THE WORLD BANK GROUP ARCHIVES

PUBLIC DISCLOSURE AUTHORIZED

Folder Title:	India - Coal Sector - Coal India - Investment and Production Planning
Folder ID:	30436852
Series:	Project lending and operational support
Dates:	01/01/1979 - 12/31/1985
Fonds:	Records of the Industry and Mining Sector
ISAD Reference Code:	WB IBRD/IDA IND-03
Digitized:	11/27/2023

To cite materials from this archival folder, please follow the following format:
[Descriptive name of item], [Folder Title], Folder ID [Folder ID], ISAD(G) Reference Code [Reference Code], [Each Level Label as applicable], World Bank Group Archives, Washington, D.C., United States.

The records in this folder were created or received by The World Bank in the course of its business.

The records that were created by the staff of The World Bank are subject to the Bank's copyright.

Please refer to http://www.worldbank.org/terms-of-use-earchives for full copyright terms of use and disclaimers.

THE WORLD BANK
Washington, D.C.
© International Bank for Reconstruction and Development / International Development Association or
The World Bank
1818 H Street NW
Washington DC 20433
Telephone: 202-473-1000
Internet: www.worldbank.org

PUBLIC DISCLOSURE AUTHORIZED

INDIA - Coal Sector
3.8 COAL INDIA INVESTMENT AND

Archives
30436852
R1986-068 Other #: 75 Box # 36662B
India - Coal Sector - Coal India - Investment and Production Planning

DECLASSIFIED
WBG Archives

(1)

SANCTIONED PROJECTS

NAME OF PROJECT	CAPTION CAPACITY M.T.Y.	SANCTIONED CAPITAL & Date Rs. Million	ACCORDING F.R. ORIGINAL COMPLET. DATE	PRODN. 79-80	ACTUAL PRODN. 79-80	EXPECTED SLIPPAGE YRS.	BROAD REASONS FOR SLIPPAGE
1	2	3	4	5	6	7	8
EASTERN COAL FIELDS —							
Chinakuri I & 2 Pits — U/G. Reconstruction Superior grade.	1.00	89.20 6/75	1982-1983	0.45	·23	~~3~~ 2-4	unexpected behaviour of the roof. Tech parameters revised now. Power + Labour
Amritanagar U/g Reconstruction Superior grade	1.14	108.50 5/76	1982-1983	0.46	0.26	2-3,	Erratic behaviour of the seams, Delay in CHP, Labour trouble. Power.
Kottadih. u/g Reconstr. Superior grade.	0.87	76.60 8/77	1981-82	0.22	0.12	~~3~~ 2-3	Involve thick seam mining. Delay in siding and Starting of 1st Slice.
Dhemo-main. u/g Reconstr. Superior grade.	1.00	119.50 8/77	82-83	0.33	0.28	2-3	C.H.P., Tech change due to behaviour of roof. Development
Ratibati u/g Reconst. Superior grade.	0.90	99.40	84-85	0.58	0.15	3	Shaft-sinking and CHP. Labour unrest Power trouble
Ninga u/g Reconst. Superior grade	1.60	152.50 8/77	83-84	0.60	0.39	3	CHP + Power Failure of Dewatering. Labour.
North Searsole. u/g Reconst. Superior grade	1.16	99.20 10/78	88-89	0.23	0.12	~~not~~ 2-3	CHP. Siding delayed by land acqn.

ECL contd.

1	2	3	4	5	6	7	8
1. J. K. Nagar U/G. Reconst. Superior	0·86	151·72 4/79	1984-85	0·20	0·12	3	Delay in Shaft Sinking, and development. CHP, Siding
1. Bankola u/g Reconst. Superior grade	1·08	81·80 4/79	1985-86	0·55	0·40	-(?)	CHP. Siding
0. Satgram u/g Reconst. Superior grade	1·20	263·70 4/79	88-89	0·20	0·12	1	Shaft-sinking CHP and Siding.
1. Perbelia u/g Reconstruction Superior Grade	0·80	125·40 4/79	86-87	0·35	0·10	2	Shaft Sinking Development CHP.
12. Bahula u/g Reconstruction Superior grade	1·20	113·50 10/79	85-86	0·70	0·72	Ahead of Schedule (?)	CHP Siding
13. Sodepur. u/g Reconst. Superior	0·55	41·60 7/78	83-84	·34	·15	-	Low progress due to power trouble May be delayed
4. Kunostoria u/g Reconst. Superior	0·66	34·70 10/78	83-84	0·49	·21	-	
15. Sketalpur u/g Reconst. Superior	0·40	47·90 7/78	84-85	0·18	·06	-	
16. Purushottampur New o/c Superior	0·46	67·30 4/79	82-83	0·11	·10	(?)	Siding zero
17. Rajmahal New o/c Power coal	5·00	874·80 7/80	85-86	-	-		

1	2	3	4	5	6	7	8	9
10) NICHITPUR – New O/C Noncoking	7/80	0·45	74·90	83-84		On Schedule		
11. Keshalpur O/C new Non-coking	7/80	0·51	71·70	83-84		on Schedule.		

NAME & TYPE OF PROJECT	DATE OF SANCTION	CAPTION CAPACITY MTY	SANCTION CAPITAL Rs. Mln	ACCORDING FR. COMPLETION DATE	PRODN. 79-80	ACTUAL PRODN 79-81	SLIPPAGE YRS.	BROAD REASON FOR SLIPPAGE
CENTRAL COAL FIELDS								
Gidi 'C' Superior grade O/C new	6/73	0·60	31·38	78-79	0·60	·50	nil,	
Jagannath O/C - Power Coal.	8/73	1·00	57·78	78-79			no Slippage.	
3. Jayanti new O/C. Power Coal	2/78	6·00	773·70	84-85	1·20	1·10	nil	
4. Bina new O/C Power coal	5/79	4·50	569·10	84-85	1·50	1·20	"	
5. Gorbi New O/C Power Coal	8/73	1·00	75·22	78-79	1·00	·90	"	
6. Taping New O/C. Coking Coal	11/76	0·60	49·90	81-82	·40	·25	No Slippage	
7. Jhingurdah new O/C Power Coal	11/77	3·00	248·70	79-80	2·80	2·02	Nil.	
8. Ramgarh new O/C Coking Coal	8/77	3·00	418·70	82-83	1·00	·29	2	Delay in washery Construction
9. Dakra O/C - Reconstn Power Coal	10/78	1·30	101·20	81-82	1·00	0·872	nil	
10 Sirka O/C O/C - new Superior Coal	10/78	1·00	131·80	82-83	·50	·36	nil.	

1	2	3	4	5	6	7	8	9
K.D Hesalong new o/c Power coal	12/78	1.50	144.40	84-85	On Schedule			
Seligara o/c - new superior coal	7/78	0.37	40.70	83-84	- on Schedule			
Kedla o/c new coking coal	9/79	1.00	173.0	82-83	.40	.24	1	Delay due to washery.
Nandira new - u/g superior	9/73	0.65	64.00	81-82	.34	.06	4	Technical difficulty Development
5. Kedla u/g - new Coking coal	6/75	1.00	97.50	82-83	.50	.12	4	Development Technical Washery.
16. Karo special new-u/g Coking coal	2/78	.50	55.0	82-83	-	-	2	Development.
7. Govindapur new u/g. Coking	7/80	0.80	165.96	89-90	—	Maybe on schedule		
18. Swang u/g - Recons. Coking	3/74	0.80	74.0	80-81	.50	.25	4	Development Technical.
9. Talcher u/g - Reconst. Superior	5/76	1.30	160.80	83-84	.39	.19	4	Management Staff Shifting Technical
0. Ara o/c - Recons. Power coal + coking	7/80	1.10	156.0	85-86	.30	.33	—	

Feasibility Report Submitted to the companies.

NAME OF PROJECT 1	TYPE - QUALITY OF COAL 2			CAPTION CAPACITY MTY 3	ESTIMATED CAPITAL Rs. LAKHS 4	DATE OF SUBMISSION 5	GESTATION PERIOD AFTER ZERO DATE Yrs. 6
Kumardihi A & B ECL	u/g	Reconst.	Superior	1·20	306·40	11/79	8
Chora ECL	"	"	"	2·22	550·90	8/80	10
Shampur ECL	"	"	"	1·60	936·90	10/80	7
Kalidaspur ECL	u/g	New	"	0·57	167·74	2/80	7
Jhanjra ECL	"	"	"	2·80	1265·60	6/80	10
Kottadih ECL	o/c	New	"	3·00	1056	8/80	5
Mandman ECL	"	"	Inferior	1·50	404·0	5/80	5
3. Block-II BCCL	o/c	"	Coking	2·50	1017	5/80	5
1. Muraidih BCCL	"	"	Inferior ~~Power~~	2·50	316·70	5/80	5
0. Kakri CCL	o/c	"	Power Coal	2·50	439·50	3/79	7
1. Selected Dhori CCL	o/c	Recons.	Power Coal	2·25	243·9	8/79	7
2. Dhori-west CCL	"	"	"	1·50	315·20	1/80	7
3. Associated Karanpura CCL	u/g	Reconst.	Superior	0·62	73·50	2/79	5
14. Churi CCL	u/g	Reconstn.	"	0·84	125·60	3/79	5
15. Ray-Bachra CCL	"	"	"	0·60	78·90	3/80	6
16. Laiyo CCL	u/g	"	Coking	0·50	88·80	1/80	8
~~17. Satpura II~~	~~"~~	~~New~~	~~Power~~				
17. Chhinda WCL	u/g	New	Superior	0·50	135·0	11/79	6
8. Dhanpuri WCL	"	"	"	0·30	57·80	2/80	5
9. Pali WCL	"	"	Power	0·50	≃ 70·80	7/80	5
0. Govinda WCL	"	Recons.	Superior	0·60	100·60	2/80	6

1	2	3	4	5	6
21. Hindusthan Kalpeth wll	U/g Reinst. Support	0·60	81·53	3/77	6
2. Orient wll	" " "	0·95	127·90	4/78	7
3. Dungapur Rayatwari wll	" " "	0·60	64·80	8/78	6
4. Silewara Expansion wll	" " Power	0·48	112·70	12/78	4
Inder P wll	" " Support	0·40	63·06	12/78	6
26. Satpura wll	" " Power	0·60	77·58	4/79	2
27. Patharkhera F.S. wll	" " "	0·30	68·20	4/79	3
28. Ballarpur wll	O/C " Power	1·25	204·60	2/80	2
Sasti wll	O/C new Power	1·00	212·70	4/80	8

1-9-7

1	2	3	4	5	6	7	8	9
Western Coal Fields					–			
1 Pipla U/g-new Power Coal	8/73	·45	51·72	80-81	·40	·19	3	Development Technical
2 Naka Char U/g new Superior	8/73	1·10	134·40	79-80	On Schedule			
3. Kutkona U/g new Superior	11/73	0·80	66·46	79-80		0·20	3-4	Development Siding
4. Jamuna U/g new Superior	11/73	1·00	75·56	79-80		·35	3-4	Development
5 Bhatgam U/g new Superior	12/74	1·00	104·10	82-83	·16	·16	2	Siding 2 CHP
6 Nandan U/g new Coking	12/74	0·60	75·40	~~81~~-82	·50	·25	2	Development Siding Washery
7 Sobhapur ~~1/75~~ U/g new Power Coal	1/75	·95	106·0	82-83	·50	·25	2	Development Technical
8 Rajgamar U/g new Superior	10/75	0·72	95·80	82-83	·24	·12	1	Development Siding ~~Stn~~
9 Sarni U/g new Power Coal	6/79	0·42	64·50	84-85	– Maybe on Schedule			
10 Bijuri U/g new Superior	12/73	0·48	45·50	82-83	·22	·22	~~2~~ 1	Development
11 Amlai U/g Recons. Superior 8/75		0·50	42·20	79-80	On Schedule			
12 Katma U/g R cons. Superior 1/74		0·60	42·50	80-81	On Schedule			

(Back Page)

1	2	3	4	5	6	7	8	9
Churcha Expn. U/G Recons. Superior	2/78	1.00	79.30	82-83	.70	.70	—	
4. Ramnagar U/G Reconst. Superior	9/78	0.72	76.50	85-86	.55	.55	—	
5. Rajnagar U/G Reconst. Superior	10/78	1.00	105.80	85-86	.35	.37	—	
N.C.P.H U/G Reconst. Superior	10/78	1.15	83.60	85-86	.50	.48	—	
Kamptee U/G Reconst. Power Coal	12/79	0.72	64.60	85-86	.41	.42	—	
Umrer Expn. O/C Reconst. Power Coal	12/79	1.84	209.90	82-83	on .27	1.20	—	
1. Kusmunda Expn. O/C new Power Coal.	10/78	6.00	783.80	84-85	.79	.40	(On schedule) (rail & linked by [illegible] construction)	
0 Chirimiri O/C new Superior	9/78	1.00	187.0	83-84	.20	.20	(on schedule)	
1. New Majri O/C new Power Coal	10/78	1.00	150.90	83-84	.30	.30	—	
2. Manikpur Exp. O/C Reconst. Power Coal	12/78	1.94	137.70	83-84	1.99	2.00	(ahead of Schedule)	
3. Durgapur O/C new Power Coal	10/78	1.80	346.50	84-85	—	on Schedule		
1. Dhanpuri O/C new Power Coal	6/79	1.25	250.30	85-86	.12	.10	—	
5. Gevra O/C new-Power	10/79	6.00	494.50	87-88	on Schedule			
. Rajnagar O/C new-Power	6/79	1.00	130.90	85-86	on Schedule			
H.C.I - O/C new Power	10/79	0.50	119.0	86-87	on Schedule.			

BHARAT COKING COAL LIMITED.

NAME & TYPE OF PROJECT	DATE OF SANCTION	CAPTION CAPACITY MTY	SANCTIONED CAPITAL Rs. Mlns	ACCORDING TO ORIGINAL COMPLETION DATE	FR PRODN. 79-80	ACTUAL PRODN 79-80	EXPECTED SLIPPAGE Yrs.	BROAD REASONS FOR SLIPPAGE
1	2	3	4	5	6	7	8	9
[illegible]lmudihi Coking Coal New U/g mine	3/74.	2·10	376·20	81-82	0·63	·45	4 (?)	Managem Technolog Personnel
	Being Revised now.							
Moonidih Revision Coking Coal U/g new	11/65 73-74	2·10	≃ 600	81-82 (?)	0·76	0·43	4	Managem Technology Equipment etc.
	Being Revised for the third time.							
North-Amlabad U/g-Reconst. Coking Coal. extremely gassy	10/80	0·72	272·80	86-87	—	—		May be on Schedule (?) Dagaskei
Katras U/g Reconst. Coking	10/79	0·90	260·40	84-85	—			May be on Schedule
Bhalgora U/g Reconst. Coking Coal	10/80	1·20	462·15	85-86	May be delayed by about 2 years to 3 yrs.			
Kharkaree-Dharmaband. U/g Reconst. Coking	10/80	0·36	93·25	85-86	May be on Schedule.			
Bhowrah-Sudamdih U/g Reconst. Coking	2/80	0·22	49·70	83-84	May be on Schedule.			
Golukdih O/C New. Coking + NC	1/77	0·72	76·80	80-81	·20	·18		On Schedule.
Kusunda O/C. Coking + Noncoking	10/78	1·00	118·50	81-82	·60	0·45		On Schedule

AME OF PROJECT & Category	Caption Capacity MTY	Sanction Capital Rs. Millions Date	According F.R Prodn. 79-80

Yearwise Projected Cost of Production and Sales Realisation during the period 1980 - 85

(Rs. per tonne)

	(Actual) 1979-80	80-81	81-82	82-83	83-84	84-85
1. Total Production (m.t)	91.42 ~~84.71~~	99.00	108.72	120.78	138.32	157.31
2. Revenue Prod. (m.t)	88.21	97.83	105.61	112.45	119.91	141.43
3. Production of Saleable Coal (m.t)	84.71	94.94	102.69	109.50	116.99	138.49
4. EMS i) Based on Present Wage Structure	43.47	46.95	50.70	54.78	59.14	63.87
ii) Impact of likely Wage Revision w.e.f.1.1.83	-	-	-	3.42	13.69	14.78
5. OMS (tonnes)	0.66	0.70	0.78	0.85	0.94	1.03
6. Salaries & Wages {3% inc / 5% prices}	65.87	67.07	65.00	68.47	77.48	76.36
7. Stores	10.91	12.88	15.12	16.20	17.32	20.57
8. Power	4.68	5.41	5.68	5.96	6.26	6.57
9. Transportation of Coal & Sand	2.68	2.50	2.50	2.25	2.25	2.00
10. Overhead	4.68	estimate 4.67	4.59	4.46	4.21	3.99
11. Interest on working capital	actual 1.22	4.63 *	4.64	4.87	5.38	5.47
12. Interest on long term loans	5.67	5.83	6.71	7.36	7.67	8.99
13. Depreciation	7.40	7.85	9.18	10.15	10.66	12.41
14. Others	3.54	3.54	3.54	3.54	3.54	3.54
15. Less Subsidy	1.40	1.40	1.40	1.40	1.40	1.40
Total	105.34	112.98	115.56	121.87	133.37	138.50
Escalation Based on actual past 5 yrs {5.82% for 7-14}	-	2.67	5.89	9.32	13.01	18.08
Grand Total	105.34	115.65	121.45	131.19	146.38	156.58
Increase over previous year (%)	18.62	9.79	5.02	8.02	11.58	6.97
Sales Realisation (Rs./t)	91.42	100.88	98.88	96.88	94.00	92.93

* based on 4 mths pdn.

3.7

Projected Losses by CIL at the existing coal prices

Year	Cost/tonne (Rs.)	Total cost (Rs.Crores)	Net sales realisation per tonne (Rs.)	Total sales realisation on saleable coal (Rs.Crores)	Total Losses (Rs.Crores)
1980-81	115.65	1097.98	100.88	957.75	140.23
-82	121.45	1247.17	98.88	1015.40	231.77
-83	131.19	1436.53	96.88	1060.84	375.69
-84	146.38	1712.50	94.00	1099.71	612.79
-85	156.58	2168.48	92.93	1286.99	881.49
Total					2241.97

Production

Total : As incorporated in the Coal and Lignite group report.

Revenue Production

It has been assumed that during 1st four years i.e. 1980-84, only 4 groups of mines will be under revenue production i.e. Existing Mines, Spillover Projects, approved reconstruction and approved new mines. From the year 1984-85 Vth group viz. projects formulated and yet to be approved has been considered to be reaching revenue stage and as such production obtained from this group in 1984-85 has been considered to be revenue production.

Saleable Coal: Revenue production thus obtained has been reduced to the extent of colliery consumption, figures for which are as per coal and lignite group report.

Capital Investment :

Existing investment upto 1979-80

A nos. of variants have been tried, but it is considered that calculations arrived at from the per tonne depreciation give the most suitable results. Accordingly the existing capital to be depreciated at the end of 1979-80 came to Rs.896 crores.

To this figure of investment after writting off 7% of the balance each year following investment figures have been added.

Rs. crores

	1980-81	1981-82	1982-83	1983-84	1984-85
1. ~~According~~ opening investment	832.82	774.52	720.31	669.88	622.99
2. 4 groups cumulative investment (spillover, existing reconstruction & new)	231.92	572.78	868.40	1112.10	1315.54
3. Projects formulated but yet to be approved	-	-	-	-	516.61 (cumulative)
4. Project yet to be formulated	-	-	-	-	-
	1064.74	1347.30	1588.71	1781.98	2455.14

These investment figures have been depreciated @ 7% and total depreciation thus arrived at has been distributed over saleable coal. ~~shown above~~.

Long term loan

As in case of investment upto 1979-80, long term loan has also been worked out from the interest impact on per tonne of saleable coal. This amount comes to Rs.457 crores. Considering 10% repayment every year and new investment to the extent of 50% from loan capital, total long term loan will be:

Rs. crores

	1980-81	1981-82	1982-83	1983-84	1984-85
1. Existing	411.30	370.17	333.15	298.84	269.96
2. 4 Groups	115.96	286.39	434.20	556.05	657.77
3. Project Formulated yet to be approved	-	-	-	-	258.31
Total	527.26	656.56	767	854.89	1186.04

Variant-I

YEARWISE PROJECTED COST OF PRODUCTION DURING THE PERIOD 1980 - 85

(Rs. per tonne)

	1979-80 (Actual)	1980-81	1981-82	1982-83	1983-84	1984-85
1. Production (m.t)	91.42	99.00	108.72	120.78	138.32	157.31
2. EMS (Rs.)						
(i) Based on present wage structure	43.47	46.95	50.70	54.78	59.14	63.87
(ii) Impact of likely wage revision w.e.f. 1.1.1983	-	-	-	3.42	13.69	14.78
3. O.M.S (tonnes)	0.66	0.70	0.78	0.85	0.94	1.03
4. Salaries and Wages	65.87	67.07	65.00	68.45	77.48	76.36
5. Stores	10.91	12.77	14.47	14.69	14.83	14.59
6. Power	4.68	5.41	5.68	5.96	6.26	6.57
7. Transportation of Coal and Sand	2.68	2.50	2.50	2.25	2.25	2.00
8. Overheads	4.68	4.67	4.59	4.46	4.21	3.99
9. Interest on working capital	1.22	4.64	4.62	4.80	5.24	5.17
10. Interest on long term loan	5.67	5.59	6.34	6.67	6.49	6.19
11. Depreciation	7.40	8.01	9.23	9.80	9.59	9.18
12. Others	3.54	3.54	3.54	3.54	3.54	3.54
13. Less Subsidy	1.40	1.40	1.40	1.40	1.40	1.40
Total	105.34	112.80	114.57	119.22	128.49	126.19
Escalation		2.66	5.77	8.86	11.88	14.50
Grand Total	105.34	115.46	120.34	128.08	140.37	140.69
Increase in cost over previous year (0/0)	18.62	9.61	4.23	6.43	9.60	0.23
Sales realisation	91.42	100.88	98.88	96.88	94.00	92.93

Variant - II

YEARWISE PROJECTED COST OF PRODUCTION DURING THE PERIOD 1980-85

(Rs. per tenne)

	1979-80 (Actual)	1980-81	1981-82	1982-83	1983-84	1984-85
1. Preduction (m.t.)	91.42	99.00	108.72	120.78	138.32	157.31
2. EMS (Rs.)						
(i) Based en present wage structure	43.47	46.95	50.70	54.78	59.14	63.87
(ii) Impact ef likely wage revisien w.e.f. 1.1.1983	-	-	-	3.42	13.69	14.78
3. O.M.S (tonnes)	0.66	0.70	0.78	0.85	0.94	1.03
4. Salaries and Wages	65.87	67.07	65.00	68.45	77.48	76.36
5. Stores	10.91	13.27	14.83	15.17	15.07	14.83
6. Power	4.68	5.41	5.68	5.96	6.26	6.57
7. Transportatien ef Coal and Sand	2.68	2.50	2.50	2.25	2.25	2.00
8. Overheads	4.68	4.67	4.59	4.46	4.21	3.99
9. Interest en werking capital	1.22	4.64	4.62	4.80	5.24	5.19
10. Interest en leng term lean	5.67	4.90	5.78	6.22	6.13	5.91
11. Depreciatien	7.40	8.38	9.38	9.85	9.54	9.03
12. Others	3.54	3.54	3.54	3.54	3.54	3.54
13. Less Subsidy	1.40	1.40	1.40	1.40	1.40	1.40
Total	105.34	112.98	114.52	119.30	128.32	126.02
Escalatien		2.67	5.76	8.88	11.84	14.45
Grand Tetal	105.34	115.65	120.28	128.18	140.16	140.47
Increase in cest ever previeus year (0/0)	18.62	9.79	4.00	6.57	9.35	0.22
Sales realisatien	91.42	100.88	98.88	96.88	94.00	92.93

Variant-III

YEARWISE PROJECTED COST OF PRODUCTION DURING THE PERIOD 1980 - 85

(Rs. per tonne)

	1979-80 (Actual)	1980-81	1981-82	1982-83	1983-84	1984-85
1. Production (m.t.)	91.42	99.00	108.72	120.78	138.32	157.31
2. EMS (Rs.)						
(i) Based on present wage structure	43.47	46.95	50.70	54.78	59.14	63.87
(ii) Impact of likely wage revision w.e.f. 1.1.1983	-	-	-	3.42	13.69	14.78
3. O.M.S (tonnes)	0.66	0.70	0.78	0.85	0.94	1.03
4. Salaries and Wages	65.87	67.07	65.00	68.45	77.48	76.36
5. Stores	10.91	13.27	14.83	15.17	15.07	14.83
6. Power	4.68	5.41	5.68	5.96	6.26	6.57
7. Transportation of Coal and Sand	2.68	2.50	2.50	2.25	2.25	2.00
8. Overheads	4.68	4.67	4.59	4.46	4.21	3.99
9. Interest on working capital	1.22	4.64	4.62	4.80	5.24	5.19
10. Interest on long term loan	5.67	4.90	5.78	6.22	6.13	5.91
11. Depreciation	7.40	8.91	9.98	10.48	10.15	9.61
12. Others	3.54	3.54	3.54	3.54	3.54	3.54
13. Less Subsidy	1.40	1.40	1.40	1.40	1.40	1.40
Total	105.34	113.51	115.12	119.93	128.92	126.60
Escalation		2.70	5.83	8.99	11.98	14.61
Grand Total	105.34	116.21	120.95	128.92	140.90	141.21
Increase in cost (0/0) over previous year	18.62	10.32	4.08	6.59	9.29	0.22
Sales realisation	91.42	100.88	98.88	96.88	94.00	92.93

Shilesh

MSG NO.11 DTD 28/10

FROM COPSP CIL CAL

FOR CMD WCL NAGPUR

RPT EXECUTIVE DIR BILASPUR

WORLD BANK TEAM (4 PERSONS) REACHING CHAMPA BY BOMBAY MAIL ON 4TH NOVR RPT 4TH NOVR(.) THEY WILL VISIT KUSMUNDA PROJECT ON 4TH NOV(.) KINDLY ARRANGE TRANSPORT AT THE STATION(.) THE TEAM WILL HAVE NIGHT STAY AT BILASPUR(.) ACCOMODATION TO BE ARRANGED(.) TEAM WILL LEAVE FOR NAGPUR ON 5TH NOV BY BOMBAY MAIL(.) TRANSPORT AND ACCOMODATION ARE TO BE ARRANGED AT NAGPUR ON 5TH(.) THEY WILL HAVE DISCUSSION WITH CMD WCL ON 6TH(.) THEY WILL LEAVE NAGPUR ON 7TH FOR CALCUTTA FOR FINAL ROUND DISCUSSION WITH CHAIRMAN COAL INDIA(.)

OVER

- -

Western Coalfields Ltd
Bisesar House,
Nagpur.

Ref No.WCL/CMD/,/Secy/2745/M

29th Oct 1980.

Secy

Copy to : Exe.Director, Bilaspur.
Copy to : Exe.Director, Nagpur.
Copy to : Dir (P) WCL Nagpur.
Copy to : Dir (F) WCL Nagpur.
Copy to : CME (HQ) WCL Nagpur.

For information & necessary action.

CMD(WCL)

3.7

Item No.17

SUB: STATEMENT SHOWING PROJECTION OF COST UPTO 1984-85 CONSIDERING NO CHANGE IN CONSUMER PRICE INDEX AND NO PRICE ESCALATION

Particulars	1979-80	1980-81	1981-82	1982-83	1983-84	1984-85
Production from Revenue Mines(MT)	24.097	25.973	28.060	32.37	34.65	40.35
Net Saleable coal "	23.444 1.84	25.283 2.12	27.360 4.26	31.62 2.23	33.85 +5.65	39.50
Overall manshifts	266.42	282.07	292.05	317.11	328.00	360.59
Overall OMS	0.90	0.92	0.96	1.02	1.06	1.12
UG OMS	0.75	0.765	0.77	0.77	0.775	0.775
O/C OMS	2.96	3.00	2.81	2.67	3.00	3.00
E.M.S	50.11	51.11	52.13	53.20	54.33	55.51
Consumer Price Index	345	345	345	345	345	345

3.7

	1979-80		1980-81		1981-82		1982-83		1983-84		1984-85	
	Amt. in Rs.lakhs	P.T.	Amt. in Rs.lakhs	P.T.	Amt. in Rs.lakhs	P.T.	Amt. in Rs.lakhs	P.T.	A mt.in Rs.lakhs	P.T.	Amt.in Rs.lakhs	P.T.
Salaries & Wages	13,350	56.94	14,417	57.02	15,225	55.64	16,870	53.35	17,822	52.65	20,017	50.67
Stores	2,549	10.87	2,692	10.65	2,848	10.41	3,158	9.99	3,314	9.79	3,867	9.79
Contractor's Payment	392	1.67	415	1.64	441	1.61	475	1.50	500	1.48	525	1.33
Power	1,068	4.56	1,094	4.33	1,121	4.10	1,180	3.73	1,210	3.58	1,390	3.52
Depreciation	1,521	6.49	1,901	7.52	2,705	9.89	3,500	11,07	4,300	12,70	5,100	12,91
Interest	1,332	5.68	1,625	6.43	2,204	8.06	2,800	8.86	3,400	10.04	4,000	10.13
Misc. Expenses	1,094	4.67	1,116	4.41	1,139	4.16	1,150	3.64	1,150	3.40	1,150	2.91
Gross Total Cost	21,306	90.88	23,260	92.00	25,683	93.87	29,133	92.14	31,696	93.64	36,049	91.26
Less: Subsidy	(-) 107	(-)0.46	(-) 100	(-)0.40	(-)100	(-)0.36	(-)100	(-)0.32	(-)100	(-)0.30	(-)100	(-)0.25
Net Total cost	21,199	90.42	23,160	91.60	25,583	93.51	29,033	91.82	31,596	93.34	35,949	91.01
Sale Value	19,495	83.15	23,214	91.82	25,171	92.00	29,090	92.00	31,142	92.00	36,340	92.00
Less:Quality Allowance/ Bad Debts.	545	2.32	630	2.49	750	2.74	869	2.75	931	2.75	1,086	2,75
Net Sale Value	18,950	80.83	22,584	89.33	24,421	89.26	28,221	89.25	30,211	89.25	35,254	89.25
PROFIT/LOSS	(-)2249	(-)9.59	(-) 576	(-)2.27	(-)11-62	(-)4.25	(-)812	(-)2.57	(-)1,385	(-)4.09	(-) 695	(-)1.75

/Sam.

ASSUMPTIONS

1. Production and OMS

Production and OMS have been taken as incorporated in the report for working group on coal and lignite.

2. Salary and Wages

EMS has been calculated from the actual salary and wages per tonne and OMS during 1979-80.

Actual wages per tonne	=	Rs.65.87
O.M.S	=	0.66
E.M.S	=	Rs.65.87 x 0.66
	=	Rs.43.47

Thereafter EMS has been increased by 8% every year to take care of increment and variable D.A.

Further in respect of likely wage revision w.e.f. 1.1.1983 growth of EMS has been assumed to be 25%.

3. Stores

3.1 Explosive : Quantity of explosives as included in the report of coal and lignite group has been considered for working out the explosive requirement per tonne of coal production. Price is based on March 1979 - quotation which has been escalated by 20%.

3.2 Timber : Based on actual cost figures of the past, cost of timber has been taken as Rs.1=00 per tonne of U.G production and this cost has been distributed over total production (both U.G and O.C).

3.3 Stores and Spares : This has been taken as 5% of total capital investment in plant and machinery. The details of calculations are as follows :

Variant-I

1. Total investment as on 31.3.80 based on depreciation cost of 1979-80 i.e. Rs.7.40 x 84.71 x $\frac{100}{7}$ = Rs. 896 crores

2. Investment on P&M @ 70% of 896 crores = Rs. 627 crores

Variant II - Investment figures are based on the agenda of items-1 of Directors(F)'s meeting held on 8.7.80. Accordingly the investment considered is

Equity	=	Rs. 806.97 crores
Plan loan	=	Rs. 480.74 "
		Rs.1287.71 crores

Investment in mines @ 80% = Rs.1029 crores

Investment in Plant and Machinery @ 70% of Rs.1030 crores = Rs.720 crores

Further investment in year 1980-81 onwards, 80% of the investment in the 4 category of mines has been taken for investment in P&M due to increasing level of mechanisation in future.

4. Power

In 1979-80 power consumption was on average 15.58 kwh per tonne of production and this is expected to rise to 18 kwh/tonne during 1980-81 (coal and lignite group draft report). The tariff is taken as applicable to 1979-80, and for later years i.e. 1981-82 cost of power has been increased by 5% to account for increasing quantity of power required.

5. Transportation of Coal and Sand

L.S figures have been provided but reduction is expected because of reducing share of U.G production resulting in reduced requirement of sand for stowing.

6. Overheads

Total overheads for the year 1979-80 have been worked out from per tonne impact of Rs.4.68 and total production of coal. Every year the total cost on overheads has been assumed to be increasing @ 8%, per tonne impact on basis of total production has been worked out.

7. Interest on working capital

Procedure as followed in project reports i.e. 4 months cash revenue expenditure has been followed here also.

8. Interest on loan capital

Variant I

Base loan capital has been worked out from the per tonne impact of interest on the saleable production during 1979-80. During 1979-80, interest on loan capital per tonne of production = Rs.5.67.

Total net saleable coal production = 84.71 million tonnes.

Interest rate being 10.5%, total loan capital at the dnd of 1979-80 works out to $= \text{Rs.}5.67 \times 84.71 \times \frac{100}{10.5}$

Variant-II

As indicated under variant-II for stores and spares

Plan loan = Rs. 480.74 crores

Loan investment

in mines @ 80% = Rs. 384.59 crores

For future this loan capital is assumed to be repaid at the rate 10% of loan at the end each year; repayment starting with the year 1980-81.

Future investment in 4 groups of mines viz. existing mines, spill over from IVth plan and approved reconstruction and new mines has been taken and 50% of this is considered to be available as long term loan.

9. Depreciation

Variant-I

The value of assets at the end of 1979-80, has been estimated from the net saleable coal production of 1979-80 and depreciation per tonne. Considering the average depreciation to be 7% of investment, this investment figure has been worked out.

1979-80 figures $= \text{Rs.}7.40 \times 84.71 \times \frac{100}{7}$

$= \text{Rs.}8955.06$ million.

Variant-II As explained under stores and spares total investment at the end of 1979-80 = Rs.1029 crores. The total depreciation as worked out for this variant has been distributed over total coal production during each of these five years.

Variant-III Total depreciation as arrived at under variant-II has been distributed over saleable coal (94% of teal production).

A part of this investment (7%) has been considered to be completely written off each year during the period 1980-85.

Further investment in these 4 groups as considered for loan capital has been added and depreciation @ 7% has been worked out, ~~distributing over the~~ total coal or ~~saleable production (94%~~ of total production).

10. Other expenditure and subsidy

These item have been taken at the same rate as actual for 1979-80. All cost figures for 1979-80 are as included in the CIL monthly report for March,1980.

11. Escalation

So far salary and wages is concerned assumptions have been shown under 2 above. Based on past 5 years data i.e. 1975-76 to 1979-80, it has been observed that simple weighted rate of escalation has been 5.82%. Accordingly same rate has been made applicable during next five years commencing 1980-81 for items other than salary & wages.

12. Sales realisation :

Considering the actual mix of the production during each of the five year 1980-85, sale price, loss on ~~as of or~~ soft coke and net sale realisation during April-June 1980, the following sale realisation per tonne is expected.

1980-81	=	Rs.100.88	1983-84	=	Rs.94.00
1981-82	=	Rs. 98.88	1984-85	=	Rs.92.93
1982-83	=	Rs. 96.88			

13. Profit/Loss

Accordingly the projected cost and sales realisation during next 5 years in respect of variant-I are likely to be as follows :

Year	Cost/tonne (Rs.)	Total cost (Rs.crores)	Net sale realisation/tonne (Rs.)	Total sale realisation (Rs.crores)	Total loss (Rs.crores)
1980-81	115.46	1143.05	100.88	998.71	144.34
1981-82	120.34	1308.34	98.88	1075.02	233.32
1982-83	128.08	1546.95	96.88	1170.12	376.83
1983-84	140.37	1941.60	94.00	1300.21	641.39
1984-85	140.67	2212.88	92.93	1461.88	751.00
		8152.82		6005.94	2146.88

In case actual production achieved is below these figures, losses are likely to increase due to impact of higher fixed costs.

Variant-III

YEARWISE PROJECTED COST OF PRODUCTION DURING THE PERIOD 1980 - 85

(Rs. per tenne)

	1979-80 (Actual)	1980-81	1981-82	1982-83	1983-84	1984-85
1. Preduction (m.t.)	91.42	99.00	108.72	120.78	138.32	157.31
2. EMS (Rs.)						
(i) Based on present wage structure	43.47	46.95	50.70	54.78	59.14	63.87
(ii) Impact of likely wage revision w.e.f. 1.1.1983	-	-	-	3.42	13.69	14.78
3. O.M.S (tonnes)	0.66	0.70	0.78	0.85	0.94	1.03
4. Salaries and Wages	65.87	67.07	66.00	68.45	77.48	[illegible]
5. Stores	10.91	13.27	14.83	15.17	15.07	14.83
6. Power	4.63	5.41	5.68	5.96	6.26	6.57
7. Transportation of Coal and Sand	2.68	2.50	2.50	2.25	2.25	2.00
8. Overheads	4.68	4.67	4.59	4.46	4.21	3.99
9. Interest on working capital	1.22	4.64	4.62	4.80	5.24	5.19
10. Interest on long term loan	5.67	4.90	5.78	6.22	6.13	5.91
11. Depreciation	7.40	8.91	9.98	10.48	10.15	9.61
12. Others	3.54	3.54	3.54	3.54	3.54	3.54
13. Less Subsidy	1.40	1.40	1.40	1.40	1.40	1.40
Total	105.34	113.51	115.12	119.93	128.92	126.60
Escalation		2.70	5.83	8.99	11.98	14.61
Grand Total	105.34	116.21	120.95	128.92	140.90	141.21
Increase in cost (0/0) over previous year	18.62	10.32	4.08	6.59	9.29	0.22
Sales realisation	91.42	100.88	98.88	96.88	94.00	92.93

Variant - II

YEARWISE PROJECTED COST OF PRODUCTION DURING THE PERIOD 1980-85

(Rs. per tonne)

	1979-80 (Actual)	1980-81	1981-82	1982-83	1983-84	1984-85
1. Production (m.t.)	91.42	99.00	108.72	120.78	138.32	157.31
2. EMS (Rs.)						
(i) Based on present wage structure	43.47	46.95	50.70	54.78	59.14	63.87
(ii) Impact of likely wage revision w.e.f. 1.1.1983	-	-	-	3.42	13.69	14.78
3. O.M.S (tonnes)	0.65	0.70	0.78	0.85	0.94	1.03
4. Salaries and Wages	65.87	67.07	65.00	68.45	77.48	76.35
5. Stores	10.91	13.27	14.83	15.17	15.07	14.83
6. Power	4.68	5.41	5.68	5.96	6.26	6.57
7. Transportation of Coal and Sand	2.63	2.50	2.50	2.25	2.25	2.00
8. Overheads	4.68	4.67	4.59	4.46	4.21	3.99
9. Interest on working capital	1.22	4.64	4.62	4.80	5.24	5.19
10. Interest on long term loan	5.67	6.90	5.78	6.22	6.13	5.91
11. Depreciation	7.40	8.38	9.38	9.85	9.54	9.03
12. Others	3.54	3.54	3.54	3.54	3.54	3.54
13. Less Subsidy	1.40	1.40	1.40	1.40	1.40	1.40
Total	105.34	112.98	114.52	119.30	128.32	126.02
Escalation		2.67	5.76	8.88	11.84	14.45
Grand Total	105.34	115.65	120.28	128.18	140.16	140.47
Increase in cost over previous year (0/0)	18.62	9.79	4.00	6.57	9.35	0.22
Sales realisation	91.42	100.88	98.88	96.88	94.00	92.93

Variant-I

YEARWISE PROJECTED COST OF PRODUCTION DURING THE PERIOD 1980 - 85

(Rs. per tonne)

	1979-80 (Actual)	1980-81	1981-82	1982-83	1983-84	1984-85
1. Production (m.t)	91.42	99.00	108.72	120.78	138.32	157.31
2. EMS (Rs.)						
(i) Based on present wage structure	43.47	46.95	50.70	54.78	59.14	63.87
(ii) Impact of likely wage revision w.e.f. 1.1.1983	-	-	-	3.42	13.69	14.78
3. O.M.S (tonnes)	0.65	0.70	0.78	0.85	0.94	1.03
4. Salaries and Wages	65.87	67.07	65.00	68.45	77.48	76.33
5. Stores	10.91	12.77	14.47	14.69	14.83	14.59
6. Power	4.68	5.41	5.68	5.96	6.26	6.57
7. Transportation of Coal and Sand	2.68	2.50	2.50	2.25	2.25	2.00
8. Overheads	4.68	4.67	4.59	4.46	4.21	3.99
9. Interest on working capital	1.22	4.64	4.62	4.80	5.24	5.17
10. Interest on long term loan	5.67	5.59	6.34	6.67	6.49	6.19
11. Depreciation	7.40	8.01	9.23	9.80	9.59	9.18
12. Others	3.54	3.54	3.54	3.54	3.54	3.54
13. Less Subsidy	1.40	1.40	1.40	1.40	1.40	1.40
Total	105.34	112.80	114.67	119.22	128.49	126.19
Escalation		2.66	5.77	8.86	11.88	14.50
Grand Total	105.34	115.46	120.34	128.08	140.37	140.69
Increase in cost over previous year (0/0)	18.62	9.61	4.23	6.43	9.60	0.23
Sales realisation	91.42	100.88	98.88	96.88	94.00	92.93

These investment figures have been depreciated @ 7% and total depreciation thus arrived at has been distributed over saleable coal shown above.

