A STUDY ON APPLICATION OF STRATEGIC PLANNING MODELS AND OPERATIONS RESEARCH TECHNIQUES IN OPENCAST MINING

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BY
KSHIROD CHANDRA BRAHMA

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DR. B.K.PAL AND DR. C. DAS

DEPARTMENT OF MINING ENGINEERING
NATIONAL INSTITUTE OF TECHNOLOGY
ROURKELA-769008
ORISSA
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CERTIFICATE

This is to certify that the thesis entitled "**A Study on Application of Strategic Planning Models and Operations Research Techniques in Opencast Mining"** being submitted by Shri Kshriod Chandra Brahma for the award of the Degree of **DOCTOR OF PHILOSOPHY** in Engineering to the National Institute of Technology, Rourkela, Orissa, India is a record of bonafide research work carried out by him under our supervision and guidance. The thesis, in our opinion, has fulfilled the requirements of the regulations of the National Institute of Technology, Rourkela and has satisfied the standard pertaining to the Degree. The results incorporated in the thesis have not been submitted to any other University or Institution for the award of any Degree or Diploma.

Dr.C.Das,
Prof. & Head (Retd.)
Dept. of Mathematics,
N.I.T.Rourkela.

Dr.B.K.Pal,
Professor,
Dept. of Mining Engineering,
N.I.T.Rourkela.

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CONTENTS

PAPERS PUBLISHED / PRESENTED

1 Venukumar, N and Brahma, K.C. (2003) "Some aspects of Petri nets and its application in Mining". MEAI-Seminar on "*Recent Trends in Mine Mechanisation Exploration to Mine Closure*". 21-22, Nov.'2003, Puri,Orissa

2 Venukumar, N and Brahma, K.C. (2004) "Petri nets and its application in Mine Automation". National Seminar on "Policy Formulation and Strategic Planning for Mineral Industry-2012" Organised by IMMA, Bhubaneswar. 10-11 January, 2004.

3 Brahma, K.C.; Pal, B.K. Das, C. (2004) "Study on application of Petri nets in opencast mining". *Proc. of the Thirteenth International Symposium on Mine Planning and Equipment Selection*, Wroclaw, Poland Ed by Monica Hardygora, Gabriela Paszkowska; Morek Sikora, A.A. Polardn Balkema Publishers, pp.3-6.

4 Brahma, K.C.; Venukumar, N.; Das, C. and Pal, B.K. (2004) "Petri net based planning and scheduling of initial activities of an opencast coal mine project". Presented at the International Seminar on "*Technology Update in Mining and Mineral Industries*". Indian Institute of Science, Bangalore. 16th & 17th October, 2004. Organised by Mining Engineers Association of India with DMG, Karnatak. pp: 183-190. Published *Mining Engineer's Journal*. April, 2005 Vol.6, No.9, pp.16-19.

5 Brahma, K.C, Pal, B.K. and Das,C. (2005) "An application of Regression Model for evaluation of blast vibration in an opencast coal mine – a case Analysis" *Proc of the Fourteenth International Symposium on Mine Planning and Equipment Selection (MPES-2005)* and the Fifth International Conference on Computer Application in the Minerals Industries (CAMI 2005). Held jointly at Banff, Alberta, Canada. Oct., 31st – Nov – 3rd, 2005, Edited by Raj K.Singhal,Kostas Fytas & Chris Chiwetelu. pp.161-165.

6 Brahma, K.C.; Venukumar, N.; Pal, B.K. & Das,C (2005). "Application of PETRI NETS for Modeling of Drilling Operation in Opencast Mines" Presented at and published in the *Proc. of International Symposium on Advances in Mining Technology and Management (AIMM,2005)* at Indian Institute Technology, Kharagpur. November 30 – December 2, 2005. Edited by Ashis Bhattacharjee. Samir Kumar Das, U.M.Rao Karanam.pp. 323-326.

7 Brahma, K.C., Pal, B.K. and Das, C. (2005). "Modeling of shovel dumper combination system in opencast mines through Petri nets". *Fifth International Conference on Operational Research for Development (ICORD-V)*. Organised by IFORS and OSRI, India at Jamsedpur. 19th - 21st December, 2005.

8 Brahma, K.C., Pal, B.K. and Das, C. (2006). "An investigation of multivariate statistical models to assess blast design strategies in an opencast coal mine", *The Indian Mineral Industry Journal*, May, 2006, pp. 119-125.

9 Brahma, K.C., Pal. B.K. and Das, C (2006). "Petri net a tool for modeling of continuous surface miner operations in opencast mines". *Fifteenth International Symposium on the Planning & Equipment Selection, 2006*.

10 Brahma, K.C., Pal, B.K. and Das, C. (2007). "A study on application of Markovian decision process in drilling and blasting operations of opencast mines". *Journal of Mines, Metals & Fuels*, Vol-55, No: 1 & 2 ,Jan –Feb, 2007 ,pp-24-29.

11 Brahma, K.C., Pal, B.K. and Das, C. (2006). "Optimization of shovel-dumper combination system by application of Queueing theory – a case study". *International Journal of Surface Mining, Reclamation and Environment* (communicated).

GLOSSARY OF ABBREVIATIONS

AI	Artificial Intelligence
BT	Billion Tonnes
B.C.C.L	Bharat Coking Coal Limited
CAGR	Compound Annual Growth Rate
CBM	Coal Bed Methane
C.C.L	Central Coalfields Limited
CHP	Coal Handling Plants
C.I.L	Coal India Limited
CMM	Coal Mine Methane
CMPDI	Central Mine Planning and Design Institute
CPM	Critical Path Method
CUM	Cubic Meter
DGMS	Director General Mine Safety
D/L	Dragline
D.P.	Dynamic Programming
DTH	Down The Hole
E.C.L	Eastern Coalfields Limited
EMP	Environmental Management Plan
FCFS	First Come First Served
FDI	Foreign Direct Investment
FIFO	First In First Out
GDP	Gross Domestic Product
GERT	Graphical Evaluation and Review Techniques
GIS	Geographical Information System
GPS	Global Positioning System

GSI	Geological Survey of India
HEMM	Heavy Earth Moving Machinery
ICRIS	Integrated Coal Resource Information System
IISCO	Indian Iron and Steel Company
IRR	Internal Rate of Return
KgOL	Kilogram of Oil Equivalent
LHD	Load Haul Dump
L.P.	Linear Programming
M.C.L.	Mahanadi Coalfields Limited
NCDC	National Coal Development Corporation
N.C.L.	Northern Coalfields Limited
N.L.P.	Non Linear Programming
OB	Overburden
OC	Opencast
OE	Oil Equivalent
OMS	Output per Man per Shift
OR	Operations Research
PERT	Program Evaluation and Review Techniques
PDM	Precedence Diagramming Method
PN	Petri nets
PNET	Probabilistic Network Evaluation Techniques
PPV	Peak Particle Velocity
PR	Project Report
S.C.C.L	Singareni Collieries Company Limited
SDL	Side Discharge Loader
S.E.C.L	South Eastern Coalfields Limited
SEFT	Static Earliest Firing Time

S.G.	Specific Gravity
SLFT	Static Latest Firing Time
SM	Surface Miner
S.R.	Stripping Ratio
TERI	The Energy Research Institute
TISCO	Tata Iron and Steel Company
TLD	Trunk Line Delay
TPN	Time Petri Nets
UCG	Underground Coal Gasification
UG	Underground
VLSI	Very Large Scale Integration
W.C.L.	Western Coalfields Limited

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INTRODUCTION

1.0 Background

Mining happens to be the second oldest industry in the world considering the agriculture as the first and the foremost. That primitive society relied nearly on mined produce that is reflected aptly through nomenclature such as Stone Age, Copper Age, Bronze Age and Iron Age. These nomenclatures rightly capture the ethos of the time that shows increasing complexity of people's society's relationship with mining produce and use of metals.

Our remote ancestors did practice mining on hard rock. Mining remained with their common occupation to earn livelihood and meet their needs. Since they had meager requirement of fuels; their major need of fuel was met mostly from dense forests on the earth. As the time passed, they required to meet ever increasing standards of living. As a result, demand of fuel was felt as extremely necessary for the existence of mankind and it kept on growing. In order to meet ever increasing demand, mining of coal took a shape in one way or the other.

Nature provides basically six sources of producing wealth as fruits of man's endeavours like parenthood, regeneration, hunting and fishing, forestry, agriculture, husbandry and mining. Out of them, mining is a source to obtain oil and coal. They are major raw materials to serve as fuel in the present time. Rapid growth of oil industry and invention and subsequent innovations of internal combustion engines and emergence of electricity as (alternate form) optional power brought about a decline in direct use of coal as energy source. However, the subsequent worldwide energy panic of the 1970's that resulted from oil embargo restored back coal to its prominent position in the world scenario. Dwindling position of oil reserves across the world is another potent reason to raise coal utilization to prime concern. Since coal continues to enjoy advantage over other energy sources, and since reserves of coal are much ample and assured the world wide in comparison to rival energy sources, coal utilization comes to the forefront. India being an oil-deficient country makes a clear choice to boost up coal production and to rely more on it as energy source.

Since industrial revolution captured the nerve of the world in the late 19th and 20th centuries, energy sectors acquired prime concern with a view to fulfilling energy requirements for survival and efficient functioning of industries. As a result, the need of coal production was felt more acutely. It comes to almost 60% of the energy resources in the Indian scenario. Out of the four major fuel sources in India – oil, natural gas, coal, and uranium – coal has the largest domestic reserve base and claims the largest share in India's energy production. The per capita energy consumption of fuel in India is the highest in case of coal that is 176.8 kg Oil Equivalent (OE), followed by that of Oil with 104.9 kg OE, of Natural Gas (25 kg OE), of Hydel power (14.6 kg OE) and of Nuclear (3.8 kg OE). It makes a total of about 325 kg OE (Kumar, 2005).

The installed electric power generation capacity in India is given in table -1.1 below (as on March, 2005):

Table – 1.1

Installed Electric Power Generation Capacity in India (MW)

Installed Capacity	Million Watts
Coal	67791
Diesel	1201
Gas	11910
Total	80902
Wind and Renewable Energy	3811
Nuclear	2770
Hydro	30936
Grand Total	118419
GROSS GENERATION – 2004-05	
Hydro	84497 GWh
Nuclear	16638 GWh
Thermal and Wind	486031 GWh
No. of Villages electrified	498286
Length of Transmission & Distribution lines	6497727 Circuit km.
Per Capita Consumption	606.2 k Wh
Loss – Transmission, Distribution, Transformation and Unaccounted energy	32.15%

(Source: www.expert-eyes.org/power/capacity.html)

The coal mining industry in India has not been fostered well by historians and researchers. It is generally neglected by the national media. There is a little documentation available on India's 300 year of coal industry. Coal mining has progressed far and ahead in the past 50 years, especially following the nationalization of the industry in the early 1970s. India today proudly claims a higher position among major coal producing countries in the world say as the third largest producer of coal after China being the first and the USA the second, with an output of 376.78 million tones in 2004-05. It applies the most advanced technologies in various spheres, from exploration to exploitation.

Coal being the primary source of energy supply in India dispatching about 73% of the total production, it also accounts for almost 60% of the electricity generation in the country. The world's third largest coal reserves of 247.86 billion tones are located in India as recorded on 1.1.2005 (Table-1.2 & Table-1.3). They serve a source to manage secure energy supplies to cater to the requirements of a growing economy, expanding population and rapid urbanization. As Kumar views, the coal reserves in India are expected to last over 235 years at the current rate of production. In contrast, the recorded reserves of oil and natural gas in the country are expected to last for hardly 17 years & 25 years respectively (Kumar Shashi, 2005).The category-wise and depth-wise break-up of reserves in India is given below (GSI Reports)

Table – 1.2

Category-wise and depth-wise break-up of coal reserves in India

Depth (in mts)	**Category-wise Reserves (in Million Tonnes)**				**% Share**
	Proved	**Indicated**	**Inferred**	**Total**	
0-300	71068.98	66510.96	14995.52	152575.46	61.6
300-600	6511.66	39448.37	17182.35	63142.38	25.5
0-600 for Jharia only	13710.33	502.09	0	14212.42	5.7
600-1200	1669.18	10628.77	5618.64	17916.59	7.2
Total 0-1200	92960.15	117090.19	37796.51	247846.85	100
% share	37.5	47.2	15.3	100	

(Source: GSI reports)

Type-wise and category-wise break-up of coal reserves are as follows:

Table – 1.3

Type-wise and category-wise break-up of coal reserves in India

Type of coal	Category-wise Reserves (in Million Tonnes)				% Share
	Proved	**Indicated**	**Inferred**	**Total**	
Prime coking	4614.35	698.71	0	5313.06	2%
Medium coking	11416.74	11754.79	1888.94	25070.47	10%
Semi coking	482.16	1003.29	221.68	1707.13	0.7
Total of coking	16513.25	13466.79	2110.62	32090.66	12.9
Non-coking	76015.11	103517.01	35317.12	214849.24	86.7
Tertiary coal	431.79	106.39	368.77	906.95	0.4
Total all types	92960.15	117090.19	37796.51	247846.85	100
% share	37.5	47.2	15.3	100	

(Source: GSI Reports)

The distribution of coal resources within the command areas of CIL, SCCL and other Stake Holders is given below in Table-1.4:

Table – 1.4

Distribution of Coal Reserves

(In BT)

Blocks	Proved	Indicated	Inferred	Total
CIL	67.7	19.4	4.6	91.7
Non-CIL (Captive)	9.5	15.9	2.7	28.1
Non-CIL (Others)	3.5	3.4	5.9	12.8
Others (TISCO etc.)	2.8	0.3	0	3.1
Un-blocked	0.8	71.9	21.6	94.3
Godavari Valley	8.2	6.1	2.6	16.9
NE Region	0.4	0.1	0.4	0.9
Total :	**92.9**	**117.1**	**37.8**	**247.8**

(Source: Kumar, 2005)

The year wise growth of estimates of coal reserves in India since 1.1.1999 to 1.1.2005 are given in Table: 1.5.

Table – 1.5

Estimates of Coal Reserves in India (Million Tonnes)

As on date	Proved	Indicated	Inferred	Total
1.1.1999	79106	88427	41219	208752
1.1.2000	82396	89501	39697	211594
1.1.2001	84414	98546	38023	220983
1.1.2002	87320	109377	37417	234114
1.1.2003	90085	112613	38050	240748
1.1.2004	91631	116174	37888	245693
1.1.2005	92960	117090	37796	247847

(Source: GSI Reports)

1.1 Demand Scenario of Coal in India

Prior to portraying the futuristic coal demand scenario, it would be appropriate to have a glance at the past trends of coal consumption vis-à-vis growth in the national GDP. It is observed that in the last two decades or so there was a gradual decline in the elasticity of demand of coal against the GDP. As a result, during the period between 1992 and 2003, the CAGR of the GDP was 5.98% while that of coal consumption was 4.15% (Kumar, 2005). The possible reasons for the decline may be presumably accounted for as (a) rise in the share of non-energy consuming sectors in the aggregate GDP, (b) tendency towards substitution of coal by alternative fuels and (c) technological innovations in coal consuming sectors leading to energy efficiency and reduction in specific consumption (Kumar, 2005).

Coal Vision 2025 estimates a demand of coal for the future upto the year 2024-25. The estimate concerns different sectors and it is based on the approach adopted by the TERI in its exercise for the purpose. It adopts an approach of econometrics and establishes relationship between the coal demand and a change in the GDP. The adopted approach indicates that the overall growth in the coal demand is expected to be 5.62% with 8% of growth in the GDP and 5.04% with 7% growth in the GDP.

Sector-wise demand of coal is assessed with the above approach for both the positions. The following table-1.6 presents the results:

Table – 1.6

Sector-wise coal demand

(Million Tonnes)

Sector	05-06*	06-07*	2011-12		2016-17		2021-22		2024-25	
			7%	8%	7%	8%	7%	8%	7%	8%
Power utilities	303.56	317	412.69	427.16	517.31	552.56	635.46	698.53	718.94	804.03
Power captive	27.35	28.26	43.26	44.33	59.89	62.96	83.50	90.04	101.93	111.60
Steel	42.05	42.7	53.14	54.24	66.57	69.47	83.87	89.52	96.54	104.50
Cement	20.22	25.40	38.44	39.39	58.18	61.06	88.16	94.82	113.13	123.47
Brick & Others.	52.47	59.82	63.52	64.51	79.57	82.11	100.72	105.62	116.54	123.41
Total	445.65	473.18	611.45	629.63	781.52	828.16	991.70	1078.54	1147.08	1267.01

(Source: Kumar Shashi, 2005)

* As per Annual Plan 2005-06 of Ministry of Coal.

Demand Estimates as Reported by Administrative Ministries

The Administrative Ministries of Coal Consuming Sectors in their presentation before the Expert Committee set up for the Coal Sector report about the current and the future requirements of coal as follows (Table-1.7) (Kumar,2005) :

Table – 1.7

Demand Estimates as per Administrative Ministries

(Million Tonnes)

Sector	2005-06	2006-07	2011-12
Power utilities	304.00	317.00	511.00
Power captive	27.35*	28.26*	44.33**
Steel	47.61	53.04	54.24**
Cement	22.71	24.47	33.58
Bricks & Others	52.47*	59.82*	64.51**
Total	**454.14**	**482.59**	**707.66**

(Source: Kumar Shashi, 2005)

* As per Annual Plan 2005-06 of the Ministry of Coal

** As per the estimates present in *Coal Vision 2025*, as the figures are not available from the Administrative Ministries.

As per the recent communication from the Ministry of Power, demand of coal for the power sector is likely to increase further in the near future. The communication indicates that by 2011-12, coal requirement for power sector will increase to 550 million tones for the Utilities and 55 million tones for Captive plant. This shows that the demand scenario for coal is highly dynamic and is expected to grow beyond what is planned about near future.

In this light, the table 1.8 presents the Indian demand supply status of coal for the Xth Five Year Plan and for the terminal year of XIth Five Year Plan. A total gap between the country's demand and the supply of coal including expected imports shows an increasing trend on the forecast. For such reason, the country has to focus on rapid expansion and high investment in the coal sector together with high productive technology, so that coal may be offered to Indian consumers at competitive prices.

Table – 1.8
Demand & Supply of Coal – Indian Scenario
(Million Tonnes)

	IX plan	**X Plan**					**Term Yr XI plan**
	01-02 Actual	**02-03 Actual**	**03-04 Target**	**03-04 Actual**	**04-05 Projected**	**06-07 Projected**	**11-12 Projected***
Demand all India	354.29	363.30	380.90	380.90	404.19	460.50	620
CIL supply	279.65	290.69	299.85	306.38	315.05	350.00	445.00
SCCL supply	38.81	33.24	33.50	33.85	35.00	36.13	35.40
Others supply	17.33	17.34	18.05	20.93	20.15	18.87	44.60
Total indigenous supply	327.79	341.27	351.40	361.16	370.20	405.00	525.00
Import coking	11.11	12.95	16.41	13.33	15.89	17.18	
Non-coking (low ash not available indigenously-mainly)	9.44	10.31	3.00	7.51	7.50	3.30	
Total import	20.55	23.26	19.41	20.84	23.39	20.48	
Demand/ supply gap (indigenous)	26.50	22.03	29.50	19.74	33.99	55.50	
Gap (between demand & supply)	5.95	0.00	10.09	0.00	10.60	35.02	95.00

(Source: Ministry of Coal reports)

*	As per working group on coal & lignite import of coal shows a decline in trend due to (i) volatile price (ii) no long term commitments (iii) fluctuation in ocean charges.

The working group on coal and lignite for the Xth Five Year Plan estimates indigenous supply of coal to remain at only 405 Mt by 2006-07. The situation may grow worse by the end of XIth Five Year Plan when the demand will grow upto 620 Mt.

1.2 Production Plan from Indigenous Sources

Considering the scenario of high demand of coal, either as per the *assessment of Coal Vision 2025*, or as per report in the estimates of the Administrative Ministries of Coal Consuming Sectors, a need is acutely felt to increase availability of coal from the indigenous sources. With this view in mind, coal production programme has been worked out for different 'Plan Periods' in future. One needs to think beyond the so far followed mechanized mining of coal deposits through conventional technology and explore further avenues of harnessing CBM and in-situ gasification of coal from inaccessible deposits. In order to supplement availability of superior quality coal of both coking and non-coking grades, the Coal India has also set up a subsidiary named "Coal Videsh" to obtain supply of coal from foreign sources. The responsibility of the subsidiary is to explore information on mines abroad and through acquisition of overseas equity in coking coal property abroad it should arrange for direct import to bridge the gap.

The coal production plan, as envisaged in the *Coal Vision 2025* document, is expected to rise to 1063 million tones per annum by 2024-25. This quantity is beyond a need of 25 million tones of coal equivalent energy that is expected to be met from the CBM-UCG initiatives that are taken up by the CIL and other companies. The production plan of the terminal years of different Plan Periods is projected as follows in the table – 1.9 (Kumar, 2005):

With higher rate of coal and its increasing trend, it is required that mining operations are standardized with scientific approach. In this context, traditional methods need to be replaced with modern techniques such as OR, Petri Net and other computer aided packages. They should be applied with systematic study and scientific analysis so that old time traditional thumb rules may be replaced with those of diagnostic, analytical and mathematical kinds. The present research deals with solutions to some selective problems of mining operation and analyzes them through strategic planning models and OR modeling techniques adopted in the opencast mining scenario.

Table – 1.9

Future Production Plan

Producing Company	Projected Production (Mt)														
	X-Plan (06-07)*			XI-Plan (11-12)**			XII-Plan (16-17)			XIII-Plan (21-22)			XIV-Plan (2025)		
	OC	**UG**	**Tot.**	**OC**	**UG**	**Tot.**	**OC**	**UG**	**Tot.**	**OC**	**UG**	**Tot.**	**OC**	**UG**	**Tot.**
ECL	23.5	11	34.5	30.5	14.5	45	43	20	53	43	27	70	43	31	74
BCCL	19.8	7.2	27.0	24.5	7.5	32	27	13	40	29	16	45	32	17	49
CCL	41.3	2.7	44.0	64.65	2.35	67	108	4	112	121	6	127	120	11	131
WCL	34.1	9.9	44.0	34.85	10.15	45	23	13	36	21	19	40	18	28	46
SECL	70.5	18	88.5	89	22	111	126	26	152	127	27	154	127	32	159
NCL	54	-	54.0	70	-	70	70	-	70	76	-	76	81	-	81
MCL	77.8	2.2	80.0	134	3	137	170	4	174	212	5	217	236	5	241
NEC	0.8	0.2	1.0	0.8	0.2	1	0.8	0.2	1	0.8	0.2	1	0.8	0.2	1
CIL Total	321.8	51.2	373	448.5	59.5	508	568	80	648	630	100	730	658	124	782
Non-CIL Areas Unblocked Areas	-	-	-	-	-	-	5	-	5	25	-	25	57	-	57
Gross CIL			373			508	573	80	653	655	100	755	715	124	839
Coal equ. CBM-UCG	-	-	-	-	-	-	-	-	5	-	-	15	-	-	25
SCCL			37.5			41			45			47			47
Others,viz captive TISCO/ IISCO etc			21			44			75			125			175
Grand Total (Domestic)			431.5			593			778			942			1086

(Source: Kumar Shashi 2005)

* (As per Annual Plan 2005-06 of the Ministry of Coal)

** Adjusted with Revised Production Plan of the CIL.

1.3 General Mining Operations

Mining may be defined as an act, process or work of extracting minerals or coal from below the natural bed of earth and transporting them to a point of process or consumption.

