Mines and Minerals Development Project (Hydrocarbon Package #07)

Review of the existing mining operations of the Barapukuria Coal Mine and Recommendation on improvements (Final)

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Author

	Name & Title
Produced by	*Prof. D.C. Panigrahi, Prof. S. Chaudhuri, Prof. S.B. Srivastava, Prof. U.K. Singh, Nazrul Islam, A.K.M. Shamsuddin, Md. Mosharraf Hossain, Md. Ehsanullah, Pukhraj Sethiya*
Reviewed by	*Yogesh Daruka*
Approved by	*Kameswara Rao*

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PricewaterhouseCoopers Pvt. Ltd.

Place: Hyderabad

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Table of Abbreviations

AFC	Armoured Face Conveyor
ARC	Armoured Rear Conveyor
ARD	Airborne Respirable dust
BAPEX	Bangladesh Petroleum Exploration and Production Company Ltd.
BCMCL	Barapukuria Coal Mining Company Limited
BGFCL	Bangladesh Gas Fields Company Ltd
BGDCL	Bakhrabad Gas Distribution Company Limited
BOGMC	Bangladesh Oil, Gas and Mineral Corporation
CMC	Chinese National Machinery Import & Export Corporation
DBT	Dry Bulb Temperature
DERD	Double Ended Ranging Drum
ECNEC	Executive Committee of National Economic Council
EFM	Empirical Formulas Method
EIA	Environmental Impact Assessment
EMRD	Energy and Mineral Resources Division
FEM	Finite Element Method
GSB	Geological Survey of Bangladesh

GTCL	Gas Transmission Company Ltd
IDC	Interest During Construction
IMC	International Mining Consultants
IRR	Internal Rate of Return
KGDCL	Karnaphuli Gas Distribution Company Limited
KW	Kilowatt
LDT	Lower Dupi Tila
LTCC	Longwall Top Coal Caving
M&P	Maintenance and Production
MGMCL	Maddhapara Granite Mining Company Ltd
Mpa	Mega Pascals
Mt	Million Tonnes
Mtpa	Million Tonnes Per Annum
NNW	North Northwest
NPV	Net Present Value
PGCB	Power Grid Company Ltd., Bangladesh
PGCL	Paschimanchal Gas Company Ltd
PME	Periodical medical examination
PSLW	Powered Support Long Wall
R&D	Research and Development
RCE	Revised Cost Estimate
RPGCL	Rupantarita Prakritik Gas Company Ltd
RQD	Rock Quality Designation
SGFL	Sylhet Gas Fields Ltd
SOP	Standard Operating Procedure
SRDI	Soil Resource Development Institute, Dhaka
SSR	Systematic Support Rule
SGCL	Sunderban Gas Company Limited
TCS	Tri-axial Compressive Strength
TEFS	Techno Economic Feasibility Study
TGDTCL	Titas Gas Transmission & Distribution Company Ltd
TPD	Tonne per day
UCS	Ultimate Compressive Strength
UDT	Upper Dupi Tila

UML	Upper Mineable Limit
UTS	Ultimate Tensile Strength
WBT	Wet Bulb Temperature
WFFZ	Water Flowing Fractured Zone
XMC	Xuzhou Coal Mining Group Company Limited

Table of contents

List of Tables

List of Figures

1. Executive Summary

1.1.1. This report on "**Review of the existing mining operations of the Barapukuria Coal Mine and Recommendation on improvements (Final)"** is prepared as a part of the Mines and Minerals Development Project (Package#07) for Hydrocarbon Unit, Energy and Mineral Resources Division, Bangladesh.

1.1.2. In this report, review of ongoing operations of the Barapukuria Coal Mining Company Limited (BCMCL) has been discussed and recommendations have been provided for improvement in mining operations. Comments on suitability of the extraction of opencast mining in the Northern part of the Barapukuria basin has also been discussed.

1.1.3. Review of the existing mining operations at the mine is done based on the reports and data collected during the site visits, observations made during visits and the subsequent discussions held with the BCMCL management, mining contractor of BCMCL and IMC, the technical consultant of BCMCL.

Brief information of Barapukuria Coal Mine

Sl. No.	Particulars	Values
1.	Coal bearing area of the mine (Sq. Km)	6.68
2.	Number of boreholes	33
3.	Borehole density (no. of boreholes/Sq. Km)	4.94
4.	No. of coal seams	7
5.	Thickness range of persistent splits of coal seams (m) :	
	(a) VII Seam (Not persistent)	0-4.2
	(b) VI Seam	29.40-40.52
	(c) V Seam	1.74-10.37
	(d) IV Seam	3.12-10.58
	(e) III Seam	0.72-2.60
	(f) II Seam	13.95-15.24
	(g) I Seam	0.46-4.57
6.	Total geological reserves (in million tonnes (Mt))	351.20
7.	Geological reserves converted into project by CMC (Mt):	
	(a) VI Seam	285.41
	(b) V Seam	16.40
	Total	301.81
8.	Recoverable reserves projected by CMC (Mt):	
	(a) VI Seam	81.46
	(b) V Seam	2.56

	Total	84.02
9.	Gross calorific value of VI Seam coal (MJ/kg)	25.68
10.	Depth of shafts (m) : Auxiliary shaft (Intake) Main shaft (Return)	 320 326
11.	Average make of water at actual (m^3/day)	35000
12.	Main ventilation fan : Rated capacity (m^3/s) Pressure (Pa)	 150 2000
13.	Major equipment as per 1st M&P contract between BCMCL and CMC-XMC consortium (Contractor): (a) DERD Shearer (nos.) (b) Powered support (nos.) (c) Road headers (nos.)	 2 194 4
14.	Rated mine output (Mtpa)	1.00
15.	Date of signing of 1st M&P contract between BCMCL and CMC-XMC consortium	4th June, 2005
16.	Date of commencement of commercial production	10th September, 2005
17.	Contracted total production during 1st M&P contract period of 71 months ending on 10.08.2005 (Mt)	4.75
18.	Achieved total production during 1st M&P contract period (Mt)	3.651
19.	Commencement of 2nd M&P contract	11.08.2011
20.	Period of 2nd M&P contract (months)	72
21.	Total production to be obtained during the 2nd M&P contract (Mt)	5.50
22.	Actual manpower in the mine: a) BCMCL: Permanent officers Permanent staff Contractual staff Outsourcing (staff & officer) b) M&P contractor: Chinese (registered) Chinese (attendance) Local workers (registered) Local workers (attendance) Total manpower (attendance)	 107 52 01 213 263 229 1127 716 1318

23.	Total capital investment (In Lakh Taka) as per revised Project Plan approved by ECNEC on 15.08.2002 : Pre-construction works (Including land) Plant & Machinery Construction Works Transport and Vehicle Manpower Cost (Including training) Other Cost (Including Design, Engineering, Installation, Consultancy, IDC, Overheads etc) Total Capital	9129.78 38717.47 54001.91 565.78 3900.42 36797.30 143112.66
24.	Operating cost (USD/tonne): in FY 2008-2009 in FY 2009-2010 in FY 2010-2011	 54.83 60.23 64.58
25.	Net profit (in Crore Taka): in FY 2008-2009 in FY 2009-2010 in FY 2010-2011	 141.40 88.39 131.98
26.	Total contract price as per new M&P contract (including local taxes, VAT, customs duty etc.) in million USD	200.976
27.	Production cost as per above new M&P contract per tonne	USD 36.54 (Foreign Component) + Taka 1394.60 (Local Component)

Table 1 : Brief information of Barapukuria Coal Mine

Note: While 2nd M&P Contract has started, Contract was not made available to consulting team and thus no review or study has been conducted on 2nd M&P contract.

1.1.4. In Section 3 parameters of the Mine from the Approved Project Report (Modification of Basic Design for Barapukuria Coal Mine Project – Petrobangla, Bangladesh Prepared by China National Machinery Import and Export Corporation) has been discussed primarily covering aspects like mine development, mode/method of development, upper limits of Seam VI, method of extraction, geological conditions, ventilation design, mode of transportation for man and material, etc.

1.1.5. In section 4, current status (December-January 2012) of Barapukuria mine has been discussed. Key operational aspects and challenges being faced in mine such as mine development (developments made so far), extraction of longwall panel, year-wise production till June 2011 and panel-wise production, major production equipment, underground mine environment, mine dewatering system, power supply status and mine safety have been discussed. It was also observed that Barapukuria mine could not achieve the rated capacity of 1.00 Mtpa till date in any operating year.

1.1.6. Section 5 of this report provides an appraisal of existing mining conditions and identification of areas for further improvement. The key findings discussed in this section are as follows:

1.1.6.1. Accordingly to CMC Report, only 81.46 Mt (i.e. 28.54%) out of 285.41 Mt of geological resources of Seam VI in the main syncline in Barapukuria Basin would be recoverable.

1.1.6.2. Geology of Seam VI in the central part of the mine is established and being worked. The northern part and the southern part of the seam are undeveloped. The northern part of the seam is moderately explored but the southern part is under-explored. Further exploration of the seam is necessary to achieve the following objectives:

- Delineation of the trend of sub-crop of Seam VI.
- Up-gradation of reserves presently categorized under Rank 'C'.
- Firming up of the geological structure, thickness and quality of Seam VI occurring in the southern part of the deposit up to Phulbari exploration block.

1.1.6.3. Upper seams (seams I, II, III, IV and V) are not adequately explored. Exploration of these seams cannot be taken up at present due to following reasons:

- These seams are subsidizing which is expected to continue till overlying strata stabilises after VI seam is fully exploited. Therefore, it will be prudent to explore these seams after depletion of VI seam, to capture their final profiles.
- It will be unsafe for the drilling crew to work over unstable (subsiding) area. And;
- There is possibility of jamming of drill column in the drill hole due to ground movement.

1.1.6.4. Piezometric observation of seam VI indicated that the water level is gradually receding, which implies that there is no charging of Gondwana formation from the Upper DupiTila formation.

1.1.6.5. In case opencast mining is done in the northern part of the deposit, large quantities of water need to be pumped.

1.1.6.6. Source of underground discharge water is the connate water from large thickness of Gondwana formation. This water has been preserved within this formation since its deposition.

1.1.6.7. For management of Dupi Tila water in the working area, further detailed study is required through modeling to review the application of the LTCC method of mining to predict its impact on the overall stability of the mine and quantity of water inflow into the mine due to unstable conditions arising out of movement/caving of the overlying ground above VI seam.

1.1.6.8. Method of Working:

1.1.6.8.1. Presence of a thick overburden of unconsolidated water bearing strata over the fractured and caved hard rock will result in development of two major problems related to mine safety during extraction of 2nd and subsequent slices of VI Seam and these problems will be gradually more pronounced as the number of slices extracted increases.

1.1.6.8.2. Possibility of ingress of water from the highly water bearing UDT aquifer into the VI Seam workings through flow paths developed due to any of the following reasons:

- Generation of fracture plane across the strained hard rock and LDT strata (particularly where it is thin) which extends to the base of UDT horizon.
- Passage of water from UDT through water transmitting faults.
- Opening of fault planes which were previously closed and non-water transmitting.

1.1.6.8.3. Studies should be made to predict, inter alia, the following:

- Increase in thickness of caved zone with the increase in cumulative thickness of slices extracted.
- Increase in thickness of water permeable fractured zone with the increase in the cumulative thickness of slices extracted.
- Make of water in underground working in each slice.
- Support resistance required in longwall face in each slice.
- Expected surface subsidence and subsidence of each of the strata

1.1.6.9. Underground Mine Environment and Mine ventilation

1.1.6.9.1. Major underground environmental problems encountered in the Barapukuria Coal Mine are as follows:

- High heat and humidity affecting the workplace environment
- Ventilation problem of the mine
- Spontaneous combustion and fire issues
- Other underground environmental problems

1.1.6.9.2. CMC Report (2000) has suggested an effective temperature of 32°C as the limiting environmental condition for this mine, whereas the effective temperature observed in some of the longwall panels is more than 32°C at number of locations.

1.1.6.9.3. Important reasons identified for high underground temperature and humidity are high geothermal gradient and high virgin rock temperature, heat liberated from the hot water oozing out from the mine, heat from auto-oxidation of coal and carbonaceous matter, heat from electrical equipment operated in the longwall panels and high temperature of surface air.

1.1.6.9.4. It is suggested that the efforts should be made for prevention of water seepage and reduction of humidity in the mine.

1.1.6.9.5. Provision for a detailed study has been recommended for installation of air cooling system for addressing the issues of high heat and humidity in the mine.

1.1.6.9.6. In addition to this, measures have been suggested to improve upon the existing mine ventilation condition for Barapukuria coal mine such as, carrying out detailed ventilation survey, developing ventilation network model, construction of a ventilation shaft, setting up of ventilation laboratory by BCMCL which can be undertaken by BCMCL in the following years.

1.1.6.10. The section further discusses issue of Spontaneous combustion of coal faced by BCMCL and the various causes for the same. Suitable recommendations to avoid fire problem in goaf has been provided.

1.1.6.11. Other underground environmental problems such as, Mine gases, Airborne respirable dust (ARD) problems, Coal dust explosion hazard and Dust problems have also been discussed in this section.

1.1.6.12. In the section, mine safety aspects and various accidents happened in mine has been discussed. An analysis of the cause of incidences occurred in the mine till July, 2011 have been presented. To improve mine safety performance, safety tools such as Risk Assessment and Management, Emergency response system, Training and retraining of workers, Periodical medical examination (PME) of workers, etc. have been recommended.

1.1.6.13. Logistics and Infrastructure

- As per CMC (2000) Report, mine infrastructure and logistics was designed for a rated capacity of 1 MTPA coal production from the mine.
- Sub-Section 5.7 of this report discusses existing surface and underground infrastructure, shaft construction and pit bottom and top areas, water pumping system, power supply and distribution.

1.1.6.14. Surface Environment, including aspects like Physical environment, Land use pattern have also been discussed in detail in this section.

1.1.6.15. Organization Structure

- Various recommendations and changes have been suggested to improve management control and monitoring over the mining operations. It is suggested to increase involvement of BCMCL employees in training programs which will help in improving their skills, develop in-house capability for mine operations.
- Proposal for initiating a Project and Planning department by BCMCL has also been recommended in this section.

1.1.7. Section 6 of this report discusses Suitability of longwall top coal caving (LTCC) method. Some of the key features of LTCC method are as discussed below:

1.1.7.1. The method envisages extraction of 6m coal in single lift immediately below 1st slice.

1.1.7.2. A new set of face equipment (specifications not known) for LTCC method has been included in the new M&P contract, the rated capacity of which will remain at 1 Mtpa.

1.1.7.3. With the assumed height of longwall face of 2^{nd} slice in LTCC method as 3m and the thickness of coal parting between floor of 1^{st} slice and roof of 2^{nd} slice longwall face as 3m, the LTCC system will be achieving a higher coal recovery compared to the conventional multi-slicing system, where the entire parting coal of 3 m (or so) will be lost in goaf with consequent increased risk of fire.

1.1.7.4. Rate of advance of the longwall face in LTCC method may be slowed down occasionally due to problem of blocky coal coming down.

1.1.7.5. Arrangements of nitrogen flushing and chemical treatment will be required to mitigate risk of fire in goaf in LTCC panels.

1.1.7.6. To ensure a regular production of 1.0 Mtpa from one set of LTCC equipment, appropriate actions have to be taken to minimize the following delays:

- Delay in face advance due to problem of jamming or damage of ARC, delay in clearing of roof coal etc.
- Delay during salvaging and re-installation of the LTCC equipment to the next longwall panel.
- Delay in taking up due maintenance and breakdown repair works

1.1.7.7. System of proper spares management should be established and effective steps for up-gradation of skills of the operation and maintenance crew should be taken to reduce the delays.

1.1.8. Section 7 discusses on the potential of opencast mining of VI seam in open window area. The key points covered under this section are summarized below:

1.1.8.1. The 'open window' area is fairly well explored and has a geological reserve of 135.18 Mt which is substantial.

1.1.8.2. Considering the current selling price of Barapukuria coal, the area between the NE-SW trending sub-crop of VI seam in the rise side and the line joining DOB 7 and DOB 11 (following seam floor contour) in the dip side can tentatively be assumed to have opencast mining potentiality with VI seam as base.

1.1.8.3. Hydro-geological studies should be carried out to predict the safe distance of opencast excavation from mine shafts, important surface structures and existing mine workings of VI seam so as not to affect their stability.

1.1.8.4. Slope stability studies for internal and external soil dumps should also be taken up to ascertain maximum height and safe slope of these dumps for dump planning.

1.1.8.5. It will be essential to fill up the residual void after productive life of the mine by re-handling the material from surface external dump. This will ensure safety of future underground workings, free the land area occupied by external dump and help in re-establishment of the ground water regime of the UDT aquifer.

1.1.8.6. Additional exploratory drilling is required in the open window area to establish resource geology before planning of an opencast mine for firming up the trend, thickness and coal quality of the sub-crop region of VI seam falling in the 'open window' area.

1.1.8.7. Environmental impact assessment (EIA) study should also be carried out based on the TEFS to assess the environmental damage to be caused by such opencast mining in the open window area.

1.1.9. Mining of VI Seam in southern side of south district has been discussed in the section 8 of this report. Due to higher initial depth of open pit in the south side compared to the north side, the problems related to slope stability and draw down created by pumping of aquifer is expected to be more severe. The cost of initial excavation is expected to increase phenomenally because of higher depth and also, the recovery of coal is likely to be much lower in the south side because of comparatively narrow pit width and flat slope of high walls on all sides of the pit. Therefore, this area, lying to the south of latitude 10°14’00” has to be worked by underground mining. A detailed planning exercise should be taken up for firming up the method of development and extraction of this area which can be done only after the geology of the area is firmed up and reserves estimated.

1.1.10. Section 9 of this report discusses Mining of upper seams which can be extracted only by opencast mining with V seam as base, after allowing some time for stabilisation of strata above VI seam following depletion of reserves of VI seam. Thus opencast mining of the upper seams may become viable only at a distant future.

1.1.11. Feasibility of adopting stowing method for extraction of VI seam has been covered in the section 10 of this report with the following key points:

1.1.11.1. Adoption of stowing in Barapukuria mine will result in many advantages while mining of extra thick VI seam as outlined below:

- Reduction in strata control problems while working in multi-slices
- Reduction in the risk of spontaneous combustion
- Improvement in face ventilation as longwall mining with panel barriers can be practiced
- Reduction in the make of water from aquifer due to reduction in ground movement

1.1.11.2. Detailed studies for establishing availability and transport of stowing material are required to be taken up. Also, supply of tailor made Powered Supports for stowing need to be ensured, if stowing is adopted in future.

1.1.11.3. Extraction of slices in ascending order with stowing will significantly improve safety and recovery but the actual implementation of the method will require more efforts and scientific studies.

1.1.12. Recommendations and conclusions have been covered in the section 12 of this report which includes recommendations for addressing issues related to areas like exploration, hydrogeology, method of mining, suitability of LTCC method of mining, safety and recovery of coal reserves, mine production capacity, mine economics, Feasibility of opencast mining of VI seam in open window area, Mining of VI seam in the southern side of south district, Mining of upper seams, Feasibility of adopting stowing method for extraction of VI seam, underground mine environment, mine infrastructure, organization structure and mines safety. This section covers the various proposed changes, studies and way forward to overcome issues pertaining to the various aspects of Barapukuria coal mine.

2. Introduction

2.1. Background

2.1.1. In 1985, Geological Survey of Bangladesh (GSB) discovered high quality bituminous Coal at a depth ranging from 118 to 509 m in Barapukuria under Parbatipur Upazilla in the district of Dinajpur. During 1987-1991, a UK based organization M/s Wardell Armstrong carried out the techno-economic feasibility study on this coal reserve under ODA financial support programme. Based upon the Wardell Armstrong report, Petrobangla undertook Barapukuria Coal Mine Development Project and the Project Plan (PP) was approved by ECNEC on April 21, 1993.

2.1.2. After the approval of the Project, a Construction Contract under supplier's credit was signed on 7th February, 1994 between China National Machinery Import and Export Corporation (CMC) and Petrobangla with a view to develop an underground mine having a capacity of 1.00 Million Metric Tons of coal per annum. To ensure proper implementation of the project and smooth functioning of the mine operation Barapukuria Coal Mining Company Limited (BCMCL) was formed and registered on August 04, 1998 under the Companies Act 1994 of Bangladesh.

2.1.3. After substantial completion of the project, the mine was taken over from the Contractor (CMC) on June 30, 2005 issuing a Conditional Acceptance Certificate. A Management, Production and Maintenances Service Contract (M&P Contract) was initialed between BCMCL and a consortium of CMC and Xuzhou Coal Mining Group Company Limited (XMC) and commercial production of coal commenced on September 10, 2005. The term of this M&P Contract was 71 months and during this period a total production of 4.75 Mt was planned to be achieved. The contract term completed on 10.08.2011 with a total production of 3.651 Mt of coal. For continuous production of coal from the mine, a new M&P Contract has been signed between BCMCL and CMC-XMC Consortium. Term of this contract commenced from 11.08.2012 and will remain effective for a period of 72 months with total target coal production of 5.5 Mt.

2.2. TOR for this Report

2.2.1. In the present module i.e. 'Review of the existing mining operations of the Barapukuria Coal Mine and Recommendation on improvements', Consultant will review existing mining operations at Barapukuria coal mine and suggest measures to improve its operational efficiency. The key areas identified for review of mining operations are based on the discussions during inception stage and subsequently during the course of engagement.

2.3. Scope of Report

2.3.1. This report focuses on review and analysis of mining operations and challenges being faced at Barapukuria Coal Mine. Further, as agreed in the inception stage, opportunity to mine the northern and southern part of Barapukuria Coal Basin and possibility of open cast mining is discussed. The scope of this report is as follows:

- To review the present mining operations in the first slice of Seam VI of Barapukuria coal mine.
- To comment on the suitability of mining method for mining at the Southern extension of the coal seam.

- To comment on the suitability of introducing Longwall Top Coal Caving method for second slice of Seam VI.
- To comment on mining method for coal seams above Seam VI, based on the studies already conducted.
- To comment on the suitability of operating the Northern extension by opencast mining method, based on the studies already conducted.

3. Summarized parameters of Approved Project Report

This section briefly discusses the parameters of the approved project report (Modification of Basic Design for Barapukuria Coal Mine Project – Petrobangla, Bangladesh, Prepared by China National Machinery Import and Export Corporation, April 2000) which was initial base to start mining activities at Barapukuria Coal Mine.

The objective of this chapter is to provide a brief summary of approved mine operation related aspects which were to act as guidelines for operating Barapukuria Coal Mining Project. However, details of the present mine workings and their appraisal has been discussed in Section 5 of this report. Same section also covers review of existing mining operations in detail.

3.1. Barapukuria Coal Deposit – General Information

3.1.1. The Barapukuria Coal Deposit is located in the north-western part of Bangladesh, between 25° 21' N & 25° 34' N and 88° 57' E & 88° 59' E, about 50 km south east of the district headquarters of Dinajpur and about 20 km east of the border with India.

3.1.2. The main north-south broad gauge railway line passes to the immediate west of the project area and the towns and railway stations of Phulbari and Parbatipur are located 5 km to the south and 20 km to the north respectively. The Madhyapara hard rock project site is about 10 km to the east of the coal mine.

3.1.3. The mine field is a part of alluvial plain formed between river Ghen and river Jamuna. Relief of the area is flat with surface elevation varying between +29 m and +32 m from MSL. The surface gently slopes from north to south.

3.1.4. The Barapukuria coal field is an asymmetrical half-graben basin with axis roughly along N-S which is truncated in the east by a major fault. The basin is of Gondwana coal bearing formation of Permian period and is preserved in a graben structure. It overlies unconformably on basement Archaean rocks.

3.1.5. According to CMC, the Geological reserves (only seam VI and seam V) of Barapukuria coal deposit are estimated at 301.81 Mt and the recoverable reserves (only seam VI and seam V) are estimated at 84.02 Mt.

3.1.6. The Barapukuria coal deposit is presently being worked by underground mining method for a rated production capacity of 1 Mtpa by Barapukuria Coal Mining Company Ltd., a subsidiary of Petrobangla. The mine is being operated by consortium of China National Machinery Import & Export Corporation (CMC) on contract basis. Presently only the VI seam (which is thick seam) is being worked in the mine by mechanized longwall technology with caving.

3.2. Summarized Parameters from the Approved Project Report

(Project Report - Modification of Basic Design for Barapukuria Coal Mine Project – Petrobangla, Bangladesh, Prepared by China National Machinery Import and Export Corporation, April 2000)

Mine Introduction

Coal Field Boundary

3.2.1. Barapukuria Coal Field has the bottom wall of fault F_b as its southern boundary, seam VI subcrop as its northern boundary, fault F_a as its eastern boundary and the bottom wall of fault F_b south of the exploratory line 2 and the seam VI sub crop north of the exploratory line 2 as its western boundary.

Designed production capacity

Working schedule

3.2.2. Present working system follows 365 working days with 4 shifts a day operations. Of these shifts, 3 are production shifts while 1 shift is for preparation and maintenance which includes 18 net hoisting hours per day.

3.2.3. The overall productivity of the mine is 1.5 tonne/man shift.

Determination of designed mine capacity

3.2.4. The main factors considered in designing the capacity of the mine are:

- The main mineable seam VI is extra-thick and is covered by more than 100 m of heavy water bearing strata leaving effective mining area of about 3 Km^2. The division of mining districts and layout of working faces are largely constrained.
- The coal field features abundant high quality coal reserves, shallow depth, gentle seam pitch, stable and simple structure which create favorable conditions for fully-mechanized mining.
- As this is the first modern coal mine in Bangladesh, the work force is inexperienced and there are insufficient skilled workers. Hence, even though the coalfield has conditions to build a medium to large scale mine, the standard unit production capacity is limited.

Mine Development – Underground Mining

Technical parameters considered for mine design

3.2.5. The Barapukuria coalfield has a medium structure and stable seam. The overall structure is an asymmetrical syncline spread approximately NNW. There are 3 faults located in this coal field. Most of the faults are normal faults. The west wing fault is dipping eastward and the east wing is the boundary fault, extending deeper and further and dipping westward independently.

3.2.6. Seam VI is the main mineable coal seam with an average thickness of about 36 m. Seam VI is weak in strength and the water bearing strata make the hydro-geological conditions of this coalfield relatively complex.

3.2.7. In addition as the geothermal gradient is high and thus the development method and roadway layout adopted should favour the lowering of the underground temperature. Since the lithological characteristics of the seam VI roof are weak and the stability is poor, the main roadways of the mine should be located in the seam VI floor bed of which lithological characteristics are hard, more stable, fire-proof and allow a low level of long term maintenance.

3.2.8. The Teritiary Dupi Tila formation is heavily water bearing. Thus, it is difficult to use incline drifts for development and vertical shaft development was used.

3.2.9. Considering the level of water inflow shown in hydro-geological exploration, in order to efficiently prevent water flooding, the vertical shaft and an inclined development method had been proposed in the CMC report on Modification of Basic Design for Barapukuria Coal Mine Project, Petrobangla, Bangladesh (April, 2000).

3.2.10. The CMC report (April, 2000) selected a two shaft option, i.e. one as main shaft (air return) and the other as the service shaft. The shaft position was selected to be between boreholes DOB 8 and DOB 11.

3.2.11. Considering high underground temperature and the complicated hydro-geological conditions, a double-entry layout was suggested in the CMC report (April, 2000). This favors temperature control, ventilation and construction safety. The main haulage roadway is located in the relatively competent floor rock of seam VI after multi-option comparison.

Extraction of Coal Seam - upper limit

3.2.12. The mining operation of this coal field was proposed to be carried out in the extra-thick seam under an abundant aquifer. For safe mining under water bodies a reasonable mining upper limit needs to be selected. The reasonable mining upper limit was determined based on the following factors:

- **Hydro-geological conditions:** The main threat to mine workings in this coal field is from the water in the Teritiary water abundant aquifer penetrating the goaf areas. The aquifer is the main factor determining the mining upper limit.

- **Physio-mechanical characteristics of seam VI roof rock bed:** The roof sandstone is a soft-medium hardness rock bed and has a lower tensile strength and a higher deformation resisting ability, but upon its encounter with water, its strength decreases.

- **Technical parameters of mining:** The cutting height of the first slice is an important factor in influencing the calculation of the two-zone height. As per the CMC report (April, 2000) a special study was conducted by relevant specialists and the number of slices for the seam VI was fixed at 8, with an accumulated mining height of 24m.

3.2.13. The CMC report (April, 2000) suggested that the mining thickness of the first slice to be mined during the initial period be limited to 2.5m. After gaining experience the mining thickness of successive coal slices can be increased to 3m.

Mode of Development

3.2.14. Based on the geological structure, hydro-geological and strata conditions and the existing conditions at the Barapukuria deposit, two options were considered for underground development. After long deliberation, demonstration and based on the opinion of experts, the CMC Report (April, 2000) recommended the development of Barapukuria Coal Mine in the following manner:

- The south belt roadway and track roadway to the west of fault F_b together with north roadway at east of fault F_b will be established at the -260m level after shaft is completed.

- The coal field is divided into 2 mining districts i.e. No.1 mining district and No.2 mining district.

- During the initial period the -260m south roadways, dip entries and inclines will be used for development of No.1 mining district.

- Later the -260m north roadways and rise (dip) entries will be used for the preparation of No.2 mining district.

Excavation Sequence

3.2.15. It was proposed that No. 1 mining district would be mined first followed by mining of No.2 district only once the conditions for mining have matured. In general the longwall face combined with the single entry mining method will be adopted for mining of seam V.

3.2.16. The proposed excavation sequence is summarized below:

Excavation Sequence	Area to be Excavated	Method of Mining	Production Capacity	Service Life
1.	Seam VI - No. 1 District	Longwall face extraction	1000 Kt/annum (Kt/a)	50 Years
2.	Seam VI – No. 2 District	Room and Pillar	600-800 Kt/annum (Kt/a)	15 years
3.	Seam V	Longwall face combined with the single entry mining method	200-300 Kt/annum (Kt/a	10 Year

Table 2: Details of Excavation Sequence Mining Method

3.2.17. The mining method for mining of Barapukuria coal deposit was selected keeping in mind:

- Presence of thick and heavy aquifer above coal seams
- To control subsidence of the overlying strata
- Proneness to spontaneous combustion of coal and goaf fire control
- Medium standard of mining and driving mechanisation to suit Bangladesh.

3.2.18. CMC report (April, 2000) compares two options of longwall mining, one with longwall mining along the strike and the other is inclined longwall mining. After discussion with experts, it was established that longwall mining along the strike is more suited to the geological conditions of Barapukuria deposit due to advantages of practical development layout, flexible mining sequence and convenient drainage mechanism.

3.2.19. The CMC report (April, 2000) suggested the implementation of the method of inclined slice longwall along the strike with caving and a metal mesh artificial false roof. The caving method provides advantage of high production and high efficiency. The sand – flush filling mining method can minimize the roof and surface subsidence but it was not selected as filling material is not available locally and would be expensive.

3.2.20. The CMC report (April, 2000) suggested that retreat type of longwall mining method should be followed. The advance per cycle was estimated at 0.6m with 8 cycles per day resulting in 120 m advance per month on an average during normal production period. It was estimated that the coal recovery in the mining district would not be less than 75% and coal recovery in mining face not less that 93%.

3.2.21. Seam VI is only planned to be mined initially and the mining of other seams will be considered only after the mine is in production and proved to be safe. Seam VI is divided into 7 natural slices in descending order named as slice A, B, C, D, E, F and G which will be mined by inclined slicing longwall along the strike.

Layout of mining district

3.2.22. The first mining district is located in the south of the central part of the coal field, where thickness of the LDT is greater than 10m.

3.2.23. The first coal face is numbered as 1101 and is located at the upper end of the south wing. The elevation of the track gate is above -257m so the water inflow from the gate and the faces can flow freely via the south track roadway down to -260m pit bottom sump. CMC report (April, 2000) suggested face 1101 to be mined first.

3.2.24. The second coal face (face no. 1110) was proposed to be located at the deep end of the north wing in No.1 mining district, near borehole CSE14. Face 1110 employs the retreat mining method but advances away from the main dip roadways.

3.2.25. The ventilation type in each of the mine face is ‘U’ type ventilation. Local fans (smaller capacity fans) were proposed to be used in the headings.

Face equipment selection

3.2.26. CMC report (April, 2000) proposed use of a double ended ranging drum shearer on each face to mine the coal. It proposed Coal loading with a combination of the drum helical scroll and the conveyor ramp plates. Loaded coal to be transported along the face by flexible scraper conveyor.

Haulage

3.2.27. The CMC report (April, 2000) recommended Belt conveyors for underground coal haulage. Auxiliary haulage involves fixed-case mine cars (1T) hauled by battery locomotive (8T). The belt conveyor system was selected due to comparatively small mine area, a super-thick coal seam, haulage requirements and short haulage distances.

Roadway Drivage

3.2.28. CMC report (April, 2000) suggested a semi-circular arch shaped cross-section for roadways and roads in the middle of the district while flat and a trapezoidal shaped cross-section for gate roads. Rock bolting and shot-crating was recommended for rock roadways and middle flat roads, steel support for gate roads and concrete arch support for chambers with a large cross section.

Roadway drivage rates

3.2.29. CMC report (April, 2000) estimated the monthly roadway advance rate as provided below:

Type of Roadway	Advance Rate
Horizontal heading in rock roadways	60m/month
Rise heading in rock roadways	55m/month
Dip heading in rock roadways	50m/month

Horizontal heading of an in-seam road by road header	300m/month

Table 3: Roadway drivage rates in Barapukuria coal mine

3.2.30. A total of 5 off headings including 3 off in-seam and 2 off rock heading were planned to ensure normal mine development and the preparation of the mining faces.