Long term loan

As in case of investment upto 1979-80, long term loan has also been worked out from the interest impact on per tonne of saleable coal. This amount comes to Rs.457 crores. Considering 10% repayment every year and new investment to the extent of 50% from loan capital, total long term loan will be:

Rs. crores

	1980-81	1981-82	1982-83	1983-84	1984-85
1. Existing	411.30	370.17	333.15	298.84	269.96
2. 4 Groups	115.96	286.39	434.20	556.05	657.77
3. Project Formulated yet to be approved	-	-	-	-	258.31
Total	527.26	656.56	767	854.89	1186.04

Production

Total : As incorporated in the Coal and Lignite group report.

Revenue Production

It has been assumed that during 1st four years i.e. 1980-84, only 4 groups of mines will be under revenue production i.e. existing Mines, Spillover Projects, approved reconstruction and approved new mines. From the year 1984-85 Vth group viz. projects formulated and yet to be approved has been considered to be reaching revenue stage and as such production obtained from this group in 1984-85 has been considered to be revenue production.

Saleable Coal: Revenue production thus obtained has been reduced to the extent of colliery consumption, figures for which are as per coal and lignite group report.

Capital Investment :

Existing investment upto 1979-80

A nos. of variants have been tried, but it is considered that calculations arrived at from the per tonne depreciation give the most suitable results. Accordingly the existing capital to be depreciated at the end of 1979-80 came to Rs.896 crores.

To this figure of investment after writting off 7% of the balance each year following investment figures have been added.

Rs. crores

	1980-81	1981-82	1982-83	1983-84	1984-85
1. According opening investment	832.82	774.52	720.31	669.88	622.99
2. 4 groups cumulative investment (spillover, existing reconstruction & new)	231.92	572.78	868.40	1112.10	1315.54
3. Projects formulated but yet to be approved	-	-	-	-	516.61 (cumulative)
4. Project yet to be formulated	-	-	-	-	-
	1064.74	1347.30	158R.71	1781.98	2455.14

Projected Losses by CIL at the existing coal prices

Year	Cost/tonne (Rs.)	Total cost (Rs.Crore)	Net sales realisation per tonne (Rs.)	Total sales realisation on saleable coal (Rs.Crore)	Total Loss (Rs.Crore)
1980-81	115.65	1097.98	100.88	957.75	140.23
-82	121.45	1247.17	98.88	1015.40	231.77
-83	131.19	1436.53	96.88	1060.84	375.69
-84	146.38	1712.50	94.00	1099.71	612.79
-85	155.68	2168.48	92.93	1286.99	881.49
Total					2241.97

Yearwise Projected Cost of Production and Sales Realisation during the period 1980 - 85

(Rs. per tonne)

	(Actual) 1979-80	80-81	81-82	82-83	83-84	84-85
1. Total Production (m.t)	91.4 ~~84.71~~	99.00	108.72	120.78	138.32	157.31
2. Revenue Prod. (m.t)	88.21	97.83	105.61	112.45	119.91	141.43
3. Production of Saleable Coal (m.t) Million Tonnes	94.7	94.94	102.69	109.50	116.99	138.49
4. EMS i) Based on Present Wage Structure	43.47	46.95	50.70	54.78	59.14	63.87
ii) Impact of likely Wage Revision w.e.f.1.1.83	-	-	-	3.42	13.69	14.78
5. OMS (tonnes)	0.66	0.70	0.78	0.85	0.94	1.03
6. Salaries & Wages	65.87	67.07	65.00	68.47	77.48	76.36
7. ~~Stores~~	10.91	12.38	15.12	16.20	17.32	20.57
8. Power	4.68	5.41	5.68	5.96	6.26	6.57
9. Transportation of Coal & Sand	2.68	2.50	2.50	2.25	2.25	2.00
10. Overhead	4.68	4.67	4.59	4.46	4.21	3.99
11. Interest on working capital	1.22	4.63	4.64	4.87	5.38	5.47
12. Interest on long term loans	5.67	5.83	6.71	7.36	7.67	8.99
13. Depreciation	7.40	7.35	9.18	10.16	10.66	12.41
14. Others	3.54	3.54	3.54	3.54	3.54	3.54
15. Less Subsidy	1.40	1.40	1.40	1.40	1.40	1.40
Total	105.34	112.98	115.56	121.87	133.37	138.50
Escalation	-	2.67	5.89	9.32	13.01	18.08
Grand Total	105.34	115.65	121.45	131.19	146.38	156.58
Increase over previous year (%)	18.62	9.79	5.02	8.02	11.58	6.97
Sales Realisation (Rs./t)	91.42	100.38	98.88	96.88	94.00	92.93

Production capacity and capital outlay in the mines of CIL During 1977, 1978 and 1979

Company/ Mine category	Year of submission	Type of mine (O.C./U.G.)	Capacity (m.t.)	Capital (Rs. lakhs)	Capital per tonne capacity (Rs.)	Remarks
(1)	(2)	(3)	(4)	(5)	(6)	(7)
A. E.C.L. I	8 pits	O.C.A	— Nil —			
I. Approved Reconstruction Projects						
1. North Searsole	10/1978 11/1977	U.G.	1.16	1358	117.07	s
2. Seetalpur	10/1977	U.G.	0.40	477	119.75	d
3. Sodepur	3/77	U.G.	0.55	416	75.64	d
4. Kunustoria	10/77	U.G.	0.66	347	52.58	d
5. J.K. Nagar	10/78	U.G.	0.86	1518	176.51	d
6. Parbelia	5/78	U.G.	0.80	1251	156.38	d
7. Satgram	5/78	U.G.	1.20	2640	220.00	d
8. Bankola	6/78	U.G.	1.08	818	75.74	s
			6.71	8827	131.55	
II. Approved New Mines:						
1. Bansra	7/78	O.C.	0.20	275	137.50	S.D
2. Kumarkhala	5/78	O.C.	0.26	400	153.85	S.D
3. Ratibati	3/78	O.C.	0.16	300	187.50	S D
4. Kapasdra (manual Quarries & U.G.)	1/79	O.C.	0.44	405	92.05	S D
5. Khoordia	2/79	O.C.	0.30	250	83.33	S D
6. Chhora	2/79	O.C.	0.20	321	160.50	SD
7. Purushottampur	3/78	O.C.	0.46	673	146.[illegible]	SD
			2.02	2624	129.90	
III. Projects formulated but yet to be approved:						
1. Kalidaspur	2/78	N/U.G.	0.18	317	176.11	S D
2. Rajmahal	12/79	N/O.C.	5.00	8746	174.92	d. S.D

D – More than 150 m deep
S – Less than 150 m. deep
S.D – Shovel dumper
d.S.D dragline, shovel dumper.

(1)	(2)	(3)	(4)	(5)	(6)	(7)
B. BCCL:						
I. Approved Reconstruction Projects.						
1. Katras	12/78	U.G.	0.90	2604	289.33	S
2. Bastakola	1/78	O.C.	0.075	81.5	106.67	S.D
3. Bhagwira South	1/78	O.C.	0.07	90.5	129.29	S.D
4. Katras Choitodih	1/78	O.C.	0.05	28.5	57.00	S.D
			0.195	200.50	102.82	
II. Approved New Mines:						
1. ~~Goluckdih~~ Tisra	3/79	O.C	0.30	198	66.00	S.D
2. South Jharia	1/79	O.C	0.24	235	97.92	S.D
3. Lodna	1/78	O.C.	0.07	81.5	116.43	S.D
			0.61	514.50	84.34	
4. Madhuband-Phularitand (Phase-I)	12/78	U.G	0.15	306	204.00	S
5. Bhagaband New incline incline XVIII top	2/79	U.G	0.18	199	110.56	S
			0.33	505	153.03	
III Projects Formulated but yet to be Approved.						
1. Keshalpur	1/79	O.C.	0.51	696	136.47	S.D
2. Nichitpur-Tetulmari	3/79	O.C.	0.45	749	166.44	"
3. Kesalgarh	12/78	O.C.	0.15	656	437.33	"
4. Nadkhurkhee	12/78	O.C.	0.25	756	302.40	"
			1.36	2857	210.07	
5. Pootkee South/Loyabad	11/77	U.G.	1.80	4450	247.22	Deep
6. Bulliary-Bagaband	8/78	U.G.	1.80	4274	237.44	Deep
7. North Amlabad	1/79	U.G.	0.72	2247	312.08	D
8. Bhagaband Incline 17B	2/79	U.G.	0.18	499	277.22	S
~~9. Nadkhurkhee~~	~~12/78~~	~~U.G.~~				
9. Sitnala	5/79	U.G.	0.15	428	285.33	S
10. Bhalgora	2/79	U.G.	1.20	3446	287.17	D
~~11. Ghaslitand~~	~~1/80~~					
			5.85	15344	262.29	

(1)	(2)	(3)	(4)	(5)	(6)	(7)
CCL:						
I Approved Reconstruction Projects:						
1. Sarubera patch	78	O.C.	0.10	168	168.00	S.D
2. Topa patch Deposit	78	O.C.	0.10	168	168.00	S.D
3. Kedla	78	O.C.	1.00	172.0	172.00	S.D
4. K D Hesland	5/78	O.C.	1.50	1444	96.27	S.D
5. Dhori patch Deposit	78	O.C.	0.10	175	175.00	
6. Sel. Dhori	3/79	O.C.	2.25	2438	108.36	S.D
			5.05	6113	121.05	
7. Khas karanpura	4/77	U.G.	0.36	257	71.39	S
8. Sirka	1/78	U.G.	0.30	156	52.00	D
9. Handidhua	78	U.G.	0.10	137	137.00	S
10. Piprodihi	1/79	U.G.	0.30	329	109.67	S
			1.06	879	82.38	
II Approved New Mines:		Nil				
~~1. R~~						
III Projects formulated but yet to be approved.						
1. Kakri	3/79	O.C.	2.50	4395	175.80	
2. Ara Reorga.	7/78	R/O.C.	1.00	1285	128.50	
3. Dhori west	1/79	R/O.C.	1.50	3152.3	210.15	S D
			2.50	4437.3	177.49	
4. ~~Asant~~ A.K.	2/79	U.G.	0.62	735 ~~735~~	118.55	S
5. Ray Bachra		U.G.	0.60	789	131.50	S
			1.22	1524	124.92	

D. WCL:

I Approved Reconstruction Projects

1. Umrer Expansion.	11/78	O.C.	1.84	2095	113.86	D.S.D
2. Ramnagar	3/77	U.G.	0.72	765.39	106.30	Sh.
3. Ambai Incline (Marginal)	3/78	U.G.	0.10	84.89	84.89	Sh
~~4. Kamptee~~		~~U.G.~~	~~0.72~~	~~673.95~~		
4. Rajnagar	3/77	U.G.	1.00	1058.41	105.84	Sh
5. Ballarpur 3 & 4 pits	77	U.G	0.40	88.07	22.02	Sh.
Average			~~2.06~~ 2.22	~~4091.76~~ 1996.76	~~57.96~~	89.94

II Approved New Mines:

1. New Majri	1/78	O.C.	1.00	1496.82	149.68	Sh D
2. Durgapur	4/78	O.C.	1.80	3465	192.50	Sh D
3. Ballarpur	8/78	O.C.	0.29	365.49	126.03	Sh D
4. Shivpuri	6/78	O.C.	0.13	107.74	82.87	Sh D
5. Kusmunda Expn.	3/78	O.C.	6.00	1309.0	~~21.81~~ 21.81	D S.D
6. Gevra	3/79	O.C.	6.00	4945.0	82.42	D S D
7. Chachai	78	O.C.	0.16	196.98	123.11	S D
8. Dhanpuri	2/79	O.C.	1.25	2409.52	192.76	S D
9. Rajnagar	1/79	O.C.	1.00	1173.34	117.33	S D
10. Hindustan Lalpeth	3/79	O.C.	0.50	1188.60	237.72	S D
11. Nakoda	10/79	O.C	0.40	480.0	120.00	S D
Average			18.53	17137.49	92.49	
12. Patansongi	77	U.G.	0.12	49.70	41.42	Sh.
13. Shivpuri	6/78	U.G.	0.18	108.0	60.00	Sh deep.
14. Sarni/Satpura II	3/79	U.G.	0.42	647.55	154.18	Sh
Average			0.72	805.25	~~118.84~~	111.84

III Projects formulated but yet to be approved:						
1. Hindustan Lalketh	3/77	R/U.G.	0.60	717.00 ~~716.99~~	119.50	Sh
2. Durgapur Rayatwari	8/78	R/U.G.	0.60	648.12	108.02	Sh
3. Orient	4/78	R/U.G	0.95	1280.0	134.74	Sh
4. Lagkura	3/79	R/U.G.	0.50	242.15	48.43	Sh
5. Indek	12/78	R/U.G.	0.75	962.00	128.26	Sh.
Average:			3.40	3849.27	114.21	

for some of the current underground projects are given below with a suitable explanation for the variations from the above norm.

	Project	Company	Type	Specific Investment	Standard Norm	Explation for Variation.
1.	Kalidaspur	ECL	Shallow	289	338/238	Mix mechanisation viz, L/W with Power Support and Mechanised Bord & Pillar.
2.	Jhanjra	"	Shallow/ Deep	452	392	Designed by the Soviet with 33% spare capacity
3.	Chinakuri 1+2 Revised	"	Deep	371	392	Reorganisation.
4.	Bhalgora	BCCL	Deep	385	392	
5.	Pootkee-Bulliary	"	Deep	458 (Provisional)	392	Difficult mining condition-horizon development underground stowing arrangement.
6.	North Amlabad	"	Deep	379	392	No shaft sinking requires degassification and complete stowing.
7.	Chhinda	WCL	Shallow (B&P)	209 270	238	78-79 Price level and intermediate B&P mechanisation.
8.	Birshingpur Pali	"	Shallow (B&P)	196	238	Without siding. Coal will be supplied to the nearby T.P.S. Avaibility of some infrastructure of the adjacent existing mine.
9.	Tandsi	"	Shallow (B&P)	359	238	Extremely difficult geological conditions and outlying location of the project.

19.2 %

21-23%

and a deep mine is shown in the Appendix I. Cost for the plant and equipment excluding those for hoisting will be almost the same for the two types of mines. Areas of saving on a shallow mine are coal-winding, capital outlay for development, project management (due to a shorter gestation period) and prospecting and boring. There will also be a marginal saving in service building, civil structures and pumps.

Conclusion

The above is an attempt towards a realistic analysis of capital requirement for a fairly large-sized underground project based on the 1979 price level. Though this will presumably escalate in future years in actual quantitative terms, the percentage distribution of different items is expected to remain fairly constant. Assessment is done here taking an idealised case, and there is bound to be variations from project to project depending on the resource availability, the geographical location and geological set-up of the project but variation is not likely to exceed a +10% range.

It may be seen that saving in capital in case of a shallow mine than a deep mine for the same output level is quite substantial, of the order of about Rs 5.3 crores. This is mainly on account of elimination of sinking and vertical transport. Barring those in the Jharia field and the western part of Raniganj field, most of the new coal projects will fall under the shallow mine category. As the coal hoisting through shaft is becoming very capital-intensive, one has to take a very cautious appraisal before suggesting skip hoisting for an underground project with a depth of about 200 m or less.

APPENDIX I

Case — I SHALLOW MINE — Depth Range — 100-300 m

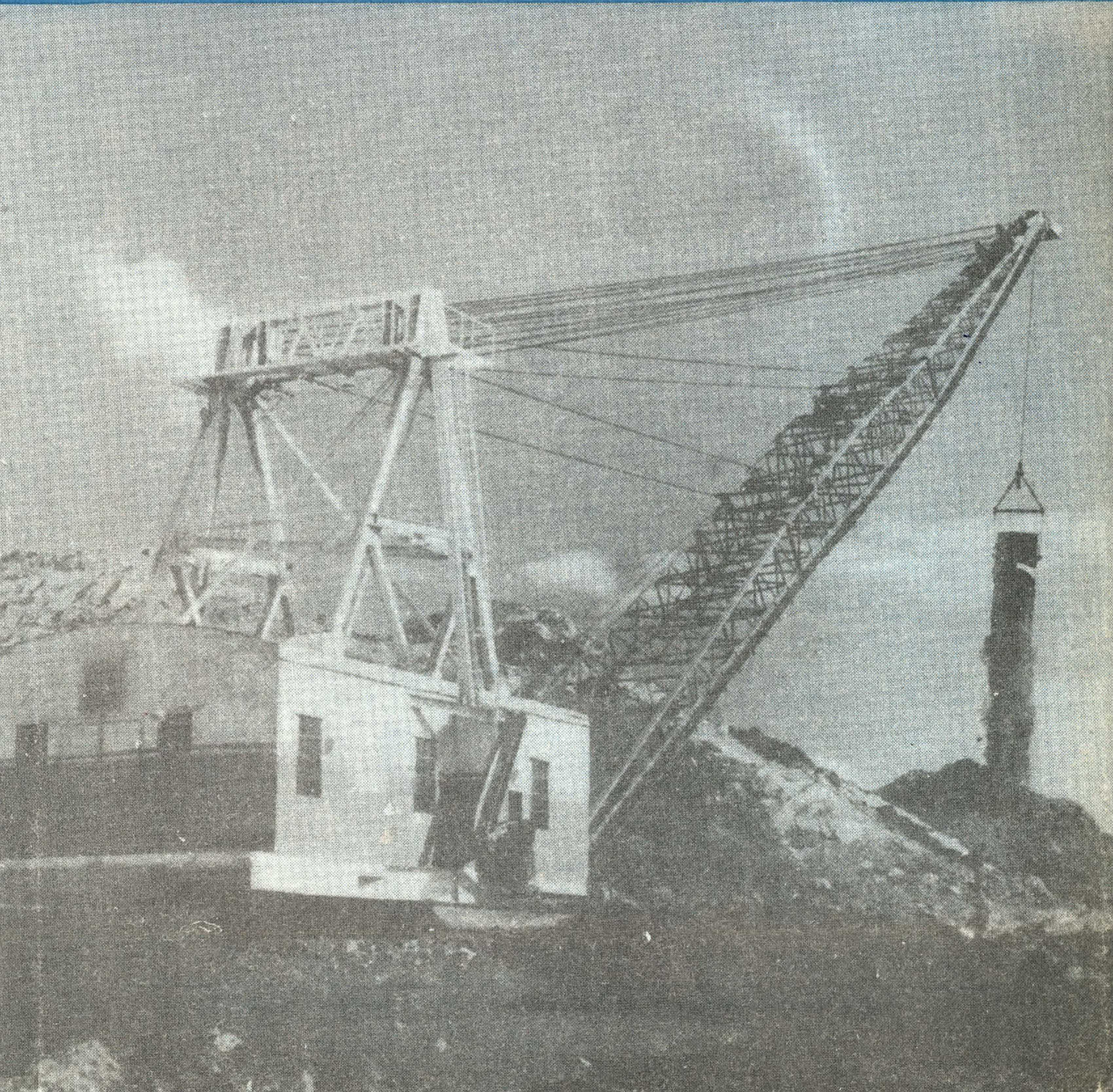
Case — II DEEP SHAFT MINE — Depth Range — 300-500 m

Rated Production 4200 tpd/1.20 Million Tonne Per Year

Items of expenditure B2P	Case I: Total capital in Rs lakhs	Case I: % of the total capital	Case I: Specific investment Rs per tonne of annual rated production	Case II: Total capital in Rs lakhs	Case II: % of the total capital	Case II: Specific investment Rs per tonne of annual rated production
1	2	3	4	2	3	4
1. Land 1·8	22	0.57	1.74 1·7	22	0.50	1.75
2. Buildings : 37	340	8.85	26 98 27	342	7.82	27.14
(a) Service	30			32		
(b) Welfare	21			21		
(c) Residential	289			289		
3. Plant and machinery 140	2821	73.48	223 88 249	2925	66 96	232.14
4 Furniture and fittings ·50	5	0.19	0.40 ·50	5	0.11	0.40
5 Railway siding	140	3.65	11.11 17	140	3.20	11.11
6. Vehicles	18	0.47	1.43 2	18	0.38	1.43
7. Prospecting and boring	26	0.68	2.07 2·5	52	1.12	4.13
8. Development and others	467	12.16	37.00	864	19.78	68.57
(a) Capital outlay	220			520		
(b) Roads and culverts	16			16		
(c) Water supply	26			26		
(d) Project Management	175			266		
(e) FR preparation and documentation	30			36		
Total	3839		~~305~~ 338	4368		.346

3.8

minetech



Code No. 015:8/MT

VOLUME	FOUR
NUMBER	3&4
SEPTEMBER	1980

EDITORIAL BOARD

Shri M Koshi	Addl CME
Shri S K Verma	Dy CME
Shri C Krishnamurthi	Dy CME
Shri A Mookerjee	Supdt (Mines)

Published by

Technical Publications Wing
Central Mine Planning & Design Institute Ltd
and Printed by Shri M Koshi, Addl CME
at the CCL Press, Ranchi

The views expressed by the authors are their own and not necessarily of CMPDI

With best compliments

Chairman-cum-Managing Director
Central Mine Planning & Design Institute Limited
Gondwana Place, Ranchi 834 008

minetech

VOLUME FOUR NUMBER THREE & FOUR

CONTENTS

Congratulations to the Prize Winners

(Minetech Prizing Scheme)

In the third year of publication of MINETECH, 29 articles appeared in the four numbers of the third volume. All the articles were adjudged for the award of five prizes, first two of Rs 500/- each and next three of Rs 300/- each.

A Committee consisting of senior mining engineers of the country went into the selection of prize-winning articles.

1. Controlled Blasting Technique

   P R Roy,
   C Krishnamurthi. — CMPDI — Rs 500/-

2. Planning of Opencast Coal Mines—Norms

   V P Talwar, — CMPDI — Rs 500/-

3. A Study on the Optimisation of Sample Size for Proximate and Ultimate Analyses of Coal Core Samples

   N Sen,
   P K Pal, — CMPDI — Rs 300/-

4. Dragline Excavators in Surface Mining

   M N Raina, — CMPDI — Rs 300/-

5. Beneficiation Characteristics of Barakar Measure Coals in the Jharia Coalfield

   G G Sarkar
   H P Das
   P N Ash — C F R I, Dhanbad — Rs 300/-

article

experience with self-advancing powered supports at moonidih colliery

A K Gulati & A K Singh

Introduction

Moonidih colliery in the Jharia coalfield was the second project undertaken in collaboration with Poland for exploitation of deeper coal seams of prime coking nature. The sinking of two 7.5 m dia. shafts was started in 1964 and completed in 1967.

Since the introduction of longwall caving technology in early 1974, a number of longwall faces of varying lengths have been extracted successfully at Moonidih by conventional solid blasting and individual friction prop support. To achieve the planned production of 7000 tpd, with this technology about 25 conventional faces were required to be operated at a time. This, however, was found to be impractical from the ventilation point of view. Besides, managing the very large manpower required to operate so many faces would also have been impracticable.

The project was reviewed and the Revised Project Report, 1978, envisaged five powered-support faces to be introduced successively in conjunction with five roadheading machines for development of panels, to achieve the target output of 7000 tpd from the mine. Each 150 m long face was to produce about 1000-1400 tpd of coal (average) depending upon the thickness of the seam.

The first powered support equipment were imported from M/s Dowty (UK) in March, 1978. Two teams consisting of engineers, managers, foremen and fitters were sent to the UK for intensive training at the Dowty works. Under the guidance of Dowty and NCB instructors the persons selected for the face were trained on the surface at a 30 m mock face for various operations of installation, operation and maintenance. The same persons were trained underground on-the-job for a period of six weeks to enable them in managing the face with full confidence. The first panel, PS-I (550 m long) was commissioned on 28.8.78 in XVII top seam (1.75 m thick). The extraction of this panel was completed on 29.9.79 and the face equipment were shifted to an advancing face in the next panel PS-2.

Project Officer, Mining Engineer, Moonidih Project, BCCL

Specification of Equipment

A brief description of the equipment deployed at the face at Moonidih is given below :

Face Support

Make : Dowty, 4 X 280 t, Rigid base chock.

Extended height	1826 mm
Closed height	1242 mm
Setting load	80 t
Yield load	280 t

Gate-end Supports

20 t Dowty, Duke props.

Extended height	1800 mm
Closed height	1245 mm
Yield load	20 t

Shearer

Make : Anderson Mavor, double-ended ranging drum shearer

Dia of drum	1220 mm
Width of drum	610 mm
Motor	150 kw

Face Conveyor

Make : Anderson Mavor, 90 kw drive at each end, 18 mm double-strand chain, 650 mm centre, complete with ramp plates and Perard cable-handling system, chain speed 0.95 m/sec.

Stage Loader

Make : Anderson Mavor, 20 m long, 650 mm chain centre, double-strand 18 mm chain, chain speed 1.08 m/sec.

Face Signalling and Communication

Davis Derby, Sivad MK-III, with provision of stop/start units, loudspeakers and microphones; central control console adjacent to the main electrical switches.

Power supply

Gate end panels and flexible power cables upto face equipment motors; 550 volt power supplied through four 200 KVA Transformers connected in parallel at at substation.

Face length	150 m
Panel length	550 m
Total No. of chocks	110 Nos. at 1.35 m centre distance
Seam hight	1.75 m

Layout of panels is shown in Fig. 1.

Selection and Training of Manpower

The face miners required for the powered support face were selected after a thorough scrutiny and personal interviews conducted for selection tests for aptitude, motivation and zeal for the new technology to be introduced for the first time in India. Annexure I shows the manpower at the face.

Fitters and electricians were also selected from the Industrial Training Institute (ITI) boys through interview and scrutiny.

The crew, thus selected, was trained at the surface installation at the colliery training centre for a period of six weeks under supervision of NCB and Dowty Engineers. The persons were again trained on—the-job underground for a period of eight weeks. Figure 2 shows the training scheme for the face crew adopted at the Moonidih mine.

A composite group of experienced miners thus selected and the ITI boys, was put on the jobs connected with the operation and maintenance of the equipment at the face on an 'all-men-all -job' basis.

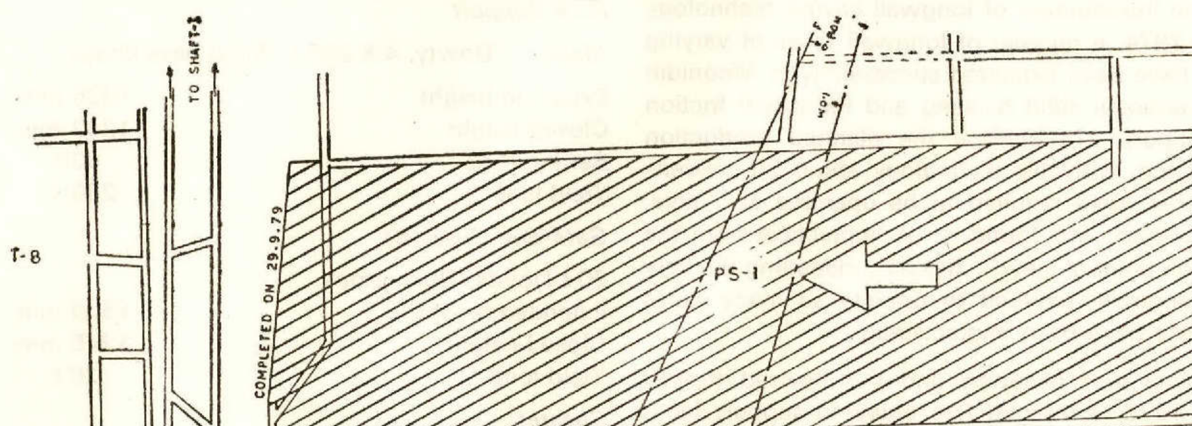


Fig. 1

Part Plan of XVII Top Seam Showing Layout of Powered Support Faces PS - I & PS (-) II

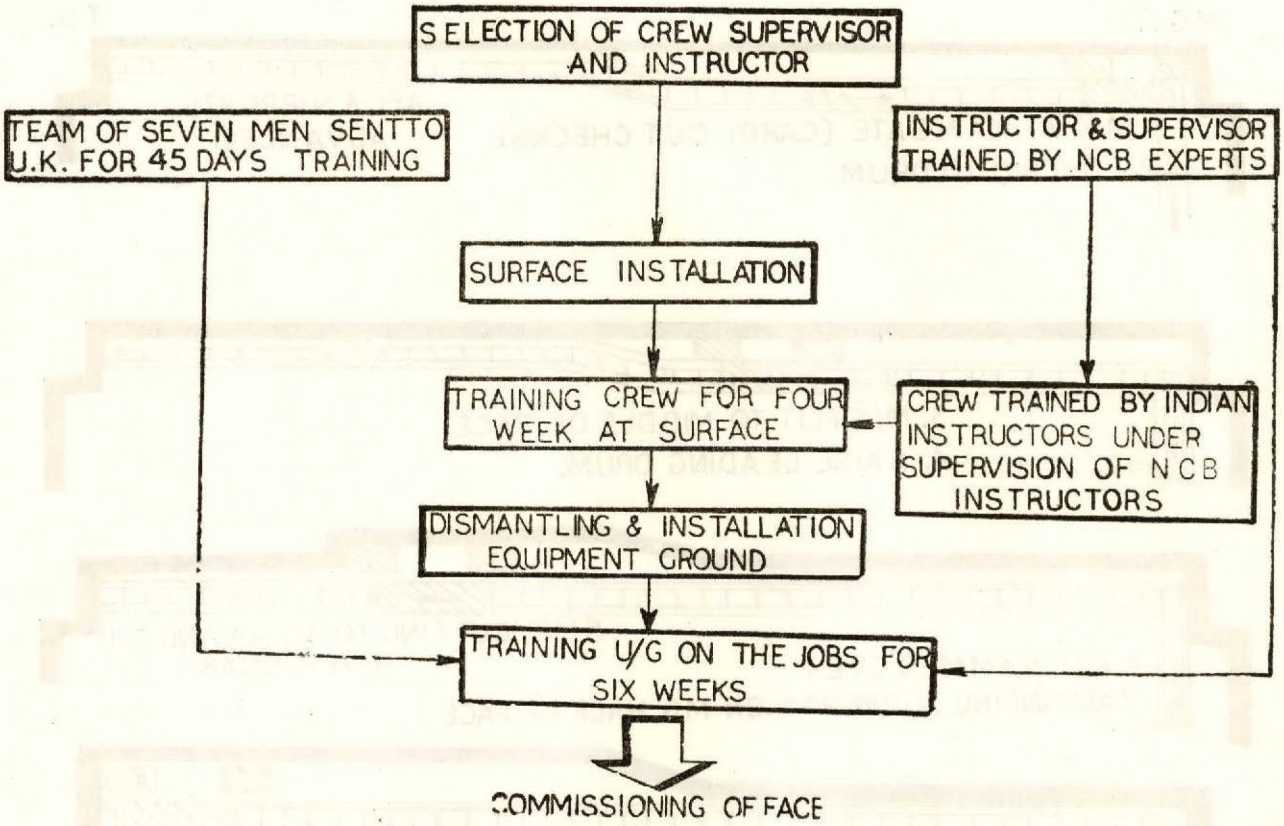


Fig. 2
Dowty Training Scheme

Performance

The performance of the first face worked at Moonidih with self-advancing powered supports (Dowty) is given in Annexture II. The salient points are given below :

	Average	Maximum
(a) Production per day (Including period to cross faults)	600 t	1800 t
(b) Advance, metres per day (Including period to cross faults)	1.6 m	4.2 m
(c) OMS (Face)	15.0 t	40 t

The half-cut method adopted with double-ended ranging drum shearer for shearing coal is shown in Fig. 3. The cycle time at face PS-I is given in Annexure-III.

Though the performance of the face in respect of production, productivity, safety etc. were very encouraging, the full potential of the equipment could not be utilised due to certain bottlenecks at loading point and horizontal and vertical transport, mainly the limited hoisting capacity.

Wage Structure & Cost

The wage-structure for the mechanised face was drawn providing sufficient motivation for the crew to achieve the optimum efficiency. The workers' unions were consulted and an agreement was signed between the management and the workers' representative. In brief, the wage structure is as below :

2 Shears or 450 t/shift :	Allowance of Rs 6.38 above Rs 14.62 = Rs 21.00
3 shears or 675 t/shift :	Allowance of Rs 6.38+ Rs 6.00 above Rs 14.62 = Rs 27.00
4 shears or 900 t/shift :	Allowance of Rs 6.38 + Rs 6.00+Rs 8.00 above Rs 14.62 = Rs 35.00
5 shears or 1125 t/shift :	Allowance of Rs 6.38+ Rs 6.00+Rs 8.00+Rs 10.00 above Rs 14.62 = Rs 45.00

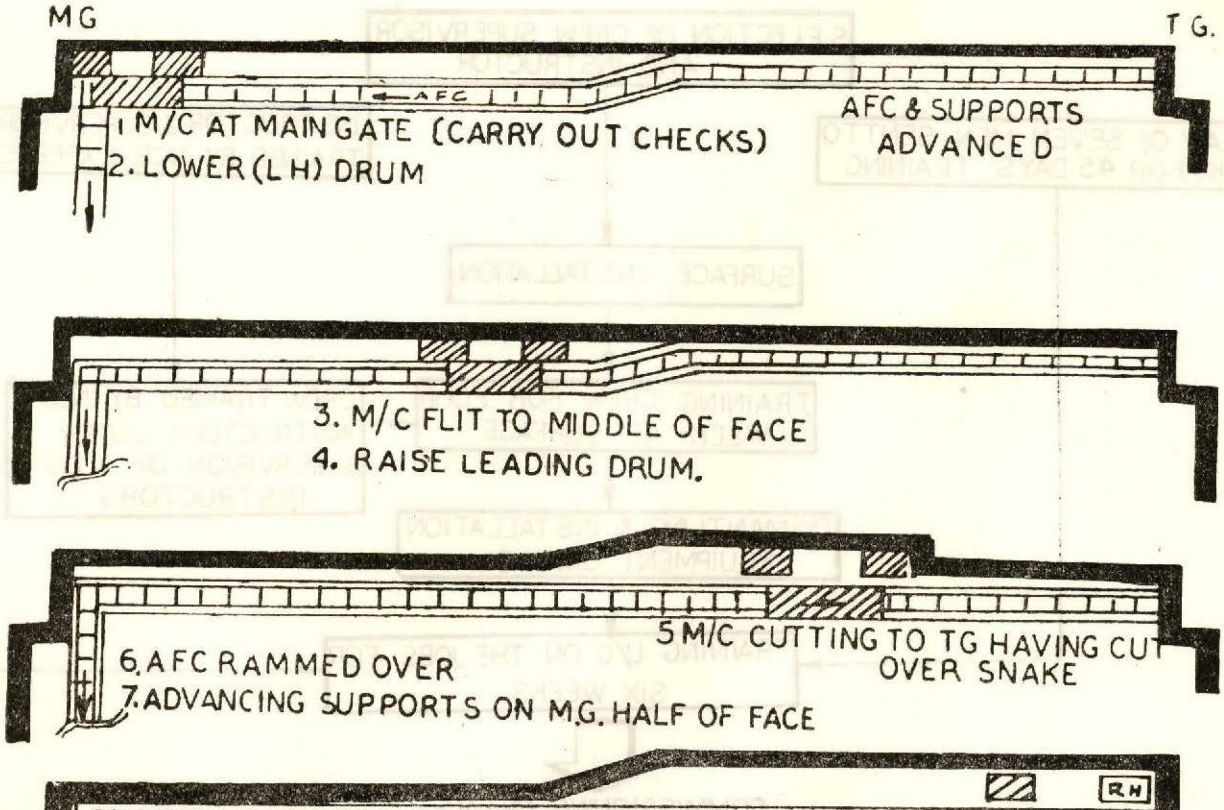


Fig. 3
Sequence of Machine Operations Half Face system

The total cost of the equipment including spares was Rs 3.5 crores.

Comparison of cost between powered support face and conventional longwall face is given in Annexure IV.

Safety

Introduction of powered supports have considerably reduced the accident rate, which is evident from the following figures :

Description	Conventional longwall	Powered support
(a) Mandays lost due to accident	373	20
(b) Rate of accident per thousand tonnes production	0.317	0.021
(c) Rate of accident per thousand manshifts worked	0.500	0.250

Salvaging Face PS-I

As stated earlier, the first face PS-I was extracted completely on 29.9.1979. The equipment salvaging started from 1.10.1979 and the second face PS-II was commissioned on 1.12.1979. Fig. 4 shows the skid trolley used for transportation of chocks. It is worth mentioning that with no earlier experience and very little know-how, the whole job was completed within two months time. Which is in no way a small achievement.

Advancing Face PS-II

The second face PS-II, which was commissioned after shifting the equipment from PS-I face is an advancing face, though the gate roads had been already driven upto about 500 m in advance. Fig. 5 shows the system of maintenance of the main gate road.

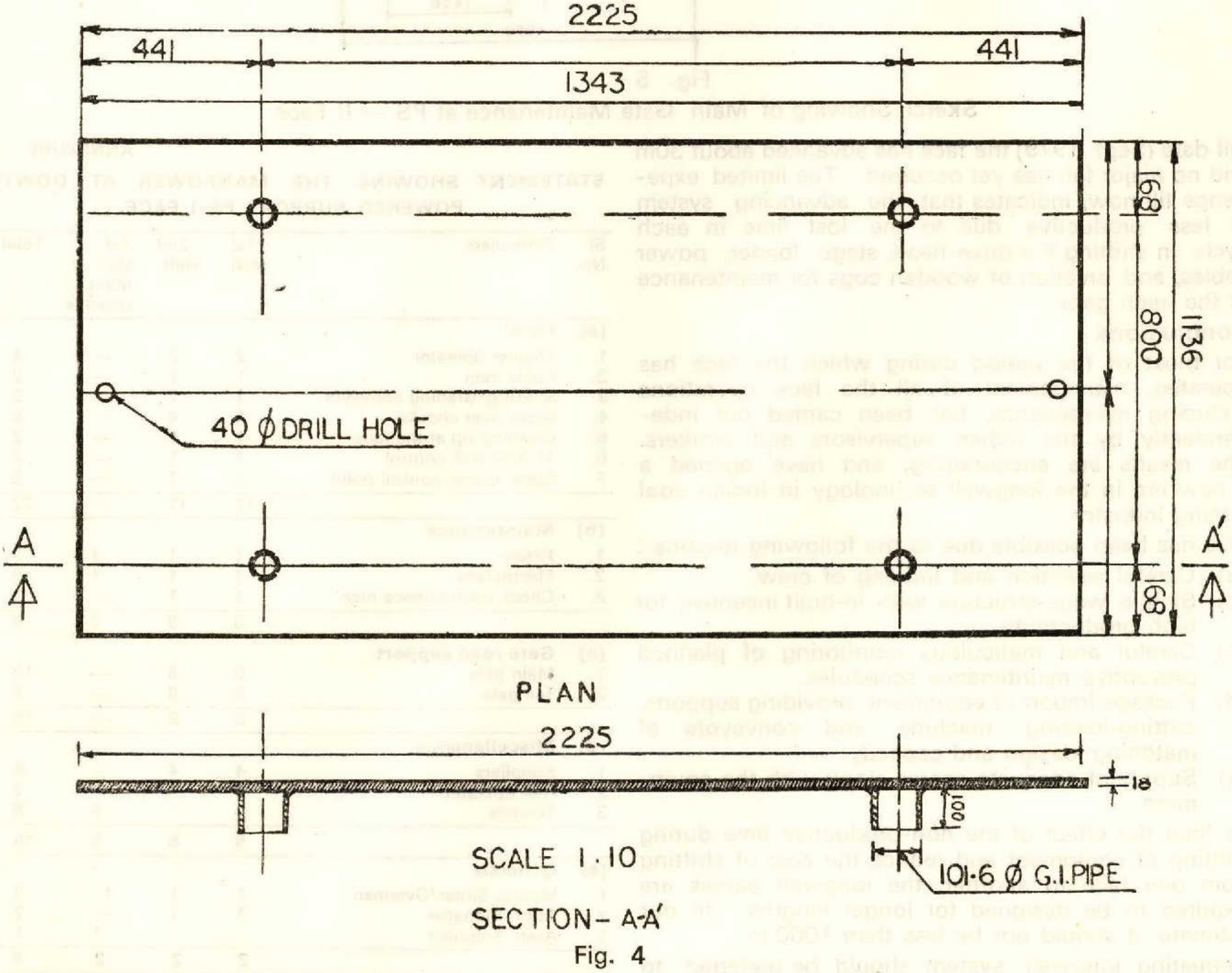


Fig. 4

Plan Showing Skid Trolley Used for Transportation of Dowty Chocks

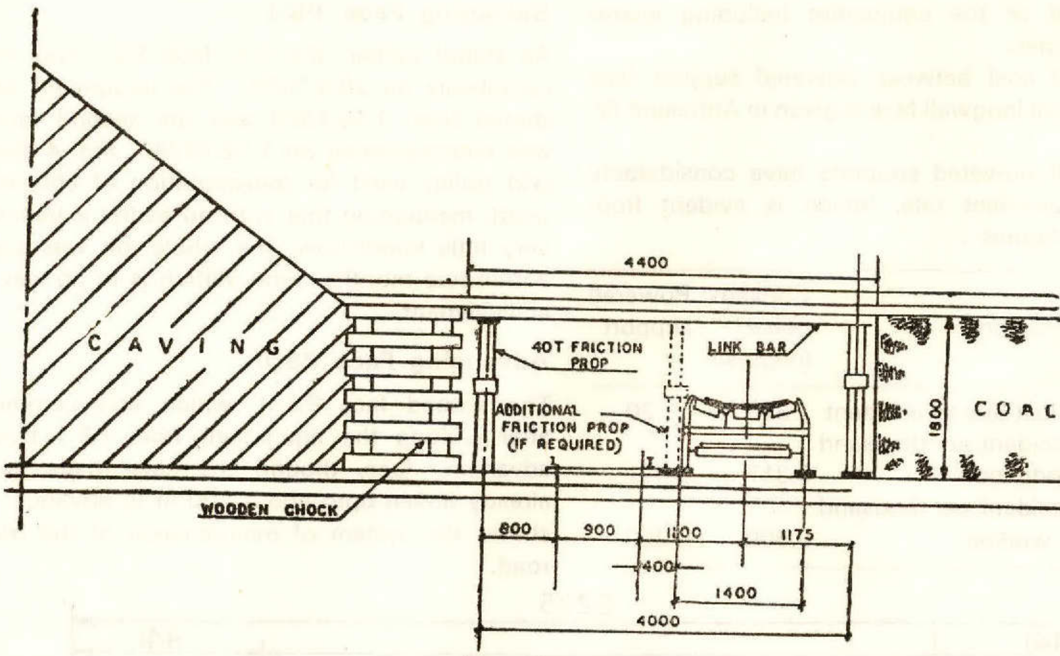


Fig. 5
Sketch Showing of Main Gate Maintenance at PS — II Face

Till date (Sept. 1979) the face has advanced about 30m and no major fall has yet occurred. The limited experience till now, indicates that the advancing system is less productive due to the lost time in each cycle in shifting the drive-head, stage loader, power cables, and erection of wooden cogs for maintenance of the main gate.