A mine, therefore, is a spot where digging operation is conducted below the earth. Its purpose is to extract minerals / coal. Such operations are conducted on the surface of the earth or underground or both on the surface as well as underground.

Under traditional methods, miners use hands and implements of wood, bone, stone and metals to dig the earth and extract minerals. With an advent of a social awareness, mining probably became more organized. Use of slaves to labour under supervisors would undoubtedly mean to meet higher production goals.

The Industrial Revolution spurred a demand for energy sources. It further intensified search for coal as it constitutes a major source to generate energy. The sequence of mining operations starts from exploration of coal deposits to be followed by those of finding, proving, developing, mining and processing and marketing of products to mine closure has to be judiciously decided. Each step requires prudent planning and design and also systematic and scientific execution.

1.4 Operational Scenario

On broad base, mining activities are divided into two parts - underground and opencast or surface mining. After a deposit is discovered, delineated and evaluated, the next step is to select a suitable mining method which is physically, economically and environmentally feasible to obtain minerals from Mine deposits. Many factors affect a selection of a method of mining. However, before choosing any method, safety and economy have to be granted due priority.

1.5 Processes involved in Opencast Mining

In recent years, the opencast mining attains a commanding height in global mineral production with a huge share of 80% of the aggregate production of mineral raw materials (Ghose, 2004). At quarrying operations overburden on the earth in the form of alluvium and rocks need to be removed. Such waste material needs to be dumped in initial stages at a place that would not be required in future for quarrying or other purposes like housing etc. Once the coal exposed is completely extracted, the area is then backfilled to the original land level. Various processes involved in the opencast mining are as under:

i) Mine planning and design,
ii) Drilling of holes,
iii) Blasting,
iv) Loading by shovels,
v) Transportation of OB & Coal,
vi) Land reclamation.

1.5.1 Mine Planning and Design

In order to conduct mining efficiently and to achieve desired goals, planning and design acquire prime concern with due regards to safety, productivity, conservation and restoration at reasonably low costs. In the opencast mining, planning and design are correlated to all phases of mining operation. The factors that are considered in planning and designing of an open pit mine are numerous and they reflect on the characteristics of surrounding conditions of a particular coal seam/ore body. Pertinent elements to be considered in planning are geology, topography, tonnage and an area of the reserves, mining equipments, economic factors like operating costs, capital expenditures, profit, types of coal seam/ore body, pit limits, cut-off grade, stripping ratio, rate of production, pit slopes, bench heights, road grades, hydrological conditions, property lines, marketing and justification for mining.

1.5.2 Drilling of Holes

With more efficient equipments coming to market in a steady flow and enhanced form technology registers a high pace in the drilling and breaking sectors. The first operation that falls in the line of unit operations conducted during the exploitation phase in surface mining is production drilling. It precedes blasting. It is associated with blasting as the two unit operations employed to break into pieces of the consolidated material in a rock form. The principles of drilling are concerned with energy employed for penetration of rocks. Usually, it is mechanical energy that goes with functional responses and interrelationship between drill systems and rocks. The utilization of mechanical energy for penetration of rocks primarily involves development of the drill system. However, emphasis is laid on efficiency and practical approach of the system in a specific working environment. The purpose of drilling is to create large or small diameter holes in the natural rock massif. Drilling of holes is a process consuming labour and high cost process, especially when drilling is done on hard rocks. All the mining operations are fully dependent on this first and basic operation of drilling. Further, drilling is a vital operation to get better blast efficiency. Drilling activity needs to be meticulously planned keeping in view the following aspects:

a) Each mine should have a decided blast size i.e. no. of rows. It is but obvious those minimums of 3-4 rows are normally planned to reduce boulders, toes and improve blast efficiency.
b) There should be a decided pattern and it should be marked on the ground distinctly so that there is no deviation in drilling patterns.
c) Checking and strict control on the above mentioned two aspects is much required and blasting should not be undertaken until the checks are conducted and recorded duly on the "Drilling Plan".

This signifies the importance of drilling. Those involved in drilling become fully aware of the accuracy of drilling and its role.

1.5.3 Blasting

Developments in blasting technology are essential to raise productivity and lower costs on reasonable grounds. The first stage in any mining operation is breaking of whole rock into fragments that are of suitable size for loading and subsequent transportation and handling. Most mines standardize their blasting practices by employing a set of guidelines for blast designs that can be reasonably guaranteed to generate acceptable results.

In the mining industry the breaking of rocks is undertaken with mechanical and chemical methods. Mechanical methods include drilling by boring machine, hydraulic hammers and many others. Chemical methods require exclusive use of explosives that is loaded in previously drilled holes. Mechanical methods are not used primarily in rocks of lower strength (38-69kPa unconfined compressive strength). For large scale breaking of rock with higher-strength, drill and blast method is by far a common technology.

A need to design blast geometry gains momentum in the recent times. It is to obtain optimum blast results. Optimum blasting is recommended to obtain proper degree of fragmentation of rocks with a lower combined cost of drilling, blasting, loading, hauling and crushing with due regards to the limits of vibration, noise and fly rocks.

There are two inherent dangers of blasting. They are vibration and a resultant fragment size. A careful study and analysis are required to find out a maximum charge/delay and total charge in a round keeping in view a distance between nearby structures. Optimum blast design may be adopted to produce a required size of the product after the blast. It can be taken up in a mine with a view to establishing a suitable blast design. Various blast design parameters such as bench height, burden, spacing, depth of hole, sub-grade drilling, max charge/hole, total charge in round etc. can be determined for a particular mine. It facilitates smooth operation for a loading machine and also small size of the blast output can be handled with a primary crusher.

1.5.4 Loading by Shovels

Shovels are used as loading machines. They are deployed to varying and extremely broad based ranges of loading work. They are designed to accommodate a wide range of working conditions. Sizes of shovels have increased incredibly to meet ever increasing requirements. There is no limitation observed on sizes of shovels and other loading machines. Availability of trucks to match mining shovels is a matter of great concern. It is an only major obstacle to increasing these capacities. Physical and economic considerations may operate as controlling factors in equipment size.

1.5.5 Transportation of OB and Coal

From time to time, a mining engineer faces a need to conduct a study of haulage or transportation. It is to determine not only the most suitable method of hauling material, but also to determine most effective and economical means or equipment to use for operation of shifting materials. The rear dumpers are mostly used in transportation of OB and coal in opencast mining project. These units have the body that is mounted on a frame of a truck. Dumping is carried out by raising the box with a hydraulic hoist system. There are common types of trucks that are capable of handling all types of material, whether blasted, ripped or loose. These units cannot be used for any road, but for off highway service, since they exceed legal width and weight limits.

The OB or coal after being loaded into dumpers or trucks, are transported to dump-yards or coal handling plants for the purpose of dumping and crushing at respective destinations.

1.5.6 Land Reclamation

After mining, the land is left in rather unattractive state of condition. A natural topography, with its unique drainage pattern is miserably altered into a series of almost parallel ridges with intervening depressions. Once natural drainage system is disrupted, there is little or no run-off left during the rainfall. Water that gets logged on a ground infiltrates through spoils and waste to a newer and higher ground water table. This calls for a need to reclaim the land area for alternate use.

A number of problems of reclamation are associated with open pits. Open pits are characteristically so varied with individual topographical conditions and the climate. In this light, each operation has to be treated individually for reclamation. Wherever possible, revegetation too may be tailored to suit each case.

The process of reclamation needs also to involve people’s participation. It is necessary that public understands the problem and also what the industry can do in planning about conservation of land. It may also focus on what it is doing and what the ultimate beneficial results would be? They need to understand that even if reclamation is a practical way, it takes time, often years, before optimum results are identified distinctly.

1.6 Objectives of the Present Research.

In recent years, mechanized mining operations have gained significance with easy and safety in operations and capability to meet a target of higher rate of production. It, thus, ensures productivity within limited period of time. For it, utmost care is needed to regulate all the activities through mathematical results and economy in operation.It may involve optimum utilization of equipments and other resources including labour. The present day scenario illustrates that each industry takes advantage of the Operations Research and various other useful modeling tools to optimize resource utilization and to save cost of production. It, in turn, boosts up overall efficiency of the project.

The Strategic Planning Models and Operations Research techniques are successfully applied in different operations of the opencast mining. However, the results achieved show that much needs to be done. The situation is becoming more and more challenging nowadays particularly after the government decided to open its doors for private organizations including the F.D.I to enter the coal mining sectors. Under such a changing scenario, it becomes imperative to realize objectives of management with maximum possible efficiency, exercising the available techniques and methodologies in project implementation. Some key tasks fall in front of a manager and a planner to choose realistic targets in terms of investment, production, productivity, safety, conservation, research and development, welfare etc. They may be operative under largely varying geo-mining conditions at different stages of mining for its different subsystems.

Attempts have been made to mechanize the opencast mining system to enhance productivity. It may be highly capital intensive. Hence, better utilization of various tools, machines and equipments is essential. Without it no one can achieve the targeted production with economic means.

In course of the present study a field survey was conducted. It indicates that no optimal planning is done on the basis of some scientific methods. No serious efforts are made so far to boost up economical production and system development. This provides a scope to conduct a thorough study into the system to obtain improved results.

In this light, objectives are laid down for the present research. The basic objective of the present study is to apply the strategic planning models and operations research techniques to the open cast mining operations under various geo-mining conditions, management grounds and a variety of work culture. In course of this research, the area under study was put to close observation in view of changing situations. In order to achieve realistic objectives from application of strategic planning models and operations research techniques in opencast mines, it needs to work out the following objectives:-

i) To consider various drilling parameters to test feasibility of automation in drilling operations in the interest of enhanced efficiency.
ii) To consider various blasting parameters to minimize probability of vibration and damage with the opencast mining.

iii) To select or allocate shovel dumper combination in view of project requirements and automation in shovel-dumper combination system.
iv) To achieve targets as laid down at a planning stage and to ensure their proper scheduling and implementation with a view to enhanced efficiency and increased output.

In view of the above objectives, the present study further attempts to explore the following operational requirements by applying various OR techniques:

i) To implement the principle of Markov chain to find out working state probability and non-working state probability of the subsystems in working stages like drilling, charging, blasting and loading.
ii) To develop a Petri net based model to involve automation in drilling operations.
iii) To identify significant decision variables and parameters that may affect the performance of geometric volume of blasting.
iv) To establish functional relationships between various decision variables for optimal blast volume.
v) To develop a multi-variate regression model to evaluate the parameters that comply with statutory needs of the opencast mining in respect of limits of blast vibration.
vi) To apply Queuing model to shovel-dumper combination system to find out optimum number of dumpers in a shovel face.
vii) To develop dynamic resource modeling of shovel-dumper combination using the concept of Petri nets to have optimum assignment of dumpers to the shovels and to allow automation in shovel dumper combination system.
viii) To develop a Petri net based model for planning and scheduling of initial activities on conversion of a traditionally used PERT chart.

1.7 Motivation

The opencast mining operation involves man, machine and processes to be employed aptly. The processes may be listed as:

i) Drilling of blast holes.
ii) Charging of holes.
iii) Blasting
iv) Loading of OB/Coal
v) Transportation of OB/Coal to different destinations

These processes and operations are characterized with an influence of number of random factors. As these factors act on, they may decrease reliability of the technological operations. What contributes to productivity is usually efficient use of the entire system i.e. machine and manpower, with proper utilization of advanced technology. The present research proposes to apply Strategic Planning models and Operations Research techniques to some important opencast mining operations. It is a fact that Operations Research models will be helpful in a choice of alternative strategies to decision making.

The present study derives its motivation from a fact that the processes that may contribute to total mining operation in the opencast mining can be absolutely modeled through OR tools and techniques. It may be by virtue of operational parameters, changes in their occurrences, conditions constraining them and finally overall objectives as identified or associated with each process. It may further vary from cost reduction to enhancement of probability of certain stochastic operation such as drilling, charging of holes, blasting, loading and transport, planning and scheduling activities of opencast mines.

1.8 Organization of the Thesis

The present thesis is organized in eight chapters followed by a list of references.

Chapter-I : The chapter deals with general aspects of coal mining as well as some processes of opencast mining. The reserves and the demand-supply scenario of coal in India is focused on. The objective and motivation of the thesis are spelt out with a view to structuring an argument.

Chapter-II : The opencast mining methods are explained in detail. Mining of coal is among the most arduous activities that man is called upon to perform. In this context, growth of coal mining in India is reviewed. Exploration, intensification and mechanization of coal mining is reviewed in relation to the periods of the pre-nationalization and post-nationalization. The opencast mining operations are discussed and the area of study is spelt out on specific grounds. The factors that affect a choice of the opencast mining method are discussed in detail. Especially for the opencast mining methods, pit design and selection of various machinery are essential for smooth operation and economical production. This aspect is discussed in detail. A global survey of the machinery used in the opencast mining and their salient features are presented with due documentation.

Chapter-III : The chapter deals with various methods used in strategic planning and design of industries. They include Petri Net modeling and OR techniques. The discussion focuses on applicability of these methods and their effect on productivity. The chapter presents a critical review of publications focusing on applications of Petri nets and OR techniques in mining and allied industries in general and with special reference to the opencast mining in particular.

Chapter-IV : The chapter deals with Markov processes with discrete index parameters that are known as the Markov chains. The application of Markov chain analysis is considered in operations like drilling, charging, blasting and loading. Further, the Petri net based modeling of drilling operation is also formulated. The technique can be adopted for automation in drilling operations. It is considered as the most difficult and hazardous job in the opencast mining operations.

Chapter-V : Various blasting parameters are studied for optimal results. The decision variables in blasting such as bench height, burden, spacing, drill hole diameter, average charge/delay etc. are taken into account to formulate a multi-variate linear regression model. An analysis is carried out to establish the interrelationship so that blasting officers may take decision with confidence in respect of the loading of explosives into the drill holes. The chapter also deals with the vibration study in blasting operations using the regression analysis techniques. The maximum charge/delay is assessed for different distances of blast site from the structures so that vibration measured in terms of peak particle velocity (i.e. in mm/sec.) does not affect the structures and complies with the statutory requirements.

Chapter-VI : The Queuing model is applied for optimum use of dumpers to shovels. The Petri net modeling tool is applied to shovel dumper combination system where dynamic resource modeling concepts are used. This technique adopts the principle of fusion place method. The optimum allocation of dumpers to shovels too is analyzed and simulated.

Chapter-VII : The Petri net based modeling of initial activities of a project was earlier formulated on the PERT network. A concept of converting the PERT to the Petri net is, therefore, elaborated and analyzed. Advantages of the Petri net over the PERT charts are widely realized and, hence, they are recommended for use so that the monitoring can be effected with scientific approach in terms of their mathematical base of analysis capability.

Chapter-VIII : The chapter draws conclusions on the application of Petri net modeling and analysis thereof. Further, it reviews scope of future research in the application of OR techniques in opencast mining and recommends modifications in the present state of mining operations.

1.9 Summary

This part of the research deals with various aspects of mining operation, mining processes, supply demand scenario of coal in India, objective of the thesis and motivation. The organization of the thesis has been briefly discussed to focus some light on the research work carried out in the entire dissertation.

In order to have clear and neutral view about mining operations, it would be appropriate if we look at the mining methods so far followed and a new method that emerges and put to application in the recent times. The present research intends to review the recent methods of the Opencast Mining against the conventional one and the Underground Mining with a view to having clear picture about the pros and cons of the method in terms of work efficiency, cost-effectiveness, productivity and profitability. Since mining activities are large scale global level operations involving huge investment, labour and infrastructure, due attention has to be paid to such salient aspects to render benefits to the nation at large.

MINE WORKING METHODS

2.1 Introduction

Earlier references to the use of coal appear in the writing of Aristotle. The life to-day with fuzzy environment of liberalization, globalization and free economy brings home eventually, the hackneyed Darwinian concept of “Survival of the fittest”. The general complaint about the mining industry is that technological advancement has not kept pace with the evolution in comparison to other engineering disciplines. But the mining industry in its pursuit of providing the mankind basic raw materials for existence and growth always has to face a task of solving vast complexity of technical and scientific problems that arise out of its constant struggle with natural forces. Coal mining is changing fast in recent years with sophistication in mechanization, automation and computer control. Man-less mining, robotics, hydro-mining and underground coal gasification are distinctly visible as emerging technologies to capture operations in the future. In the present time of technological explosion, a mining engineer, a production manager, a planner and designer have to strive hard to keep pace to escape an allegation of being obsolete. In this light, the chapter focuses on various aspects of mine working methods.

2.2 Growth of coal mining industry in India

The first published reference of mining of coal in India dates back to the year 1774 when coal mining was initiated on a small scale at Raniganj coalfields. However, initially some set backs were felt due to poor quality of coal and lack of adequate transport facilities. Almost after five decades, there was a spurt in mining activities at the Raniganj coal fields. Following it, coal mines were opened up in quick succession during the 20th century in different parts of the country. They were connected with the Railways. As a result, coal mining gradually received considerable momentum.

The coal industry faced many vicissitudes. It then stabilized by the second half of the 19th century. By 1900, the coal production rose to 6.12 million tones. After India's independence, the Five-Year Plans projects implemented by the Government of India recognized the importance of energy in the interest of development of the country. Initially, it was the railways that remained a chief consumer of coal. It used coal in steam locomotives irrespective of its quality. As the steel industry developed, a new thrust occurred in the form of exploitation of coking coal from Indian coalfields. The power sector soon overtook these sectors and it became a major consumer of coal.

The new millennium brings forth changes in the Indian coal industry. It then finds itself ensnared along with the rest of the coal industry globally. Conflicting requirements of escalating demands of energy and conditions of Kyoto Protocol pose serious challenges to the coal industry, especially in developing countries that struggle with energy shortages.

The Opencast mining has witnessed a sea-change in the last 20 years. It contributes to raise the overall production of coal from 30% to 80% with a capacity of individual mines to reach in excess of 10 Mty. This unprecedented growth rate of nearly 12% per annum bears a testimony of successful design and execution of high capital projects that involve state-of-the-art of heavy mining machinery. With an emphasis on bulk coal production with the opencast mining during this period, the underground working played a supportive role. If continued with market demands in this sphere of the industry that centered more on the quality. Though the production from underground mines hovered around 57 Mty, a rich and varied experience could be acquired with several experiments and technological changes.

A map of India shows various coal fields (Fig.2.1). A map showing coalfields of Orissa is presented in Fig.2.2. A study of various parameters is conducted in relation to two coalfields in Orissa, i.e. Talcher and Ib-valley. The locations of various opencast projects operative and planned as well as those of virgin geological blocks are projected in Fig. 2.3 and 2.4 for Talcher and Ib-valley coalfields respectively.

2.3 Methods of mining

Coal mining methods can be broadly classified as the opencast or strip mining and the underground mining. The choice of techniques available for these two methods is guided by geological, technical, economic and environmental considerations. The geological factors are dominant to begin with. Depicting types of mechanical equipments together with deployment of labour force it can be efficiently used in winning coal. Seam characteristics such as thickness, depth, ratio of coal to overburden, inclination or dip of the seams, surface strata conditions, volume of coal that can be recovered, multiplicity of seams, etc. all these earmark possible methods and a range of techniques that are geologically feasible. Metallurgical characteristics such as coal quality, presence of shale, sand and other impurities, together with mining characteristics such as ventilation requirements, water-logging and fire conditions, etc., may obstruct a way to choose techniques. It further modifies the choice options. Finally, economic feasibility alone can guide mine designers and planners about a range of techniques that may formulate an investment plan for capacity expansion.

2.3.1 Opencast mining

The opencast mining uses shovel-dumper or dragline techniques. It is quite an advanced technique as compared to manual quarrying. These methods have advantages when coal seams are quite thick and are available at shallow depths. The Opencast mining delivers almost 40 per cent of the world's coal output and, consequently, it claims a very high level of investment as compared to other mining systems. In India, opencast production of coal has increased from 31 percent in 1979-80 to 50 percent in 1984-85. It claimed almost an 80 percent of the total by the year 2004-05. In this method, basically both overburden and coal are excavated after blasting rocks. They are then transported to stockpiles. Blasting operations depend upon geological structure of the rocks. Blast holes are generally drilled vertically, though the angle drilling often provides security to a high wall in the opencast mining. The maximum size of a blast hole varies from 160mm to 315mm in diameter. The size of a hole and a type of a drill used depend on site conditions. Rotary drills are generally used for drilling large diameter holes or for hard rock formations. In softer formations, augurs and drag bits are generally used. Percussion drills can be used upto 125mm diameter. Blast holes go as deep as 30-35 meters.

Excavation and transportation form a next stage in opencast operations. These operations can be carried out jointly or separately by different methods. It depends upon the prevalent conditions at a site. There are choices available whether to deploy draglines, shovels, dozers, scrapers or their combinations. It further depends on a type of material to be handled. Draglines perform two functions: excavating and conveying. With buckets of sizes varying from 10 cubic meters to 168 cubic meters and a boom length varying from 60 to 300 meters, the operations can be most economical, provided appropriate choice of the size and the capacity is made depending upon a size and a type of overburden and coal to be handled. Draglines can be utilized efficiently whenever seam or overburden thickness remains uniform around 10-15 meters over a large area. Its efficiency goes down if it is to be operated on multiple benches. Draglines are currently used in several coalfields in India, mostly in the Northern Coalfields Ltd.

In opencast fields, a shovel and dumper combination is used in excavation and transportation respectively. The conventional rope shovels as well as hydraulic shovels are used even though the former is cheaper in cost. The capacity of a shovel varies from 3 to 20 cum (size of the bucket). The size may be optimally chosen depending upon the haulage distance over which the dumpers travel and also inclinations of a road (affecting the turn around rate), and the amount of overburden to be handled. The recent developments in continuous conveyor haulage indicate that dumper transportation techniques are less economical.

Normally, it takes about 4 to 6 years to develop an opencast mine. The production may begin and the development continues in certain sections. Apart from this, the mine planning is much less complicated and the cost of coal extracted is generally lower than that obtained with an underground operation. Land Reclamation and removal of overburden may form two major cost components of this method, whereas technical parameters are the quality of coal and the coal overburden ratio. Since the opencast mining is not selective, the quality of coal is generally inferior to that obtained from the underground mining. On the other hand, the method has advantages of large scale mining and greater flexibility in production management. The elements of mining system are depicted in Fig.2.5 that elaborates on a location of equipment in a bench in the operational area.