Selection of roadway heading machines

3.2.31. The roadway heading machines were selected considering following conditions:

- Geological Conditions
  - Floor conditions
  - Coal or Rock hardness
- Cross- sectional area
- Roadway curve radius
- Roadway slope

3.2.32. Based on the analysis of the above factors the CMC report (April, 2000) recommended the use of the model ELMD-75A in-seam heading machine. The technical specifications of the in-seam machine recommended are as follows:

Parameter	Remarks
Cutting capacity	$80m^3/h$
Drivage cross section	6-16 m^2
Cutting hardness	f<5
Working slope	±120
Ground pressure	0.14 Mpa
Minimum turning radius	6.5m
Total power	130KW

Table 4: Technical specifications of model ELMB-75A in-seam heading machine

Selection of bridge belt

3.2.33. The CMC report (April, 2000) recommended use of bridge belt of type JZP-100A with the following specifications:

Parameter	Remarks
Conveying capacity	200 t/h
Conveying length	16.5m

Belt width	650mm

Table 5: Technical specifications of bridge belt type JZP-100A

Transport equipment selection

3.2.34. The CMC report (April, 2000) recommended use of two-way dual purpose extendable belt conveyor type SJ-8011 with the following:

Parameter	Remarks
Capacity	200 t/h
Length	1000m
Width	800mm
Capacity for carrying 1 steel	2 pieces/min
Power	2 x 40 KW
Voltage	660/1140v
Speed	2 m/sec

Table 6: Technical specifications of two-way dual purpose extendable belt conveyor type SJ-8011

Equipment selection for drilling and blasting headings

Drilling machine selection

3.2.35. Drilling machines in rock roadways include three types pneumatic rock drill, hydraulic rock drill and electric rock drill. The CMC report (April, 2000) recommended the YTP26G pneumatic rock drill for rock roadway drivages and AS-12 wet type coal drill for in-seam headings.

Loading and transport equipment

3.2.36. Based on the coal mining experience in China and the auxiliary haulage transport method, the CMC report (April, 2000) recommended the usage of scraper rock loader, mine cars (1t) and a dispatching winch for the loading and transporting operation.

3.2.37. The ratio of mining to driving, drivage ratio and waste rock ratio is summarised in the following table:

Parameter	Remarks
Ratio of mining to driving	1:2.5
Drivage ratio	18.67 m/Kt
Waste rock ratio	7.5%

Table 7: Ratio of mining to driving, drivage ratio and waste rock ratio

Ventilation

Ventilation System

3.2.38. The Barapukuria coal mine is characterized by low gas emission, high geothermal gradient and the coal is prone to spontaneous combustion and dust explosion.

3.2.39. The CMC report (April, 2000) suggested the implementation of an exhaust ventilation system on the grounds of reduced capital investment, economic results, less occupied land, less residual coal and ensuring safety and production.

3.2.40. Fresh air to flow from the service shaft, via pit bottom, main track roadway, mining district track dip entry and belt gate to the working faces and driving faces. The used air to return from the working and driving faces via the belt gate, mining district air return dip entry, main air return roadway and is finally exhausted from the main shaft.

Ventilation for heading and chambers

3.2.41. The CMC report (April, 2000) recommended a forcing system of ventilation for driving. According to the driving section, distance and temperature control of the driving face, local axial fans are selected and should be installed in the intake air stream.

3.2.42. The CMC report suggested a separate ventilation system for the underground explosives store with the return air flow directly entering the main air return roadway. Other electro-mechanical chambers should be supplied with sufficient fresh air or designed with an intake air supply.

Temperature control measures

3.2.43. The Barapukuria coal mine faces acute thermal problems due to high rock surface air temperature, underground rock heat, high thermal gradient, oxidation heat, hot water and mechanical heat from electro-mechanical equipment.

3.2.44. The CMC report (April, 2000) has suggested the following temperature control measures:

- Use the dip developing layout with the main air intake roadways in the shallow part of the mine avoiding the high temperature zone of the deep part and shorten the length of the air intake roadways.
- A separate air return is to be used as far as possible for large electro-mechanical equipment chambers avoiding the heat from the equipment entering the intake air flow.
- Increase the air quantity for the faces and headings to a standard of $15m^3/s$ and $6m^3/s$ respectively.
- Adopt double-entry ventilation, reduce long ventilation distances and minimize the air loss.
- Stop off coal face districts after mining for the prevention of spontaneous combustion and reducing the temperature.
- Adopt ascentional ventilation for coal faces, where the conditions are suitable, avoiding the heat from electro-mechanical equipment in the gate entering the intake air flow and increasing the air temperature on the working face.

- Spray insulation material on coal roadways to reduce the oxidation heat.
- Propose a mechanical mine cooling system whose design document should be prepared separately.

Disaster prevention and safety device

Prevention of gas explosion

3.2.45. The CMC report (April, 2000) suggested following measure to prevent gas explosion:

- Adequate ventilation to quickly disperse the accumulated gas.
- Adequate gas monitoring and control.
- Prevention of ignition/gas ignition and explosion from mining.

Prevention of dust explosion

3.2.46. The main measures to prevent dust explosion are the prevention of airborne dust, deposited dust re-entering the air stream in explosion, isolation of igniting sources and controlling the propagation of an explosion.

3.2.47. The CMC report (April, 2000) suggested the following measures to reduce dust production:

- Water injection, water spraying and other dust suppression measures.
- Fitting effective cutting and external sprays to mining machinery
- Use of wet drilling
- Fitting water sprays to each transfer point.
- Cleaning the walls of the roadways with water and white washing at regular intervals.
- Reasonable air flow velocities.

The CMC report (April, 2000) also suggested the common measures to control ignition sources and prevent the propagation of dust explosion.

Measures to prevent spontaneous combustion

3.2.48. The CMC report (April, 2000) suggested the following measures to prevent spontaneous combustion.

- **Mining Technique:** Select a suitable mining method in accordance with the suitable stipulations, proper roadway layout and mining sequence to minimize residual coal pillars. After coal extraction the worked out areas must be sealed and grouted immediately.
- **Ventilation:** Select a proper ventilation system and for each mining district production should be carried out only when a separate ventilation system has been formed. Properly locating and constructing ventilation structures.

- **Fire-proofing with chemical agents:** As working faces advance, chemical solutions such as Calcium Chloride should be sprayed over the mined-out area and over the walls of cross cuts and connecting roadways.
- **Fire-proofing by grouting:** Due to the prevailing conditions and technical characteristics of slice mining yellow mud grouting is to be used to prevent spontaneous combustion and to form a secondary roof for the next slice with the metal mesh.

Emergency systems and measure

3.2.49. The CMC report (April, 2000) suggested the following emergency systems and measures:

- Air-flow reversal for the whole mine ventilation system
- Local airflow reversal of the mining district
- Installation of ladder ways in the main shaft and the service shaft for the emergency egress of the underground personnel.

Hoisting

3.2.50. The Barapukuria coal mine has a planned production capacity of 1 Mtpa. The underground access is via two vertical shafts, a main shaft and a service shaft. The mouth of both main and service shaft are at an elevation of +33.6 m. The main roadway haulage level at the shaft is at -260m.

Main shaft hoisting

3.2.51. The CMC report (April, 2000) has analyzed three design options for Main shaft hoisting as presented below:

Option	Description
Option I	Single rope hoist (Ø 3.5m) and skips (8t)
Option II	Single rope hoist (Ø 3.0m) and skips (8t)
Option III	Multi - rope hoist (Ø 2.8 * 4m) and skips (9t)

Table 8: Options for main shaft hoisting

3.2.52. The CMC report (April, 2000) suggested the adoption of Option 1 for Barapukuria coal mine i.e. 2JK-3.5/20B single-rope hoist, 8t skips, YPR800-10/1180 motor, hoisting speed of 5.4m/s and hoisting capacity of 1240kt/year.

3.2.53. The design parameters for the main shaft are as presented below:

Parameter	Remarks
Designed production capacity	1.0 Mtpa
Shaft mouth elevation	+33.6m
Loading level	-267.023m
Unloading level	+46.0m

Working system	300 days per year
Hoisting time	14 hours per day
Hoisting Container	8t skip (steel guide), load weight Q=8000kg, dead weight Qc=5000kgt

Table 9: Design data for main shaft hoisting

Service shaft hoisting

3.2.54. The CMC report (April, 2000) has analyzed three design options for service shaft hoisting as shown in table below:

Option	Description
Option I	Ground mounted multi rope hoist (Ø 3.25m * 4) and double-deck four car (1t) cages
Option II	Tower type multi rope hoist (Ø 3.25m *4) and double-deck four car (1t) cages
Option III	Single rope hoist (Ø 3.5m) and double-deck four car (1t) cages

Table 10: Options for Service shaft hoisting

3.2.55. The CMC report (April, 2000) suggested the adoption of Option 1 for Barapukuria coal mine i.e. JKMD-3.25*4 ground mounted multi rope hoist and a pair of double deck four car (1t) cages, including one wide cage and one narrow cage, two decks for man riding and one deck for hoisting waste rock.

3.2.56. Planned maximum hoisting speed of 7.4 m/s, maximum operation time for a shift is 2.86 hours and the travelling time for workers going underground in a shift is a maximum of 32 minutes.

3.2.57. Design parameters for the service shaft are shown below:

Parameter	Remarks
Shaft mouth elevation	+33.6m
U/G haulage level	-260m
Maximum number of personnel in a shift	627
Waste rock	90t/shift
Support	6 supply cars/shift
Equipment	6 runs/shift
Explosives	2 runs/shift
Pit wood	10 cars/shift
Sand, gravel and cement	6 cars/shift
Metal mesh	6 supply cars/shift

Hygienic car	2 runs/shift
Others	8 runs/shift

Table 11: Design data for service shaft hoisting

Mining district drainage

3.2.58. Water from the faces and goafs flows by gravity to the track dip entry or down to the rock mother road via the drainage boreholes. It then flows to the -430m track cross-cut via the lower flat of mining district or via the drainage boreholes and to the -430m sump where the water can either be pumped up to the -260m level or discharged directly up to the surface.

Pumping

3.2.59. The CMC report (April, 2000) recommended a combined system of direct and indirect drainage.

3.2.60. It suggested that the existing -260m pumping system be retained and used keeping 5 off sets of pumps and 3 off sets of pipelines in the service shaft unchanged. 4 sets, type D500-57 x 6, to be installed in their original positions and one additional pump to be installed in the spare position.

3.2.61. A new pump station at the -430m level was proposed to be equipped with 8 off sets of pumps. Of the 8 sets, 5 sets are type D500-57 x 4 capable of discharging most of the water from the -430m level to the -260m sump. 3 off pipelines, ∅ 325 x 9 mm will be installed along the drainage incline.

3.2.62. The other 3 pump sets, type D500-57 x 9 are capable of discharging the remainder of the water directly up to surface. 2 off drainage pipelines, ∅ 325 x 14mm, proposed to be installed to surface via the drainage incline and the main shaft.

Power Supply

Power Source

3.2.63. As per the CMC report (April, 2000), the mine shall be provided with power from RANGPUR and STAIDPUR power stations via 2 off 33kv power transmission lines, one from each station.

3.2.64. The main contract specifies that the contractor will build a power generating station equipped with 2 off 2.0 MW diesel generators, acting as emergency power supply.

3.2.65. However, only one 33kV power line from RANGPUR, via the hard rock mine to the mine has been constructed since the commencement of construction in 1997. In the CMC report (April, 2000), BCMCL was requested to ask the appropriate authority of Government of Bangladesh to consider establishing a new 132/33kV substation in the area to supply power directly to coal mine and the adjacent hard-rock mine.

Power transmission and transformers

3.2.66. The 33kV substation has been established at the north-east side of the mine site. It is equipped with the main transformers, 2 off 10 MVA and one incoming power transmission line from Rangpur. The diesel generator station with 2 of 2.0 MW generator sets is located nearby.

Surface coal handling System

3.2.67. The coal from the mining district was planned to enter no. 2 underground coal bunker, with diameter of 7m. Coal feeders, type K-4, underneath the bunker to feed coal onto transfer belt to No. 1 bunker (diameter 5m). No.1 bunker to feed coal onto two sets of parallel belt conveyors, which are 1000mm wide.

3.2.68. The coal was then proposed to be fed into skip weighing and loading equipment. The transport capacity of each conveyor is 450t/h, which can meet the requirements of the maximum hoisting capacity of 325t/h. The total effective volume of No.1 and No.2 bunkers is 800 tonnes.

3.2.69. A coal feeder type k-4 was proposed to be fitted beneath coal receiving bunker. A belt conveyor was proposed to be used to carry raw coal to a silo. ROM coal was planned to go to transfer point (tower) via a belt 1000mm wide and thence to the silo via a silo belt 1000mm wide.

3.2.70. The capacity of the belt conveyor is compatible with the mine capacity of 1 Mtpa. The designed conveying capacity is 286 t/h and the actual capacity can reach up to 325 t/h.

Surface Transportation

3.2.71. The roadways linking the mine site with the national highway was proposed to be designed and constructed by Petrobangle. The coal produced from the mine was to be mainly supplied to the adjacent power plant. About 250,000 tonnes of coal was planned to be dispatched by railway. Coal supply to other nearby areas was proposed by trucks.

Manpower

3.2.72. The CMC report (April, 2000) estimated the total man power (men on books) requirement of 3,086 staff and workers. The break-up of the man power requirement is as follows:

Item	Personnel	Total	Payroll factor	Men on Books
1	Production	2022		
(1)	Underground	1567	1.35	2115
a	Face	538		
b	Heading	334		
(2)	Surface	455	1.25	569
2	Management	200	1.00	200
Total Production		2222		2884
3	Service	144	1.00	144
4	Others	58	1.00	58
Total		2424		3086

Table 12: Break up of Manpower

Technical and Economic Parameters of Barapukuria Coal Mine

3.2.73. The main technical and economic parameters of the mine are summarized in the table below:

Name	Unit	Index
Capacity		
○ **Yearly**	Kilo tonnes	1000
○ **Daily**	tonnes	3333
Reserves		
○ **Geological**	Kilo tonnes	301810.8
○ **Industrial**	Kilo tonnes	154722.1
○ **Mineable**	Kilo tonnes	84020.7
Service life No. 1 Level	Year	64.6
Working Schedule		
○ **Working face production days/annum**	Days	300
○ **Shifts/day**	Days	3
Coal Quality		
○ **Grade**	%	Weakly coking coal
○ **Ash**	%	14.31-19.9/16.25
○ **Volatile Matter**	%	35.32-37.9/36.71
○ **Sulphur**	%	0.35-0.58/0.52
○ **Moisture**	%	2.94-3.75/3.19
○ **Calorific Value**	MJ/kg	26.77-25.58/27.71
Usage		Power station/domestic Coal
Coal seam condition		
○ **No. Of mineable seam**		2
○ **Total thickness of mineable seam**	m	Seam VI 29.4 -41.0/36.4
○ **Dip of Seam**		6-17°
Coal Field		
○ **Strike (Length)**	Km	4.9
○ **Slope Width**	Km	0.3-1.90
○ **Area**	Km^2	6.68
Access Mode		Shafts and Incline

Elevation	m	-260
○ **Auxiliary Level**	m	-430
Section		
○ **No. Of Section**	Nos	1
○ **Fully Mechanised face**	Nos/m	2/225
○ **No. Of Headings**	Nos	5
○ **Mining Method**		
○ **Mining Machinery**		
Main roadway haulage mode and equipment		
○ **Coal Haulage**		Belt Conveyor B=1000m
○ **Auxiliary Haulage**		Battery loco & mine car
Hoisting Mode and Equipment		
○ **Main Shaft**		2JK-3/20B Single rope hoist
○ **Service Shaft**		JKMD-3. 25*4 Ground multi rope winder
Ventilation Mode and Equipment		
○ **Gas classification**		Low gas
○ **Air Quantity**	m^3/s	110
○ **Negative Pressure**	pa	1106
○ **Fan no. & type**	2 sets	GAF-26. 6-13.1.1
Water drainage mode & Equipment		
○ **Normal Flow**	m^3/h	1200
○ **Max. Flow**	m^3/h	1600-1800
○ **Pump No. & type**		D500-57 *6 5 sets, D500-57 *4 5 sets, D500-57*9
Compressed air equipment		HPY18-10/7-K 5 sets, HPY-3/6 2 sets
Power Supply		
○ **Total capacity of mine transformers**	KVA	2*10000
○ **Annual Electricity Consumption**	MWh	46
○ **Electrical Consumption per tonne**	kWh	45.97
Total areas of buildings and structures on mine site	m^2	30644.8
Mine floor areas		

○ **Mine site**	m²	
○ **Temporary dump site**	m²	
Construction Period	Month	48
Men	Person	3086
Productivity		
○ **Overall Productivity**	Tonne/manshift	1.5
Total investment	10000USD	19575
Investment per annual tonne	USD/t	195.75

Table 13: Technical and Economic indices of the Barapukuria Coal Mine

Limitations of the summarized parameters of Approved Project Report

3.2.74. To fulfill the objectives of the works covered under the scope of this report, field visits were done by Consultant's team and discussions were held between Consulting team and the management of Barapukuria mine, representatives of CMC-XMC consortium working at the mine and IMC (BCMCL consultant) team posted in the mine so that all the information related to the geological, hydro-geological, physico-mechanical, mine planning and operational aspects which are necessary for such review work can be obtained expeditiously.

3.2.75. However, the complete set of information, particularly the geological plans and mine plans prepared by CMC/CMC-XMC consortium (except those available in A-4 size within reports) as well as performance data relating to the main mining equipment, certain important ventilation data and some other important data could not be obtained. The discussions held in the mine between consulting team and CMC-XMC team also yielded very limited result. Further, any technical report or details on implementation of LTCC method was not made available including second M&P Contract signed between BCMCL and CMC-XMC consortium. Therefore, the present review report has been prepared based on the limited available information. Findings and recommendations of this report may vary if new information is brought to light and variation may be material. Further, this report has been prepared considering technical and operations parameters observed at the mine and brought in light during discussions. Thus, commercial contract between BCMCL and its Contractor has not been studied or reviewed and commercial arrangements agreed between BCMCL and its Contractor may prove to be limiting factor on implementation of some of the recommendations.

3.2.76. Further, BCMCL may note that the recommendations made in this report are based on the observations made by consulting team during site visits and information furnished by BCMCL team in writing or verbally during the site visits held till first week of February 2012 and thus any developments post that are not taken into account. Any information communicated verbally has been considered by consulting team as accurate and has not been verified independently.

3.2.77. Further, the implementation of any of the recommendations would also need to take into account the new M&P contract signed between BCMCL and CMC-XMC consortium which was not available with consulting team.

4. Review of Present Status of Mine

4.1. Mine development

4.1.1. A construction contract (BCMP-77) was signed between China National Machinery Import and Export Corporation (CMC) and Bangladesh Oil, Gas and Minerals Corporation (BOGMC) on 7th February, 1994 [BCMCL was formed later on 04.08.1998] for development of Barapukuria Coal Mine Project for a capacity of 1.00 Mtpa. CMC completed the basic design for the mine in 1995 and started mine construction on 01.06.1996.

4.1.2. Development of the main shaft was started in December, 1996 and completed in October, 1997 and that of auxiliary shaft was started in April, 1997 and completed in November, 1997. The two shafts were interconnected in December, 1997 and construction of the pit bottom completed. However, construction work suffered due to inundation of the mine on 05.04.98 during development of roadways in the north wing mining district. Though the mine was dewatered on 03.06.98, the construction restarted during the first week of August, 2000 after modification of the basic design of Barapukuria mine by CMC in April, 2000. Acceptance test run of four road header systems and two longwall faces (1110 and1101) were completed on 29th May, 2005. Mine construction was substantially completed on 30th June, 2005 along with development of the above two Longwall faces in seam VI.

4.1.3. No development work has been taken up for working the other coal seams lying above seam VI.

4.2. Extraction of longwall panel

4.2.1. A management, production and maintenance (M&P) contract was signed between BCMCL and consortium of CMC and Xuzhou Coal Mining Group Company Limited (XMC) on 4th June, 2005 for a period of 71 months at a contract price of USD 82.30 million. During contract term, a total production of 4.75 Mt was agreed to be achieved from seam VI of Barapukuria mine by mechanised longwall mining. Per contract, commercial production of coal started from 10.09.2005 and the contract ended on 10.08.2011. The total production achieved during the contract period was 3.651 Mt against the targeted 4.75 Mt.

4.2.2. The entire coal bearing area of the mine (6.68 sq. Km) has been sub-divided into three major parts as follows:

- The Northern part (2.81 sq. Km)
- The Central part (3.00 sq. Km)
- The Southern part (0.87 sq. Km)

4.2.3. At present, Central Part of seam VI (average thickness 36.14m) is being worked by longwall mining with caving.

4.2.4. The thick seam is proposed to be worked in multiple slices in descending order and presently the top most slice of 2.7m-3.0m height is being extracted.

4.2.5. A coal barrier of around 6m - 10m width is being left between adjacent longwall panels so that the barrier yields easily and complete caving of roof is facilitated.

4.2.6. The longwall panels have been laid out on the north and south sides of the central development roads driven in rock about 20m below the floor of seam VI. On the north side, the panels have been laid up to a line (Figure 1) where the thickness of the Lower Dupi Tila (LDT) formation is at least 10m as it was apprehended that a lower thickness of LDT will not be adequate to arrest inflow of water into the workings from Upper Dupi Tila (UDT) aquifer. On the south side, where the LDT formation has thickened, longwall panels of 1000 m-1200 m length were envisaged. However, in practice, the panels could not be developed up to the planned length due to adverse mine environmental conditions arising out of heat and humidity coupled with percolation of hot water (up to 47°C) from the overlying strata. Consequently, the mining area in the central sector got reduced to about 1.96 Sq. Km (Figure 2).

4.2.7. Extraction of longwall panels in the 1st slice (top most slice) of Seam VI started on 10.09.2005. Since then 12 longwall panels have been worked in the 1st slice of VI seam in the central sector - 5 nos. in the south side and 7 nos. in the north side. Out of these, 10 panels have already been mined out, one is under extraction (likely to be mined out by March, 2012) and one panel is sealed off due to fire.

4.2.8. The various important geo-mining and hydro-geological parameters of these longwall panels along with the tonnage of coal extracted have been provided in table below.

Production Year (From 1st July to 30th June)	**Production (In Mt)**
Up to June, 2004	0.091
2004-05	0.087
2005-06	0.303
2006-07	0.388
2007-08	0.677
2008-09	0.828
2009-10	0.705
2010-11	0.667
Total	**3.746 Mt**

Table 14: Year-wise Coal Production in the Mine since Inception

(Source: Annual Report 2010-11, page 33 of the English version)

4.2.9. It can be observed that mine never achieved rated capacity of 1.00 Mtpa till date.

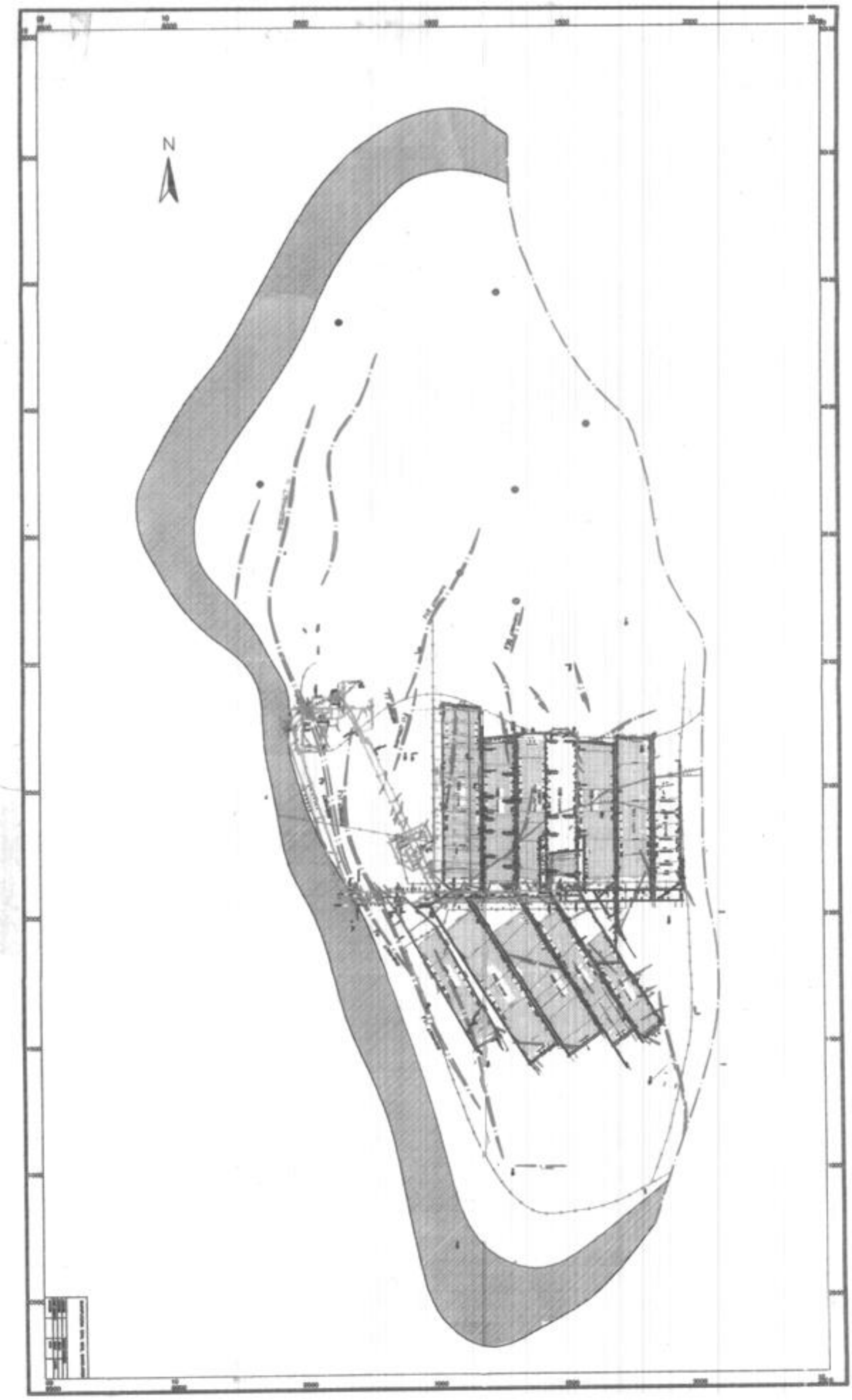

Figure 1: Location of Longwall Panels of 1st Slice in VI Seam

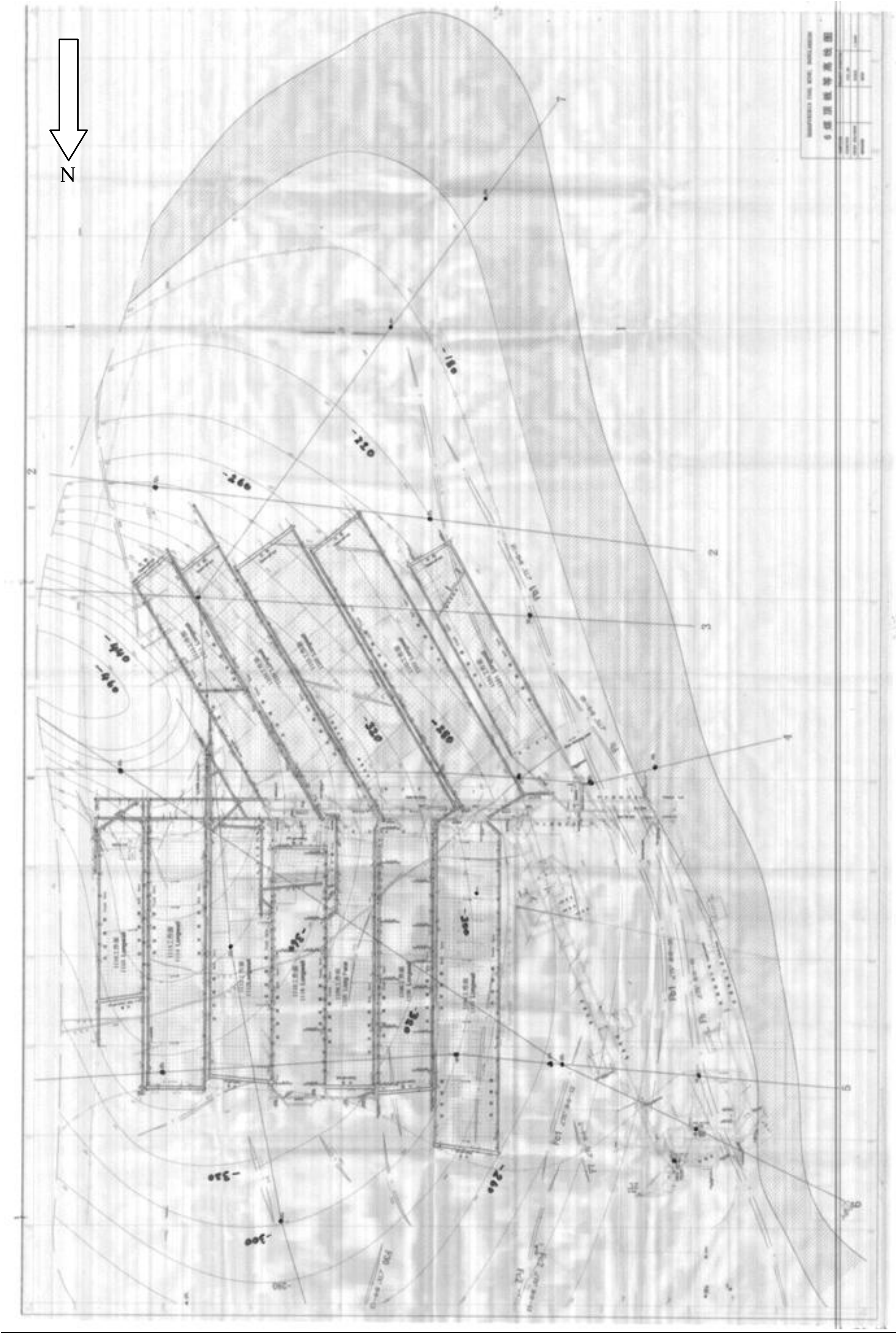

Figure 2: Mining Area in the Central Sector Showing Roof Contours of VI Seam

Panel Name (Arranged in sequence of Mining)	Dimension of Panel (m)			Start and Finish Dates	Total Production (Tonnes)	Retreat Per Day (m/day)	Water Inflow (cu.m/hr			Water Temp. (°C)	Remarks
	Panel Length	Face Length	Face Height				Initial Stage	Closing Stage	Present Stage		
1110	44.37	118	2.7	10.09.05 to 28.09.05	33452.05	3.74 (adv.)	36	28	05	37-41	Advancing face in the north side. Spontaneous Combustions occurred on 29.09.05 & the panel was sealed off on 14.10.05.Several attempts made to reopen have failed. Equipment recovered on 15.08.08 but the panel is still sealed off. Amount of coal left due to fire is estimated at 227799 t.
1101	415.9	105	2.7	18.11.05 to 28.02.05	185544.36	4.47	460	738	156	45-47	Retreating face in the south side. Mined out worked with wire netting at floor. Encountered 6 faults and a dyke.
1106	562.75	121	2.72	07.05.05 to 22.09.06	265116.03	4.05	60	123	36	35-36	Retreating face in the north side. Mined out worked with wire netting at floor. Encountered 2 faults and a dyke.
1109	558.54	106	2.68	07.03.07 to 19.11.07	226599.69	2.16	86	549	199	42-45	Retreating face in the south side. Mined out. Encountered 5 Faults. Face was highly inclined(23° to 27°) Coal left 22809 t.
1103	609.85	141.84	2.94	14.01.08 to 04.06.08	362045.57	4.26	83	295	75	37-41	Retreating face in the south side. Mined out. Encountered 6 Faults and a dyke. Coal left 76810 t.
1104	667.50	14.62	2.9	26.08.08 to 18.02.09	410166.12	3.83	108	121	62	33-37	Retreating face in the north side. Mined out. Encountered 3 Faults and a dyke.
1114	547.2	127	2.98	07.03.09 to28.07.09	299340.19	3.8	87	95	62	37-39	Retreating face in the north side. Mined out. Encountered 2 Faults and a dyke.
1105	560	165	2.9	12.08.09 to 16.03.10	382666.29	2.47	40	105	108	39-41	Retreating face in the south side. Mined out. Encountered 9 Faults and no dyke.
1108	550	108 86	2.9 2.9	25.02.10 to 09.11.10	237010.76	2.14	40	105	108	39-41	Retreating face in the north side. Mined out. Encountered 10 Faults and a dyke. Amount of coal left 15509 t due to fishing activities
1112	537.7	138.3 116.6	2.97 2.97	21.11.10 to 05.04.11	306895.42	3.98	10	65	65	40-41	Retreating face in the north side. Mined out. Encountered 2 Faults and 2 dykes. Amount of coal left 15478 t due to fishing activities
1111	542.65	98.5	3.00	22.04.11 to 01.10.1	219384.57	3.32	305	525	525	42-45	Retreating face in the south side. Mined out. Encountered 4 Faults and no dyke.