Conclusions

For most of the period during which the face has operated, management of all the face operations including maintenance, has been carried out independently by the Indian supervisors and workers. The results are encouraging, and have opened a a new era in the longwall technology in Indian coal mining industry.

This has been possible due to the following reasons :

(a) Careful selection and training of crew.
(b) Simple wage-structure with in-built incentive for high productivity.
(c) Careful and meticulous monitoring of planned preventive maintenance schedules.
(d) Package import of equipment, providing supports, cutting-loading machine and conveyors of matching design and capacity.
(e) Supply of adequate spares along with the equipment.

To limit the effect of the non-productive time during shifting of equipment and reduce the cost of shifting from one face to another, the longwall panels are required to be designed for longer lengths. In our estimate, it should not be less than 1000 m.

Retreating longwall system should be preferred to the advancing system.

ANNEXURE – I

STATEMENT SHOWING THE MANPOWER AT DOWTY POWERED SUPPORT PS–I FACE.

Sl No.	Particulars	1st shift	2nd shift	3rd shift Main-tenance	Total
(a)	**Face**				
1.	Shearer operator	2	2	—	4
2.	Cable man	1	1	—	2
3.	Shaking/Grading conveyor	1	1	—	2
4.	Move over chocks	4	4	—	8
5.	Cleaning up at the face	1	1	—	2
6.	At AFC D/E control	1	1	—	2
7	Stage loader control point	1	1	—	2
		11	11	—	22
(b)	**Maintenance**				
1.	Fitters	1	1	1	3
2.	Electricians	1	1	1	3
3.	Chock maintenance men	1	1	—	2
		3	3	2	8
(c)	**Gate road support**				
1.	Main gate	5	5	—	10
2	Tail gate	3	3	—	6
		8	8	—	16
(d)	**Miscellaneous**				
1.	Suppliers	4	4	—	8
2.	Belt operator	1	1	—	2
3.	Tyndals	–	–	6	6
		5	5	6	16
(e)	**Officials**				
1.	Mining Sirdar/Overman	1	1	1	3
2.	Asstt. Manager	1	1	—	2
3.	Asstt. Engineer	–	–	1	1
		2	2	2	6
	Grand total	29	29	10	68

ANNEXURE-II

Performance of Powered Support Face PS-I

Sl. No.	Particulars	Sept. 1978	Oct. 1978	Nov. 1978	Dec 1978	Jan. 1979	Feb. 1979	March 1979	April 1979	May 1979	June 1979	July 1979	Aug. 1979	Sept. 1979
(a)	**Production**													
1.	Average production tpd	430	480	816	912	959	620	305	352	299	505	800	903	439
2	Total production in tonnes	9870	11230	22257	23728	24936	13021	7590	8820	8067	12900	20800	23470	10970
(b)	**Manpower**													
1.	Total manshifts worked	1194	1180	1162	1305	1357	935	1138	1403	1410	1431	1299	1254	1340
2.	Output per manshift (face)	8.26	9.52	18.29	18.18	18.38	13.9	6.98	6 29	5.72	11.39	16.00	18.6	8.18
(c)	**Face advance**													
1.	Total No. of cuts	53	64	97	105	116	47	31	37	34	61	93	110	57
2.	Total advance of face (metres)	27 0	32 9	62.2	60 5	64.72	28 02	18 0	22 0	16 0	31 0	54.00	56.0	33.00
3	Average advance per cut (metres)	0.50	0.51	0 63	0.57	0.55	0 59	0 56	0 60	0.47	0 50	0.58	0.51	0 58
4	Average per day advance (metres)	1.08	1.32	2.39	2.33	2 59	1 12	0.72	0.85	0.61	1.19	2 10	2.16	1.37
	Remarks	On the job training						Crossing of two faults						

ANNEXURE-III

Operational Times at Powered Support Face

A. Travel time

Inbye	25 min
Outbye	25 min
To mid point of face and out	5 min
	55 min

B. Preparation time

Tools on face	5 min
Tools down	5 min
Time for waiting	15 min
	25 min

C. Machine running time/shear

Shearing half face 3 m/min	25 min
Flitting half face @ 4.5 m/min	15 min
Shearing other half face @ 3 m/min	25 min
Flitting half face @ 4.5 m/min	15 min
Ends cutting	15 min
	95 min

D. Machine ancillary time

Check/change picks	15 min
Other delays—check horizon, raise, lower drums	10 min
	25 min

E. Operational delays at face

Adjust horizon	5 min
Miscellaneous	10 min
Lumps	10 min
	25 min

F. Operational delays outbye — 15 min

G. Total shift time — 480 min

H. Available shift time $= 480 \text{ min} - (55+25)$ — 400 min

I. Time required for one shear — 160 min (C+D+E+F)

J. No. of cuts/shift $\frac{400}{160} = 2.5$

K. No. of cuts/day (2 shifts) 5

ANNEXURE-IV

Comparison of Production Costs at Loading Points

Description	Powered support (Rs)	Conventional (Rs)
Wages	4.86	23.52
Stores	2.16	6.40
Power	1.20	1.60
Interest	12.28	5.92
Depreciation	8.64	4.96
Total cost/tonne	29.14	42.40
Capital invested/tonne of annual production	80.13	45.60

safety in coal sector-an appraisal

H S Ahuja

To understand the present state of safety in coal mines, it will be pertinent to see what was its status at the time of nationalisation. The late S Mohan Kumaramangalam had remarked in his posthumous book,—Coal Industry in India — Nationalisation and Tasks Ahead.

"Unscientific and unsafe mining, in other words 'slaughter mining' which the coal mine operators in the Private sector resorted to, gave them good quality coal at comparatively cheaper price, since they did not adhere to principles of safety or scientific exploitation of the mineral. The exploitation of labour — in the form of non-payment of statutory wages, non-payment of provident fund dues, non-payment of VDA, employing 'forced labour' gave them scope to amass further wealth.

"The mining practices adopted by colliery owners were in total disregard of the principles of mineral conservation and safety.

"In the case of underground mines, inclines were driven in a most unscientific manner, expediency being the rule. Mines being small in size and numerous, the barriers were innumerable and valuable quantities of coal remained locked in these barriers. While the selective mining provided high profits to the owner it resulted in great loss of national asset. In many collieries galleries were driven beyond safe limits. 'Robbing' of pillars was rampant.

"Often, very high and wide galleries were driven leaving small pillars which resulted in collapse of working places in several cases. Ventilation was often poor; so was supporting. Precautions were not taken against coal dust.

"The workers were forced to work under the most adverse conditions; safety regulations being thrown to winds in several collieries........."

While this may be true of a large number of small coal mines, it did not really give due credit to a majority of bigger organised units in the private coal industry which had set much higher standards of safety and were generally law abiding. The so-called smaller mines contributed only about 5% or so of the annual output of coal, approximately 70 million tonnes in 1973.

Following the holding of First & Second Safety Conferences during 1958 and 1966 respectively, a number of important recommendations had been made to improve the status of safety in coal mines and implementation thereof was on the anvil. Mention here may be made of some of the more important safety measures that had already been taken.

(1) Use of 'mug battis' had been stopped, and electric cap lamps introduced both to imporve lighting (and ventilation) and as a safegaurd against ignition/explosion hazard.

(2) All coal mines had been declared gassy (w.e.f. 1.4.67) and precautionary measures against explosion hazard like installation of mechanical ventilators, stone dust barriers, treatment of coal dust with water within 90 m zone taken in a large number of mines.

(3) Supply of footwear and helmets to a large number of workers was made in most of the bigger units.

(4) Setting up of training centres for vocational training of workers had started.

(5) National Council for Safety in Mines had been set up for safety education and propaganda.

(6) The institution of Mines Safety Weeks had been firmly established commencing from 1962-63 bringing about noticeable improvements in the general working conditions of a large number of mines and in the matter of increasing safety consciousness of workers.

Deputy Director General of Mines Safety (Central Zone), Dhanbad

Nationalisation of all coal mines except TISCO and IISCO* captive mines (NCDC & Singareni Collieries Co. were already in public sector) in 1971-73 had certainly generated a valid expectation of much greater advances in safety, both technically and organisationally, so as to improve further and at a faster pace the general working and environmental conditions in the coal mines.

With a unified control of the coal industry in public sector, financial resources were made available to implement the safety measures. and infrastructural facilities created for proper and scientific working and management of all mines. In this regard the following are worthy of note.

(1) Internal safety organisations have been set up in all the four subsidiaries of Coal India Limited for carrying out internal safety audits and regular inspection of mines to pinpoint shortcomings and recommend remedial measures etc.

(2) The Central Mine Planning & Design Institute with its four Regional Institutes, set up as a subsidiary of Coal India Limited is now in a position to draft Feasibility/Project Reports of new mines on scientific lines providing all safety requirements required by law or as per recommendations of DGMS and other agencies.

(3) Supply of safety equipment and materials is now generally uniform and better in all mines.

(4) Staffing of managerial and supervisory personnel is now much better than before.

(5) The Government of India in the Ministry of Energy have created a Sectoral Budget for Safety and drawn up a time-bound action programme as an advance action for implementation of recommendations of Safety in Coal Mines Committee set up by the said Ministry in the wake of Chasnalla colliery disaster in 1975. This Committee is understood to have made a number of important recommendations for bringing about positive improvements in safety measures in mines and these are presently said to be under active consideration of the Government.

(6) Group Training Centres have been set up by all the subsidiaries of Coal India for vocational training of workers, and formulation and execution of special training programmes have been undertaken for shotfirers and statutory supervisory staff, roof support & strata control supervisors, middle-level management officials, ventilation officers and safety officers.

(7) For tackling surface fires and blanketting of subsidence areas, each Area has been equipped with bulldozers.

(8) Provision of a large number of Burnside boring machines for putting long holes underground to guard against danger of inundation from old workings and dewatering of such workings.

(9) Pick mining has been replaced with winning coal by use of explosives in bord & pillar workings and this has done away with the hard labour and drudgery of the miner. (At the time of nationalisation, about 60% of coal in BCCL mines was won by picks).

(10) Holding of regular co-ordination meetings with DGMS officers in which progress in implementation of various safety measures is monitored, causes of accidents discussed and analysed and preventive steps duly identified. Decisions taken at such meetings are by and large being implemented.

It must be said that even today, a number of mines continue to work with violations, which show almost the same pattern as before nationalistion. While reviewing the progress of implementation of the recommendations of the Third Conference on Safety in Mines held at Mosaboni on 10th September, 1979, the reconstituted Review Committee dwelt upon in detail the recurring violations in strata control, ventilation, explosives, coal dust suppression, electrical and mechanical engineering (Appendix II).

The existence of violations in the nationalised sector mines, is perhaps due to overemphasis on production at the cost of safety. One reason appears to be the higher fixation of production targets for mines which do not have the required capacity or places of work to give the desired output. Another reason could be the frequent transfer of managers/agents and their desire to show good production during their short tenure at a mine for personal gains.

The following Table shows the position before and after nationalisation of coal mines of notices under Section 22 and prohibitory orders under Section 22(3) or Section 22(1A) of Mines Act issued by officers of DGMS, which is an important index of the safety condition in mines and state of

*IISCO mines were taken over by the Government in 1975.

implementation of safety requirements, direction and control of mines.

TABLE

Year	Notices issued	Orders issued
	Section 22(1)	Sec. 22(1A)/Sec. 22(3)
1966	50	184
1967	177	89
1968	18	119
1969	27	132
1970	58	66
1971	124	60
1972	171	119
1973	140	59
1974*	426	34
1975	74	56
1976	49	31
1977	29	25
1978	40	44

* Special drive regarding winding and FLP features.

The overall position is hopeful as reflected firstly through the creation of necessary infrastructure and financial resources for working mines safely and secondly from the general downward trend in the number of accidents (fatal and serious) and the rate of fatalities and serious injuries to work-persons in mines (Appendix I A & B).

Some broad aspects of the problems in the industry and the remedial measures are briefly discussed below.

Support of Ground

This problem area is the single largest cause of fatalities (40%) in coal mines every year. There is now a concerted effort towards use of better quality timber and gradual increase in the use of steel supports in permanent roadways and friction and hydraulic props etc. as temporary supports near working faces. Of late a new form of temporary support, called 'Safari Support', has been developed in collaboration with CMRS and this has provided a quick and easily erectable means of supporting freshly exposed ground within 9 m of working faces in development workings, at loading points (in both development and depillaring areas), and while heightening of galleries by lifting floor coal. Considerable interest has been generated in the coal mines of one of the subsidiaries of CIL in this form of support which is nothing but a crossbar (wooden or steel) pinned into the coal sides with special type of steel clamps. This company has decided to go in a big way during the current year towards adoption of this and other forms of supports for freshly exposed ground so as to ensure that no workperson has to work under unsupported roof in the potentially unsafe and vulnerable areas.

The ILO multi-disciplinary Team (PIACT Mission, under International Programme for the Improvement of Working Conditions), which visited India in November-December, 1978, has recommended that multi-skill face working be adopted as soon as possible in bord & pillar workings and that in each working place at least four screw-jack props be provided so that each coal filler can erect them as soon as he has cleared sufficient coal to provide the floor space. It has also been recommended that each colliery company set up, as a start, a small strata control/mechanisation branch for development and adoption of suitable strata control methods.

Technology

In India today over 98% of the coal output from underground workings is won from bord and pillar workings. This system of mining is likely to continue for another 10-15 years. There are about 20 longwall working faces and one powered support face today. There are firm plans with CIL for more number of mechanised self-advancing powered support faces and more mechanised open pits with 5:1 overburden to coal ratio in the near future. It may be mentioned here that the spectacular reduction in the number of fatal accidents due to fall of roof and sides in UK from 240 in 1947 to 6 in 1977, due to the installation of modern self—advancing powered supports on mechanised longwall faces. Quick-setting supports are to be adopted near working faces, both in development and depillaring areas.

Explosion and Other Hazards

A proper organisation for coal dust suppression and treatment in mines is yet to be built up and continued availability of stone dust of right quality and required quantity at regular intervals ensured and continuous monitoring of environmental conditions with respect to danger from inflammable gas and fires has still to be provided in most of the degree III and fiery mines to guard against danger of explosions in mines. The position has of late become specially difficult due to shortage of electricity and frequent interruption of power increasing the potential danger. While the danger of inundation, fires and subsidences had been identified minewise by the Bagchi Committee appointed under the aegis of Safety in Coal Mines Committee of the Ministry of Energy, implementation of a number of its recommendations has yet to be completed.

Organisation

In the organisational plane, it is necessary that much serious thought is given to the following shortcomings.

(i) The internal safety organisations do not appear to enjoy the status and importance they ought to have.

(ii) The status and authority of the present day managers of mines is not of a sufficiently high level.

(iii) Majority of the managers of public sector mines do not have enough experience.

(iv) There is an overall shortage of qualified mine surveyors affecting seriously the standard of mineplans.

(v) There is a growing evidence of lack of devotion to duty on the part of supervisory staff.

(vi) Very little accountability for safety at different levels of supervision and management is demanded.

(vii) There is as yet no well-organised R&D wing in the different public sector units.

Workers' Involvement

Real progress in the promotion of safety and health of workers will, however, be not achieved without the full co-operation and commitment of the employees and trade unions in good measure. It is a historical fact that in almost all other advanced countries the trade unions have been at the forefront of all demands for major advancements in the raising of safety standards and improved working conditions whether through enactment of safety legislation or codes of good practices, and Indian counterparts are no exception. The recently noticed wrath of workers against the mine officials following a fatal accident at a mine is a manifestation of their anger and anguish against unsafe conditions in their respective mines. This energy, however, needs to be properly channelised by encouraging 'demands', but in an orderly manner—for safe working conditions so that accidents can be avoided.

APPENDIX 1

Comparative Statement of Fatal & Serious Accidents in Coal Mines

A. Fatal Accidents

Year	No. of fatal accidents	No. of persons killed	No of persons s/inj.	Death rate per 1000 persons** employed	Death rate per Mill. tonnes of coal raised
1	2	3	4	5	6
1961	222	268	36	0.65	4.81
1966	196	222	37	0.52	3.14
1971	199	231	45	0.60	3.05
1976	209	296	35	0.58	2.82
1977	216	237	41	0.47	2.25
1978*	162	185	32	0.37	1.76
1979*	143	179	37	0.36	1.70

B. Serious Accidents

Year	No. of S/Acc.	No of persons S/Inj.	Serious injury rate per 1000 persons** employed	Serious injury rate per Mill. tonnes of coal raised
1	2	3	4	5
1961	3515	3605	8.88	64.71
1966	1934	2008	4.71	28.50
1971	1460	1542	4.03	20.39
1976	2800	1877	3.68	17.94
1977	2093	2177	4.38	20.97
1978*	1902	1963	3.95	18.72
1978 (upto Nov.)	1953	1802	—	—
1979 (upto Nov.)	1692	1758	—	—

(a) Including persons seriously injured in fatal accidents also.
* Provisional and subject to further revision.
— Serious accidents data lag by one month.
** Rate of 1978 & 1979 have been calculated on the basis of estimated employment data of 1978 and hence are provisional and subject to revision.

Appendix -II

LIST OF REPEATED VIOLATIONS AS OBSERVED BY DGMS

Strata Control

1. Heightening and widening of original galleries beyond permissible limits.
2. Excessive splitting of pillars
3. Lag in stowing — large scale voids.
4. Failure to support workings as per STR; support in longwall face scanty; collapse of parting.
5. Benches in overburden and coal not formed in accordance with statutory provisions.

Ventilation

1. Inadequate ventilation due to — non-construction of ventilation stoppings; inadequate capacity of fan; improper ventilation circuit.
2. No sequence control and interlocking arrangements in the auxilliary fans.
3. Underground development being done without providing for mechanical ventilator (in some cases).
4. Sectionalisation not done (to be completed in a number of mines).

Coal Dust Suppression

1. Non-provision of stone dust barriers.
2. Arrangements for spraying working faces and roadways within 60 m of faces with water not provided (from permanent source of supply).
3. Acute stone dust deficiency.

Explosives

1. Solid blasting being carried out without permission/after withdrawal of permission/continued in degree II gassy mine.
2. Solid blasting with instantaneous detonators.
3. Shotfiring without water spraying.
4. Unapproved exploder used in degree III gassy mine.

Others

1. Self-rescuers not provided in degree II gassy mine.
2. Working being made in close proximity of suspected water-logged workings.
3. Workers employed for seven days in a week.
4. Accumulation of fallen coal not removed; roadways not treated with incombustible dust.
5. Depillaring without isolation/preparatory stoppings.
6. Fire-fighting equipment not provided; fire-fighting organisation not set up.
7. Protective works against fire not up-to-date.

Electrical Engineering

1. FLP features not provided/disturbed in underground equipment.
2. Cable boxes not filled with compound.
3. Earth leakage relays not provided/rusted and inoperative.
4. Oil circuit breakers being used with very little oil.
5. Flexible cables connected to shovels etc. defective and damaged; not supported and protected.

Mechanical Engineering

1. Automatic contrivances not provided in winding.
2. Brake of winding engine not effective.

MINETECH, a quarterly technical journal of Coal India Limited (A Government of India Enterprise) is regularly published by Central Mine Planning & Design Institute Limited. It contains research and review articles on various aspects of the science and art of mining and allied subjects contributed by the Engineers, Planners & Designers employed within and outside Coal India Limited including foreigners. It also covers Spotlight, Notes and News, DGMS Circulars and Readers' Response and other features.

Rate of Subscription

	For one year of four numbers	One number
Inland	Rs 40/-	Rs 10/-
Foreign	U S $ 12	U S $ 3
	U K £ 6	U K £ 1.5

The above rates are effective from Volume V. They include postal charges with foreign air mail. 50% concession rates are offered to officers of Coal India, Central Government Departments & bonafide students.

SUBSCRIPTION FORM

To
The Editor, MINETECH
Technical Publications Wing
Central Mine Planning & Design Institute Ltd
Gondwana Place
Ranchi 834 008

From :

Name of the Subscriber (Block Letters)..

Postal Address..

....

....

I/We enclose a bank draft of Rs......................................(U S $..................................for the year/Volume............................drawn in favour of Central Mine Planning & Design Institute Ltd, Ranchi towards..................................subscription of your journal MINETECH.

Signature..

Guide for Accepting Articles for Publication

1. Articles should be typed on one side of a foolscap size sheet of paper with double spacing leaving a margin on the left hand side. A brief summary of the article should be given in the beginning.
2. Two copies of articles are to be sent to the Editor. Unpublished articles will not be returned to authors unless asked for.
3. Diagrams should be clear and be drawn in Indian ink on tracing paper or smooth white board suitable for blockmaking. Letterings on diagrams should be large enough on the original so that the letters are legible after reduction while blocks are made.
4. Diagrams should be numbered and have captions. Captions should be typed/pasted behind photographs. A list of diagrams/photos should be attached at the end of the article.
5. Diagrams should use cross-hatching.
6. Metric system in weights and measures should be used.
7. Standard symbols and abbreviations can be used only in formulae. They should be spelt in full in the body of the text.
8. References should be fully documented and typed at the end of the article.
9. Thirty free reprints of the published article will be sent to the author if specially asked for.
10. Every effort will be made to acknowledge the receipt of articles and their suitability for publication. They may be edited for reasons of clarity and space.
11. The Editorial Board reserves the right to reject any article without assigning any reason whatsoever.

analysing objectives of coal india for corporate planning

Dr Basudeb Sen

'Corporate Planning' has been defined differently by different proponents. For the purpose of the present paper, however, we may adopt the following definition. "Corporate Planning is a continuous process through which a corporate body seeks to ensure that its different constituent parts and levels find, for their implementation, a set of plans which are not only internally consistent but also consistent with the (explicit or implicit) corporate identity and the corporate objectives/goals and at the same time appropriately adapted and responsive to the changing environmental conditions under which the corporate body and its parts operate."

Evidently, corporate objectives play an important role in 'Corporate Planning' (CP). They are fundamental to the determination of what a company wants to do and how.

For the purpose of 'CP' and other reasons, corporate organisations now-a-days generally have a statement/ list of corporate objectives. It is necessary that the nature of the objectives, their inter-relationship and relevance in the context of emerging internal and external environment are analysed.

It is proposed to make such an analysis for Coal India Limited (CIL). The Project Black Diamond, 10-year Perspective Plan (1978-79 to 1987-88) of CIL has listed seven objectives for CIL. Each of them may be examined on the basis of the following two criteria :

(A) Whether the listed objective is really an independent and disparate objective, (i.e. it is not subsumed by any other listed objective or, if the other objectives were not in the list, this particular objective would itself be able to claim its status as the sole objective to be pursued) ?

(B) Whether the objective under examination reflects the company's identity, or reflects an ultimate condition for the continued existence of the company ?

Econometrician, Coal India Ltd, Calcutta

Depending on the degree to which any listed objective satisfies the criteria in the context of the current socio-economic environment, the objective will be categorized as primary or subsidiary or a self-imposed restriction.

Now, considering the first objective in the list (i) *"To promote the development and utilisation of coal reserves in the country for meeting the present and likely future requirement of the national economy with due regard to the need for conservation of non-renewable resources and safety of mine workers,"* it is easily seen that this particular objective (i) satisfies both the criteria. It is an independent and disparate objective as the achievement of this objective need not imply achievement of any other listed objective. It also reflects the company's identity. In fact, the nationalisation of the coal industry and the subsequent formation of CIL was guided by the above need to promote scientific development and utilisation of the country's coal reserves.

The second of CIL's objectives is (ii) *"To raise the productivity of coal mining and related activities through introduction of improved technology, streamlining of organisation and management and improving the skills and motivation of the work-force."* All these help to achieve a faster growth of production that may be necessary to meet the country's requirements of coal. It may also be necessary for the purpose of reducing cost of production and thereby help generate surplus. In other words, this objective (ii) helps in the achievement of the objectives (i) and (iii) (referred to below). Also, if there was no other objective, it is difficult to conceive of a situation when an organisation would be guided by a single objective as the objective (ii). Clearly, therefore, this objective (ii) does not satisfy criteria (A) and (B).

The third objective is (iii) *"To generate surplus by optimum utilisation of productive capacity, improving efficiency of operations and adopting appropriate cost* *reduction and cost control method."* This objective is not subsumed by another objective and can even be the single objective of some organisations. Therefore, it satisfies criterion (A). However, generation of surplus may not be regarded as reflecting a company's identity, but may reflect an ultimate condition of an organisation's continued existence. There may be doubts as to whether in the public sector profit should be an objective or not. Yet, if we look at the history of the commercial organitions in any country, it is difficult to find examples of continued existence of an organisation perenially in loss. Moreover, even in a planned socialist economy the nation expects the public sector companies to contribute to the growth of investible resources. This need not be confused with profit making. Thus, it may be said that (iii) satisfies the criterion (B) at least in some restricted sense.

The fourth objective of CIL is (iv) *"To make efficient arrangements for marketing and supply of coal so that coal, coke and other similar derivatives are available to consumers throughout the country conveniently and at reasonable prices."* This is really a part of the objective (i). Even if the coal production is adequate in quantity, in the absence of efficient marketing the requirements of the country may not be met. Although (iv) could be the sole objective of a trading organisation, in view of the continued existence of the private sector coal trade, system of sponsorship in coal distribution, and the railways' effective control over transportation of coal, it is doubtful whether (iv) could have been regarded as the sole objective of a coal producing organization. Thus, it is difficult to accept that (iv) fully satisfies the criterion (A). For similar reasons, it does not satisfy the criterion (B) either.

The fifth objective is (v) *"To promote research and development activities on a continuing basis in the areas of coal mining, beneficiation, new coal-based products or by-products, fuel technology or any other areas having bearing on conservation, development or utilization of coal reserves of the country."* Future requirements of coal in the national economy will be influenced by the R&D activities with regard to coal mining technology, coal preparation and utilisation. This objective, therefore, is partly subsumed by objective (i) and partly by (iii). Moreover, in the present and the near-future external environmental conditions it does not appear realistic to establish an organisation with the sole objective of promotion of R&D implied by (v), and may satisfy neither criterion (A) nor (B).

The sixth objective is (vi) *"To improve suitable facilities for training with a view to upgrading the knowledge and skills of employees in different categories and enabling them to make full use of their capabilities."* It may be regarded as subsidiary to objectives (i) and (iii). Training and development facilities to help raise the productive potential of the human and physical assets of employees may facilitate a faster rise in productivity and gains in terms of cost reduction and surplus generation. But, it is difficult to conceive of organisation which will function with the sole objective of training and development of its employees. Therefore, objective (vi) does not satisfy the criteria (A) and (B).

The seventh objective reads, (vii) *"To look after the welfare of the employees and promote the establishment and maintenance of healthy relations between management and workers, and foster a sense of fellowship and belonging to the company among personnel at all levels."* This may help to raise productivity and reduce the production stoppages or go-slows, and contribute to realization of the objective (i). But again, (vii) cannot be regarded as the sole objective of any commercial organisation, though a socially conscious organization might impose on itself a moral obligation to look after the employees' welfare, and maintain healthy industrial relations.

At this point it may be worthwhile to review very briefly how far these objectives really influenced CIL's activities and operations in past. Although the targets of production could not be achieved in the past years, the major concern has always been to attain the set targets on production and capacity expansion on the basis of anticipated demand. As for productivity, output per manshift rose appreciably in the initial years, but has been stagnating along with the production in the last few years. CIL's history, so far, has been one of continued and increasing losses. However, for a major part, the losses are explained by the limited or no freedom that CIL enjoys in selecting its producing points for supplies to consumers linked by the concerned authorities, and in adjusting its sales prices to inflationary cost escalations. The Railways' rules concerning directional movement of traffic and the linkages established by national linkage authorities act as restrictions to the choice of CIL to locate projects in economically exploitable areas, and to produce more from the profitable or low-cost mines and less from uneconomic ones. The company's concern was to control the rising trend in the cost of production . A government committee examined various issues relating to CIL's costs and the scope for economy. There was a continued and strong pressure on the company to achieve a cost reduction. The company had to repeatedly undertake exercises to present its case for a price rise, and explain the losses in terms of uneconomic prices. However, during the period between 1974 and 1979, pithead prices of coal were raised thrice depite the objections from various quarters. The guiding factor that seems to have influenced the price rise decisions, is the need to compensate the company for the cost escalations beyond its control, and help the company to maintain at least a break-even status.

With regard to the objective (v), efforts made in different directions are yet to contribute significantly to the operations of the company, to the growth of demand in new applications, or to a more efficient use of coal. Training and management development have been given considerable emphasis, and the company has a much larger proportion of trained and skilled employees than it had at the time of nationalisation.

As for the objective (vii) relating to the employees' welfare and healthy industrial relations, it may be said that despite growing welfare expenditure and major wage revisions, industrial disputes and high absenteeism have at times, seriously and adversely affected the production.

Regarding objective (iv), in the immediate post-nationalization period there was a scarcity of coal. A large step-up in production turned the situation into one of plenty when marketing efforts were directed towards sales promotion. But in recent years, a crisis developed where despite the record pithead stocks, consumers suffered from inadequate supplies.

The past experience clearly shows that by far the most important problems with which CIL had to deal with, related to the listed objectives (i) and (iii).

When stocks began to accumulate, or the production targets were not met, or when the market showed symptoms of scarcity, the company's arrangements for marketing and coal supplies came under review. The marketing organization was under close examination and went through stages of expansion, or reorganization. Either the requirements of the country were not being met, i.e. objective (i) was not satisfied, or the growing pithead stocks in danger of spontaneous heating implied rising costs of inventory and risk of a loss of output, i.e. when the situation was against the satisfaction of the objective (iii) relating to surplus generation.

In terms of priority perhaps the objectives (v) and (vi) come last.

On the basis of the above, the seven corporate objectives may be categorised as follows :

(1) Primary objectives : (i) and (iii).

(2) Subsidiary objectives : (ii), (iv), (v) and (vi).

(3) Self-imposed restriction or constraining objective : (vii).

To check the appropriateness of the above categorization, a final criterion may be employed — the criterion of public image. If objective (i) is not satisfied, the public image of the company is adversely affected. With objective (iii) unsatisfied, it is likely to be equally adversely affected. So long as objectives (i) and (iii) are satisfied, any strong public criticism against low productivity, ineffective promotion of R & D, inadequate training, or ineffective marketing — concerning objectives (ii), (v), (vi) & (iv) respectively, is unlikely.

As regards objective (vii), public criticisms of neglect of employees' welfare or unhealthy industrial relations are also less likely if objectivves (i) and (iii) are satisfied. On the other hand, public resentment against growing employees' welfare expenditure or higher wages for better industrial relations is certain if objectives (i) and (iii) are not satisfied.

The analysis so far has been directed primarily to an examination of the nature of the corporate objectives. For the purpose of CP however, more detailed understanding of the inter-relationships among the objectives is essential.

When dealing with the primary objectives, conceptually there are two different ways of formulating a Corporate Plan.

First, each of the objectives may be given a weight and the sum of the weighted value of the objectives may be sought to be maximised. Second, one of the objectives may be sought to be maximised while considering the others as binding constraints. Here, the objective (vii) relating employees' welfare and industrial relations has to be treated as a binding constraint. Of the two primary objectives (i) &

(iii), either could be treated as a binding constraint. Thus, one may maximise the satisfaction of objective (i), subject to the constraint of generating a specified surplus, or one may think of maximising the surplus subject to satisfying the country's coal requirements up to a specified level.

But there are two difficulties to adopt the second formulation. In the first place, a public sector undertaking like CIL can only have a target of meeting the entire requirements of the economy irrespective of the expected contributions from other coal producers, so far as objective (i) is concerned. Secondly, as the linkages of existing as well as prospective coal consumers are also influenced by considerations external to CIL, maximisation of the surplus is subject to the limitations imposed by linkages of production centres to consumers. This is not to suggest that maximisation of the surplus subject to targets on objectives (i) and (iii), is not possible. In the short run, the more convenient formulation would be one of maximisation of the satisfaction of objective (i), (i.e. meeting the requirements of the country) subject to generating a specified surplus say, even a non-negative surplus, and subject to improving employees' welfare and industrial relations to some specified extent.

Whatever be the formulation of a plan, the interrelations among the three objectives — two primary and one self-imposed restriction, have to be analysed.

Improvement of employees' welfare and healthier industrial relations may help maintain continuity in production and thereby help attain objective (i). They may also help generate greater surplus through maintenance of and/or improvement in productivity levels. But so long as they are dependent on higher welfare expenditure and periodic upward revision in wages and benefits to employees, they may have a negative effect on the generation of surplus. Before drawing up a corporate plan therefore, the exact cost implications of (vii) on the objectives (i) and (iii) need to be worked out.

So far as the interrelations between the primary objectives are concerned, normally one would expect that the greater production implied by objective (i) would lead to a greater generation of surplus. Moreover, a quick rise in production over the years will call for a fast changeover to improved technologies of production as well as deployment of more sophisticated and efficient plant and equipment. This in turn, is likely to contribute towards lowering the cost of production and hence generation of a greater surplus. But in the case of the coal industry, in the present environment, there are at least two forces which may make objectives (i) and (iii) conflicting in nature. Firstly, in the case of the tied-up future consumers, economically less attractive linkages at the current coal prices and existing technology would imply forgoing of more attractive opportunity, and the producer have to bear the consequent adverse impact on surplus generation. Secondly, with the present state of mechanisation and level of technology a fast changeover to higher technologies and a fast rate of mechanisation may not be associated with commensurate economies in cost of production in the short run. The work force has to be adequately trained before the full benefits of mechanisation and technology improvement are obtained. The period of learning through trials may turn out to be relatively long. In other words, in the short run such mechanisation and technology improvements may lead to a higher rather than a lower cost of production. Thus, before drawing up appropriate long-term corporate plans, it is necessary to work out the effects on the cost as well as the growth rate of production over different periods of time, associated with different possible rates of mechanisation and technological advancement that may be considered for implementation.

There is yet another area where the interrelation between objectives (i) and (iii) may be conflicting. The capacity expansion of the company has to match the growth in the country's requirements of coal. The capacity expansion projects in the coal industry are generally characterised by relatively long gestation periods (6 to 10 years). Moreover, with the introduction of mechanisation and advanced technology, the optimum economic size of the projects is increasing. All this implies (a) a capacity expansion in future will have to be in relatively large discrete jumps and (b) capacity expansion projects have to be undertaken much ahead of the growth of demand. Therefore, unless the demand forecasts are reasonably accurate, there is likely to be either a shortage or an excess of capacity. Shortage of capacity in relation to demand means that objective (v) remains unsatisfied. Carrying an excess capacity has an obvious adverse impact on objective (iii), i.e. surplus generation. The forecasting of demand in the past in the coal industry has seldom been accurate. On many occasions the demand forecasts were scaled down rather than up. This would produce an adverse impact on the capacity expansion projects. On the other hand, slippages or delays in the construction of projects have an adverse effect on the cost of production. If the construction is not delayed or if slippages do not occur, there would be an excess capacity. The probable effects of such conflicts between objective (i) and (iii) must be worked out before a long - range corporate plan can be drawn up.

So far as objective (iii) of surplus generation is concerned, it is necessary to look at it not only from the cost side but also the earnings side. The probable sources of income, the possiblity of increasing the income either through increased sales or increased prices, and even the scope for diversification need to be examined in detail. Although prices are officially controlled by the Government and the experience has been one of strong resistance to coal price rise from different quarters on various grounds, the coal prices have been increased. If one goes by the behaviour of pithead coal prices during 1974 to 1979, it will be noticed that genuine cost increases were allowed, even if after much delay, to be reflected in the pithead prices. It is necessary that before drawing up the long-term corporate plans, an estimate, of the possible rate of price increase is made. Past experience in regard to coal price behaviour in conjunction with forecasts of likely escalation of different cost components may be used with advantage.

As regards increased volume of coal sales, it will be ultimately limited by the country's requirements of coal. In any case, an estimate of the difference between sales revenue and cost of production will be necessary for planning. If it indicates that there will continue to be a gap between income and expenditure, the scope for closing it through additional income from possible diversification schemes, may have to be considered. The issue of diversification is related to, among others, two of the subsidiary objectives, (iv) and (v). It is necessary to have an assessment of the results of R&D efforts and the scope for marketing of coal-based products from low-temperature-carbonisation (LTC), processed solid fuels, etc. The diversification, however, need not be limited to coal utilisation projects; manufacture of inputs required by coal industry (like explosives, power, construction materials) can also contribute to the surpluses of the company. Again, an examination of the assets (both physical and human) of the company may reveal opportunities for diversification into different types of consultancy services, computer services, etc. It must be emphasised here that the issue of diversification and scope for different diversification schemes would be seriously considered if the company had laid down a specific target in regard to objective (iii). This is bound to be difficult to achieve while remaining in the existing business of producing coal only.

The subsidiary objectives are, in a sense, the means to the attainment of the primary objectives. The long-term corporate plan can only be drawn up if a detailed assessment of the extent to which the company can depend on these means, is made. For example, in respect of objective (iv) regarding marketing in the given environment of sponsorship by Government authorities and the Railways' role in programming allotment of wagons for coal transport, the extent to which marketing arrangements on the part of coal producers can be effective, has to be realistically assessed. Similarly, in respect of the coal of lower grades which are relatively plenty, the issue of decontrol of its distribution has to be examined. In case of such decontrol the consequent demand on the company's marketing efforts has to be assessed. With regard to objective (v) on R&D, it is necessary to review the adequacy of the flow of funds and scientific talents. The pertinent questions are (a) whether the allocation of funds for R&D in coal sector vis-a-vis other sectors has been adequate and (b) whether the coal sector could attract scientific talents at the desired rate. Based on these reviews, it is also necessary to work out the implications of alternative programmes of R&D on diversification schemes as well as the growth of coal demand.

Similarly, the cost implications of the alternate programmes of training and development have to be estimated. Forecasting is also necessary in respect of the impact of such programmes on the ability of the company to raise production and productivity. It is necesary to know the likely costs and benefits of the programmes and actions concerning training and development, introduction of improved technology, improvements in organisation and management, and imparting greater motivation. In other words, it is necessary to work out the interrelations among subsidiary objectives (ii) and (iv) and the primary objectives (i) and (iii). There could be different feasible rates at which training could be imparted and improved technology adopted. It is also noted that it may take some time before the full benefit of such training and mechanisation could be reaped. It is necessary to forecast the period that will be needed before a real break-through in this regard may be expected.

In conclusion, it may be pointed out that the exercise in analysing the nature of objectives and inter-relations among them, undertaken in this paper has been only illustrative. Nevertheless, the purpose would have been served if the illustrations were able to show the relevance of the analysis to the process of corporate planning. In particular, the analysis brings up the conflicts among objectives, which have to be resolved. But how the results of such an analysis are utilised in the formulation of the corporate plan, and how the conflicts among objectives are resolved through decisions taken at the highest levels of a corporate organisation, form part of a distinctly separate stage in the corporate planning process.

an investment analysis for a high-capacity underground project

T P Basu

Introduction

After the nationalisation of the coal industry in 1973, more than a hundred investment proposals, big and small, for coal projects, both reconstruction and new, have been sanctioned. The caption capacity of all such projects will be 115 million tonnes per annum with a projected capital investment of about Rs 1505 crores. In most cases, it has been found, that the gap between the projected capital and the actual expenditure is widening. The reasons for this are many, and it is not intended to go into their details, in this paper. However, it may be worth noting that even in identical conditions, the specific investment differs from project to project. This normally should not occur, if the projects are formulated during the same time period; though admittedly, the specific investment will vary from year to year as it is dependent on escalation of the prices and the capital cost index.

An attempt has been made in this paper to present briefly an overall view of planning a high capacity project, and a methodology of estimating the capital investment involved therein.

Major Areas of Investment and the Percentage Distribution of Costs

The following capital cost elements constitute the total capital of a project. An analysis of the feasibility reports prepared from 1971 to 1976 shows that their percentage distribution has more or less remained within a certain range.

Cost elements	Percentage distribution
Land	2.94%
Buildings	13.15%
Plant and machinery (P&M)	70.00%
Furniture and fittings	0.21%
Railway sidings	4.50%
Vehicles	0.12%
Development capital outlay and capitalised revenue combined	7.75%
Prospecting and boring	1.33%

Addl. CME, CMPDI, Ranchi

It may be seen from the above that the principal elements of cost items are P&M, buildings, development, capital outlay and capitalised revenue combined. These three items constitute more than 90% of the total capital. Analysis of the current feasibility reports shows that there is no appreciable difference in the percentage distribution of the above capital items except in buildings where it has been lower which is possibly, due to improved technology in building structures.

It has been observed that reconstruction projects formulated since 1971 show a lower capital cost to the extent of 25-34% depending upon the type of the particular project in comparison with that of a new mine.

Planning Strategy for a High—Capacity Underground Mine

The present day planning for underground mines has taken a distinct shift towards application of mechanised longwall from the previous practice of bord and pillar method. Demand for larger output of coal has necessitated the projection of larger units. The bord and pillar method with its inherent constraints would not be able to meet this desired demand for production. It has become imperative for a larger underground mine to have the maximum concentration of working places so that for a given production level, manpower employed is the minimum. This is only possible from mechanised longwall faces, as manpower cannot replace machines.

For the estimation of capital investment, some basic conditions are assumed in the following example which are commonly met with while planning for different coal deposits in our country.