2.3.1.1 Production equipment and methods

First, the mine boundaries and the various mine parameters need to be decided. The total quantity of coal to be extracted and total amount of overburden (OB) to be removed is calculated. A selection of the production equipments is the most important aspect of design in an opencast mining operation. Many factors, both physical and economical, have to be given careful attention. The decision will then affect the type, size and number of the equipments allotted for operations, such as draglines, shovels, dumpers, drills, bulldozers etc. A continuously increasing size of equipments for the overburden removal is influenced by the following important factors.

1. High production requirements to meet demands.
2. Need to handle increased depth of overburden.
3. High stripping ratio.

Normally a selection is made between shovels-dumpers and draglines or a combination based equipments looking to the situations governed by natural conditions.

2.3.1.2 Shovel dumper combination

This combination of equipments is most suitable for hard rock strata. Common shovel-dumper combinations being used are:

1) 3-3.5cum hydraulic shovel with 35t dumpers.
2) 4.6-5 cum rope shovel with 35/50 t dumpers.
3) 10cum rope shovels with 85-120 t dumpers.
4) 14.5cum hydraulic shovel with 170t dumpers.
5) 20.00cum shovel with 170t dumpers.

The tables: 2.1 & 2.2 show the shovel-dumper productivity of overburden and coal used in Indian opencast mines.

Bigger size of shovels with smaller dumpers, or vice versa, do not provide an optimum cycle time. Hence, it does not give optimum output. Transport of OB or coal by dumpers is a costly operation. If the volume and distance covered cross certain limit, it is better to replace/reduce use of dumpers with adopting a crusher-conveyor system for coal and OB. As far as coal is concerned, there are many opencast mines which use feeder-breaker-conveyor system to transport coal. It may restrict the use of dumpers in a mine from shovel to feeder-breaker only. The feeder-breaker can be shifted from one place to another with extension of a belt conveyor. (As the mine progresses).

Table – 2.1
Shovel & dumper productivity for overburden

OB	**HYD**	**ERS**	**ERS**	**HYD**	**HYD**	**HYD**	**ERS**	**ERS**
	4.3 cum+ 50T R/BODY	**4.6 cum+ 50T R/BODY**	**5.0 CUM 50T R/BODY**	**6.1 CUM+ 50T R/BODY**	**9.5CUM 85T R/BODY**	**9.5CUM 120T R/BODY**	**10CUM 85T R/BODY**	**20CUM 170T R/BODY**
LEAD(Km)	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**
0.5	0.2667	0.2285	0.2325	0.2850	0.4554	0.6463	0.4148	0.9126
0.75	0.2325	0.2006	0.2025	0.2451	0.3927	0.5694	0.3594	0.8066
1	0.2107	0.1823	0.1833	0.2200	0.3532	0.5191	0.3242	0.7397
1.25	0.1922	0.1669	0.1671	0.1992	0.3203	0.4762	0.2946	0.6739
1.5	0.1783	0.1551	0.1549	0.1837	0.2958	0.4437	0.2726	0.6240
1.75	0.1674	0.1459	0.1454	0.1718	0.2768	0.4180	0.2554	0.5846
2	0.1587	0.1386	0.1378	0.1623	0.2617	0.3974	0.2417	0.5531
2.25	0.1516	0.1325	0.1316	0.1546	0.2494	0.3804	0.2306	0.5271
2.5	0.1456	0.1274	0.1264	0.1482	0.2392	0.3661	0.2213	0.5053
2.75	0.1385	0.1213	0.1202	0.1405	0.2269	0.3489	0.2101	0.4799
3	0.1323	0.1159	0.1148	0.1339	0.2162	0.3338	0.2004	0.4581
3.25	0.1268	0.1113	0.1099	0.1280	0.2069	0.3206	0.1920	0.4386
3.5	0.1219	0.1071	0.1057	0.1229	0.1987	0.3089	0.1844	0.4216
3.75	0.1176	0.1034	0.1020	0.1185	0.1915	0.2984	0.1778	0.4063
4	0.1139	0.1001	0.0987	0.1143	0.1850	0.2889	0.1719	0.3927
4.25	0.1104	0.0971	0.0956	0.1107	0.1791	0.2804	0.1665	0.3804
4.5	0.1072	0.0944	0.0929	0.1074	0.1739	0.2727	0.1617	0.3691
4.75	0.1036	0.0913	0.0897	0.1036	0.1679	0.2638	0.1561	0.3565
5	0.1003	0.0884	0.0868	0.1002	0.1623	0.2555	0.1510	0.3449
5.25	0.0972	0.0857	0.0842	0.0971	0.1572	0.2480	0.1463	0.3343
5.5	0.0943	0.0833	0.0818	0.0941	0.1524	0.2410	0.1420	0.3244
5.75	0.0918	0.0810	0.0795	0.0914	0.1481	0.2346	0.1379	0.3152
6	0.0894	0.0789	0.0773	0.0889	0.1441	0.2285	0.1342	0.3068
6.25	0.0871	0.0769	0.0753	0.0866	0.1402	0.2228	0.1307	0.2988
6.5	0.0849	0.0751	0.0735	0.0844	0.1368	0.2175	0.1275	0.2914
6.75	0.0829	0.0734	0.0718	0.0823	0.1335	0.2126	0.1245	0.2845
7	0.0811	0.0718	0.0702	0.0804	0.1304	0.2079	0.1217	0.2780

(Source: CMPDIL reports)

Table – 2.2
Shovel & dumper productivity for coal

COAL LEAD (km)	4.3 cum+ 50T Coal Body 1	4.6 cum+ 50T Coal Body 2	5.0 CUM 50T Coal Body 3	6.1 cum+ 50T Coal Body 4	8.5 cum 85T Coal Body 5	9.5 cum 85T Coal Body 6	10.4 cum 85T Coal Body 7
0.5	0.3077	0.2762	0.2769	0.3519	0.5681	0.5945	0.6221
0.75	0.2698	0.2437	0.2427	0.3046	0.4964	0.5194	0.5397
1	0.2453	0.2223	0.2205	0.2746	0.4502	0.4711	0.4877
1.25	0.2244	0.204	0.2016	0.2496	0.4109	0.43	0.4436
1.5	0.2086	0.1901	0.1875	0.2308	0.3814	0.3992	0.4106
1.75	0.1962	0.1792	0.1763	0.2163	0.3583	0.375	0.385
2	0.1863	0.1704	0.1673	0.2047	0.3399	0.3558	0.3646
2.25	0.1781	0.1631	0.16	0.1952	0.3248	0.3399	0.348
2.5	0.1712	0.157	0.1539	0.1873	0.3122	0.3267	0.3341
2.75	0.1631	0.1496	0.1464	0.1778	0.2968	0.3106	0.3173
3	0.1558	0.1432	0.14	0.1696	0.2835	0.2967	0.3028
3.25	0.1495	0.1375	0.1342	0.1624	0.2719	0.2845	0.29
3.5	0.144	0.1326	0.1293	0.1561	0.2616	0.2738	0.2787
3.75	0.1389	0.128	0.1248	0.1504	0.2524	0.2642	0.2688
4	0.1346	0.1241	0.1208	0.1454	0.2442	0.2555	0.2598
4.25	0.1305	0.1204	0.1171	0.1409	0.2369	0.2479	0.2518
4.5	0.1268	0.1171	0.1139	0.1367	0.2301	0.2408	0.2445
4.75	0.1226	0.1133	0.1101	0.132	0.2224	0.2327	0.2362
5	0.1188	0.1097	0.1066	0.1277	0.2153	0.2253	0.2285
5.25	0.1151	0.1065	0.1034	0.1237	0.2087	0.2184	0.2214
5.5	0.1119	0.1035	0.1004	0.1201	0.2027	0.2122	0.2148
5.75	0.1088	0.1007	0.0976	0.1166	0.1971	0.2062	0.2088
6	0.1059	0.0981	0.0951	0.1135	0.1919	0.2008	0.2032
6.25	0.1033	0.0957	0.0927	0.1105	0.187	0.1957	0.198
6.5	0.1009	0.0935	0.0905	0.1078	0.1825	0.1909	0.1931
6.75	0.0986	0.0913	0.0883	0.1052	0.1782	0.1865	0.1885
7	0.0964	0.0894	0.0864	0.1028	0.1742	0.1824	0.1843

(Source CMPDI reports)

A use of feeder breaker conveyor system enables the use of smaller and simpler machines to load and transport coal. This increases a number of production faces for a given output and thus makes the quality of ROM coal, more uniform. A trend of an increased use of these equipments for the opencast coal mining is likely to continue because of its obvious advantages. Another incidental advantage of using several feeder breakers distributed at many places in the mine is that such a system avoids concentration of coal dust at one point and, thus, becomes environment-friendly.

It has been experienced that in India opencast mines that are deeper than 70m take acute problems of maintaining dumpers. Such a deep and large mine may profitably accommodate crusher-conveyor system for OB transport that may limit dumper movement within a mine from shovel to crusher and conveyor point/hopper to dump yard. A use of shovel-dumper-crusher conveyor system has been adopted for OB at W.C.L mines.

2.3.1.3 Dragline

The machine is used to remove soft or well fragmented material from above the coal seam and to dump it directly or through rehandling of the removed material on a previously decoaled area in a mine. It is a machine that loads as well as transports and dumps the OB within a mine area from where coal is extracted.

The single seam situation is best suited to dragline mining and balancing diagram of a dragline is presented in Fig: 2.6.

It is a common experience that contamination of coal by OB occurs more in a dragline mine than in a shovel dumper mine. It is due to the fact that, coal roof is not properly cleaned by a dragline. This, however, can be ensured by running a dozer above the coal bench. Greater amount of care has to be taken, therefore, during mining to ensure quality control of coal.

The quantum of OB that is removed by dragline every year (including rehandling, if any) determines the size of a dragline to be used. The size of a dragline is indicated by the capacity of bucket and its boom length.

2.3.1.4 Surface miner

In the early 1980s, mechanical rock cutting machines were introduced in the opencast mining industry. The initial applications were in lignite, coal, limestone and gypsum (Schimm, B, 2004).

Conventional opencast mining operations are invariably associated with drilling, blasting, loading, crushing and transportation activities. The efficiency of operations depends mostly upon the efficiency of drilling and blasting. Facing constantly public criticism and resentment for problems like blast damages, vibration, and noise etc., the opencast mining all over the country looked for equipment which would help to eliminate these problems. In the early 90's, introduction of continuous surface miners could find solution to these problems in Europe and elsewhere which was the beginning of eco-friendly mining operations (Biran K.K, 2000). It is a viable alternative to rock breakage eliminating drilling, blasting, loading and crushing operations. It may take care of complaints associated with these activities. The first surface miner was introduced in India in 1993. The surface miner has been successfully deployed in limestone and coal mines in India. It is now proved to be a revolutionary technology in the present era. In India, the first surface miner was introduced at coal mines in 2001. It was first seen at the Lakhanpur opencast project, Mahanadi Coalfields Limited, a subsidiary of Coal India Limited. The author was closely associated with the project in the capacity of Supdt. Of Mines/Manager at that point of time. The researcher got an opportunity to review the performance of a surface miner. An idea of using / applying a surface miner at mining operations occurred because of close proximity of a village. It blocked up some 7 lakh tonnes of coal for more than five years. Successful application of a surface miner at the Lakhanpur Opencast Project resulted in improved quality with selective mining and eco-friendly extraction of coal. It further prompted the mining communities in India, both private and public sectors, to use this versatile equipment in increasing numbers in order to meet their requirements of coal. The schematic diagram of a surface miner is presented in fig. 2.7.

Surface miners cut coal or rock with tungsten carbide tipped replaceable multi-cutting teeth that are mounted on a heavy duty cutting drum positioned to operate under the weight of a machine. The cutting drum is also called milling drum. It is equipped with picks across the full width of the machine in the form of helix. It facilitates propelling of the cut material towards the center of the machine. A cutting drum located between two sets of crawlers rotates in an upmilling direction and a layer of predetermined thickness (10mm to 200mm) of the deposit is cut and crushed by the picks. The thickness of the layer to be cut is controlled by means of an "Electronic Depth Control System". It electronically senses the depth of a cut to be executed.

Materials that is cut and crushed is picked up and transferred to a primary ribbed-conveyor belt. It is then taken to a discharge boom that extends out of the main body of a machine. It can be slewed on both sides and its height can also be altered as per the requirement of equipment to be loaded. The cutting drum is followed by a scraper blade which gathers any material left on the floor. It ensures clean and smooth floor without any undulations. Dust suppression is ensured by means of water spray arranged on cutting drum. It serves the dual purpose of cooling of picks that increases its life utility and it further leaves the working environment totally dust free.

A surface miner has two sets of crawlers. The front set has its own steering cylinders that facilitate negotiations on sharp turns. Each crawler unit and the cutting head can be raised or lowered by means of hydraulic cylinders provided for the purpose. These cylinders are controlled from the central control unit through sensors. A surface miner has a capacity to negotiate a maximum gradient upto 14^{o} (Raghavendra Rao, 1997). A machine should always be propelled along the gradient. It should never go across for safety considerations.

"Compressive Strength" is a major parameter that determines the suitability and efficiency of the surface miner technology. Strata with compressive strength of 5 MPa to 40 MPa are ideally suited to the operation of a surface miner. However, occasional encounter of strata upto 80MPa can be negotiated partly compromising with the rate of production and partly suffering from the damage of picks. This type of situation is put to study and reviews. Now in mining operations, rocks upto 80MPa can be cut at economic rates with the use of higher version of surface miners. Depending on the rock structure and its geo-mechanics, rock with a compressive strength upto 200MPa may also be cut. However, in general the compressive strength above 100 MPa substantially reduces the output. An increased wear of cutting tools has to be taken into consideration in this regards (Schimm, 2004). For hard and compact rock formations, a number of research works and pilot tests are carried out to suggest a novel method of blast free physico-chemical weakening of rock mass by means of surface active chemical substances in advance of actual excavation work. The method is commonly called "Surface Application System" (SAS).

Inclination of strata being mined, however, severely restricts the scopes of an application of surface miners. It can be generally stated that strata with inclination upto 10^o from the horizontal can be mined with surface miners without its appreciable fall in the output levels. Contamination of pay mineral and dilution loss are dependent on inclination of strata. It can be shown as:

Inclination	Likely contamination & dilution/loss
2^o	4.0%
5^o	8.6%

(Source: Manufacturers' Brochure)

Another parameter that limits the scopes of application of surface miners is the "water content" in the strata. In case water content exceeds the plastic limit, an operation of a surface miner is severely restricted.

Another strata parameter that determines wear of cutting tools of a surface miner is "free silica content" if it exceeds 10% it brings about excessive and uneconomic wear of cutting tools. In this respect, it is an important limiting factor.

The optimum working length differs much depending on the traveling speed and hardness of a rock. With harder rocks the traveling speed remains low (e.g. 5m/min.) and the optimum working length is approx 250m for a 2200SM model Wirtgen make. With soft rocks, the traveling speed remains high (e.g. 20m/min), and the optimum working length is approx. 900m or more.

There are a number of surface miner manufacturers across the world. The technical data of major manufacturers of surface miners are presented in table no.2.3 below:

Table – 2.3

Technical data of different models of surface miners

Sl. No.	Item	KRUPP	Wirtgen (Germany)				Hunon Company		Bitelli SF200m
		200R	2200	2600	3700 SM	4200SM	Easi miner 1018M	Eas iminer 1224	
1	Overall length (m)	43	16.8	19.27	25.3	28	-	28.35	15.4
2	Overall width (m)	7.0	2.75	3.1	5.0	5.5		4.87	2.5
3	Overall height (m)	13.6	4.8	4.75	8.2	8.2		7.85	3.8
4	Length of discharge conveyor (m)	15.6	8.82	10	12	15	13.72	15.25	7.2
5	Height of discharge (m)								
i)	Minimum	3.9	3.00	3.6	4.3	5.07			2.0
ii)	Maximum	12.55	4.8	6.445	7.0	8.5	7.10	7.92	3.9
6	Slewing angle of discharge conveyor (deg.)	!93	45-55	!90	!90	!90	!110	!105	!40
7	Diameter of drum bucket wheel (m)	4.8	1.115	0.95	1.4	1.86	1.52	1.22	2.0
8	Rated output	2000 1.cum/hr	70 bcum/hr	360 bcum/hr	860 b cum/hr	1250 bcum/hr	1200 tph	1633 tph	250 tph
9	Cutting depth (max) (m)	2.9	0.24	0.25	0.6	0.8	0.457	0.61	0.2
10	Cutting width (m)	7.1	2.2	2.6	3.7	4.2	3.5	4.14	2
11	No. of picks/bucket	4x15	76	Variable	Variable	Variable	38	44	87
12	Cutting speed (m/sec)	1.53	0.45	0.41	0.33	0.33	0.76	0.76	4.5
13	Loading conveyor belt width (m)	1.6	1.0	1.0	1.8	1.8	1.37	1.83	0.9
14	Loading capacity (t/hr)		550	845	2400	2400	1.83m		8.00

(Source: Manufacturers brochures and leaflets)

A Surface miner is called “the total mining machine”. It can excavate, size and load material in one single go without prior face preparations. The advantages and limitations of using a surface miner may be outlined as below:

Advantages

- λ No drilling and blasting is required.
- λ Direct loading of cut material (coal) into the truck for transportation to the siding directly.
- λ No chance of fire as the cut leaves behind a hard surface.
- λ No primary crushing is required as the size is <100mm.
- λ Minimum deployment of men and machines at a face.
- λ A surface miner leaves behind smooth surface and, thereby, it reduces the transportation cost with less wear and tear of tyres.
- λ Maintenance of minimum types of machines and hence small inventory of spares.
- λ Easy to control and monitor production to enhance productivity.
- λ Uninterrupted production is ensured on a sustained basis.
- λ Eco-friendly method of mining of coal reducing or minimizing no hazards of dust, noise, vibration, fly rock, air blast, etc.
- λ Selective mining of coal becomes possible. It, thus, improves the grade of coal and eliminates dirt bands from coal seam more than 10cm and dumps those in the dump yard.
- λ Better and concentrated area of supervision.

Limitation

- λ Rocks of compressive strengths more than 100MPa are not economically viable to the surface miner technology.
- λ Maximum gradient is 14^o where it can work.

2.3.2 Underground mining

While a choice of coal mining technology is necessarily deposit-specific and based on the economics of investment allocations, there is no better alternative to the underground mining for conditions like the deposit is deep-seated and the ratio of overburden to coal is too high for any cost effective mining.

Basically, the underground mining involves operations like reaching a coal seam either through an incline or a shaft cut through the overburden and lifting it to the surface. Development of tunnel roads may be an essential requirement to reach the seam to an appropriate location. It needs to consider the seam thickness, strata conditions and methods of underground mining. Coal cutting, blasting, picking, loading, transporting are carried out at different stages of mining. They may vary in methods and the intensity of mechanization. The underground mining requires a huge labour force at all these stages in varying proportions depending upon the technique applied. Thus, the mining operation becomes more labour intensive as compared to the opencast mining method. It takes about 8-12 years to fully develop an underground mine.

There are two distinct types of mechanization applied in the underground mining. They are the bord and pillar method and the longwall method. The bord and pillar method is the most popular and highly practiced system of mining in India since the inception of the coal industry. The method is suitable particularly in India due to hard and strong roof conditions. The method consists of driving a series of parallel roads or tunnels to be connected with cross-roads. In the process, a cluster of pillars are constructed to separate the tunnels. There are two stages in this method. The development stage is meant to leave the panels and the depillaring stage is meant to extract coal from these panels.

In case of longwall mining, an advance or retreat method is adopted. In the 'advance' method, coal cutting is synonymous with mine development and hence the gestation lag between mine design and coal production is less. In the 'retreat' method, mine development up to the boundaries of the coal seams is carried out first. Coal production is carried out in a retreat phase. With this, more geological information at a production stage is available and, hence, the production plans can be suitably implemented. Particularly, when a mine is highly mechanized, maintaining high capacity utilization is important. Because of it, the retreat method ranks better. In the either option, while coal is removed, it leaves a void or a decoaled area (goaf). It can be either left as it is or allowed to break down or collapse. This method of mining is known as caving. Alternatively, the void can be filled up with sand, crushed stone, etc., so that the roof does not collapse; thereby it protects the surface strata and the habitat. This method is known as stowing.

2.4 Mechanization: Pre-Nationalization Years

The growth of the coal industry in the pre-independence era and during first two decades after India's independence remained very slow (table-2.4) (Kumar, 1996). The country's annual production of coal in late sixties and early seventies hovered around 70Mt.

Table – 2.4

Coal production in India Pre-Nationalization Era

Year	Coal Production (Mt)
1850	0.12
1860	0.30
1870	1.02
1880	1.74
1890	2.46
1900	6.12
1910	12.25
1920	17.09
1930	22.68
1940	29.85
1950	32.51
1960	55.66
1970-71	72.94*
1972-73	77.22**

(Source: Kumar, 1996)

* Nationalization of Coking coal (1971)

** Nationalization of non-coking coal (1973)

Most of the Indian mines relied on their early years on working in shallow deposits and small areas that did not need any application of mechanical power. Labour was available to them in abundance and cheaply. The first recorded instance of an application of machinery was the use of a 10HP steam winder in 1852 at the Raniganj coalfields. Further, in 1920s, steam power was fairly popular in the industry.

Electricity as a source of power came in to use since 1906 at the Sodepur Colliery of the Bengal Coal Company with 400KVA power station. The first coal cutting machine called a bar coal cutter which was introduced in India between 1906 and 1907. In 1922, some 40 coal cutting machines were used in Indian coal mines. They increased to 125 in 1925, 400 in 1950 and 700 in 1960. In the 1950's and 1960's, the machinery used in Indian coal mines were limited to only handhold electric drills and coal cutting machines. Even in larger mines, the Pick mining was confined to smaller mines and was gradually replaced with blasted coal. It became substantially cheaper with increased productivity.

During the period of 1900 to 1950, a low degree of mechanization is evident during this period. It is from the fact that only about 9% of the total output was machine-cut coal in 1945 and only about 16% of the mines were electrified though they produced about 65% of the total output.

2.5 Mechanization: Post-Nationalization Years

The manual "Bord & Pillar" method of mining is still the predominant system of underground coal production. A major portion of manpower is engaged in manual loading of blasted coal into coal tubs or mine cars of one or two tonne capacity. This involves carrying loaded baskets to some distance.

During two decades of the post independence era, scenario of coal experienced stagnation, the production at a level of around 73 million tones. But from the year 1971-72 and with first Nationalization of coal mines in India it has witnessed progressive increase to rise as high as around 376.78 million tones in 2004-05. It registered India as the 3rd largest coal producer in the World. The trend of production in the successive Five Years Plans is projected in the table-2.5 below.