				1							
1116	385	105	2.98	24.11.11 (Working)	N.A.	-	87	95	62	37-39	Retreating face in the north side. Under extraction. Encountered 2 faults and 2 dykes. Panel shortened by 200m thereby losing 200000 t of coal. Last face in first slice
1204											Retreating face in north side under development in 2nd slice. 4 meter parting left below the goaf of panel 1104 of 1st slice

Table 15: Longwall Faces of Barapukuria Coal Mine – Geo Mining and Operational Parameters

4.2.10. The 1st slice of the Central region is exhausted. The development of the first panel (panel 1204) of 2nd slice of VI seam had started leaving a coal parting of about 3m below the 1st slice in the north side. Extraction of this panel is likely to commence from April, 2012.

4.2.11. To maintain continuous coal production from the mine and for ascertaining safety of the mine, a new M&P contract has been signed on 6th August, 2011 by BCMCL with CMC-XMC consortium. The new contract has commenced from 11.08.2012 and will remain effective for a period of 72 months. The new M&P contract provides for extraction of coal from 2nd slice of the Central part of the mine and includes, inter alia, the following (Source: Annual Report 2010-11, page 30 of the English version):

- Total coal production of 5.5 Mtpa over period of 6 years.
- Adoption of longwall top coal caving method (LTCC) for extraction of 6m thickness of coal in one lift.
- A new set of face equipment for longwall top coal caving method.
- Procurement of a 3 MW capacity generator.
- Procurement of 3 pump sets capable of lifting water from –430 m level and 2 additional pump sets.
- Replacement of two sets of high capacity motor for main fan.
- Drilling of 5 no. of geological boreholes in the south side of the central part of the coal basin.
- Installation of an additional deep tube well.

4.2.12. The 1st slice of VI seam has mined out area of about 1.90 sq. km, excluding an area of 0.06 sq.km in panel no. 1110 presently left out due to fire. The thickness of extraction of 1st slice varies from 2.7m to 3.0m at depths ranging from 250m to 450m. This has resulted in a subsidence of about 2.5 sq.km (627.73 acre) area on the surface with a maximum depth of subsidence trough of around 1.5m (As reported by BCMCL management during site inspections). No subsidence measurement has been taken up in the mine. The entire subsided area is now full of water up to the surrounding original surface level.

4.2.13. The first subsidence occurred on 11th May, 2006 in Kalupara and Balarampur villages. In the same area, second subsidence occurred on 7th May, 2008 (1101 and 1103 face area). Till date, a total of 592 families involving 3684 people, 276 nos. business enterprises and 276 nos. of landless poor/vulnerable/marginal families have been affected by subsidence. Compensation is being paid to the affected families as per approved compensation package.

4.3. Major production equipment

4.3.1. The approved Project Report provided for two sets of DERD Shearer with AFC, Stage loader and crusher and 184 sets of Powered Supports (including 10 sets of end supports) for operation of two longwall faces at a time. In addition to these, 4 sets of Road headers were provided for development of longwall panels.

4.3.2. The Shearer were commissioned in September, 2005. One Shearer was deployed in panel 1110 in the north and the other in panel 1101 in the south. But the longwall panel in the north (1110) was stopped on 29.9.2005 due to spontaneous heating in the panel. The panel was sealed off on 4.10.2005 and the Shearer, Powered supports and other equipment deployed in the panel were not salvaged. These equipment could be salvaged from the face on 15.8.2008 (i.e., after about three years) and were subsequently rehabilitated and reused. Also, out of the 4 road headers, only 2 are now in working condition and the other two have been reportedly cannibalised.

4.3.3. Due to unavailability of record/ data for equipment/ machine worked hours, maintenance hours, breakdown hours and idle hours for any equipment with the BCMCL management, only Shearer-wise production data have been compiled as provided in the following table:

Machine	Date of first deployment	Cumulative no. of days worked till 01.10.2011	Total Coal (Production in Mt)
Shearer No. 1	18.11.2005	1153 days	1.891
Shearer No. 2	10.09.2005	613 days	1.037

Table 16: Shearer Wise Production Data of Barapukuria Coal Mine

4.3.4. The average production/day of the shearer no. 1 and no. 2 work out to 1640 TPD and 1692 TPD respectively. This average daily productivity of shearer matches with the estimates made in the mine design report.

4.3.5. The shearer No. 1 worked for 1153 days out of 1784 working days achieving an utilisation of 65% and the Shearer No. 2 could work for 613 days out of 927 days achieving an utilisation of around 66%.

4.3.6. As per BCMCL estimate, total length of development achieved by the two road headers is 16723m (till 01.10.2011) resulting in production of around 0.239 Mt of coal.

4.3.7. Due to unavailability of data on the total number of days worked by the road headers, actual productivity of these machines could not be ascertained.

4.4. Underground mine environment

4.4.1. The maximum designed capacity of mine ventilation fan is 150 m^3/s. Though the air quantity circulated in main return is 120 m^3/sec. The intake air quantity is 105 m^3/s, out of which a regulated quantity of about 20 m^3/s is being sent to the longwall face being presently worked (No. 1116). The intake air quantity in the development panel (No. 1204) of the 2nd slice is about 17 m^3/s and the balance air is diverted to the sumps and other areas of the mine.

4.4.2. Mine is low on gas content with only 0.025 m^3/t of methane content in Seam VI. The gas percentage in the longwall return gate usually ranges from 0.02% to 0.5%. However, a gas percentage of as high as 1.34% has also been recorded in the air return junction (junction of longwall face and return gate road) of longwall panel.

4.4.3. The VI seam coal is very much susceptible to spontaneous combustion and the incubation period, as indicated by BCMCL management, is between 4-6 weeks. No scientific study has yet been conducted to determine the propensity of this coal to spontaneous combustion.

4.4.4. Concentration of CO is closely monitored in the mine and it is usual to find a CO concentration between 50 ppm to 100 ppm in the working areas, particularly in coal roof cavities caused due to fall. Due to the propensity of coal to spontaneous combustion, air velocity in the panel roadways had to be restricted resulting in reduced quantity of air in the face.

4.4.5. The usual ranges of ambient air temperature, air velocity and humidity as observed in some of the longwall panels are furnished below:

Longwall panel	Measurement date	DBT, °C	WBT, °C	Relative Humidity, %	Av. Air Velocity, m/s
1116	28.11.11	30.5-35	29.75-34.5	95-97	-
1111	22.03.11	27.5-36	26.5-36	93-100	2.67-3.63
1112	22.02.11	26.5-33.25	26.0-33	96-98	1.85-2.37

Table 17: Usual ranges of ambient air temperature, air velocity and humidity of Barapukuria Coal Mine

4.5. Mine dewatering

4.5.1. Average make of water in the mine is 1459 m^3/hr. There is almost no seasonal variation in the quantity of water inflow in the mine.

4.5.2. There are two sumps in the mine. The main sump is having a capacity of 7600 m^3 and is located at -430m level. There are three pumps (including one stand-by pump) each of capacities 500 m^3/hr x 513m head and five pumps (including two stand-by pumps) each of capacities 500 m^3/hr x 228m head installed in the main sump.

4.5.3. Thus, the total pumping capacity of the mine is 2500 m^3/hr against a requirement of around 1500 m^3/hr.

4.5.4. The other sump at -260m level has a capacity of 5500 m^3. This sump receives water from the discharge of three working pumps of capacities 500 m^3/hr x 228 m head installed in -430m level main sump. At -260m level a total of five pumps (including one stand-by pump) of capacities 500 m^3/hr x 342m head have been installed.

4.6. Power supply

4.6.1. Presently the mine is receiving power at 33 KV from Power Grid Company Ltd., Bangladesh (PGCB). A thermal power plant of 2x125 MW commissioned in February 2008 supplies power to PGCB.

4.6.2. Two transformers each of 10 MVA capacity have been provided for power supply to the Barapukuria mine which supply power at step down voltage of 6.3 kv. The actual power consumption of Barapukuria Mine is presently in the range of 7-8 MW

4.7. Mine safety

4.7.1. There has been some mine incidences involving fatal and serious injuries in the mine, particularly in the longwall panels. Year wise number of fatalities in the mine reported during last 10 years have been summarised below:

Year	Number of fatalities	Cause
2002	1	Roof fall
2004	1	Side fall
2007	4	Heat stroke (1), fall of objects (2), machinery (1)
2008	1	Electricity
2010	1	Roof fall

Table 18: Year wise number of fatalities in Barapukuria Coal Mine

4.7.2. Presence of fire and emission of noxious gases has been reported as the goaves of adjacent longwall panels are connected due to virtual absence of barrier between the panels.

4.7.3. Panel 1110 in the north side is under active fire. The isolation stoppings constructed in the panels in the rib pillars are neither explosion proof nor adequately sealed as these have been breached to allow escape of strata water from the goaved out panels.

4.7.4. The mine is also prone to bumps characterised by sudden fall of roof and sides. Several incidents of bumps have been recorded and therefore, the stress levels in the roof and sides are regularly measured by BCMCL's mining contractor and steps are taken to de-stress coal and rock strata.

4.7.5. The mine is highly watery and the average rate of pumping from the mine during the last four years works out to about 1470 m^3/hr. There had been one incidence of mine inundation in 1998 during development of roadways through faulted zone in the north-wing mining district, resulting in drowning of the pit bottom development workings. The mine was dewatered within two months time. The pumping system of the mine has since been strengthened to deal with the high make of water in the mine.

4.7.6. Based on the discussions, it is understood that Bangladesh does not have any mine safety regulations. BCMCL's Consultant and Contractor follow regulations from their country of origin. Thus, there is no standard official safety norm or practices that are required to be followed in mine but miner is following self framed norms. Also, the mine does not have any organization for maintaining safety.

4.7.7. Based on the above points 4.7.1 to 4.7.6, the mine is considered to be potentially unsafe, though the numbers of fatal accidents occurred are few.

5. Appraisal of mining conditions and identification of areas for further improvement

5.1. Exploration of the deposit

5.1.1. As per the CMC Report on Modification of Basic Design Parameters, in Barapukuria basin, out of a geological resource of 285.41 Mt of Seam VI in the main syncline, only 81.46 Mt (i.e. 28.54%) would be recoverable. The rest of the resources of Seam VI, which is the main seam of Barapukuria deposit, will be lost due to different design loses and mining loss.

5.1.2. Out of this recoverable reserve of 81.46 Mt, 64.80 Mt has been classified as proved reserve (111) and probable reserve (112). The balance 16.66 Mt has been classified as pre-feasibility resource (221 and 222).

5.1.3. In addition to the above, Barapukuria basin contains 40.24 Mt of indicated resource (332), 21.06 Mt of inferred resource (333) and 43 to 64 Mt of reconnaissance resource (334).

5.1.4. An appropriate mining technology need be developed to exploit the resource of VI seam in the 'open area' in the northern part of existing Barapukuria mine workings so that the estimates of recoverable reserve in this area can be firmed up.

5.2. Geology and Reserve

5.2.1. Barapukuria mine area has been explored by 33 boreholes drilled by different agencies as given below:

- By GSB (Initial Exploration): 7 boreholes of GDH series
- By Wardell Armstrong: 13 boreholes of DOB series
- By CMC (Supplementary Exploration): 13 boreholes of CSE series

5.2.2. In addition to the above boreholes, extensive geophysical surveys have also been carried out in the mine area.

5.2.3. The borehole densities in different seams in the mine on the basis of the Supplementary Exploration Report prepared by CMC are as follows:

Seams	No. of boreholes	Area (sq. km.)	Borehole density/ sq. km.	Remarks
VI	33	5.80	5.68	Area considered in modified mine design by CMC is 6.68 sq. km., giving a borehole density of 4.94/sq. km.
V	14	2.40	5.83	Area of the persistent split section of the seam (average thickness-4.78m) considered

IV	11	1.89	5.82	Area of the persistent split section of the seam (average thickness-8.82m) considered
III	5	1.26	3.97	Area of moderately persistent split section of the seam with average thickness of 1.59m considered
II	4	1.02	3.92	Area of the persistent split section of the seam (average thickness-14.44m) considered
I	1	-	-	Intersected in 3 boreholes out of which in 2 boreholes thickness is less than 0.5m and in 1 borehole 4.57m

Table 19: Borehole densities in different seams

5.2.4. In the Supplementary Exploration Report prepared by CMC, the area of VI Seam has been divided into ‘open window area’ i.e., the area where the Lower Dupi Tila (LDT) aquiclude is absent in the northern side of the deposit and ‘non open window area’ i.e., the area where the LDT aquiclude is present in the central and southern side of the deposit. However, in actual practice, the mine has been divided in three parts as discussed below. The number of boreholes in each of the these parts along with the borehole densities in the respective parts is given below:

Part of the mine	Area (sq. km.)	No. of boreholes	Borehole density /sq. km.	Remarks
Northern part	2.81	12	4.27	Northern part of the mine with LDT thickness of 10m or less. Not being worked at present.
Central part	3.0	20	6.67	Central part of the mine with thickness of LDT of more than 10m. This area is being presently worked in VI Seam
Southern part	0.87	1	1.15	Southern part of the mine. Not being worked at present.

Table 20: Number of boreholes in each of the above parts along with the borehole densities

5.2.5. Coal reserves under areas of influence of GDH series boreholes (done by GSB) has been treated by CMC as Rank ‘C’ reserves (i.e. with low geological confidence level) due to absence of geophysical logging, reliable borehole coordinates, coal analysis data and coal recovery data. Out of the total geological reserves of VI Seam around 60% has been categorized as Rank ‘C’ reserves in the Supplementary Exploration Report prepared by CMC.

5.2.6. The geological reserve and recoverable reserve by underground mining as estimated by CMC in its report on ‘Modification of Basic Mine Design’ are as follows:

Seam	Geological Reserve (Mt)			Recoverable Reserve (Mt)	Recovery %
	Rank A + Rank B	Rank C	Total A+B+C		

VI Seam open window area (below 0-10m thickness of LDT aquiclude)	55.95	79.23	135.18	16.66	12.3
VI Seam non open window area	58.37	91.86	150.23	64.80	43.1
Total VI Seam (5.8 sq. km.)	114.32	171.09	285.41	81.46	28.5
V Seam	-	16.40	16.40	2.56	15.6
Total	114.32	187.49	301.81	84.02	27.8

Table 21: Geological reserve and Recoverable reserve by underground mining as estimated by CMC in its report on 'Modification of Basic Mine Design'

5.2.7. Seam VI has already been worked in the central part of the mine and therefore, geology of the seam in this part of the mine is established. The northern part and the southern part of the seam are yet to be worked. The northern part of the seam is moderately explored but the southern part is under-explored. Further exploration of the seam is necessary to achieve the following objectives:

- Delineation of the trend of sub-crop of Seam VI.
- Up-gradation of reserves presently categorized under Rank 'C'.
- Firming up of the geological structure, thickness and quality of Seam VI occurring in the southern part of the deposit up to Phulbari exploration block.

5.2.8. Five new boreholes have been proposed by CMC–XMC consortium in the southern part of the mine as per the new M&P contract. A few more additional boreholes need to be drilled in this area to firm up the geology of VI seam in the south side of Barapukuria mine up to the boundary of Phulbari exploration block. Additional boreholes are also necessary in the northern and in the virgin area of central part of the seam to achieve the above objectives. New boreholes should not be drilled over or in the vicinity of already worked area of the seam. All new boreholes need to be geo-physically logged before plugging these holes effectively.

5.2.9. After completion of the above exploration job and analysis of coal cores, the geological plans and sections of VI seam have to be prepared afresh incorporating areas up to the boundary of Phulbari exploration block and geological reserves recalculated. The new geological plans should include floor contour plan, roof contour plan, isochore plan and isograd plan of the seam including the area between Barapukuria and Phulbari exploration blocks. Also, floor contours of the 1st slice workings of VI seam need to be drawn using the existing survey data to facilitate control of the level of working horizon of the 2nd slice.

5.2.10. The boundaries of different sectors of the VI seam have to be redefined and geological reserves should be estimated sector-wise based on the new geological plan to be prepared excluding the geological reserves already depleted in VI seam due to mining done.

5.2.11. The upper seams (seams I, II, III, IV and V) are also not adequately explored. Most of the areas covered by these seams are occurring vertically above the present mining area of VI seam. Exploration of these seams cannot be taken up at present due to following reasons:

- These seams will continue to subside further till the strata stabilises after completion of mining of VI seam. Therefore, it will be better to explore these seams after depletion of VI seam, to capture their final profiles.
- It will be unsafe for the drilling crew to work over unstable (subsiding) area and,
- There is possibility of jamming of drill column in the drill hole due to ground movement.

5.3. Hydrogeology of the Mine Area and its impact

5.3.1. This comprises mainly of defining the hydrogeology of the main lithological units and to predict/estimate the general underground water in-flows, as well as, management of DupiTila waters in Barapukuria coal mine area.

Hydrogeology

5.3.2. A large number of boreholes were made to investigate the hydrogeology of the area.

5.3.3. The drilling of investigation boreholes began in December 1989 and was completed in April 1990. This comprised of 23 observation wells and exploratory pump test boreholes to evaluate the geology, hydrogeology and geotechnical characteristics of the coal bearing Gondwana formation and the unconsolidated DupiTila formation.

5.3.4. Cores were obtained mainly from Gondwana formation and to a lesser extent from the DupiTila formation. Geophysical logging, packer testing, core analysis and pump testing were also included.

5.3.5. The lithostratigraphy of the areas showing the formations and their thicknesses is as follows:

Formation	Thickness (M)
Soils, Alluvium	0 – 1
Madhupur Clay	3 – 15
~~**Unconformity**~~~~~~~~~~~~~~~~~	
Upper DupiTila	95 – 125
Lower DupiTila	0 – 80
~~**Unconformity**~~~~~~~~~~~~~~~~~~~	
Gondwana	150 -500
~~**Unconformity**~~~~~~~~~~~~~~~~~~	
Basement Complex	Beyond 500

Table 22: Lithostratigraphy of mine area

5.3.6. The DupiTila aquifer system is one of the major groundwater reservoirs of Bangladesh. It consists of the Upper DupiTila sand aquifer (average thickness: 100m) and the underlying lower DupiTila clay aquiclude (thickness 0 to 80m – 0m in northern part and 80m in southern part).

5.3.7. The DupiTila aquiclude core analysis indicates permeability between 10^{-4} and 10^{-5} m/day. Madhupur clay aquiclude (3 to 15m thick) forms the upper confining layer to the sand aquifer.

5.3.8. Groundwater movement is towards south-west. Hydraulic gradients are low, and are of the order of 0.0005 and the groundwater velocities are on average 0.02m/day except near the pumping centres where the velocities are about 0.1m/day.

5.3.9. The rate of lateral flow is about 7 l/s per kilometre of aquifer width. The permeability varies from 15 to 20 m/day.

5.3.10. However in the Lower-more argillaceous part of aquifer, permeability are of the order of 6m/day. Specific yields are in the range of 25% to 30% and storage coefficient of the order of $4x10^{-3}$. Recharge is mainly by infiltration from the rainfall which varies from 370 to 660 mm/annum but where Madhupur clay is well developed it is low and is of the order of 280mm/annum.

5.3.11. The Gondwana formation is a poor aquifer and has a saturated thickness of 150m in the north and 500m in the south, and consists of feldspathic kaolinised sandstone, inter-bedded with coal, clays and mudstone.

5.3.12. Five coal seams were identified in the area, the deepest coal seam (Seam VI) being the thickest (36m on an average). The aquifer occurs in a structural basin which is truncated to the east by a major north-south fault. In the northern part of the area Lower DupiTila clay is absent and the Gondwana aquifer is directly overlain by the Upper DupiTila sand aquifer.

5.3.13. Groundwater levels in both aquifers are about 8m below surface during dry season which rapidly rise during the rainy season. Piezometric levels in the Gondwana are in general below those of the DupiTila.

5.3.14. Although during the dry season water level in DupiTila can be higher. Thus depending upon the time of the year, the groundwater can potentially flow from the DupiTila into the Gondwana and vice-versa. Hydraulic continuity between the two aquifers is greatest in the north where Lower DupiTila clay aquiclude is thin or absent.

5.3.15. In the south, where the aquiclude is 20 to 80m thick, the hydraulic connection is poor and possibly negligible. Chemical character of the groundwater of the two aquifers is similar. The measurement of pH value indicates the groundwater to be acidic to slightly alkaline in nature.

5.3.16. The groundwater movement is in south- east direction, towards the centre of the coal basin and appears to follow the broad structural trends of the basin. Hydraulic gradients are on an average 0.0002 and velocities 0.0007 m/day.

5.3.17. Due to anisotropic nature of the sandstone and coal, the permeability varies between 0.0001m/day and 0.4 m/day.

5.3.18. The coal horizons have been found to be generally more permeable than the sandstone. The recharge to the Gondwana is from downward vertical leakage from the overlying DupiTila and the average rate is under 0.2 l/sec per sq.km area. Inflows are between 1 and 30 l/sec depending upon the permeability.

5.3.19. Ground water movement within Gondwana deposit appears to be somewhat more complex than that within the DupiTila. In general groundwater elevation is higher in the north, corresponding with the area where Gondwana deposit and the coal strata sub-crop into the DupiTila sands. The major N-S fault to the east appears to act as a barrier to flow. The distribution of the piezometers was not extensive enough to confidently define groundwater flow in the southern part of the project area.

5.3.20. The piezometric observation of seam VI indicated the water level as -40 m (approx.) below mean sea level in year 2000 and as -290 m (approx.) below mean sea level in year 2008. This shows that the water level is gradually receding, which implies that there is no charging of Gondwana formation from the Upper DupiTila formation. The results after year 2008 were not made available by BCMCL.

5.3.21. Groundwater recharge occurs through infiltration of rain falls, flood waters, stream and canal flow and irrigation returns. It mainly occurs in-between May and July of the year, as indicated by the sharp rise of water levels during this period. After July, water levels remain stable suggesting rejection of recharge because the aquifer is filled to its full capacity. In the project area where the Madhupur clay is relatively thick, infiltration may be limited by the permeability of the clay. Infiltration tests on Madhupur clay have indicated a percolation loss of 1.5mm/day which, during the monsoon season, could amount to a vertical recharge of 280mm.

5.3.22. As the Gondwana aquifer occurs below the DupiTila aquifer system, it cannot receive direct recharge from rain fall. Therefore recharge to the Gondwana formation is indirect from the overlying Upper DupiTila aquifer. In the northern part of the project area where the Lower DupiTila aquiclude is absent, the Gondwana formation receives recharge by percolation from the overlying DupiTila sand aquifer. In the southern part, the DupiTila clay aquiclude prevents recharge to the Gondwana except in very minor quantities, by downward vertical leakage.

5.3.23. Water level fluctuations: Groundwater within the DupiTila aquifer system is close to ground surface throughout the year. Lowest observed natural groundwater level during March 1990 was approximately 8m B.G.L.

Estimates of underground water inflows

5.3.24. Permeability within the Gondwana is considerably less than that of the overlying DupiTila aquifer. Ground water within the coal seam will converge on the area of excavation and will cause increased movement towards the mine cavities including lateral flow along the coal seam.

5.3.25. **Lateral Seam Inflow:** The proximity of mine excavations to the area of seam sub-crop into the DupiTila deposit on the northern side will influence the volumes of groundwater inflow. Within the vicinity of the sub-crop of seam VI, the thickness of Gondwana formation overlying coal, approach a minimum. Furthermore, the seam is in direct hydraulic continuity with a large reservoir of groundwater which serves to maintain high Gondwana groundwater levels. The wide sub-crop of seam VI on the northern part provides a large surface area over which infiltration may be induced into coal strata. It is assumed that groundwater generally flows along the seam dip.

5.3.26. **Vertical Movement:** Groundwater flow will not be restricted to movement along the alignment of coal strata. A component of vertical flow will also occur towards the seam excavations.

5.3.27. The rate of movement will relate to the thickness and types of the surrounding lithologies, particularly those which overlie the coal. The compressional and tensional forces developed in response to mining, will cause modifications to the distribution and magnitude of permeability within the vicinity of the seam excavation. These types of permeability changes are related to detailed geotechnical properties of roof strata and excavation techniques.

5.3.28. Highest vertical seepage is expected in areas where the Lower DupiTila clays are absent and where only a thin cover of Gondwana sandstone occurs. These areas are also most susceptible to high induced infiltration from subcrop.

5.3.29. The estimates of vertical seepage are based on in-situ permeability of typical lithologies undisturbed by mining operations. The structural response of the sandstones to the underlying excavations is critical to the actual permeability and discrete flow paths that may be induced. Should a direct communication with the DupiTila sands is developed through tension cracking, mine inflows will be considerably higher than the estimated quantities.

5.3.30. **Inflows during mining:** Under natural conditions groundwater is introduced into the coal seam by infiltration at the sub-crop with DupiTila sand and by vertical movement to the surrounding Gondwana host rocks. Drainage of mine excavations within the seam will cause a readjustment of a natural groundwater flow pattern.

The groundwater within the coal seam will cause increased movement towards the mine cavities including lateral flow along the coal seams. The groundwater flow will not be restricted to movement along the alignment of coal strata; a component of vertical flow will also occur towards the seam excavations.

The proximity of the mine excavations to the area of seam VI sub-crop into the DupiTila deposits will fundamentally influence the volumes of groundwater inflow.

5.3.31. The open window in the north is the main recharging area of the roof sandstone of seam VI. The aquiclude Lower DupiTila (LDT) thickness increases gradually from north to south. Coal measure itself is poor watery and is a poor aquifer.

5.3.32. The roof sandstone of seam VI has larger unit water inflow of 0.14 l/s in the north near the open window while in other areas where LDT aquiclude is present, the unit water inflow is less than 0.1 l/s. The underground water flow direction is the same as that of the water of UDT sand layer which flows from north-east to south-west.

5.3.33. Flows during the mining of seam VI have been tentatively estimated to be between 10 and 100 l/s. Flows were found to decrease with increasing distance from the coal seam sub-crop against the Upper DupiTila sand aquifer.

5.3.34. The estimates of inflow into the mine panels are tentative and do not take account of the time variance of flows or the mining induced changes of permeability of deposits overlying the coal excavations. For planning purposes particularly with regard to mine inflows, a margin of safety over and above the maximum estimated flows of three times or more should be considered.

Conclusions

5.3.35. In the long term and in the absence of serious structural weakening of roof strata, ground water flow towards the mined area is expected to comprise of induced infiltration at sub-crop, lateral flow to the coal seam and vertical movement through adjacent lithologies. In the initial stages of mining, there expected to be a general readjustment of ground water flow patterns in response to the localized ground water depression and drainage of the initially saturated lithologies.

5.3.36. The detailed pattern of the ground water flow that will eventually develop within the coal reserve and aquifers is expected to be complex, subject to the various boundary effects at the extremities of the seam. The hydraulic interrelationships between the aquifer systems, accurate assessment of the various patters of movement and of the volumes of ground water flow, as they vary in time and space, will require detailed numerical modeling. At present based on the simplified approach given above, total seepage into the panels in seam VI is expected to range between 10 and 100 l/s

5.3.37. In case opencast mining is done in the northern part of the deposit, huge quantity of water would be required to be pumped out. Radial flow calculations to estimate the pumping out volumes necessary to depress the ground water surface within the DupiTila towards its base to approximately 110m assumed an infinite extent of DupiTila. The volumes show that large ground water withdrawals (in excess of 8,000 to 10,000 l/s) will be necessary to dewater the DupiTila formation to its base over the area in which excavation can take place. The scheme would require more than 150 wells in the DupiTila, each pumping at about 60 l/s. This water could be utilised by the power plant located in the area, if found suitable after appropriate technical studies. However, the environmental impact of ground water depression and the possible land subsidence must be assessed and considered.

Management of DupiTila waters in Barapukuria coal mine area

5.3.38. In Barapukuria coal mine area the watery properties of the different aquifers become poor downwards. The upper sand horizon of Upper DupiTila (UDT) is highly watery while the lower one is moderately watery with an apparent decrease. The water levels were measured at 22 boreholes in the Barapukuria coal mine area by Wardell Armstrong in 1991 for the study of hydro-geological characteristics, water level fluctuation and hydraulic relationships between aquifers. The analytical results of the pumping tests data by Wardell Armstrong revealed that the upper and lower horizons of the UDT aquifer are thick and its recharge condition is good. The pumping test data analysis of sandstone of Seam VI roof and floor also reveals that transmissivity of the aquifer is good.

5.3.39. Rainfall is the main source of recharge to the UDT aquifer. Water levels during dry season and monsoon change as a cycle. Average monthly rainfall varies from 6 to 59mm in the dry season i.e. between the months of November and April. The water levels fall subsequently during this period and minimum elevations vary between +22.60m to +23.89m in the months of March and April.

5.3.40. On the other hand average monthly rainfall varies from 236 to 482mm in the monsoon season between the months of May and September and it is about 87% of total annual rainfall. During that time water level keeps rising and the maximum elevation ranges between +30.83m and +30.11m. Range of water level fluctuation is 7.04 to 7.70m.

5.3.41. **Measurement of discharge water from underground:** From year 2000 to 2002 some water came from water inrush point and some water came from deep inclined roadway, -430m level, and 1101 belt gate and track gate roadways. With respect to development and production from the mine, amount of discharge water decreases from inrush point and increases from development points of the roadways and production faces. Average monthly water discharge from 2005 to 2011 was recorded, and this varied from 1,000m^3/hr. to 1,300m^3/hr.

5.3.42. **Measurement of water table in the DupiTila formation:** From the beginning of Barapukuria coal mine development, the measurement of water table of DupiTila formation is going on in bore holes SHOB 4 and SHOB 6. The elevation of water table measured in the boreholes between years 2000 and 2011, varied from 19.74mto 23.57m in SHOB 4 and 20.61 m to 24.72 m in SHOB 6.

5.3.43. Measurement of water table in the Gondwana formation: Water table elevation in the Gondwana formation was measured in the boreholes CSE-14 and CSE-15 from year 2000 to year 2009. According to the measurements the water table elevation of the Gondwana formation varied from -27.87m to -361.85m in CSE-14 and from -35.36m to -297.92m in CSE-15.

Conclusion:

5.3.44. The principal results on the hydrogeology of the Barapukuria coal mine area and the source of underground discharge water, are summarised as follows:

- In relation to hydro-geological conditions of this area, Barapukuria coal basin has two major aquifers: one, the Upper DupiTila formation whose thickness varies from 102 to 136m and the other, Gondwana formation with average thickness of 360m.

- At the end of year 2003 when the mine development was nearly complete and preparations for production of coal from different faces was going on, the average water discharge was about 1,022.75m^3/hr. Maximum water came from inrush point, -260m level, deep inclined roadway, -430m level and 1101 belt gate and track gate roadways.

- At the end of October 2011 when first slice of all faces except 1116 completed, the average discharge water was about 1,480.04m^3/hr. Maximum water came from 1111, 1101,1105,1103 faces and mother rock road way. The rate of discharged water rapidly decreased from the above mentioned points except 1101 face.

- During the year 2000 when the development of this mine was going on, the elevation of water table of the DupiTila formation was 23.57m and 24.61 m above mean sea level, as measured in two separate observation boreholes, SHOB-4 & SHOB-6 respectively. In the middle of 2008, the water table of DupiTila formation was 19.74m above mean sea level (SHOB-4) and at the end of the year the borehole was sealed. In the middle of 2011, the water table of the formation was measured as 20.61m above mean sea level (SHOB-6).

- Also during the year 2000 when development of the mine was going on, the elevation of the water table of the Gondwana formation was -27.87m and -35.36m below mean sea level as measured in two separate observation boreholes (CSE-14 and CSE-15). At the end of 2011, water table elevation of Gondwana formation was measured in the same borehole sand was found to be -361.85m (CSE-14) and -297.92m (CSE-15). It indicates that the elevation of water is decreasing.

- The rate of lowering of water table from 24.09m to 20.17m, in DupiTila formation and from -31.61m to -329.00m in Gondwana formation, may clearly indicate that there is no connectivity between the DupiTila formation and Gondwana formation.

- It may be mentioned here that the main cause of drawdown of DupiTila water table (yearly average of 0.40m) is because of more discharge for irrigation and for other industrial purposes than recharge by infiltration during rainy season. It is the actual phenomenon not only in the Barapukuria mine area but in the whole of Bangladesh.

- Thus the source of underground discharge water is the connate water from large thickness of Gondwana formation. This water has been preserved within this formation since its deposition.

5.3.45. The water inflow from the strata into the mine is almost constant. The piezometric record shows that the water is gradually receding. Between the years 2000 and 2008, the water level receded from about -40 m below mean sea level to -290 m below mean sea level. This indicates that there is no increase in the water inflow into the mine.

5.3.46. The pumping records indicate that the quantity of water being pumped up is almost same throughout the year; which means, there is no seasonal variation in the water inflow into the mine. The water level recession is 0.3 to 0.5 m per year, as observed.

- The Lower DupiTila (LDT), an aquiclude, does not permit the vertical movement of water from Upper DupiTila (UDT) into the Gondwana formation, in the central part. However, in the northern part where the LDT aquiclude is absent, there is direct contact of the UDT with the Gondwana formation. Here, there is chance of lateral movement of the water through the Gondwana formation towards the central part of the seam where mining is being done.

- The Gondwana formation itself is full of water and is saturated, and therefore it offers resistance to water flow from UDT to Gondwana, and the quantity of water in Gondwana is expected to remain same even if it is getting drained out into the mine. The capacity of the Gondwana formation to hold the water in it is made up from the UDT. Thus, the quantity of water inflow to the mine shall remain almost constant; however, there may be increase in inflow when the seam meets any fault plane connecting the seam with UDT.

- The sandstone of seam VI roof is a watery aquifer. Pumping test indicates that specific discharge is 0.008 to 0.14 l/s.