— Area of the mine — 4-5 km² with about 3 kms along strike and 1.5 km along dip.

— Presence of multi-seams in succession with a thickness range between 1.5 m and 3.5 m.

- Reasonable level of availability of infrastructures in the form of roads, power lines, railways, for which no extra high capital to be borne by the project.
- Gradient 6° - 8° .
- Surface free and cavable.
- Total reserves in the above thickness range 10-15 million tonnes./seam

These conditions are mostly met in the major coalfields like Raniganj, Jharia, Central and other coal fields. The mine is planned from the grassroots level for a production of 4200 tonnes per day with a maximum attainable capacity of 5300 tonnes per day. Two depth ranges of the incrop are assumed viz., 100 and 300 m.

Under normal geological conditions of such a 'mine take', the above production should be available by working one seam at a time, if there are no quality and reserve constraints; otherwise, two or more seams will have to be worked at the same time. In the latter case, the capital expenditure will tend to be more for duplication of trunk transport equipment and also arrangement for skip loading (with one or two pit-bottom loading points depending on the parting between the seams).

The production will be available from three powered support longwall faces each being 120-150 m long (depending on the conditions prevalent) and main dip development. The trunk roads will be located as far as possible along the central axis of the mine so as to balance the length of the longwall panels on either side of the trunk road. The faces will be retreating from the boundary towards the central trunk road.

From the experience with the British and German mines, the minimum rate of drivage of gate roads for any longwall panel is estimated to be 3-4 times the rate of face advance. Assuming a face progress of 6-8 shears per day, i.e. 3.6-4.8 m per day, progress of the gate roads will be about 12-19 m per day. Regarding drivage, a cautious choice has to be made between the single-heading and the double-heading with chain pillars. In the British and continental practice single-heading drivage is done employing very high water-gauge auxiliary fans for ventilation. Presently these fans are not available in India, and hence there is a tendency among the planners to go in for the double-heading development. There is no reason why the high water-gauge fans cannot be imported alongwith the powered support equipment. For the purpose of planning, single-heading drivage is adopted here. It is proposed to use two road headers with necessary back-ups including bridge conveyor and other equipment for each panel.

Planning for the Access to the Mine

The two depth ranges of incrop considered are, Case-I-100 m and Case-II-300 m.

In Case-I, assuming a gradient of 6°-8°, the maximum depth at the dipmost region of the mine-take will be about 300 m and in Case-II, about 500 m.

In Case-I, entry will be by an incline and one shaft. To ensure the minimum sterilisation of coal, the coal may be touched in this case at the minimum depth possible, say about 125 m. There may be two such inclines serving as intakes, and the shaft as the return. With a gradient of 1 in 4, the length of each drift will be about 500 m with an average clear cross-section of about 15 m². The excavation size will be about 19 m². This will allow about 200 m³ of air per second. One incline will be exclusively for ventilation, the other will house the conveyor and one haulage track. The shaft will be of 6.5 m dia fitted with a cage winder for men and material.

In Case-II, the entries will be through shafts only the depth of each being about 400 m with diameters 7 m and 6.5 m respectively. The larger-diameter shaft will be fitted with a 10 t skip and automatic loading and unloading arrangements. The other shaft will have cage winding for men and material transport.

Underground Transport Modules

The each longwall power support face, heavy duty AFC of 450 tonnes capacity will deliver through a stage loader to a 900 mm wide gate-road belt conveyor.

In the gate-road drivage, a road header will deliver the coal to an outbye belt conveyor 800 mm wide, through the bridge conveyor.

The conveyors at the power support face and the back-up conveyors in the gate roads as well as those for the road-headers should have sufficient capacity to take a peak load of 450 tonnes per hour so that the high-cost face machinery work at the maximum efficiency.

The 1200 mm heavy-duty trunk conveyor will have steel-cord belting receiving load from either side. Two gate belts will have two bunker conveyors of about 50-100 tonnes capacity at the inbye-end. The bunker conveyors will be of reversible type with hydraulic motor drivage having provision for automatic stoppage when full.

The same steel.cord trunk conveyor system will be extended to raise coal through the drift in Case-I, with a continuity of transport right from the face to the surface CHP.

Production Assessment

(a) Longwall Face

Maximum production with 6 shears	— 1260 tpd
Maximum production with 8 shears	— 2260 tpd
Average production taking 50 days for shifting, i. e. 250 days of actual mining per year and with about 90% efficiency from three faces	— 3600 tpd
(b) Gate road drivage from three panels @ 185 tonnes from each panel	— 550 tpd
(c) Main dip drivage conventional	— 100 tpd
Total	— 4250 tpd
say 4200 tpd	
Annual average	1.26 million tonnes
Maximum attainable	1.6 million tonnes

Investment Analysis — Broad Break-up

As mentioned before, the main items for capital expenditure are plant and equipment, civil structures and buildings, development including revenue expenditure during project management, railway siding, land, prospecting and boring. These items will now be analysed one by one on the basis of the planning strategy outlined above. The costs are by and large based on the 1979 level (Appendix I).

Plant and Machinery : These are further divided into the following broad sub-heads showing the requirement of capital against each. The basis of the price taken is the latest tender quotations or purchase.

A general breakup of the main items of plant and machinery is given in Appendix II. Details of prices of individual equipment is beyond the scope of this paper.

Building and Civil Structures : Under this head, fall service buildings, buildings for welfare amenities and residential housing colony.

For the size of the underground project outlined a total number of about 13-15 service buildings are to be constructed. Depending on their specified plinth area on duty demand, the total capital for construction of these buildings will be about Rs 32 lakhs, the estimate being based on the prevailing cost index at the industrial belt in Asansol, Jharia and other coalfields of Bihar. Taking the building cost index as 100 base in Delhi on 1.10.76, the same is now 180 for Asansol and Jharia. This may be slightly less for outlying fields especially in MP and Orissa (144 in the Ib area). The common types of service buildings are office, stores, workshop, sub-station, vocational training centre, vehicle sheds (garages and cycle sheds), winding engine rooms, lamp room, fan house, haulage engine room, pit office and explosives magazine.

The buildings for welfare amenities normally constitute 16-19 units for which total capital will be about Rs 21 lakhs. These buildings are — pit-head baths, canteen, hospital and dispensary, workers' institute, club, first-aid centre, school, post office and bank, rest house, shopping centre, co-operative store, rest-shelter, wash rooms etc.

The number of residential buildings and the size of the colony will depend on the manpower employed in the project. For a project of this size and degree of mechanisation, an overall OMS of about 2 tonnes may be expected to be achieved. Thus, with a daily production of 4200 tonnes, the working strength will be 2100 and taking a 18% absenteeism due to various reasons (including leave, sickness etc.) the total manpower on roll will be about 2478. The total number of residential buildings to accommodate this population with 55% satisfaction will be about 1363. For optimum economy, without sacrificing the living comfort of the workers, the type-mix of the quarters will constitute about 60% miners' quarters and hostel accommodation, 22% type 'A', 15% type 'B', 2% type 'C' and the rest 1%, 'D' type quarters. The present-day cost of a miner's quarter is about Rs 16,800 and that of the 'A', 'B', 'C' and 'D' types about Rs 23,500, Rs 36,000, Rs 50,000 and Rs 1 lakh each respectively. With the above cost of the individual types, the average cost of each of the type-mix quarters will work out at about Rs 22,450. Thus, the capital required for residential buildings will amount to Rs 306 lakhs. Another 5% should be added to this sum for designing a modern colony with parks, playgrounds, sewerage etc., and the total cost for a township will be approximately Rs 321 lakhs. About 10% reduction may be effected if double/triple storeyed buildings are constructed and in that case, the capital required will be Rs 289 lakhs.

Prospecting and Boring : According to the Indian Standard Specifications about 4-5 boreholes per km^2 are required for regional proving. For detailed proving, 6 boreholes per km^2, or 27 boreholes in all will be necessary for this mine-take of 4.5 km^2 area. In addition, about 4-5 boreholes may have to be drilled as 'probe' holes at the shaft or the incline site for precise delineation of faults and other disturbances, if any.

In Case-I, the depth of the deposit being 100-300 m, each borehole will have an average depth of about 200 m. This will fix the total meterage of drilling for 36 boreholes at about 7,200 m. In Case-II, the depth of the deposit being 300-500 m total meterage will work-out to be 14,400 approximately.

The present drilling cost is about Rs 350 per metre, according to which the capital required for drilling will workout to be Rs 26 lakhs in Case-I and Rs 52 lakhs in Case-II.

Siding : For loading 4200 tonnes of coal per day, two loading points will be needed, assuming the conventional six-hour mechanised loading from the CHP. Theoretically, one loading point should be sufficient for a daily loading of two super-rakes. However, the new project is likely to be loacted comparatively far away from the existing pilot yards, and considering the present constraints in movement of the pilots, two loading points are envisaged. It is assumed that at a time one super-rake (44 box wagons) will be loaded.

The siding for the off-take of the above production, will have a total length of about 1700 m for each loading line. There will be two loading lines and one escape line for the engine. The minimum length of the track for the escape line will be about 1100 m. This length is based on the stipulation by the railways that a clear standing length (CSL) of a super-rake should not be less than 690 m and the length of the shunt not less than 72 m.

The length of the assisted siding between the takeoff point at the main line and the colliery takeoff at the siding has to be considered also, as the railways often refuse to bear this expenditure. As this length will entirely depend on the location of the project, a length of 2 km is taken here as a general estimate. Thus, the total length of the track works out at about 7 km. The present-day cost estimate for each km of track in a moderately level terrain is about Rs 20 lakhs, out of which material cost excluding carriage etc. is about 25%. Thus, the total capital to be provided for siding is about Rs 140 lakhs.

Land : Normally, land does not constitute a major share in the total investment, but is separately dealt with here as this is one of the critical items for starting a project. Acquisition of land has become a problem with new dimensions, especially in Raniganj, Jharia and some other coal belts. The planners should clearly indicate the area of land required for the project.

For an underground project, land is required for entries to the mine, construction of service buildings, explosives magazine, civil structures, welfare amenities, township, railway siding and roads in the project. At the entry site a clear space of about 300 m on all sides of the opening should be available. For the required number of service buildings (excluding explosives magazine) about 7 hectares, and for the township to accommodate about 1368 quarters, 55 hectares of land will be required. For the explosives magazine 28 hectares of land is necessary to comply with the Indian Explosives Regulations. For the railway siding about 20 hectares of land should be provided.

Thus, the total surface area of land will be about 140 hectares, and at the prevailing rate of about Rs 16,000 per hectare, the total capital to be provided with, is about Rs 22 lakhs.

Development of Project

This will comprise sinking and drifting, arrangement of water supply, construction of roads and culverts and project construction cost.

Capital Outlay : This will be different for shallow and deep mines and estimated separately below :

In Case-I (Shallow Mine)

Two drifts of each 500 m length and a shaft of 200 m depth will be required	
Cost of drivage of a total of 1000 m @ Rs 10,000/m	Rs 100 lakhs
Cost of sinking a 200 m deep shaft 6.5 m dia., with shaft-collar, inset etc. @ Rs 60,000/m	Rs 120 lakhs
Total	Rs 220 lakhs

In Case-II (Deep Mine)

Cost of sinking a 400 m deep shaft-7 m dia with shaft-collar, inset etc. suitable for installation and operation of 10 te skip to deal the entire output of 4200 tpd @ Rs 70,000/m	Rs 280 lakhs
Cost of sinking a 400 m deep shaft 6.5 m dia with shaft-collar, inset etc. for men and material winding Rs 60,000/m	Rs 240 lakhs
Total	Rs 520 lakhs

Roads and Culverts : For a 4.5 km^2 area of the project, roads of a total length about 9 km have to be constructed, out of which a length of about 3 km connecting the office, store, workshop, pits etc. should be built for heavy vehicles and of Grade 'A'. The cost for this type of road is Rs 1.75 lakhs/km and that for the ordinary type Rs 65,000/km approximately.

The length of the approach road from the main highway will depend on the actual location of the project. However, a token length of 2 km is taken here for the purpose of calculation. The rate of construction of such roads is Rs 1.40 lakhs/km approximately. The cost for construction of the roads only will be about Rs 10 lakhs.

Additionally, about 60% of this cost is provided for culverts, drains, drain-crossings etc.

Thus, the total capital required for roads and culverts will be about Rs 16 lakhs.

Water Supply : The basis of estimation of total water requirement and the capital thereof is the total manpower employed and the size of the housing colony and township in the project. This will also depend on the distance of the source of water.

According to a broad estimate based on the provisions under the ISS (135 litres/day/person) total potable water demand including process and storage loss will be about 1.16 million litres per day.

At the rate of Rs 300 (actual and notional), the estimated requirement of capital will be about Rs 26 lakhs.

Project Management

The expenditure on project management is a very important factor in the total capital investment of the project, and is incurred during the construction phase of the project till it is brought upto the revenue-earning phase. This period may constitute 6-8 years for an underground deep shaft mine of the project size under discussion. The delay in completion may permanently damage the economic viability of the project throughout its entire run. All the essential development works involving shaft-sinking and drifting, power supply arrangement, construction of essential service buildings, 40-50% of residential housing, siding and CHP, main approach roads, a major portion of the water supply arrangement etc. should be completed within this period. Installation and commissioning of winders, main fan, construction of underground sump and main drainage system, and drivage of main underground roadways should also be completed during this period alongwith the procurement of most of the P&M items.

The specific operations incurring cash flow are administration and supervision of the above construction works, detailed engineering documentation and drawing specifications, and ordering and inspection of the P&M items. The backup expenditure includes that for temporary housing,water supply, power and stores.

About 70-80% of the development capital is spent during the period, and the cost of the capital, (i.e., interest charges,) constitutes a major component. The development capital generally comprises 10.5% of the loan capital, i.e. the total excluding the capital for residential housing and welfare amenities.

It is not meant to show here the details of estimation of the development capital, but with a normal level of operational efficiency, this capital for a new project will constitute about 6% for a shallow mine and 8% for a deep mine of the total project capital.

However, the implementing authority has to proceed with utmost caution during the development phase. Any delay beyond the estimated time-frame will severely upset the entire investment pattern. This delay is caused not only due to inefficient management, but due to inadequate resource-planning. Planners should focus their attention on five basic resources viz., water, energy, land, material and manpower (called the WELMM approach). Substantial cash flow could only be released after tying up availability of each of the five resources.

Investment Rating

Based on the above analysis, a broad breakup of the total capital requirements for a shallow mine and a deep mine is shown in the Appendix I. Cost for the plant and equipment excluding those for hoisting will be almost the same for the two types of mines. Areas of saving on a shallow mine are coal-winding, capital outlay for development, project management (due to a shorter gestation period) and prospecting and boring. There will also be a marginal saving in service building, civil structures and pumps.

Conclusion

The above is an attempt towards a realistic analysis of capital requirement for a fairly large-sized underground project based on the 1979 price level. Though this will presumably escalate in future years in actual quantitative terms, the percentage distribution of different items is expected to remain fairly constant. Assessment is done here taking an idealised case, and there is bound to be variations from project to project depending on the resource availability, the geographical location and geological set-up of the project but variation is not likely to exceed a $\pm 10\%$ range.

It may be seen that saving in capital in case of a shallow mine than a deep mine for the same output level is quite substantial, of the order of about Rs 5.3 crores. This is mainly on account of elimination of sinking and vertical transport. Barring those in the Jharia field and the western part of Raniganj field, most of the new coal projects will fall under the shallow mine category. As the coal hoisting through shaft is becoming very capital-intensive, one has to take a very cautious appraisal before suggesting skip hoisting for an underground project with a depth of about 200 m or less.

APPENDIX I

Case — I SHALLOW MINE — Depth Range — 100-300 m

Case — II DEEP SHAFT MINE — Depth Range — 300-500 m

Rated Production 4200 tpd/1.20 Million Tonne Per Year

Items of expenditure (828)	Case I: Total capital in Rs lakhs	Case I: % of the total capital	Case I: Specific investment Rs per tonne of annual rated production	Case II: Total capital in Rs lakhs	Case II: % of the total capital	Case II: Specific investment Rs per tonne of annual rated production
1	2	3	4	2	3	4
1. Land 1·8	22	0.57	1.74 1·7	22	0.50	1.75 1·7
2. Buildings : 37	340	8.85	26.98 27	342	7.82	27.14 28
(a) Service	30			32		
(b) Welfare	21			21		
(c) Residential	289			289		
3. Plant and machinery 140	2821	73.48	223.88 249	2925	66.96	232.14 251
4 Furniture and fittings ·50	5	0.19	0.40 ·50	5	0.11	0.40 ·5
5. Railway siding 20·00	140	3.65	11.11 17	140	3.20	11.11 17
6. Vehicles 2·00	18	0.47	1.43 2	18	0.38	1.43 2
7. Prospecting and boring 4·00	26	0.68	2.07 2·5	52	1.12	4.13 4·5
8. Development and others 33	467	12.16	37.00 39	864	19.78	68.57 81
(a) Capital outlay	220			520		
(b) Roads and culverts	16			16		
(c) Water supply	26			26		
(d) Project Management	175			266		
(e) FR preparation and documentation	30			36		
Total 238·00	3839		~~305~~ 338	4368		~~346~~ 392

APPENDIX II

LIST OF MAJOR P&M ITEMS FOR A NEW MINE PRODUCING 4200 TPD

		Rate/ Nos.	Total Amount in Rs lakhs
1.	Equipment for 150 m long power support longwall face with about 1.5-2.3 m height range comprising 138 numbers 6 x 240/ 2 x 280 t rigid base power support with 4 Nos. of anchor chocks including the following :	3 sets Rs 531 lakhs each	1593
(a)	DE shearer with all electricals ;		
(b)	Power pack with all electricals and other ancillaries ;		
(c)	400 Nos. of hydraulic props for gate road support ;		
(d)	2 x 900 metres long, 900 mm belt conveyor with 450 t Ph peak capacity complete with all electricals and PVC belting ;		
(e)	Heavy duty AFC-2 x 90 kw-450 tph peak capacity ;		
(f)	Heavy duty stage loader of equivalent capacity ;		
(g)	Endless haulage and material trolley, winch etc. for material transport ;		
(h)	50-100 t bunker conveyor ; and		
(i)	All other ancillaries and spare parts for 2 years.		
2.	Equipment for main dip drivage with haulage, rails, tubs etc.	One unit Rs 6 lakhs	56
3.	Equipment for drivage of gate roads for longwall panel consisting of two road-headers, bridge conveyors, gate conveyor 1500 metres long X 800 mm wide with belting and other machinery.	3 sets Rs. 145 lakhs	435
4.	Trunk transport for both coal and material comprising 1000 metres X 1200 mm steelcord belt conveyor, material haulage, manriding haulage and 500 metres $\times$1200 mm conveyor for level transport.		100
5.	For deep shaft mine — pumping and drainage consisting of 5 Nos. of main pumps, 1000 metres for heavy duty pipe range, intermediate and face pumps and pipelines.		24
	For shallow mine similar equipment		19
6.	Ventilation equipment consisting of :		
	2 Nos. of main ventilating fans of 250 m^3 capacity per second with stand-bye motor, high watergauge auxiliary fans, ducting etc.		40
7.	For deep shaft mine — 2 Nos. winders (one with 10 t skip and the other with cage), head frame, pit top and pit bottom arrangements.		220
	For shallow mine — 1200 mm steel cord belt conveyor, material haulage, men & material transport equipment etc.		121
8.	Surface Coal Handling Plant to deal with 1.2-1.5 million tonnes of coal per annum with conventional loading arrangement onto super rakes.		200
9.	Standard unit work shop		10
10.	Surface and u/g communication		12
11.	Electricals — 10% of the above P & M cost		190
12.	Contingency 5%		95
Total	For deep shaft mine	Rs. 2925 lakhs	
Total	for shallow m.ne	Rs. 2821 lakhs	

selection of self-advancing supports for longwall faces

A G Watwe

Introduction

To improve the productivity in the mines with a greater concentration of production and to achieve a high percentage extraction, Indian coal mining industry is progressively increasing the application of the longwall system. To attain optimum production and productivity levels commensurate with safety at the longwall face, the emphasis is on employment of self-advancing powered supports. Self-advancing supports involve considerable capital investment per face (about Rs 2 to 3 crores for a face length of 150 m) and, therefore, it is necessary to have a proper insight into the strata behaviour associated with the longwall system and the factors governing the choice of supports. The characteristics of various types of supports also need careful consideration. This paper discusses in general the behaviour of strata at a longwall face and the basis for selection of appropriate supports.

Strata Behaviour at a Longwall Face

From the studies made in Europe it has generally been observed that an increase of the normal stress in the strata can first be detected about 100 m ahead of the longwall face. The increase in the stress level is quite rapid in the region lying between 15 to 20 m and the peak stress is attained in the region lying between 1 to 4 m ahead of the face (Fig. 1).

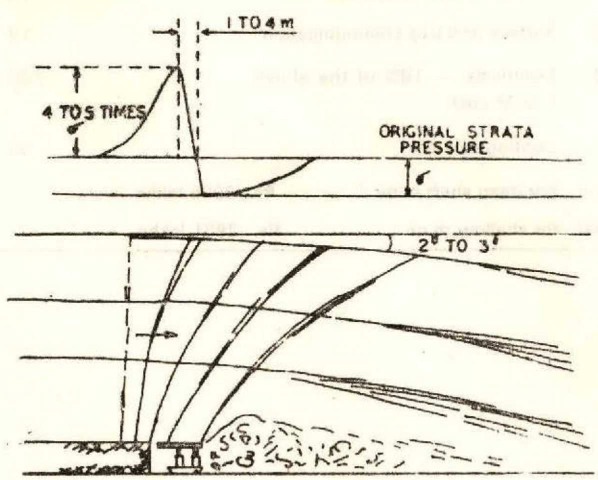


Fig. 1

Addl. C.M.E., CMPDI, Ranchi

These distances are only indicative and would vary according to the actual composition and properties of the strata. The peak stress might attain a value as high as 5 times that of the normal stress existing before starting the mining operations. In the zone of peak stress the immediate roof may begin to fail, and this results in a rapid fall in the intensity of stress on the immediate roof which has yielded. In the supported portion of longwall face the supports have to withstand only the weight of the immediate roof. Behind the supports where the main roof gradually settles on the broken debris of the immediate roof, the stress level slowly increases. In the region of the peak stress the immediate roof yields and begins to slope. The angle of the slope generally varies from 2° to 3°. Due to the transverse pressure arch across the longwall panel the peak stress is not uniform along the face. It is greater near the gate roads and decreases from the gate roads towards the middle of the face.

It is observed that the minimum convergence due to the slope of roof beds cannot be avoided even with the strongest supports. Generally,the convergence is found to be of the order of 30 to 40 mm per metre of advance of the face. However, if the thrust provided by the supports is too low, it may lead to excessive bed separation and a much greater convergence. Presence of a strong and massive roof will lead to irregular convergence, though the average rate of convergence over a few weeks, would be of the same order. Convergence is also not uniform along the face. Convergence is less in the neighbourhood of the gate roads, and is more towards middle of the face. If the face advance is fast the amount of convergence is expected to be less as it would not have enough time to take place. If the immediate roof and the floor are weak the supports may bite into them and thereby increase the convergence.

The height to which caving is likely to extend will depend upon the bulking factor associated with the immediate roof. In UK it is generally assumed that when the immediate roof breaks, its volume increases by 50%. On this basis it is assessed that height upto which caving might extend could be upto 4 times the thickness of extraction when measured from the floor of the face. As the supports at the face are to deal with the immediate roof, the support resistance will depend on the height of caving which is related to the bulking factor. Cavability is also largely influenced by the existence of joints, fissures, cleavage planes etc. and their direction in relation to the face.

At the face, the immediate roof converges, and simultaneously moves laterally towards the goaf. The frictional resistance between the canopy of the supports and the strata causes the broken roof blocks to tilt towards the goaf.

Most of the observations on longwall faces in India relate to faces equipped with individual friction or hydraulic props. The face advance is slow compared to similarly equipped faces in Europe, and is at a far lower rate compared to faces equipped with self-advancing supports. It will, therefore, be difficult to correlate the Indian observations with the European observations.

Assessment of Support Requirement

There are a number of empirical methods for the assessment of the support requirements at the longwall face, some of which are discussed below.

Method Adopted by US Bureau of Mines

In this method it is assumed that the immediate roof acts like a cantilever beam and it breaks in front of the face at a distance equal to the seam height. The suggested method of calculation is given below (Fig. 2).

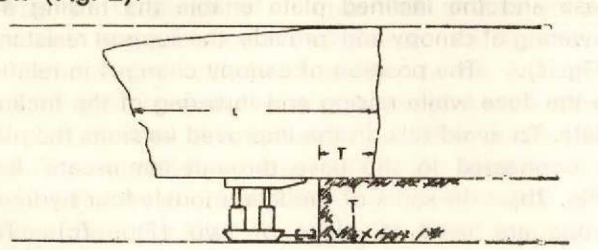


Fig. 2

Here, $P = L\,d\,i\,H$

when P = load to be supported

L = length of the beam

d = density of the rocks

i = support interval

and H = thickness of the beam

Wilson's Method

In this method it is assumed that the immediate roof breaks along the face either vertically or at an angle. The area of roof to be supported is assumed equal to the working area, and the caving height from the roof of longwall face is assumed equal to about twice the thickness of extraction. The requirement of the support resistance varies with the angle of caving as illustrated in Fig. 3.

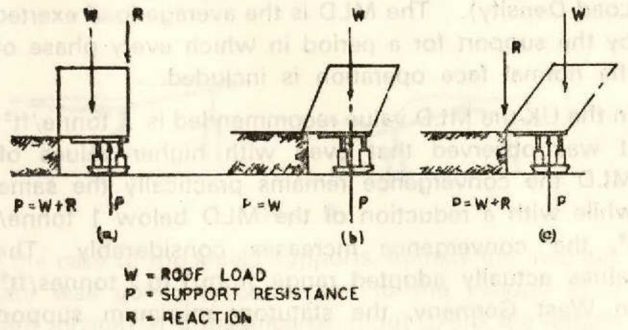
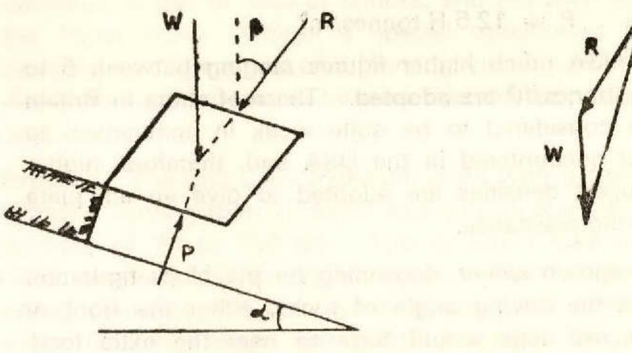


Figure 3

For equilibrium, summation of all vertical forces is equal to zero, and summation of moments at any point is also equal to zero. It is assumed that all the legs of the support are uniformly loaded and the resultant thrust of the support acts at a point equidistant from the front and rear legs. When the roof load and the support resistance are co-axial, the requirement of support resistance is minimum as in case (b) of Figure 3. In actual practice, as shown in case (a) the forward legs, while in case (c), the rear legs would be loaded more. The loads on the individual legs can be worked out on the basis of the principle of equilibrium.



$$P = W\left(\frac{\sin\alpha}{\tan\phi} + \cos\alpha\right)$$

Fig. 4.

If the seam is steeply dipping, the support resistance should be adequate to prevent lateral sliding of the beds. The requirement of support resistance for such a situation can be worked out as shown in Fig. 4. It has been mentioned previously that the load distribution depends upon the location of the canopy with respect to the broken roof block. Again, it is obvious that the load distribution will also depend on the position of the canopy with respect to the face. Thus, the load will vary during different activities at the face. The support requirement is, therefore, considered in relation to the MLD (Mean Load Density). The MLD is the average load exerted by the support for a period in which every phase of the normal face operation is included.

In the UK the MLD value recommended is 1 tonne/ft^2. It was observed that even with higher values of MLD the convergence remains practically the same while with a reduction of the MLD below 1 tonne/ft^2, the convergence increases considerably. The values actually adopted range from 1 to 2 tonnes/ft^2. In West Germany, the statutory minimum support capacity for the caving longwall faces in seams dipping at less than 18^0 is calculated according to the following relation.

$$P = 8\,H \text{ tonnes/m}^2$$

when P = Support capacity in tonnes/m^2 (.093 tonnes/ft^2)

H = Thickness of seam in metres

In actual practice, however, a higher load capacity of the order of 1.5 to 2 times the recommended value is adopted.

In France the load bearing capacity of supports is calculated on similar lines. The equation adopted for this purpose is as follows :

$$P = 12.5\,H \text{ tonnes/m}^3$$

In USA much higher figures ranging between 6 to 12 tonnes/ft^2 are adopted. The roof strata in Britain are considered to be quite weak in comparison to that encountered in the USA and, therefore, higher support densities are adopted to give an adequate caving resistance.

As shown earlier, depending on the block figuration and the caving angle of rocks, either the front or the rear legs would have to bear the extra load. Therefore, to have a universal application the supports should be chosen for an MLD value equal to twice the value considered necessary.

Types of Self-advancing Supports

Frame type : This consists of two or three hydraulic props attached to a common roof canopy and base. Its ability to move forward is independent of the conveyor. The frames are connected in pairs, and each frame is capable of advancing with the help of an anchorage provided by the other frames. Fig. 5

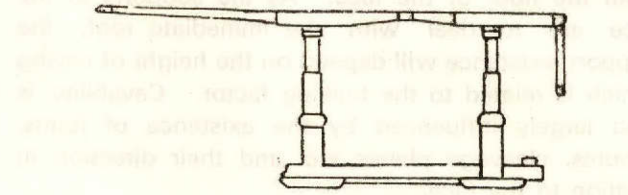


Fig. 5

Chocks : These consist of four to six hydraulic props with a common canopy and base (Fig. 6).

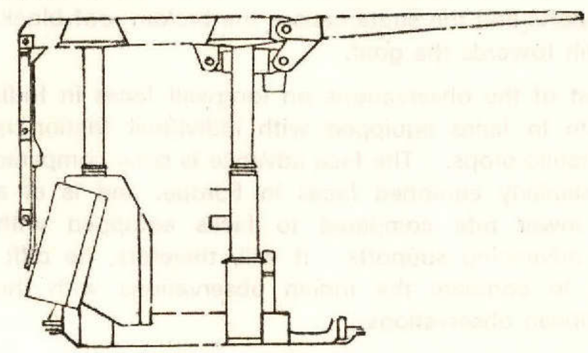


Fig. 6

The advance of the chocks is linked with the face conveyor movement as each chock is connected to the face conveyor through a double acting push/pull hydraulic cylinder.

Shields : These consist of an inclined steel plate having one end hinged to the base and the other end connected through a hinged joint to a roof canopy. Two hydraulic props acting between the base and the inclined plate enable the raising and lowering of canopy and provide the support resistance (Fig. 7). The position of canopy changes in relation to the face while raising and lowering of the inclined plate. To avoid this, in the improved versions the plate is connected to the base through lemniscate' links (Fig. 7b). In some of the later models four hydraulic props are used in place of two (Fig. 7c). This arrangement overcomes the problems of instability of the canopy. Moreover, as the plate in this case is connected to the rear end of the canopy the problem of loose debris accumulating in between the rear

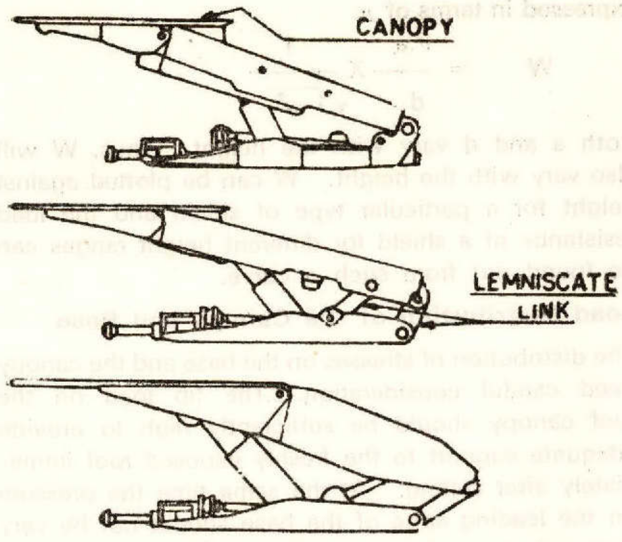


Fig. 7

end of canopy and the plate is also overcome. It is observed that the pressure on the loading edge of the base is quite high in the case of two-legged shield support.

Chock shields : This is a combination of chock and shield type supports to offer advantages of both types.

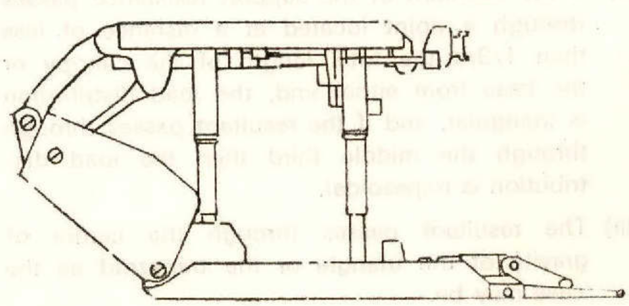


Fig. 8

Support Elements

Canopy : Canopies of frames and chocks may either be in one piece or in two pieces. In the case of a canopy the articulated joint may be located ahead of the forward leg of the support and the front part of canopy is activated by either a special capsule or by additional legs. With the location of joint in between the legs there are encountered, as is evident from the Fig. 9. In UK the use of supports with the articulated joint located in between the two rear rows of legs is no longer permitted. With the location of the joint ahead of the forelegs it becomes possible to provide a high tip load and keep the front portion of canopy in contact with the roof.

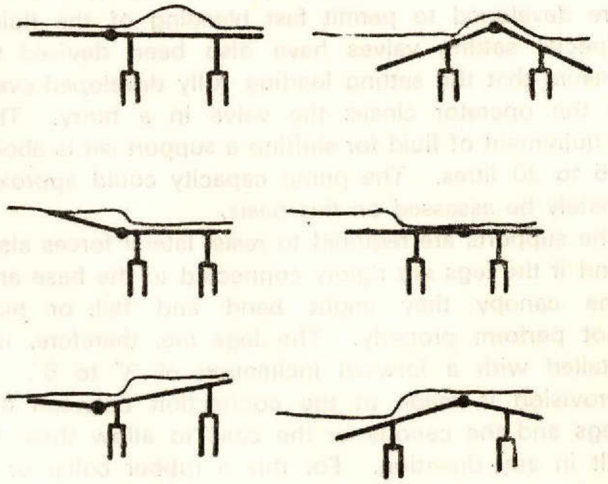


Fig. 9

In the case of the shield supports, initially the inclined plate was generally connected to the canopy at a point located at a distance of about 2/3rd the length of the canopy from the forward tip. Difficulties were faced in this arrangement due to roof debris coming in between the rear end of the canopy and the inclined member. Inspite of numerous modifications the problem was not completely overcome. In the later models, the connection between the inclined plate and the canopy is located at the rear end of the canopy. Further, four props are provided instead of two, and at least two of the props support the roof canopy directly.

In the case of shields, the roof is fully covered and no debris can fall either from the roof or through the inclined shield inside the supported space. The coverage is less in case of chocks, and the least for the frame types. Unless a special attachment is provided at the rear end of a frame or chock support, debris from goaf are liable to come into the supported area.

Props : The props are hydraulic jacks with a bore diameter varying from 10 to 30 cms and an operating pressure of 70 to 350 bar. The cylinders may be either single-acting or double-acting. With single-acting cylinders, the prop when released will retract under its own weight, and therefore takes a longer time to retract. For obtaining a greater range, extension pieces of an internal or external type, are provided. Double telescopic props have also been introduced with a special arrangement to ensure a uniform yield.

To withstand shock loads, rapid-yielding type valves are developed to permit fast bleeding of the fluid. Special setting valves have also been devised to ensure that the setting loading fully developed even if the operator closes the valve in a hurry. The requirement of fluid for shifting a support set is about 15 to 20 litres. The pump capacity could approximately be assessed on this basis.

The supports are required to resist lateral forces also, and if the legs are rigidly connected to the base and the canopy they might bend and fail, or may not perform properly. The legs are, therefore, installed with a forward inclination of 3° to 5°. A provision is made at the connection between the legs and the canopy or the base to allow them to tilt in any direction. For this a rubber collar or a spring-loaded system may be used permitting the legs to regain the original position when the lateral thrust is removed.

Support resistance does not vary with the height for the chock and frame supports but it varies with the height in case of shield supports (Fig. 10).

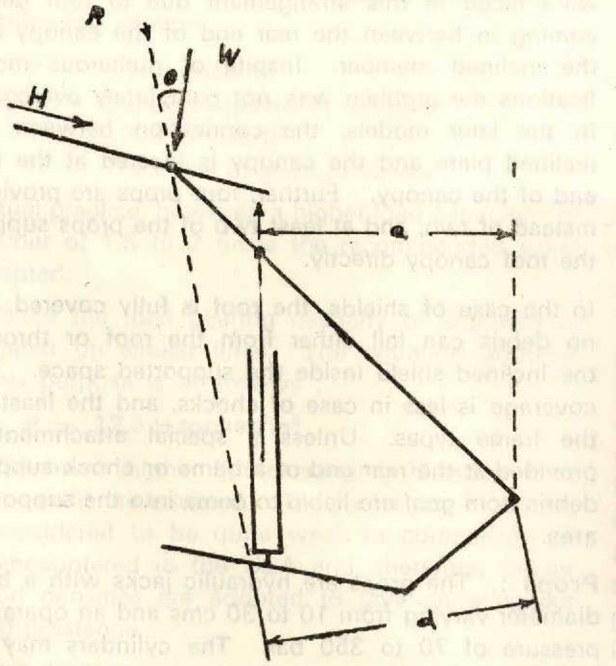


Fig. 10

Taking moments,

$$R.d = P.a$$

$$\mu = \tan \Theta \text{ (Coefficient of friction)}$$

$$W = R.\cos \Theta$$

$$W = \frac{P.a}{d} \cos \Theta$$

Expressed in terms of μ,

$$W = \frac{P.a}{d} \times \frac{1}{\sqrt{1-\mu^2}}$$

Both a and d vary with the height. Thus, W will also vary with the height. W can be plotted against height for a particular type of shield and the load resistance of a shield for different height ranges can be found out from such a curve.

Load Distribution of the Canopy and Base

The distribution of stresses on the base and the canopy need careful consideration. The tip load on the roof canopy should be sufficiently high to provide adequate support to the freshly exposed roof immediately after setting. At the same time the pressure on the leading edge of the base should not be very high, otherwise the edge will bite into the strata and cause difficulties in advancing the supports.

An accurate analysis of the stress distribution in the canopy and base of powered supports is very complicated. However, for practical purposes the stress distribution is worked out on the following assumptions :

(i) The stress distribution is linear.

(ii) If the resultant of the support resistance passes through a point located at a distance of less than 1/3rd the total length of the canopy or the base from either end, the load distribution is triangular, and if the resultant passes through through the middle third then the load distribution is trapezoidal.

(iii) The resultant passes through the centre of gravity of the triangle or the trapezoid as the case may be.

With these assumptions the stress distribution could be worked out as shown in Fig. 11.

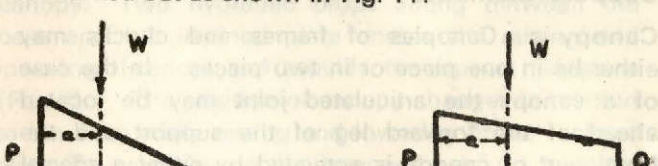


Fig. 11

Here, W = Resultant of support resistance
P = Maximum stress
Q = Minimum stress
L = Length

i = Support interval
a = distance of the CG of the area from the end subjected to maximum stress.

Triangular distribution	Trapezodial distribution
$W = 1/2\,L\,P\,i$	$W = \frac{P \times Q}{2}\,Li$
$a = \frac{L}{3}$	$a = \frac{L\,(P+2\,Q)}{3\,(P+Q)}$
$P = \frac{2\,W}{3\,ai}$	$P = \frac{2\,W\,(2L-3a)}{L^2\,i}$
	$Q = \frac{2\,W\,(3a-L)}{L^2\,i}$

Setting and Yield Loads

The setting load will depend on the pressure provided by the hydraulic pump, while the yield load represents the maximum permissible load for which the components are designed. Once the support is set at the setting load the support resistance will increase with the convergence till the yield load is reached.

Self-advancing supports are advanced at short intervals of about 0.6 m, and normally the convergence may be as low as 25 mm. Crushing of any loose materlal above the canopy and below the base may easily account for about half of such convergence, the other half being accommodated by the compression of the fluid. If a support is extended through about 0.75 m about 10 mm of compression would be needed to increase the setting pressure of 100 kg/cm² to a yield pressure of 400 kg/cm². This does not take into account the expansion of the hose pipes. It will be obvious that for an efficiently operated longwall face with self-advancing supports the supports might not attain the yield load if initial setting load is low. To have proper utilisation of the support capacity it will be better to keep the setting load as close to the yield load as possible. It is, however, observed that the performance of yield valves is not uniform and it also deteriorates with use. It is, therefore, generally recommended that setting load should be about 80% of the yield load. In UK the setting loads are kept still lower due to generally weak roof conditions.

Looped hydraulic supply lines of sufficient diameter are provided at the face to avoid excessive pressure drop in the supply lines. Such excessive pressure drop would cause considerable reduction in the setting load.

Operating Range of Supports

While selecting the operating range of supports the following factors need to be kept in view :

(i) the variation in thickness of the deposit ;

(ii) the anticipated convergence ;

(iii) possible development of steps in roof and floor due to improper cutting (about 5 cm may be provided on this account) ;

(iv) possible accumulation of loose debris below the base and above the canopy (about 5 cm may be allowed for this reason also).