Table-2.5
National coal production in the terminal years of Successive Five Year Plans

Terminal Year	**Five Year National Plan**	**Production (million tones)**	**Annualized Growth Rate over Previous Plan**
1973-74	IV	78.17	
1978-79	V	101.95	5.23%
1984-85	VI	147.41	6.34%
1989-90	VII	200.89	6.39%
1996-97	VIII	285.63	5.15%
2001-02	IX	327.79	2.79%
2002-03	1st Year Xth Plan	341.27	4.11%
2003-04	2nd Year Xth Plan	361.17	5.83%
2004-05	3rd Year Xth Plan	376.78	4.32%
2006-07 (Projected)	X	405.00	4.37%

(Source: Kalam, 2005)

The CMPDIL drew number of project reports employing different types of face loading equipments, viz.scraper, loader or slusher to collect blasted coal and scrap it onto a light duty chain conveyor near the face and load-haul-dumpers to load the blasted coal at the face, to haul and dump it into the district transport system. During 1975-77 indigenously manufactured slushers were introduced on trial runs. 27 such devices were introduced in development districts and 10 in depillaring districts. The device was simple in operation and was cheap at use. The results were satisfactory in fairly flat seams (upto 10 degrees) and were having an average thickness of 2 to 3 meters. Simultaneously, trial runs were carried out with imported crawler mounted side discharge loaders (0.6 cum) capacity of two different models in 12 mines involving 28 machines. The result was rather mixed, as one model did not function satisfactorily and the other did give encouraging results. It was found that in fairly flat (10 degrees or less) and moderately thick seams (1.8 to 3m), with good floor and roof conditions, the machine yielded expected results. However, instead of fast moving development of face drivage, it was preferred for depillaring and drifting.

Side Discharge Loaders (SDLs) and Load Haul Dumpers (LHDs) are now indigenously manufactured. These machines are presently used extensively at underground mines in India. In order to obtain more coal per round of blast in development faces, an Auger-cum-drill has been developed indigenously. These machines are yet to be accepted by users. Currently, machines with 19 Auger-cum-drill are on the roll of the CIL

In addition to loading machines and drilling equipments, the steps are taken for improving the Bord & Pillar system. The Roof Bolting and Rope Stitching methods of support too are introduced on a large scale to facilitate the application of loading machines in both development and depillaring areas. In the Bord & Pillar system deployment of continuous miners and road headers are actively considered for speedy work of depillaring and development in suitable geo-mining situations. After the Nationalization of coal mines in India, most noteworthy achievement was integration of the coal sector. Small mines were amalgamated to make reasonably larger units. Out of 750 coal mines that were nationalised, some 430 coal mines were reformulated. The manpower employed at mines was more than 750,000 at the time of nationalization. It is now (2005-06) rationalized and reduced to a reasonable strength of 4, 60,000.

2.6 Brief Scenario of Opencast Mining in India

The coal industry has succeeded in adopting and absorbing the state-of-the-art technology in the opencast mining. The Shovel-dumper system remains the mainstay technology. Shovels of $10m^3$ capacity in combination with pay load dumpers of 120 tonne capacity form a common system in several of large capacity opencast mines, like Gevra, Dipka, Jayant, Dudhichua, Nigahi, Sonepur Bajari etc. The largest dumper put to use in India is that 170 tonne capacity and it is at the Rajmahal project. A number of mines deploy walking draglines for stripping operations. The Piparwar opencast mine where mobile crushing and conveying system is successfully operating has added another feather to the value of this technology upgradation. Electric rope shovels of upto $25m^3$ bucket capacity is working successfully. A wide range of hydraulic excavators prove successful for selective mining and medium hard strata condition. A wide scale application of surface continuous miners at several projects of the MCL is extremely encouraging both from the point of view of selective mining as well as enhanced output. Draglines upto $30m^3$ bucket capacity and 96 m boom length are found quite effective especially due to thick seam situations.

Some technological milestones in the field of the Opencast Mining in India may be spelt out as follows (table – 2.6)

Table – 2.6

Technical Milestones in Opencast Mining

1990-91	Inpit crushing and conveying at **Padampur Project** of WCL.
1993-94	Commissioning of $25m^3$ Shovel and mobile crusher at **Piparwar Mine**.
1994-95	Inpit curshing and conveying at **Ramagundam Opencast Mine Phase-II**
1999-2000	Surface continuous miner at Lakhanpur OC Project at **MCL**

(Source: Kalam, 2005)

The application of surface miners (448 kW) proves a great success on several counts. Production capability and quality enhancement by selective mining acquire validity with its contribution to improved economics through application of this innovative technology. The trials conducted at a pioneering stage have bolstered confidence of the coal industry. Using the state-of-the-art technology of surface miners coupled with rapid ash analysis with ash probe the industry, opens up a wider horizon for quality enhancement.

The CIL was assigned with a task that was both ambitious and highly demanding. Looking at the urgency as well as the vital importance of the task in hand, the CIL rightly placed its priorities and thrust on opening up a number of surface coal mines. Large scale and extensive coal exploration activities were then conducted to identify shallow coal reserves.

High demand of coal can be admirably met with a phenomenal increase in the coal production that may result from the opencast mines to meet the ever growing needs of the economy.

2.7 Global Advancement in Opencast Technology

Progressive advancement in design of new diesel engines, truck tyres and transmission systems has led to continuous increase in the size of haul trucks. While in the early eighties the largest haul truck was designed for 170 tonnes pay load at the end of the decade a wide scale use of trucks of 240 tonnes capacity was witnessed in the West. A world wide trend of surface coal mining shows preference for trucks with heavier payload. A fleet size would depend on site-specific conditions. Most mines in the USA enhanced the fleet size of trucks from 240 tonnes to 270 tonnes by the end of the century while some had gone even upto 340 tonnes size with gross horse power of 2700. With the development of commercial haul trucks of 325 to 360 tonnes pay load by the start of this decade, it may occur to one that time is not far when 400 tonnes capacity haul trucks shall be widely used abroad.

After trucks, excavators upto $50m^3$ bucket capacity are available to load large size dumpers in 3-4 passes. However, high capital requirement for such loaders and reliance on electric power promotes selection of hydraulic machines or wheel loaders. In recent years, hydraulic excavators with $25m^3$ size gain wider acceptance. Manufacturers develop larger machines (upto $28m^3$ capacity/1800 hp) to be used as primary loaders at several locations.

Large draglines too find considerable acceptance in major surface coal mining areas. But their application is highly specialized. These machines find their wider application in Australia, South Africa, Canada and India with the largest number in Australia. The bucket of some of these machines is $122m^3$ with boom length 109.7 to 128m and with maximum digging and dumping height of 60m (Kalam 2005).

Recent developments in conveyor capability and performance enable coal mining technology to advance at an accelerated pace with it, while they remarkably reduce the capital requirement and operating costs. Improvement in belt and drive technologies too allow longer flights and higher lifts. Further, the high angle conveying solves several problems of conveying coal and other materials. In some areas, high-angle conveyors enable engineers to design systems that can lift coal and other materials vertically into silos and load/unload ships at rates measured in thousands of tones per hour.

High capacity dozers (upto 860hp) are operating successfully at several mines. In the next few years, one may see a launch of dozers of excessive size of 1000hp. Drilling operations are brought to an arena of electronification and automation. As the development of GPS, remote diagnostics and other peripherals is accelerating; drills may acquire full automation in near future.

2.8 Technology Shift from Underground to Opencast

Phenomenal growth can be attained in coal production in the country only with laying greater emphasis on the Opencast Mining Technology. Both from the considerations of volume of production as well as cost of production, the Opencast Mining prove its supremacy. Additionally, conservation with greater recovery of coal earns to the Opencast Mining wider acceptability for mine operators. An exercise carried out two years back by the Working Group on Coal shows that in India the cost of coal per billion calorie works out to US $ 5.476 for underground mines and US $ 2.18 for opencast mines. It is against the global mining cost of US $ 2.42 and US $ 3.54. This may be the reason that the Opencast Mining flourishes well in India. The share of the Opencast Mining increased from 26% (20.77 million tones) in the year 1974-75 to 82.2%

(298.41 million tones) in 2003-04. Against it, the share of the underground Mining declined from 74% (58.22 million tones) to 17.38% (62.76 million tones) during the same period. The table-2.7 shows the technology wise break up of coal production in the last 11 years (Kalam, 2005).

Table – 2.7

Technology-wise national coal production past-11 years

Year	Opencast		Underground		Total production (Mt)
	Production (Mt)	% Share	Production (Mt)	% Share	
1993-94	174.71	70.25	73.98	29.75	248.69
1994-95	185.79	72.08	71.98	27.92	257.77
1995-96	201.52	73.71	71.89	26.29	273.41
1996-97	218.37	75.48	70.95	24.52	289.32
1997-98	231.37	77.02	69.03	22.98	300.40
1998-99	228.75	77.15	67.76	22.85	296.51
1999-00	237.28	78.02	66.83	21.98	304.10
2000-01	247.63	78.94	66.07	21.06	313.70
2001-02	262.97	80.23	64.82	19.77	327.79
2002-03	277.67	81.37	63.58	18.63	341.25
2003-04	298.41	82.62	62.76	17.38	361.17

(Source: Kalam, 2005)

The coal production from the Opencast Mines in India was 16.40 million tones in the year 1973-74. It has been a spectacular achievement in the post nationalization period that has witnessed an average growth in opencast coal production at the rate of 12.0%. It is unparalleled in the history of mining. At the same time, considerable efforts and investment of US $ 1.46 billion are employed in the underground mines in order to maintain the production level with a different technology mix. The enormous task of reorganizing and restructuring underground mines that are haphazardly and unscientifically managed was undertaken during the nationalization. Along with an additional problem of surplus manpower and socio-economic factors, has reportedly increased the production from opencast mines. Overall performance of opencast and underground mines in India is outlined in the table 2.7.

2.9 Growth in productivity

At Opencast mines, employees' productivity has reportedly improved steadily in the post Nationalization period. It is evident from the table-2.8. There has been an average growth of 9.1% in the man productivity in the last 23 years.

Table – 2.8
Man productivity of opencast and underground mines

Year	Coal India Limited			Singareni Collieries Company Limited		
	Underground	Opencast	Overall	Underground	Opencast	Overall
1981-82	0.55	1.90	0.77	0.85	3.43	0.92
1982-83	0.52	1.99	0.79	0.78	4.16	0.88
1983-84	0.53	1.97	0.81	0.77	3.47	0.86
1984-85	0.52	2.07	0.87	0.77	3.50	0.87
1985-86	0.53	2.24	0.92	0.87	3.54	0.92
1986-87	0.55	2.46	0.99	0.82	3.08	0.95
1987-88	0.54	2.68	1.08	0.78	3.94	0.94
1988-89	0.57	2.88	1.17	0.76	3.70	0.95
1989-90	0.55	3.08	1.21	0.71	3.81	0.96
1990-91	0.53	3.31	1.30	0.65	4.74	0.96
1991-92	0.53	3.70	1.40	0.67	4.48	0.98
1992-93	0.55	3.80	1.46	0.70	4.46	1.04
1993-94	0.55	4.00	1.52	0.71	4.38	1.05
1994-95	0.56	4.35	1.63	0.69	3.63	1.08
1995-96	0.56	4.73	1.75	0.74	3.66	1.23
1996-97	0.57	5.12	1.86	0.72	2.96	1.19
1997-98	0.57	5.07	1.93	0.76	3.50	1.31
1998-99	0.59	5.52	2.03	0.75	3.92	1.31
1999-00	0.61	5.46	2.11	0.75	4.43	1.42
2000-01	0.63	5.92	2.30	0.79	7.29	1.50
2001-02	0.64	6.09	2.45	0.85	6.74	1.66
2002-03	0.69	6.30	2.67	0.86	7.66	1.89
2003-04	0.68	6.66	2.82	0.86	7.69	1.81
2004-05 (Provisional (April-December)	0.68	7.11	2.93	0.82	8.24	1.86

(Source: www.coal.nic.in)

Note:NCL and SECL were set up w.e.f 1.1.1986,MCL was carved out of SECL w.e.f 3.4.1992

2.10 Technical characteristics of coal seams

The technical characteristics of Indian coal seams and associated strata vary widely. They require different approaches to exploit coal in different situations. The technologically relevant characteristics are mentioned below:

- Thickness
- Association
- Inclination
- Quality
- Strength of coal seams
- Liability in spontaneous heating
- Gassiness
- Hydrogeology

2.11 Factors Affecting a Choice of Opencast Mining Methods

The basic principles governing a selection of a mining method are technical feasibility and economic expediency. Technical feasibility should ensure that expected production is achieved with safety. In Safety due regards is demanded mostly by the law. It also helps the economy of production and enhances the present reputation of a mine with reports of minimized hazards involved in mining. For a given mineral deposit, there may be several technically feasible methods of mining. A choice of a right method depends on assured efficiency or economy of the method. Such a method gives the maximum financial advantage, so it becomes an obvious choice. Financial advantage is then taken as the total profits earned over whole life of a mine.

However, maximization of economic benefits would not always be a sole consideration to select a mining method. In case of strategic minerals that are usually in short supply in a country, an utmost care has to be taken to ensure economic conservation of these minerals. The fullest extraction of such precious mineral would be preferred to maximization of financial benefits. It has to be treated as a governing factor while selecting a mining method, provided it is technically feasible.

Based on the above principles, a choice of an opencast mining method would depend on the following major factors:

- Nature of mineral deposit
- Stripping ratio
- Desired production and degree of mechanization
- Allowable dilution, degradation and loss of mineral
- Capital available
- Cost of mining
- Surface topography
- Climate
- Availability of labour and equipment
- Management efficiency

2.12 Strategies for future opencast technology in India

In view of an increase in demand of coal production from opencast mines, the strategies to be adopted in future would be :

- Opening more green-field projects of high unit capacity (12-25mtpa) to suit the deployment of higher size equipments.
- Forging 'annual maintenance contracts' along with spare parts availability with suppliers of HEMM.
- Replacing obsolete equipments with a higher quality fleet that would reduce the cost of operation and maintenance.
- Ensuring machine availability and its optimum utilization through a use of IT-enable systems, for enhancing overall productivity and to make them more cost competitive.
- Introduction of surface miner on a wider scale to permit selective mining for quality control as well as for bulk production at a reduced cost. The future is poised for larger capacity surface miners suitable for cutting harder coals and interbands in several coalfields. Technological developments may be applied to find an exact level on which a surface miner may work. It may call for incorporation of Global Positioning System (GPS) into machine design. Along with the digital mining plan, it can be used on board. The machine may provide operational data. This will facilitate separation of different grades of coal and reject stone

bands. This task is carried out presently manually with individual experience.

- The future opencast mines should aim at planning its operations in such a way that the entire gamut of operations generates no waste in the process. The cycle of operations may include methane exploration and its extraction followed by top-soil extraction and preservation, removal of overburden, coal extraction, backfilling of OB material in the quarried area, technical reclamation, top soil topping and finally biological reclamation to restore the land to its original form.

These strategies may enhance the value of the Opencast Mining Method with enhanced productivity, reduced cost, economic conservation of material and above all safety assurances in operations. They call for open minded attitude from Indian miners to receive and apply recent technical know how and professional commitment to ensure quality and efficiency at coal mining over financial capabilities and technical efficiency, what count most is commitment to professional ethics to allow cost efficient and utility efficient production of coal that forms the backbone of the energy sector in the present context. However, it would be wise what others speak out of their experience of the method. It may count as expert opinion and an experienced person's view to render valuable insight. With this view in mind, we would move further to conduct a review of the literature on the subject and application of the method.

COALFIELDS OF INDIA

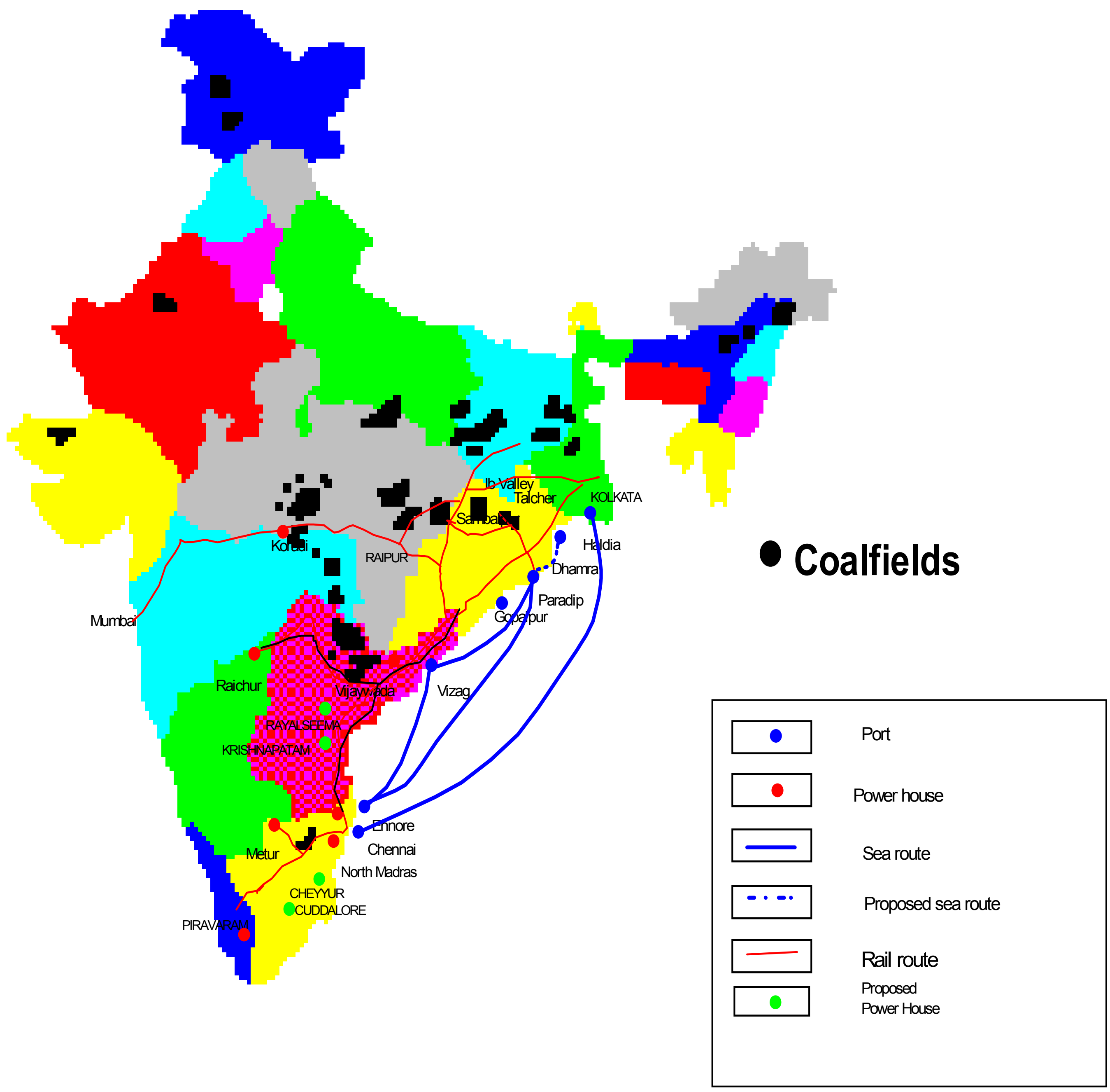
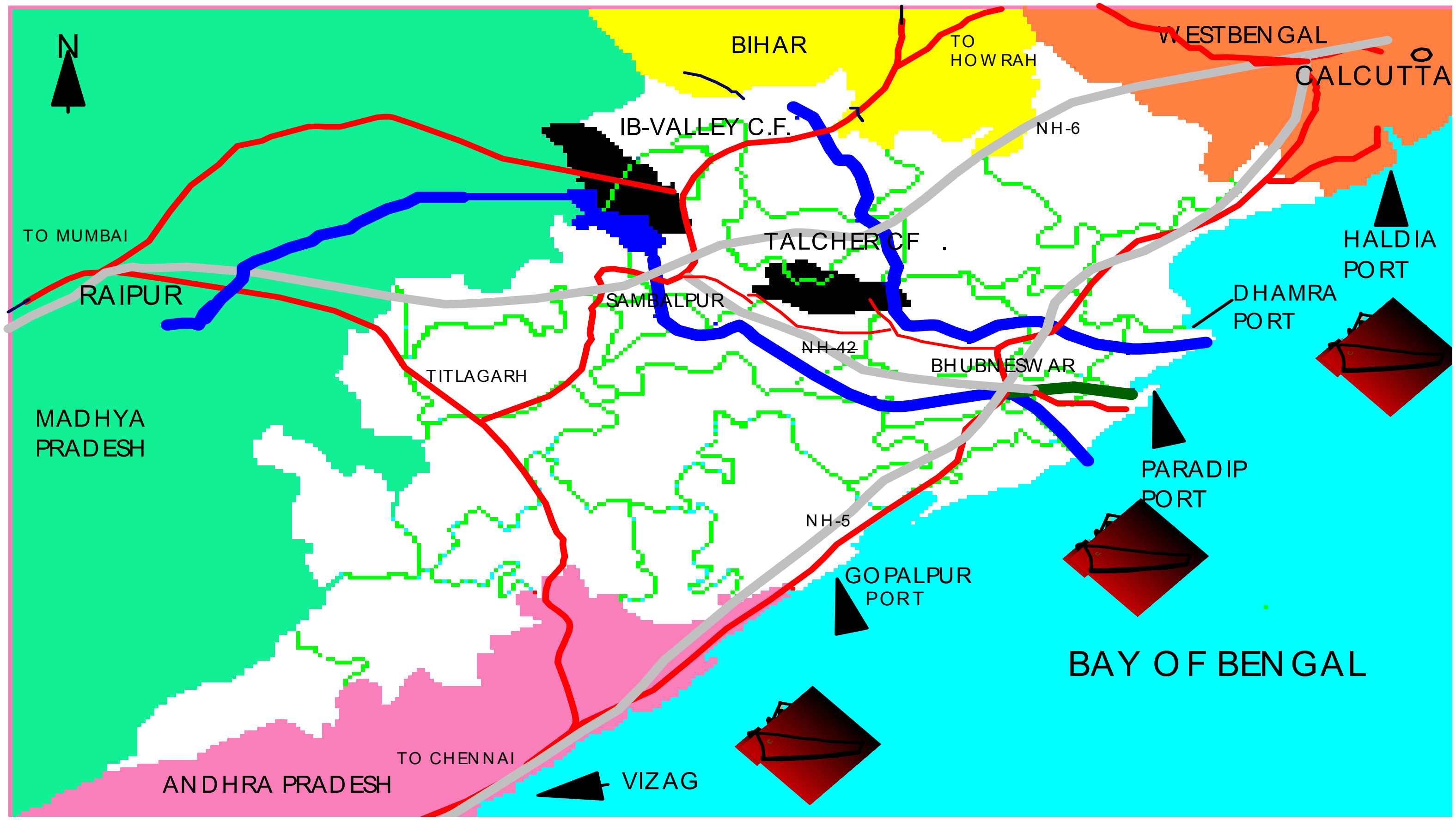