- The average quantity of water discharged from underground workings of Barapukuria coal mine for the last 04 years is as follows:

Year	Quantity of water discharged (m3/ hr)
2008	1460.779
2009	1520.890
2010	1427.168
2011	1468.444

Table 23: average quantity of water discharged from underground workings of Barapukuria coal mine for the last 04 years

Average water discharge for the last four years is 1469.32 m^3/hr. (1470 m^3/hr. approx.). The actual measure provided by BCMCL, is 1459 m^3 per hour (about 1,500m^3/hr. or 36,000m^3/day). As such the pumping capacity is more than the water inflow of the mine. There is almost no change in the quantity of water inflow with seasonal variation.

- There are 3 nos. of 9-stage 1120 kW pumps installed at -430 m level; 2 running and 1 stand-by. Capacity of each one is 500 m^3/hr. These pumps pump the water via air return to -291 m main shaft pit bottom to surface (+33m level) through main shaft; maximum permissible delivery head: 513m.There are two discharge lines; the pipe dia. of each one is 325mm with wall thickness of 14mm.

- Another set of 5 nos. of 4-stage 500 kW pumps have been installed at -430 m level; 3(+1) running and 1 stand-by. Capacity of each one of these pumps is 500 m^3/hr. These pumps pump the water from -430 m pump house to -260 m level sump through three discharge pipe lines; the pipe dia. of each one is 299mm with wall thickness of 8mm; delivery head of 228m.

- At -260 m level, in total there are 5 numbers of 6-stage 800 kW pumps. Out of the 5 pumps, 1 is stand-by and the remaining 4 are meant for pumping. However, for 1 of these 4 pumps, pipe line has not been installed and thus there are only 3 pumps which are in operation. These 3 pumps pump water from -260 m level to surface (+33m level) via -256m level auxiliary shaft pit bottom. The pipe diameter is 325mm with 9mm wall thickness. The maximum permissible delivery head is 342m. Capacity of each pump is 500 m^3/hr.

- The sump capacity at -260m level is 5,500 m^3 and that for -430m level is 7,600 m^3. The daily water outflow from the mine is approximately 1470 m^3/ hr.

- The present total pumping capacity is 2,500 m^3 per hour whereas the Barapukuria mine is pumping only about 1,500 m^3 per hour.

5.3.47. To tackle the expected increase in water inflow during mining of the second slice by LTCC method, pumping capacity shall be doubled, and the pumping shall be done directly from -430m level to surface.

5.3.48. There are 9 faults discovered in the minefield. Out of these 3 are major faults with throw of more than 10 metres. All of them are normal faults, mostly along the strike and lie in N-S direction. Because faults in the minefield possess considerable water transmissibility, precaution should be taken while approaching these faults to avoid sudden inrush of water into the mine. Procurement of adequate numbers of long hole underground directional drilling machines for the mine is recommended for safely draining out water under pressure in advance whenever needed/planned.

Recommendations

5.3.49. Further detailed study is required through modeling, preferably numerical modeling or any other suitable method, to review the application of the LTCC method of mining to predict its impact on the overall stability of the mine and quantity of water inflow into the mine due to unstable conditions arising out of movement/caving of the overlying ground above VI seam. If it is expected that there might be a sudden inrush of water into the mine workings, adequate precautionary measures may be planned and implemented to safeguard the life and property in the event of such sudden inrush of water into mine workings.

5.4. Mining Methods

5.4.1. Report on 'Modification of Basic Design for Barapukuria Mine Project' prepared by CMC (April, 2000) envisages mining of VI Seam (29.40 m-40.52 m thick with an average of 36.14 m) by multi-lift Longwall mining with metal mesh artificial roof and full caving.

5.4.2. Stowing method has not been considered by CMC in view of non-availability of stowing material in the locality and high cost of stowing (page 68, Modification of Basic Design for Barapukuria Mine Project). According to the study undertaken by the CMC specialists, VI Seam will be extracted in 8 slices with aggregate mining thickness of 24m.

5.4.3. The mining thickness of the 1st slice was envisaged in the Modification of Basic Mine Design Report as 2.5m and the slice thickness was to be increased up to 3m after gaining experience at Barapukuria. It has been planned to work the seam first in the 'non-open window area' (central and southern part) where the thickness of the LDT aquiclude is more than 10m.

5.4.4. The northern part of the VI Seam has been designated as 'open window area' because the thickness of LDT aquiclude in this area varies from Nil-10m and mining in this part of the seam has been deferred till a suitable method is selected to work the thick seam below highly water bearing UDT aquifer.

5.4.5. The prime consideration in planning of the Barapukuria deposit has been prevention of the water inflow to the mining area from the UDT aquifer through the fracture zone created due to caving of successive slices of VI Seam. Accordingly, the limiting upper most level in VI Seam up to which the mining can be extended so that the lowest level of the UDT aquifer is not disturbed even after caving of 8 slices (24m) has been calculated by CMC and adopted in the report.

5.4.6. According to the stipulations and experiences of mining in China the height of coal/rock water barriers is obtained by following empirical formula:

$$Hsh = Hli + Hb$$

Where,

Hsh – height of water barrier (m)

Hli – maximum height of permeable fracture zone (m)

Hb – thickness of the protection layer (m)

5.4.7. According to the hydrogeological conditions, the physico-mechanical characteristics, seam dip angle and cumulative mining thickness (of slices mined), the maximum height of fracture zone Hli is calculated by the following two empirical formulas and the larger value is taken:

$$Hli = 100\sum M \div (1.6 \sum M + 3.6) + 5.6 \qquad (1)$$

$$Hli = 20(\sum M)1/2 + 10 \qquad (2)$$

Where, ∑M – cumulative mining thickness (m)

5.4.8. The applicable conditions of the above two empirical formulas are 1-3 m for a single slice cutting height and 15m of cumulative mining thickness.

The thickness of the protective layer (Hb) is calculated according to the following formula

$$Hb = 4 * \sum M/n$$

Where, n – slice number

5.4.9. Considering the difference between the geological and mining conditions in Bangladesh and Chinese experience, a buffer height Hh is added to the height for the water barriers for calculating the results. Hh has been taken as 30m in this design. Thus, the formula for water barrier height is:

$$Hsh = Hli + Hb + Hh$$

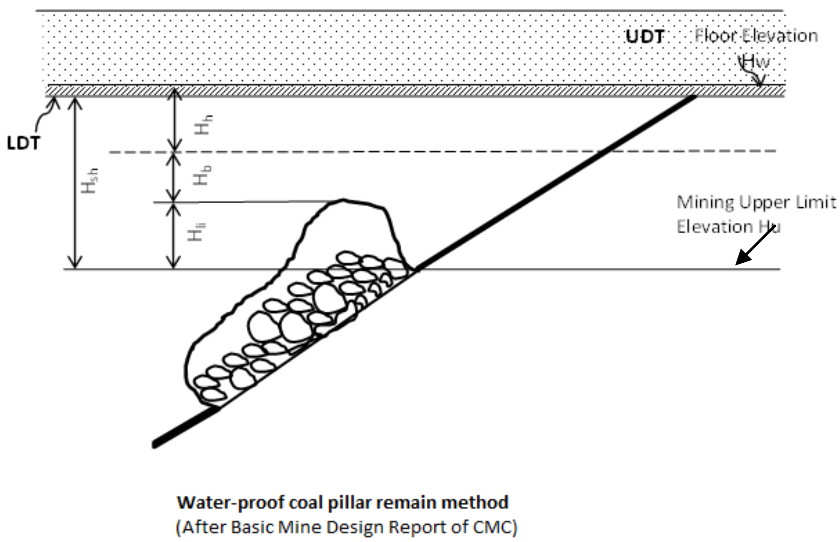

Figure 3: Water-Proof coal pillar remain method

5.4.10. The calculated results of the coal/rock water barrier heights according to the above formula and different cumulative mining thicknesses as given in the Basic Mine Design Report has been presented in table below:

Cumulative mining height (m)	Slice	Water permissible fracture zone height (m)			Protective layer, Hb (m)	Buffer layer, Hh (m)	Coal-rock pillar height, Hsh (m)
		Formula 1	Formula 2	Taken value			
2.5	1	38.5	41.6	41.6	10	30	81.6
5.0	2	48.7	54.7	54.7	10	30	94.7
7.5	3	53.7	64.7	64.7	10	30	104.7
24.0	8	62.7	108.0	108.0	12	30	150.0
30.0	10	63.7	119.5	119.5	12	30	161.5

Table 24: Water-proof Coal Rock Pillar Height Calculation

5.4.11. According to the calculated results, when the cumulative mining thickness is 24m (8 slices), the elevation of the mining upper limit should range between -162.5m to -256.3m. Table below gives the details of this estimate done by CMC using both finite element method (FEM) and empirical formulas method (EFM). The EFM results have been considered as this gives a higher thickness range of safe parting.

No.	∑M (m)	Hf (m)	Hw (m)	Hli (m)	Hum (m)	Hli - Hf	Hup

			(-82.5)				
M1	2.5	20	-84.7	41.6	-114.7	+21.6	-166.3
	5.0	37	-87.5	54.7	-165.3	+17.7	-183.2
	7.5	60	-90.0	64.7	-191.3	+4.7	-196.0
	24	100	-106.5	108.0	-248.3	+8.0	-256.3
	30	113	-112.5	119.5	-267.3	+6.5	-273.8
			(-81.2)				
M2	2.5	20	-83.4	41.6	-143.4	+21.6	-165.0
	5.0	32	-86.1	54.7	-159.1	+24.9	-182.6
(P2)	7.5	42	-88.7	64.7	-172.0	+22.7	-194.7
	24	110	-105.2	108.0	-257.0	-2.0	-255.0
	30	122	-111.2	119.5	-275.0	-2.5	-270.5
			(-82.5)				
M4	2.5	22	-84.7	41.6	-148.5	+19.6	-168.1
	5.0	40	-87.5	54.7	-169.3	+14.7	-183.2
	7.5	63	-90.0	64.7	-194.8	+1.7	-196.5
	24	80	-106.5	108.0	-228.3	+28.0	-256.3
	30	110	-112.5	119.5	-264.3	+9.5	-273.8
			(-79.9)				
M5	2.5	30 (1 face)	-82.1	41.6	-150.9	+11.6	-162.5
	2.5	38 (3 faces)	-82.1	41.6	-150.9	+3.6	-162.5
(P4)	5.0	40	-84.9	54.7	-163.7	+14.7	-180.9
	7.5	60	-87.4	64.8	-186.2	+4.8	-191.0
	24	120 (reaches LDT)	-103.9	108.0	-262.7	-12.0	-250.3
			(-82.5)				
	2.5	30	-92.3	41.6	-162.0	+11.6	-173.6
M7	5.0	50	-95.5	54.7	-196.5	+4.7	-192.0
	7.5	70	-98.0	64.8	-209.8	-5.2	-204.6
	24	120 (reaches LDT)	-114.5	108.0	-276.3	-12.0	-264.3

Table 25: Comparison of the calculation results of FEM and EFM (Reproduced from Table 2-3-4 of Basic Mine Design of Barapukuria Mine formulated by CMC)

Note: ΣM – the cumulative mined thickness

The maximum of damage zone heights obtained by FEM

H_w – the UDT bottom elevation

H_{li} – the maximum of WFFZ obtained by EFM

H_{um} – the UML determined by FEM

H_{up} – the UML determined by EFM

FEM – Finite Element Method

EFM – Empirical Formulas Method

UML – Upper Mineable Limit

WFFZ – Water Flowing Fractured Zone

Physico-mechanical properties of VI Seam and strata above VI Seam

5.4.12. The lithologies and physico-mechanical properties of main horizons of the strata section at Barapukuria mine area along with their significance are presented in the Table below:

Horizon	Thickness range (m)	General lithology	Physico-mechanical properties	Remarks
Madhupur Clay	4.30-15.90 (Avg. 10.72)	Mainly brown yellow sandy clay horizons	-	Surface layer. Negligible bending ability.
Upper Dupi Tila (UDT)	90.70-126.82 (Avg.104.41)	Medium sand beds, inter-bedded with fine sand, pebbly grit and thin clay horizons. The content of clay increases with depth	-	Unconsolidated and highly water bearing strata. No bending ability.
Lower Dupi Tila (LDT)	0-80.14 (Avg.28.88, generally 15m in the middle of the mine field)	Greyish mudstone, silty mudstone, quartz sandstone and inter-bedded with 1-2 lignite seams	UCS of sandstone ranges from 3-41 MPa (Avg. 15 MPa). UTS of sandstone ranges from 1-1.5 MPa (Avg. 1.3 MPa). UCS and UTS of mudstone are 19 MPa and 0.5 MPa respectively.	Poorly consolidated rock, easy to smash. Limited bending ability. Aquiclude.
Base of LDT to base of V Seam	0-110m	Quartso-feldspathic, medium and coarse grained sandstones, interbedded with occasional siltstones	UCS of sandstone ranges from 0.20-41.05 MPa (Avg. 11.58 MPa). UCS of mudstone is 18.72	The UCS of sandstones indicate weak to moderately strong rock material. The UCS of coal and mudstones indicate moderately strong materials.

		and mudstone horizons and four main coal seams. In some sandstones the feldspars have been kaolinised making the sandstone friable	MPa and that of V Seam is 19.48 MPa. UTS of sandstone ranges from 1.13-1.46 MPa (mean 1.28 MPa). UTS of mudstone are 0.48 MPa. TCS of sandstone ranges from 6.20-6.50 MPa (mean 6.35 MPa). Av. RQD varies from 52% to 74%.	UTS of sandstone and mudstone suggests moderately weak rock in tension. UTS values also indicate weak to moderately weak strata. Lower RQDs relate to coal horizons and weathered horizons close to sub-crop.
Below seam V to top of VI Seam	70-130 m.	Strata below seam V tends to be dominated by massive sandstones with minor mudstones and siltstone horizons. The sandstones are of similar lithology as encountered above seam V – quartzo-feldspathic, kaolinised, variably weathered and either weak, moderately weak or moderately strong.	UCS of sandstone ranges from 0.40-55.32 MPa (mean-18.4 MPa) and that of mudstone ranges from 34.04-35.52 MPa.`UTS of sandstone ranges from 0.06-3.56 MPa (Avg. 1.44 MPa). TCS of sandstone ranges from 6.6-12.30 MPa (mean-9.45 MPa) at confining pressure of 3000-6000MPa. Young's Modulus of sandstones ranges from 1350-20630 MPa (Av. 11060 MPa). Poisson's ratio 0.0908-0.3861 MPa (Av. 0.2223 MPa).Av. RQD values range from 70% -93%.	UCS values of sandstone indicate that sandstones are generally weak to moderately strong and occasionally strong. Sandstones have low UTS.TCS values of sandstone indicate moderately weak rock. The mudstone samples indicate strong material. The Young's Moduli values of sansdtone indicate a low degree of elasticity and a tendency to resist deformation, suggesting that blocky caving would occur in unsupported excavation. RQD values indicate strong ground.
VI Seam	29.40-40.52 (Avg.36.14)	Coal with thin dirt bands	UCS of coal ranges from 5.71-24.73 MPa (Avg. 13.67 MPa) . The Young's Modulus values	UCS values indicate weak to moderately strong material. RQD values indicate that extensive support will be required in underground

		range from 3201-3239 MPa and a Poisson's Ratio of 0.1919-0.2705 MPa. Av. RQD values range from 6%-53%	excavation. However, the lower zones have higher RQD values. Young's Modulus and Poisson's Ratio values indicate a low degree of elasticity and a tendency for caving to occur.

Table 26: Summarised Physico-mechanical properties of VI Seam and strata above VI Seam

UCS : Ultimate Compressive Strength

UTS : Ultimate Tensile Strength

TCS : Tri-axial Compressive Strength

RQD : Rock Quality Designation

5.4.13. Approximate thicknesses of Gondwana strata comprising weak to medium strong rock and upper coal seams and that of LDT clay above roof of VI seam in the central part of the mine over the existing longwall panels have been estimated in Table below:

Panel No.	Approximate ranges of RL of roof of VI Seam (m)	Approximate ranges of RL of floor of LDT (m)	Approximate ranges of RL of floor of UDT (m)	Approximate thickness range of parting between floor of LDT and roof of VI Seam (m)	Approximate thickness range of LDT (m)
South side:					
1101	-220 to -252	-150 to -120	-80	70-132	70-40
1103	-245 to -300	-150 to -120	-80 to -85	95-180	70-35
1105	-270 to -350	-150 to -120	-80 to -90	120-230	70-30
1109	-305 to -370	-145 to -120	-80 to -100	160-250	65-20
1111	-340 to -420	-150 to -140	-90 to -100	190-280	60-40
North side:					
1104	-260 to -320	-100 to -120	-90	160-200	10-30
1106	-305 to -340	-100 to -120	-90	205-220	10-30

1108	-320 to -360	-100 to -120	-90	220-240	10-30
1110	-330 to -385	-105 to -140	-90 to -100	225-245	15-40
1112	-345 to -405	-110 to -140	-90 to -100	235-265	20-40
1114	-348 to -410	-110 to -140	-90 to -100	238-270	20-40
1116	-348 to -405	-120 to -145	-90 to -100	228-260	25-45

Table 27: Approximate thicknesses of LDT and Gondwana strata above the roof of VI Seam

5.4.14. It may be noted from Table 27: Approximate thicknesses of LDT and Gondwana strata above the roof of VI Seam that the thickness of Gondwana strata in the south side panels is less compared to that in the north side panels whereas the thickness of LDT horizon is more in the south side panels compared to that in the north side panels.

5.4.15. It may also be observed from Table 26 that a considerable thickness of strata from surface at Barapukuria mine comprising Madhupur clay, UDT and LDT horizons are unconsolidated ground or poorly consolidated rock having nil to limited bridging capacity over excavations. The consolidated strata comprising mainly sandstones of medium strength having bridging capacity over excavations occur above VI Seam and below the TDT formation.

5.4.16. The width (w) to average depth (d) ratios of the caved workings of 1st slice of VI Seam considering only the thickness of the consolidated strata above the roof of VI Seam in place of depth have been worked out as under:

	W/D ratio (approximate)	W/D ratio considering thickness of hard strata only (approximate)
South wing panels:		
1101	0.45	1.19
1103-1111 (Adjacent barrier less panels)	1.42	2.77
North wing panels:		
1104-1108 (Adjacent barrier less panels)	1.20	1.81
1112-1116 (Adjacent barrier less panels)	1.30	1.61

Table 28: width (w) to average depth (d) ratios of the caved workings of 1st slice of VI Seam

5.4.17. The relationship between s/t ratio (ratio of subsidence to thickness) and w/d ratio according to Subsidence Engineers' Handbook of UK is presented below (This is valid for UK but may be generally considered for other places also). The above values of w/d ratio in Barapukuria mine read with the graph below explain the high magnitude of subsidence observed in the mine.

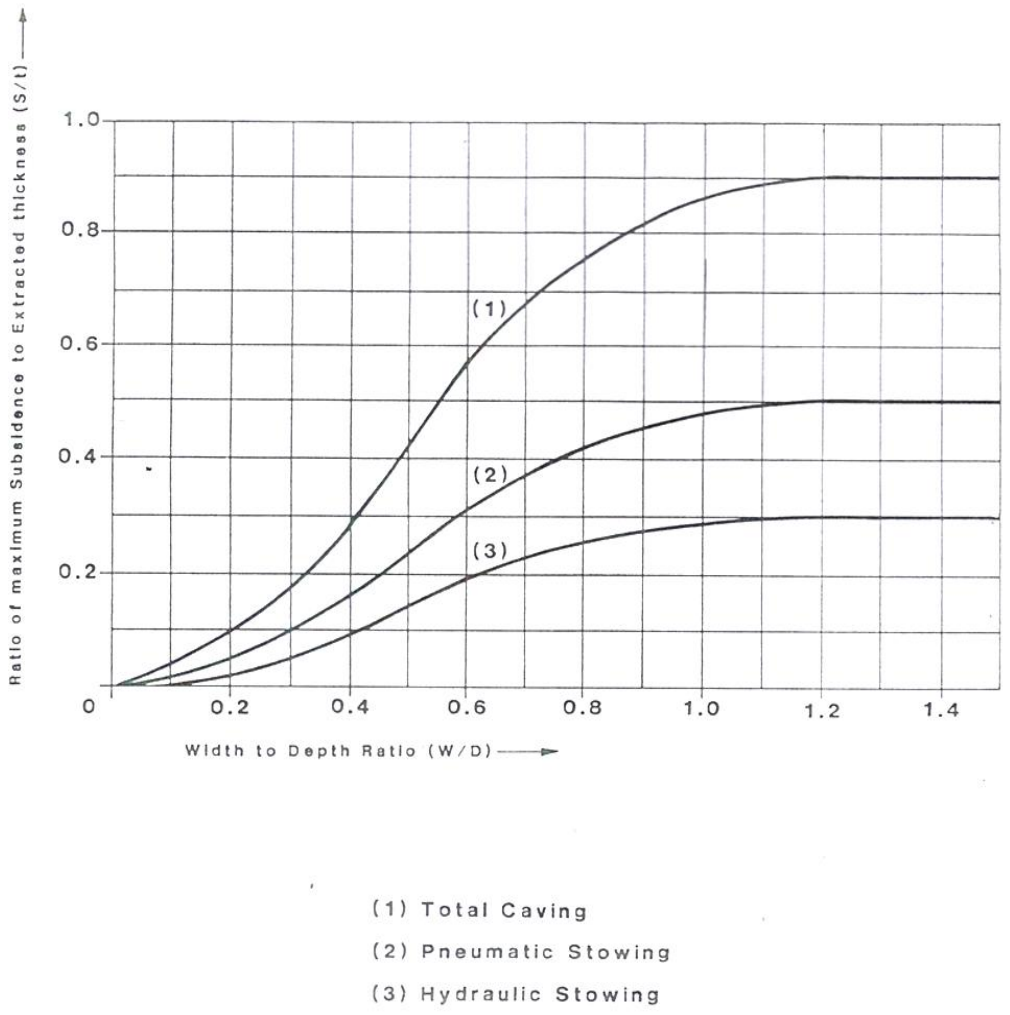

Figure 4: Subsidence – Width/Depth curves for different treatment of the goaf. (After S.E.H.) Graph taken from Barapukuria Coal Deposit, Stage 2, Feasibility study by Wardell Armstrong

5.4.18. Presence of a thick overburden of unconsolidated water bearing strata over the fractured and caved hard rock will result in development of two major problems related to mine safety during extraction of 2nd and subsequent slices of VI Seam and these problems will be gradually more pronounced as the number of slices extracted increases.

5.4.19. The first obvious risk is the possibility of ingress of water from the highly water bearing UDT aquifer into the VI Seam workings through flow paths developed due to any of the following reasons:

- Generation of fracture plane across the strained hard rock and LDT strata (particularly where it is thin) which extends to the base of UDT horizon.
- Passage of water from UDT through water transmitting faults.
- Opening of fault planes which were previously closed and non-water transmitting.

5.4.20. The other possible danger to the mine workings might result from dead weight of the thick unconsolidated strata. These strata, being unconsolidated do not have any bridging capability and hence a component of the dead weight of these strata may be transmitted to the caved Gondwana rocks filling the goaf of VI Seam below which mining has to be done in different slices.

5.4.21. It may be noted that the safe thickness of coal/rock water barrier above the roof of VI seam as has been calculated in the Basic Mine Design report is based on two limiting conditions – (i) individual slice thickness will not be more than 3 m and (ii) cumulative thickness of all slices will not be more than 15m. However, both of these limitations of the empirical formula used will be breached as the individual thickness of slice will be 6m in LTCC method and the cumulative thickness of slices will be 24m (both in conventional and LTCC method).

5.4.22. Also, the formula considered was based on experience of strata conditions and hydro-geological conditions of mines in China, over which certain additional thickness was added on ad-hoc basis to allow for the difference in strata conditions and hydrogeology of Barapukuria mine with those of Chinese mines. However, after working of the 1st slice in Barapukuria mine, valuable geological, hydro-geological and geo-technical data regarding the mine are now available.

5.4.23. Therefore, detailed numerical modeling study and/or other scientific studies are required to be carried out considering the data available and experience gained during working of the 1st slice, to predict the stability and behavior of the strata lying above VI Seam during extraction of 2nd and subsequent slices covering full thickness of VI seam.

5.4.24. Two sets of such studies should be made – one set considering descending slices with caving (for conventional multi-slicing method and LTCC method separately) and the other set considering ascending slices with hydraulic sand stowing (for conventional multi-slicing with barrier between panels) to predict, inter alia, the following:

- Increase in thickness of caved zone with the increase in cumulative thickness of slices extracted.
- Increase in thickness of water permeable fractured zone with the increase in the cumulative thickness of slices extracted.
- Make of water in underground working in each slice.
- Support resistance required in longwall face in each slice.
- Surface subsidence

5.4.25. The above studies for all the slices covering the full thickness of VI seam must be taken up and completed on priority basis for firming up the method of working of 'non-open window' area of the seam. A short term, slice-wise approach to mine planning must be avoided and a view in totality should be taken for selecting a mining system with an objective to achieve mine safety and conservation in the long term.

5.4.26. In addition to the present uncertainties relating to the future projections of ingress of water, stress level in strata and strata behavior; there is problem of existing fire in one of the panels in the 1st slice. It is imperative that the fire in the 1st slice be effectively dealt with and other protective measures taken to prevent further occurrence of fire in the goaves of 1st slice in future.

5.4.27. Therefore, it is recommended that the above-mentioned studies should be carried out, their results analyzed and all necessary actions taken before extraction of 2nd slice for safety of the mine workers, mine property and conservation of coal resources.

5.4.28. The comments offered and opinions expressed in subsequent sections relating to mining methods must be viewed in the background of the overall approach of total planning backed by the results of the scientific studies suggested and stipulations made above, and not in isolation.

5.5. Underground Mine Environment

5.5.1. The following underground environmental problems are faced by the Barapukuria Coal Mine of BCMCL:

- Heat and humidity problems affecting the workplace environment
- Ventilation problem of the mine
- Spontaneous combustion and fire problems
- Other underground environmental problems

The challenges faced in each of these areas and the possible solutions which could be thought of are presented in the subsequent sections of this report.

Heat and humidity problems affecting the workplace environment

5.5.2. Barapukuria Coal Mine has got a typical underground mine environment with regards to high temperature and humidity. A study of the recent ventilation records reveals the following temperature readings at various locations in the longwall panels, viz. 1116, 1111 and 1112, and these are presented in Table 29, Table 30, and Table 31. In addition, the environmental condition in -260mL is also presented in Table 32.

Station Reference	Location	Area of Roadway, m^2	Average Air Velocity, m/s	Airflow, m^3/s	DB Temp, °C	WB Temp, °C	Effective Temp, °C	Relative Humidity, %
A	25m from the first T. Support of B/G	8.88	2.58	22.91	30.50	29.75	25.47	95.00
B	25m from the T. Junction of B/G	8.71	2.50	21.78	31.75	31.00	27.47	95.00
C	25m from the T. Junction of T/G	9.53	2.30	21.92	34.50	34.00	32.20	97.00
D	25m from the first T. Support of T/G	9.68	2.25	21.78	35.00	34.50	33.03	97.00
E	Longwall face	-	-	-	33.50	33.00	-	97.00

Table 29: Ventilation data measured on 28.11.2011 in Panel 1116

Station	Location	Area of Roadwa	Average Air	Airflow,	DB Temp,	WB Temp,	Effective Temp,	Relative Humidi

Reference		y, m²	Velocity , m/s	m³/s	ºC	ºC	oC	ty, %
A	30m from the first T. Support of B/G	9.12	2.75	25.08	27.50	26.50	20.75	93.00
B	30m from the T. Junction of B/G	9.09	2.67	24.23	31.75	31.50	28.00	98.00
C	30m from the T. Junction of T/G	9.12	3.42	31.16	35.00	34.50	32.00	97.00
D	490th Trapezoidal Steel Support of T/G	8.14	3.63	29.55	36.00	36.00	33.75	100.00
E	Open off cut	-	-	-	31.75	31.50	-	97.00

Table 30: Ventilation data measured on 22.03.2011 in Panel 1111

Station Reference	Location	Area of Roadway , m²	Average Air Velocity , m/s	Airflow , m³/s	DB Temp, ºC	WB Temp, ºC	Effective Temp, ºC	Relative Humidity , %
A	25m from the first T. Support of B/G	9.69	2.37	22.96	26.5	26.00	20.00	97.00
B	30m from the T. Junction of B/G	8.82	2.08	18.37	27.8	27.20	23.00	96.00
C	10m from the T. Junction of T/G	8.20	2.07	16.97	33.25	33.00	30.80	98.00
D	10m from the first support of T/G	8.33	1.85	15.41	33.25	33.00	31.00	98.00
	Longwall face (HPRS # 06)	-	-	-	29.50	28.50	-	93.00
	Longwall face (middle)	-	-	-	31.00	30.25	-	95.00
	Longwall face (HPRS # 90)	-	-	-	32.50	32.00	-	97.00
	Goaf	-	-	-	33.50	33.50	-	100.00

Table 31: Ventilation data measured on 22.02.2011 in Panel 1112

Station Reference	Location	Area of Roadway , m²	Average Air Velocity , m/s	Airflow , m³/s	DB Temp, ºC	WB Temp, ºC	Effective Temp, ºC	Relative Humidity , %
A20	10m downward from Track Incline	14.06	1.20	16.87	31.40	27.90	26.73	81.80
A1	10m from -260m S/S door	14.06	2.40	33.74	32.60	28.00	26.01	77.10
A3	10m from coal spillage roadway	15.58	2.50	38.95	31.80	27.90	25.33	80.30
A4	10m from -260 water sump west gate	15.44	3.12	48.17	33.00	28.10	25.63	76.00

A5	15m from explosive house 2nd door	15.01	5.95	89.31	32.30	28.00	-	78.10
A6	10m from T.B.C. roadway	15.90	6.25	99.38	31.80	27.90	-	80.00
A8	Track Dip Upper Yard	12.95	5.42	70.19	30.40	29.00	-	92.30
F28	South Belt Conveyor roadway	15.16	4.42	67.01	32.60	32.60	-	100.00
F29	Connecting roadway of Transfer Belt & South Belt	10.70	5.20	55.64	32.50	32.50	-	100.00
F30	Transfer Belt Conveyor Roadway	14.70	4.43	65.12	32.70	32.70	-	100.00
F35	10m downward from M. shaft connecting roadway	14.18	4.30	60.97	32.70	32.70	-	100.00

Table 32: Ventilation data measured on 16.10.2011 in -260mL

5.5.3. An analysis of the ventilation data presented in Table 29, Table 30, Table 31 and Table 32 reveals the following serious heat and humidity problems:

- The wet bulb temperature (WBT) of longwall face in the Panel 1116 is 33°C with 97% humidity. Towards the tail gate side the WBT has increased to a level of 34°C with 97% humidity.

- In the longwall Panel 1111 the maximum WBT of 36°C with 100% humidity have been encountered.

- In the longwall face 1112 the maximum WBT at the face is 32°C with 97% relative humidity.

- In addition to the WBT, effective temperature is also recorded which is derived from combination of three main parameters, viz. temperature, humidity and air velocity which in turn affect the thermal environment of the mine and has been considered as measure of workplace environment by many countries. The CMC Report (2000) has suggested an effective temperature of 32°C as the limiting environmental condition for this mine. It may be mentioned here that the effective temperature in the longwall panels, viz. 1116, 1111 and 1112, as presented in the aforementioned tables is more than 32°C at number of locations.

5.5.4. Analysis of the information available in Wardell Armstrong Report, CMC Reports (1995, 2000) and data gathered during the field visit reveals the following important reasons for high underground temperature and humidity:

- **High geothermal gradient and high virgin rock temperature:** The temperature of the isothermal zone of Barapukuria Coal Mine is measured at about 25.5°C at a depth of about 30m. According to borehole logging data, the geothermal gradient of roof strata above Seam VI ranges from 2.92°C/100m to 4.97°C/100m with an average value of 3.83°C/100m, which gradually increases from the south and the west to the basin centre. The geothermal gradient within the Seam VI ranges from 5.96°C/100m to 22.44°C/100m, and 10.46°C/100m on average, which is quite high with a significant variation gradually rising from basin edge to basin depth with an increasing rate (CMC Report, 1995). The virgin rock temperature of the main mine district at the lower part of Seam VI is about 40°C. Since the geothermal gradient of the mine is quite high, the mine belongs to the category of high geothermal mine. In this case the heat from the strata is transferred to mine air mostly by convention.

