digwarhi mines has the highest susceptibility towards spontaneous combustion among these mines. And with an average CPT value between 140-160°C, the other three mines are moderately susceptible to spontaneous combustion.

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