Fig 2.1: Coalfields of India

A MAP OF ORISSA SHOWING COALFIELDS



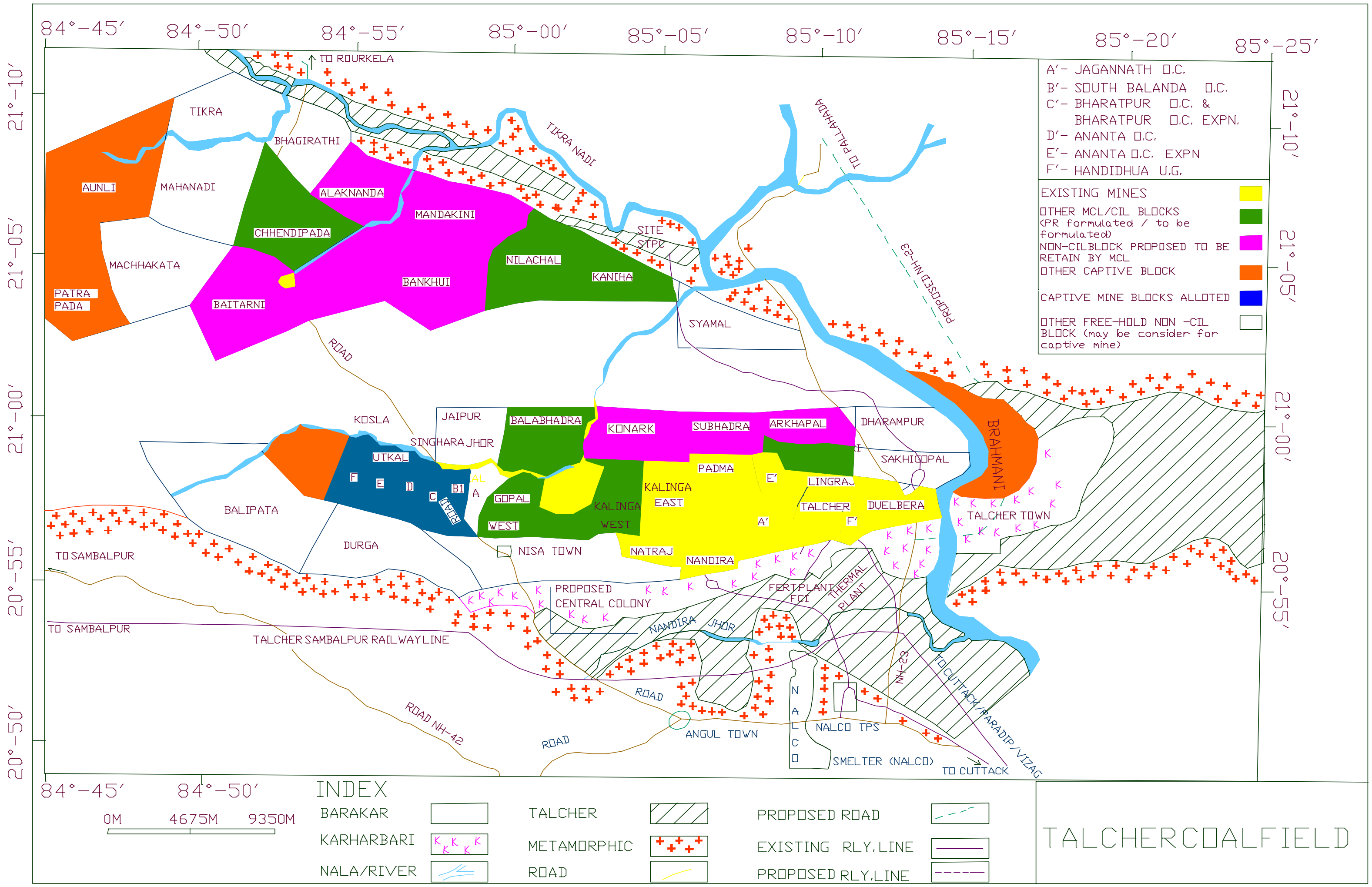


Fig.-2.3 Talcher Coalfields

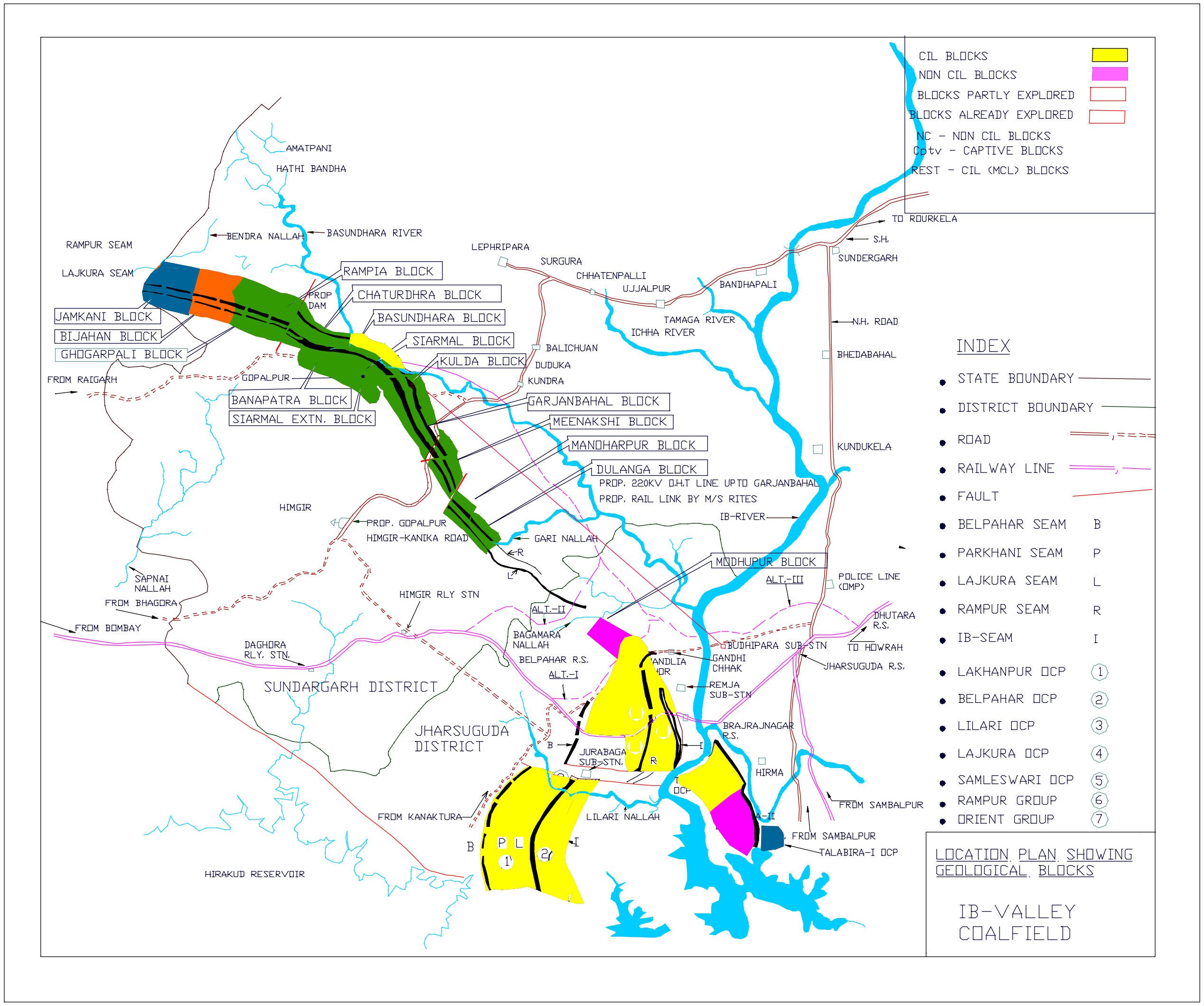


Fig. - 2.4: Ib valley coalfields

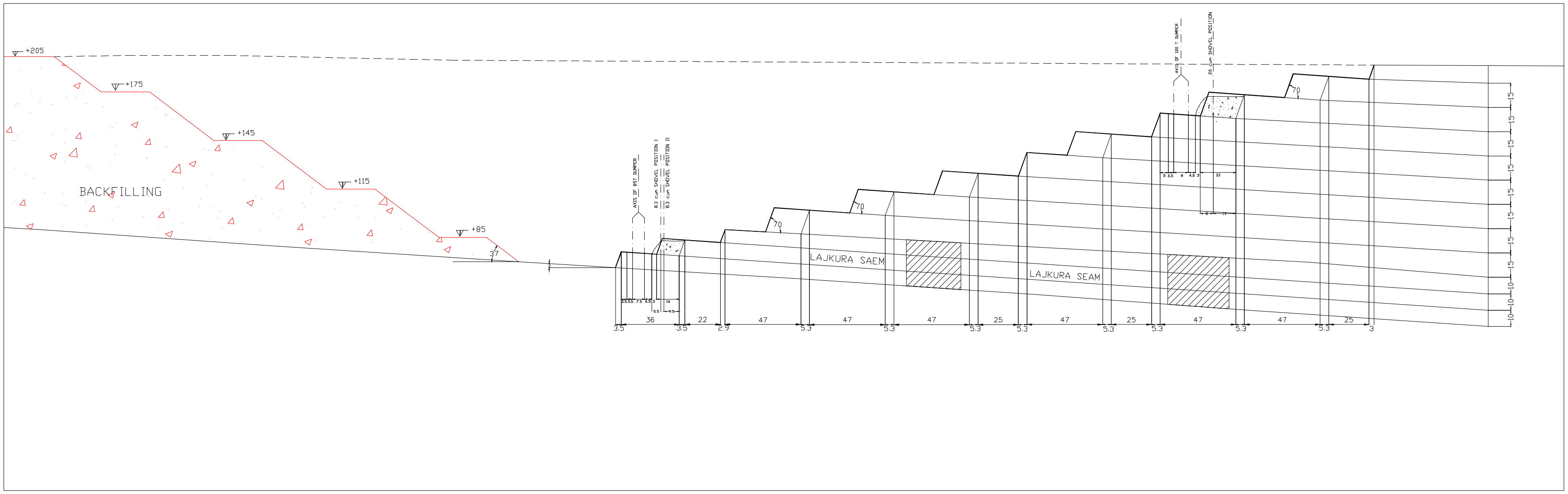
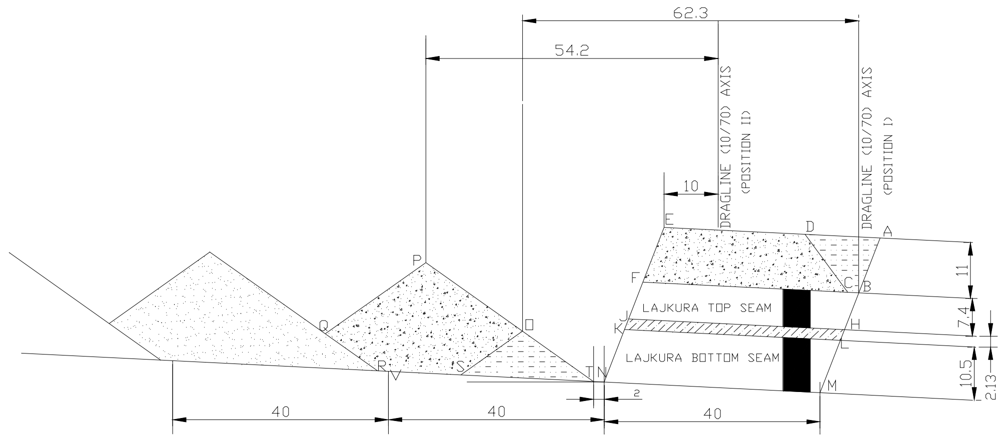


Fig.-2.5: Elements of Mining



LEGEND

	LAJKURA SEAM
	INSEAM PARTING
	DRAGLINE MAIN CUT & DUMP
	DRAGLINE KEY CUT & DUMP

PARTICULARS		CUT	DUMP	AREA	QUANTITY PER ANNUM
DRAGLINE	KEY CUT	ABCD	OST	88	1.26 Mcum +.04 Mcum (misc.REHANDLING) =1.30 Mcum
	MAIN CUT	CDEF	OPQRSO	352	
COAL BY SHOVEL & DUMPER		BFJH & KLMN	CHP	715.2	2.80 MTY
PARTING BY SHOVEL		JHLK	INTERNAL DUMP	85.2	0.24Mcum

DRAGLINE BALANCING DIAGRAM

Fig.- 2.6 : Dragline Balancing Diagram

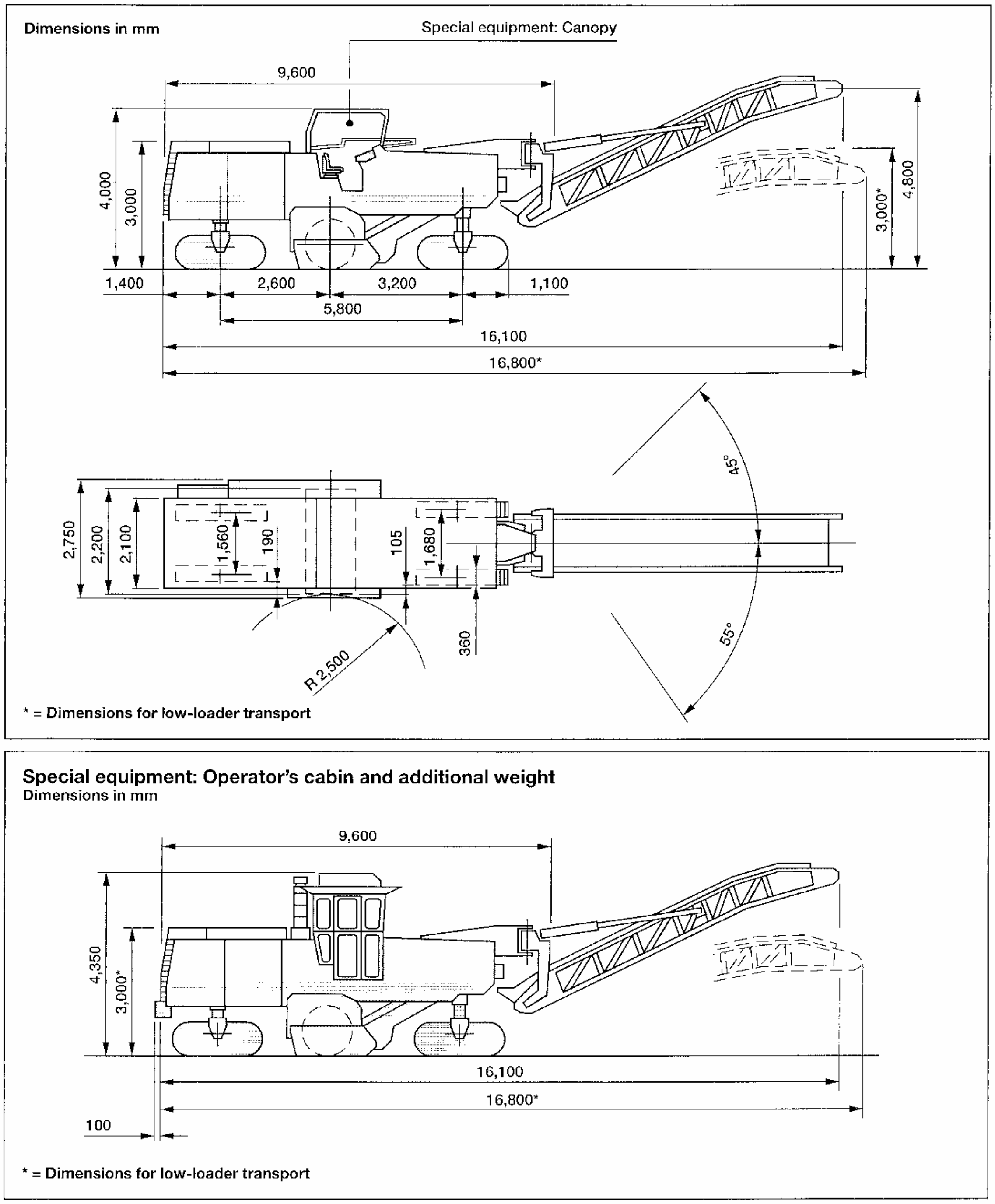


Fig- 2.7: Schematic diagram of surface miner

LITERATURE REVIEW

3.1 Introduction

Industrial venture calls for proper and scientific planning to achieve the target efficiently and economically. To improve productivity and to be competitive in global markets, due importance has to be granted to basic and applied research that aim at solving technological and management problems pressing hard upon entrepreneurs. Engineering management today is better advanced and more comfortable with mathematical and computer models. Automation of basic administrative functions and word processing encourages acceptance of personal computers for spreadsheet and small data base applications.

Operations Research (OR) emerged as major field of study during the World War-II. It was accepted for application with a view to solving pressing military problems such as use of the radar to detect an enemy's air craft and initiate appropriate actions at appropriate time. Its successful experience emphasized models to guide decision making and planning and to improve system design and performance. The OR has evolved a wealth of knowledge with numerous successful applications in the public and private sectors. In the Indian context, it can be said that this traditional method has coupled with recent technological advancements and positioned the OR with strategic significance to be able to improve the nation's ability to compete.

At present, some Hi-technologies such as computer integrated engineering robotics, flexible manufacturing and other novel technologies pose challenges – challenges that call for innovations in technology management. In order to meet challenges, the nation has to invest in basic and applied research in engineering management.

3.2 OR MODEL

Operations Research (OR) is an application of scientific method, techniques and tools to study of systems. The methodology employed consists primarily of defining the system under consideration, constructing a mathematical model of the system and manipulating the variables of the model in order to obtain optimum solutions to the problem of interest. It works on a philosophy of quantification that rests upon ability to identify and quantify pertinent variables and to discover interrelationships that prevail among these variables in a system.

The development of high speed computers has greatly increased confidence of engineering management to employ the OR as an aid to decision making, planning and control. Faster computers enable more efficient solution procedures. With better training, they afford for management of the information. As a consequence, industries invest in the OR to support and improve the quality of logistics planning to go parallel to investment in engineering based improvement to production and distribution.

3.3 Some OR accomplishments

Over the last five decades, the OR creates a wealth of knowledge in modeling and solution of mathematical programming problems. They are:

1. Linear programming
2. Non-linear programming
3. Linear integer programming
4. Markov Chain model
5. Regression model
6. Queueing model
7. Petri Net model
8. Network analysis

3.3.1 Mathematical Programming Models

Enormous progress has been made using large scale mathematical programming models to route raw materials, components and finished goods optimally between production plants and warehouses.

One such technical achievement is a use of approximation methods to analyze models with non-convex cost curve representing the economies of scale that typically arises in trucking operations in coal industries.

3.3.1.1 Linear Programming Model (LP)

Linear programming being one of the oldest disciplines within the OR continues to be one among the most active tools of solution provider. With high speed computers and multi-processors, it is now possible to construct and solve linear programs. It was not that much impossible a few years ago. To cite an example, given the location and demands of various grades of coal, the problem of mining and distributing coal in an optimal fashion has been formulated as a linear programming problem. This is the contribution of the US Geological Survey (Watson et al. 1988). Future developments on this model may extend the planning horizon upto 25 years. Solving large scale linear programs on Super computers begins to recede in the recent times and parallel processors allow opportunities for improved competition. It may lead to an increase in a size of linear programs that can be solved and that too with significant reduction in the time for solution. More important advances derived from new computer technologies are those that are made possible with new algorithms. A glaring example is the interior point method of N.Karmarkar (1986). Linear programming models may be successfully applied in a variety of problems such as production planning, transportation problems, scheduling and many other cases. (Achoff and Sasieni, 1968; Hadley, 1962; Wagner, 1975; Rao, 1977; Hillier and Lieberman, 1987; Taffler, 1979; Verma and Gross, 1978; Norbert, 1965).

3.3.1.2 Non-linear Programming (NLP)

Unlike LP, the NLP involves non-linear objectives and/or constraints. If an objective function is a quadratic one and the constraints are linear, the model is known as quadratic programming model in which the optimal solution is obtained by applying Beale's method or Lemke's method. (Theil and Van De Panne, 1960). Non-linear programming optimizes non-linear functions of several variables subject to (non-linear) constraints on the variables. Problems of this type occur in essentially all scientific, engineering and economic applications. The sequential quadratic programming and sequential linearly constrained methods are used reliably and rapidly to find local optima. One example is the optimal power flow problem of optimizing the distribution of electrical power over a network. The general purpose sequential quadratic programming and sequential linearly constrained methods are applied to solve optimal power flow problems.

3.3.1.3 Linear Integer Programming

If at least one of the decision variables is restricted to be non-negative integer, the LP problem is called a linear integer programming problem. The model becomes an all integer or mixed integer linear programming, depending on whether all or some of the decision variables are restricted to be non-negative integers. An integer linear programming problem can be stated as,

Optimize (Maximize/Minimize)

$$Z = \sum_{j=i}^{n} CjXj$$

Subject to

$$\sum_{j=i}^{n} aijXij \propto bi, \text{ for } i = 1,2,3, \ldots\ldots\ldots\ldots\ldots\ldots, m$$

$X \chi R^n$ and $Xj \, ƒ0$, j=1,2,3……………………n

If further Xj are restricted to be integer then it is called pure inter linear programming problem.

In case Xj = 0 or 1 then it is called zero – one inter program

3.3.1.4 Markov chain model

Markov decision process deals with behaviour of dynamic systems over time. Such behaviour is referred to as a stochastic process. A stochastic process is defined to be an indexed collection of random variables (X_t) where the index't' runs over a given set 'T'. The random variable (X_t) represents the state of the stochastic process at time't'. A set of possible values that the random variable $\{X_t, t \chi T\}$ is called the state space of the stochastic process.

Markov process models have been used extensively in marketing to study brand switching. It is used in the study of equipment maintenance and failure problems. The technique is also useful to the personnel department in determining future manpower requirements of an organization. Basically Markov process helps us to identify

i) a specific state of the system being studied .
ii) a state transition relationship.

An occurrence of an event at a specified point in time (say, period n) puts a system in a given state, say E_n. If, after a passage of one time unit, another event occurs (during the time period n+1), the system moves from state E_n to state E_{n+1}. The probability of moving of system from one state to another, or to remain in the same state in a single time period is called transition probability (p_{ij}) It is the probability that shows the system presently in state E_i and in future in state E_j at some latter step (usually not time). The mathematical format of the process is described by means of state transition matrix as follows:

State Transition Matrix

A state transition matrix is a rectangular array which summarizes the transition probabilities for a given Markov process. In such a matrix, the rows identify the current state of the system being studied and the columns identify the alternative states to which the system can move.