- **Heat liberated from the hot water oozing out from the mine:** The high temperature water inflow is one of the major causes of increase in the heat and humidity problems of the mine. Water transfers its heat to the mine air by evaporation and thereby increasing the latent heat of the air. In addition, it also increases the relative humidity of air. Earlier studies indicated that in Barapukuria Mine, the normal water inflow is 368 m^3/hr and the maximum inflow is as much as three times of the normal inflow and equal to 1160 m^3/hr (CMC Report, 1995). However, the average quantity of water pumped out at present from the mine is in the range of 1470 m^3/hr. Thus geophysical studies should be taken up to identify existence of any possible source of high temperature in and around the mine. As per the report of Geological Department, the quantity of water percolation and its temperature from different longwall faces are summarised in table below:

Longwall face	**Quantity of water inflow from the face, m^3/hr**			**Water temperature**
	Initial stage	Closing stage	At present	
1101	**Total water inflow from roadways and face: 460-738.0**			Track gate roadway: 45-47.6°C Belt gate roadway: 36-41°C Drill hole water temp. at the end of belt gate: 46°C
1103	83.0	295.0	75.0	**In belt gate: 41°C** **Track gate: 37°C**
1105	40.0	105.0	108.0	**In the face: 39-41°C**
1109	86.0	549.0	199.0	**At 615m distance of the track gate roadway: 45°C** **From belt gate within 650-750m: 45°C** **In the face: 42-45°C**
1111	305.0	525.0	525.0	**In the face: 42-45°C**
1104	108.0	121.0	62.0	**In the face: 33-37°C**
1106	60.0	123.0	36.0	**In the face: 40°C** **Water inflow from middle station-I: 46°C**
1108	64.0	91.0	43.0	**In the face: 41°C**
1110	36.0	28.0	05	**In the face: 37-41°C**
1112	10.0	63.0	65.0	**In the face: 41°C**
1114	87.0	95.0	62.0	**In the face: 37-39°C**
1116	No water inflow was observed from this face			-

Table 33: Water percolation from different longwall faces and its temperature

- A study of Table 33 reveals that the temperature of the percolated water is between 33 to 47.6°C. The highest water temperature varying in the range of 36 to 47.6°C has been recorded in the longwall panel 1101.
- It may be mentioned here that the temperature of the water is quite high and is more than the virgin rock temperature of the strata at this horizon. Therefore, authenticity of this

temperature recording is to be validated by fresh studies of virgin rock temperature and temperature of percolated water.

- During the mine visit, it has been observed that in order to handle this percolated water in the main intake airways close to downcast shaft bottom, water drains have been made and these drains are properly covered.

- The percolated water from the longwall panels flow through the intake airway, i.e. through the dip side gate and the main intake airway near the panels. The ventilation system of the panels being antitropal, air picks up the moisture at these places due to evaporation of the openly flowing water and increases the humidity of the air entering the face.

- **Heat from auto-oxidation of coal and carbonaceous matter:** In a number of longwall panels, the concentration of CO is in the range of 100-180 ppm towards the Tail Gate. It indicates that spontaneous combustion is taking place in the goaf areas of longwall panels. This spontaneous combustion is an exothermic process and releases heat. The heat released from this process might be getting added to the intake air after the middle position of the longwall face resulting in further deterioration of the workplace environment.

- **Heat from electrical equipment operated in the longwall panels:** A number of electrical machines run in the longwall panel. These machines are positioned either in the Gate Road or in the longwall face. The friction losses and inefficiency of these machines in the form of heat gets added to the intake air. These machines are belt conveyor, stage loader, transformers, power pack, lump breaker, AFC motors, shearer etc. These high rated electrical machines mostly produce sensible heat which is added to the intake air further deteriorating the temperature of intake air at the longwall face.

- **High temperature of surface air:** Bangladesh has a subtropical climate, with an annual average temperature of 24.8°C and an annual average relative humidity of 74.8%. From April to September, the average temperature is 28.1°C and the average relative humidity is 79.25%. The mine area is subtropical, characterized by hot, humid, sweltering and abundant in rain with relatively arid cool winter. The temperature during April and May is highest in a year, normally ranges from 34°C to 35.6°C, with the maximum of 37.5°C. In winter, from November to February, the temperature normally ranges between 11.6°C and 27.5°C, with minimum of 10°C (CMC Report, 2000). This high surface temperature in most part of the year also contributes for heat and humidity problems in underground.

Recommendations and Conclusions

5.5.5. The following arrangements should be made for prevention of water evaporation and reduction of humidity in the mine:

- The water percolated in the intake airways should be channelized through the covered drains as far as practicable, so that it does not come in contact with the intake air for maximum time of its flow.

- The channelized water through the covered drains should be diverted towards the return circuit, which will reduce the humidity at the working faces.

- The proper planning of drainage of percolated water should be done in such a way that it comes in the minimum intake routes of the mine. This can be achieved either by reorganizing the water drainage system or the ventilation circuit of the mine.

5.5.6. The heat and humidity added to the mine air can be diluted by increasing the air quantity flow to the working areas. This will entail larger pressure difference across the longwall faces. If the pressure across the longwall face increases, leakage of intake air to the goaf area also increases. This may cause spontaneous combustion in the goaf resulting in fire in the longwall faces. Thus appropriate studies and modifications would be required to implement this.

5.5.7. Presently, the intake air to some of the longwall faces has been decreased to reduce this pressure difference across the longwall faces. Therefore, an optimum quantity of air should be allowed into the longwall faces to avoid this fire problem. With this optimal air quantity flow in the panel, efforts should be made to achieve whatever improvement in the mine climate as possible.

5.5.8. If the heat and humidity problem is not resolved by optimizing the air quantity in the longwall panels, a detailed study should be carried out and air cooling system should be installed in the panels for solving the heat and humidity problems.

Ventilation issues in the mine

5.5.9. The Barapukuria Mine has adopted exhaust ventilation system which is favourable for ventilation control and reducing air leakage in the ventilation system.

5.5.10. In order to ventilate the mine, two high capacity exhaust fan systems each of 150 m^3/s capacity and developing 2000 Pa pressure are in operation. Out of the two main fans, one is kept as standby. These fans exhaust the air to the atmosphere through two numbers of vertical evasees of circular cross-section. The main mine fans are directly coupled with the motor shaft with following specifications:

Item	Value description
Diameter of fan casing	2660 mm
Diameter of fan hub	1412 mm
Clearance between the fan casing and blade tip	2.7+0.9 mm
No. of stages	01
No. of blades	20
Regulating range of blade	-15° to +15°
Material of blade	HF-1
Fan speed	750 rpm
Capacity flow	150 m^3/s
Fan motor	Synchronous motor, Mfg. by M/s Sanghai Electrical Mfg. Works, China
Motor type	TD630-8, 6 kV
Motor speed	750 rpm
PF	0.9
V	6000
A	75

Phase	3
Hz	50

Table 34: Specification of main mine fan

5.5.11. The mine has two shafts each with net diameter of 6m and cross section of 28.3 m^2, viz. Auxiliary Shaft and Main Shaft. The Auxiliary Shaft with 320m depth is used for air intake, man-riding, hoisting and lowering of materials, rocks and equipment; whereas, the Main Shaft with 326m depth is used for air return of the whole mine and hoisting of coal.

5.5.12. The Main Shaft is also equipped with metal ladderway for emergency exit and a pair of 8 tonne plate gate skips.

5.5.13. The Auxiliary Shaft is equipped with a pair of multi-rope cages, double deck and 4 cars of 1 tonne each. Both the shafts dug up to -260 m are located in the industrial area, serving for the whole life period of the mine. These shafts are smoothly lined with concrete and have been made waterproof.

5.5.14. As per the record available in the mine, about 6300 m^3/min (105 m^3/s) of fresh air flows through auxiliary shaft, via shaft bottom, main track roadway, mining district track dip entry and belt conveyor gate, respectively into working faces and driving faces. The total return air amounting about 7200 m^3/min (120 m^3/s) returns from the working and driving faces via track gate, mining district air return dip entry, main air return roadway and finally exhausted through the main shaft.

5.5.15. Out of the total intake air, regulated air quantity of about 1100-1200 m^3/min is supplied to the longwall faces to prevent air leakage into the goaf and minimize spontaneous combustion problem.

5.5.16. The roadway cross-section of this mine varies in the range of 8.5 to 12.6m^2 (CMC Report, 1995) with roadway bending radius larger than 9m. The roadway dip angle of Barapukuria Coal Mine is generally less than 12°. Semi-circular arched shaped roadways in rock are used for main intake and trapezoidal section is used for gate roads.

5.5.17. Independent ventilation system is arranged in each mining district. The "U" type ventilation method is adopted in each mining face. Forced ventilation mode using auxiliary fans (smaller capacity fans) is adopted in the development headings supplying about 5 m^3/s of air.

Conclusive analysis of the ventilation system

5.5.18. High pressure loss in the roadways is prevalent due to increase in development activities in the mine since its commencement and in addition due to the reduction of cross sectional area of these roadways, even though major portions of these roadways being sealed off on a continuous basis.

5.5.19. The air quantity reaching at the longwall face is in the range of 20 m^3/s. This quantity is not sufficient for ventilating the longwall panel with all electrical machines and for a mine having high heat and humidity problems.

5.5.20. The total pressure developed by the fan for handling an air quantity of 120 m^3/s is in the range of 2000 Pa. This pressure is considered very high for a mine having fire problems.

5.5.21. The pressure developed by the fan is very high and as a result it increases the power consumption by the fan. Since this is a continuous running machine, the annual power consumption and the power cost is expected to be very high.

Recommendations and Conclusions

5.5.22. In order to improve the ventilation system of the mine, the following measures should be adopted:

- A detailed ventilation survey (pressure-quantity and temperature survey) should be carried out in the mine.
- Ventilation network model of the mine should be developed and an exhaustive computer simulation exercise should be carried out by using different variants for reorganising the ventilation system of the mine.
- One of the conditions worth simulating by using the ventilation network model is the construction of a ventilation shaft at a suitable location, which will reduce the air travel distance, minimize air pressure loss and improve the ventilation of the mine. It is expected that in this condition, the pressure requirement for ventilating the mine may reduce to a significant extent which will reduce the total power consumption of the system, and save a large amount of energy cost for the company.
- The ventilation system should be reorganized in such a way that the ventilation pressure requirement of the system is the minimum. This will have a favorable effect on the fire problems of the mine and reduce the occurrence of new fires.
- A well equipped ventilation laboratory belonging to BCMCL should be set up in the mine with arrangements for chemical analyses of mine air samples, gas chromatography, temperature monitoring, dust monitoring, ventilation surveys, determination of inseam CH_4 content etc.

Spontaneous combustion and fire problems

5.5.23. The mine has faced an incidence of mine fire due to spontaneous combustion. Smouldering mine fire was experienced in face 1111 with high temperature (Annual Report, 2009-10).

5.5.24. Longwall panel 1110, which was being worked out by longwall advancing method, was sealed off due to mine fire. It is evident from the history of the mine that Barapukuria Coal Mine is prone to fire and a number of occurrences of fire have been reported. Presence of poisonous CO gas was detected in the belt gate roadway of 1111 and 1116 faces in all the times (Annual Report, 2010-11).

5.5.25. The results of the air sample analysis at return junction and air return of different longwall panels obtained from BCMCL is presented below:

Longwall panel	Date	Tail gate junction closed to the face		Air return (Tail gate)	
		CO (ppm)	CH4 (%)	CO (ppm)	CH4 (%)
1116	01.01.2012-10.01.2012	10-60	0.06-0.14	0-5	0.04-0.08
1111	14.09.2011-24.09.2011	30-80	0.24-0.34	5-7	0.02-0.04
1112	06.04.2011-17.04.2011	5-120	0.08-0.90	4-80	0.04-0.50
1108	04.11.2010-15.11.2010	5-180	0.06-1.34	5-20	0.04-0.14
1105	20.03.2010-30.03.2010	7-130	0.04-0.46	5-40	0.04-0.28

Table 35: Air sample analysis results of the air at return junction and air return of different longwall panels

5.5.26. From the table above it is evident that highest amounts of CO and CH4 were found in the return air of 1111 longwall panel and the minimum concentration of CO and CH4 in this panel is observed at 30 ppm and 0.24% respectively. The maximum concentration of CO at the tail gate junction is observed at 180 ppm and in some cases it has increased to 100 ppm. This is slightly higher for safe operations of the longwall panels.

5.5.27. The higher concentration of CO in these operating panels is due to occurrence of spontaneous combustion of coal in the goaf. In these cases, the occurrence of fire in the panels mainly depends upon two important parameters, viz. the intrinsic characteristics of coal and the extraneous conditions imposed on it because of working conditions of the mine. It is worthwhile to analyse both the parameters separately and find out the corrective measures to be taken to reduce the occurrence of fire.

Intrinsic properties of coal

5.5.28. The moisture content of coal on air dry basis ranges from 2.31% to 5.36% with an average value of 3.19%.

5.5.29. As per the CMC report, the volatile matter content of Seam VI is relatively high and the coal is easy to combust. On the basis of exploration result, CMC has reported that the dry ash free volatile matter of different zones of Seam VI is less than 37% in most of the cases except in some zones over 37%. The volatile matter content of coal varies from 35.32% to 37.9% and average is 36.71%. Presence of high volatile matter content in the coal accelerates the rate of oxidation and increases the susceptibility of coal to spontaneous combustion.

5.5.30. Based on the analysis of 370 samples, it has been found that the average ash content of Seam VI is 17.82% and apparent density is 1.45 (Wardell Armstrong Report). Since the ash content of the coal is low, this favours the spontaneous combustion and fire susceptibility of coal.

5.5.31. The sulphur content of Seam VI coal varies in the range of 0.61% to 0.67% with an average value of 0.64%. The low value of sulphur may not be an influencing parameter for increasing the susceptibility of coal.

5.5.32. The false roof of Seam VI has the thickness of 0.11m to 2.20m, consisting of carbonaceous mudstone and mudstone and is collapsible soft roof belonging to poor stable bed. The main roof bed is composed of medium to coarse grained sandstone and pebbly gritstone, and constructs medium to difficult caving roof, belonging to medium stable and locally stable rock bed.

5.5.33. Floor of Seam VI has the thickness between 0.18 and 1.66m. It has 2 types of beds, one with poor stability composed of greyish black carbonaceous siltstone/mudstone and mudstone, containing plant fossil fragments while other has medium stability composed of siltstone and medium to fine sandstone.

5.5.34. In the present case the false roof and floor of the seam are generally of low thermal conductivity. These strata are not expected to readily conduct away the heat and favours spontaneous combustion due to accumulation of heat. However, this need to be validated by on field measurement of thermal conductivity of roof rock and floor rocks both in situ and in the laboratory.

5.5.35. Seam VI is characterised as high inertinite content and also with relatively high fusain content. The statistics of the maceral content of different zones of Seam VI reveals that vitrinite content is high in three zones with average values of 42.3%, 44.0% and 37.4%; while vitrinite content is the lowest in one zone with average value of 25.6%. Generally the coal with high vitrinite content is considered more prone to spontaneous combustion. Since some of the zones have high vitrinite content, this may be initiating the spontaneous combustion.

5.5.36. However, the other important parameters such as cleat intensity and extension, which act as the paths for oxygen intake to the coal bed should be studied in detail for further ascertaining the role of geological parameters on spontaneous combustion.

5.5.37. The coal of Seam VI is highly friable and the friable coals are always more susceptible to spontaneous combustion.

5.5.38. Seam VI is very prone to spontaneous combustion and has the incubation period in the order of 35 days (4-6 weeks) as informed by the mine management. This may be due to coal characteristics favouring spontaneous combustion, viz. high volatile matter, low ash, and high vitrinite content etc. as revealed from the overall analysis of the characteristics of Seam VI coal. The exact incubation period of the seam should be determined from field measurements.

5.5.39. As reported by the mine management, the crossing point of coal is 170 °C and according to the general practice these coals should be considered as the least susceptible to spontaneous combustion. This is contrary to the actual evidence that the coal is highly susceptible.

Extraneous condition imposed in the mine workings

5.5.40. **Coal loss:** During the mine visit it was observed that a significant amount of coal is left in the roof of 1st slice of 1116 longwall panel. As per IMCL, this loss of coal is due to undulation of the roof and is unavoidable. In addition, the coal is friable in nature and hence there is always a possibility of spontaneous combustion of the left out coal after the panel is mined out. Also, as per the mining practice, a barrier pillar of 6m-10m thickness is usually left between the panels. Such a lesser thickness barrier pillar accompanied by the friable nature of coal facilitates leakage of air into the worked out panels and leads to spontaneous combustion of coal.

5.5.41. **Ventilation pressure difference:** The ventilation system of the mine has been designed in such a way that there is a high pressure requirement by the system for ventilating the mine. This high pressure difference between the intake and return circuits will cause higher leakage of air into the goaf areas of sealed off panel as well as into the goaf areas of working panels. These leakages will always trigger fire in the sealed off as well as operating panels.

5.5.42. **Method of mining:** The method of mining also plays an important role in governing the occurrence of fire. In the present case, quite a significant amount of coal is lost in the 1st slice. After the introduction of LTCC, coal loss in the goaves is likely to increase. In addition, there will be a lot of fractures in the strata which will assist the spontaneous combustion.

5.5.43. **Improper sealing off worked out panels:** It was found that the worked out panels are not completely sealed off and some openings are purposefully left towards the bottom portion of the stoppings for the purpose of water drainage. These openings may be acting as air leakage path to the goaf for causing spontaneous combustion. In such cases airtight stoppings with U-shaped gully traps (water seals) should be used for drainage of water without air leakage.

5.5.44. **Improper monitoring of fire areas:** The worked out panels and the existing fire areas of the mine are not properly monitored for assessing the initiation and status of fire. It may be noted that in order to assess the status of fire, a number of gas ratios, viz. Graham's ratio, Young's ratio, Willet's ratio, Morris ratio, oxygen content etc. are used internationally. The trend of these ratios over a period of time should be monitored very closely on day to day basis for taking ameliorative measures against spontaneous combustion and fire which is not being practiced currently in the mine.

Recommendations and Conclusions

5.5.45. The worked out panels should be properly sealed with explosion proof stoppings. Wherever water accumulation is taking place behind the seals, gully traps or suitable draining arrangements should be provided and precautions should be taken so that only water is allowed to come out without ingress of air into the sealed off areas.

5.5.46. All the sealed off panels irrespective of occurrence of fire should be monitored by collecting the air samples behind the seals. The fire area air sample analysis should be carried out by using microprocessor-based gas chromatographs. The different fire ratios, viz. Graham's ratio, Young's ratio, Willet's ratio, Morris ratio, oxygen content etc. should be calculated for all the air samples drawn from the sealed off panels and the trend of these ratios should be monitored very closely for assessing the initiation of fresh fire and the status of existing fire.

5.5.47. The air leakage around the seals should be monitored by using tracer gas technique or suitable scientific method. In addition, pressure difference across the seals should also be monitored. If the pressure difference across the seals is high, pressure balancing should be carried out for those seals to avoid the leakage of air into the fire areas.

5.5.48. The sealed off panels under fire should be closely monitored and if the fire activity is increasing, inertisation of the goaf areas should be adopted so that the fires in those panels are not propagating further.

5.5.49. While working the 2nd and subsequent slices, fire may play a major role for successful operation of these panels. Therefore, in order to avoid the occurrence of fire in those panels, pro-active inertisation should be adopted, i.e. while the longwall faces are retreating, the back side of the goaf is flushed with the inert gas continuously for avoiding the occurrence of fire in the goaf with sufficient capacity of inertisation system.

5.5.50. A fresh study on the proximate analysis, maceral content, cleat intensity and extension, thermal conductivity of roof and floor rocks and crossing point temperature of the coal of Seam VI should be carried out and its susceptibility to spontaneous combustion and incubation period should be re-established for properly planning the mining operation.

5.5.51. As Seam VI is very much susceptible to spontaneous combustion, rigorous R&D efforts should be initiated in the mine level to deal with the problem of spontaneous combustion effectively. An in-house R&D set up should be established in the mine for this purpose.

Other underground environmental problems

5.5.52. In following part of this section, environmental problems other than those discussed above, i.e. heat and humidity, ventilation and fire has been discussed. The other environmental problems based on information gathered from the existing reports and field visits to Barapukuria Coal Mine are as follows:

- Mine gases
- Dust problems

Mine gases

5.5.53. A study of different reports reveals that the CH_4 gas content of Seam VI is 0.025 cm^3/g.

5.5.54. However, the report is not very specific regarding the method adopted for determining the gas content of coal.

5.5.55. It may be mentioned here that the gas content of coal is determined either by direct or by indirect methods.

5.5.56. Present practice is that the gas content is determined by both the methods for reliably authenticating the gas content of the seam.

5.5.57. In addition to the gas content of the seam, it is a standard practice that borehole gas survey in the operating mine should be conducted at a regular interval depending upon the rules and regulations of the country. In some countries in the world, the borehole gas survey is carried out once in a month or once in three months or once in four months intervals. This helps the practicing engineers of the mine in advance of their workings if there is a variation in gas content of the seam.

5.5.58. It may be noted here that the borehole gas survey has not been carried out in the mine ever since it's commencement.

5.5.59. In addition to CH_4, some of the seams in other countries also contain H_2S and other gases. In this particular case it has not been studied thoroughly whether other gases are emitted from the seam or not. Therefore, the presence of other gases should be checked thoroughly for ensuring safety of the persons working in the mine.

Dust problems

5.5.60. The coal dust generated in the mine may create two types of problems:

- airborne respirable dust problems
- coal dust explosion hazard

Airborne respirable dust (ARD) problems:

5.5.61. The airborne respirable dust is mainly produced due to the cutting by shearer in the longwall face and coal transportation through conveyor belts. As per discussions with BCMCL's management, water spraying is done at the longwall face for reducing the ARD. But there are no arrangements for dust suppression in the gate roads and belt transfer points.

5.5.62. Since the mine is naturally wet with high water percolation rate, ARD may not be a major problem in the mine. However, all the precautions against ARD should be taken to avoid the occurrence of dust related diseases to the mine workers.

5.5.63. ARD affects the people and leads to long term suffering before the natural death. Keeping these points in view, the monitoring of ARD is done in all the mines of the world at a specified time interval as per the statute.

5.5.64. It may be mentioned here that presently there is no arrangement or instrument for monitoring ARD concentration in the mine. In addition, there is no statute to guide the mine operator for undertaking such monitoring in the mine.

5.5.65. In addition to the level of ARD, the free silica content of ARD is also an important parameter. In Barapukuria Mine, the records indicate that the free silica content of ARD has not been determined at any point of time. It clearly reveals that the Barapukuria Mine is neither having the facility for measuring the ARD concentration nor has measured the ARD concentration in the past. This is one of the serious safety lapses in the system.

Coal dust explosion hazard:

5.5.66. Barapukuria is a naturally wet mine and wet coal dust may not take part in the coal dust explosion.

5.5.67. In most of the cases, coal dust explosion is initiated by firedamp explosion. Since the CH_4 content of the seam is very less, the chances of occurrence of normal firedamp explosion is less.

5.5.68. However, the water barriers are also put in the mine as a precaution against explosion. These barriers are used in the main intake and return airways of the mine for suppression of mine explosion. Each barrier contains 25 number of shelves and each shelf carries 3 number of water troughs of each 60 litre capacity and supported on the cross bars at a height of 4m from the floor level.

5.5.69. However, the Barapukuria mine is a fire prone mine and a number of fires are existing in the worked out panels. Even if the seam is not gassy, these fires may give rise to CH_4 and other hydrocarbon gases at different stages of heating.

5.5.70. In addition, it may also produce hydrogen as a distillation product in some of the cases. The combined lower explosive limit of hydrogen and other gases is less than the CH_4 concentration limit. The heat of the fire also acts as a source of ignition, which may trigger the gas explosion in the sealed off panels. In these circumstances, there is a chance of explosion in the sealed off panels. If such explosion occurs, it may sometimes breach the seals and propagate in other parts of the mine. Therefore, all the precautions for dust explosion should be taken in Barapukuria Mine.

5.5.71. One of the important precautions taken in most of the mines of the world is the sampling of dust from the roof, sides and floors of the roadway by following certain procedure. The inert matter and the VM content of the dust are determined at regular interval of time as per the statute. In case of Barapukuria Mine, there is neither any practice of dust sampling nor any statute in vogue for mandating the dust sampling.

5.5.72. The above mentioned points are to be noted for implementation to reduce the explosion hazard in the mine.

Recommendations and Conclusions

5.5.73. The gas content of the seam has been determined at the initial stage. Since the mine has been working for a number of years, the gas content of seam should be determined by both direct and indirect methods for authenticating the gas content of the seam.

5.5.74. The guidelines for carrying out borehole gas survey in the mine should be developed. Further, in case of increase in gas emission, the gas survey at a regular interval should be carried out for taking preventing measures.

5.5.75. In addition to CH4, other gases emitted by Seam VI should be measured at regular interval of time to avoid the sudden occurrence of other hazardous gases in the mine atmosphere.

5.5.76. The statute for ARD sampling should be developed and the method of ARD monitoring including the type of instrument to be used should be defined in the statute.

5.5.77. The free silica content of ARD should be determined from time to time as a normal practice.

5.5.78. Dust suppression arrangements along the gate belt conveyor and transfer points should be installed for reducing the ARD concentration in the mine atmosphere.

5.5.79. In order to prevent the occurrence of coal dust explosion, the guidelines for dust sampling should be developed and it should include the method of sampling, its frequency, parameters to be determined from the samples, etc.

5.6. Mine Hazards and Safety

Mine Hazards

5.6.1. Barapukuria coal mining operation is done in a very thick seam under water-abundant aquifers. Therefore, the mine management has to take a number of precautions against some of the possible hazards which may be due to water inundation, fire, gas/dust explosion, roof fall/caving, emission of harmful gases etc.

Hazard due to water inundation

5.6.2. Since commencement of mine in June 1996, the mine has witnessed a severe water inrush on 05.04.1998, which was finally dewatered on 15.06.1998 by pumping. Risk of inundation of the mine is mainly from the UDT aquifer above the Seam VI, which is heavily water bearing with water flowing through the fissures and faults providing a direct make-up of water in Seam VI.

5.6.3. In addition, Gondwana Formation above Seam VI also acts as an aquifer for water percolation to the mine. Seam VI itself is weak and also contains water. Its lower section contains more water than the middle and upper sections. Therefore, the hydro-geological conditions of this coalfield are relatively complex. These sources together make the normal water inflow for the mine of about 1200 m^3/h and a maximum of 1800 m^3/h (CMC Report, 2000). The average pumping of water from the mine during the last four years (2008-2011) is around 1470 m^3/hr.

5.6.4. In addition, water from the local accumulation in goaf areas and unplugged exploratory boreholes are the possible sources of risk. Also, during the visit by consultancy team, it was observed that a total area of about 250 Ha above the mined out area has been subsided to a depth of around 1.5 m and filled with large quantity of water.

5.6.5. In case of extraction of the 2nd and subsequent slices, there may be further subsidence and if the crack/potholes extend to the bottom of the reservoir, the whole quantity of water will seep into the mine workings causing inundation of the mine.

5.6.6. This huge accumulation of water may pose immediate inundation threat for the mine if the area is not emptied of water or necessary timely measures are not taken. As precautionary measure, this subsided area should always be kept dewatered and fenced. The other measured to avoid the water inundation are mentioned in the hydrogeology section of this report.

Fire hazard

5.6.7. There is a serious risk of fire hazard in the mine. The details of this hazard along with the recommendations for combating this hazard have been given in this report under the heading "Spontaneous combustion and fire problems".

Gas/dust explosion

5.6.8. The previous study reports indicate that the mine is less gassy and naturally wet. Therefore, in the normal circumstances there is less chance of gas and dust explosion hazards. However, in case of fire in the goaf areas and subsequent increase in different gas concentrations and presence of source of ignition, the risk of explosion is very high. It may be noted that the fire problem would be more acute in the 2nd and subsequent slices, and risk of this explosion hazard will also go up. Other details and the precautions to be taken against this explosion hazard are mentioned in Section "Coal dust explosion hazard" of this report.

Roof fall/caving

5.6.9. During the extraction of 1st slice, most of the fatal accidents have occurred in the mine due to roof and side falls. It may be noted that in the 2nd and subsequent slices, the roof and side management are expected to be more challenging. Unless due precautions are taken, the frequency and severity of accidents will be manifold while working the 2nd and subsequent slices. The precautions which will help avoiding accidents due to this aspect have been mentioned in Section "Mining Method and Production Potentiality".

Emission of harmful gases

5.6.10. In the present case, the emission of harmful gases may occur from two sources, viz. inseam and leaking from fire areas. The emission of gas from inseam should be checked by conducting gas surveys at a regular interval of time. It may be noted that there may be variation of gases from patch to patch within the coal seam. Therefore results of few patches should not be extrapolated to whole seam without thorough investigation. The possible leakage of harmful gases from fire areas and the precautionary measures to be adopted have already been described under the heading "Spontaneous combustion and fire problems".

Mine accidents and safety

5.6.11. The fatal accidents (major, dead) which the mine witnessed during 03.06.2002 to 31.07.2011 are summarised in table below.

Sl. No.	Date	Location	No. of Victims	Cause
1.	03.06.2002	-430 Outer sump	01	Roof collapse while carrying out maintenance work under unsupported area
2.	10.03.2004	1101 Longwall face (winch room)	01	Side collapse in the open off cut (winch room)
3.	26.04.2007	Mother rock roadway	01	Heat stroke due to high temperature and humidity while monitoring the sealed face 1110 through the Mother Rock Roadway
4.	26.05.2007	1109 Longwall face	02	Falling of big coal lump through the AFC
5.	17.09.2007	1103 Belt gate	01	Sudden driving of cutting drum of the road header by a Chinese worker
6.	23.09.2008	1104 Longwall face	01	Electric shock causing severe burning of the whole body
7.	11.05.2010	1108 Belt gate R/W	01	Roof collapse caused falling of steel beam

Table 36: A summary of fatal accidents (major, dead) occurred in the Barapukuria Coal Mine during 03.06.2002 to 31.07.2011

5.6.12. Based on the analysis of accident statistics of Barapukuria Coal Mine for the aforementioned period, it is found that the major (dead) accidents which led to the death of 08 (eight) miners are due to collapse of roof and side walls, hit by falling/rolling coal lump, coal bump, heat stroke, roadheader operation, electric shock etc.

5.6.13. Amongst all, accidents due to collapse of roof and side walls resulted maximum number of fatalities leading to death of 03 miners.

5.6.14. During this period, the other causes of accidents in the mine resulting in minor or serious bodily injury to the miners are due to collapse of roof, mine cars, slusher machine, steel supports, belt conveyors, etc. It may be noted that some of the major and minor accidents are also caused due to high temperature and humid mine atmosphere. The following important points are recommended for improvement of safety in Barapukuria Mine.

Recommendations and Conclusions

5.6.15. Presently to reduce the mine accidents and improve safety, many countries of the world are following the approach of "Risk Assessment and Management". In Barapukuria Coal Mine, this risk assessment and management technique should be implemented for improving the safety in the mine.

5.6.16. Emergency response system should be designed and emergency organization plan should be formulated and implemented.

5.6.17. Training and retraining of workers should be put in place for improving their knowledge in the domain of their working and awareness about safety.

5.6.18. Periodical medical examination (PME) of workers should be introduced to know the status of their health with respect to dust related and other diseases.

5.6.19. In hot and humid environment, apart from heat stroke, people are prone to other types of accidents due to their mental state under exhaustive conditions. Therefore, the mine climatic condition, especially with reference to heat and humidity should be improved as discussed earlier.

5.6.20. The systematic support rule (SSR) should be formulated and imposed for reducing the accidents due to roof and side falls.

5.6.21. Safety rules, bylaws, standard operating procedure (SOP) and code of practices should be formulated and implemented.

5.6.22. The recommendations for preventing the occurrence of other hazards as described in previous sections should be implemented.

5.6.23. Some of the important plans, viz. water danger plan, ventilation plan with all ventilation control devices (stoppings, airlocks, regulator, air-crossing, etc.), dust sampling plan etc. should be prepared and updated at regular intervals.

5.6.24. Regular subsidence monitoring and treatment of the subsided area on the surface should be undertaken on priority basis. A survey organization should be set up at the mine level for day to day underground survey and also for subsidence survey.

5.6.25. Organization at the mine level should be developed for regular safety monitoring with full time management.

5.7. Logistics and infrastructure

5.7.1. In following paragraphs, surface and underground mine infrastructure and logistics of Barapukuria mine has been appraised.

Coal Transportation system

Transportation system out of mine

5.7.2. Men and material are transported from production and development faces using trunk conveyor systems to the pit bottom and automatically loaded into the main shaft skip winding system. Upon reaching surface, coal is transported to power station using conveyor system.

5.7.3. To store coal produced at higher rate of production, Bunkers are provided both underground as well as surface. A Silo of 15 m diameter and 2500 tonnes is provided on the surface.

Underground Coal Transportation System

5.7.4. Figure 5 provides typical arrangement for transportation of coal from development and extraction faces in a Longwall mine.

5.7.5. In case of a development face, coal is being cut by advancing road-header machines which is then transported through a system of belt conveyors before being hoisted onto the surface by skips

5.7.6. In case of panel extraction, coal is cut and caved simultaneously by using DERD shearer machine which is then transported through a system of belt conveyors, before being hoisted onto the surface.