Criteria of Selection

Apart from the load-bearing requirements, some of the other considerations for selection of supports are given below :

(i) low cost ;

(ii) large operating range ;

(iii) ability to negotiate steps and cavities ;

(iv) provision of minimum floor pressure at the forward edge of base ;

(v) provision of maximum protection from roof and goaf ;

(vi) stability against lateral forces ; and

(vii) any other considerations.

(i) Cost : Costs of frame and chock supports with low load-bearing capacities normally are in the same range, and are cheaper than shields which are 30% to 50% more costly for the same load-bearing range. Chock shields are more expensive than shields.

(ii) Operating range : Generally, extension of height over the minimum to the extent of 90% is possible in case of frames and chocks while in case of shields it is of the order of 150% for two-legged shields and 200% for four-legged shields. The operating range of shields is greater than that frames and chocks. However, the support resistance offered by the shields varies with a change in height.

(iii) Negotiating steps and cavities : Frame and chock supports having an articulated canopy with an articulated joint located ahead of the front legs, are well suited for negotiating steps and cavities. Shields and chock shields with an articulated extension in the front portion of the canopy also meet this requirement adequately.

(iv) Minimum floor pressure at the forward edge of base : This requirement can be examined only on the basis of stress analysis carried out taking into account the characteristics of the support and the roof. However, the chock shields and five-or six-legged chocks meet this requirement more satisfactorily compared to the other supports.

(v) Maximum protection from the roof and goaf : Amongst all the types, the frame offers the least protection from the roof while shields and chock-shields offer the maximum. Also, unless a special attachment is provided, the frames and chocks do not provide adequate protection against goaf debris, while shields and chock shields provide the maximum. For very weak and friable roofs, the shields or chock shields would be the best choice from the point of protection against falling/rolling rock pieces.

(vi) Stability against lateral forces : Shields and chock-shields are the most stable against lateral forces both parallel and perpendicular to the face. (Chocks are much more stable compared to frames). Application of frames therefore may be considered for thin seams only, while the chocks and shields for medium-thick and thick seams respectively.

(vii) Other considerations : Four-legged shields or chock-shields and five-or six-legged chocks offer better protection to the men against fall of coal from the face because they provide a free passage-way between the legs. These supports also offer easier movement at the face.

The support resistance for five-or six-legged chocks is distributed on five or six legs, and therefore, the resistance per leg is lower compared to that offered by four-legged chocks. Moreover, the supported span for five-or six-legged chocks is greater compared to four-legged chocks. In situations where the roof is not easily cavable and the greater load is shared by the rear legs, four-legged chocks would be more suitable. Supported span for shields is lower compared to chocks, and therefore, lower support resistance is required compared to chocks when deployed in similar working conditions.

For very weak roofs, immediate forward supports (IFS) are sometimes introduced. These supports stand at a distance of one web backwards from the face conveyor. There are, however, some disadvantages associated with this system.

Some supports are also provided with special attachment to provide immediate support to the coal face if the coal is very weak.

For extraction by sub-level caving method, special types of shield supports permitting efficient and safe extraction of coal from the caved area are required to be deployed.

REFERENCES

1. Coal Mine Ground Control — *S S Peng.*
2. Roof Control in Longwall Face — *Dr Mont H Irresberger.* Colliery Guardian, April, 1978
3. Various Aspects of Longwall Roof Support — *A H Wilson.* Colliery Guardian, April, 1978.
4. Method of Mining Thick Seams in France—Indo-French Symposium on Coal Mining, April, 1979, Ranchi
5. Design of Support Systems for Future Longwall Faces — An Overview *Ajoy K Ghose* — Journal of Mines, Metals & Fuels May, 1976.
6. Some Recent Developments in Powered Roof Support, — *A D Allen* — Journal of Mines, Metals & Fuels — May, 1976

a mathematical model for controlled blasting parameters

S N Jog, Dr R Kumar & P R Roy

Introduction

The controlled blasting technique has been evolved with a view to arresting the flying of rock fragments and to control the ground vibrations produced due to the blasting operation. Need for controlled blasting has been described in detail by several authors [1 2 3] The USBM[4] and others have conducted extensive field studies of ground vibration due to blasting and have suggested 50 mm/sec as the safe limit of the peak particle velocity. The same has also been accepted by the Indian Standards Institution for deciding the demaging effects due to blasting vibration. Langefors and Kihlstrom[5] have also suggested certain safe limits of the peak particle velocity.With the objective of arresting the flying of rock-fragments, within these safe limits, experiments were conducted by the Explosive Utilisation Wing of CMPDI in several coal mines. Based on the results of these experiments, efforts have been made to develop a simple methematical model which will primarily be employed as the starting point for further experimentation in controlled blasting, and generation of larger data.

Approach for Developing a Mathematical Model

Simplifying Assumptions : As the data available for analysis are limited, the following assumptions have been made. With the availability of larger data the analytical model developed here could be further improved.

(i) Large variations in the properties of the rock met with in the mining areas, affect the magnitude of vibration caused during blasting. A due consideration to physico-mechanical properties of these rocks is necessary. However, due to non-availability of adequate data on physico-mechanical properties of the rocks, the analysis has been done by classifying the rocks merely as shale, sand stone and coal.

(ii) The make and composition of the explosive and its characteristics such as weight strength, bulk strength, velocity of detonation etc. do influence the blasting operation. However, for the present study no distinction has been made between different types of explosives, assuming that the type of explosives generally used in the mines does not have a very significant effect on the change in the vibration level.

Financial Relationship

(a) Generally, the bench height which depends on the machinery in use at the mine (such as, shovel etc.) determines the depth of the hole. Though the maximum bench height is normally fixed in a quarry, the bench height, and therefore, the depth of the holes vary with the working site in the quarry. The bench height may also vary from one quarry to another with a consequent change in the depth of the holes. As such, the hole depth (D) has been considered as an independent variable. Further, the diameter of the hole remaining constant, the burden can be increased with the increase in depth of the hole.

i. e. $(B)_d \propto D^p$

Where $(B)_d$ = Optimum burden in metres for a given hole diameter, d

D = depth of hole in metres

p = Undetermined constant.

(b) Since the peak particle velocity is a function of maximum charge of explosive per delay it has been considered for calculation of the optimum burden in a drilling pattern for controlled blasting. Therefore, the following functional relation is considered :

$$(B)_d \propto m^q \qquad \text{—(2)}$$

where m = Maximum charge per delay (in kg).

q = Undetermined constant.

Sr. Sc. Asstt. Supdt. (M), Sr. Blasting & Drilling Engineer, CMPDI, Ranchi

(c) The equations (1) and (2) may be combined to get
$(B)_d = D^p m^q r$ —(3)
where r is another undetermined constant.

(d) Spacing between the holes in a drilling pattern is assumed to be a function of the burden.
i.e. $S = f(B)$ (4)
where S = Spacing between the holes in metres

Analysis

General : Controlled blasting experiments in conjunction with the measurement of vibration produced were carried out at several open cast coal mines. In addition, preliminary studies to establish the technique were conducted in other coal mines. The data obtained from the experiments also included the recorded values of blasting vibration, and they were analysed statistically to evaluate the undeter mined constants in equation (3).

Reliability of the Data

The general procedure adopted for the controlled blasting experiments had been as follows :

The blasting parameters were evolved on the basis of general theoretical considerations and practical experience. These were modified further to suit the actual rock formation at site. Thus the optimum blasting parameters (viz., spacing and burden for a given depth of hole) which would produce satisfactory breakage with the minimum vibration by using an optimum explosive charge were determined by trials. Then studies were made with various delay intervals to find out the optimum delay interval to control the vibration. The next step in the process was to determine explosive charge per delay which would produce a ground vibration at the nearest surface structure to be protected, with a particle velocity within the safe limit (50 mm/sec.). The results of the experiments carried out at various mines are summarised in Table 1.

TABLE 1

Sl. No.	Identification of sample space	Total No of expts.	No. of expts considered for analysis (sample size)	General rock type
1	2	3	4	5
1.	S_1	24	24	Shale
2.	S_2	14	10	Shale
3.	S_3	13	10	Coal
Total		51	44	

Analysis of the Data

The data generated from these experiments (The experimental data are included in Table 4) have been analysed statistically using the principles of multiple rsgression analysis. For the purpose of analysis eqn(3) has been changed to the following linear relation, by taking logarithms.

$$Y = pX_1 + qX_2 + K \quad (5)$$

Where $Y = \log (B)_d$
$X_1 = \log D$
$X_2 = \log m$
$K = \log r = \text{Const.}$

The values of the undetermined constants p, q and r, are determined for two different hole diameters in shale and one hole diameter in coal and these are given in Table 2.

TABLE 2

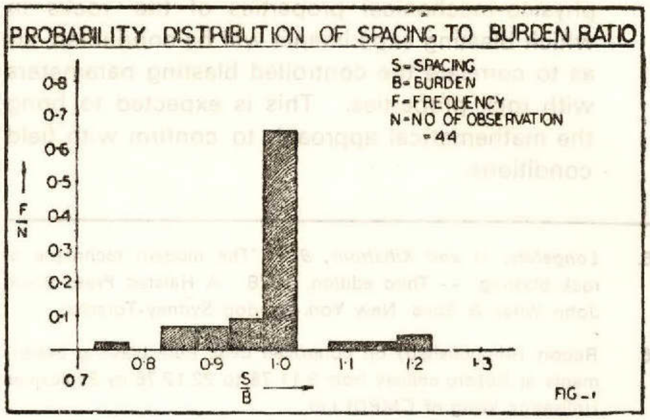
Constants p, q and r for Various Hole Diameters

1. No.	d in mm	r	p	q	Rock type	Identification of samples space
1.	243	1.08	0.67	0.08	Shale	S_1
2.	156	2.02	0.32	0.05	Shale	S_2
3.	156	3.14	0.22	0.025	Coal	S_3

The data have also been analysed to find out the relation between spacing and burden. For this purpose, the ratio of spacing to burden has been calculated for all the 44 experiments considered for analysis. Table 3 gives the frequency of spacing to burden ratio at intervals of 0.05 of each. The probability distribution of S/B values has been shown in Fig. 1.

TABLE 3

Sl. No.	S/B	Frequency f	Probability f /N
1.	0 725 - 0.775	1	0.023
2.	0 775 - 0.825	–	
3.	0.825 - 0 875	3	0 068
4.	0 875 - 0.925	3	0.068
5.	0 925 - 0.975	4	0 091
6.	0.975 - 1.025	29	0.659
7.	1.025 - 1.075	–	—
8.	1.075 - 1.125	1	0.023
9.	1.125 - 1.175	1	0.023
10	1.175 - 1.225	2	0.045
11.	1 225 - 1.275	–	—
		44	1.000



Results

There exists a definite relationship between burden, hole depth and maximum explosive charge per delay for a given hole diameter and rock type. Following are the equations for different hole diameters in two rock types viz. shale and coal.

1. $(B)243=1.08\ D^{0.67}\ m^{0.08}$—for shale
2. $(B)156=2.02\ D^{0.32}\ m^{0.05}$—For shale
3. $(B)156=3.14\ O^{0.22}\ m^{0.025}$—For coal

The above equations have been used for calculating burden values (B') in the respective sample spaces. Table 4 gives the calculated burden against the actual burden for comparison.

There also exists a definite relationship between spacing and burden. From the analysis the probability of having the spacing equal to burden with allowance of $\pm$ 2.5% is 66%.

TABLE 4

Comparison of Experimental (Actual) Values with the Calculated Values for Burden

Sl. No.	Identification of sample space					
	S		S		S	
	B Meteres	B' Meters	B Metres	B' Metres	B Metres	B' Metres
1	1	3	4	5	6	7
1.	5.49	5.60	6 41	5 70	5.49	5 73
2	5.49	5.56	5.19	5 44	5.49	5 79
3.	5.49	5.67	5.49	5.33	6.10	5.66
4.	6.10	5.36	5.49	5.70	5.49	5.73
5.	4.58	4.61	5.49	5.57	6.10	5 68
6.	5.04	5.77	6.41	5.51	4.58	5.70
7.	6.10	5.85	5.49	5.47	4.58	4.89
8.	6.10	5.85	5.49	5.39	6.10	5.29
9.	6.41	5.84	5.34	5.10	6.41	5.93
1	2	3	4	5	6	7
10.	5.40	5.55	4.58	5.47	6.10	5.75
11.	4.58	5.55				
12.	5.69	5.86				
13.	6.10	5.45				
14.	5.19	5.23				
15.	5.19	5.39				
16.	5.19	5 18				
17.	4.58	5 22				
18.	5.49	5.34				
19.	5.49	5 22				
20.	5.49	5.45				
21.	5.49	5.90				
22.	5 49	5.45				
23.	5 45	5 45				
24.	4.58	5.13				
$\Sigma\ (B-B')^2$		4.016		2.103		2.935
$\bar{B}$		5.425		5.538		5.644
N		24		10		10
$\Sigma\ (B-\bar{B})^2$		6.068		4.147		3.754
R		0.58		0.70		0.47

Note :

$$R = \sqrt{1 - \frac{(B-B')^2}{(B-\bar{B})^2}}$$ is the coefficient of multiple correlation.

when $\bar{B}$ is the mean value of burden.

Discussions/Conclusion

(i) Multiple correlation co-efficients have been calculated for the three sample spaces. These are recorded in Table 4. From these values it appears that there exists a positive correlation between the variables under consideration (i.e., burden, depth and max charge of explosive per delay). Though the correlations observed are not very good, the equtions arrived at in this paper can be used for estimating burden where similar conditions exist in future experimentation.

(ii) The data used for analysis have been based on the situations where the shot to gauge distance has been to a minimum of 45 m and therefore, the applications of the above method for limiting the particle velocity to 50 mm/sec shall be valid for cases where the structures are situated at distances beyond 45 m of the blasting site.

(iii) In most of the cases included for the analysis, the optimum delay interval to reduce/cancel the effect of two consecutive shots have been found to vary between 75 to 100 milliseconds. Therefore, for the design of controlled blasts, the delay interval between two consecutive shots shall be kept in this range.

(iv) The analysis of the data has provided a mathematical relationship for evolving the controlled blasting parameters. These parameters, would, however, need modifications on the basis of further trials. This would save considerable time in conducting trials of controlled blasting and would help in the safety of surface structures lying close to opencast mines.

(v) Further experimentation will help in generating of additional data for analysis, as also in testing the validity of these equations.

(vi) Efforts shall be made to determine the relevant physico-mechanical properties of the rocks in which blasting experiments will be conducted, so as to correlate the controlled blasting parameters with rock properties. This is expected to bring the mathematical approach to confirm with field conditions.

REFERENCES

1. *P R Roy* : "Controlled Blasting Technique" — unpublished paper read at DGMS on Aug. 1, 1978.
2. *Dr R Kumar* : "Application of controlled blasting in opencast mines" — unpublished paper read at BHU Varanasi on Feb. 9 1979.
3. *P R Roy & C Krishnamurthy* : "Controlled blasting technique" Minetech, Vol. III, No. 3, pp. 19-23.
4. *USBM Bulletin 656, 1971* : 'Blasting vibration and other effects on structures.'
5. *Longefors, U and Kihstrom, B* : 'The modern technique of rock blasting' — Third edition, 1978 : A Halsted Press Book John Wiley & Sons New York-London-Sydney-Torento.
6. Report (unpublished) on controlled deep hole blasting experiments at Bokaro colliery from 9.11.78 to 22.12.78 by Explosives Utilisation Wing of CMPDI Ltd.
7. Report (unpublished) on controlled deep hole blasting experiments at Khudia colliery (ECL) from 10.12.78 to 31.1.79, Explosives Utilisation Wing, CMPDI Ltd.

rock breaker in primary crushing

K Krishnamachari

Introduction

The maximum lump size which a primary crusher in a CHP can take is 1200 mm X 900 mm. The grizzly over the run of mine (ROM) coal receiving pit is designed to receive lumps of coal upto a maximum size referred above to avoid jamming, resulting in a possible breakdown of the crusher. In actual practice lumps with the ROM coal from a quarry are received whose dimensions sometimes extend to 1500 mm or even more. This immediately results in jamming of the grizzly over the ROM receiving pit where the coal haulers from the quarry discharge their load.

The 'no-coal' hours due to jamming of receiving pit in one of the opencast coal mines detected in the first year 1975 was 25 hours. It was about 200 hours in 1976 and in the region of 400 hours in 1977-78. The problem is, therefore, serious enough to deserve attention.

Solutions to the Problem

Some of alternatives put on trial to eliminate the oversize lumps, are given below :

(i) In one of the opencast mines some years back, an experiment was made by fabricating a partition in a 4.6 m^3 capacity bucket of the shovel. This helped in preventing oversize lumps entering the bucket but at the same time it also resulted in an overall, decrease in the coal production.

(ii) In some of the opencast mines secondary blasting at the quarry coal face was adopted to tackle problems of oversize lumps. This, no doubt, helped in technically solving the oversize lump problem but the consequent increase of the cost of blasting per tonne of coal won was discouragingly high.

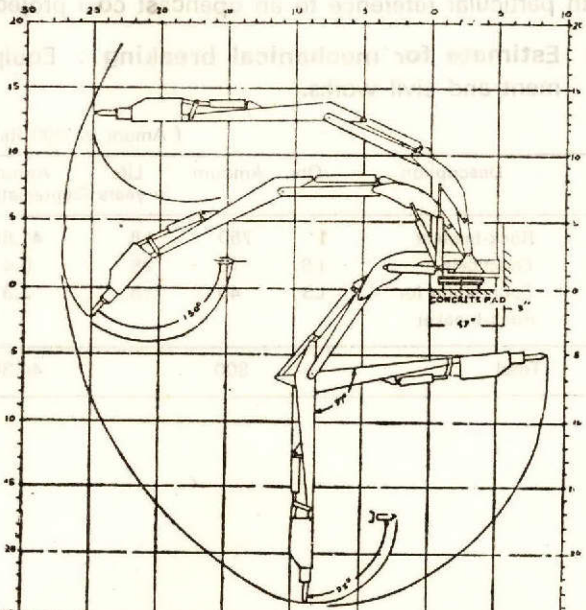
(iii) In a further investigation, one suggestion was made, to allow the grizzly structure to rest on suitably designed robust springs so that the vibration started by the motor, the gear box, and the eccentric crank and connecting rod system (as provided in a motorised reciprocating grizzly feeder) would clear the jam. This system could only clear the jam caused by coal or shale lumps trapped in the apertures between grizzly members, but not the jamming caused by oversize lumps.

(iv) The present practice is to utilise six to ten persons per shift for manually breaking the oversize lumps with sledge hammers which involves safety hazards.

Rock Breaker Adopted in Other Countries

The practice adopted in over 100 opencast mines in the USA and Canada is utilising a hydraulic rock-breaking system to break oversize lumps and dislodge bridged/trapped material over the grizzly.

The device, is either mounted on a concrete base near the ROM receiving pit of the opencast mine, or of a mobile type and comprises essentially the following (shown in Figure below).



Executive Engineer, CMPDI, Ranchi

(i) Booms : A set of booms each articulated by hydraulic metal cylinders has a maximum horizontal reach of approximately 8 m and a maximum vertical reach of 6 m. The boom swing (rotation) is upto 270°. The front boom has a hammer capable of 180° rotation. The hydraulic system includes an oil flow of 28 gpm at 160 kg/cm².

(ii) Power unit : A 40 HP motor operates an oil pump (variable volume and pressure compensated) with an oil reservoir of 60 gallons capacity.

(iii) Hammer : A hammer attached to the front boom is designed for 6 blows per second. An impact energy of 179 kg m is imparted per blow. This, through a 100 mm diameter X 650 mm long demolition tool, can break the oversize lumps on the grizzly within a few seconds.

A number of projects in India have been envisaged to use this type of equipment. Presently this machine is not manufactured in India. One machine is being imported for trials, and it is expected that once its efficiency and usefulness is proved in Indian mining conditions, a demand will be built up allowing the entrepreneurs in the country to come forward to manufacture the machine indigenously.

Economics

The techno-economic study has been made as a comparison between manual and mechanical breaking with particular reference to an opencast coal project.

(i) Estimate for mechanical breaking : Equipment and civil works.

(Amont in '000 Rs)

Sl. No.	Description	Qty.	Amount	Life in years	Annual Depreciation
1.	Rock-breaker	1	750	18	41.66
2.	Electricals	LS	10	28	0.4
3.	Foundation for Rock-breaker	LS	40	18	2.3
	Total		800		44.36

(ii) Manpower (3 shifts)

(Amount in '000 Rs)

Sl. No.	Description	Cat / scale	Persons	Annual wages: Mech breaking	Annual wages: Manual breaking
1.	Rock-breaker operator	IV	3	28.227	—
2	Coal breaker (Manual)	IV	3 x 10	—	282.27
	Total			28.227	282.27

(iii) Comparison

(Amount in '000 Rs)

Sl No.	Description	Total Annual Cost: Mechanical breaking	Total Annual Cost: Manual breaking
1.	Wages	28 227	282.27
2	Power (estimated)	30.240	—
3.	Stores 10% of equipment cost	7.5	—
4.	Depreciation	44.36	—
5.	Interest on loan capital 10.5%	78.750	—
6.	Interest on working capital 11%	7.256	—
7.	Cost of the sledge hammers required per year 10 x Rs 150	—	1.50
	Total	196 333	283.770

(iv) Saving : From the tables given above a net saving with the use of the rock-breaker instead manual breaking is about Rs 283,770—196,333 —Rs 87,437/-. This does not include the indirect gain due to the increase in production by reducing the 'no-coal' hours at the grizzly.

Advantages of a Rock Breaker

Mechanical breaking is more efficient and faster than manual breaking.

Manual breaking is hazardous to the persons standing and working on the grizzly.

Besides the direct economic advantages out of the profits for the additional production, the cost of the equipment can be recovered in a few years. The rock breaker is also capable of raking the pile on the grizzly, to separate the oversize lumps from the fines and manoeuvring of oversize lumps to expose the fracture zones and the flatter surface of the oversize to the breaker.

decision-making and the role of analysis

S P Mukherjee

Introduction

Any organisation trying to achieve a goal (fixed or changing, formally stated or informally accepted) has to carry out various actions or operations, through its various elements, committing various resources under various constraints. Operations are preceded as well as succeeded by decisions or choices. In practice, such decisions are not necessarily deliberate and rational. With some operations the decision-making problem is whether 'to do or not to do,' while with most others we should decide 'to do best.' Quite often a problem of the first kind is followed by one of the second.

In a statistical quality control process, a decision between 'no inspection' and 'sampling inspection' of some incoming materials, ending in favour of the second alternative, leads to the problem of deciding on (i) the number of units to be sampled (ii) the method of sampling and (iii) the method of inspection. This comprehensive decision now leads to the actual operations of inspection resulting in some quality measures for the inspected units. The operation (of inspection) necessitates a second decision between such alternatives as (i) accept the consignment of materials (ii) reject the consignment (iii) accept the materials for a different (from intended) use (iv) have more inspection for a better decision etc. A decision to choose the fourth alternative has to be followed by the operation of further inspection.

Decisions are judged only through the consequences produced by operations/actions based on them and nothing can vouchsafe a perfect implementation or translation of a decision into an operation. Consequences are judged by individuals/institutions who can rarely rise above their idiosyncracies.

And the implementation of a decision is only in relation to an environment. The diversity in environments and in judgements, let alone the difference in perceived objectives (as distinct from stated ones), of different decision-makers in solving a decision-problem makes up the colourful canvas of decision-making.

Deptt. of Statistics, Calcutta University

Decision-making

Broadly speaking, decision-making is concerned with seeking better ways to :

— uncover and select goals consistent with the organisation's interest ;

— design and choose alternatives to achieve these goals ; and

— ensure that the alternative(s) selected is (are) properly implemented.

Decision-making gains from a quantitative analysis of decisions but is much more than any form of analysis. Policy analysis is the generic name for a comprehensive analysis of decisions, policies or strategies.

Policy analysis usually proceeds through the following steps :

(1) Clarify the issue ;
(2) Identify the objectives ;
(3) Develop measures of effectiveness ;
(4) Determine a criterion for comparison ;
(5) Determine the environment ;
(6) Identify the alternatives ;
(7) Formulate models ;
(8) Collect necessary data ;
(9) Carry out the comparison ;
(10) Examine the analysis for sensitivities ; and
(11) Summarise and, where appropriate, recommend.

Decision Taxonomy

Decisions do not admit of a unique taxonomy. Level of difficulty, importance of consequences, frequency of occurrence, time span of influence, operational/technological/investment characteristics and the like provide possible bases for classification of decisions.

Personnel selection is scientifically difficult as distinct from inventory reordering since it is difficult to evaluate the effects of various selection policies and to measure employee quality, if not efficiency.

A daily production scheduling decision has less effect on the company than a capital investment decision. The second is also relatively unique since economic climate, competitive position etc. change fast enough with repetitions of such decisions, while the former may have a repetitive solution. A daily production plan influences a much smaller time span than a new product decision.

An operational decision is concerned with the best way to utilise given resources to operate a given system e.g., inventory control; a technological decision is concerned with the design specification of the system e.g., design decisions for a transformer; an investment decision is concerned with the level and nature of facilities e.g., number and size of warehouses.

The taxonomies may be correlated e.g., their importance may increase as we move from operational through technological to investment decisions. Policy decisions restraining the selection of personnel for certain posts to personnel within the organisation may indirectly appear as a constraint on a lower level operational decision. Only a cross-taxonomy can bring to relief all relevant attributes of a decision situation.

Some Important Elements

The five most important elements in a decision problem are :

(1) The Objectives : The objectives are what a decision-maker seeks to accomplish or to attain by means of his decision. Unfortunately, they are not even fully perceived by the decision-maker, let alone be unambiguously communicated to the analyst. In addition, even for the individual decision-maker, and certinly for a composite one, the goals are likely to be multiple and they may often be conflicting.

(2) The Alternatives : The alternatives are the opinions or means available to the decision-maker by which, it is hoped, the objectives can be attained. Alternatives need not be obvious substitutes for one another, nor perform the same specific functions. Also, the alternatives are not merely the options known to the decision-maker at the start, they include whatever additional options can be discovered or invented later.

(3) The Impacts : The designation of a particular alternative as the means of accomplishing the objective implies a certain set of consequences. Some of these are benefits and contribute positively to the attainment of the objectives; others are costs. In addition, there are the spillovers or externalities, which may affect the attainment of the decision-maker's (or the affected group's) certain other objectives. Many, but by no means all, costs can be expressed adequately in monetary or other quantitative terms.

(4) The Criteria : A criterion is a rule or standard by which to rank the alternatives in order of desirability. A frequently used example would be : given a fixed objective, (rank first) the alternative that can accomplish it at the least cost.

(5) The Model (or models) : The heart of any decision analysis is the existence or creation of a process that can predict, or at least, indicate what impacts would be generated and to what extent the objective would be attained, if an alternative were to be implemented. This role is fulfilled by a model.

The Role of Analysis

Given the set of alternatives and some criterion whereby to assess the extent to which an alternative achieves the objective. the decision analysis helps fixing the target or objective as the criterion value for the optimum alternative, inflated to some extent to allow for a fuller or better implementation or exploitation of each alternative (reaching a higher value for the criterion). If the objective be specified, greater attention should be paid to the generation of appropriate alternatives.

A better communication with the environments and a manipulation of the constraints may help better implementation.

The ability to devise a criterion that can closely reflect the perceived objective may become essential in situations where deployment of different criteria may lead to varying optimal decisions (strategies).

While the usual analysis is restricted merely to those alternatives which satisfy some given constraints, it must be noted that such constraints are not always inviolate, though violation of constraints is necessarily associated with some penalty which detracts from achieving the objective. It may be possible in some cases to incorporate such penalties (using a suitable trade off relation) in the objective function, and to optimise this modified objective function within the the bigger class of alternatives which do not necessarily satisfy constraints.

Not infrequently, it so happens that an analysis of alternatives already identified (and not necessarily generated), since an analysis based on modelling need not consider the alternatives physically — leads to an optimum that itself meets (is expected to meet) the objective only poorly. Sometimes the decision-maker may willingly ignore a number of available alternatives that seem to him as having low prospects (possibilities) of achieving the objective(s).

Methods of Analysis

Operations research, cost-benefit analysis, cost-effectiveness analysis, and systems analysis are often used for decision making (analysis). The distinctions among these are rather arbitrary and arise largely from their origins.

Operations research seeks to use scientific methods to assist decision-makers in getting the most out of available resources. Unlike science, its purpose is not merely to understand or to predict, but to manipulate the real world more effectively. The term systems analysis began to be applied to the broad 'higher level' studies that looked into aspects that OR workers had usually considered 'given' (the objective for instance), and accepted models that seemed hardly scientific to some.

Cost-effectiveness is a form of systems analysis in which the alternative actions, or systems under consideration, are compared in terms of two of the consequences, rupee or resource costs and the effectiveness, associated with each alternative. The preferred alternative is usually taken to be either the one that produces the maximum effectiveness for a given level of cost or the minimum cost for a fixed level of effectiveness. On occasions, when the scale of the operation is not specified, the preferred alternative is taken to be the one for which the ratio C/E, of cost to effectiveness, is the least, but this sometimes leads to error. Whereas we are often able to use cost-effectiveness to rank competing alternatives for the same goal, we cannot use it to compare alternatives that seek different goals.

Cost-benefit analysis is, in theory at least, a much more powerful tool for decision-making than cost-effectiveness. Ideally, all costs and benefits should be identified, converted to monetary units, and taken into account in the evaluation.

This means costs and benefits for the life of the project, not just for the immediate future. One of the defects of the cost-benefit analysis is that the format does not consider the distributional effects of costs and benefits.

As an alternative to systems analysis and the systems approach, Lindblom suggests 'disjointed incrementalism', a refinement of a process he had earlier referred to as 'muddling through', which he describes in the following manner :

"The incrementalist feels he can afford to make only minor changes and to make mistakes because policy-making is serial and fragmented. Problems are never solved; instead some analysis is done, a decision is made, unanticipated adverse consequences show up, more analysis is done and more decisions are made to remedy the adverse consequences, etc., ad infinitum."

Concluding Remarks

A deep understanding of Art and Science differentiates the first from the second by putting the former as a function of feelings and the latter as a function of thoughts. Placed in this context, decision-making is as much an art as a science. Feelings are as amenable to analysis as thoughts are, though forms and fruits of analysis differ in the two cases.

References

1. Lindblom, C. E. Quade, E. S. Raiffa, H — Decision Analysis, Addison Wesley(1970)
2. White D. J. — Decision Methodology, John Wiley & Sons, New York (1978)
3. Simon H. A. — The New Science of Management Decision Harper & Row. (1960)

treatment of borewell water

K R Bulusu, B N Pathak & D N Kulkarni,

Introduction

The Forest Development Corporation of Maharashtra has a Field Station at Kopela, 25 kms from Sironcha, Chandrapur district. There are about 500 people in residence, mostly floating forest labour. FDCM constructed a domestic borewell of 150 mm dia in place of two dug wells to meet the domestic potable water demand. The borewell is 61 m deep and the water level at the time of testing was at 4.6 m. The yield varied between 3.63 to 4.54 m^3/hr with an average of 4 m^3/hr. NEERI, Nagpur undertook the physico-chemical analysis and bacteriological examination of the water. While bacteriological quality was satisfactory, the physico chemical characteristics were far from satisfactory.

TABLE 1

Analysis Report of Kopela Borewell Water Sample Collected by the DM, FDCM, Sironcha on 19.3.1978 (WD-349)

Physical Characteristics

Appearance	Turbid	pH	4.9
Colour	Reddish yellow	Langelier Index	4.5
Turbidity, units	200	Conductivity uS/cm	330

Chemical Characteristics

(All values shown below are in mg/l)

Dissolved solids	140	Sodium, as Na	9
P-alkalinity*	0	Potassium, as K	8
M-alkalinity*	5	Iron, as Fe	8
Total hardness*	92	Manganese as Mn	0.2
Alkaline hardness*	5	Chlorides, as Cl	6
Non-alkaline hardness*	87	Sulphates, as SO_4	84
Free CO_2 *	283	Flourides, as F	0.5
Permanganate value**	1.6	Nitrates, as N	0.8
Calcium, as Ca	20	Phosphates, as PO_4	0
Magnesium, as Mg	10.2	Silica, as SiO_2	8

Probable Composition of Residue

$CaCO_3$	5	$CaSO_4$	61	$CaCl_2$	0	$Ca(NO_3)_2$	0	SiO_2	8
$MgCO_3$	0	$MgSO_4$	50	$MgCl_2$	0	$Mg(NO_3)_2$	0	Fe_2O_3	11
Na_2CO_3	0	$NaSO_4$	0	NaCl	10	$NaNO_3$	0	Mn_2O_3	0.3
K_2CO_3	0	K SO_4	0	KCl	0	KNO_3	6	Al_2O_3	–

* as $CaCO_3$

** as Oxygen, 4 hrs at room temperature.

Based on these tests, NEERI declared the water unsatisfactory for domestic purposes and informed FDCM that a treatment is necessary to make the water potable.

A mine laboratory was set up for the purpose of field studies and meaningful corrective treatment.

Field Studies

The tube well is housed in a hall measuring 10.85 x 3.65 x 3.95 m and is presently fitted with hand-pump. The tube well is to be ultimately coupled to a diesel engine for 8 hours a day operation to obtain an estimated yield of 4 m^3/hr of water on an average. The tube well was in disuse for long owing to the bad quality of water. Hence, the hand pump was operated for 3 hours on 9.1.1979 and 4 hours on 10.1.1979 prior to commencing sampling for tests. The results of the analysis of samples collected are summarised in Table 2.

The taste and odour of the water were repulsive and it was difficult to retain water in the mouth for even a minute, because of its extreme astringency and objectionable odours.

Soon after the sample was drawn from the tube well, mild effervescence was observed due to the escaping gases. In minutes, the sides of the beaker were covered by tiny bubbles, which on stirring with a glass rod coalesced and escaped.

TABLE 2

Physico-Chemical Characteristics of Water Samples Collected from Kopela Tube Well

Date of collection	9.1.79	10.1.79	10.1.79
Time of collection	1600 hrs	1000 hrs	1300 hrs
Odour	Metallic	Metallic	Metallic
Appearance*	Clear	Clear	Clear
Colour*	Colourless	Colourless	Colourless
Temperature °C	28	29	29
Conductivity, uS. cm	440	490	490
Dissolved solids mg/1	264	294	294

Scientists, National Environmental Engineering Research Institute, Nagpur

TABLE 2

Physico-Chemical Characteristics of Water Samples Collected from Kopela Tube Well

Langelier Index	2.67	2.99	—
pHs	8.37	8.39	—
pH	5.7	5.4	5.6
Total acidity, mg $CaCO_3$ /l (pk 8.3)	159	173	198
P-alkalinity, mg $CaCO_3$ /l	0	0	0
M-alkalinity, mg $CaCO_3$ /l	50	46	46
Total hardness mg $CaCO_3$ /l	70	76	—
Calcium mg $CaCO_3$ /l	50	55	—
Magnesium mg $CaCO_3$ /l	20	21	—
Chlorides, mg Cl /l	27	—	—
Sulphates, mg SO /l	100	130	—
Manganese mg Mn. /l	0.10	Traces	Traces
Total iron, mg Fe. /l	30	45	40
Ferrous iron mg Fe. /l	30	45	40
Copper, mg Cu. /l	—	1.8	1.8
Sulphide, mg S. /l	0.1	Traces	Traces
Ammonia, mg N. /l	7.0	7.0	—
Organic matter, mg. /l	1.6	2.4	—
Silica as SiO_2, mg. /l	14	12	—

* Appearance and colour gradually changed on keeping overnight.

A perusal of the results in Table 2 confirms the water as extremely aggressive to iron & steel. It was typically characterised by low pH, moderate hardness and bicarbonate alkalinity, excessive total acidity and dissolved iron. Presence of objectionable quantities of ammonia, peaty organic matter, sulphide, copper and zinc made the water more objectionable to users.

Presence of such large concentrations of iron was due mostly to the dissolved carbondioxide and to a certain extent to the complex organic matter. The fact that intermittent vigorous aeration extended over several hours did not reduce the soluble organic matter suggests the presence of iron in complex organic combinations.

Dissolved iron concentration was excessive and varied between 30 and 45 mg. /l hence in the determination of free carbondioxide figure by the alkalimetric titration method, both carbondioxide and dissolved iron (and possibly acid organic matter also) were included. Hence the results of the alkalimetric titrations were, therefore, more correctly estimated as total acidity than free carbondioxide.

Continuous pumping at maximum capacity for eight hours did not reveal much reduction in the dissolved iron content of the samples that were collected at intervals. It was, therefore, fairly certain that most of the iron was inherent in water and only a small quantity was derived from the metals of the borehole tubing.

Intermittent but rigorous stirring for over four hours and overnight exposure to atmosphere resulted in the development of opalescence in water. A deposit was noticed after 24 hours both on the sides and bottom of the container, with a metallic sheen spread over the surface of the water layer. Even such samples contained iron a part of which was in solution and the remaining in insoluble or colloidal forms.

To observe the varying quality of water on pumping, samples were collected every two hours and examined for conductivity, pH, total acidity, total alkalinity, total hardness and iron. The variation in parameters other than iron was not that significant as evidenced by the results below :

Conductivity	440-490 uS. cm^{-1}
pH	5.3 - 5.7
Total acidity	159-200 mg CO_2/l
Total alkalinity	46-50 mg CO_2/l
Dissolved iron	30-45 mg Fe ./l

Objections to Using Such Water

The presence of zinc, copper, sulphides, ammonia and excessive iron in water is objectionable owing to the production of discolouration, turbidity, deposit, taste, odour and unknown toxicity. The water showed an excessive astringency and characterized by metallic or bitter taste; combination of the iron in such waters with tannin, imparts an inky colour to tea in infusions and vegetable preparations, besides unknown toxicity imparted by combination of ammonia with copper. The water is thus highly undesirable for culinary and other domestic purposes. It is objectionable for laundry purposes since it gives rise to 'iron moulding' to linen and other fabrics being washed.

Essential Treatment for Such Waters

The tube-well water requires to be treated to correct pH, copper, zinc, ammonia, or remove sulphurous odours, excessive carbondioxide and dissolved iron. Troubles due to the presence of all these are, therefore, avoided by removing them from water before distribution, particularly since it contains more than traces of objectionable components.

Treatment by Aeration and Filtration

Results of the study revealed that vigorous aeration for one hour, contact oxidation and filtration did not reduce the dissolved iron concentration to below 30 mg./l and the pH remained around 5.8 $\pm$ 0.1. Other objectionables did not reveal any significant change in their concentration.

Hence the conventional approach viz., aeration, contact oxidation, sedimentation and filtration, to the iron removal problem was not pursued. This was more so, when the question of iron removal is under consideration, it is important to decide what other treatment of water, if any, is necessary, or desirable to correct other undesirable parameters.

It is for example, inadvisable to remove iron yet leave sufficient free acidity, copper, zinc, ammonia etc. The water requires purification from not only these components, but also requires organic and bacterial clean-up, for which suitable measures are necessarily to be combined with those required for iron. With this in mind, various treatments were explored before finally arriving at the following appropriate treatment.

Most Suitable Treatment

The decision as to the most suitable method of treatment out of the various examined was influenced by the characteristics of water in respect of its gaseous, mineral and organic constituents. Adequate analyses were made and experimental treatment carried out before finalising the method comprising treatment with bleaching powder, sodium aluminate and lime followed by sedimentation. This has been found to be very effective in removing all objectionable constituents viz., iron, total acidity, ammonia, copper and zinc from the tube-well water.

The appropriate dose of chemicals were arrived at on the basis of several trial runs. Below 200 mg/l lime and 20 mg/l sodium aluminate, the results were not encouraging. Jar tests were repeated with these chemical doses and the results are shown in Table 3.

TABLE 3

Jar Tests Using Sodium Aluminate & Alum (10 min Flocculation and 30 min Settling)

	Jar Test No. 1					Jar Test No. 2				
	1	2	3	4	5	1	2	3	4	5
Lime dose, mg..../l*	0	100	150	200	250	0	150	200	250	300
Sodium aluminate, mg..../l	0	20	20	20	20	0	20	20	20	20
Conductivity, μS.Cm.../l	420	500	520	470	390	490	595	580	445	450
pH	5.8	6.6	6.9	8.2	8.6	5.4	6.9	8.2	8.4	8.6
P-alkalinity, mg $CaCO_3$.../l	0	0	0	4	10	0	0	6	6	6
M-alkalinity, mg, $CaCO_3$.../l	50	92	104	68	52	46	109	108	66	47
Total hardness mg, $CaCO_3$.../l	70	192	198	190	130	76	187	199	151	128
Calcium hardness mg, $CaCO_3$.../l	—	132	152	130	80	22	136	151	116	100
Magnesium mg, $CaCO_3$.../l	—	60	46	60	50	54	51	48	35	28
Ammonia, mg, N..../l	5.50	0.70	0.45	0.30	0.10	7.00	0.80	0.30	0.10	0
Copper, mg Cu..../l	2.0	1.5	0.8	0.3	0.2	1.8	0.6	0.24	0.3	0.3
Dissolved iron, mg Fe..../l	30	3.50	2.00	0.18	0.15	45	6.0	0.10	0.10	0.10
Zinc, mg Zn..../l	1.6	—	—	0	—	1.6	—	0	—	—

* Purity of the lime used is low. Contains 29±1% CaO only.

A perusal of the results confirm that the dissolved iron persisted in the samples even after agitation for 20 min at 60 rpm during flocculation followed by a 30-min sedimentation. Since plain aeration and sedimentation did not prove effective, addition of chemicals was necessary.

Results revealed that application of 20 mg/l sodium aluminate in combination with 200-250 mg/l lime (commercial, with 29±1% CaO content) produced satisfactory water after treatment. The lime also reduced the total hardness considerably as could be seen from the results. The dissolved iron was brought down to below 0.2 mg/l. Copper, zinc, ammonia and aggressivity were within acceptable limits.

On the basis of these studies, the treatment recommended was mixing with 20 mg/l sodium aluminate, 200-250 mg/l lime (29±1% CaO content) and 5 mg/l bleaching powder (30% chlorine content) for 20 minutes through flocculation to ensure reaction with the undesirable constituents, followed by plain sedimentation for 30 minutes.