Let, E_i = state i of a stochastic process; (i = 1,2,…,m) and p_{ij} = transition probability of moving from state E_i to state E_j in one step.

Then, a one stage state-transition matrix P can be described as given below:

$$P = \begin{array}{cc} & \begin{array}{cccc} E_1 & E_2 & \dots & E_n \end{array} \\ \begin{array}{c} E_1 \\ E_2 \\ \vdots \\ E_m \end{array} & \left\{ \begin{array}{cccc} P_{11} & P_{12} & \dots & P_{1n} \\ P_{21} & P_{22} & \dots & P_{2n} \\ \vdots & \vdots & & \vdots \\ P_{m1} & P_{m2} & \dots & P_{mn} \end{array} \right\} \end{array}$$

In the transition matrix of the Markov chain, p_{ij} = 0 when no transition occurs from state i to state j ; and p_{ij} = 1 when the system is in state i, it can move only to state j at the next transition.

Each row of the transition matrix represents one-step transition probability distribution over all states (Swarup, et.al 2001). It means

$$P_{i1} + p_{i2} + \dots + p_{im} = 1 \qquad \text{for all } i \text{ and } 0 \leq p_{ij} \leq 1.$$

Transition Diagram

A transition diagram shows the transition probabilities that can occur in any situation. Such a diagram is given in fig.3.3. The arrows from each state indicate possible states to which a process can move from a given state. The following transition matrix corresponds to the diagram (fig.3.3):

$$P = \begin{array}{c|ccc} & E_1 & E_2 & E_3 \\ \hline E_1 & 0 & P_{12} & 0 \\ E_2 & 0 & P_{22} & P_{23} \\ E_3 & P_{31} & 0 & P_{33} \end{array}$$

A zero element in the matrix indicates that the transition is impossible.

3.3.1.5 Regression Model

The purpose of drilling and blasting operations in the opencast mining is rock fragmentation. The major thrust is to maintain safety, minimize an overall cost and maximize a value of resulting products. Requirements of efficiency, timeliness and accuracy have to be satisfied simultaneously. The advent of a large scale mining during the last two decades has brought about significant changes in an approach to blast designs. Computer simulations of blasting process and prediction of blast results in advance are now standard practices in major mining operations. The introduction of computer-aided blast designs is also greatly facilitated by recent advances in explosive technology. Modeling of a blasting process is not a new technology. A blasting model means any relation between blast design and blast results, and therefore, it must be inherent to all practical blasting operations since their advent. Several blast prediction models of varying sophistication are proposed for current use by many researchers.

Blasting in overburden and coal in open pit mines is considered more a science than an art. A lot of research has been done in areas related to surface mine blast design and analysis. But a little of this research seems to have been applied in blasting practices. Very often blasters prefer to use simple empirical formula for their blast design. The varying mining conditions at different locations either at a same mine or at other mines compels a mine operator to design a blasting pattern on the basis of site-specific conditions. With modifications of an established formula on the basis of data obtained from field blasting, optimum design parameters can be determined. Poor and improper blast design lead to problems like poor primary fragmentation, expensive secondary blasting, back-breaks, generation of poisonous and noxious fumes, vibration, fly rock and air blast. Because of poor fragmentation, loading of improperly sized material becomes a problem for excavating equipments, thus it results into frequent breakdowns of machines. (Paul, et.al. 1987). Mishra (2004) views in this light that the primary objective of blasting in a mine is to properly fragment the rock mass so that the fragmented rock mass can be handled properly by excavating equipment.

There are four elements in a blasting system. They interact to produce the desired results. The elements are :

- ☞ explosives
- ☞ rock
- ☞ blast geometry
- ☞ cost

Interaction among the first three elements determines blast results. They have to be weighed against the cost of production. Relevant properties of explosives are its density and detonation properties and those of rock are its strength, density, geological characteristics, in-homogeneity, etc. The blast geometry includes borehole diameter, hole depth, spacing, burden, stemming, coupling and mode & timing of initiation of explosives. The blasting system is thus a very complex one in which several interacting parameters determines the final blast results and the cost of the mining operation (Mohanty, 1988). A number of studies are conducted by experts in the field of mining. They hail both from academics and practical fields. They are concerned with optimum blast design parameters (Tatiya, 2000; Brahma, 1999; Hustralid, 1999; Woof, 2004; Dhar, 1993; Shenoy, 1994).

Regression analysis is an useful tool of operations research. It can be applied in mining for prediction of certain unknown parameters with use of dependent controllable variables. Some statistical studies are made by Sprott David (1988), Singh, et.al. (1993), Jog et.al. (1980), Singh et.al. (2003), Diwedi et.al. (2003), Sastry (2003), Adhikari (1994), Ramlu et.al. (2002), Singh V.K. (2004), etc. in respect of blast vibrations and their effect on structures. Balbas Anton alongwith Garcia J.I.Diaz (1995) studied a spatial relation between laws of vibration from blasting. A number of case studies have also been done to formulate a model suitable for a particular mine having definite rock characteristics. The solution of multivariate linear regression models with their linear equations have been discussed (Kothari,1978; Sastry,1989).

The maximum velocity from the position of rest is termed as peak particle velocity (ppv) and is expressed in mm/sec. This ppv is the measure of ground vibration. The present research discusses it in view of prediction and evaluation through use of regression analysis techniques.

3.3.1.6 Queueing Model

Mechanism of queueing theory involves mathematical study of queues or waiting lines. The flow of customers from finite/infinite population towards service facilities forms a queue (waiting line) on account of capability of a striking a perfect balance between service facilities and the customers. Waiting is needed either of service facilities or at customer's arrival.

The basic process assumed by most queueing models is in the following lines. Vehicles requiring service are arranged over time by an input source. These vehicles enter a queueing system and join a queue. At certain times, a member of a queue is selected for service by some rule known as the service discipline. This process is depicted as follows (Fig: 3.4):

The mathematical formulation of a queueing system is described below:

There are three assumptions in queueing systems:

a) Service is provided on FIFO
b) Customers arrive at random but at a certain average rate.
c) A queuing system is in a steady state condition.

The number of arrivals per unit of time is a random variable with Poisson distribution:

$$F(x) = P(X = x) = \frac{e^{-\lambda} \lambda^{x}}{x!}$$

Where $x = 0, 1, 2 \ldots\ldots\ldots\ldots\ldots\ldots$

$\lambda > 0$

Where E(X) = Expected value

Var (X) = Variance of a poisson random variable

X = number of arrivals per unit time

λ = average number of arrivals per unit of time.

The time between consecutive arrivals has an exponential distribution with the parameter μ

f (t) = $\mu e^{-\mu t}$ $t > O$

$\mu > O$

E (T) = $1/\mu$

Var(T)= $1/\mu^2$

T = time between consecutive arrivals.

The cumulative distribution of an exponentially distributed random variable is useful in simulation. It is given by

$$
\begin{aligned}
f(t) &= Pr(T \leq t) \\
&= \int_o^t \mu e^{-\mu x} dx \\
&= 1 - e^{-\mu t}
\end{aligned}
$$

In particular, if the time between consecutive arrivals at a service facility has an exponential distribution with parameter λ. That is

$$f(t) = \lambda e^{-\lambda t} \qquad t \geq 0$$

$$\text{then } f(x) = \frac{e^{-\lambda} \lambda^x}{x!} \qquad \lambda >) \quad x = 0, 1, 2 \;\ldots..$$

In summary, if the number of arrivals per unit of time has a Poisson distribution with mean λ, then the time between consecutive arrivals has an exponential distribution with mean $1/\lambda$. The queueing system in this situation is said to have a Poisson input, and customers are said to arrive according to Poisson process.

Queuing theory applications

Loading and hauling of excavated materials represent a very significant component of the total operating cost of a surface mine. Material handling system is different at each surface mine. It ranges from simple to complex depending on the size, number and type of load/haul units, number of coal/OB dump positions, incorporation of crusher and belt conveyors, requirement of washing or blending etc. Different views are available in this regards projecting a variety of options. Interactions of all these components in a real life situation can be best evaluated by a computer simulation (Singhal, et.al, 1986). A non-pre-emptive goal programming dispatching model that provides an efficient basis for maximizing production and maintaining coal/ore quality characteristics within a prescribed limit is formulated, developed and validated with data from an operating mine (Temeng, et.al, 1998).

Many authors apply various linear programming formulations to maximize production, minimize the number of trucks for a particular production or minimize the operating cost of dumpers (Lizotte and Bonates, 1988, Lizotte et.al,1987, Li, Z.,1990, Munirathinam, et.al,1994).

Some aspects are studied in view of materials transportation by rear dump trucks in opencast mines studied (Rai,2001). Applications of Genetic Algorithons for efficient vehicle allocation in opencast mines too are discussed (Maulik, et.al. 2001).

The queuing theory application determines an economical number of dumpers matching with shovels. Different authors deal with this application (Kesimal, 1998, Huang, et.al, 1994; Ogbonlowo, et.al, 1988; Srinivas, et.al 1989, etc.).

3.3.1.7 Petri net Model

Petri nets are graphical and mathematical modeling tools applicable to many systems. An analysis of Petri Net can reveal significant information about the structure and dynamic behaviour of a modeled system. This information can then be used to evaluate a modeled system and suggest improvements or changes. Thus, the development of a theory of Petri Nets is based on the application of Petri nets in modeling and design of systems.

The Petri nets proves to be a promising multi-focal tool to describe and study systems that are characterised as being concurrent, asynchronous, distributed, parallel, non-deterministic, and/or stochastic. As a graphical tool, the Petri nets can be used as a visual communication aid similar to flow charts, block diagrams and networks. In addition, tokens are used in these nets to simulate dynamic and concurrent activities of systems. As a mathematical tool, it is possible to set up state equations, algebraic equations and other mathematical models governing the behaviour of systems. The Petri nets can be used by both practitioners and theoreticians. Thus, they provide a powerful medium of communication between the two: practitioners can learn from theoreticians how to make their models more methodical and theoreticians can learn from practitioners how to make their models more realistic.(Murata,1989).

The concept of the Petri nets had its origin in Carl Adam Petri's doctoral dissertation, "Kommunikation mit Automaten" [Communication with automata] submitted in 1962 to the Faculty of Mathematics and Physics at the Technical University of Darmstadt, West Germany. The dissertation was written while C.A Petri worked as a scientist at the University of Bonn. Petri formulated a basis for a theory of communication between asynchronous components of a computer system. He was particularly concerned with description of casual relationships between events. The work of Petri came to the attention of A.W.Holt and others working at the Information System Theory Project of Applied Data Research, Inc. in the United States. Early developments and applications of Petri nets are found in the record of the 1970 Project MAC conference on concurrent systems and parallel computations. From 1970-1975, computation structure group at Massachuttes Institute of Technology (M.I.T.) was most active in conducting Petri net related research and produced many reports and theses on the Petri nets. Since then, Europeans were very active in organizing workshops and publishing conference proceedings on the Petri nets. The European workshop on Application and Theory of Petri nets used to be held every year at different locations in Europe: 1981, Bad Honnef, West Germany; 1982, Varenna, Italy; 1983, Toulouse, France; 1984, Aarhus, Denmark; 1985, Espoo, Finland; 1986, Oxford, Great Britain; 1987, Zavagoza, Spain etc. The latest international conference (i.e. 26th) on "Application and Theory of Petri nets and other models of concurrency" was held at Miami, Florida during 20-25, June, 2005. It was organized by Florida International University, School of Computer Science.

The basic elements of the Petri net models are "places", "transition", "directed arcs" and "tokens". In graphical representation, places are drawn as circles, transitions as bars or boxes, directed arcs as arrows and tokens as black dots (or coloured dots) inside the places(fig-3.1). The Petri net is a particular kind of directed graph or digraph weighted and bipartite together with an initial state called initial marking, Mo. If there is a directed arc connecting a place to a transition, the place is described as the input place to the transition so that P_1 represents the input place to transition. Similarly, if there is a directed arc connecting from a transition to a place, then the place is an output place of that transition ($P_2$to T_0) in fig 3.2. A single place can be connected to a single transition with more than one arc. This weighted connection is represented by having the arc labeled with a natural number called arc weight. By default, unlabelled arcs are weighted one (fig. 3.2). Conforming to these basic rules, multiple places and transitions can be connected to form very complex net to model the static view of the complex system.

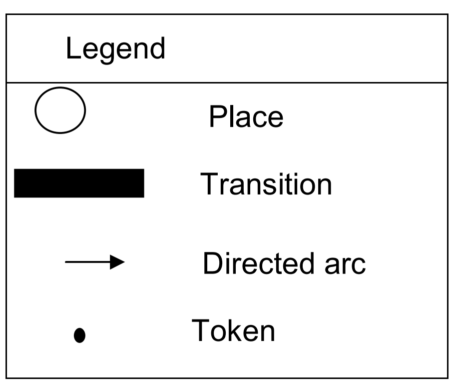


Fig – 3.1: Petri net elements

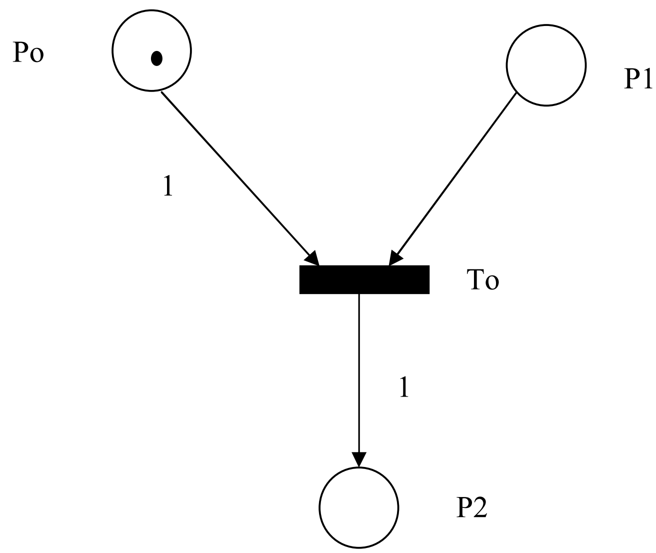


Fig – 3.2: Petri net structure

Petri net consists of two parts:

2) the net structure that represents the static part of the system and

3) a marking that represents the overall state of the structure.

The token distribution among the places of a Petri net is called its marking which represents dynamic behaviour of a system. When one or more tokens reside in a place, the place is said to be marked, otherwise unmarked. The number of tokens in a place represents a local state of a place so that marking of the net represents an overall state of a system. Dynamic behaviour of a system is then modeled by a flow of tokens and firing of transitions. Superficially, transition firing means that tokens in input places are apparently moved to output places. In order to simulate dynamic behaviour of a system, a state or a marking in a Petri net is changed according to the following transition (firing) rule:

1) A transition is said to be enabled if each input place has at least as many tokens as the weight of an arc connecting them.

2) Enabled transition may be fired by removing from each input place number of tokens equal to the weight of an arc connecting them.

3) When the transition is fired, tokens will be added to output places connected to the transition. A number of tokens to be added to each output place is equal to the weight of an arc joining them.

For step 2, it should be noted that the enabled transitions are never forced to fire. In practical modeling, transitions can be related to external conditions that determine whether they may fire or not when enabled.

The transition firing rule described above is illustrated in the Fig. 3.8, with the help of well known chemical reaction. For instance, three tokens in input place H_2 and two tokens in input place N_2 indicate that three units of H_2 and two units of N_2 are available. After transition firing, the same transition is no longer enabled. This change in marking is shown in table below the figure 3.8.

The above described mechanism is usually called the "firing rule" or informally "token game". While a token game governs a dynamic behaviour of the Petri net models, the meaning of the process is determined by the net interpretation. When applied to different domains, the net elements may represent different things. Conditions that are modeled by places and events are modeled by transitions. Inputs of a transition are preconditions of a corresponding event; and outputs are post-conditions. Some typical interpretations of transitions and places are given below (table 3.1):

Table 3.1

Interpretations of transitions and places in Petri net modeling

Input places	**Transition**	**Output places**
Preconditions	Event	Post conditions
Input data	Computation step	Output data
Input signals	Signal processor	Output signals
Resources needed	Task or job	Resources released
Conditions	Clause in logic	Conclusion(s)
Buffers	Processor	Buffers

(Source: Murata,1989)

A Petri net is a 5 – tuple mathematical equation

PN = (P,T,F,W,Mo),

Where

$P = \{p_1, p_2 \ldots\ldots p_m\}$ is a finite set of places

$T = \{t_1, t_2, \ldots\ldots t_n\}$ is a finite set of transitions

$F \subseteq (P \times T) \cup (T \times P)$ is a set of arcs (flow relation)

$W: F \rightarrow \{1,2,3 \ldots\ldots..\}$ is a weight function

$Mo: P \rightarrow \{0, 1, 2, 3 \ldots\ldots..\}$ is the initial marking

$P \cap T = \Phi$ and $P \cup T \neq \Phi$

A Petri net structure N = (P, T, F, W) without any specific initial marking is denoted by N.

A Petri net with a given initial marking is denoted by (N, Mo).

Classification of Petri nets

A large number of different Petri net models are defined by different researchers. Their extensions have more functioning rules than the ordinary Petri nets and, therefore, can be applied to more domains. The recently extended classification of Petri nets is enumerated in the fig.3.5.

Formulation of Time Petri nets:

The Time Petri nets (TPN) are most widely used models for real time system specification and verification. In TPN, even synchronization is represented by a set of pre and post conditions associated with each individual action of the modeled system and timing constraints are expressed in terms of minimum and maximum amount of time elapsing between the enabling and the execution of each action. This allows a compact representation of the state space and an explicit modeling of concurrency and parallelism The SLFT and SLFT are considered in the TPN formulation.

Where SEFT in the Static Earliest Firing Time

SLFT is the Static Latest Firing Time.

A state of a TPN is a pair S = (M,I)

Where,

1) “M” is a markingΦ

2) “I” is a firing interval set which is a vector of possible firing times. The number of entries in this vector is given by a number of transitions enabled by marking “M”.

The fundamental and most widely used method for analyzing PN’s models, like many other models, is the reachability analysis. It permits automatic transition of behavioural specification models into a state of transition graphs made up of a set of states, a set of actions and a succession relation associating states through actions.

Modeling Features of Petri nets

The basic concepts of PN’s that are useful in modeling a system are discussed below: Fig: 3.6 and 3.7:

Conflict

The structure of a Petri net having two or more output transitions from a single place is referred to as a conflict, decision or choice depending on application. Two events are in conflict if the either can occur, and not both, and they are concurrent if both the events can occur in any order without conflicts.

Concurrency

Two transitions are said to be concurrent if they are actually independent i.e. one transition may fire before or after or in parallel with the other.

Confusion

A situation in which conflict and concurrency are mixed up is called confusion.

Synchronization

In a distributed processing system, resources and information are shared among several processors. This sharing must be controlled or synchronized to ensure the correct operation of overall system. PN’s have been used to model a variety of synchronization mechanisms including mutual exclusion, producer-consumer problems, etc. Mutual exclusion is a technique of defining entry and exit code so that at most one process accesses a shared data object and needs protection from interference by other processes and it is called a critical section.

Confusion

It is a situation where concurrency and conflicts co-exist. In the fig.3.7, both t_1 and t_3 are concurrent, while t_1 and t_2 as well as t_2 and t_3 are seen to be in conflict

Dead-lock situation

In a particular situation, activities may start but they are not in a position to start. In fig. - 3.7 both transitions t_1 and t_2 are shown as waiting for the other to fire, but the neither can process.

Merging

Different activities merge to develop a passive state. For example in Fig.- 3.7 arrival of several parts from several sources to a centralised warehouse is represented.

Priorities

The concept of priority is denoted by an inclusion of a special arc called inhibitor Arc. An inhibitor arc that extends a place P_2 to a transition t_2 has a small circle, in stead of an arrowhead at the transition. This notation is borrowed from switching theory, where a small circle means "not". In presence of the inhibitor arc, a firing rule is changed as follows: In fig.-3.7, the arc connecting place P_2 and transition t_2 is called inhibitor arc. The transition t_2 is enabled if P_1 has a token and P_2 does not have a token. This enables to give priority to t_1 over t_2.

Properties of Petri nets

Petri nets as mathematical tools possess a number of properties. These properties, when interpreted in the context of a modeled system, allow the system designer to identify the presence or absence of an application domain specific functional properties of a system under design. Two types of properties can be distinguished: behavioral and structural properties. The behavioral properties are those that depend on an initial state, or marking, of a Petri net. The structural properties, on the other hand, do not depend on the initial marking of a Petri net. These properties depend on a topology, or net structure, of a Petri net. In this section, we provide an overview of some of the most important behavioral properties from a practical point of view. The behavioral properties discussed in this section are reachability, boundedness, liveness, reversibility home state (Fig-3.9).

A. Reachability

An important issue in designing distributed systems is whether a system can reach a specific state, or exhibit a particular functional behavior. In general, the question is whether the system modeled with Petri nets exhibits all desirable properties, as specified in the requirements specification, and it does not have any undesirable ones.

In order to find out whether a modeled system can reach a specific state as a result of a required functional behavior, it is necessary to find such a sequence of firing of transitions which would result in transforming a marking M_o to M_i, where M_i represents the required functional behavior. It should be noted that real systems may reach a given state as a result of exhibiting different permissible patterns of functional behavior. In a Petri net model, this should be reflected in the existence of specific sequences of transition firings, representing the required functional behavior, which would transform a marking M_o to the required marking M_i. The existence in the Petri net model of additional sequences of transition firing which transform M_o to M_i indicates that the Petri net model may not be reflecting exactly the structure and dynamics of the underlying system. This may also indicate the presence of unanticipated facets of the functional behavior of the real system, provided that the Petri net model accurately reflects the underlying system requirements specification. A marking M_i is said to be reachable from a marking M_o if there exists a sequence of transitions firings which transforms a marking M_o to M_i.

B. Boundedness

The Petri net property that helps to identify in a modeled system the existence of overflows is a concept of boundedness. A Petri net is said to be k-bounded if the number of tokens in any place p, where p $\in$ P, is always less or equal to k (k is a nonnegative integer number) for every marking M reachable from the initial marking Mo, M $\in$ R(Mo). A Petri net is safe if it is 1-bounded.

C. Liveness

The concept of liveness is closely related to a complete absence of deadlocks in operating systems. A Petri net (N, Mo) is said to be live (or equivalently Mo is said to be a live marking for N) if, no matter what marking has been reached from Mo, it is possible to ultimately fire any transition of the net by progressing through some further firing sequence. This means that a live Petri net guarantees deadlock-free operation, no matter what firing sequence is chosen.