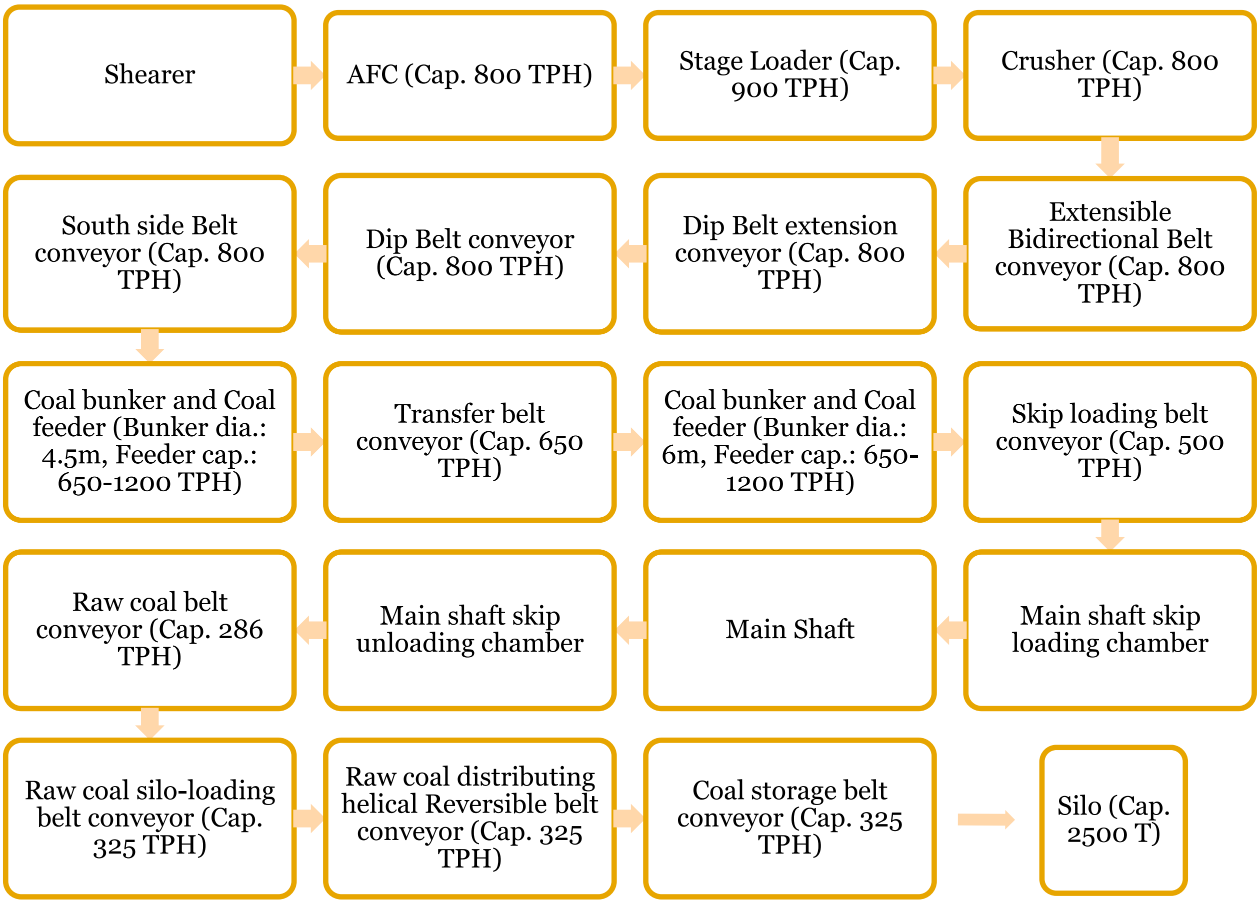

Figure 5: Layout showing Underground Coal Transportation System

Surface Coal Transportation System

5.7.7. Coal, after being transported through the underground coal conveying system, is hoisted onto the surface with skips of 8 tonnes from where it is being transported to the coal stockpile area. The flow diagram below represents the surface coal transportation system.

5.7.8. Overall, the complete coal transportation system is mechanized and has been designed to achieve a production capacity of 1 Mtpa of coal from the mine (as per the report of Wardell Armstrong).

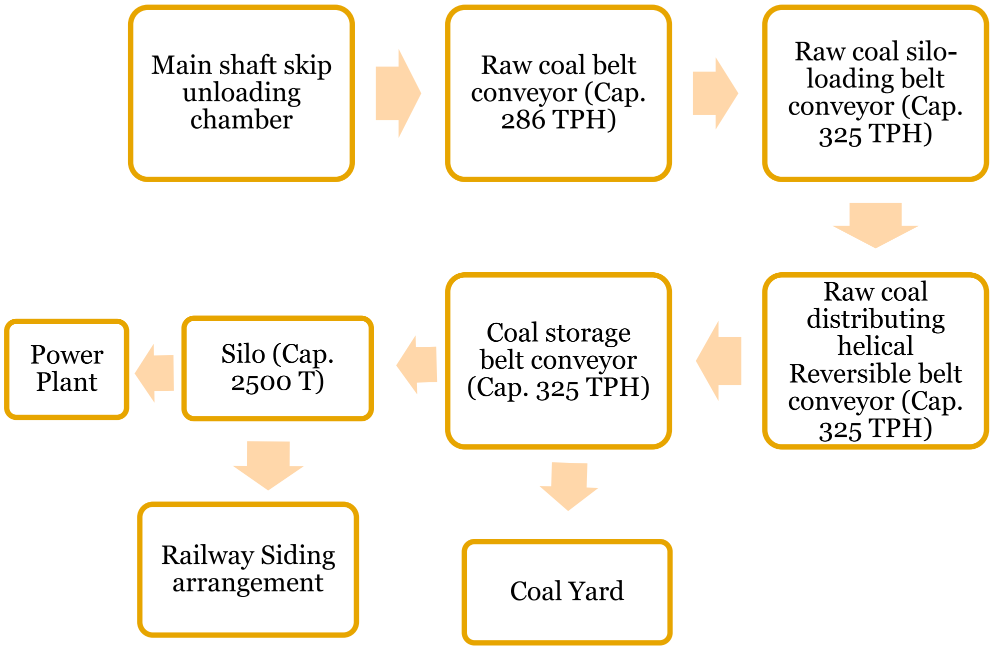

Figure 6: Layout showing Surface Coal Transportation System

Mine infrastructure

Underground infrastructure

Pit-Bottom

5.7.9. Layout of Pit Bottom is vertical type with circular system comprising of shaft, main roadways, access to the mining district, coal and rock transportation system.

Service Shaft Area

5.7.10. The service shaft area is at an elevation of -260 m and has establishment for the signaling chamber and car-pushing chamber. The main sub-station, main pump house and pipe roadways are located at the West Side of Service Shaft area. The waiting room for personnel has been built at the East side of the area, which houses tools room and pit bottom clinic. In addition, a passage connects the waiting room to the shaft ladder way as an emergency exit. There is provision for firefighting equipment as well.

Main Shaft Area

5.7.11. The skip loading chamber and loading belt conveyor roadway is located at the -260 m level. There is a vertical bunker (No. 1) built above the belt conveyor roadway. No. 2 vertical bunker is located to the west of a fault with the belt head chamber. An explosive store is provided near the empty car shunting line. Inside the car return line is a dispatching room, firefighting materials and locomotive repair room.

Main Water Sump

5.7.12. Within the pit bottom area the water sump is at the lowest elevation of -260 m level, having a sump capacity of 5500 m^3. In addition to this, there is another sump area at -430 m level with a sump capacity of 7600 m^3. This sump has been connected to the coal extraction area by a 1:200 gradient road to ensure that all the water from the district will be collected at the sump by gravity.

Power Supply

5.7.13. The main power supply for the mine is from Power Grid Company Ltd. Bangladesh (PGCB). A thermal power plant of capacity 2*125 MW was commissioned in the month in February 2008 which supplies power to PGCB. Two transformers of capacity 10 MVA each have been provided for supplying power to the Barapukuria mine. Although the total capacity is 20 MVA, the maximum permissible power consumption for the mine operations and other surface facilities is 16 MVA.

5.7.14. With current operations, actual power consumption at mine is in the range of 7-8 MW. Power is supplied in mine through step down transformer of 33 kV /6.3 kV. On the mine surface, a Bus-bar of capacity 6.3 kV receives this power and distributes this power to the following surface installations:

- Main shaft
- Auxiliary shaft
- Fan house
- Power Sub-station at -260 m level (underground Sub-station)
- Power Sub-station at -430 m level (underground Sub-station)
- BCMCL residential buildings
- Capacitor plant
- Lighting plants

5.7.15. The line diagram below describes the power supply to the mine and its flow to the sections and various working panels in the mine:

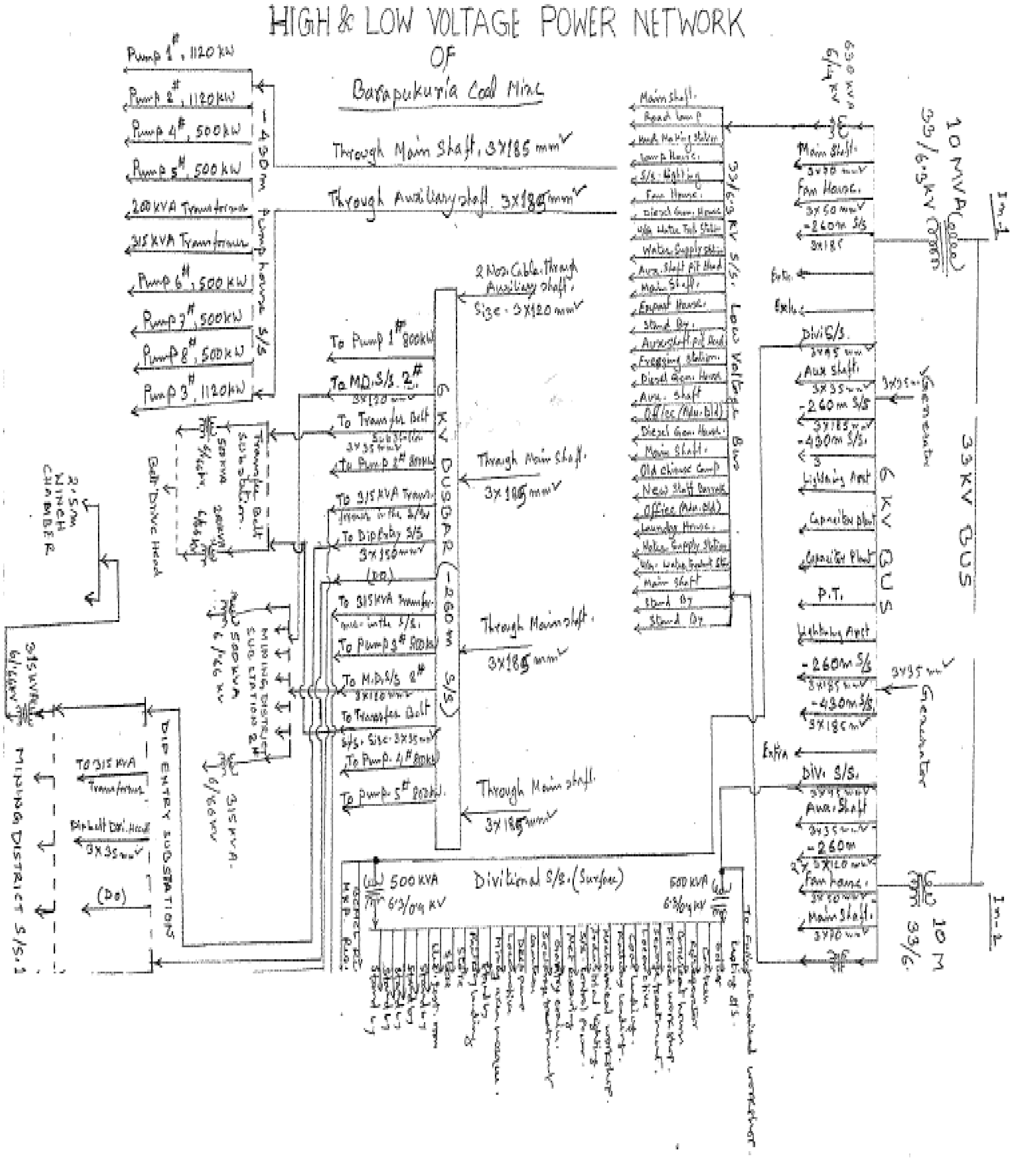

Figure 7: Line diagram describing the power supply to the mine and its flow to the sections and various working panels

Underground Power Distribution:

5.7.16. From the surface power station, power is supplied to the pit bottom sub-station located at -260 m level having Bus-bar to receive power at 6.3 kV. Power from this Bus-bar is distributed to following underground installations:

5.7.17. **Pump level at -260 m level:** There are 5 number of pumps installed at the -260 level. Power is being fed to these pumps station from this bus-bar.

5.7.18. **Transformer and conveyor system:** Power from this bus-bar is being fed to another bus-bar which distributes incoming power to the following installations:

- Face transformer of variable ratings of 6.3 kV/1140 V (Extraction and Development face) and 6.3 kV/ 660 V (Development face).
- Belts drive head unit transformer of rating 6.3 kV/ 660 V.

5.7.19. Pit Bottom lighting via lighting transformer of rating 6.3 kV/ 127 V.

5.7.20. Man-riding transformer of rating 6.3 kV/ 660 V.

5.7.21. The second sub-station has been installed at the -430 m level. A bus-bar of incoming power capacity of 6.3 kV distributes the power from this sub-station to the following feature:

- Lighting stations through a transformer of rating 6.3 kV/ 127 V.
- Pump level at -430 m level

There are 8 number of pumps installed at -430 m levels. Power is being fed to these pumps from this bus-bar.

5.8. Surface environment

Physical Environment

Climate

5.8.1. The Barapukuria Coal Mining Project area is under the typical monsoon climate prevailing in the country. It has three main seasons:

- Summer/ Pre-monsoon – March to April
- Rainy season/ Monsoon – May to October
- Winter season – November to February

5.8.2. The rainy season extends over a period of June to October and is hot and humid accounting for about 84% of the annual rainfall. The winter is predominantly cool and dry, whereas the summer is hot and dry interrupted by occasional heavy rainfall. Humidity in the air at that time is very low.

Temperature

5.8.3. The average minimum temperature occurs in November through January is generally 12.9°C. While the average maximum temperature is 33.2°C recorded in the month of April. The highest recorded temperature is 43.9°C. The ever lowest recorded temperature in the area is 1.1°C.

Rainfall

5.8.4. The total mean annual rainfall received by the area is about 1929 mm. Seasonal distribution of rainfall is skewed with majority of rains received during monsoon season.

Months	**Rainfall (mm)**	**Percent**
Rabi (November – February)	31	1.6
Pre-monsoon (March – May)	281	14.57
Monsoon (June – October)	1617	83.83
Total	1929	100.00

Table 37: Seasonal Rainfall at Parbatipur (Source: SRDI, Dhaka)

Wind

5.8.5. The available data on the wind directions and speeds for the region (Dinajpur-Rangpur) indicates that the wind blows predominantly from East to West (40%), west to east (26%), and from north-east (18%). The wind speed rarely exceeds 8 m/s and mostly the wind is calm for most part of the year. (Source: EIA of Barapukuria Coal Mine Development Project, 2005)

Air Quality

5.8.6. The project site is located in a rural setting without any kind of industrial activity. The air quality in existence has been designated as normal not to warrant any concern for human health or environment degradation, as per the EIA of Barapukuria Coal Mine Development Project, 2005.

Ambient Noise

5.8.7. The project area was quite noiseless as there were no activity other than agriculture and small business prior to start of mining activities and setting up of power plant. Noise level in the process area is within the allowable limits as per standards set by Environment Department.

5.8.8. The noise level as recorded over time at mine is presented below:

S. No.	**Observation Time**	**Sound Level in decibel (dB)**		
		Lowest	Highest	Allowable Limit
1	Morning (8.30 A.M.)	30	60	60-70 dB
2	Afternoon (12.45 P.M.)	25	60	60-70 dB
3	Evening (6.30 P.M.)	20	55	60-70 dB

Table 38 : Sound level near the Parbatipur, on May 15, 2005. (Source: EIA of Barapukuria Coal Mine Development Project, 2005.)

Land Use Pattern

5.8.9. Out of the total land, 80% are cultivable land followed by 15% homestead and the remaining 05% consisting of orchard, ponds and other fallow lands. The table below shows the detailed land use pattern of the surveyed Unions.

Name of the Upazila	Land Holding Pattern (%)				
	Cultivable land	Homestead	Orchard	Others	Total
Parbatipur	80%	15%	3%	2%	100%

Table 39: Detailed land use pattern of the surveyed unions (Source: EIA of Barapukuria Coal Mine Development Project, 2005.)

5.9. Organizational structure

Present organization structure

5.9.1. The present organization structure of the Barapukuria Coal Mining Company Limited flows from the Bangladesh's Energy and Mineral Resources Division, through Petrobangla and subsequently rolls down to the organization present at their site area.

5.9.2. The organization chart below present the reporting relationship of BCMCL with EMRD along with relative position with other related entities.

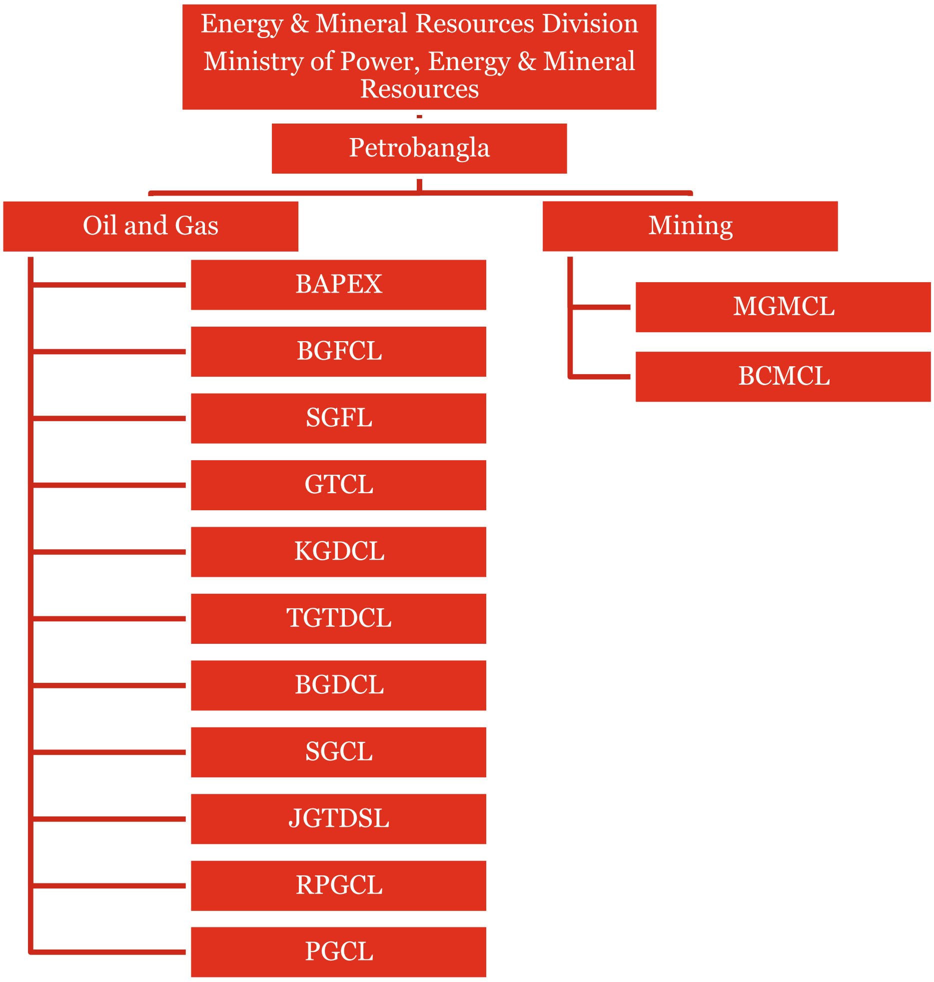

Figure 8: Organizational Chart of EMRD

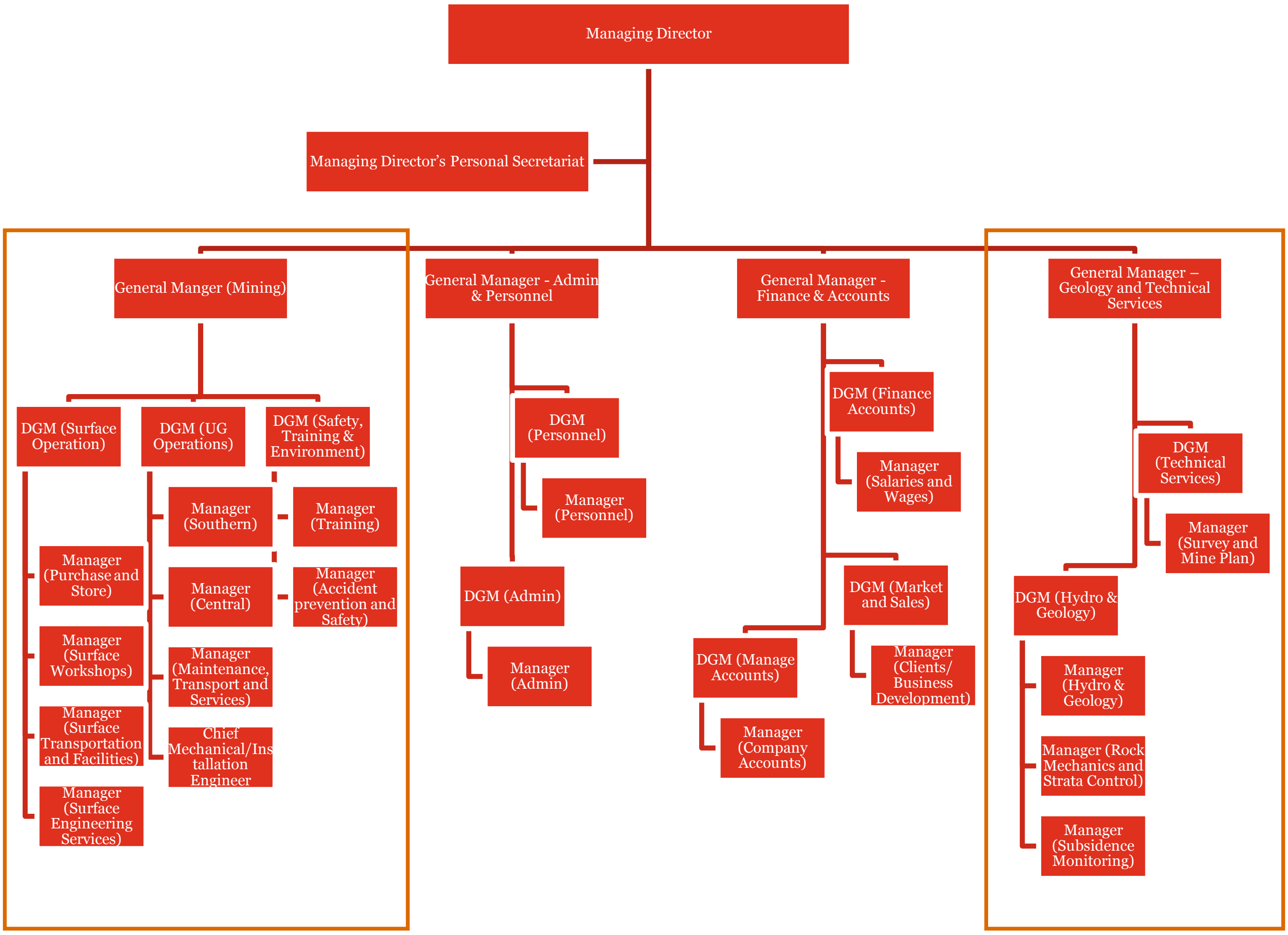

Figure 9: Organization Structure of Barapukuria Coal Mining Company Limited

Recommendations:

5.9.3. As discussed earlier, Barapukuria coal mine faces several challenges including safety aspects, high water make, weak rock conditions, high heat and humidity etc. which raises safety concerns. Further, we understand BCMCL is considering adopting LTCC method. Both longwall and LTCC methods of working specially in thick seam are specialized methods and thus BCMCL management may consider appointment of technical experts with the experience in the similar technologies to support mine oversight.

5.9.4. Further, enhancement of the strength and capabilities of technical departments for planning and operations of the underground mine is required to ensure smooth operations and supervision. Some of suggested changes are discussed below:

Proposed Changes in Structure of General Manager, Geology and Technical Services

5.9.5. The structure defined under General Manager, Geology and Technical Services may be redrawn as follows:

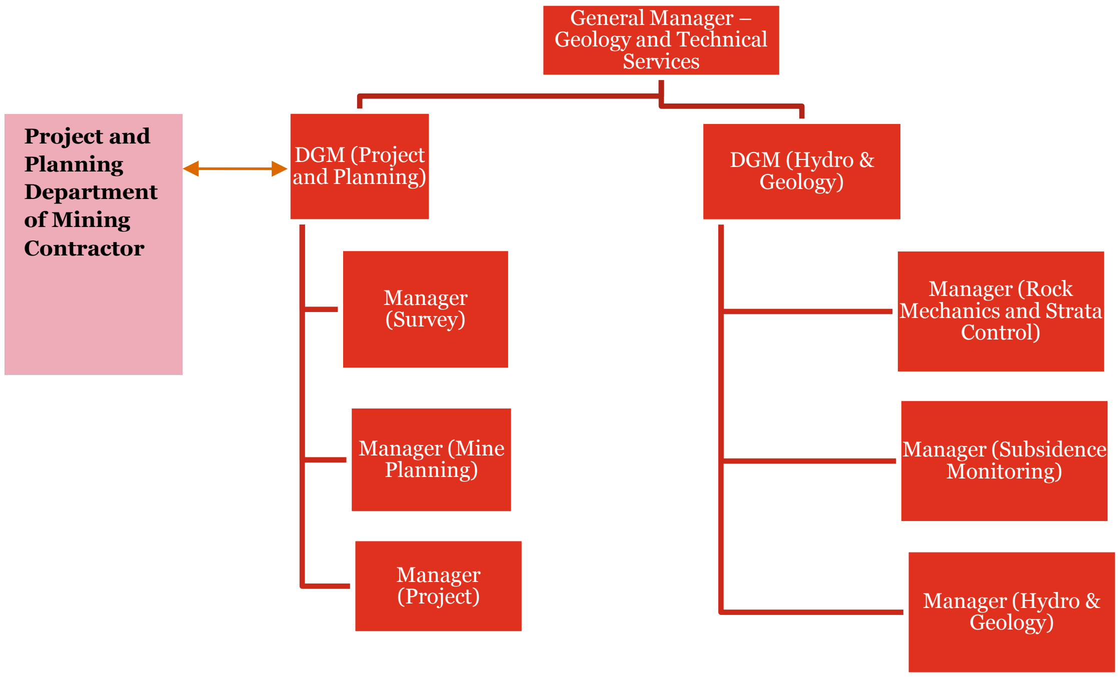

Figure 10: Proposed Structure of General Manager, Geology and Technical Services

5.9.6. As per the Structure proposed above, there would be a dedicated team for Project and Planning focused on the Mine Planning and Scheduling along with the Project works to be carried out in the mining operations. The department would develop monthly/ quarterly/ six-monthly and yearly mine plans from time to time necessary for smooth mining operations. The plans and schedules so developed would be in-line with the Original (Approved) mining plan, as per the Contractual terms.

5.9.7. Further, the Project and Planning Department will perform Subsidence Monitoring on a routine basis, preferably, once in a fortnight.

5.9.8. The department needs to work in close coordination with the Project and Planning department of the mining Contractor and should meet at least on weekly basis to discuss the progress of the mine. This meeting suggested in addition to the general production meeting of the mine and should be focused on the mine's future development activities and the present progress to have discussions at micro level.

5.9.9. The department will also appraise BCMCL management on the progress of mine working and compliance with M&P Contract and mine plan on weekly/ fortnightly/ monthly/ yearly basis. Deviation from mine plan and agreed contract should be reported on priority.

5.9.10. All the reports/ plans generated by this department must be in English language or a language understandable by BCMCL management so as to have an access to these reports/ plans.

5.9.11. For initial period, this department may also be operated with consultants appointed to perform BCMCL's duties along with responsibility of knowledge sharing and skill development to BCMCL officials. This step would further ensure training of engineers, miners and maintenance technicians of employees of BCMCL so as to develop and strengthen in-house capabilities of BCMCL.

Proposed Changes in Structure of General Manager, Mining

5.9.12. The structure under General Manager, Mining is proposed as below:

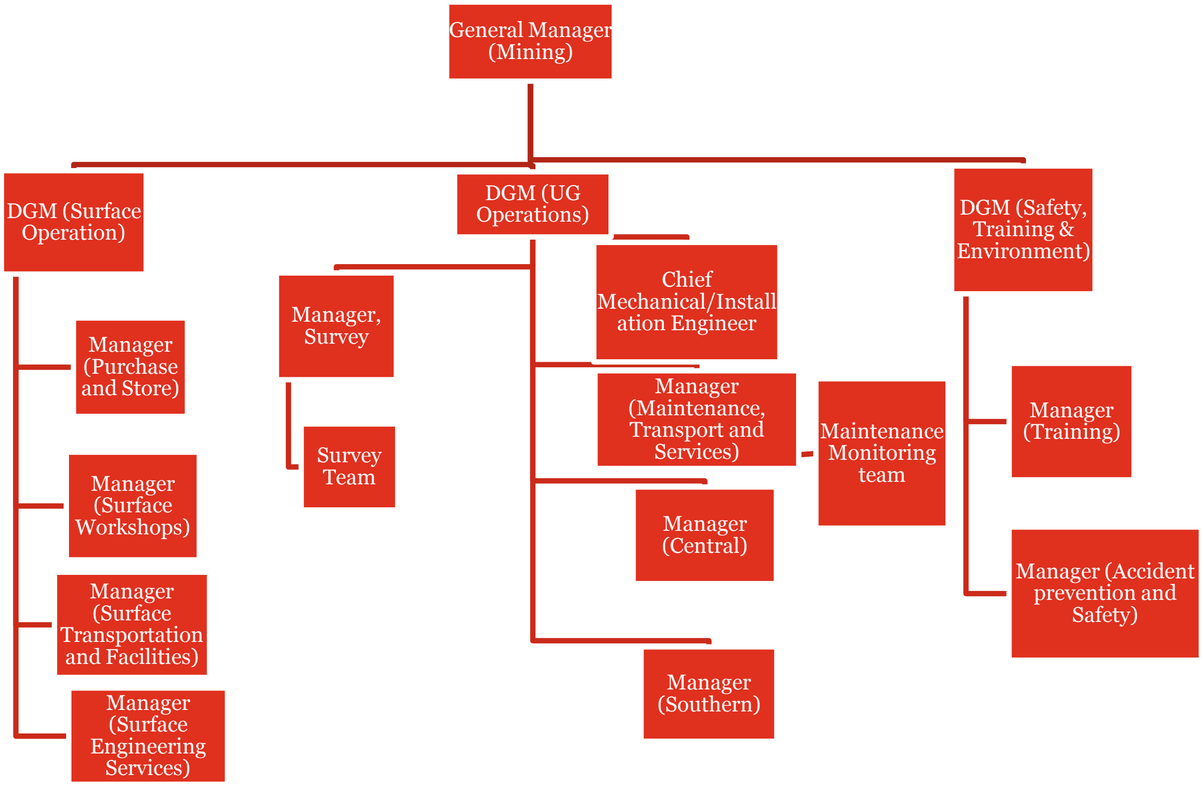

Figure 11: Proposed Changes in Structure of General Manager, Mining

5.9.13. The above changes are proposed to facilitate the monitoring of the mining activities.

5.9.14. The Manager Survey should be entrusted with following responsibilities:

- Survey and measurement of daily progress of the coal mining face
- Monitoring of production from coal faces by measuring face progress
- Provide a cross-check of the coal production, being measured by belt weighing system, installed on the surface.

5.9.15. This department will work in close coordination with the Project and Planning Department as proposed in the Section 5.9.5.

5.9.16. The Maintenance Monitoring team will have a crucial role to play in the existing operating scenario at BCMCL.

5.9.17. The key responsibility areas for this team would be:

- Generate data for equipment wise breakdown for every shift
- Capture these data to produce daily equipment availability and utilization
- Maintain these data for monthly/ quarterly/annual reporting

- Assist the Purchase and Store department in inventory planning
- Maintain a record for equipment maintenance schedule.
- Evaluate operations performance against agreement

Roles for Training Department

5.9.18. The Training department should have an enhanced role by training and certifying BCMCL's employees from time to time. This will improve in-house capabilities for BCMCL in all operational and mining support related activities. Training department should also be entrusted with responsibility to develop knowledge repository with the help of BCMCL's Contract and BCMCL's technical consultant.

6. Suitability of longwall top coal caving method

6.1. Background for Longwall Top Coal Caving (LTCC) Method

6.1.1. The 2nd slice of VI seam is proposed to be worked by longwall top coal caving (LTCC) method as per agreed terms of the new M&P contract signed between BCMCL and CMC-XMC consortium. The method envisages extraction of 6m coal in one lift immediately below 1st slice.

6.1.2. However this report has been prepared by Consulting Team based on the information made available to it till first week of February 2012. Till the time, consulting team did not receive detailed parameters of the mining system proposed under LTCC. Though we understand that BCMCL has signed new M&P Contact with CMC-XMC consortium and thus implementation of recommendations may need to be done within the framework of new contract.

6.1.3. It is therefore, assumed that the height of the longwall face in 2nd slice will be 3m and the top coal (sublevel coal) above the roof of this longwall face and below the floor of the 1st slice will be 3m. As the 2nd slice longwall face and the powered supports advance, the top coal or sublevel coal will be allowed to cave on an armored rear conveyor (ARC) located in the goaf side behind the powered supports and the ARC will deliver coal to the gate belt conveyor. Therefore, a new set of face equipment (specifications not known) for longwall top coal caving method has been included in the M&P contract. The rated capacity of the mine will remain at 1 Mtpa.

6.1.4. Adoption of the proposed LTCC method under the geo-mining conditions obtained at Barapukuria mine vis-a-vis continuation of the existing multi-slice longwall mining method has been analysed in this section from following perspectives.

- Method of mining, safety and recovery of coal reserves
- Mine production capacity, and
- Economics of mining

6.1.5. However, it must be mentioned here that the scope of the present study is limited only to an indicative assessment of the above two mining systems and detailed scientific and feasibility studies must be taken up by BCMCL for an exhaustive evaluation of these two systems and also of alternate mining systems with stowing for safe and efficient extraction of all the slices of the thick VI seam.