Bucket Experiments

To verify these findings and also to assess the acceptability of the taste of the water after the said treatment, large-scale experiments were made in 16 litre-buckets by administering the above treatment followed by manual stirring and settling.

The supernatant was tested by the investigating team and all those present at the site and was found to be acceptable.

The treated sample was examined at Nagpur for copper, chromium, lead, cadmium, zinc, arsenic, iron and manganese by Atomic Absorption Spectrophotometry. None of these metals could be detected indicating their absence in the treated water.

Proposed Plant — A Fill & Draw Type

The estimated requirement of water is for an anticipated population of 800 at Kopela. The FDCM had hinted that the station is likely to be abandoned in another ten years or so.

The Manual on Water Supply and Treatment recommended for rural communities, where house service connections are not contemplated and the supply is through stand-posts, a rate of 40 lpcd. (litres per capita per day) Accordingly, the total requirement of water for a population of 800 is 32 m^3. The average capacity of tube well is 4 m^3/h and in one shift operation of eight hours, the requisite quantity of water can thus be obtainable.

Since the requirement is low, a batch type operation is recommended. For batch-wise settling or the fill and draw system, as it is called, simple hopper-bottom square tanks are used. In all, two hopper bottom tanks of 16 m^3 capacity each, are proposed with a straight depth of 2 m (0.2 m free board). The hopper bottom depth is taken as 1 m enough to hold the settled sludge after treatment.

Each tank is provided with a strirer paddle to facilitate mixing of the chemicals with water. The paddle can be operated either manually or by a diesel drive. A schematic arrangement is shown in the drawing. The stirrer is hinged on a bottom support and is equipped with a gear drive that operates at 10-20 rpm. The vertical paddles are made out of wooden planks and are fitted to the horizontal MS flats. The paddles comprise a net-work of the following :

(i) three horizontal MS flats at 0.6-0.7 m spacing; and

(ii) vertical wooden strips arranged systematically as shown in the drawing. The strips are 25 mm wide and are of three lengths viz., 2.50, 2.25 & 2.00 m.

If diesel drive is available a common drive to strirers in both hopper bottoms is recommended, otherwise,

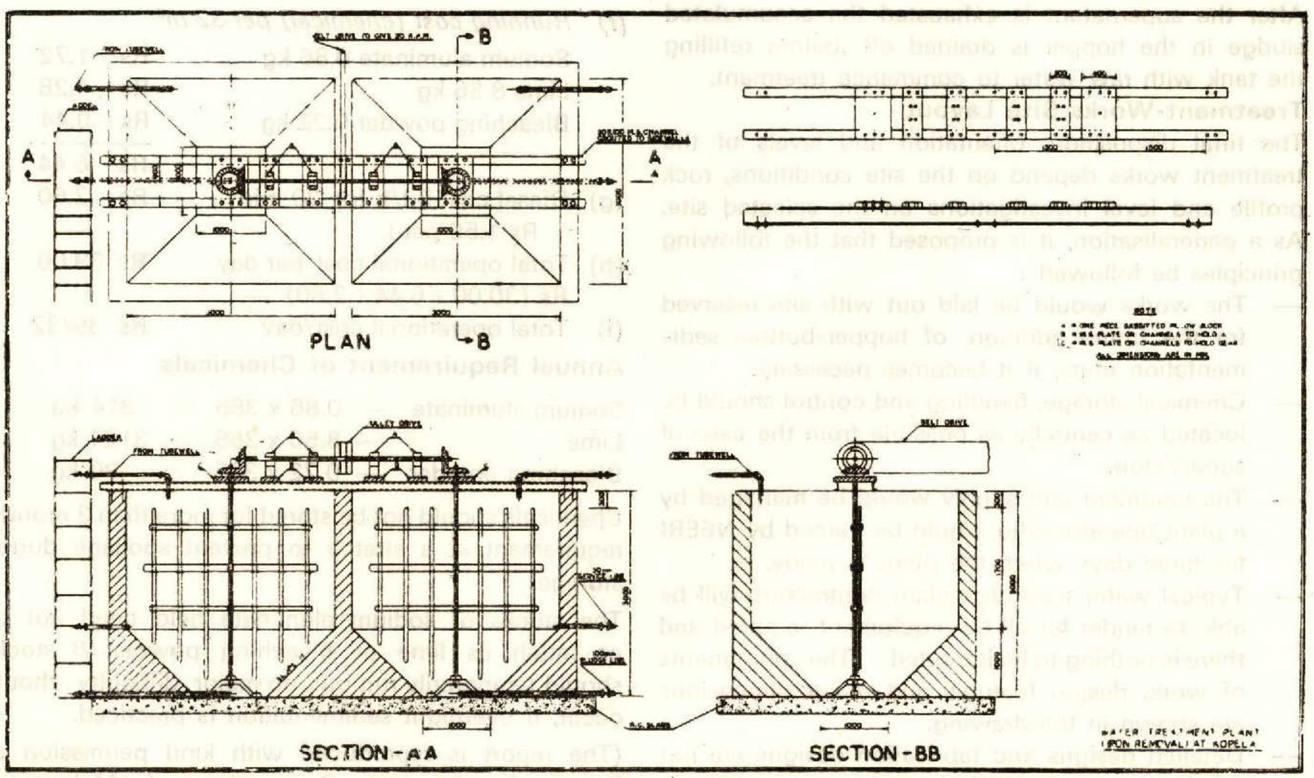


Figure 1

the paddles are to be operated separately and manually by making suitable provisions for hand rotation of the stirrers.

While one tank is in use, the other can be made ready.

The quantity of chemicals required for each fill of the of the tanks are :

Lime (29±1% CaO)/(fill) (tank) = (200 g/m^3) x (21.90 m_3) = 4.38 kg

Sodium aluminate/(fill) (tank) = (20 g/m^3) x (21.90 m_3) = 0.43 kg

Bleaching powder/(fill) (tank) = 0.11 kg

The quantity is based on the volume of water in each tank, i.e. 21.90 m^3 (tank 16.2 m_3+hopper 5.7 m^3).

How to Treat ?

The tanks are filled with raw water leaving approximately 0.2 m free board and the mixing started by keeping the paddles in motion. The chemicals are made into a slurry or solution form in separate polythene buckets of 10 l capacity each and added to the water while mixing is in progress. The mixing is continued for 20 minutes at about 20 rpm. The contents are allowed to settle without disturbance overnight or for two hours at least.

The settled supernatant is ready for use and can be drawn though service connections and standposts.

After the supernatant is exhausted the accumulated sludge in the hopper is drained off ,before refilling the tank with raw water to commence treatment.

Treatment-Works Site Layout

The final disposition, orientation and levels of the treatment works depend on the site conditions, rock profile and level investigations on the selected site. As a generalisation, it is proposed that the following principles be followed :

— The works would be laid out with site reserved for the future addition of hopper-bottom sedimentation units, if it becomes necessary.

— Chemical storage, handling and control should be located as centrally as possible from the case of supervision.

— The treatment and supply would be managed by a plant operator who would be trained by NEERI for three days when the plant is ready.

— Typical water treatment plant contractors will be able to tender for all the equipment required and there is nothing to be imported. The components of work, design features and major dimensions are shown in the drawing.

— Detailed designs and fabrication designs are not within the purview of the reference and these can be got done departmentally.

Cost of Treatment (Based on 1979 Rates)

(a)	*Capital cost*	
	Civil work	Rs 40,000/-
	Machinery & piping	Rs 5,000/-
	Diesel engine	Rs 10,000/-
		Rs 55,000/-
(b)	*Depreciation*	
	Civil works @ 5% p.a.	Rs 2,000/-
	Machinery @ 10% p.a.	Rs 500/-
	Diesel engine @ 20% p.a.	Rs 2,000/-
		Rs 4,500/-
(c)	Interest on the capital @ 10% on capital cost	Rs 5,500/-
(d)	*Maintenance*	
	Civil works @ 2%	Rs 800/-
	Mechanical works @ 5%	Rs 750/-
(e)	*Operational cost*	
	Depreciation	Rs 4,500/-
	Interest on capital	Rs 5,500/-
	Maintenance	Rs 1,550/-
	Total	Rs 11,550/-
	or	Rs 30/- per day (approx.)
(f)	*Running cost (chemical) per 32 m^3*	
	Sodium aluminate 0.86 kg	Rs 1.72
	Lime 8.56 kg	Rs 4.28
	Bleaching powder 0.22 kg	Rs 0.44
		Rs 6.44
(g)	Diesel cost 5 l/h for 20 min Rs 1.56 per l.	Rs 2.60
(h)	Total operational cost per day Rs (30.00+6.44+2.60) —	Rs 39.00
(i)	Total operational cost/day	Rs 39/32

Annual Requirement of Chemicals

Sodium aluminate	— 0.86 x 365	=	314 kg
Lime	— 8.56 x 365	=	3124 kg
Bleaching powder	— 0.22 x 365	=	80 kg

Chemicals should not be stored for more than 3 months requirement at a stretch to prevent spoilage during storage.

The stocks of sodium aluminate held need not be as much as lime or bleaching powder. If stocks should temporarily run out, no major difficulty should occur, if overnight sedimentation is practiced.

(The report is reproduced with kind permission of Forest Development Corporation of Maharastra, Sironcha).

production of substantially low-ash coke from coal-extract/solvent-refined coal

M Prasad, S B Chowdhury, S Mazumdar, A K Roy, D K Banerjee, A C Dutta & S Banerjee

Introduction

Recent constraints on the availability of low-ash petroleum-coke[1] much in demand, particularly for production of pre-backed electrode or Soderberg paste have led to attempts for development of its substitute from alternate sources such as coal-tar or coal-tar pitch by method of delayed coking [2,3,4]. Commercial production of pitch-coke either by delayed coking or in coke ovens [5,6,7] is well known[8]. Results are also reported by Allred [9], wherein coal-extract, an alternate of coal-tar pitch or petroleum-pitch residue, is carbonized and low-ash coke produced. Similarly, Jasienk [10] has also reported some studies relating to evaluation of low-ash coal-extract coke. Aluminium industry, in India, particularly is in a very disadvantageous position because the increased demand of petroleum coke is not entirely met from internal resources. The low-ash petroleum-coke also finds extensive use in other non-ferrous electro-metallurgical industries e.g., production of carbides of calcium, magnesium and silicon. Available information reveals that the total petroleum-coke requirement in various industries of the country by 1983 will be about 3,40,000 tonnes out of which only 2,20,000 tonnes will be available from Indian refineries, and the shortfall of 1,20,000 tonnes has to be met through import. The import of raw petroleum-coke with low sulphur content required for aluminium and other electro-metallurgical industries will result in a heavy drain of foreign exchange.

Hence some technical innovation for its substitution appears to be necessary.

Considering the factors mentioned above, investigations were undertaken in Central Fuel Research Institute, for finding out alternative sources as well as developing a technology for production of substantially low-ash coke which may be utilised as a petroleum-coke substitute in aluminium and various other non-ferrous electro-metallurgical industries. In a previous publication [11] covered by an Indian patent [12] a process had been described wherin medium pitch from coal-tar having a softening point 73.5°C was chosen as the raw material for production of low-ash coke. In the present case, instead of coal-tar pitch, coal-extract/solvent-refined coal was made in an attempt to produce low-ash coke which may primarily serve as a petroleum-coke substitute in the aluminium industry. Since a coking coal is a more likely source of formation of low-ash coke of sufficient strength and other physico-chemical properties required for electrode-grade carbon, the processing was initially restricted to coking coal. However, the possibility of producing coke of suitable nature from semi-coking and/or non-coking coal extracts are also being explored, and details of these investigations will be published later.

Experimental Procedure

For production of solvent-refined coal/coal extract, washed and air-dried powdered coking coal from Dugda washery No. 1 (analysis shown in Table 1) was mixed with an aromatic polynuclear anthracene oil solvent and subjected to extraction under low pressure, or at its own vapour pressure at a temperature of 380-410°C in a pressure vessel for a specified period. The extract slurry was thereafter filtered to remove the mineral matter and undissolved coal. The filtrate, i.e. the dilute extract alongwith excess solvent was then distilled under vacuum to remove excess solvent thereby producing a concentrated extract. Thus the solvent-refined coal (SRC) in the present case might be considered as a solution of coal mass or extracted coal dispersed in solvent media resembling coal-tar pitch, the proportion of coal in it being about 36%.

This concentrated coal-extract or SRC was then carbonized at different temperatures in a 1 kg/batch charging-capacity coker and the yields of coke and by-products such as, gas and liquor were determined.

Scientists, CFRI, Dhanbad

Detailed analyses of coke, by-products, tarry liquor and gas produced by pyrolysis of concentrated coal extract at 650°C (±15°) were carried out, and their properties compared with those of petroleum-coke and pitch-coke. The process as presented in the flowsheet (Fig. 1) consisted of two parts, the extraction and the pyrolysis.

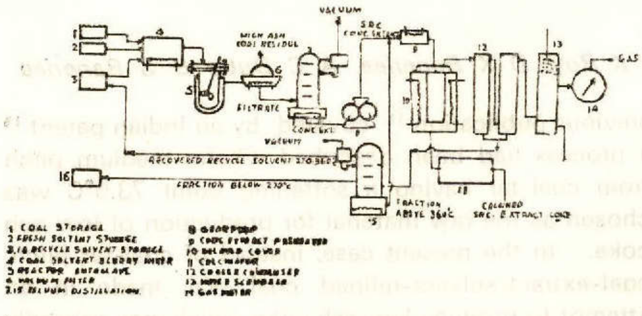


Fig. 1

Production of Low ash SRC-Extract Coke

About 1 kg of concentrated extract having the characteristic properties shown in Table 1 was pre-heated to 200°C (±15°) and then allowed to flow in a thin liqiud stream into a carbonizer retort electrically heated to 650°C. The flow of the liquid extract was controlled in such a manner that the total period of charging was prolonged or delayed. After charging, the retort temperature decreased at the beginning which thereafter again rose till the desired carbonization temperature was attained. The coke mass, i.e. the charge after pyrolysis, was kept at this temperature for about 30 minutes for heat-soaking. The gross energy requirement for the carbonization process including the heat-soaking period was calculated to be roughly about 4200 to 4500 kcal/kg of charge.

At the end of the experiment the coker retort was cooled and the coke product discharged. The by-product tarry liquor was recovered through condensation in the water cooler condensors, and the amount of gas generated determined by means of a gas meter. Later, the coke mass, tarry liquor and gas were all subjected to various physico chemical analyses for evaluation of their properties. The average material balance values of some typical sets of experiments are shown in Table 2.

Results and Discussions

The material balance values as presented in Table 2 indicate that a variation in carbonisation temperature range from 500°C to 675°C does not have much influence on the yield of either the primary product coke, or the by-product tarry liquor. Excepting in one case, the average coke yield was within 30 to 33%, whereas tarry liquor yield remained within the range of 61 to 64%. However, the gas generation at a carbonization temperature of 500°C was low, about 2.5% (by wt) whereas at higher temperatures of 550°C to 675°C it was found to lie within the range of 3.6 to 4.8%. When calculated on volume basis, for 1 kg of coal-extract charge carbonised at 650°C the quantity of gas yield was about 80 litres at NTP.

The recovered tarry liquor by-product comprised partly tar from the coal mass, and partly excess recoverable anthracene oil solvent originally present in the concentrated coal-extract charged. An average result dealing with the fractional distillation of some typical tarry liquor samples presented in Table 3 indicated that the anthracene oil fraction having the boiling point 270°C to 360°C constituted about 87-88% whereas the fraction above the boiling point of 360°C constitute about 7 to 8%. From the latter when recycled to the carbonizer retort, the fraction between 270°C to 360°C might be reutilised in the system for solvent extraction of coal.

The complete analysis of a typical gas generated during the carbonization of coal-extract at 650°C, was carried out both in the Orsat and the gas-liquid chromatographic apparatus (GLC), as shown in Table 4, to determine its different components. The gas contained about 58% hydrogen, about 22% methane and about 8.8% ethane. Although the quantity of gas generated was small, its calorific value was as high as about 6239 k cal/m^3, and could therefore be suitably used as a supplementary combustion fuel in the system.

Coke, the principal pyrolysis product of SRC/coal-extract, was of a very low ash content ranging from 0.4 to 0.8% (as received). The details of the proximate and ultimate analyses of a typical coke sample, produced by carbonizing the coal-extract at a temperature of 650°C is shown in Table 5. For the sake of comparison, similar analyses of pitch and petroleum-cokes are also included in the same Table. According to Table 2, the average yield of coke from coal extract may be stated to be about 32%. It has been pointed out earlier that the concentrated coal-extract is a solution of coal or dispersion of coal mass resembling coal-tar pitch in the highly aromatic anthracene oil solvent media. Hence the total quantity of coke yield is partly the pyrolysis product of extracted coal mass or the pitch-like substance originally present as the dispersed phase in the concentrated coal-extract and solvent media, and partly the carbonization residue of the polymerised components of thermally treated anthracene oil solvent. It may be interesting to note that thermally treated anthracene oil (b.p. 300-360°C) when carbonized alone at a temperature of 650°C (±15°) resulted in a coke yield of only 0.8 to 1.0%. It was thus presumable that during extraction, a part of this solvent anthracene oil in presence of coal substance developed polycondensed and polymerised high molecular hydro-carbons which during pyrolysis in the delayed coker contributed to the coke formation. It has been pointed out earlier [11] that coal-tar pitch when carbonized at 650°C in the delayed coker, produced coke to the extent of 53%. Therefore, out of the 32% coke produced from the total mass of concentrated coal extract, a major part, i.e. about 20% was assumed to be contributed by the 36% extracted coal mass or the pitch like substance, present in the mixture of coal-extract and solvent. The rest, about 12% was contributed by the polymerised or polycondensed aromatic hydrocarbons contained in the anthracene oil solvent of the mixture, perhaps formed during solvent extraction operation as well as during the thermal treatment in the delayed coker unit. During the process of carbonization, besides coke formation, a part was distilled off and another part decomposed to gas and lower boiling fractions having a boiling point below 300°C, as manifest in the analysis data (Table 3.)

However, a clear distinction regarding the source of coke, i.e. coal solvent and polycondensed intermediate complexes, as well as the pyrolysed products from each of these sources constituting the total coke mass was difficult to ascertain.

The overall yield of coke from the coal-extract was comparatively low, the reasons may be on one hand, due to the low Conradson Carbon residue of 29.6% compared to that of coal-tar pitch of 47-50% and on the other, to the near absence of free carbons in the coal-extract as indicated by the low quinoline insolubles of only 1.3% as against about 8% in pitch.

The physical strength of uncalcined coal-extract coke and petroleum-coke as determined by the Hardgrove index values are presented in Table 5. The low values (27-29) indicate higher physical strength of coal-extraction coke over the petroleum-coke with a value of 47. The volatile matter content of extract-coke is much lower than the petroleum-coke and varies within the narrow limits of 3 to 4%. The real and apparent densities of green coal-extract coke, pitch-coke and petroleum-coke as also shown in the Table 5, are more or less similar the real and apparent densities being 1.7 and 1.8, and 1.11 and 1.21 respectively. However, the porosity of pitch-coke is greater than that of either coal-extract coke or petroleum-coke. The sulphur content of extract-coke is about 0.63%. The quality of SRC/extract-coke was further assessed by optical microscopic examination of polished surface sections using polarised light technique. This examination provided useful information concerning the form of microporosity and the orientation of the grains 17. The nature of surfaces of two typical extract coke samples carbonised at 500° C and 650° C (Table 2) respectively are shown in Figs. 2(a) & (b). It may be observed

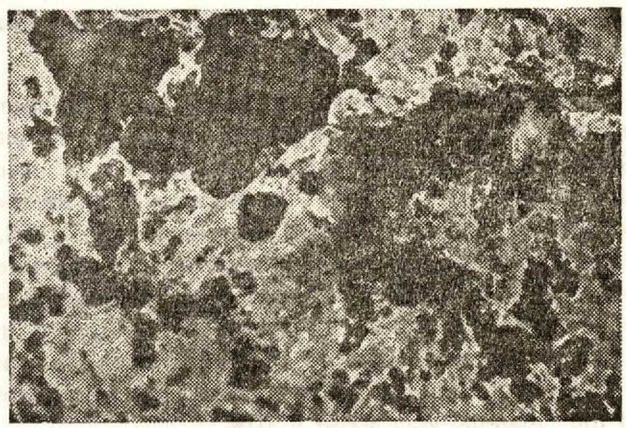


Fig 2 (a)

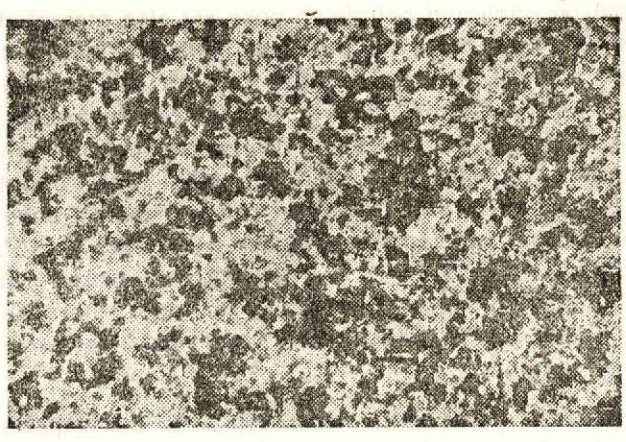


Fig. 2 (b)

Fig 2 Nature of polished surface of SRC — extract coke carbonized at (a) 500°C (b) 650°C
(Microphotograph magnification by 13R ×)

that the coke samples have developed a texture containing only mosaic type regions. These were indicative of the ordering of pre-graphitic layer of carbon atoms. The coke produced at carbonization temperature of 500° C had assorted big and small non-uniform pores, Fig. 2(a); whereas the other produced at 650° C contained small uniformly distributed macro pores resembling the mosaic structure, Fig. 2(b). In this connection, it may be worthwhile to mention that Jasinenko reported the coal-extract coke to be flaked, leafy, fibrous as well as anisotropic in nature, whereas Ramsay and Pitt 13 reported a coke surface having a texture of both mosaic and lamellar regions. Although the reasons for this variation from those reported by Jasienko and Pitt are not clear, it may, however, be presumed that the conditions prevailing during the process of coal extraction, the solvent used and the method adopted during the extract pyrolysis, were responsible for this variation in the coke texture.

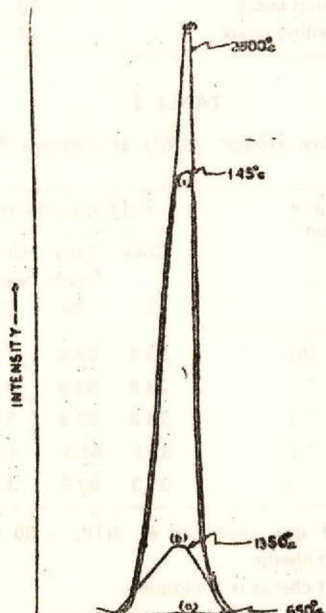
For production of anodes needed in the aluminium industry either as prebaked electrodes or in the Soderberg paste, calcined petroleum-coke is used as a carbon filler material. In order to compare the properties with the calcined petroleum-coke, SRC/extract-coke samples were calcined to a temperature of about 1350° C, and various physico-chemical tests undertaken, the results of which are presented in Table 6. It was observed that after calcination at a temperature of 1350° C, the total loss in weight of the material was about 9-10%.

A comparison between the properties of calcined coal-tar pitch and petroleum-coke are also presented in the Table 6. The proximate analysis indicated that the calcined extract-coke had an ash content varying from 0.6 to 0.8% whereas in the petroluem-coke it varied from 0.2 to 1.5%. On the contrary, the ash content of calcined pitch-coke was high, about 2.3-2.5%. The real density of calcined extract-coke pitch-coke and petroleum-coke were 10.3×10^2, 8.96×10^2, 8.72×10^2 respectively, expressed in $Ohm/mm^2/m$. It is interesting to note that the variations in the electrical and resistivity values were only marginal.

It may be mentioned that along with the physico chemical tests such as real density, electrical resistivity and also microscopic examination, another very useful parameter for determining the suitability of low-ash SRC/extract-coke in aluminium industry, is its reactivity with oxygen. In the aluminium electrolysis bath oxygen is generated near the suspended anodes. This oxygen in turn reacts with the carbon of either prebaked or Soderberg electrodes resulting in its consumption through disintegration, and falling into the bath. It is therefore, necessary that electrodes should have very low ash content as it goes into solution in the metal in the electrolysis bath. As there is no specific test for determining the coke reactivity with oxygen, according to the suggestion by Yanko[14], an oxygen-containing gas viz., CO_2, was used and reactivity of calcined extract-coke, petroleum-coke and pitch-coke (calcined at 1350°C) determined. It may bee pointed out that the method adopted for determining the critical air blast (CAB) value for blast-furnace coke is not applicable in this case. The reactivity required in an electrode-coke is very much a matter of compromise between conflicting requirements. It is well known that pitch is used as a binder in the production of anode which in the finished material, is also converted to coke having high reactivity. To equalise the reactivities of the binder and the filler, the latter should have a low reactivity so that the oxidation in the finished electrode/anode mass can be controlled, and its rapid disintegration avoided. The test results as presented in Table 6, indicate that among all the three types of calcined coke samples, extract-coke had the minimum reactivity value of 43-44 (in cc of $CO/100$ cc of CO_2), whereas in case of calcined petroleum-coke and calcined pitch-coke the reactivity values are higher i.e. about 77 and 129 respectively. It may be pointed out that in an attempt to simulate the thermal conditions in the aluminium electrolysis bath, the reactivity tests were conducted at 950°C and the conversion of CO_2 to CO at this temperature noted. It was observed that the reactivity of all the three calcined coke substances were widely different. The porosity or pore-size distribution as well as the pore volume, coke structure, i.e. crystal growth or degree of graphitisation during calcination, and perhaps the allotropic modification of carbon, are all responsible for the characteristic variation of reactivity. Thus calcined SRC/extract-coke may be suitably used as a carbon filler material in production of the electrode anode much needed in the aluminium industry.

The structural change, i.e. change in the graphitisation characteristics or the crystal growth, takes place with increase of calcination temperature. The extract-coke samples were calcined at different temperatures, 1350, 1450 and 2800°C and the X-ray diffraction patterns for each was examined. The X-ray analysis also enabled the determination of the degree of preferential orientation of graphite crystallites at different temperatures of calcination.

For the sake of comparison, a sample of green extract-coke was also examined in the same manner. Diffraction profiles Figs. 3(b), (c) and (d) of calcined coke samples clearly indicate the increase in the graphitisation characteristics, or ordering and crystal growth of extract-coke with increase of calcination temperature. A comparative measurement indicated the position of peaks of extract-coke calcined at different temperatures. It may also be observed in the Fig. 3, curve (a) that the green coke was not at all graphitised whereas curve; (b) indicates a low degree of graphitisation at 1350°C. The curves



(c) and (d) on the other hand indicate the advanced stages of graphitisation at 1450°C and 2800 C respectively. A critical examination will further reveal that although there was a marked increase in the graphitisation of extract-coke calcined at 1450° C than at 1350°C, the increase in graphitisation at 2800°C was not significantly higher than that at 1450°C. However, the appropriate temperature for calcination will have to be ascertained on the basis of economic factors involved, as well as on actual performance of anode produced from this carbon filler in the aluminium electrolysis bath.

The ash content of the calcined extract-coke was similar to calcined petroleum-coke and low ranging between 0.6 and 0.8%. Its main constituents, as can be seen in Table 7, and calculated on coke basis are 0.12% SiO_2, 0.32% Fe_2O_3 and 0.04% Al_2O_3 respectively. Besides these, other components existing in the ash are Ca and Mg in the form of their oxygen compounds, the total quantity being about 0.042%. An emission spectroscopic analysis for determination of trace elements indicated that the extract-coke ash does not contain Va, Mo, Bi and Cd. Likewise, Ni and Cr are also absent.

Conclusion

The SRC/coal-extract with the softening point of 48.8°C. and containing about 36% pure coal in the solvent anthracene oil, resembling soft pitch, produced 30-33% substantially low-ash coke on pyrolysis in a delayed coker in a similar manner as already reported in a previous publication[11]. A part of this coke was produced from carbonisation of pure coal mass, and another part was the carbonization residue product of polymerised and polycondensed high molecular aromatic compounds, formed from anthracene oil in presence of coal. The coke produced had a very low ash content ranging between 0.4 and 0.8%, the main constituents of ash being the oxides of Si, Al and Fe. An optical microscopic examination of the polished surface of extract-coke produced at 650°C under polarised light showed a mosaic structure having uniform micropores. The physical strength of green extract coke determined by Hardgrove index was higher than that of green petroleum-coke. For the purpose of comparing with the physico-chemical properties of calcined petroleum-coke used for the production of anodes, SRC/extract-coke was calcined at higher temperature and its physical properties namely real density, apparent density and electrical resistivity were determined. They indicate a close similarity with those of calcined petroleum coke. The calcined extract-coke had lower reactivity against CO_2 than petroleum-coke or pitch-coke. In X-ray analysis a well-advanced stage of graphitisation was noticed at 1450°C whereas a low degree of graphitisation was found to take place at 1350°C. Notwithstanding the improved quality of calcined extract-coke, as indicated by various physico chemical tests, it may finally be mentioned that it is difficult to predict the performance of an electrode produced from this carbon filler in laboratory investigations alone, as the thermal and mechanical conditions prevailing in actual aluminium electrolysis furnace are difficult to define and stimulate. Hence for a real evaluation, semi-commercial pilot plant tests appear to be absolutely necessary.

REFERENCES

1. Petroleum Refinery Engineering by Nelson 4th Edition International Students Edition, p. 73.
2. *Rice R L et a* : Industrial Engineering Chemistry, Prod. R/D Vol. 7 No. 4(1968). p. 270.
3. *Gambre, A J et a* : Chemical Engineering Progress Vol 65 No 5 (1959) p. 75
4. *Huttinger. K J, J Bitumen Teere & Aspha t,* Vol II(1970) p 48.7
5. *Oschepkova, N N et a*, Coke and Chemistry USSR. Vol. 2(1970) p 25.
6. *Smirnov B N et a* Coke and Chemistry USSR No 12(1969) p. 21
7. *Jurkiewicz, J et a* Koke Smola Gaz Vol I, No 6(1962) p. 219.
8. *Yashushi Takenchi et a* . Kogya Kagaken Zassi (Japan) Vol 68 No 5.
9. *A red. D V,* Oil snf Gad joutnsl Feb. 12 (1973), p. 66.
10. *Jasienko, S.* Fuel Vol 57, No. 3 (1978) p. 131.
11. *Banerjee, S et a* Chemical Age of India, Vol 26 No 11(1975) p. 849.
12. Indian patent specification No 2038/Ca/74, dated 12 9 74.
13. *John, C V, Rumsey and Geoffrey J Pitt* Fuel Vo 57, No. 3 (1978) p. 155.
14. *Yanko E A et a*, Coke and Chimestry USSR Vol. 6(1973) p. 34 38
15. *Mante C L* Carbon and Graphite Handbook Inter-science Publishers, John Willey & Sons (N Y) USA, (1968) p 156
16. Proceedings of "All India Seminar" on Carbon Technology held in Korba, 4-5 March 1978 p 1 organised by M/s Bharat Aluminium Co Ltd Korba and Indian Institute of Metals Korba Chapter Key note Address by T B Singh
17. Mochida et al Fuel Vo1 58 No. 7(1979) p 482

TABLE 1

Physico Chemical Properties of Solvent-refinsd Coal (SRC) or Concentrated Coal-extract

1	Feed stock	Solvent-refined Coal (SRC)/ Concentrated coal-extract
2	Water content :	
	(a) Volume %	0 15
	(b) Weight %	0.15
3	Ash content (as received) %	0.06 0.08
4	Dens ty as 60 C (gm/cc)	1.07 — 1.14
5.	Specific heat	0 36 0 45
6	Softening point (°C)	48.8
7.	Matter insoluble in quinoline (%)	1.3
8	Matter insoluble n Toluene (%)	10 86
9	Conradson carbon (%)	29 6
10.	Viscos ty (Reswood No. 1)	
	60°C	431.5 sec
	80°C	181 0 "
	100°C	86.5 "
	110°C	61.0 "
11.	Calorific value (kcal/kg)	9031.6
12.	Ultimate analysis (d a f)	
	Carbon (%)	91.10
	Hydrogen (%)	6 10
	Sulphur (%)	0 63
	Oxygen (%)	1.08
	Nitrogen (%)	1 09
	H/C ratio (atomic)	0 80
13.	(a) Proximate analysis of raw Dugda Washed Coking Coal (Washery No 1) (Air-dried basis) :	
	Moisture %	2 5
	Ash %	18.1
	V M %	22 9
	Fixed carbon %	56 5
	(b) Ultimate analysis (d a f)	
	Carbon %	86.18
	Hydrogen %	5 22
	Sulphur %	0 72
	Nitrogen %	2 14
	Oxygen %	5 74
	Caking index	20
	Swelling index	7

TABLE 2

Pyrolysis of Coal-extract (SRC) at Different Temperatures*

Sl No.	Temparatura of carbonisation (°C)		Yield of products Coke (%)	Tarry liquor (%)	Gas by weight (%)	Unaccounted loss %
1	500°	(± 15)	30 6	62 4	2 6	4.4
2	550°	(± ")	28.8	63 9	3 8	3.5
3	600°	(± ")	28 8	63 9	3.8	3.6
4.	650°	(± ")	32 2	61.1	4 8	1 9
5.	675°	(± ")	31.3	62.9	3.6	2.2

Volume of gas produced at NTP, — 80 litres per kg of coal-extract charge

* Coal extract charge is 1 kilogram.

TABLE 3

Fractional Distilat.on of Tarry Liquor Recovered During Pyrolysis of Coal Extract (Carbon sation Temperature 650°C)

Temperature ranges (°C)	Weight (%)
0°-170°	3.1- 2.4
170°-230°	0.8- 0.5
230°-270°	1.1- 1 1
270°-300°	2.3- 2.6
300°-360°	84.6-85 6
360° and above	8.1- 7.8

TABLE 4

Composition of (typical) Gas Obtained During Pyrolysis of Coal Extract at 650°C

Constituents	Volume (%)
CO_2	1 4
$C_n H_m$	1.6
O_2	—
CO	1 6
H_2	57.8
CH_4	21.5
C_2H_6	8.8
C_3H_8	2.7
N_2 (by difference)	4.6
	100 00
Ca value (kcal/m^3)	6239

TABLE 5

Analysis and Properties of Coal-extract (SRC), Coke Pitch-coke and Petroleum-coke (Coal Extract and Pitch Carbonised at 650°C)

Properties	Coal-extract coke (SRC coke)	Pitch-coke	Petroleum-coke*
Proximate analysis			
Moisture %	1.4 - 2.1	1.8 - 3.3	0.5 - 1 2
Ash %	0 4 - 0.8	0.8 - 1.4	0.26- 0.73
VM (%)	3.0 - 3.7	2 9 - 3.3	11.4 -11.8
FC	91.5 -93 5	92.6 -95 3	88.2 -88.6
Ultimate analysis			
Carbon %	89 78-91 60	92 9 -93.4	92-27.92.29
Hydrogen %	2 05- 2.63	0 78- 1.50	4.01- 4.18
Sulphur %	0.54- 0.59	0.37- 0.39	1.02- 1.64
Nitrogen %	1.87- 2.04	1 23- 1.32	0.95- 1.53
Oxygen % (by difference)	1.5 - 2.81	0.84- 1 56	1.00- 1 11
Real density (gm/cc)	1 61- 1 73	1 73	1 8
Apparent density	1.11	1 09- 0.95	1 21
Porosity (%) (average value)	33 7	41.0	32 7
Hardgrove Index (HGI)	27-29	—	47

*Carbon and graphite handbook by C L Mantel (1968) (15).

TABLE 6

Analysis and Properties of Calcined Coal-extract Coke, Pitch-coke and Petroleum-coke (Calcined at 1300°-1350°C)

Properties	Coal-extract coke (SRC)	Pitch—coke	Petroleum coke
Moisture (%)	0.1 - 0.2	0 4 - 0.5	Negligible
Ash %	0.6 - 0.8	2.3 - 2.5	0.2 - 1.5
Volatile matter %	0.6 - 0 7	0 3 - 1.2	Below 0.2
Fixed carbon %	96 3 -98.6	95.8 -97 0	97.0
Real density (gm/cc)	2.08- 2.11	1 93- 2.03	2.08- 2.13
Electrical resistivity (Ohms/Sq mm/m) (at a pressure of 42 5 kg/cm)	10.3 X 10^2	8.96 X 10^2	8 72 X 10^2
Reactivity against carbon dioxide (CO_2) (cc of CO/100 cc of CO_2)	43-44	129	72

TABLE 7

Composition of SRC/Coal Extract Coke Ash (Ash Content of Coke — 0.6 %)

Constituents	Coke basis (% by weight)
SiO_2	0.12
Fe_2O_3	0 312
Al_2O_3	0.046
TiO_2	0.0077
P_2O_5	0.0024
SO_3	0.0081
CaO	0.0094
MgO	0 033
Rest undetermined	0 059

beneficiation trends of indian coals

H C Dutta & A N Banerjee

Introduction

The technology of beneficiation of coal was adopted in India in 1951, when the first washery was erected by Tata Iron & Steel Company in West Bokaro with an installed annual capacity of 0.57 million tonnes. Since then, 14 washeries in the coking coal sector have been set up (excluding the non-coking coal washery set up by the Associated Cement Company, at Nowrozabad) with a total installed capacity of 25.62 million tonnes of raw coal input per annum. New washeries are either under construction or in the planning stage to meet the future demand of the steel plants. An attempt has been made in this paper to evaluate the performance of the existing washeries, and the feasibility of acceptance of new technologies to beneficiate the inferior grade coals in the future washeries.

Coal Characteristics and Industrial Needs

Coking coal, as mined, rarely meets the quality parameters for direct use by the consumers. This is due to the presence of impurities in the coal body depending on the nature and type of deposition during initial stages of the formation of coal. These impurities have to be removed to the extent tolerable before consumption.

Steel plants consuming coking coals have laid down strict specifications for the coals used for coke making. The normal ash content of coals charged to ovens for making metallurgical coke is expected to be at the level of $17 \pm 1\%$. The average ash content of raw coking coals in the present production is around 22-26% which is likely to rise to 28% due to mechanised mining and exploitation of more of the inferior grade coals which will need washing. The situation in the power generation industry, the major consumer of the non-coking coal, is different. Without adopting the costly process of washing, they may find it convenient to design the combustion system to burn high-ash coals.

Dy. CE (W), Regional Director CMPDI.

However, from the point of conservation of the high-grade non-coking coal and for utilisation of the sizable reserves of inferior coal, serious thought has to be given to beneficiate the low-grade non-coking coals for the major consumers like power stations, railways, cement industries as per their specified requirements.

There is no controversy over the technology to be adopted to bring down the ash content of ungraded coal (35% and above) to a level of $32 \pm 3\%$. A simple deshaler will efficiently serve this purpose. It is the acceptance of the beneficiated non-coking coal by the consumers that has to be considered from the economic point of view.

Existing Washeries

As already stated, there are 14 washeries in operation at present, in the coking coal sector with an installed capacity of approximately 6500 tonnes per hour equivalent to 25.62 million tonnes/annum raw coal input, based on 4000 working hours in a year. A list of existing washeries with the technologies involved is given in Annexure 1. A close study of the washing schemes adopted reveals the following :

(a) Heavy Media (HM) separation process has been adopted in all the washeries except the Lodna washery of BCCL.

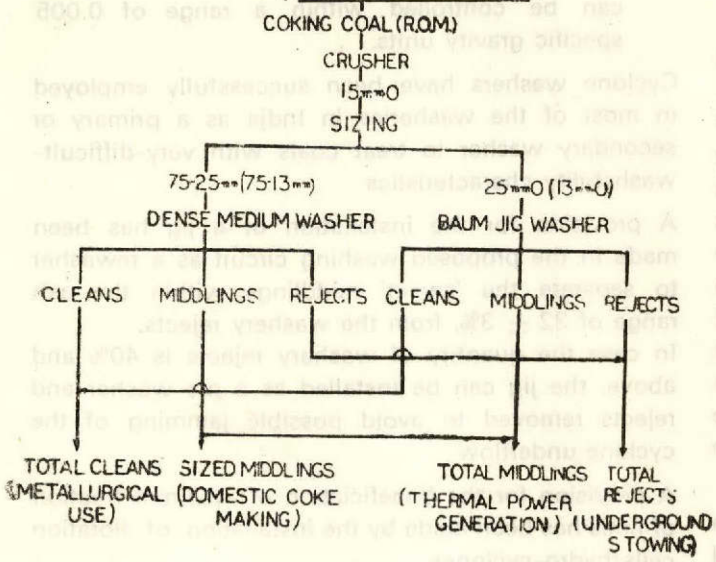
(b) Multistage/composite circuits have been designed with different combinations of Baum jig, HM bath, HM cyclone and thickener-cum-filter depending on the washability characteristics of the coal to be treated. Flotation cells in Kathara and hydro-cyclones in Dugda-II have been added to the respective washery circuits for beneficiation of the fines.

According to the general scheme of washing, which has so long been adopted in India, all the composite washeries can be broadly classified into three categories illustrated in Flow-Sheets No. 1, 2 & 3.