Liveness is an ideal property for many systems. However, it is impractical and too costly to verify this strong property for some systems such as the operating system of a large computer. Thus, we relax the liveness condition and define different levels of liveness as follows. A transition t in a Petri net (N, Mo) is said to be :

0) dead (L0-live), if t can never be fired in any firing sequence in L (Mo).
1) L1-live (potentially firable), if t can be fired at least once in some firing sequence in L(Mo).
2) L2-life, if, given any positive integer k, t can be fired at least k times in some firing sequence in L(Mo).
3) L3-life, if it appears infinitely, often in some firing sequence in L(Mo).
4) L4-live or live, if t is L1-live for every marking M in R(Mo).

A Petri net (N, Mo) is said to be Lk-life if every transition in the net is Lk-live, k=0, 1,2,3,4. L4-liveness is the strongest and corresponds to the liveness defined earlier. It is easy to see the following implications: L4-liveness = L3-liveness = L2-liveness = L1-liveness, where = means “implies”. We say that a transition is strictly Lk-live if it is Lk-live but not L(k+1)-live, k=1,2,3.

D. Reversibility and Home State

A Petri net (N, Mo) is said to be reversible if , for each marking M in R(Mo), Mo is reachable from M. Thus, in a reversible net one can always get back to an initial marking or state. In many applications, it is not necessary to get back to an initial state as long as one can get back to some (home) state. Therefore, we relax the reversibility condition and define a home state. A marking M' is said to be a home state if, for each marking M in R(Mo) M' is reachable from M.

Analysis method of Petri nets

Methods of analysis for Petri nets may be classified into the following three groups: 1) the coverability (reachability) tree method, (2) the matrix-equation approach, and (3) reduction or decomposition techniques. The first method involves essentially the enumeration of all reachable markings or their coverable markings. It should be able to apply to all classes of nets, but is limited to "small" nets due to the complexity of the state-space explosion. On the other hand, matrix equations and reduction techniques are powerful but in many cases they are applicable only to special subclasses of Petri nets or special situations.

A. The Coverability Tree

Given a Petri net (N, Mo), from the initial marking Mo, we can obtain as many "new" markings as a number of enabled transitions. From each new marking, we can again reach more markings. This process results in a tree representation of markings. Nodes represent markings generated from Mo (the root) and its successors, and each arc represents a transition firing, which transforms one marking to another.

The above tree representation, however, will grow infinitely large if a net is unbounded. To keep a tree finite, we introduce a special symbol ω, which can be though of as "infinity". It has the properties that for each integer n, $\omega > n$, $\omega \pm n = \omega$ and $\omega \geq \omega$.

For a bounded Petri net, the coverability tree is called the rechability tree since it contains all possible reachable markings.

B. Incidence Matrix and State Equation

The dynamic behavior of many systems studied in engineering can be described with differential equations or algebraic equations. It would be appropriate if we could describe and analyze completely the dynamic behavior of Petri nets by some equations. In this spirit, we present matrix equations that govern the dynamic behavior of concurrent systems modeled by Petri nets. However, the solvability of these equations is somewhat limited, partly because of the non-deterministic nature inherent in Petri net models and because of the constraint that solutions must be found as non-negative integers. Whenever matrix equations are discussed, it is assumed that a Petri net is pure or is made pure by adding a dummy pair of a transition and a place.

Incidence Matrix:

For a Petri net N with n transitions and m places, the incidence matrix A = (aij) is an n x m matrix of integers and its typical entry is given by

$$aij = a^{+}_{ij} - a^{-}_{ij}$$

where $a^{+}_{ij} = w(i,j)$ is a weight of an arc from transition i to its output place j and $a^{-}_{ij} = w(j,i)$ is a weight of an arc to transition i from its input place j. We use A as the incidence matrix instead of its transpose A^T because A reduces to the well-known incidence matrix of a directed graph for marked graphs, a subclass of Petri nets.

Petri net application

The practical application of Petri nets to the design and analysis of systems which can be accomplished in several ways. One approach considers Petri nets as an auxiliary analysis tool. For this approach, conventional design techniques are used to specify a system. This system is then modeled as a Petri net and this Petri net model is analyzed. Any problem encountered in the analysis point is studied and remodeling is done. A design needs to be modified to correct the flows. A modified design can then be modeled and analyzed again. This cycle is repeated until the analysis reveals no unacceptable problems. This approach can also be used to analyze an existing operative system.

The conventional approach described above for using Petri nets in a design of a system requires constant conversion between a designed system and a Petri net model. In an alternate approach, the entire design and specification process is carried out in terms of Petri nets. Analysis techniques are applied only when it is necessary to create a Petri net design error-free. Then the problem is to transform the Petri net representation into an actual working system (Peterson, 1981).

These two approaches of using Petri nets in a design process provide different types of problems for Petri net researchers. In the first case, modeling techniques must be developed to transform systems into Petri net representation; in the second case, implementation technique must be developed to transform Petri net representations into systems. In both the cases, we need analysis techniques to determine the properties of Petri net model.

The Petri net has been proposed for a wide variety of applications. It is due to generality and permissiveness inherent in the system. They can be applied informally to any area or system that can be described graphically like a flow chart and that needs some means of representing parallel or concurrent activities. However, a careful attention has to be paid to a trade-off between modeling generality and analysis capability. In applying the Petri nets, it is often necessary to add special modifications or restrictions suited to a particular application. It has two successful application areas: performance evaluation and communication protocol. Promising areas of applications include modeling and analysis of distributed software system, distributed data base systems, concurrent and parallel programs, flexible manufacturing/industrial control systems, discrete event systems, multiprocessor memory systems data flow computing systems, fault tolerant systems, programmable logic and VLSI arrays, office information systems, formal languages and logic programs. Other interesting areas of applications are local area networks, legal systems, human factors, neural networks, digital filters, decision models, medical and health related problem areas, chemical and civil engineering problems, etc. (Murata, 1989).

The use of computer-aided tools is a necessity for practical applications of Petri nets. Most Petri net research groups have their own software packages and tools to assist the drawing, analysis, and/or simulation of various applications. Object oriented Petri nets can be considered as a special kind of high level Petri nets which allow for the representation and manipulation of an object class. In this class of nets, tokens are considered as instances or tuples of instances of object class which are defined as lists of attributes. Attempts are made to combine Petri nets with other techniques, such as neural networks, fuzzy logic, etc. (Zurawski et.al, 1994). Fuzzy Petri nets have been used for knowledge representation and reasoning.

The application of Petri nets in mining industry is very rare. Prof. V.Konyukh, a renowned scientist of Russian Academy of Mining Sciences, carried out research and published number of papers on application of Petri nets in mine automation and robotics. Functional modeling and Petri nets are used in order to simulate mining robotic system (Konyukh, 2002). Robotics based mining can be simulated by using Petri nets as a movement of so called tokens through the transitions $t \in T$ with some delays in places $p \in P$ for a time of the technological operations. Konyukh developed a robotized long wall mining system using Petri nets (Konyukh, 2002).

Gosine (1999) observes that application of Petri nets deals with a control of mobile robots in an unstructured environment like mining using discrete event modeling system. Sawhney discusses and analyzes hybrid scheduling technique utilizing concepts of Petri nets to model the work tasks in construction projects incorporating the risk and uncertainty in time and cost estimates. He also discusses how it is simulated dynamically (Sawhney, 1997).

3.3.1.8 Network Analysis

Network analysis is a technique that is related to sequencing problems which are concerned with minimizing some measures or performance of a system such as the total completion time of the project, the overall cost and so on. This technique is useful for "describing elements in a complex situation for the purpose of designing, planning, coordinating, controlling and making decision". Network analysis is specially suited to projects which are not routine or repetitive and which will be conducted only once or for a few times". (Kothari, 1978). A project in the context of our study is a "one time" operation which has a well defined "end point" in the time horizon. In an opencast mine, the end point can be a date of achieving completion capacity of a project. At a corporate level, the end point can be a date when one company takes over the management of a sick unit. In the context of marketing, it can be a date on which a new product is sold in a market on commercial basis (Mustafi, 1988). Network analysis has for long played a significant role in electrical engineering. However, there has been a growing awareness that certain concepts and tools of network theory are useful also in many other fields as well (Hillier & Liberman, 1987).

Network scheduling is a technique used for planning and scheduling large projects in fields of construction, maintenance, fabrication, purchasing, computer system installation, research and development designs, etc. This technique is a method of minimizing trouble spots, such as, production bottlenecks, delays and interruptions, by determining critical factors and coordinating various parts of overall job (Swarup. et.al. 2001). By analyzing a network, which is a graphic representation depicting 'activities' and 'events', planning, scheduling and control of a project becomes much easier. One such OR tool. It is used on large scale projects to aid management in expediting and controlling utilization of personnel, materials, facilities and time input of a project is Program Evaluation and Review Techniques (PERT). The PERT was developed in 1958-59 as a research and development tool for the U.S.Navy Polaris Missile program. The PERT analysis is applied in number of areas of the computer industry, motion picture industry and military projects

(Gillett,1979). The Critical Path Method (CPM) is another planning and scheduling tool which was developed for use in construction projects. The past experience is used in it to obtain time and cost estimates of various phases of a project.

The basic components of a network are:

Activity – An activity is a task, or an item of work to be done. It consumes time, effort, money and other resources. It lies, between two events; one that is called ‘proceeding’ and another that is ‘succeeding’. An activity is represented with an arrow whose head indicates a sequence in which events occur.

Event – An event represents the start (beginning) or completion (end) of an activity. As such it consumes no time. It has no time duration and does not consume any resources. It is also known as a mode. An event is not complete until all the activities flowing into it are completed. An event is normally represented on a network with a circle, a rectangle, a hexagon or some other geometric shape.

Several observations are available as regards to application of Network analysis. A few of them are summarized as under. Jardine observes that scheduling uses interactive graphics that are used with in Computer Applications in mineral industry (Jardine G.H et.al, 1988). It discusses major advances in mine scheduling. An open pit planning and scheduling system were put to deliberations at the first Canadian conference on Computer Applications in Mineral Industry. This approach of planning and scheduling attempts to approximate complex optimization, while producing more practical results with much less effort. Mining engineers understand it and menu drivers of multi-window screen require little program documentation (Gerson, 1988). An overview of computer techniques is utilized in short range planning. At a number of mines operating presently, practical and innovative computer solutions need to be developed in close association with operating personnel at a mine site (Edmiston, 1988).

In a fast moving mine production scenario, target based planning and scheduling become key issues in order to arrive at a position that renders desired output. As scheduling of a project at an opencast coal mine requires connections like hierarchical decomposition of project activities, risk and uncertainty at time of activity, cost estimates and modeling of dynamically allocated resources. Traditional network techniques such as PERT and CPM are currently used in the mining industry. They provide limited modeling versatility. Performance evaluation of repetitive automated manufacturing systems is studied with a concept of converting PERT network to Petri net models (Campos Javier, et al, 1990). The concept of the Petri net modeling from the PERT network is discussed in general (Peterson, 1981) in the "Petri net theory and modeling of systems". Various applications of the Petri net for different systems are elaborated (Murata, 1989, Zurawski, et al, 1994). The Petri net based simulation of construction schedules are also studied for evaluation of Petri net model and PERT network (Sawhney, 1997). Ang. et al. (1975) developed a technique called Probabilistic Network Evaluation Technique (PNET). This technique applies probability theory to reduce number of possible critical paths and evaluates expected project duration based on representative paths in the network. .Woolery and Crandall (1983) and Ahuja and Nandakumar (1985) provide a stochastic network model for scheduling. All these observations prepare a background with which an attempt is made here to apply the Petri modeling and to review it against a traditional method of the PERT network that is currently in use for planning and scheduling activities at opencast coal mine projects (Fig-3.10).

3.4 OR Applications

Operations Research techniques in the Indian coal mining industry is relatively new and just at an infant stage. Even in advanced countries, use of computers and operations research techniques used to be slow initially. Number of theoreticians and practitioners involved at the mining industry try time to time to apply these techniques in various mining operations like production planning, inventory management, scheduling, transportationand decision support systems.

A survey was conducted on the use of computers at mining operations by Grayson et al. (1989) at the Department of Mining Engineering, West Virginia University. It reveals that only 63.5% responded in affirmation and out of it, some 13.5% of engineers use computers for mining applications. It is observed that a large number of mines are fully computerized whereas small mines have only 44% of computerization. Despite it, sophisticated engineering applications are yet to see a real value.

Mine planning is based on large heuristic and empirical knowledge and, therefore, it belongs to one of the most complex engineering problems. A model-based approach enables us to represent knowledge from an examined area of the mine planning and makes it transparent, recordable and reusable (Martens, et.al. 1997). A process of surface coal mine planning involves selection of coal property, making decisions regarding appropriate mining method, selection of types and number of equipments; and producing mine designs to make an optimum use of equipments and manpower (Chhipa et al, 1995). A development of an effective open pit mine scheduling procedure generates and evaluates an extraction sequence of mining blocks over short periods of time. It becomes highly desirable in today's competitive and high risk mining world with low commodity prices (Elevli, 1995). With this views, optimizing mine life and design capacity are studied in detail using OR techniques. The problems associated with production scheduling in open pit mines are addressed once an ultimate pit limits is defined (Fytas, et.al, 1993).

The principal structure of economic and mathematical models is determined to choose efficient alternative plans for long term development of existing mines. It is done with objective functions with a system of constraints and nomenclatures of input and output variables.

In order to achieve the best all round results, economy-mathematical models of planning a total system, right from the highest to the lowest hierarchical levels of decision making, are reviewed through different OR modeling techniques. The optimum obtained by the computer solution of the models would provide a basis for objective decision making in the fields of selection of optimum plan for developing the industry (Sinha and Sharma, 1976).

Industrial and economic-financial activities of mining and processing enterprises in many respects depend on quality of planning of mining and control of mining operations (Dzharlkaganov, et.al, 2004).

An OR model applies to a system a concept augmented by mathematical modeling. It helps to optimize production planning and scheduling for a group of underground coal mines. A case study conducted by Javed and Sinha reveals it (Jawed & Sinha, 1985).

A linear programming model is developed with constraints based on statistical data available from industry. The results of the investigation are tested with a model so developed and which conforms to the pattern existing in the industry (Ray and Mazumdar, 1980).

In the present circumstances these techniques have greater importance for reasons like competitive demands in coal markets, a low operating costs, high productivity parameters and multicriterion decision making with conflicting objectives. Numerous unseen variables are observed during mining operations, either underground or opencast. Some times, it becomes cumbersome and irritating to find which goal has to be given priority in our mines. It thereby calls for a use of optimization technique. These techniques, however, attract attention of academicians and practitioners in India too, but it is too slow. Over the years, very few publications appear to speak on the application of optimization technique in a mining area, particularly in the context of the Indian environment.

Scholars like Mutmansky (1973), Seegmiller (1973), Douglas (1981) and Gardener (1984) document effectiveness of the operations research methods as an industrial tool for decision analysis. They indicate that coal industries all over the world have been slow, until recently, in accepting the operations research techniques and computer applications as compared to other production and processing industries. It is noteworthy, however, that mining industry, particularly the coal mining industry in the United Kingdom was among of the first to establish its own operations research group as far back as 1947 and since then there has been a steady growth (Rivette 1956; Cook 1956).

Several views are given in relation to application of the OR model to Indian Mining System. Here are some few. Sinha and Sharma (1976) emphasize the need of a total system planning for exploration of coal in India. They have developed mathematical models for various levels of operations. Jawed and Sinha (1985, 1989a) indicate a scope of application of the OR techniques in various areas of coal mining. They emphasize a fact that even the OR application of sub-systems may yield a huge benefit under the situation when no consideration is granted initially to optimal planning.

Sinha and Mukherjee (1983) use linear programming to develop a long term plan. Linear programming is reportedly used successfully at one of the Indian opencast mines. Mukherjee and Prasad (1992) further propose a methodology for optimal planning of coal transportation system from collieries to washeries.

A group of researchers comprising of Ray and Mazumdar (1980), Ray (1984),Bordia (1978) and Sinha & Ray (1972) highlights the application of mining investment appraisal techniques and the principles of mine system design for a techno-economically feasible planning in coal mines. Sinha particularly deals with a use of optimal planning process for exploitation of mineral resources and several constraints.

Another group of researchers to include Murthy (1989), Sinha and Ray (1978), Dhar and Sharma (1978) and Jawed and Sinha (1989a) indicate possible applications of optimization techniques for specific nature of mine systems planning.

Shulman (1989) goes little further to apply the operations research methods for optimization of non-linear function for exploitation efforts at iron ore mines in Egypt. He uses Kuhn-Tucker conditions and forms an opinion that an equivalent linear programming problem can be formulated and solved for this purpose. Grayson (1989) developes a menu driven linear programming model based on user friendly computer programmes to assist mine parameters in determining optimal long wall panel dimensions for a given set of operating conditions. Both Shulman and Grayson are of the view that sensitivity analysis must be performed by varying parameters over an expected practical range during the optimization process.

Researchers like Hartley and Spence (1985), on the other hand, review another dimension of the D.R.model. They review that the implementation of Management Information System (MIS) provides timely and accurate information. It is used on regular ground in the United Kingdom and United States to assist a management in the decision making. Mathematical models are used properly for an analysis of data that are relevant to decision making process. It can be useful in formulating better decisions. A possible explanation for general dissatisfaction may include a lack of proper attention to designing tools for practical application, concentration more on developing a new theory by academia and least on application and improvement of existing methods. General resistance to change is expressed by management personnel currently, particularly at middle and top levels. They are not qualitatively prepared to use the tools. This opinion is supported by Grayson et. al., (1989) and it is based on the results of the survey that was carried out and the interviews conducted with executives operating at production industries. Another primary reason for lack of application may be related to a dynamic structure of production inventory system itself. It often allows insufficient time for decision analysis when it seems to be moving out of control. A routine process of demand and production forecasting, planning, scheduling and control of operations as followed at the coal industry in the past are to a large extent, traditionally undertaken even today, of course, on a piece-meal basis. It is mostly without much application of the OR techniques and it consequently results in sub-optimal performances.

It may eventually be found that a literature survey on the application of operations research techniques in coal mining industry like the one as above reveals that so far no such significant work is carried out as regards to their application in production optimization in the field of opencast mines. Considering importance of mine production planning at existing mines or at new mines, there are scopes for development of suitable strategic planning models with application of OR tools for the benefit of the mine management. In opencast mines, an optimal performance of such man machine system is essential for better utilization of high investment shovel and dumper, manpower, drilling and explosive resources. In this light, the next chapter deals with modeling of drilling operation to focus on the significance of OR models in enhancing efficiency and productivity at coal mining.