6.2. Method of mining, safety and recovery of coal reserves

6.2.1. The 1st (topmost) slice of the thick VI seam has already been worked with DERD shearer and powered supports with an extraction height of 2.5-3m.

6.2.2. The approved project document envisaged use of iron mesh netting on the floor of all the panels, but in practice, such floor matting were laid only in two panels and the practice had to be abandoned for poor workmanship (as reported during Barapukuria visit). However, later it was informed that such floor matting could not be practiced due to high inclination of the face.

6.2.3. The rate of coal extraction from the panels was greatly affected due to occurrence of several faults and dykes, formation of roof cavities and frequent occurrence of self heating in coal in addition to the poor environmental condition of the workings, particularly in the panels of southern wing. With all these constraints, a production level of 0.67 Mtpa - 0.82 Mtpa could be achieved from the 1st slice against a target of 1.0 Mtpa.

6.2.4. As per the approved project document, the 2nd slice was to be worked below the 1st (topmost) slice after leaving a coal parting, the thickness of which is not mentioned.

6.2.5. Assuming a parting thickness of 3m and a height of extraction of 2nd slice as 3m, the present mining system could have been continued with the existing sets of longwall equipment.

6.2.6. With the assumed height of longwall face of 2nd slice in LTCC method as 3m and the thickness of coal parting between floor of 1st slice and roof of 2nd slice longwall face as 3m, the LTCC system will be achieving a higher coal recovery compared to the conventional multi-slicing system, where the entire parting coal of 3 m (or so) will be lost in goaf with consequent increased risk of fire.

6.2.7. The capability of the de-stressed coal parting or sublevel coal in LTCC method is not expected to pose much problem (except during initial period till some advance of the 2nd slice longwall face is achieved) considering the physico-mechanical properties of VI seam coal and the pressure of broken strata acting on the coal parting.

6.2.8. In any case, regular caving of parting coal is also a pre-requisite for the successful operation of conventional multi-slice mining.

6.2.9. However, the rate of advance of the longwall face in LTCC method may be slowed down occasionally due to problem of blocky coal coming down on the rear conveyor or even boulders of roof rock coming on the rear conveyor due to absence of iron mesh netting in most of the panels of the 1st slice causing jamming of ARC which may involve manual intervention.

6.2.10. Snapping of ARC chain may also be observed during the operation. Such situation, apart from causing delay and reducing output, may warrant leaving some of the caved parting (sub-level) coal in goaf thereby increasing the risk of fire as the coal is very much susceptible to spontaneous combustion.

6.2.11. Arrangements of nitrogen flushing and chemical treatment will therefore, have to be kept to mitigate risk of fire in goaf in LTCC panels as in case of conventional multi-slice panels.

6.2.12. LTCC method will involve extraction of 6m of coal in 2nd slice (cumulative 8.5-9m) compared to 3m (cumulative 5.5-6m) in case of conventional multi-slice mining.

6.2.13. As the empirical formula for determining the height of the fracture zone considered in the report of Basic Mine Design of Barapukuria mine assumes individual height of slices to be not more than 3m, fresh scientific studies are required to be carried out to determine the height of fracture zone and safe thickness of coal/rock parting above VI seam to prevent disturbance to the UDT aquifer horizon before application of LTCC method.

6.2.14. Further, additional data regarding behaviour of UDT, LDT and Gondwana strata as available from the experience gained during extraction of 1st slice in the mine may be utilised in the present studies to tailor the new model to suit to Barapukuria conditions.

6.2.15. However, it can be generally said that extracting the 2nd slice with 6m thickness (in LTCC method) will result in higher thicknesses of caved and fractured zones in Gondwana rocks compared to that with 3m height of extraction of 2nd slice (as in conventional multi-slicing method) and therefore, the following are to be reassessed during the above scientific studies for application of LTCC method:

- The support resistance required at the longwall face in LTCC method in 2nd and subsequent slices, and
- The quantum of flow of water from Gondwana aquifer in the goaf of 2nd and subsequent slices worked by LTCC method.

6.2.16. As the successive slices will be extracted by caving in descending order, the thickness of the caved zone will increase depending on the cumulative seam thickness extracted and the bulking factors of roof rock and un-extracted parting coal in the goaves. Thus the dead weight of the caved zone will increase while successive lower slices are worked, more so if LTCC method is adopted. Also, a component of dead weight of UDT and LDT strata over the caved zone will be transmitted to the caved rocks as the UDT and LDT horizons do not have ability to bridge the width of the caved zone. Thus, the support resistance required will increase as successive lower slices are extracted.

6.2.17. Therefore, the specifications of the powered supports of LTCC panels should preferably be selected in such a manner that these can serve for two or three successive slices without being replaced before their stipulated life.

6.2.18. In addition to the above considerations, it must also be added that adoption of LTCC method will involve use of more sophisticated mining equipment and will require manpower with advanced skills compared to those required in the operation of conventional multi-slice mining system, which is presently being practiced in the mine and in which indigenous manpower have largely been trained.

6.3. Mine production capacity

6.3.1. A study of the periods of operation of individual panels of the 1st slice shows that the two shearers have never worked in the mine simultaneously except for a brief period between 25.2.2010 and 16.03.2010.

6.3.2. But, it must also be mentioned that one of the two longwall sets was not available from end of September 2005 to middle of August 2008 (almost for 50% of the contract period) as the equipment were trapped within a panel which had to be sealed off due to fire.

6.3.3. The mechanical condition of the shearers, powered supports and other face machineries are not known due to absence of information at the mine level but the average daily production from the shearers are not below that envisaged in the Basic Mine Design Report of CMC.

6.3.4. In addition, out of the four Road-headers procured for developing longwall panels, only two are in working condition and the other two are reportedly beyond rehabilitation.

6.3.5. Considering all these, it can be said that the rated production of 1.0 Mtpa could be achieved by conventional multi-slice longwall system by deploying two sets of PSLW equipment in two faces in 2nd slice, provided the face equipment were refurbished/overhauled and two more road headers were added replacing the unserviceable ones for timely development of longwall panels.

6.3.6. After completion of the rated life of the existing face equipment, these could be replaced with updated versions of the equipment.

6.3.7. To ensure a regular production of 1.0 Mtpa from one set of LTCC equipment, appropriate actions have to be taken to minimize the following delays:

- Delay in face advance due to problem of jamming or damage of ARC, delay in clearing of roof coal etc.
- Delay during salvaging and re-installation of the LTCC equipment to the next longwall panel.
- Delay in taking up due maintenance and breakdown repair works

6.3.8. It is recommended that a system of proper spares management should be established and effective steps for up-gradation of skills of the operation and maintenance crew should be taken to reduce the above delays.

6.4. Economics of mining

6.4.1. For any investment decision, it is necessary to evaluate comparative economics of the feasible technical alternatives. In the present case also, it is necessary to carry out cash flow analyses for at least next 10 years (with 1st year as 2011-12) considering a rated production of 1.0 Mtpa for the existing longwall multi-slice system and the proposed LTCC system to compare the economics of these two systems.

6.4.2. For this purpose, the operating and capital costs of the existing system need to be updated and a revised cost estimate (RCE) of the existing mine should be prepared. The economics of the two systems may then be compared in terms of their Financial IRRs and Economic IRRs or in terms of their NPVs.

6.4.3. Further, it is desirable to carry out sensitivity analyses of the FIRRs or NPVs of the two systems to ascertain the risks involved in each system. These financial analyses must, therefore, be carried out for both the systems before taking a final decision regarding selection of a system.

6.4.4. Considering the M&P contract for the Barapukuria mine to be valid for a period of 5 years, these analyses may also be mapped for such a smaller period to assess the risk of disengagement of BCMCL with the present contractor after the completion of the contractual period.

6.4.5. The Economic Analysis can be undertaken by Consultant of International repute with strong experience in mining. The economic analysis shall be based on the technical feasibility study and detailed project report of the LTCC method and the equipment proposed. Thus a detailed technical feasibility study is required to prepare economic feasibility.

7. Feasibility of opencast mining of VI seam in open window area

7.1.1. In a major part in the north of Barapukuria mine area, the thickness of Lower Dupi Tila (LDT) formation (aquiclude) varies from 0-10m only which is overlain by the highly water bearing Upper Dupi Tila (UDT) formation.

7.1.2. Underground mining of VI seam in this area is likely to face serious problem due to uncontrollable flow of ground water in the mine from the UDT horizon due to absence/low thickness of LDT horizon. This area has been designated as 'Open Window' area in the Basic Mine Design report prepared by CMC. The area has been proposed to be mined by room and pillar method in 4 slices with parting of 10-12m between the slices as per the modified basic mine design report.

7.1.3. The 'open window' area is fairly well explored and has a geological reserve of 135.18 Mt which is substantial.

7.1.4. The main shafts and the surface infrastructure of the mine are located in the south western part of this 'open window' area. The shafts were sunk through the UDT horizon by adopting freezing method.

7.1.5. VI seam sub-crops along the entire 1.8 Km long NE – SW boundary in the rise side of the mine in open window area and then the sub-crop turns by around 90° to continue along the western boundary of the mine through open widow area and beyond.

7.1.6. The seam strikes roughly along NE - SW direction in the open window area and the minimum depth of VI seam roof in the NE-SW sub-crop zone has been projected to be about 98m. The thickness of VI seam gradually increases from 30m in the western side to 36m towards the eastern side in the open window area Figure 12: Roof contour and Isochore of VI seam (approximate) in Open Window area.

7.1.7. Coal/ OB thickness ratios in the boreholes of DOB series drilled in the open window area are given below:

Borehole no.	Overburden to roof of VI seam (m)	Thickness of VI seam (m)	Depth to floor of VI seam (m)	Coal: OB thickness ratio	Remarks
DOB 12	118.65	30.84	149.49	1:3.847	Rise most bore hole. LDT absent, UDT thickness- 93.9m.
DOB 1	131.80	29.40	161.20	1:4.483	LDT thickness -12.9m, UDT thickness - 99.90m
DOD 13	162.17	35.37	197.54	1:4.585	LDT absent, UDT thickness- 94.70m.
DOB 6	163.35	30.37	193.72	1:5.379	LDT absent, UDT thickness- 98.10m.
DOB 7	199.65	38.05	237.70	1:5.247	LDT absent, UDT thickness- 123.40m.

DOB 11	180.64	33.13	213.77	1:5.452	LDT thickness – 11.84m, UDT thickness – 108.51m
DOB 8	195.80	21.63 (seam faulted)	217.43	1:9.052	LDT absent, UDT thickness- 107.84m.
GDG 41	270.15	36.27 (VI seam) 6.40 (V seam) 8.69 (IV seam)	321.54	1:5.261	Upper seams (IV & V) are present in this borehole. LDT thickness – 12.04 m UDT thickness – 108.51 m

Table 40: Coal - Overburden thicknesses ratio at the base of VI Seam in open window area

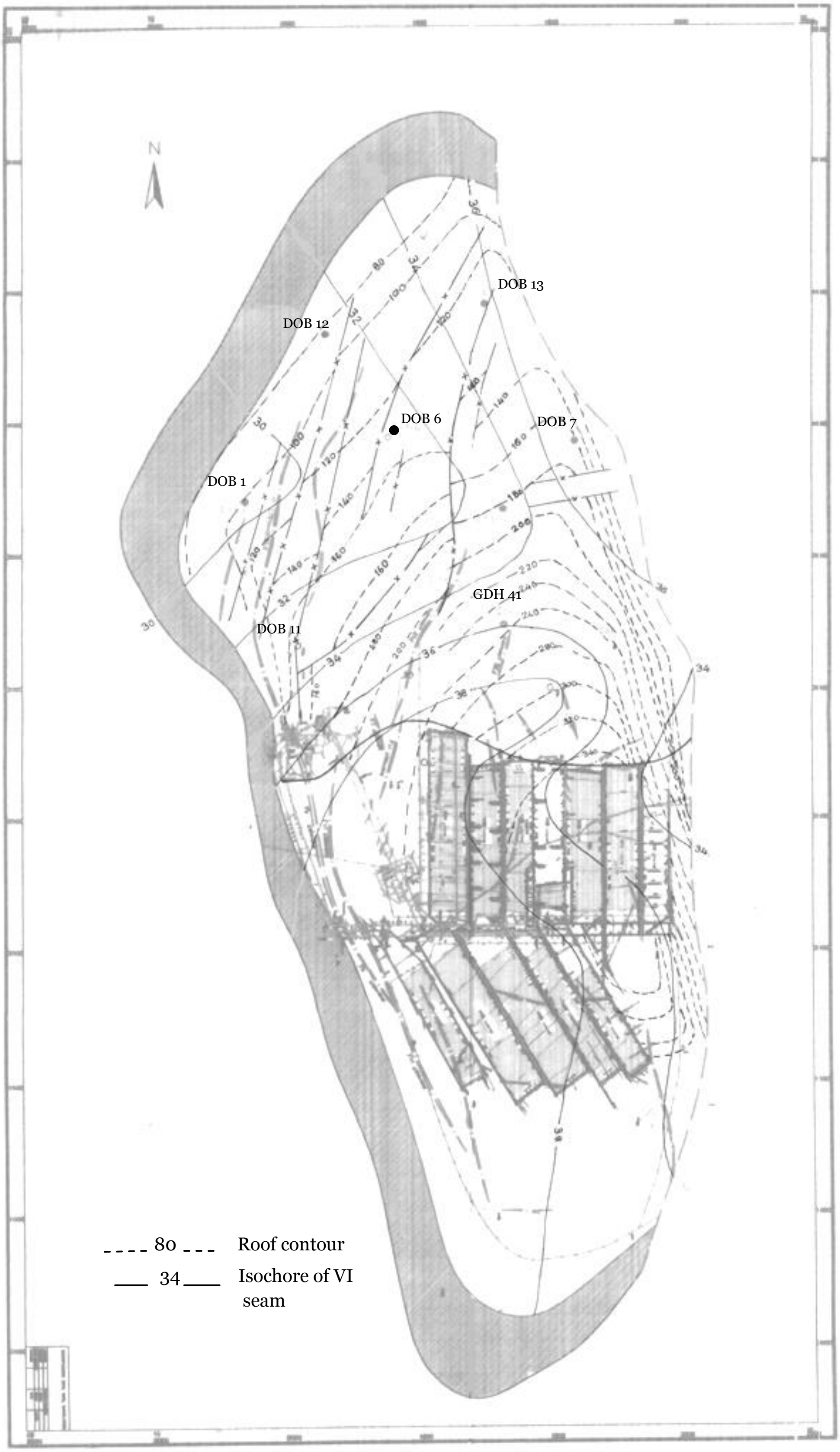

Figure 12: Roof contour and Isochore of VI seam (approximate) in Open Window area

7.1.8. It is apparent from Table 40 that the coal:overburden thickness ratio is increasing from 1:3.85 near sub-crop of VI seam at the rise side where the depth is around 150m at VI seam floor to 1:5.45 up to a depth of about 238 m at VI seam floor in the open widow area.

7.1.9. Considering the selling price of Barapukuria coal, the area between the NE-SW trending sub-crop of VI seam in the rise side and the line joining DOB 7 and DOB 11 (following seam floor contour) in the dip side can tentatively be assumed to have opencast mining potentiality with VI seam as base and therefore, further site specific studies as indicated below may be taken up for identifying a feasible area for opencast mining within this region.

7.1.10. Opencast mining in the open window area is fraught with severe technological and environmental challenges as outlined below.

7.1.11. Large scale dewatering of UDT aquifer through advance bore well (required to be drilled around opencast excavation for advanced dewatering) and through mine sump pumping during opencast mining will create significant draw down of water in the aquifer which may result in ground subsidence in the vicinity of opencast excavation. Therefore, hydro-geological studies must be carried out to predict the safe distance of opencast excavation from mine shafts, important surface structures and existing mine workings of VI seam so as not to affect their stability.

7.1.12. Opencast mining in the area will involve excavation of 94m to 120m thick UDT formation consisting of mainly sand beds, silt etc. The safe slope angles (particularly during rains) of the quarry high walls in UDT formation on all sides of the quarry (i.e., on dip side, on rise side and on two flanks) should be identified by slope stability studies. In addition, slope stability studies for internal and external soil dumps should also be taken up to ascertain maximum height and safe slope of these dumps for dump planning.

7.1.13. No data regarding ground bearing capacities of the alluvial layer at the top (usually 10-12m thick), of UDT sand bed (usually around 100 m thick) and of LDT bed (usually less than 10m where present) is available. Geo-technical studies need to be carried out to assess the ground bearing capacities of Madhupur clay, selected layers of UDT and LDT formations, by collecting fresh samples if necessary. However, the bearing capacities of these beds are expected to be sufficiently low to support the pressure exerted by earth moving equipment. It is also doubtful whether it will be possible to form individual benches in the UDT formation even during dry seasons. Thus there is need to develop an appropriate technology for excavation of such grounds.

7.1.14. In addition to the above, severe environmental degradation may occur due to the following reasons:

- Lowering of ground water table in the surrounding areas.
- Sterilization of land by external spoil dumps, as considerable initial excavation is involved to touch coal seam at a depth of 118m, high stripping ratio of the mine and low angle of repose of dump material.

7.1.15. It will therefore be essential to fill up the residual void after productive life of the mine by re-handling the material from surface external dump. This will ensure safety of future underground workings, free the land area occupied by external dump and help in re-establishment of the ground water regime of the UDT aquifer.

7.1.16. The decision on application of opencast mining technology in a part of the 'open window' area should be taken up only once the results of the above mentioned hydro-geological, geo-technical and slope stability studies are obtained and also a technology for safe excavation of around 100m thick unconsolidated sandy aquifer bed (UDT) is established and, thereafter a techno-economic feasibility study (TEFS) report is prepared using these data and information to arrive at the opencast mine boundaries, other geo-mining parameters of the mine and economics.

7.1.17. In addition to the above, additional drilling is required to be carried out in the open window area before planning of an opencast mine for firming up the trend, thickness and coal quality of the sub-crop region of VI seam falling in the 'open window' area.

7.1.18. An environmental impact assessment (EIA) study should also be carried out based on the TEFS to assess the environmental damage to be caused by such opencast mining in the open window area.

8. Mining of VI Seam in southern side of south district

8.1.1. In the report on Basic Mine Design, the area of VI seam in the mine was divided into three mining districts -North, Central and South (Figure 13: Grid Plan Showing VI Seam Area of Barapukuria Mine). The Central district was limited between latitude 10°37'00 in the north and latitude 10°21'40" in the south (1560 m long along strike) and the South sector was limited between latitude 10°21'40" in the north to sub-crop of VI seam in the south (1200m long along strike).

8.1.2. In the modification of basic mine design, only two districts were envisaged – North district and South district. The South district was formed by combining the central and south districts of basic mine design and the North district came under 'open window' area.

8.1.3. However, the sub-crop position of VI seam in the south side shown in the basic mine design report and that shown in modified mine design report varies considerably.

8.1.4. It is possible that VI seam of Barapukuria mine extends up to Phulbari mine in the south. Therefore, the geology, structure and other details of VI seam in the area between Barapukuria and Phulbari mines need to be firmed up by additional drilling before any mine planning for this area is taken up. The geological plans should, inter alia, show the location of dumps, surface infrastructures and mine boundaries as envisaged in project report of Phulbari opencast mine along with its lease areas.

8.1.5. As per the present geological map given in the modified basic mine design report, the minimum depth of VI seam sub-crop in the south is around 230m. It is not advisable to start an opencast operation with such high initial depth and other difficulties associated with opencast mining as mentioned earlier in this report. Due to higher initial depth of open pit in the south side compared to the north side, the problems related to slope stability and draw down created by pumping of aquifer will be more severe. The cost of initial excavation will increase phenomenally because of higher depth and also, the recovery of coal is likely to be much lower in the south side because of comparatively narrow pit width and flat slope of high walls on all sides of the pit. Therefore, this area, lying to the south of latitude 10°14'00" has to be worked by underground mining. A detailed planning exercise has to be taken up for firming up the method of development and extraction of this area and this, in turn, can be done only after the geology of the area is firmed up and reserves estimated.

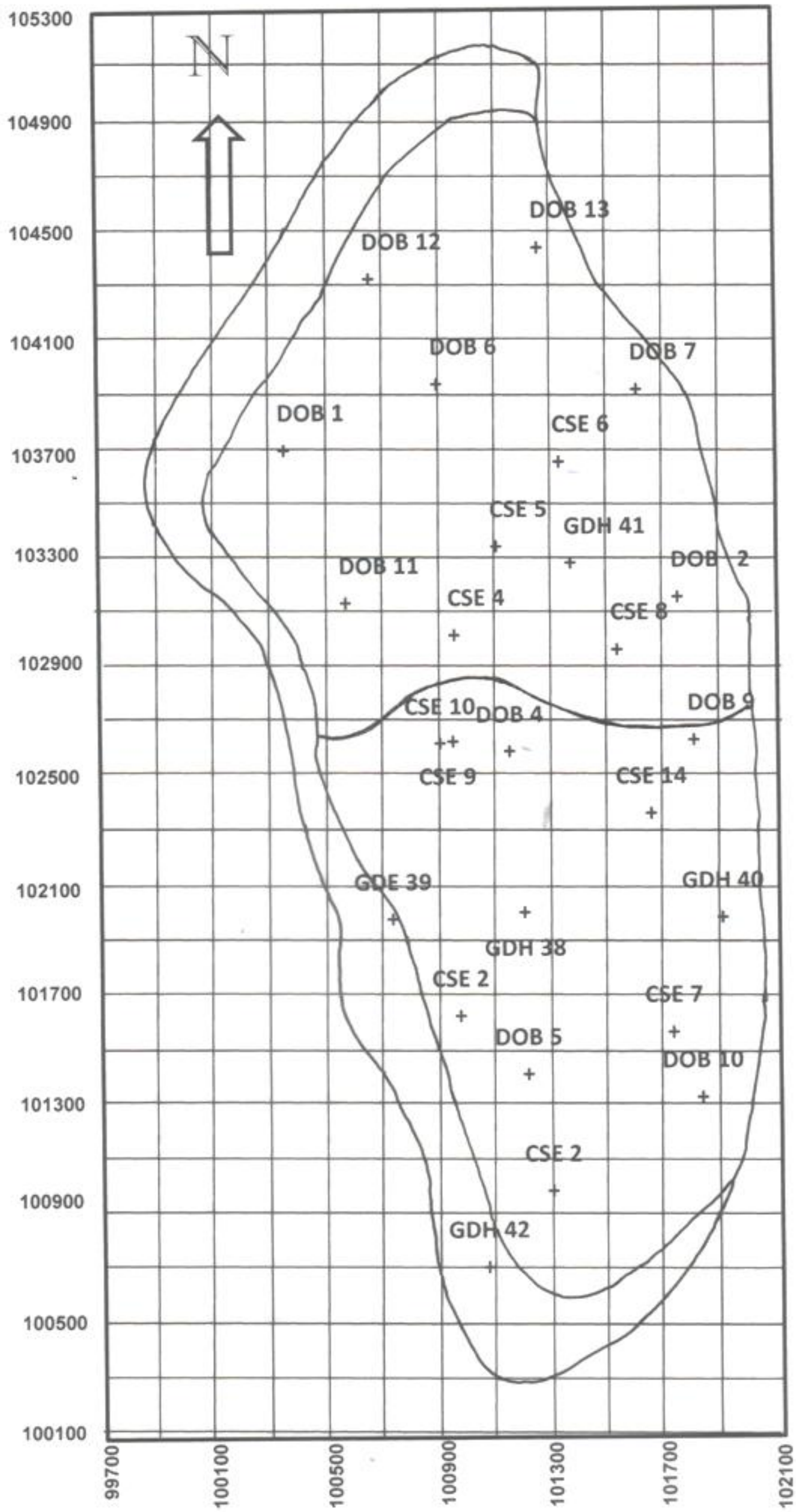

Figure 13: Grid Plan Showing VI Seam Area of Barapukuria Mine

9. Mining of upper seams

9.1.1. In Barapukuria leasehold area, there are five coal seams above VI seam. These seams are, from top downwards, Seams I, II, III, IV and V (Figure 14: Cross-sections Showing Upper Seams)The thicknesses of these seams along with thicknesses of their partings with upper seams and inferred reserves as estimated by CMC in their geological report are given in table below:

Coal seam	Thickness(m) Min-Max range Average	Average thickness of parting with upper seam (m)	Inferred reserve (Mt)	Remarks
I	4.57	198.43 (roof of seam to surface)	1.63	Developed only in one borehole (GDH 40). No quality data. Parting with LDT- 13.42m. LDT thickness- 44.50m.
II	13.95-15.24 14.44	54.27	21.06	Intersected in 4 boreholes in an area of spread of 1.02 km2 and close to UDT. More data required.
III-1	0.72-2.60 1.59	9.87	2.86	Low thickness of split sections and impersistent. Not workable by underground method.
III-2	0-1.16 0.65	2.55		
IV-1	0-0.46 0.27	19.24	23.84	Lower split has good thickness and persistent. Intersected by 11 boreholes in an area of spread of 1.89 km2. Has potential for economic exploitation.
IV-2	3.12-10.58 8.82	2.31		
V-1	0-0.63 0.33	16.30	16.40	Middle section is persistent with wide variation of thickness. Intersected by 14 boreholes in an area of spread of 2.40 km2.
V-2	1.74-10.37 4.78	4.89		
V-3	0-2.74 1.27	4.34		
Total inferred reserves of upper seams			**65.79**	

Table 41: Thicknesses of upper seams and their inferred reserves

9.1.2. Major parts of areas of these upper seams are vertically above the worked out area of VI seam and thus it is expected that these seams must have been damaged by subsidence which has already occurred. With the mining of subsequent slices of VI seam by caving, these virgin upper seams will further be damaged.

9.1.3. These seams can only be extracted by opencast mining with V seam as base, after allowing some time for stabilisation of strata above VI seam following depletion of reserves of this seam. Approximate minimum and maximum depths of floor of V seam considering pre-mining surface and underground levels are respectively 150 m and 340 m. The stripping ratio of the mine with V seam as base is likely to be very high considering the average thicknesses and areas of these upper seams.

9.1.4. Further, exploration of upper seams is not yet complete and hence no concrete picture of structure, reserves and quality of these seams are available at present. The structure of the coal seams will also undergo substantial changes from the present after the occurrence of the predicted maximum subsidence of around 24m due to mining of VI seam. In addition, there are technological problems of excavation of the 100m thick water bearing unconsolidated sandy UDT beds and poorly consolidated clayey LDT bed by opencast mining as mentioned earlier in this report.

9.1.5. In view of the above factors, opencast mining of the upper seams may become viable only at a distant future.

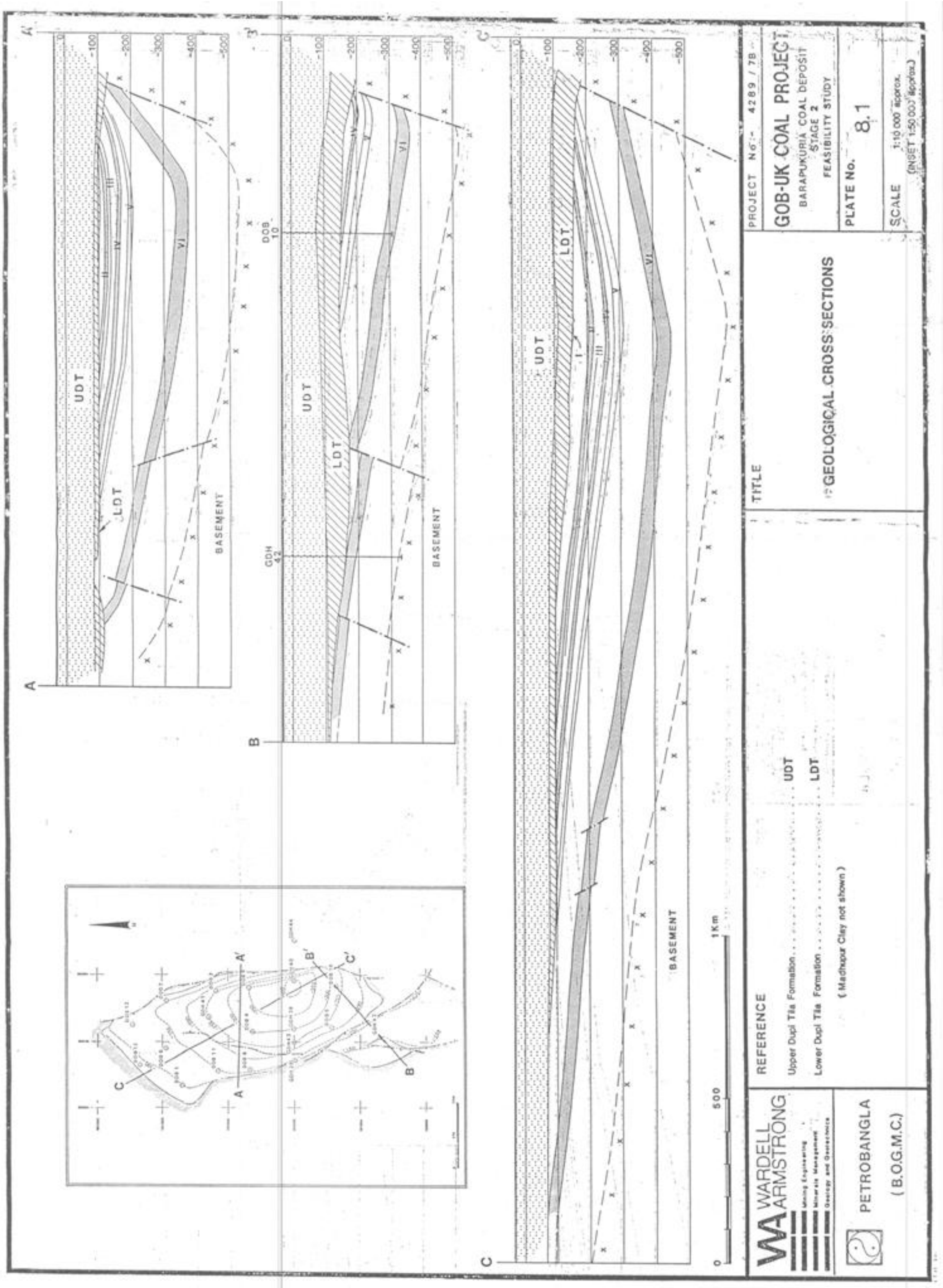

Figure 14: Cross-sections Showing Upper Seams

10. Feasibility of adopting stowing method for extraction of VI seam

10.1.1. Hydraulic sand stowing has been successfully used in many countries for extraction of thick seams in multi-slices adopting longwall technology with individual supports. Adoption of stowing in Barapukuria mine will result in many advantages in mining of the extra thick VI seam as outlined below:

- Reduction in strata control problems while working in multi-slices
- Reduction in the risk of spontaneous combustion
- Improvement in face ventilation as longwall mining with panel barriers can be practiced
- Reduction in the make of water from aquifer due to reduction in ground movement.

10.1.2. Thus, for safety of miners and for conservation of coal, adoption of stowing may be given careful consideration in Barapukuria even if that results in increase in cost of mining.

10.1.3. However, the following constraints are to be overcome for adoption of stowing in Barapukuria mine:

- Sand transportation from Jamuna river would be required which is located at a distance of 50-60 km.
- Design of Powered supports would need modification to accommodate the stowing pipes and to have rear support extensions etc.
- Production process will slow down due to extra shift time consumed for stowing operation. Hence, additional longwall sets will be required for achieving the same production level of 1Mtpa. Also, the existing powered supports will have to be replaced by newer versions.
- Additional capital investment would be required for transportation of sand from river, installation of stowing arrangements in the mine etc.
- Additional operating cost will be incurred for carrying out stowing operation.

10.1.4. There are problems of fire, strata control and water inflow in the mine which is likely to be more pronounced during mining of subsequent slices of the thick VI seam. It is therefore imperative that other alternative methods like mining of slices in ascending order (in VI seam) with stowing may be considered at this stage for ensuring mine safety and conservation of coal reserves at Barapukuria for the future.

10.1.5. Adoption of stowing in mechanized longwall face worked with DERD shearer and Powered Supports would be possible after addressing issues highlighted in para 10.1.3 above. Therefore, detailed studies on availability and transport of stowing material should be taken up, supply of tailor made Powered Supports for stowing faces has to be ensured after discussion with the manufacturers and thereafter, a detailed mine planning need to be undertaken to work out the number of stowing panels required for maintenance of production level along with economics.

10.1.6. Extraction of slices in ascending order with stowing will greatly improve safety and recovery but actual implementation of the method will require tremendous efforts in identification of stowing material and establishing stowing technology.

10.1.7. It must be noted that Barapukuria presents a unique case with one of the most difficult mining conditions in the world. Hence, solutions to these problems cannot be found within the domain of conventional mining technologies practiced elsewhere. It is therefore imperative that innovative technology for stowing in powered support faces should be developed in collaboration with the equipment manufacturers to suit the local conditions. Also, other related studies should be initiated immediately. The economics of the entire system should be worked out after all the technical issues are resolved and appropriate decisions taken for implementation.

11. Issues of strategic importance

1. Barapukuria mine is the first coal mining venture of Bangladesh. Therefore, it is important that it sets a good example for the development of coal sector of the country. It is, in this premise, certain issues of strategic importance have been identified and discussed below.