FLOW SHEET-1

COMPOSITE WASHING SCHEME FOR COKING COAL

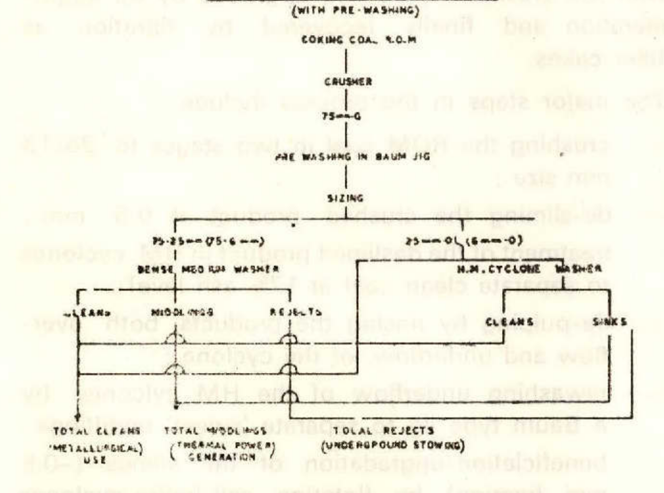
(WITHOUT PRE-WASHING)



FLOW SHEET-2

COMPOSITE WASHING SCHEME FOR COKING COAL

(WITH PRE-WASHING)



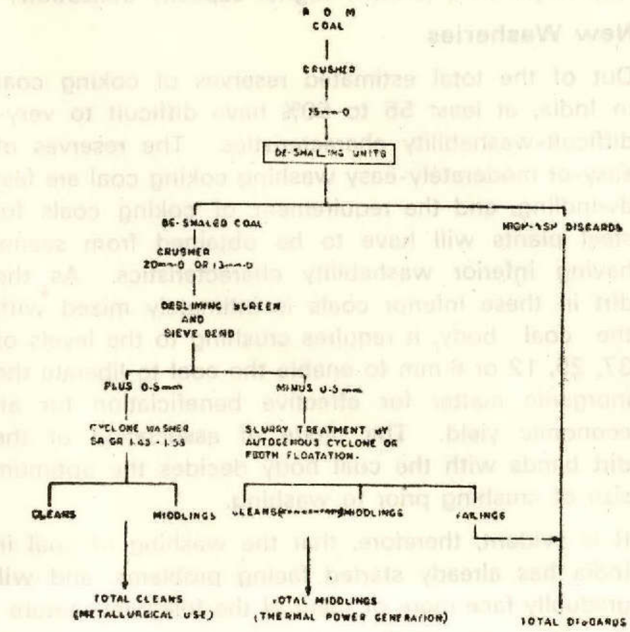
Limitations in the Present Washeries

Due to depletion of reserves of the upper-seam superior-quality coals in the Jharia coalfields, exploitation in a higher percentage is gradually being done from the lower seams (X seam and below). The main production will be obtained in future from lower seams only, and consequently the percentage of ash in coal will go up with an accompanying decrease in VM and the caking index.

The existing washeries primarily designed to beneficiate coals with comparatively better washability

FLOW SHEET-3

WASHING SCHEME FOR UPGRADING INFERIOR COKING COAL



characteristics are now being operated with constraints, mainly the following.

— due to deterioration in the quality of the raw coal feed, the ash of the fines (–0.5 mm) have also gone up from 15-16% to about 21%. As this fraction (10-15% of the total coal) is mixed unwashed with the cleans, the quality of the cleans deteriorate.

— due to an increase in the percentage of fines (–0.5 mm), the fine coal circuit in the existing washeries mostly operate overloaded.

— due to an increase in the percentage of 'near gravity material' (NGM) at the operating gravity of cut, efficiency of separation has deteriorated in some of the existing circuits affecting the quality of the washed product.

In addition to the above, overloading of the middlings/rejects-conveying circuits often takes place, thereby limiting the feed rate.

Steps have already been taken to overcome the constraints generally by (i) increasing the capacity of the fine coal recovery circuit, (ii) upgradation of fines by incorporating flotation cells, and (iii) conversion of the two-product washeries into three-product ones with the completion of the balancing facilities contemplated, the washeries under the

Central Coal Washeries Organisation (CCWO) will improve the quality of the washed product and it may be possible to attain higher capacity utilization.

New Washeries

Out of the total estimated reserves of coking coal in India, at least 55 to 60% have difficult to very-difficult-washability characteristics. The reserves of easy-or moderately-easy washing coking coal are fast dwindling, and the requirement of coking coals for steel plants will have to be obtained from seams having inferior washability characteristics. As the dirt in these inferior coals is intimately mixed with the coal body, it requires crushing to the levels of 37, 25, 12 or 6 mm to enable the coal to liberate the inorganic matter for effective beneficiation for an economic yield. The mode of association of the dirt bands with the coal body decides the optimum size of crushing prior to washing.

It is evident, therefore, that the washing of coal in India has already started facing problems, and will gradually face more of them of the following nature :

— The entire ROM coal is required to be beneficiated in different sizes with emphasis on treatment of fines, and consequently a composite multistage washing technology is to be adopted.

— The dirty slurry (–0.5 mm fraction), of the order of 15 to 25% is to be beneficiated to improve its quality. This involves handling, recovery (by thickener-cum-filter process), de-watering, drying and mixing the filter cakes to clean.

— Due to the finer crushing, a large quantity of small coal (–12 mm) is produced and to dewater this, arrangements for large spin-drying mechanisms are to be installed.

A proposed composite multistage washing circuit incorporating the above is shown in Annexure 4. This is an elaboration of the standard circuit already suggested by CFRI, Dhanbad.

The main features of this washing circuit includes the use of HM cyclones as the main washer for the reasons stated below.

— cyclones can achieve a sharp separation at any specific gravity within the range normally required with coals of very-difficult-washability characteristics, i.e. where the ash-forming dirt is intergrown and large percentage of near gravity material is persent (even 70% and above).

— They can handle fluctuation in feed, bothfo quality and quantity.

— They can maintain the gravity of separation that can be controlled within a range of 0.005 specific gravity units.

Cyclone washers have been successfully employed in most of the washeries in India as a primary or secondary washer to treat coals with very-difficult-washability characteristics.

A provision for the installation of a jig has been made in the proposed washing circuit as a rewasher to separate the 'sweet' middlings within the ash range of 32 ± 3%, from the washery rejects.

In case the quantity of washery rejects is 40% and above, the jig can be installed as a pre-washer and rejects removed to avoid possible jamming of the cyclone underflow.

A provision for the beneficiation of –0.5 mm fraction of fines has been made by the installation of flotation cells/hydro-cyclones.

The –0.5 mm fines already separated may be mixed with the crushed reject, and upgraded by oil agglomeration and finally recovered by filtration as filter cakes.

The major steps in the process include :

— crushing the ROM coal in two stages to 25/13 mm size ;

— de-sliming the crushed product at 0.5 mm ;

— treatment of the deslimed product in HM cyclones to separate clean coal at 17% ash level ;

— de-pulping by rinsing the products, both overflow and underflow, of the cyclone ;

— rewashing underflow of the HM cylcones by a Baum type jig to separate 'sweet' middlings ;

— beneficiation/upgradation of the slimes (–0.5 mm fraction) by flotation cell/hydro-cyclones and recovery by thickener-filter process as filter cakes.

An alternative arrangement has been made to crush the rejects further to –100 microns, and again wash this product by the process of 'Oil Agglomeration.' This will help in reclaiming the optimum possible carbonaceous matter, and to reject mainly the shaly portion.

New Concepts

Due to intimate nature of association of minerals with the coal, conventional washing equipment like

A FLOW SHEET FOR A COMPOSITE COAL WASHERY

Baum jigs are generally not capable of efficient washing of Indian coals except for the purpose of prewashing or rewashing for the separation of extraneous dirt.

No new coal-cleaning equipment have been developed in the past few years, though many units have been modified and improved.

An improved type of jig termed as Batac jig is described below :

Batac Jig

The Batac jig manufactured by Humboldt Wedag of West Germany, is claimed to be an improvement to conventional Baum jig. Compared with the standard design, the Batac jig exhibits two fundamental differences.

(1) The water vibrations are initiated from a series of air chambers under the jig bed and not from the air chamber located by the side of the jig bed.

(2) The movement of water, unlike in the conventional jig, is regular all along the Batac jig bed.

The Batac jig, with its sensitive electronically controlled valves and discharge mechanism, reportedly performs an excellent separation even in the case of fine grained and difficult-to-wash ROM coal with high specific gravity material.

Its higher capacity, high efficacy and significant space saving qualities have already made this machine the standard coal-cleaning unit in West Germany.

If the high performance Batac jig can be useful in beneficiation of inferior grade Indian coals with difficult-washability characteristics, it will open a new era in the history of beneficiation of Indian coals.

Switching over to beneficiation by Batac jig will have the following far-reaching advantages.

— Washing by the heavy media process can be completely eliminated thereby doing away with the troublesome media preparation, storage and regeneration circuit.

— The washing circuit will be simple, and thereby the capital investment will be lesser.

— Due to the absence of abrasive magnetic pulp, wear and tear of the equipment and pipes will be less which will reduce the maintenance cost.

On the other hand, the Batac jig is basically the same as any other conventional jig as far as the principle of operation is concerned. For the purpose of separation, both utilises the falling velocity by the gravitational acceleration.

Batac jig cannot overcome the inherent limitations of a jig to become a versatile washer like the heavy media cyclone. Like a conventional jig, it also operates efficiently only at comparatively high specific gravities of separation in a condition of low near-gravity material at the point of separation,

An attempt has been made below to make a comparison of the type of coal which has been beneficiated by Batac jig in West Germany and the USA and the type of coal available in India for beneficiation.

The performance data of the jig of coal samples in West Germany are given in Tables A.1, A.2, A.3.

On analysing the above data, the following observations may be made.

(a) The near-gravity material ($\pm$0.1 sp gr fraction) is only 5.6 at the sp gr of cut 1.50.

(b) 45% by weight reports as cleans at a specific gravity of cut, 1.40, and 43.4% as rejects at sp gr of cut, 1.90.

The fraction between 1.40 and 1.90 sp gr of cut is only 10.5%.

Evidently, there exists a clear-cut difference between the carbonaceous matter and the impurities, i.e. the impurities present are not inextricably mixed with the coal body and therefore easily separable.

Lower wittaning 'B' seam (USA) raw coal crushed to minus 3/8 inch (3/8 X 28 mesh). (Also see Table B.1.)

	A	B	C
Dp – Separating density	1.50	1.65	2.00
Ep – Cart probable	0.075	0.092	0.140
I – Imperfection	0.15	0.14	0.14
Clean coal yield %	78.3	83.7	90.1
Clean coal ash %	5.72	6.95	6.64
Refuse ash %	53.3	62.8	72.7

On analysing the above data, it is observed that,

(a) The near-gravity material ($\pm$0.1 sp gr fraction) is only 11.61 & 7.13 at sp gr of cut 1.45 and 1.50 respectively.

(b) 69.2% by weight reports as clean at a sp gr of cut as low as 1.35 and 13.70% reports as reject at a clean cut of 1.80.

The fraction in between the range of 1.35 is only 17.1%.

On analysis of the data of coal from Greenwich Colliery (B.2) it is observed that :

(a) The separation gravity is 1.56 and above for the 3/4" & 28 mesh, 2.00 and above for –29 mesh fraction.

(b) The following reduction in ash% could be achieved :

Test 1 : Reduction from 30.16% in ROM to 8.38% in cleans, a reduction of 21.78% at sp gr of cut 1.81.

Case 2 : Reduction from 32.84% in ROM to 7.41% in cleans and a reduction of 25.43% at sp gr of cut 1.5.

Case 3 : Reduction from 30.64% in ROM to 7.50% in cleans and a reduction of 23.14% at sp gr of cut 1.585.

The above reduction of ash with such a high yield could only be achieved when the raw coal treated is easily washable, i.e. there is a clear demarcation between the coally substance and the impurities. In other words, the impurities are not intimately mixed with the carbonaceous matter and they are easily separable.

The washability characteristics of some of the difficult to-wash Indian coking coals collected from collieries of BCCL and ECL are detailed in Tables C 1, C 2, C 3 and C 4 in Annexure 2. The above four samples are four coking coals in the Jharia coalfields as well as blendable coals in Raniganj, with difficult washability characteristic.

On the basis of the above comparative studies made on the beneficiation of coal by Batac jig in West Germany and the USA and the washability characteristics of Indian coal, it may be observed that :

The coal characteristic, the mode of association of dirt with the coal body, is distinctly different in Indian coal from the coals from West Germany and USA.

The dirt is intimately disseminated throughout the coal body in Indian coals, whereas the dirt is easily separable in form of straight bands from the coals from West Germany and the USA.

Oil Agglomeration

This comparatively new process developed by the CFRI, is quite promising for upgrading inferior Indian coals with difficult-washability characteristics.

Laboratory tests conducted by CFRI with a number of lower seam coals containing upto 28 to 30% ash, have yielded 70 to 75% recovery of clean coal with 17 to 19% ash, while with the versatile 'cyclone-cum-floatation' process ,only about 35 to 40% recovery could have been achieved at the same ash level.

Bharat Coking Coal Limited, Dhanbad have installed a 2-tph plant at one of the collieries with the object of developing this process and the results so far obtained are reported to be encouraging.

On the commercial acceptability of the tests being conducted in this plant, this technique can be incorporated in the existing and the new washeries intended for treating low-grade coals and re-treating the middlings and re-treating rejects of the present washeries.

REFERENCES

1. Batac jig — A New Improved Baum Type Jig, For Cleaning Coarse and Fine Sizes of Coal. : *Raymond. — E Zimmerman,* Vice President, Paul Weir Co.
2. Greenwich Collieries : A Variation in the Art of Coal Washing by Jigs : *Charles E Tackett,* Manager, Pennsylvania Mines Corporation, (World Coal Vol. 4 No. 2, Feb. '78).
3. Studies on Coal Preparation in India : Central Fuel Research Institute Dhanbad.
4. Coal for Steel – An over-view : *P R Sinha,* General Manager, CCWO, Dhanbad.
5. ISI Bulletin : Vol. 21, No. 1, p. 5-9 & 15, 1969.

ANNEXURE 1

Sl. No.	Name of the washery	Name of the owner	Year of completion/commissioning	Nominal input in tonnes per hour	Capacity input in million t./year	Principal washing systems	Feed size in mm
1	2	3	4	5	6	7	8
Prime coking coal sector							
1.	Durgapur	HSL	1960	360	1.50	H M Washer (Drewboy) & Feldspar Jig	76.0
2.	Dugda-I	HSL	1961	600	2.40	H M Washer (Tromp) and Baum jig	76.0
3.	Dugda-II	HSL	1968	700	2.40	H M Cyclone & Hydro-cyclone	13.0
4.	Bhojudih	HSL	1962 (Exp. 1964)	500	2.00	H M Washer (Leebar) Baum h g & H M Cyclone	76.0
5.	Patherdih	HSL	1964	500	2.00	H M Washer (Barvoys) Baum jig & H M Cyclone	76.00
6.	Lodna	BCCL	1955	70	0.40	Feldspar jig (Acco)	13.0
7	Jamadoba	TISCO	1952	350	1.44	H M Washer (Chance) and H M Cyclone	76.0
8.	Chasnala	TISCO	1968	550	2.00	H M Washer (Leebar) and H M Cyclone	76.0
9	Durgapur	DPL	1967	360	1.35	H M Cyclone	13.0
Medium coking coal sector							
10.	Kargali	CCL	1958 (Exp 1966)	650	2.72	H M Washer (Wimco) Baum jig & H M Cyclone	76.0
11.	Kathara	CCL	1969	800	3.00	H M Washer (Drewboy) Cyclone & Flotation	76.0
12.	Swang	CCL	1970	250	1.00	H M Cyclone & Hydro-cyclone	20.0
13.	Gidi	CCL	1970	800	2.84	H M Washer & Baum jig	150.0
14	West Bokaro	TISCO	1951 (Exp. 1973)	160	0.57	H M Washer (Chance) and H M Cyclone	76.0

ANNEXURE 2

TABLE A-1

Batac Jig Performance On Coal From Sachsen Colliery 'A' West Germany

Float and sink analysis crushed to 80 mm

Sq. gr. of cut	Weight %	Cumulative weight %
— 1.40	45.1	45.1
1.40 — 1.50	2.7	47.8
1.50 — 1.60	2.9	50.7
1.60 — 1.80	5.1	55.8
1.80 — 1.90	0.8	56.6
1.90	43.4	100.00
	100.0	

TABLE A-2

Guaranteed and Test Analysis of Products
(Batac Jig Performance)

Sp gr	Clean coal %		Middlings %		Refuse %	
	Guaranteed	Test	Guaranteed	Test	Guaranteed	Test
— 1.45	96.9	97.5	14.5	12.2	—	—
1.45 — 1.80	3.0	2.5	66.5	77.0	1.0	1.2
+ 1.80	0.1	—	19.0	10.8	99.0	98.8

TABLE A-3

Guaranteed and Test Results of Probable and Imperfection
(Batac Jig Performance)

	Clean/middlings		Refuse/middlings	
	Guaranteed	Test	Guaranteed	Test
Sp gr (Dp)	1.46	1.458	1.37	1.866
Ecart probable (Ep)	0.049	0.045	0.096	0.087
Imperfection (I)	0.107	0.098	0.100	0.100

TABLE B-1

Batac Jig Performance on Raw Coal from Lower Kittaning 'B' Seam, Pennsylvania, USA
(Crushed to 3/8" X 28 mesh)

Sp gr	Direct		Cumulative	
	Wt %	Ash %	Wt %	Ash %
1.35	69.20	4.08	69.20	4.08
1.40	6.21	12.94	75.41	4.51
1.45	2.42	17.92	77.83	5.21
1.50	1.56	21.50	79.39	5.53
1.55	1.42	25.66	80.81	5.88
1.60	1.73	30.89	82.54	6.41
1.70	2.18	38.06	84.72	7.22
1.80	1.58	44.21	86.30	7.99
+1.80	13.70	67.18	100.00	16.02

TABLE B-2

Batac Jig Performance on Coal From Greenwich Collieries
(Performance Test Data Summary)

Description	Test-1	Test-2	Test-3	Test-4	Test-5	Test-6
Separating gravity (Dp)	1.81	1.85	2.00	2.07	1.585	1.56
Actual Ep	0.097	0.083	0.190	0 185	0.080	0.072
Guaranteed Ep	0.113	0.118	0.254	0.274	0.085	0.082
Imperfection	0.120	0.097	0.190	0.173	0.136	0.129
Per cent yield	69.60	84.80	86 55	90.91	67.98	63.85
Raw coal ash (%)	30.16	19.82	18.16	16.05	30.64	32 84
Clean coal ash %	8.38	8.21	8.53	9.33	7.50	7.41
Refuse ash (%)	80.0	84.27	80.18	77.55	79.77	77.75

TABLE C-1

Washability Characteristic of V, VI, VII Seams Combined from Western Sector of Jharia Coalfield (Barora Colliery)
(Crushed to — 13*mm*)

Size (mm)	Cut at sp gravity	Weight %	Ash %	NGM*	Weight %	Total (ROM) Ash %
— 13 to 0.5	— 1.3	4.3	4.6			
	1.4	21.5	10.5	3 1.5		
	1.5	35.8	14.7	29.3		
	1.6	50.8	18.5	34.7		
	1.7	70.5	23.2			
	1.8	85.8	26.9			
	1.8	14.2	55.0		97.3	30.9
— 0.5	—	—	—		2.7	23.4
					100	30.6

TABLE C-2

Washability Characteristic of V, VI, VII Combined Seam from Eastern Sector of Jharia Coalfield (Ghanoodih Colliery)
(Crushed to — 13 mm)

Size (mm)	Cut at sp. gr.	Weihgt %	NGM	Ash %	Total Wt %	(ROM) Ash %
— 13 to 0.5	— 1.3	2.9		5.3		
	1.4	14.3	36.6	13.0		
	1.5	39.5	60.2	19.5		
	1.6	74.5		24.9		
	1.7	89.5		27.4		
	1.8	95.0		28.5		31.6
	1.8	5.0		62.2	91.8	
— 0.5	—	—	—	—	8.2	20.4
					100	30.6

* NGM — Near Gravity Material.

TABLE C-3

Washability Characteristic of Coal from Laikdih Seam Area No. III, BCCL, Jhunkundar Colliery
(Crushed to — 13 mm)

Size (mm)	Cut at sp. gr.	Weight %	Ash %	Total Wt %	ROM Ash %
— 13 to 0.5	— 1.30	7.55	6.81	92.25	22.3
	1.35				
	1.40	44.30	13.13		
	1.45				
	1.50	75.45	16.98		
	1.55				
	1.60	85.15	18.63		
	1.70	94.0	20.16		
	1.80	95.65	20.62		
	1.80	4.35	59.00		
— 0.5	—	—	—	7.75	21.03
				100	22.2

TABLE C-4

Washability Characteristic of Coal from Ha'nal Seam (Sodepur Colliery) Area No. 1, ECL
(Crushed to — 75 mm)

Size (mm)	Cut at sp. gr.	Weight %	Ash %	Total Wt %	ROM Ash %
— 75 to 0.5	1.3	2.5	3.6		
	1.4	.75	14.5	98 1	25.8
	1.5	74.9	17.7		
	1.6	82.9	19.2		
	1.7	87.3	20.4		
	1.8	90.1	21.3		
	1.8	9.9	66.5		
— 0.5	—	—	—	1.9	19.3
				100.0	25.6

SCIENCE & TECHNOLOGY PROJECTS

Four science and technology projects were completed during the year 1979-80 funded by S & T grants of Department of Coal.

● Amlabad Degasification

To raise 2000 tonnes of coal per day from one of the gassiest seams of this country, help of Soframines (F) was sought in investigating the problem and suggesting necessary measures to deal with the same. A total expenditure of Rs 16.56 lakhs was incurred on the project. A degasification and Mining Scheme has been prepared.

● Pipeline Transport

A state of art report was prepared for transport of coal as well as sand. The report reveals that pipeline transport of coal is economical with higher throughput and longer transport distance. The system presupposes the ability of the consumer to use high moisture coal (9 to 10%). The pipeline transport of sand is not an economical proposition due to very high rate of pipe wear.

A sum of Rs 1 lakh was spent on the project.

● Chemical Eradication of Green Growth

The investigation report suggests the use of chemicals like 'Grammoxone' and 'Fernoxone' for eradication of weed in preference to manual cutting. Cost of chemical spray has been estimated below 20 paise/ m^2 spray. A sum of Rs 10,000/- was spent on this project.

● Hot Briquetting

The project was under taken with a view to promote the use of non-coking coal or weakly-coking coal for production of formed coke. The project was, however, deferred as the hot briquetting plant of British Steel Corporation at Scunthorpe was reported to be facing lot of technical problems.

A total sum of Rs 1.12 lakhs was spent towards testing charges for coal and other associated work.

SINKING OF SHAFTS BY DRILLING & WIDENING TECHNIQUE

A number of existing mines are to be reconstructed to increase the production capacity. Some of these schemes involve sinking of new shafts or the reconstruction of existing shafts. It is anticipated that work of sinking of about 30 new shafts and reconstruction of 20 existing shafts is to be taken up by Coal India during the next 5 years.

The sinking technology adopted hitherto was rather labour intensive resulting in a slow rate of progress. For a quick development of new coal mining projects, it is necessary to achieve faster rates of shaft sinking by adopting suitable mechanisation.

Most of the equipment required for mechanised shaft sinking are not indigenously available. There is also an acute shortage of trained manpower experienced in mechanised shaft sinking operations. In view of this situation, sinking of shafts is proving to be a major hindrance in the development of new coal mining projects. The situation regarding drift drivage is more satisfactory. The basic equipment for the conventional mechanised drivage of drifts is already available in India. The drifting operation is simpler compared to shaft sinking and is similar in many respects to the development of coal galleries. Trained manpower, therefore, is not as scarce as in the case of mechanised shaft sinking.

Recently, a drilling technique for drivage of staple pits and raises has been developed. In this, first a small diameter hole is drilled from the workings in the upper level to the workings in the lower level. Subsequently, the borehole is widened from the lower level upwards, when the rock cuttings fall down to the lower level and are disposed of through the existing workings. Similar methods can also be adopted for establishing a new shaft opening from existing underground workings. With this technique, large diameter shafts can also be driven. While drilling there is a possibility of deviation of the bore-hole, and suitable precaution may be taken against it.

New shaft sinking and shaft reconstruction are to be shortly taken up in the Jharia and Raniganj coalfields. In most of the shaft sinking sites, mining activity is already going on in the upper seams, and the new or reconstructed shafts are proposed for working the deeper seams. In such situations, shaft sinking could be carried out by a combination of drilling and widening technique. This broadly consists of following operations and illustrated in the drawing (Fig. 1).

(i) Drivage of a narrow drift from the existing workings in the upper seam to the lower seam and upto the proposed location of the shaft ;

(ii) With the help of a raise-boring machine, boring of a small diameter hole from surface upto the lower seam at the proposed site of the shaft ;

(iii) Reaming of the borehole to a diameter of 1.2 m in the upward direction allowing the rock cuttings to fall down. The rock cuttings are cleared through the drift and upper seam workings ; and

(iv) Widening and lining of the shaft from the surface to desired diameter downwards and handling the debris in the same manner as in case of (iii) above.

While widening the shaft, the muck as well as the water will be allowed to go down through the 1.2 m diameter borehole and, therefore, the widening and lining can be done at a faster rate than conventionally done. Any deviation of the borehole if small, can be corrected while widening the shaft to the desired diameter.

For the purpose of techno-economic analysis, estimates for the conventional and the combined drilling and widening techniques have been compared for a specific case. The general parameters of the shaft and a comparison between the two techniques are indicated below :

TABLE 1

Sinking Parameter

Depth of proposed shaft	220 m
Finished diameter of the proposed shaft	6.5 m
Present depth of workings	180 m
Number of insets/openings in the shaft	4
Type of outfitting	With rigid guides
Length of the drift to reach 220 m level from the 180 m level	225 m

TABLE 2

Comparison of Salient Aspects of the two Systems

	Conventional method of sinking	Sinking by drilling and widening technique
Capital required for plant and equipment	Rs 1.05 crores	Rs 1.35 crores
Time required for completion of the shaft including insets & outfittings	24.75 months	22 months
Expenditure on shaft sinking, inset opening and out fitting	Rs 1.62 crores	Rs 1.32 crores

In case the level difference between the existing workings and the proposed shaft bottom is greater, the expenditure for driving the drift as well as handling the muck will increase. If two new shafts are to be developed adjacent to each other, the expenditure on the development of drift could be apportioned amongst the shafts. In case of skip-winding shafts a portion of the temporary drift could be subsequently utilised for the purposes of cleaning the shaft bottom as well as for providing access to the skip loading level. This technique of drilling and widening of shafts will also be ideally suited for development of peripheral ventilation shafts or stowing shafts for the existing mines.

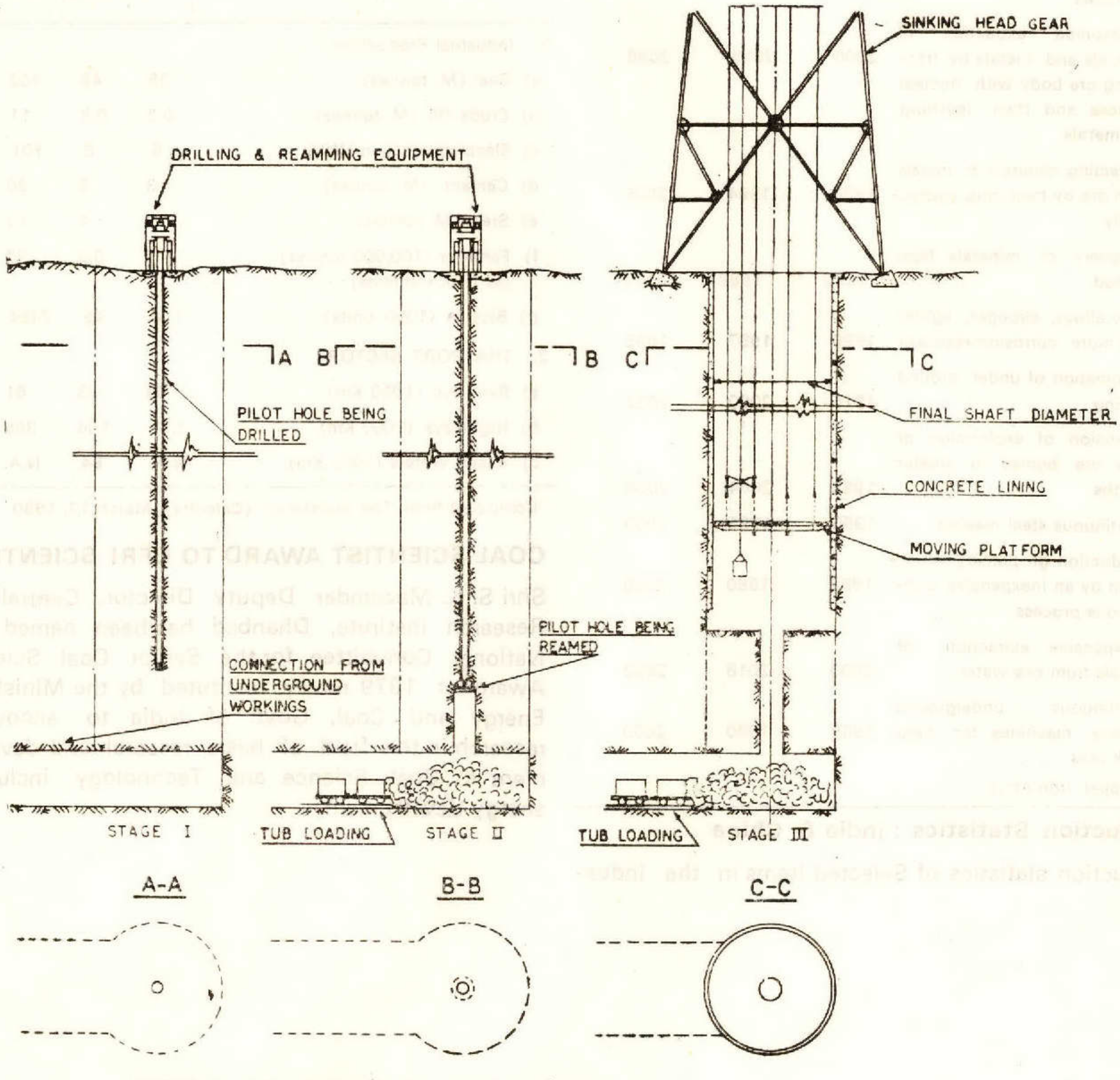
From the techno-economic comparison, it would be observed that the difference both in the time and cost frame is only marginal.

In USA the technology for blind hole drivage of shafts of 7-8 m diameter is being developed. Similar technique is also being adopted in Canada. With this technique 7.5-12 m of lined shaft is being completed every day.

Shaft Design Wing, CMPDI

Technological Breakthrough in R & D

The biennial survey 'Technological Breakthrough and Widespread Application' conducted by the Mc-Graw-Hill Co., USA, published recently, shows that most of the time tables have been moved into future by five to fifteen years projected before. The delay is attributed in part to fading R & D expenditure. The survey is based on data supplied by more than 200 firms and researchers and the prognostications cover fields ranging from aerospace, chemicals, electronics and medicine to mining and metallurgy. The target dates correspond to the logical stages in the development of new technologies-breakthrough, economic feasibility and widespread application. Some of them concerning mining and metals industries are listed below :



SCHEME FOR SINKING BY RAISE BORING

NOT TO SCALE

Technology	Year of Break through	Year when economically feasible	Year when in wide spread application
1. Use of Lasers ultrasonics and high-frequency currents for drilling, crushiog and grinding rocks	1980	1992	2005
2. Widespread extraction of minerals and metals by fracturing ore body with nuclear devices and then leaching the metals	2000	2015	2038
3. Extracting minerals & metals from ore by fracturing hydraulically	1990	1994	2005
4. Recovery of minerals from seabed	1982	1988	2000
5. New alloys, stronger, lighter and more corrosion-resistant	1980	1987	1995
6. Automation of under ground mining	1915	2000	2032
7. Extension of exploration of new ore bodies to greater depths	1996	2004	2008
8. Continuous steel making	1990	1995	2000
9. Production gf primary Aluminium by an inexpensive continuous process	1990	1980	2008
10. Inexpensive extraction of metals from sea water	2000	2018	2032
11. Continuous underground mining machines for hard rock ores (copper, iron etc.)	1985	1990	2000

Production Statistics : India & China

Production statistics of Selected Items in the Industrial and Transport Sectors of India and China in 1950 and 1978.

Source : Mc Qraw-Hill Publications Co., Economics Deptt.

I t e m (Units)	1950		1978	
	India	China	India	China
1. Industrial Produdtion				
a) Coal (M. tonnes)	35	49	102	618
b) Crude Oil (M. tonnes)	0.3	0.3	11	104
c) Electricity (billion kWh)	6	6	101	257
d) Cement (M. tonnes)	3	2	20	65
e) Steel (M. tonnes)	3	1	10	32
f) Fertilizer (100,000 tonnes) (of NPK nutrients)	0.2	0.2	27	87
g) Bicycle (1000 units)	114	45	2464	8540
2. TRASPORT SECTOR				
a) Railways (1000 Km)	54	23	61	59
b) Highways (1000 Km)	N.A.	104	395	890
c) Inland Waters (1000 Km)	N.A.	84	N.A.	136

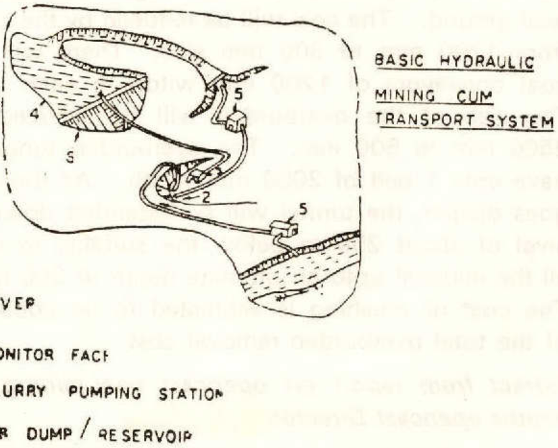
Computed from 'The Statesman' (Calcutta), March 17, 1980

COAL SCIENTIST AWARD TO CFRI SCIENTIST

Shri S. K. Mazumdar Deputy Director, Central Fuel Research Instirute, Dhanbad has been named by a National Committee for the Senior Coal Scientist Award for 1979 newly instituted by the Ministry of Energy and Coal, Govt. of India to encourage research in the field of basic research and development in Coal Science and Technology including energy develepment.

Hydraulicking in Openpits

Depending on the nature of the top overburden and topography of the surface, hydraulicking can be easily adopted either after side-casting by dragline or after drilling and blasting. After scraping the top soil and stacking separately for future reclamation, hydraulicking is done in benches of 20-25 m height. In the next hard formation, it will be still possible to do hydraulicking after suitable drilling and blasting. Figure 1 gives a typical hydraulic mining-cum-transport system and figures 2 and 3 show the two different applications.



6 - RIVER

2 - MONITOR FACE

3 - SLURRY PUMPING STATION

4 - O B DUMP / RESERVOIR

5 - WATER PUMPING STATIONS

6 - EMBANKMENT

Fig.1

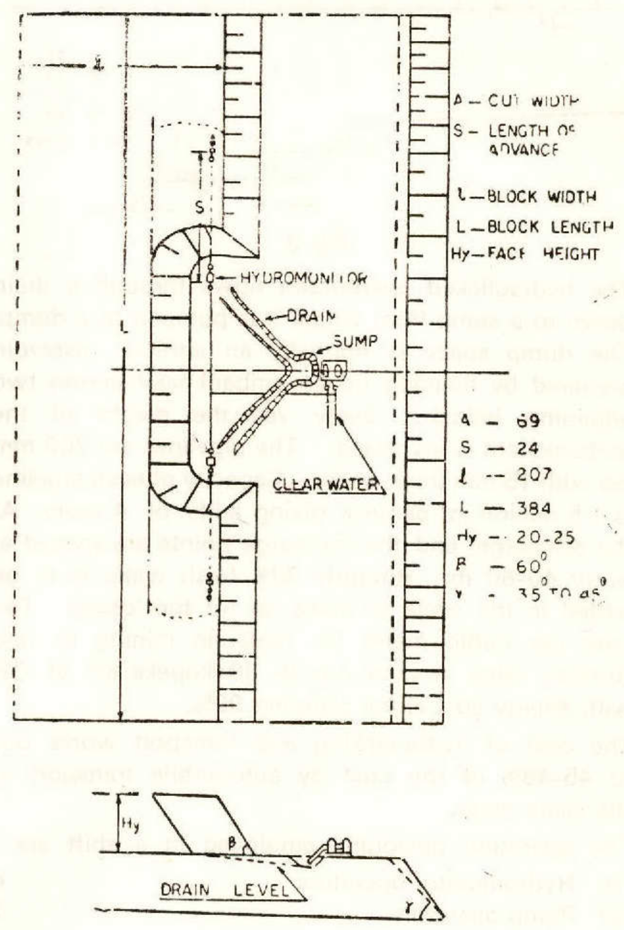


Fig. 2

Hydromonitors use 10-20 atmosphere nozzle pressures under various strata conditions. Table 1 gives the nozzle pressure, and diameter, slurry/solid percentage and water consumption.

Face Height — 20 m							
Soil category	I	II	III	IV	V	VI	VII
Nozzle pressure in 10 atmos							
(a) GMD-250	1	1.6	1.7	1.8	1.9	1.9	2
(b) TMH-350	1.4	1.4	1.6	1.6	1.7	1.7	1.8
Nozzle diameter (M)							
(a)	0.125	0.110	0.110	0.110	0.110	0.110	0.110
(b)	0.160	0.160	0.160	0.169	0.160	0.155	0.155
						— 0.160	—0.160
Ratio of Dry wt / Wet wt	0.35	0.35	0.35	0.35	0.40	0.40	0.35
Water consumption m³ /m³	3.46	3.65	4.5	5.6	7.1	7.9	11.1

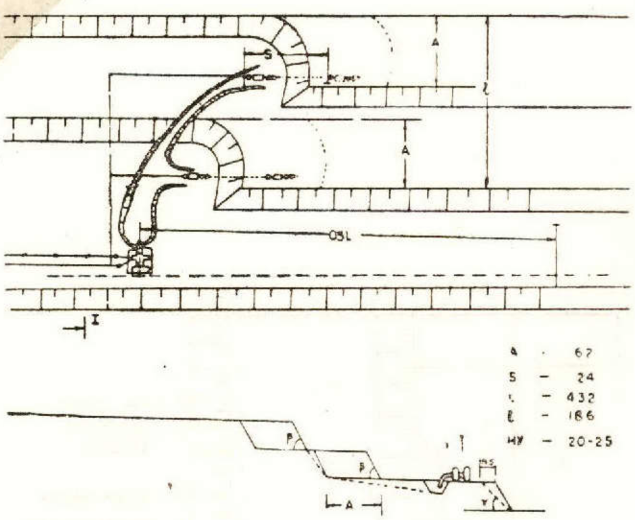


Fig 3

The hydraulicked overburden flows through a drain down to a sump from where it is pumped to a dump. The dump space is normally an artificial reservoir, prepared by building up an embankment across two adjoining hillocks. Every year the height of the embankment is increased. The pipelines are 700 mm dia with 15 mm thick walls. Capacity of each pipeline is 1.5 million m^3 per year giving a life of 4 years. At the discharge end the discharge points are spaced at every 40-60 m. Roughly 30% fresh water is to be added in the cycle to make up for the losses. The cost per cubic metre for hydaulic mining in one Russian mine worked out to 60 kopeks/m^3 of OB with energy cost alone claiming 90%.

The cost of hydraulicking and transport works out to 45-48% of the cost by automobile transport in the same mine.

The operating personnel employed in a shift are :

(i)	Hydromonitor operators	4
(ii)	Pump operators	7
(iii)	Attendant at embankment	1
(iv)	Supervisor	1
		13

In addition to this, there is a repair brigade of 10 persons, 2 engineers and a crane operator in general shift.

The slurry pumps have capacities varying from 4000 to 6500 m^3/hr and heads varying from 65 to 90 m. The clear water pumps are 3000 m^3/hr capacity with 197 m head.

Hydraulicking is done during non-winter months of April to October. The following are the figures for a typical monitor operation in Cat. II soil.

Monitor type		GMD-250
Production per hour	m^3	390
Volume of water per hour	m^3	1760
Volume of clay worked from one position of monitor	m^3	32000
Time for above (working hours)		82

A Technological Solution for opencast pit of 120 metres depth.

In one of the opencast mines, October-50, in USSR the present depth of the quarry is 120 metres and the coal seams are dipping at 75° to the horizontal. In the reconstruction programme for this mine, it has been proposed to install a crusher within the quarry at about 150 metres below the surface, for crushing the overburden and coal before transporting them out by belt conveyors through tunnels inclined at 15° to the horizontal through high wall and foot wall ground. The coal will be reduced by the crusher from 1200 mm to 300 mm size. There will be 2 coal conveyors of 1200 mm witdh in each tunnel. The size of the overburden will be reduced from 2500 mm to 500 mm. The overburden tunnel will have only 1 belt of 2000 mm width. As the quarry goes deeper, the tunnel will be extended down to a level of about 250 m below the surface, to handle all the material upto an ultimate depth of 300 metres. The cost of crushing is estimated to be about 15% of the total overburden removal cost.

Extract from report on opencast coal mining grater depths opencast division, CMPDIL.

Blasting Problems in Golukdih opencast, BCCL

Recently, a round of deep shotholes (12) were blasted in the third OB Bench over developed X seam workings of Golukdih opencast project (BCCL). A blown-out shot and the collapse of the parting into the coal gallery beneath resulted in a coal dust explosion. The X seam in this property is about 20 m thick with two sections developed, in bord and pillar method, one along the roof of the seam and the other along the floor leaving a parting of 13.5 m. The overburden is 30 m thick. Mechanised opencast mining has been resorted to win the coal in both the sections. There are three overburden benches and the coal pillars are attacked from the top to bottom.

On the day of blasting, the round in the third OB bench consisted of 12 holes of depths ranging from 9.0 to

12.6 m, burden of 5 m and spacing of 7 m. One pilot hole was made to test the thickness of the parting which was 3 m. The hole was later plugged and blasted alongwith the other 11 shotholes. The average charge per hole was 179 kg Supergel and Aquadyne were used. Soon after the blasting, large quantities of gases and flames were witnessed coming out of the developed pillars in both top and bottom section workings. The flames coming out of the galleries burnt bushes upto a distance of 45 metres in the opencast pit. Brown fumes were observed coming out from the two pot holes created in the floor of 2nd bench. Evidence of coal dust explosion was seen by charred coal dust on the side of galleries and timber props.