2. These issues can be broadly classified under the following heads:

- Mining technology
- Mine safety
- Conservation of coal resources
- Mine management

11.1. Mining technology

11.1.1. The mine is presently working the central part of the leasehold area. The northern part is not being worked as the area is lying below the 'open window' area. The southern part cannot be worked at present because there is no approach to this area from the central part which is served by the existing entries and development roadways.

11.1.2. The central part being worked at present is faced with multitude of problems like:

- poor ventilation of workings, existence of active fire and presence of CO in working places
- high make of percolation water from aquifers, which is expected to increase during mining of subsequent slices,
- strata control problems, which will also increase during mining of subsequent slices,
- under utilisation of installed mine capacity, etc.

11.1.3. These problems are likely to be compounded during working of 2nd and subsequent slices of VI seam. The future prospect of mining in the central part is dependent on how effectively and expeditiously the above safety and production related issues are addressed. Therefore various studies recommended in this report should be carried out expeditiously through independent agencies of repute so that appropriate strategies can be developed to deal with these problems.

11.1.4. Production from the mine may be greatly increased in medium term if techno-economic feasibility of opencast mining in a portion of 'Open Window' area is established while addressing social concerns and environmental clearance obtained. Therefore, the relevant studies as suggested in this report should be taken up on priority basis.

11.2. Mine safety

11.2.1. The mine has the presence of active fire in one of the panels which is not effectively sealed off and presence of CO is often recorded in working places which poses potential danger to mine workers. The coal is susceptible to spontaneous combustion and there is great possibility of occurrence of fire in goaf during mining of 2nd and subsequent slices in spite of taking precautionary measures.

11.2.2. Fire in goaf will be difficult to control as the goaves of adjacent panels are connected due to virtual absence of barriers. Rigorous R&D efforts at mine level are required to be taken up to find out an effective and foolproof solution to this problem so as to reduce the potentiality of unsafe condition in the mine caused due to the presence of active fire and hence, presence of CO. In the medium term application of stowing method may be considered.

11.2.3. The mine has a history of inundation. There exists an apparent risk of ingress of water into the mine workings from the highly water bearing UDT aquifer and this risk will increase with extraction of successive slices with caving. Scientific studies for prediction of possible future make of water in different stages of mining must be carried out and adequate protective measures must be taken in the mine to deal with any emergent situation arising from inrush of water.

11.3. Conservation of coal

11.3.1. Conservation of coal resources of Bangladesh is of strategic importance. Therefore, the mining method adopted must be oriented towards conservation of coal.

11.3.2. Extraction of thick seams by underground mining is associated with poor recovery of coal. The problem is compounded if thick seams occur in a multiple seam deposit as in Barapukuria.

11.3.3. Application of stowing method ensures safe and planned recovery of coal, protects the upper coal seams from damage and reduces surface subsidence thus preventing damage to agricultural land. All these are important for successful and environment friendly mining in Barapukuria.

11.3.4. The necessary studies related to adoption of stowing may therefore be initiated immediately.

11.3.5. From the point of view of conservation of coal resources, opencast mining is most preferred option wherever feasible. The studies suggested for opencast mining in the north side of Barpukuria may therefore be taken up on priority. However, opencast mining should not be attempted at the cost of environmental damage and/or social strife.

11.3.6. Based on current plan of Barapukuria mine, planned recovery of coal from VI seam is around 28.5%. However, the recovery percentage can substantially improve if after detailed studies open cast mining is found to be feasible and implimentable in a part of open window area in the north and/or if a decision is taken for adoption of stowing method after resolving all the constraints as envisaged in the report.

11.4. Mine management

11.4.1. Barapukuria mine is being operated by M&P contractor employing key personnel from People's Republic of China. The official distribution of work and responsibilities between the M&P contractor and BCMCL is not known to our team. However, from discussions with BCMCL management it appears that the local management have little access to vital data relating to performance of the machines, mine survey data, mine ventilation data, strata stress monitoring data or mine cost data. Plans and sections and other records are mostly maintained in Chinese.

11.4.2. For ensuring effective monitoring of operations and control by the BCMCL management, it is highly desirable that necessary organisation should be created at mine level for surveying, preparation of mine plans and sections, for conducting ventilation surveys and mine air analyses and for monitoring of safety status.

11.4.3. Also, all reports/records including machinery performance data must be made available to BCMCL management in English and language understood by local management as and when these are prepared. Similarly details of cost analysis data may also be submitted by M&P contractor to the local management. Plans and other records should also be maintained both in English and in Chinese.

11.4.4. In spite of long exposure of BCMCL engineers and mine workers to the sophisticated mining practice being followed in Barapukuria mine, there has been limited development of their expertise in respective fields. It is therefore recommended that a Training Need Analysis (TNA) study be taken up in the mine to identify the training needs of engineers, miners and maintenance technicians. Based on the results of this study, a structured programme for capacity building should be developed and implemented by the BCMCL management. A collaborative approach with other countries may also be considered for ensuring availability of trained indigenous manpower.

12. Recommendations and Conclusions

12.1. Exploration

12.1.1. The northern part of Seam VI is moderately explored but the southern part is under-explored. Further exploration of the seam is necessary to achieve the following objectives:

- Delineation of the trend of sub-crop of Seam VI.
- Up-gradation of reserves presently categorized under Rank ‘C’.
- Firming up of the geological structure, thickness and quality of Seam VI occurring in the southern part of the deposit up to Phulbari exploration block.

12.1.2. We understand that in new M&P contract, five new boreholes have been proposed by CMC–XMC consortium in the southern part of the mine. It is recommended that a few more additional boreholes are drilled in the northern, central (in virgin area) and southern part of Seam VI to achieve the above objectives. No fresh borehole should be drilled over or in the vicinity of already worked area.

12.1.3. All new boreholes are to be geo-physically logged before plugging these holes.

12.1.4. Fresh geological plans and sections of Seam VI for the entire area of the seam up to the boundary of Phulbari exploration block need to be prepared incorporating the data obtained from the new boreholes. The boundaries of different sectors of the area of Seam VI should also be re-defined and sector wise reserves re-estimated on the basis of new geological plans excluding the reserves already depleted.

12.1.5. Also floor contours of the 1st slice workings of VI seam need to be drawn using the existing survey data to facilitate control of the level of working horizon of the 2nd slice.

12.1.6. The upper seams (seams I, II, III, IV and V) are also not explored adequately. Most of the areas of the upper seams occur vertically above the present mining area of Seam VI. Therefore, exploration of these seams should be taken up later after depletion of Seam VI to avoid problems during drilling due to ground movement arising out of mining in Seam VI.

12.2. Hydrogeology

12.2.1. Barapukuria coal basin has two major aquifers: one, the Upper DupiTila formation whose thickness varies from 102 to 136m and the other, Gondwana formation with average thickness of 360m. The variation of water levels in these two aquifers during the last decade is given below:

Aquifer formations and observation bore wells made to study water levels.	**Water levels (m) above (+) and below (-) Mean Sea Level**		
	2000	2008	2011
UDT formation			
SHOB 4	+23.57	+19.74	-

SHOB 6	+24.61	-	+20.61
Gondwana formation			
CSE 14	-27.87	-	361.85
CSE 15	-35.36	-	297.92

Table 42: Variation of water levels in these two aquifers

12.2.2. From the above table it can be concluded that the source of underground discharge water is the connate water from large thickness of Gondwana formation. This water has been preserved within this formation since its deposition.

12.2.3. It has also been found that the water inflow from the strata into the mine is almost constant and there is no seasonal variation in the water inflow into the mine. This means that UDT water is not contributing to the mine inflow. The draw down observed in UDT water level in the above table is mainly due to use of UDT water for irrigation and industrial purposes and such draw down of UDT water level is also observed in other areas of Bangladesh.

12.2.4. At the end of year 2003 when the mine development was nearly complete and preparations for production of coal from different faces was going on, the average water discharge was 1,022.75 m^3/hr. At the end of October 2011 when first slice of all faces except 1116 was completed, the average discharge water was 1,480.04 m^3/hr. This water inflow is expected to increase during mining of the 2nd slice by LTCC method. Therefore, the required pumping capacity of the mine shall be nearly doubled to tackle this water inflow.

12.2.5. There are several faults in the mining area and these faults have considerable water transmissibility. Precautions should be taken to prevent sudden inrush of water while approaching these faults to prevent sudden inrush of water. Procurement of adequate numbers of long hole underground directional drilling machines for the mine is recommended for safely draining out water under pressure in advance in case of necessity.

12.2.6. It is further recommended that detailed studies should be carried out through modeling, preferably numerical modeling or any other suitable method, to review the application of LTCC method of mining to predict its impact on the overall stability of the mine and quantity of water inflow into the mine due to unstable conditions arising out of movement/caving of the overlying ground above Seam VI. If it is expected that there is possibility of sudden inrush of water into the mine workings, adequate precautionary measures may be planned and implemented to safeguard the life and property in the event of such inrush of water into mine workings.

12.3. Method of mining

12.3.1. Presence of a thick overburden of unconsolidated water bearing strata over the fractured and caved hard rock will result in development of two major problems related to mine safety during extraction of 2nd and subsequent slices of VI Seam and these problems will be gradually more pronounced as the number of slices extracted increases.

12.3.2. The first obvious risk is the possibility of ingress of water from the highly water bearing UDT aquifer into the VI Seam workings through flow paths developed due to any of the following reasons:

- Generation of fracture plane across the strained hard rock and LDT strata (particularly where it is thin) which extends to the base of UDT horizon.
- Passage of water from UDT through water transmitting faults.
- Opening of fault planes which were previously closed and non-water transmitting.

12.3.3. The other possible danger to the mine workings might result from dead weight of the thick unconsolidated strata. These strata, being unconsolidated, do not have any bridging capability and hence a component of the dead weight of these strata may be transmitted to the caved Gondwana rocks filling the goaf of VI Seam below which mining has to be done in different slices.

12.3.4. It may be noted that the safe thickness of coal/rock water barrier above the roof of VI seam as has been calculated in the Basic Mine Design report is based on two limiting conditions – (i) individual slice thickness will not be more than 3m and (ii) cumulative thickness of all slices will not be more than 15m.

12.3.5. However, both of these limitations of the empirical formula used will be breached as the individual thickness of slice will be 6m in LTCC method and the cumulative thickness of slices will be 24m. Also, the formula considered was based on experience of strata conditions and hydro-geological conditions of mines in China, over which certain additional thickness was added on ad-hoc basis to allow for the difference in strata conditions and hydrogeology of Barapukuria mine with those of Chinese mines.

12.3.6. However, after working of the 1st slice in Barapukuria mine, valuable geological, hydro-geological and geo-technical data regarding the mine are now available. Therefore, detailed numerical modeling study and/or other scientific studies are required to be carried out considering the data available and experience gained during working of the 1st slice, to predict the stability and behavior of the strata lying above VI Seam during extraction of 2nd and subsequent slices covering full thickness of VI seam.

12.3.7. Two sets of such studies should be made – one set considering descending slices with caving (for conventional multi-slicing method and LTCC method separately) and the other set considering ascending slices with hydraulic sand stowing (for conventional multi-slicing with barrier between panels) to predict, inter alia, the following:

- Increase in thickness of caved zone with the increase in cumulative thickness of slices extracted.
- Increase in thickness of water permeable fractured zone with the increase in the cumulative thickness of slices extracted.
- Make of water in underground working in each slice.
- Support resistance required in longwall face in each slice.
- Surface subsidence

12.3.8. The above studies for all the slices covering the full thickness of VI seam must be taken up and completed on priority basis for firming up the method of working of 'non-open window' area of the seam. A short term, slice-wise approach to mine planning must be avoided and a view in totality should be taken for selecting a mining system with an objective to achieve mine safety and conservation in the long term.

12.3.9. In addition to the present uncertainties relating to the future projections of ingress of water, stress level in strata and strata behavior; there is problem of existing fire in a panel in the 1st slice. It is imperative that the fire in the 1st slice be effectively dealt with and other protective measures taken to prevent further occurrence of fire in the goaves of 1st slice and other slices in future.

12.3.10. Therefore, it is recommended that the above mentioned studies should be carried out, their results analysed and all necessary actions taken before extraction of 2nd slice, for ensuring safety of the mine workers, mine property and conservation of coal resources.

12.4. Suitability of longwall top coal caving method

12.4.1. The 2nd slice of VI seam is proposed to be worked by longwall top coal caving (LTCC) method as per agreed terms of the new M&P contract signed between BCMCL and CMC-XMC consortium. A new set of face equipment (specifications not known) for longwall top coal caving method has been included in the M&P contract. The rated capacity of the mine will remain at 1 Mtpa.

12.4.2. Adoption of the proposed LTCC method under the geo-mining conditions obtained at Barapukuria mine vis-a-vis continuation of the existing multi-slice longwall mining method has been analysed in this report (only indicative assessment done) from the following perspectives.

- Method of mining, safety and recovery of coal reserves
- Mine production capacity, and
- Economics of mining.

12.4.3. It is recommended that detailed scientific and feasibility studies must be taken up by BCMCL for an exhaustive evaluation of these two systems and also of alternate mining systems with stowing for safe and efficient extraction of all the slices of the thick VI seam.

Method of mining, safety and recovery of coal reserves

12.4.4. As per the approved project document, the 2nd slice was to be worked below the 1st (topmost) slice after leaving a coal parting, the thickness of which is not mentioned. Assuming a parting thickness of 3m and a height of extraction of 2nd slice as 3m, the present mining system could have been continued with the existing sets of longwall equipment.

12.4.5. The detailed parameters of LTCC method are not yet known to consulting team. Assuming that the height of longwall face of 2nd slice in LTCC method will be 3m and the thickness of coal parting between floor of 1st slice and roof of 2nd slice longwall face will be 3m, the LTCC system will be achieving a higher coal recovery compared to the conventional multi-slicing system, where the entire parting coal of 3m (or so) will be lost in goaf with consequent increased risk of fire.

12.4.6. The cavability of the de-stressed coal parting or sublevel coal in LTCC method is not expected to pose much problem (except during initial period till some advance of the 2nd slice longwall face is achieved) considering the physico-mechanical properties of VI seam coal and the pressure of broken strata acting on the coal parting. In any case, regular caving of parting coal is also a pre-requisite for the successful operation of conventional multi-slice mining.

12.4.7. However, the rate of advance of the longwall face in LTCC method may be slowed down occasionally due to problem of blocky coal coming down on the rear conveyor or even boulders of roof rock coming on the rear conveyor due to absence of iron mesh netting in most of the panels of the 1st slice causing jamming of ARC which may involve manual intervention.

12.4.8. Snapping of ARC chain is also not ruled out. Such situation, apart from causing delay and reducing output, may warrant leaving some of the caved parting (sub-level) coal in goaf thereby increasing the risk of fire as the coal is very much susceptible to spontaneous combustion. Arrangements of nitrogen flushing and chemical treatment will therefore, have to be kept to mitigate risk of fire in goaf in LTCC panels as in case of conventional multi-slice panels.

12.4.9. LTCC method will involve extraction of 6m of coal in 2nd slice (cumulative 8.5 to 9m) compared to 3m (cumulative 5.5 to 6m) in case of conventional multi-slice mining. As the empirical formula for determining the height of the fracture zone considered in the Basic Mine Design of Barapukuria mine assumes individual height of slices to be not more than 3m, fresh scientific studies are required to be carried out to determine the height of fracture zone and safe thickness of coal/rock parting above VI seam to prevent disturbance to the UDT aquifer horizon before application of LTCC method. Also, additional data regarding behaviour of UDT, LDT and Gondwana strata as available from the experience gained during extraction of 1st slice in the mine may be utilised in the present studies to tailor the new model to suit to Barapukuria conditions.

12.4.10. However, it can be generally said that extracting the 2nd slice with 6m thickness (in LTCC method) will result in higher thicknesses of caved and fractured zones in Gondwana rocks compared to that with 3m height of extraction of 2nd slice (as in conventional multi-slicing method) and therefore, the following are to be reassessed during the above scientific studies for application of LTCC method:

- The support resistance required at the longwall face in LTCC method in 2nd and subsequent slices, and
- The quantum of flow of water from Gondwana aquifer in the goaf of 2nd and subsequent slices worked by LTCC method.

12.4.11. As the successive slices will be extracted by caving in descending order, the thickness of the caved zone will increase depending on the cumulative seam thickness extracted and the bulking factors of roof rock and un-extracted parting coal in the goaves. Thus the dead weight of the caved zone will increase while successive lower slices are worked, more so if LTCC method is adopted. Also, a component of dead weight of UDT and LDT strata over the caved zone will be transmitted to the caved rocks as the UDT and LDT horizons do not have ability to bridge the width of the caved zone. Thus, the support resistance required will increase as successive lower slices are extracted. Therefore, the specifications of the powered supports of LTCC panels should preferably be selected in such a manner that these can serve for two or three successive slices without being replaced before their stipulated life.

12.4.12. In addition to the above considerations, it must also be added that adoption of LTCC method will involve use of more sophisticated mining equipment and will require manpower with advanced skills compared to those required in the operation of conventional multi-slice mining system, which is presently being practiced in the mine and in which indigenous manpower have largely been trained.

Mine production capacity

12.4.13. A study of the periods of operation of individual panels of the 1st slice shows that the two shearers have never worked in the mine simultaneously except for a brief period between 25.2.2010 and 16.03.2010. But, it must also be mentioned that one of the two longwall sets was not available from end of September 2005 to middle of August 2008 (almost for 50% of the contract period) as the equipment were trapped within a panel which had to be sealed off due to fire.

12.4.14. The mechanical condition of the shearers, powered supports and other face machineries are not known due to absence of information at the mine level but the average daily production from the shearers are not below that envisaged in the Basic Mine Design Report of CMC.

12.4.15. Also, out of the four Road-headers procured for developing longwall panels, only two are in working condition and the other two are reportedly beyond rehabilitation. Considering all these, it can be said that the rated production of 1.0 Mtpa could be achieved by conventional multi-slice longwall system by deploying two sets of PSLW equipment in two faces in 2nd slice, provided the face equipment were refurbished/overhauled and two more road headers were added replacing the unserviceable ones for timely development of longwall panels.

12.4.16. After completion of the rated life of the existing face equipment, these could be replaced with updated versions of the equipment.

12.4.17. On the other hand, to ensure a regular production of 1.0 Mtpa from one set of LTCC equipment, appropriate actions have to be taken to minimize the following delays:

- Delay in face advance due to problem of jamming or damage of ARC, delay in clearing of roof coal etc.
- Delay during salvaging and re-installation of the LTCC equipment to the next longwall panel.
- Delay in taking up due maintenance and breakdown repair works

12.4.18. It is recommended that a system of proper spares management should be established and effective steps for up-gradation of skills of the operation and maintenance crew should be taken to reduce the above delays.

Economics of mining

12.4.19. For any investment decision, it is necessary to evaluate the comparative economics of the available alternatives. In the present case also, it is necessary to carry out discounted cash flow analyses for at least next 10 years (with 1st year as 2011-12) considering a rated production of 1.0 Mtpa for the existing longwall multi-slice system and the proposed LTCC system to compare the economics of these two systems. For this purpose, the operating and capital costs of the existing system has to be updated and a revised cost estimate (RCE) of the existing mine has to be prepared.

12.4.20. The economics of the two systems may then be compared in terms of their Financial IRRs and Economic IRRs or in terms of their NPVs, before taking a final decision regarding selection of a system.

12.5. Feasibility of opencast mining of VI seam in open window area

12.5.1. Seam VI sub-crops along the entire 1.8 Km long NE – SW boundary in the rise side of the mine in open window area and then the sub-crop turns by around 900 to continue along the western boundary of the mine through open widow area and beyond. The seam strikes roughly along NE - SW direction in the open window area and the minimum depth of VI seam roof in the NE-SW sub-crop zone has been projected to be about 98m.

12.5.2. The thickness of VI seam gradually increases from 30m in the western side to 36m towards the eastern side in the open window area. Coal:Overburden thickness ratios increase from 1:3.85 near sub-crop of VI seam at the rise side where the depth is around 150m at VI seam floor to 1:5.45 up to a depth of about 238 m at VI seam floor in the open widow area.

12.5.3. Considering the selling price of Barapukuria coal, the area between the NE-SW trending sub-crop of VI seam in the rise side and the line joining DOB 7 and DOB 11 (following seam floor contour) in the dip side can tentatively be assumed to have opencast mining potentiality with VI seam as base and therefore, further site specific studies as detailed below may be taken up for identifying a feasible area for opencast mining in this region.

12.5.4. Opencast mining in the open window area is fraught with severe technological and environmental challenges as outlined below.

- Large scale dewatering of UDT aquifer through advance bore well and through mine sump pumping during opencast mining will create significant draw down of water in the aquifer which may result in ground subsidence in the vicinity of opencast excavation. Therefore, hydro-geological studies must be carried out to predict the safe distance of opencast excavation from mine shafts, important surface structures and existing mine workings of VI seam so as not to affect their stability.

- Opencast mining in the area will involve excavation of 94m to 120m thick UDT formation consisting of mainly sand beds, silt etc. The safe slope angles (particularly during rains) of the quarry high walls in UDT formation on all sides of the quarry (i.e, on dip side, on rise side and on two flanks) have to be estimated by slope stability studies. In addition, slope stability studies for internal and external spoil dumps should also be taken up to ascertain maximum height and safe slope of these dumps for dump planning.

- No data regarding ground bearing capacities of the alluvial layer at the top (usually 10-12m thick), of UDT sand bed (usually around 100 m thick) and of LDT bed (usually less than 10m where present) is available. Geo-technical studies need to be carried out to assess the ground bearing capacities of Madhupur clay, selected layers of UDT and LDT formations, by collecting fresh samples if necessary. However, the bearing capacities of these beds are expected to be sufficiently low to support the pressure exerted by earth moving equipment. It is also doubtful whether it will be possible to form individual benches in the UDT formation even during dry seasons. Thus there is need to develop an appropriate technology for excavation of such grounds.

12.5.5. In addition to the above, severe environmental degradation may occur due to the following reasons:

- lowering of ground water table in the surrounding areas

- sterilization of land by external spoil dumps, as considerable initial excavation is involved to touch coal seam at a depth of 118m, high stripping ratio of the mine and low angle of repose of dump material.

12.5.6. It will therefore be essential to fill up the residual void after productive life of the mine by re-handling the material from surface external dump. This will ensure safety of future underground workings, free the land area occupied by external dump and help in re-establishment of the ground water regime of the UDT aquifer.

12.5.7. It is therefore recommended that the decision on application of opencast mining technology in a part of the 'open window' area should wait till the results of the above mentioned hydro-geological, geo-technical and slope stability studies are obtained and also. a technology for safe excavation of around 100m thick unconsolidated sandy aquifer bed (UDT) is established and, thereafter a techno-economic feasibility study (TEFS) report is prepared using these data and information to arrive at the opencast mine boundaries, other geo-mining parameters of the mine and economics.

12.5.8. An environmental impact assessment (EIA) study should also be carried out based on the TEFS to assess the environmental damage to be caused by such opencast mining in the open window area.

12.5.9. BCMCL may arrive at an appropriate decision regarding open pit mining of VI seam in the northern part of the deposit after completing the above studies and other activities discussed in this report.

12.6. Mining of VI seam in the southern side of south district

12.6.1. It is possible that VI seam of Barapukuria mine extends up to Phulbari mine in the south. Therefore, the geology, structure and other details of VI seam in the area between Barapukuria and Phulbari mines need to be firmed up by additional drilling before any mine planning for this area is taken up. The geological plans should, inter alia, show the location of dumps, surface infrastructures and mine boundaries as envisaged in project report of Phulbari opencast mine along with its lease area.

12.6.2. As per the present geological map given in the modified basic mine design report, the minimum depth of VI seam sub-crop in the south is around 230m. It is not advisable to start an opencast operation with such high initial depth and other difficulties associated with opencast mining as mentioned under para 6.4 above.

12.6.3. It is therefore recommended that this area, lying to the south of latitude 101400 has to be worked by underground mining. A detailed planning exercise has to be taken up for firming up the method of development and extraction of this area and this, in turn, can be done only after the geology of the area is firmed up and reserves estimated.

12.7. Mining of upper seams

12.7.1. In Barapukuria leasehold area, there are five coal seams above VI seam which is being extracted now. These seams are, from top downwards, Seams I, II, III, IV and V. Major parts of areas of these upper seams lie vertically above the worked out area of VI seam and must have been damaged by subsidence which has already occurred. With the mining of subsequent slices of VI seam by caving, these virgin upper seams will be further damaged.

12.7.2. These seams can only be extracted by opencast mining with V seam as base, after allowing some time for stabilization of strata above VI seam following depletion of reserves of this seam. Approximate minimum and maximum depths of floor of V seam considering pre-mining surface and underground levels are respectively 150 m and 340 m. The stripping ratio of the mine with V seam as base is likely to be very high considering the average thicknesses and areas of these upper seams.

12.7.3. Exploration of upper seams is not yet complete and hence no concrete picture of structure, reserves and quality of these seams are available at present. In addition, there are technological problems of excavation of the 100m thick water bearing unconsolidated sandy UDT beds and poorly consolidated clayey LDT bed by opencast mining as mentioned earlier.

12.7.4. In view of the above factors, opencast mining of the upper seams may become viable only at a distant future.

12.8. Feasibility of adopting stowing method for extraction of VI seam

12.8.1. Adoption of stowing in Barapukuria mine will result in many advantages in mining of the extra thick VI seam as outlined below:

- Reduction in strata control problems while working in multi-slices
- Reduction in the risk of spontaneous combustion
- Improvement in face ventilation as longwall mining with panel barriers can be practised
- Reduction in the make of water from aquifer due to reduction in ground movement.

12.8.2. However, the following constraints are to be overcome for adoption of stowing in Barapukuria mine :

- Sand transportation to mine from Jamuna river located at a distance of 50-60 km.
- Modification in design of Powered supports to accommodate the stowing pipes and rear support extensions etc.
- Production process will slow down due to extra shift time consumed for stowing operation. Hence, additional equipment will be required for achieving the same production level of 1Mty.
- Additional capital investment will also be required for transportation of sand from river and for installation of stowing arrangements in the mine etc.
- Additional operating cost will also be incurred for carrying out stowing operation.

12.8.3. There are problems of fire, strata control and water inflow in the mine which are likely to be more pronounced during mining of subsequent slices of the thick VI seam. It is therefore imperative that other alternative methods like mining of slices in ascending order (in VI seam) with stowing may be considered at this stage for ensuring mine safety and conservation of coal reserves at Barapukuria for the future.

12.9. Underground mine environment

Heat and humidity problems

12.9.1. The following arrangements should be done for prevention of water evaporation and reduction of humidity in the mine:

- The water percolated in the intake airways should be channelized through the covered drains as far as practicable, so that it does not come in contact with the intake air.
- The channelized water through the covered drains should be diverted towards the return circuit, which will reduce the humidity at the working faces.
- The proper planning of drainage of percolated water should be done in such a way that it comes in the minimum intake routes of the mine. This can be achieved either by recognizing the water drainage system or the ventilation circuit of the mine.

12.9.2. The heat and humidity added to the mine air can be diluted by increasing the air quantity flow to the working areas. This will entail larger pressure difference across the longwall faces. If the pressure across the longwall face increases, leakage of intake air to the goaf area also increases. This may cause spontaneous combustion in the goaf resulting in fire in the longwall faces. Presently, the intake air to some of the longwall faces has been decreased to reduce this pressure difference across the longwal faces. Therefore, an optimum quantity of air should be allowed into the longwall faces to avoid this fire problem. With this optimal air quantity flow in the panel, efforts should be made to achieve whatever improvement in the mine climate as possible.

12.9.3. If the heat and humidity problem is not solved by optimizing the air quantity in the longwall panels, a detailed study should be carried out and air cooling system should be installed in the longwall panels for solving the heat and humidity problems.

Ventilation problems of the mine

12.9.4. In order to improve the ventilation system of the mine, the following measures should be adopted:

- A detailed ventilation survey (pressure-quantity and temperature survey) should be carried out in the mine.
- Ventilation network model of the mine should be developed.
- Exhaustive computer simulation exercise should be carried out by using different variants for reorganizing the ventilation system of the mine.
- One of the conditions worth simulating by using the ventilation network model is the construction of a ventilation shaft at a suitable location, which will reduce the air travel distance, minimize air pressure loss and improve the ventilation of the mine. It is expected that in this condition, the pressure requirement for ventilating the mine may reduce to a significant extent which will reduce the total power consumption of the system, and save a large amount of energy cost for the company.
- The ventilation system should be reorganized in such a way that the ventilation pressure requirement of the system is the minimum. This will have a favorable effect on the fire problems of the mine and reduce the occurrence of new fires.

- A well equipped ventilation laboratory belonging to BCMCL should be set up in the mine with arrangements for chemical analyses of mine air samples, gas chromatography, temperature monitoring, dust monitoring, ventilation surveys, determination of inseam CH4 content etc.

Spontaneous combustion and fire problems

12.9.5. The worked out panels should be properly sealed with explosion proof stoppings. Wherever water accumulation is taking place behind the seals, gully traps or suitable draining arrangements should be provided and precautions should be taken so that only water is allowed to come out without ingress of air into the sealed off areas.

12.9.6. All the sealed off panels irrespective of occurrence of fire should be monitored by collecting the air samples behind the seals. The fire area air sample analysis should be carried out by using microprocessor-based gas chromatographs. The different fire ratios, viz. Graham's ratio, Young's ratio, Willet's ratio, Morris ratio, oxygen content etc. should be calculated for all the air samples drawn from the sealed off panels and the trend of these ratios should be monitored very closely for assessing the initiation of fresh fire and the status of existing fire.

12.9.7. The air leakage around the seals should be monitored by using tracer gas technique or suitable scientific method. In addition, pressure difference across the seals should also be monitored. If the pressure difference across the seals is high, pressure balancing should be carried out for those seals to avoid the leakage of air into the fire areas.

12.9.8. The sealed off panels under fire should be closely monitored and if the fire activity is increasing, inertisation of the goaf areas should be adopted so that the fires in those panels are not propagating further.

12.9.9. While working the 2nd and subsequent slices, fire may play a major role for successful operation of these panels. Therefore, in order to avoid the occurrence of fire in those panels, pro-active inertisation should be adopted, i.e. while the longwall faces are retreating, the back side of the goaf is flushed with the inert gas continuously for avoiding the occurrence of fire in the goaf.

12.9.10. A fresh study on the proximate analysis, maceral content, cleat intensity and extension, thermal conductivity of roof and floor rocks and crossing point temperature of the coal of Seam VI should be carried out and its susceptibility to spontaneous combustion and incubation period should be re-established for properly planning the mining operation.

12.9.11. As Seam VI is very much susceptible to spontaneous combustion, rigorous R&D efforts should be initiated in the mine level to deal with the problem of spontaneous combustion effectively. An in-house R&D set up should be established in the mine for this purpose.

Other underground environmental problems

12.9.12. The gas content of the seam has been determined at the initial stage. Since the mine has been working for a number of years, the gas content of seam should be determined by both direct and indirect methods for authenticating the gas content of the seam.

12.9.13. The guidelines for carrying out borehole gas survey in the mine should be developed and in addition the gas survey at a regular interval should be carried out for taking preventing measures if there is an increase in the gas emission in the mine.

12.9.14. In addition to CH_4, other gases emitted by Seam VI should be measured at regular interval of time to avoid the sudden occurrence of other hazardous gases in the mine atmosphere.

12.9.15. The statute for ARD sampling should be developed and the method of ARD monitoring including the type of instrument to be used should be defined in the statute.

12.9.16. The free silica content of ARD should be determined from time to time as a normal practice.

12.9.17. Dust suppression arrangements along the gate belt conveyor and transfer points should be installed for reducing the ARD concentration in the mine atmosphere.

12.9.18. In order to prevent the occurrence of coal dust explosion, the guidelines for dust sampling should be developed and it should include the method of sampling, its frequency, parameters to be determined from the samples, etc.

Mine Hazards and Safety

12.9.19. Presently to reduce the mine accidents and improve safety, many countries of the world are following the approach of "Risk Assessment and Management". In Barapukuria Coal Mine, this risk assessment and management technique should be implemented for improving the safety in the mine.

12.9.20. Emergency response system should be designed and emergency organization plan should be formulated and implemented.

12.9.21. Training and retraining of workers should be put in place for improving their knowledge in the domain of their working and awareness about safety.

12.9.22. Periodical medical examination (PME) of workers should be introduced to know the status of their health with respect to dust related and other diseases.

12.9.23. In hot and humid environment, apart from heat stroke, people are prone to other types of accidents due to their mental state. Therefore, the mine climatic condition, especially with reference to heat and humidity should be improved as discussed earlier.

12.9.24. The systematic support rule (SSR) should be formulated and imposed for reducing the accidents due to roof and side falls.

12.9.25. Safety rules, bylaws, standard operating procedure (SOP) and code of practices should be formulated and implemented.

12.9.26. The recommendations for preventing the occurrence of other hazards as described in previous sections should be implemented.

12.9.27. Some of the important plans, viz. water danger plan, ventilation plan with all ventilation control devices (stoppings, airlocks, regulator, air-crossing, etc.), dust sampling plan etc. should be prepared and updated at regular intervals.

12.9.28. Regular subsidence monitoring and treatment of the subsided area on the surface should be undertaken on priority basis. A survey organization should be set up at the mine level for day to day underground survey and also for subsidence survey.

12.9.29. Organization at the mine level should be developed for regular safety monitoring with full time management.

12.10. Mine infrastructure

12.10.1. Detailed recommendations covered in section 5.7.

12.11. Surface environment

12.11.1. Detailed recommendations covered in section 5.8

12.12. Mine organization

12.12.1. Detailed recommendations covered in section 5.9.