Since there was no serious damage to the stoppings and the supports etc., it was inferred that the explosive pressure developed was low. The parting being 3 to 4 m against the burden of 5 m, might have yielded and also the plugged hole had blown out allowing the hot gas and flame to pass through the underground galleries. The fragmentation was satsifactory.

The Explosives Utilisation Wing of CMPDI is now engaged in fixing the controlled blasting parameters for this quarry. A number of precautions have been suggested for dealing with such problems in future. The important ones are :

(a) The spacing of the holes in the bench lying immediately above the coal seam shall be so adjusted as far as possible that the blast holes do not come over the underground galleries. This will require accurate surveying of underground galleries and correlation with surface.

(b) The depth of the holes in the last OB bench shall be such as to leave at least 2.0 m thickness of OB above the coal seam. The pilot hole put for this purpose shall not be charged, and left blank.

(c) No delay action detonator in coal shall be used unless permitted by DGMS in writing.

(d) The shot holes in coal shall not be drilled within 3 m of the lower section.

(e) The bottom of all holes in the last OB bench and/or in coal shall be filled with water ampoules or moist sand for at least 0.6 m length.

(f) When underground galleries are inter-connected with quarry operations, it will be necessary to withdraw the work-persons from the mine at the time of blasting.

Energy Position in the Developing World

In half of the world's developing nations whose population amounts to about 22 % of the Third World, two thirds of the primary fuel consumption is supplied by biomass. Indonesia, Nigeria, Vietnam, Burma, Ethiopia and Bangladesh constitute this group. Seventeen countries led by India, Pakistan, Phillipines, Thailand and Morocco are in a group where the combustion of fossil and biomass fuels is about equal contain 30% of the Developing World's population. About 17 developing nations rely on traditional fuels for a quarter to third of their primary needs. Brazil, Columbia, Algeria, Turkey are important countries in this group. Only 10 % of the Third World's population including Mexico, South Korea, Iran, Argentina and Taiwan live in some 30 countries where the transition from biomass to fossil fuels has been virtually completed and where traditional energies provide less than 10% of the nation-wide consumption.

Traditional Energies in K cal/Capita/Day

Country	Fuel wood	Crop by products	Dung	Total	Modern energies
1	2	3	4	5	6
Bangladesh	200	1900	1100	3200	500
Nigeria	7000	600	—	7600	1700
Indonesia	6500	200	—	6700	3400
India	2400	500	1000	3900	4200
Brazil	5800	1100	—	6900	12800
Turkey	2600	1300	2000	5900	12100
China	1100	2000	—	3100	8300
Taiwan	100	200	—	300	29200

(Energy International Dec. '79, p. 29)

Annual consumption of biomass energies in the developing world, on a rough count use firewood over 7×10^{15} K cal, crop residues nearly 1.5×10^{15} K cal and dung 0.4×10^{15} K cal. These three in aggregate account for about 1.3 billion tonnes of hard coal or approximately 900 million tonnes of crude oil. As per UN statistics, developing countries consume annually just over 1 billion tonne of coal equivalent in coal, oil and natural gas.

Research into the efficient use of renewable energy resources is a long and costly process, and best affordable by the richest countries. In the Third World, where the extensive use of biomass impedes economic development, the need is for an expanded recovery of fossil fuels and large scale harnessing of hydropower.

Some Aspects of Vertical Hydraulic Conveying of Large Coal from a Depth of 850 m

D Jordan

All the Hansa hydro-pit, coarse-grain raw coal up to 60 mm max. grain has been brought to the surface hydraulically over a difference of level of 850 m since September 1977. Lifting is done via a tubular sluice, the so-called 3-chamber pipe-feeder, by means of which up to 400 t/h can be brought to the surface in one stage. Once the work of adjustment has been completed the 3-chamber pipe-feeder operates faultlessly and has so far shown itself to be a hoisting installation that is efficient and reliable. Vertical hoisting can be carried out by water pressure from below ground, from the surface or in a mixed operation of from below ground and from the surface. Existing water inflows, that would otherwise have to be brought to the surface by a separate pumping action, can be utilised for the lifting operation. The energy costs per ton of raw coal for a lifting height of 850 m vary between 0.84 DM and 1.32 DM and therefore lie within the range of the costs wich also occur with conventional hoisting.

At the Hansa hydro-mine, coal has been hydro-mechanically extracted and hydraulically hoisted from steep formations since September, 1977. At the present time, the plant is in the build-up phase, on the way to achieving the planned production of 3500 tons of saleable coal, 5000 tons of raw coal and at the moment is producing for up to 12 hours a day.

It is still too early to give an overall verdict on vertical hydraulic hoisting. The daily period of operation is still too short, because the capacity has not yet been fully tested. All the same, results so far show that the system works reliably and to some extent far exceeds the planned performance data. Over and above this, some interesting and in part unexpected operational results have been produced, which have had a very positive effect on the profitability of vertical hoisting. However, before going into all this in detail I must first explain the essential characteristics and the technical relationships of the process.

Mining Commissioner, Siemag Transplan GmbH. *By kind courtesy, the paper was earlier read at a Symposium of the South African Institute of Mining & Metallurgy & printed in Coal International Supplement, April 1979.*

The Technical Relationships of the Process

From the complex of tasks which the Hansa hydro-mine has to fulfil, one can distinguish 5 areas :

(a) Supply of spraying water for the hydromechanical extraction ;

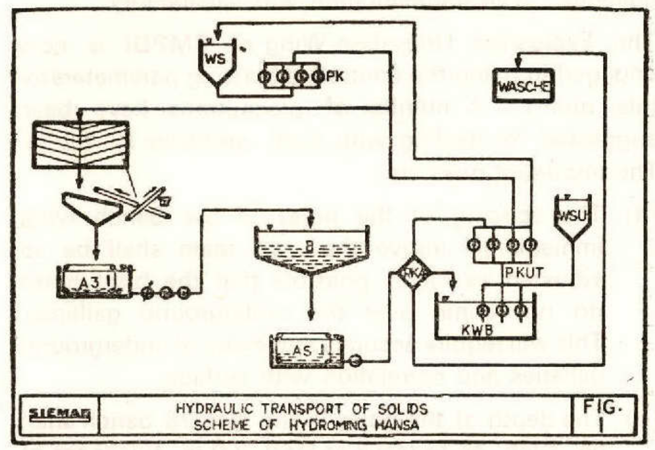
(b) Horizontal hydraulic conveying underground ;

(c) Vertical hydraulic hoisting to the surface ;

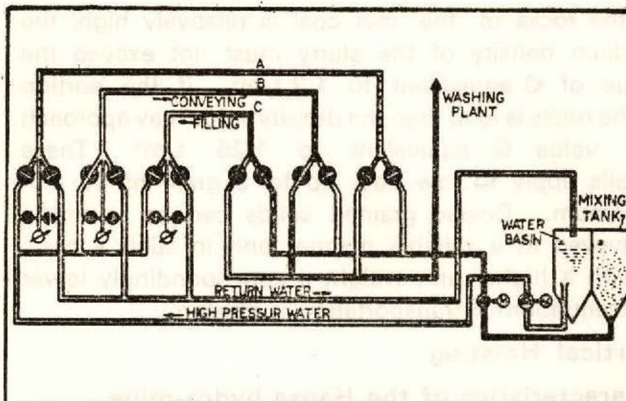
(d) Water clarification underground ; and

(e) Preparation in the washing plant on the surface.

Fig. 1 shows in simplified form the technical relationships of the process. We see the two pump chambers,



one on the surface and one underground, each equipped with 4 high-pressure pumps. The pumps supply the workings with spraying water and ensure the vertical hydraulic hoisting. The raw coal released with the water jets flows in channels with the spraying water that now serves as transport medium, through a combined screen-crusher to separate out grains over 60 mm and into the feeding basin, whence it is conducted to the feeding pumps, connected in series. From here, the slurry proceeds via a roughly 3 km long horizontal pipe into one of the raw coal basins at the shaft. Finally, the coal is drawn out of the raw coal basin into a feeding basin, sluiced into the 3-chamber pipefeeder and forced in an uninterruped flow 850 m to the surface.



SIEMAG HYDRAULIC TRANSPORT 3-CHAMBER PIPE-FEEDER FIG. 2

Not shown on the diagrammatic representation are the secondary circuits and the water clarification underground. In the rest of this article, only the vertical hydraulic hoisting to the surface will be reported on.

The 3-Chamber Pipefeeder

The heart of the vertical hoisting

The process of hydromechanical extraction with water jets was tried and tested many years ago and the winning methods have been employed successfully, inter alia, in Canada and Russia. In Germany, too, many tests were run at a series of pits, in which the process proved itself. The problem with German conditions, however, lay hitherto in how to hoist by hydraulic means the coarse grain coal with its up to 40% proportion of waste over great differences in level. Here, all previous attempts to overcome great hoisting heights with reciprocating pumps or several channel gear pumps connected in series foundered for technical or economic reasons.

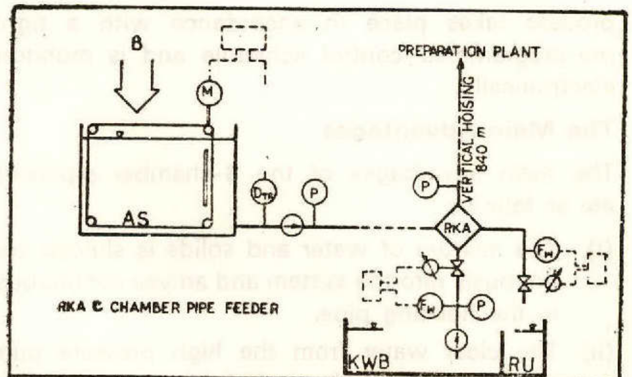
The tasks set before the Hansa mine were :

(i) The overcoming of a difference of level of 850 m corresponding to the depth of this mine ;

(ii) Coarse grain hoisting of the valuable coking coal, as far as possible, i.e. maximal grain diameter up to 60 mm, in order to keep preparation and dewatering costs within bounds ;

(iii) The economic optimisation of vertical hoisting, i.e. hydraulic hoisting with minimum water.

With today's state of the art, these requirements can be met only with a pipe sluice, the 3-chamber pipe-feeder (Fig. 2).



SIEMAG HYDRAULIC TRANSPORT OF SOLIDS SCOPE OF CONTROL FOR THE VERTICAL HOISTING FIG. 3

Mode of Operation

The mode of operation of the 3-chamber pipefeeder will be explained briefly with the aid of the diagrams. Fig. 2 shows the slurry basin with the water/solids mixture, the clear water tank, the two high and low pressure pumps, together with the pipefeeder and its 3 pipe-chambers, each 350 m in length. While chamber C in this case is filled with coal slurry by the low pressure pump under 3-5 bar, chamber B, which was already filled in an earlier cycle, is now emptied under high pressure. At this, the high pressure pumps clear water in the opposite direction into the chamber and forces the slurry that has filled up there into the vertical hoist pipe. This work cycle, depending on hoisting speed and the length of the chambers, lasts about 1-2 minutes; at Hansa the work cycle is approximately 80 seconds.

The third chamber, in this case chamber A, is in a rest position. It is already filled with slurry and is still under low pressure. The master valves on the inlet and outlet side are shut. Before the start of the hoisting process, the equalising valve to the pressure pipe opens and builds up the requisite hoisting pressure in the chamber. Now the master valves (closed at the moment) can open — they form the link between the high pressure pump and the hoisting pipe — and the hoisting process begins.

At this point, chamber B has finished the hoisting process and the master valves on the inlet and outlet side close. Then the equalising valve opens that forms the link with the low pressure pipe and reduces the high pressure in the chamber again. The chamber is relieved of pressure and then stands ready for the next filling process. The entire filling and hoisting process takes place in accordance with a tightly pre-programmed control schedule and is monitored electronically.

The Main Advantages

The main advantages of the 3-chamber pipefeeder are as follows :

(i) The mixture of water and solids is sluiced continuously into the system and arrives continuously in the hoisting pipe.

(ii) The clear water from the high pressure pump arrives, almost in its entirety, back in the return flow tank after the hoisting process and is ready to act again as the high-pressure water supply.

(iii) The solids do not come into contact with the sensitive high pressure pump, so that multistage centrifugal pumps with high degrees of efficiency can be utilised.

(iv) Wear-resistant plate valves are used.

(v) The system is so controlled that the high pressure valves are moved only in the clear water stream.

Regulating Density

Reliable hoisting of solids with minimum water can be maintained only when, in proportioning the solids, an optimal mixture in the hoisting pipe results and blockages are safely avoided. The solution chosen at the Hansa mine is represented diagrammatically in Fig. 3. The slurry feed velocity is measured in the return flow of clear water of the pipe-chamber feeder. The velocity at this point is the same as in the feeding pipe, therefore this is a good place to install the gauge and so protect it against wear. The high-pressure water velocity in the high pressure pipe is recorded immediately before the pipe feeder. The density gauge is fitted on the suction side of the feed pump and gives the input values for regulating the speed of the scraper conveyor and thus for controlling the amount of solids. The regulation of density occurs automatically in accordance with a pre-set value.

Density Values for Coal/Rocks/Water Mixtures

In the case of a three-substance mixture, the density value presents an unequivocal indication regarding the solid content of the slurry only in such a case, if the relation of the volume streams of coal and rocks remains constant. In practice, the maximum portion of the raw coal is subject to considerable variations caused by the seams being worked. If the portion of the rocks in the raw coal is relatively high, the medium density of the slurry must not exceed the value of G equivalent to 1.2 t/m^3. If the portion of the rocks is low, then the density value may approach the value G equivalent to 1.25 t/m^3. These details apply to raw coal up to a grain of 60 mm maximum. Coarse grained solids can be vertically conveyed in a reliable manner only in such a case, if with a higher unit weight a correspondingly lower concentration of transportation is chosen.

Vertical Hoisting

Characteristics of the Hansa hydro-mine

In Table 1 are shown the most important characteristics of the vertical hoisting at the Hansa hydro-mine. The plant is organised for a planned daily output of 5000 tons of raw coal (corresponding to a saleable output of 3500 tons), which is hoisted 850 m to the surface in one stage, with a maximal grain of 60 mm.

TABLE 1

Hydraulic Transport of Solids — Characteristics of the Vertical Hoisting Process at the Hansa Mine

1. General characteristics	
Hoisting height	850 m
Projected capacity/day	5,000 t coal
Projected capacity/h	250 t coal
Content of rocks	30% weights
Max. diameter of solids	60 mm
Pipe diameter	250 mm
2. Characteristics of the 3-chamber pipefeeder	
Length of pipe-chambers	356 m
Diameter of pipe-chambers	250 mm
Number of hydraulic valves	12 pieces
Number of compensation valves	6 pieces
Cycle time of chamber	80 s
Feeding velocity	4.5 m/s
Hoisting velocity	4.8 m/s
3. Characteristics of high pressure pumps and electric motors	
Underground	425 m^3/h
	120 bar
Power (inst.)	2,100 kW
Surface	425 m^3/h
	40 bar
Power (inst.)	800 kW

In organising the plant, the starting point was in average transport concentration (delivered concentration by volume) of some 20%, which is equal to an hourly output of 250 tons. Experience has shown that, even with the relatively large grain size of 60 mm, hoisting can be accomplished with higher transport concentrations. As can be seen from Table 2, the highest value so far attained stands at 34% transport concentration, corresponding to an out put of 458 tons per hour. Thus this peak value is more than 80% higher than the planned value of 250 tons. Today, the hourly output stands, as a rule, at over 300 tons per hour and has settled at something like 330-340 tons per hour.

As Table 2 also shows, the proportion of waste is way above the initially expected and therefore planned value of more than 33% and in fact, brought the average density of solids to a value of 1.64; however, because of the lower transport concentration of 20%, the value for the average density of slurry at 1.128 did not vary substantially from the planned value.

From this production derive the data in Table 4. The most significant results here are the transport concentration, standing at 25%, which corresponds to a water/solids ratio of 3:1 and the average density of slurry, which works out at 1.15 t/m³. In what follows, we shall now discuss the three possible alternatives of vertical hoisting and the energy costs per ton of raw coal. In doing so, we shall ignore the mining and horizontal conveying aspects.

Vertical Hoisting from Underground

Fig. 4 depicts the technical relationships of the method of vertical hoisting with the supply of high-pressure water from underground, and in particular the feeding station, the pipe feeder and the high pressure pumps with the booster pumps. The 3-chamber pipefeeder

TABLE 2

Selected Data on Production at the Hansa Hydro-mine (as October 1978)

Date	Production (tons raw)	Running-time RKA* (hrs)	Output RKA* (t/h)	% Rock	Average density of solids	Transported concentration C (%)	Average density of slurry	Remarks
—	5,000	20	250	20	1.535	19.27	1.203	Planning data
15.8.78	755	1.39	458	27.35	1.591	34.0	1.201	Highest RKA* output
20.10.78	3,544	10.17	350	28.42	1.600	25.9	1.555	Highest daily output to date
24.10.78	2,245	8.7	277	33.29	1.64	20.0	1.125	Highest percentage of rocks

* RKA — ROHRKAMMERAUFGEBER — pipe-chamber feeder/3-chamber pipefeeder.

Table 3 presents the data for a normal hoisting day, as typical for the build-up stage of the hydro-mine. The output of 2132 tons, which at this time came from only one mining section, was hoisted to the surface in something over 6 hours. With an average proportion of rocks in the raw coal of 29%, that gives an average density of solids of some 1.6 t/m³.

TABLE 3

Hoisting Data at 19 June 1978

Content of raw coal		
Coking coal	1,157 t	54%
Middlings	60 t	3%
Rocks	535 t	25%
Slurry coal	380 t	18%
	2.132 t	100%
Percentage of coal and rocks		
Coal		71%
Rocks		29%
Densities		
Coal		— 1.4 t/m
Rocks		— 2.5 t/m
Raw coal		— 1.6 t/m
Capacity of pipefeeder		
Raw coal		340 t/h
Slurry		850 m /h
Hoisting time		6.27 h

TABLE 4

Calculated data for the solids and slurry (19 June 1978)

Known : Quantity of solids $M_s = 340$ t/h
Quantity of slurry $V_{TR} = 850$ m⁷/h

1. Volume flow rate of solids

$$\bar{V}_s = \frac{M_s}{\varrho_s} = \frac{340}{1,6} = \underline{213 \text{ m}^3/\text{h}}$$

2. Solids concentration (by volume)

$$C_T = \frac{\bar{V}}{\bar{V}_{Tr}} = \frac{213}{850} = 0,25 = \underline{25\%}$$

3. Relation water/solids

$$\frac{\bar{V}_f}{\bar{V}_s} = \frac{\bar{V}_{Tr} - \bar{V}_s}{\bar{V}_s} = \frac{850-213}{213} \underline{3:1}$$

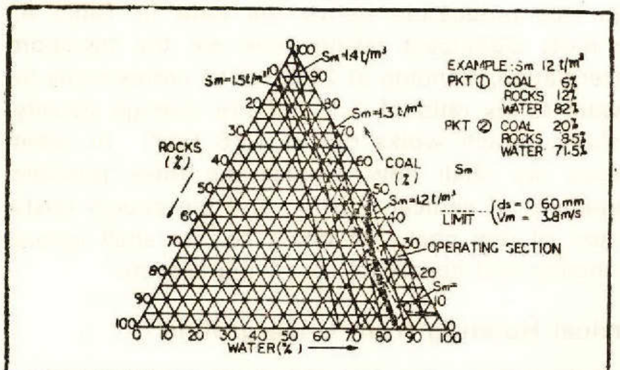
4. Average velocity of slurry

$$\bar{V}_{TR} = \frac{V_{TR}}{A_{pipe}} = 4,82 \text{ m/s}$$

5. Average density of slurry

$$\varrho_{TR} = \frac{\varrho_s \cdot V_s + \varrho_F \cdot \bar{V}_f}{\bar{V}_{Tf}} + \underline{1,15 \text{t/m}^3}$$

Index : S = solids ; f = fluids ; Tr = slurry.



SIEMAG | ROCKS COAL WATER VOLUME PERCENTAGE OF DIFFERENT DENSITIES OF RAW COAL | FIG. 4

carries out the sluicing. The mixture of water and solids is sluiced via the low pressure pump out of the feeding basin into the pipe feeder and is then emptied into the hoisting pipe by means of both high pressure pumps. The hoisting pipe is 850 m long and leads, with a short, horizontal stretch, direct into the preparation plant on the surface.

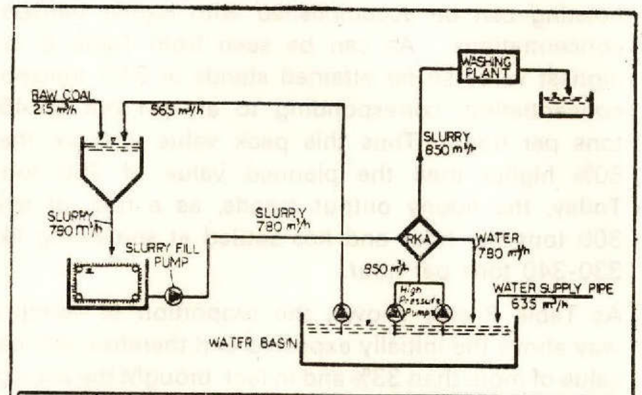
The high-pressure water pumps, each with a capacity of 425 m³ per hour at 120 bar, are driven by high-tension motors which, below ground, have an installed output of 2100 kw and, above ground, 800 kw.

Vertical Hoisting from the Surface

Fig. 5 shows the possibility of vertical hoisting with high pressure water from the surface. The feeding process to the pipe-chamber feeder takes place in the same way as in hoisting with high-pressure water from underground. The hoisting, however, is carried out using another technical process. In this case, the high pressure pumps, as shown on the diagram, are above ground and are connected direct with the 3-chamber pipefeeder via a high pressure pipe in the shaft.

Whereas, with the pumps installed underground, a pressure of 100-110 bar must be produced to force the coal slurry to the surface from a depth of 850 m, a pump pressure from the surface of about 40 bar is sufficient since only the friction losses of the pipe line have to be compensated for.

With these hoisting variants, part of the high-pressure water that is necessary for the hoisting process remains underground. This is that part of the backflow water out of the pipefeeder that is not consumed underground in fluidising the solids. At Hansa, this portion is conducted to the clear water basin and is available for the supply of spraying water for mining.



SIEMAG | HYDRAULIC TRANSPORT HYDRAULIC HOISTING BY PUMPING FROM UNDERGROUND | FIG. 5

If the mining operation has no need to hgih pressure water, this portion is pumped, during breaks in hoisting, by one of the high pressure pumps underground via the hoisting pipe into the water reservoir on the surface. The basic advantages of this solution are as follows :

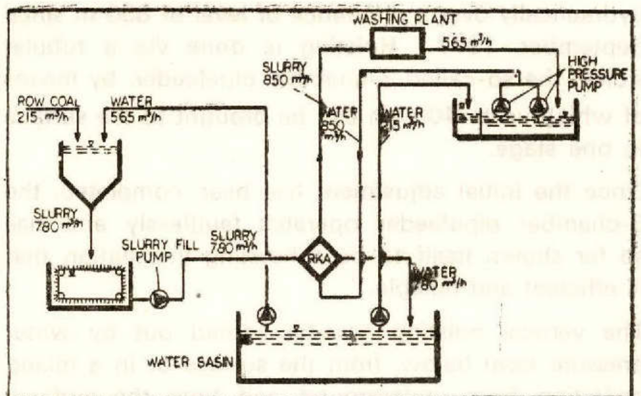
(a) The high-pressure water pumps for the hoisting are located in easily accessible sites on the surfare, where placement conditions and available space are generally more favourable. To this extent, this alternative is comparable in principle to conventional hoisting by means of skip or cage, where this winding gear is likewise above ground.

(b) The capacity of the high-pressure water pumps is small in comparison with the capacities of pumps in the underground arrangement and affects favourably the investment and operating costs.

The balance of quantities is also clear from Fig. 5. As the high pressure water for the hoisting comes from the surface, and the return flow from the 3-chamber pipefeeder is only partly used in the fluidisation process, the surplus water of 215 m³ per hour must be re-pumped to the surface.

Vertical Hoisting from Above and Below Ground

In the third variant, which has proved itself particularly at the Hansa hydro-mine and is commonly emoployed, the supply of high pressure water comes from above and below ground. Under this arrangement (Fig 6), a high pressure pump on the surface, hoists by utilising the geodetic drop, and a pump underground. A prerequisite for such modus operandi is that both pumps be synchronised in operation. The distinction of this method of operation lies in the favourable possibility of harmonising the high pressure water supply with inflows of varying quantities.



SIEMAG HYDRAULIC TRANSPORT HYDRAULIC HOISTING BY PUMPING FROM SURFACE FIG. 6

As with the alternatives already described, the return flow from the pipe-chamber feeder is used in the fluidisation and in the renewed high pressure water supply. Since, however, there is only one hoisting pump underground, the water requirement — and therewith the quantity needed to be supplied from the mine buildings, is smaller.

If the pit has little or no inflows, then a practically balanced economy of water can be achieved by means of a corresponding increase in the efficiency of the pumps above and below ground. In this case, the capacities of the individual pumps must be mutually adapted to the given conditions.

The advantages of the hoisting alternative thus lie essentially in :

(a) Its great flexibility and adaptability to mine water inflows of various quantities ;

(b) The possibility of installing an economic hydraulic hoisting system even in mines without water inflows ; and

(c) The favourable, economic overall solution, particularly with regard to energy costs.

In Germany, the advantages of this alternative have led to ideas of hoisting coal and waste hydraulically, even when they are mined in the traditional manner, i.e. dry, by conventional methods of winning or blasting.

Energy Cost for the 3 Hoisting Alternatives

Finally, we shall discuss the three hoisting alternatives with respect to their energy costs. From Table 5, Column 1, the feeding of the 3-chamber pipefeeder— irrespective of which hoisting alternative is being used — requires about 450 kW driving power. This output is made up from a number of large and smaller motors, the most important of which are specified here.

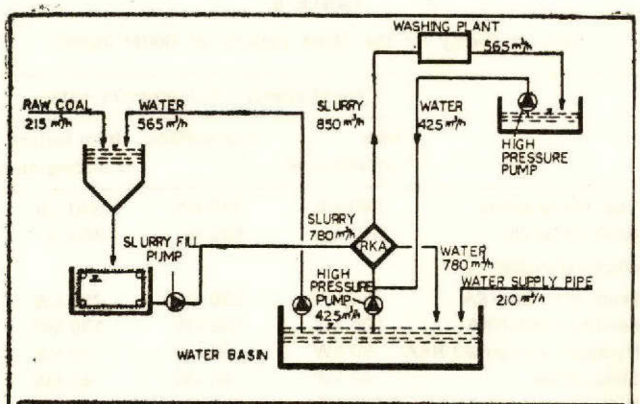
TABLE 5

Cost of Energy for the Three Variants of Water Supply

	Cost of energy : high pressure water		
	from underground	from surface	from surface & underground
Quantity of solids	340 t/h	340 t/h	340 t/h
Hoisting height	850 m	850 m	850 m
Electrical power			
Feeding pump RKA	250 kW	250 kW	250 kW
Feeding basin RKA	130 kW	130 kW	130 kW
Hydraulic equipment RKA	30 kW	30 kW	30 kW
Slide valves	40 kW	40 kW	40 kW
High pressure pump 1	2,100 kW	800 kW	2,100 kW
High pressure pump 2	2,100 kW	800 kW	800 kW
Booster pumps	160 kW	160 kW	160 kW
Total	4,810 kW	3,010 kW	3,510 kW
Required power	4,570 kW	2,860 kW	3,335 kW
West Germany			
Cost per kWh	0.10 DM	0.10 DM	0.10 DM
Energy cost/t raw coal	1.32 DM	0.84 DM	0.98 DM

Hoisting alternative 1, with both high pressure water pumps underground, requires a total driving power of 4810 kW. This provides an efficient power of about 4570 kW, so that in Germany, at a price of 0.10 DM per kilowatt-hour, energy costs work out at 1.32 DM per ton of raw coal.

Table 5, Column 2, shows the corresponding data for hoisting alternative 2, i.e. with the high pressure water supply from the surface. To the 450 kW driving power for the feed process of the pipefeeder are added the power of both high pressure water pumps, each of 800 kW. The power of these high pressure water pumps are substantially lower than those of the underground pumps (hoisting alternative 1), which amount to 2100 kW each. Of course, for the feedback operation of hoisting alternative 2, a further pump of about 800 kW has to be employed to neutralise the underground water surplus. With an efficient output of 2860 kW, the energy costs per ton of raw coal under German conditions work out at 0.84 DM. The third hoisting alternative, with high pressure water supplied from above and below ground, is shown with its data in Table 5, Column 3. The total efficient output of 3335 kW is relatively only a little higher than the output of 2860 kW with high pressure water supplied from the surface (hoisting alternative 2), although in this mine water inflows are also pumped to the surface with the hydraulic hoisting and obviate the need for a separate pumping



HYDRAULIC TRANSPORT
HYDRAULIC HOISTING BY PUMPING FROM UNDERGROUND & FROM SURFACE FIG. 7

operation. Energy costs per ton of raw coal amount, under German conditions, to about 0.98 DM.

Summary

At the Hansa hydro-mine, coarse-grain raw coal upto 60 mm maximum grain has been hoisted to the surface hydraulically over a difference of level of 850 m since September, 1977. Hoisting is done via a tubular sluice, the so-called 3-chamber pipefeeder, by means of which up to 400 t/h can be brought to the surface in one stage.

Once the initial adjustment has been completed, the 3-chamber pipefeeder operates faultlessly and has so far shown itself to be a hoisting installation that is efficient and reliable.

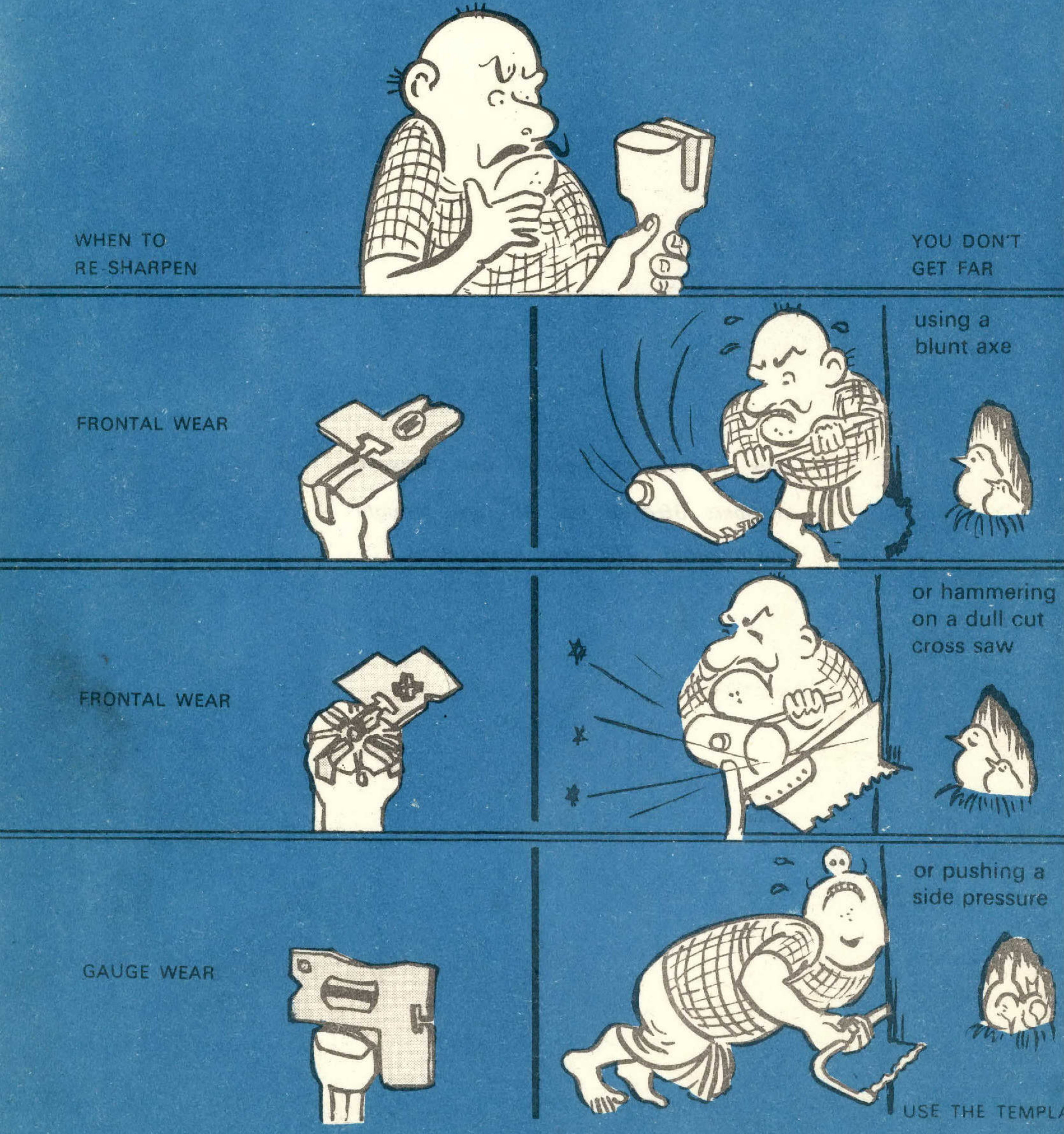
The vertical hoisting can be carried out by water pressure from below, from the surface or in a mixed operation from underground and from the surface. Existing water inflows, that would otherwise have to be brought to the surface by a separate pumping action, can be utilised for the hoisting operation. The energy costs per ton of raw coal for a lifting height of 850 m vary between 0.84 DM and 1.32 DM and therefore lie within the range of the costs which also occur with conventional hoisting.

INDEX TO MINETECH

VOLUME IV

Registration No. 33337/78

Due to reasons beyond our control, numbers 3&4 of Vol. IV are brought out together. The delay is regretted.



WHEN TO
RE-SHARPEN
YOU DON'T
GET FAR
FRONTAL WEAR
using a
blunt axe
FRONTAL WEAR
or hammering
on a dull cut
cross saw
GAUGE WEAR
or pushing a
side pressure
USE THE TEMPL

CENTRAL COALFIELDS LIMITED

SCHEMEWISE INVESTMENT PROGRAMME
(1980-81 to 1984-85)

(Rs lakhs)

		79-80 (Actual)	80-81	81-82	82-83	83-84	84-85	80/81 - 84/85
1.	Existing Mines	1212	1480	733	547	562	377	3699
2.	Approved Reconstruction Projects.	1353	2358	2975	1687	1293	970	9283
3.	Approved New Projects.	1748	2482	6012	5683	3346	1403	18926
4.	Projects formulated but yet to be approved.							
	a) New	68	266	651	713	1183	1845	4658
	b) Reconstruction	152	343	1510	1912	2208	1738	7711
5.	Projects yet to be formulated.							
	a) New	135	500	1131	1770	2676	3595	9672
	b) Reconstruction	611	722	1549	1857	2309	1040	7477
6.	Washeries							
	a) Existing	146	395	300	250	200	200	
	b) Approved New	259	689	1100	1100	1400	1000	
	c) Proposed New	-	-	-	50	150	200	
7.	Others							
	a) Existing	127	304	300	290	310	280	
	b) Approved	87	241	300	350	270	250	
	c) Yet to be formulated	188	597	608	1010	1090	1025	
	d) Exploration	-	356	350	325	300	380	
	e) Safety equipment	-	89	150	200	210	205	
	TOTAL	6086	10822	17669	17744	17507	14508	

78250

WESTERN COALFIELDS LIMITED

GROUPWISE, MINEWISE INVESTMENT 1980-81 - 1984-85

(GROUPS AS ON 1-4-1980)

(Rs. in lakhs)

Groups	Target Capacity	Project Cost	Total Expen-diture upto March'80	INVESTMENT 1980-81	1981-82	1982-83	1983-84	1984-85
I. SPILLOVER IVTH PLAN PROJECTS	2.58	2458	2414	227	481	286	259	255
II. APPROVED RECONSTION PROJECTS	9.00	8092	2408	904	2114	2127	1481	1469
III. (a) APPROVED NEW MINES	22.79	30271	7469	4866	7712	6098	5228	5185
III (b) APPROVED WASHERIES/NON-COAL PROJECTS	-	638	640	50	32	50	-	-
III (c) APPROVED EXPERIMENTAL SCHEMES	0.675	467	360	30	-	-	-	-
III (d) EXPLORATION BY CMPDIL	-	-	277	323	342	372	402	450
IV EXISTING MINES			20045	1878	2780	2292	2998	2926
SUB TOTAL (A)	35.045	41926	33613	8278	13461	11225	10368	10285
VII PROJECTS AWAITING APPROVAL	7.30	16891	-	873	1606	1995	2903	2880
VIII PROJECTS TO BE FORMULATED	17.02	-	-	50	978	2304	2816	2793
TOTAL (A) & (B)	59.365	58817	33613	9201	16045	15524	16087	15958
IX. ADDL. INVESTMENT FOR NEW AREAS IN PENCH/KANHAN NOT INCLUDED IN 5 YEAR PLAN DOCUMENT				100	700	725	725	750
GRAND TOTAL	59.365	58817	33613	9301	16745	16249	16812	16708

WESTERN COALFIELDS LIMITED

GROUPWISE, MINEWISE INVESTMENT 1980-81 - 1984-85

(GROUPS AS ON 1-4-1980) (Rs. in lakhs)

Groups	Target Capacity	Project Cost	Total Expenditure upto March'80	INVESTMENT 1980-81	1981-82	1982-83	1983-84	1984-85
I. SPILLOVER IVTH PLAN PROJECTS	2.58	2458	2414	227	481	286	259	255
II. APPROVED RECONSTION PROJECTS	9.00	8092	2408	904	2114	2127	1481	1469
III. (a) APPROVED NEW MINES	22.79	30271	7469	4866	7712	6098	5228	5185
III (b) APPROVED WASHERIES/NON-COAL PROJECTS	-	638	640	50	32	50	-	-
III (c) APPROVED EXPERIMENTAL SCHEMES	0.675	467	360	30	-	-	-	-
III (d) EXPLORATION BY CMPDIL	-	-	277	323	342	372	402	450
IV EXISTING MINES			20045	1878	2780	2292	2998	2926
SUB TOTAL (A)	35.045	41926	33613	8278	13461	11225	10368	10285
VII PROJECTS AWAITING APPROVAL	7.30	16891	-	873	1606	1995	2903	2880
VIII PROJECTS TO BE FORMULATED	17.02	-	-	50	978	2304	2816	2793
TOTAL (A) & (B)	59.365	58817	33613	9201	16045	15524	16087	15958
IX. ADDL. INVESTMENT FOR NEW AREAS IN PENCH/KANHAN NOT INCLUDED IN 5 YEAR PLAN DOCUMENT				100	700	725	725	750
GRAND TOTAL	59.365	58817	33613	9301	16745	16249	16812	16708

Eastern Coalfields Ltd.

Capital Outlay For 1984-85

Groups	(Rs in crores)
I Spillover IV Plan Projects	—
II Approved Reconstruction	16.60
III Approved New Mines	—
VI Existing Mines	35.20
VII Projects formulated but yet to be approved	22.00
VIII Projects yet to be formulated	115.30
(A) TOTAL Mining Projects	189.10
(B) Washeries	—
(C) Others	—
(D) Exploration	1.00
TOTAL ECL	190.10

INVESTMENT PROGRAMME.

(BCCL)

(figures in Rupees lakhs).

PROJECTS/SCHEMES	1980-81	1981-82	1982-83	1983-84	1984-85	1980-85 (Total).
Existing Mines/Units.	1855	2285	2225	2125	2100	10590
Spillover IV Plan Projects.	988	1200	885	566	400	4039
Approved Reconstruction Projects.	508	2160	2958	2700	2900	11226
Approved New Mines	546	707	491	147	--	1891
Approved others	1099	1147	1075	1000	1000	5321
Approved Washeries	773	300	900	2300	1600	5873
Projects formulated and yet to be approved.	1580	2890	3585	3549	3835	15439
Projects yet to be formulated.	40	4280	6186	6241	8935	25682
Exploration & Project Planning to be done by C.I.L.	322	336	367	382	400	1807
Capital Equipment in Stores.	--	--	--	--	--	--
Other items including hydraulic mining.	--	130	400	600	400	1530
GRAND TOTAL:	7711	15435	19072	19610	21570	83398

Groups	Particulars	Expenditure upto 31.3.79	Capital requirement as proposed in Draft 5 yr. Plan					Capital requirement as proposed now as recommended by Dept. of Coal				
			79-80	80-81	81-82	82-83	83-84	RE 79-80	BE 80-81	81-82	82-83	83-84
I.	Spill-over IV Plan				N I L							
II.	Approved Re-organisation Projects	2112	1731	2705	2007	2404	1073	1473	1907	2470	2800	2800
III.	Approved New Mines	1504	1214	304	400	183	10	1234	1030	300	115	10
IV.	Approved Washeries			NIL								
V.	Approved Others	230	293	233	104	00	7	100	183	223	50	-
VI.	Existing Mines & Infrastructures	13500	3113	4582	5000	4368	4475	2750	3139	5004	4370	4470
VII.	Projects formulated & yet to be approved	-	779	703	755	1370	1370	203	1030	1332	1273	1105
VIII.	Projects yet to be formulated	-	11	1393	3700	7688	7248	-	010	4579	7074	8105
	Others	271	-	-	-	-	-	-	-	-	-	-
	TOTAL :	17713	7191	10090	13711	16081	15293	5760	8707	14503	16280	16550
	CMPDI:											
	Exploration	-	319	312	317	342	300	319	312	317	342	300
	Planning	-	-	-	-	-	-	122	104	110	110	110
	GRAND TOTAL	17713	7510	11002	13328	16423	15649	6201	9123	14930	16740	17032

3.8