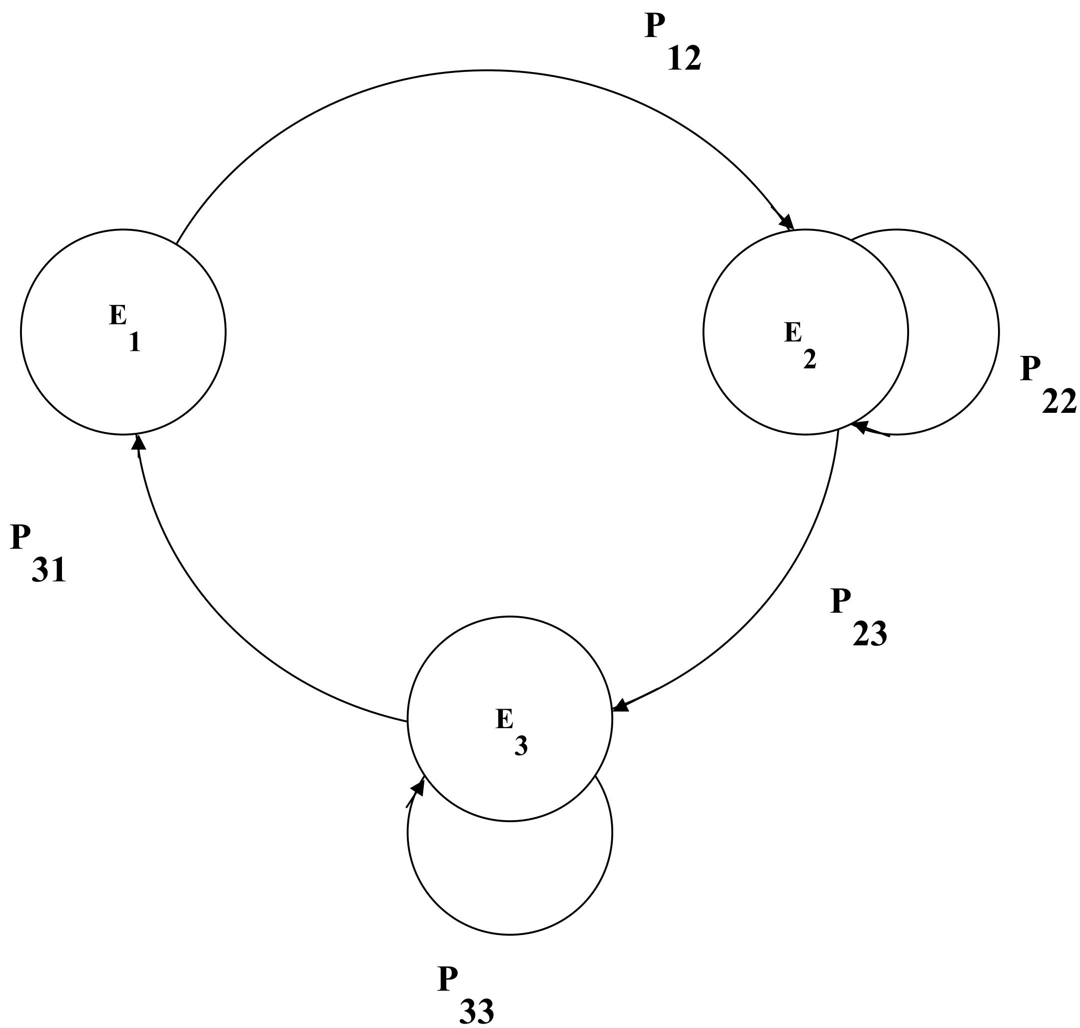


Fig.-3.3: Transition Diagram

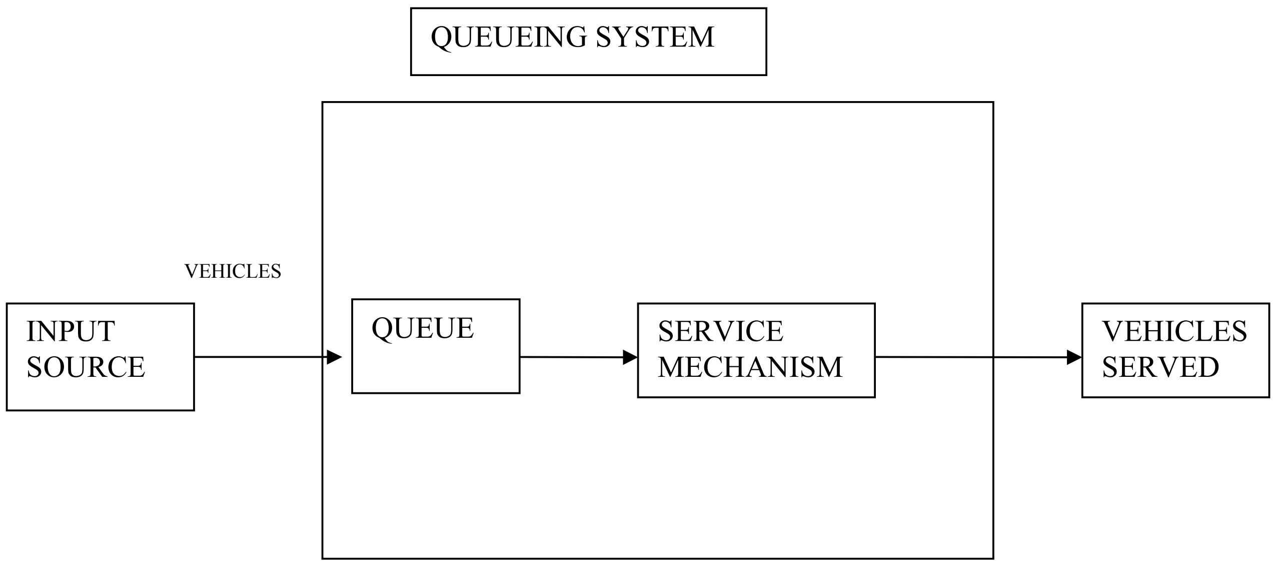


Fig 3.4 Basic Queueing Process

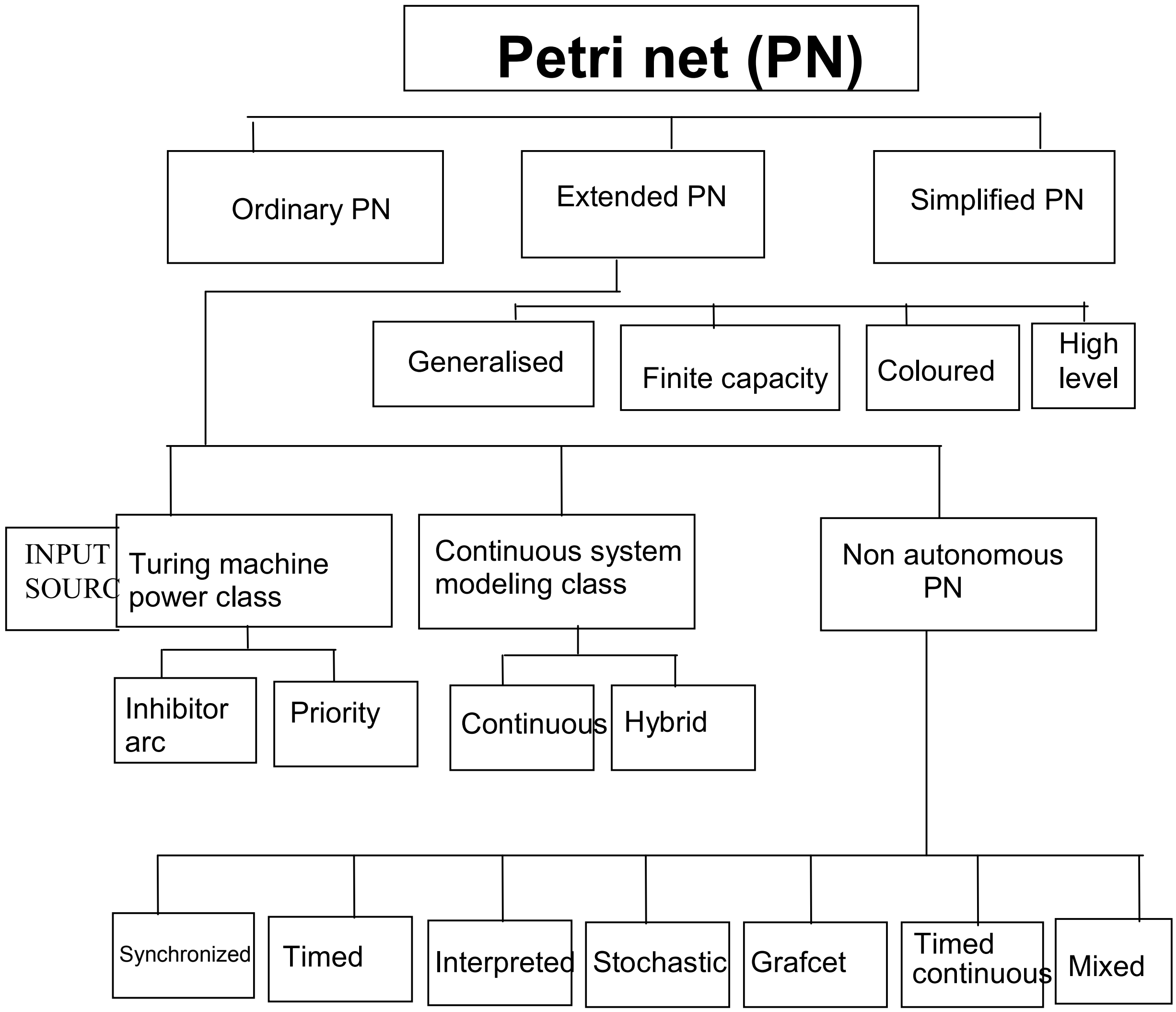


Fig – 3.5 Taxonomy of Petri nets

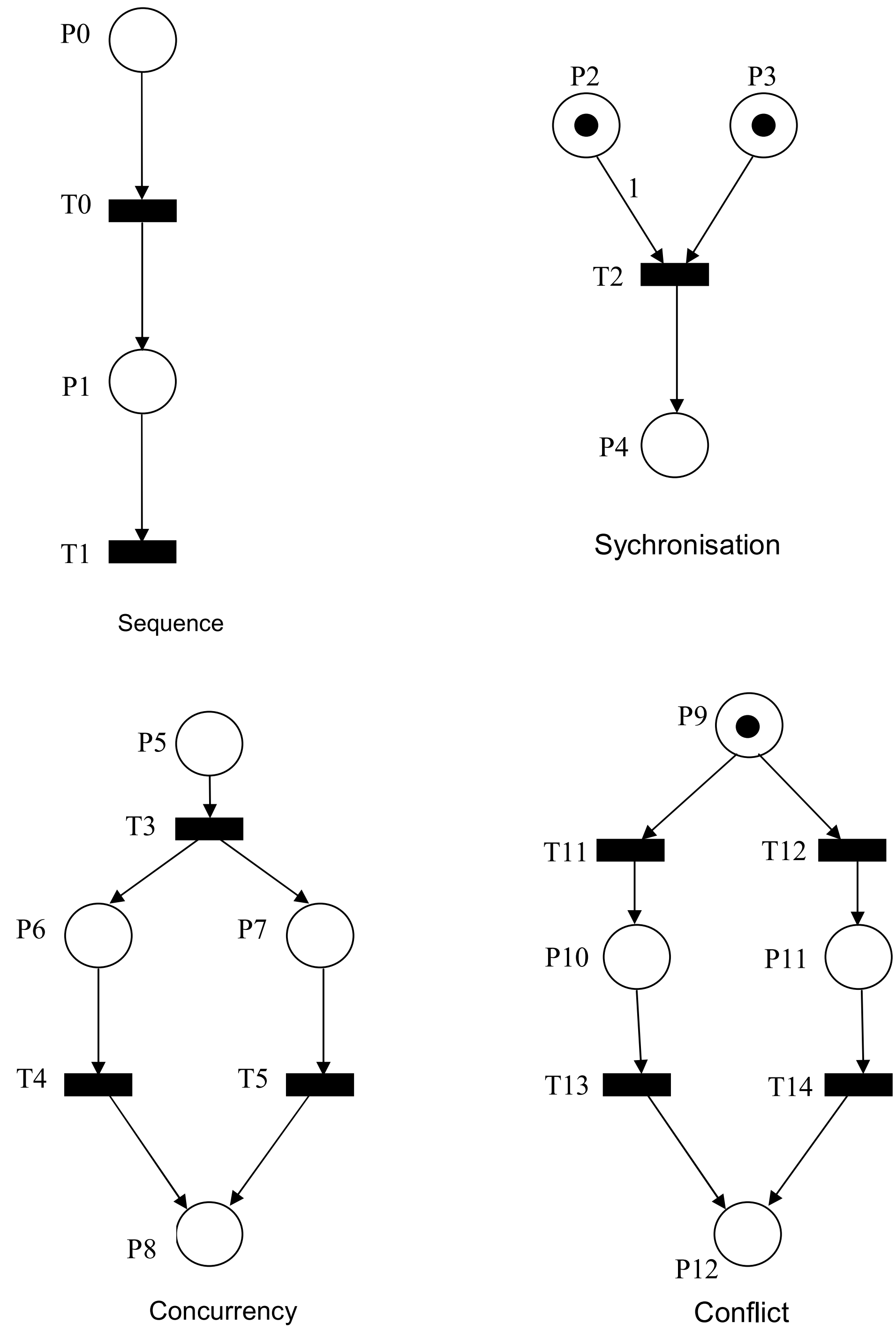
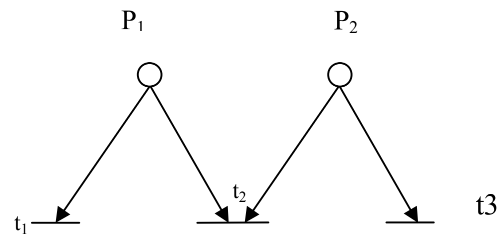
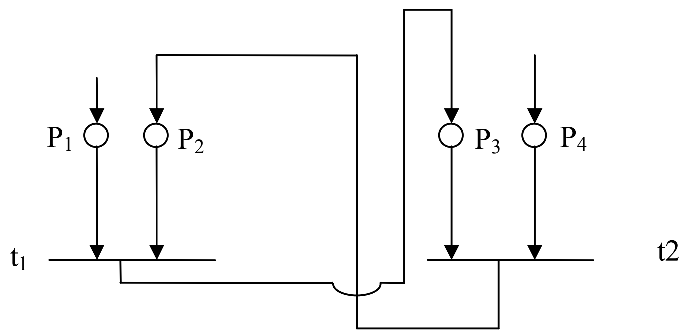


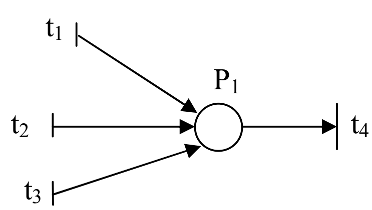
Fig. – 3.6: Modeling features of Petri net (I)



Petri Net illustrating confusion



Dead-lock situation



Merging of transitions

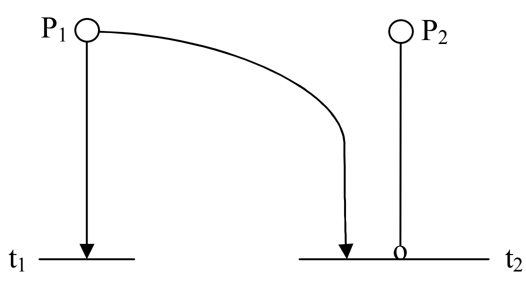
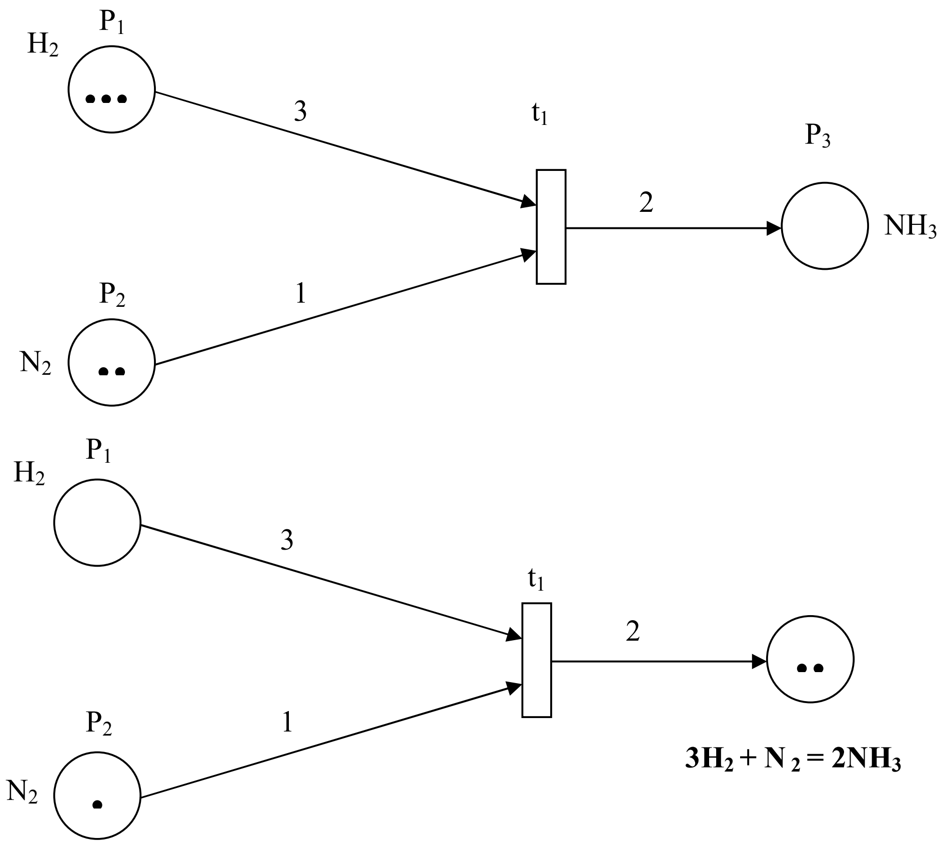


Illustration of priority

Fig. – 3.7: Modeling features of Petri net (II)

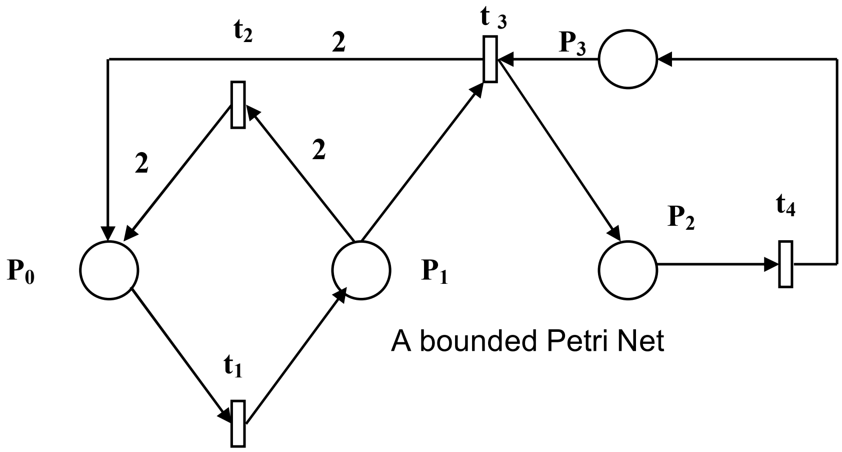


An illustration of transition firing rule.

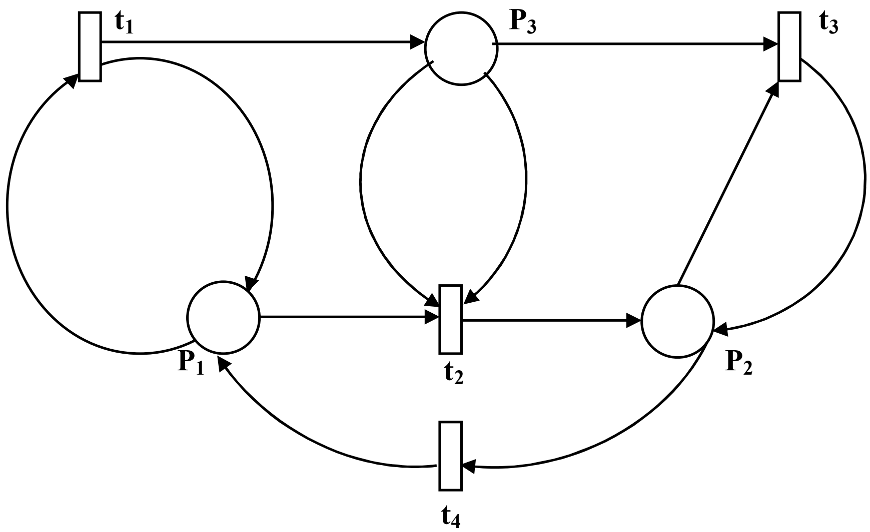
Change in marking

W(t/P) / W(p,t)	M o	M^1
0 / 3	3	0
0 / 1	2	1
2 / 0	0	2

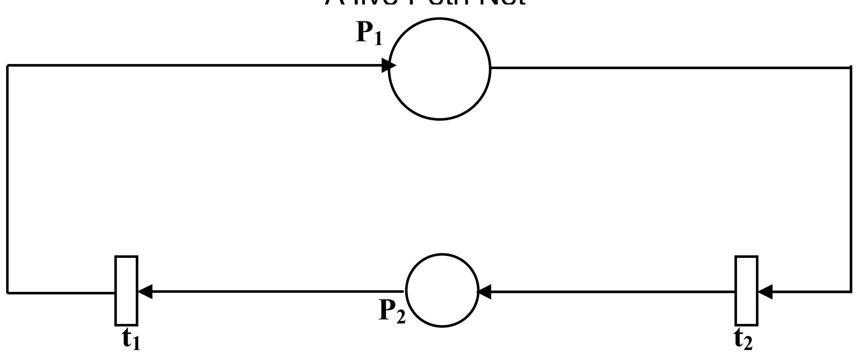
Fig – 3.8: Interpretation of Firing Rule.



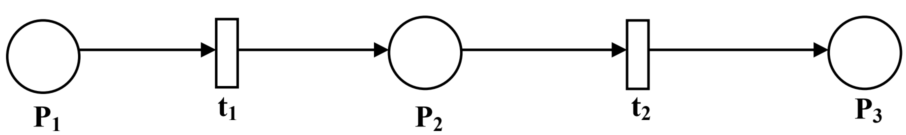
A bounded Petri Net



A live Petri Net

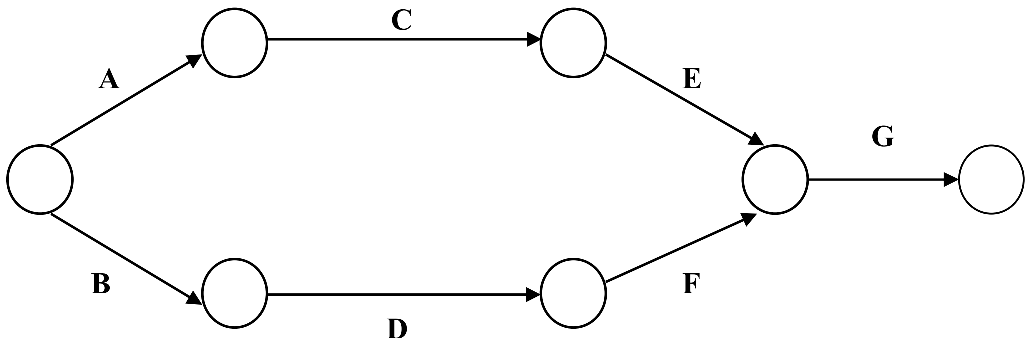


A reversible Petri Net

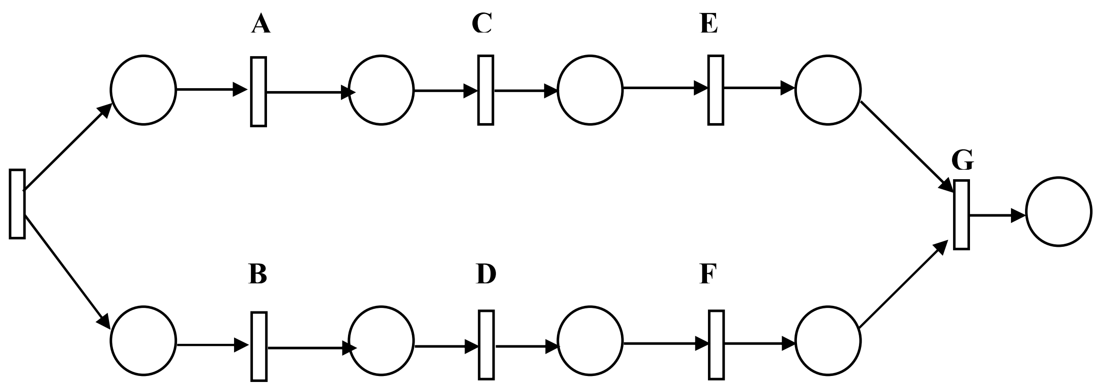


Petri net showing sequential execution

Fig – 3.9: Properties of Petri nets



PERT Diagram.



Petri net diagram of a project.

Fig 3.10: Simple PERT conversion into Petri net.

MODELING OF DRILLING OPERATION

4.1 Introduction

A demand for minerals has increased in India with an increase in number and dimensions of opencast mining industry. In order to meet an ever increasing demand, a trend to set up large opencast projects has begun. Accordingly, bigger projects of opencast mines are planned currently. With the current level of production is expected to be doubled in the next fifteen years and it may keep pace with a burgeoning demand that arises from a rapid growth attained in industries like power, steel, non-ferrous, cement, chemicals and other sectors. This, in turn, involves an increased quantum of excavation volumes. The basic equipments so far available are drill machines and other equipments to match the size of the drill machines. Drilling operation is a major activity in the field. It is time consuming and involves higher costs. As all mining operations are dependent totally on the first and basic operation of drilling, various techniques to improve this operation may ensure to generate more yield per meter of drilling. In this context, different OR techniques have been tried on practical grounds with considerable success. Hence, it is recommended in the interest of enhancing the efficiency level and productivity.

The principles of drilling are basically concerned with energetic task of rock penetration with a use of mechanical energy. It may also involve functional responses and interrelationships between drill systems and rocks. A good understanding of the basic principles of the system and its components is required, it involves the following parameters:

1) Strength characteristics of rocks.
2) Mechanics of penetration
3) Major factors of penetration rate
4) Engineering properties related to drillability.
5) Drillability determination.

The purpose of production drilling is to provide a cavity for placement of explosives. So far, no optional method is devised to serve as concrete alternative to blasting that may be effective to fragment a very resistant and hard rock insitu. However, some forms of excavation like ripping, continuous miner and bucket wheel are found to be suitable even without drilling and blasting for softer rock, such as weathered shale or mudstone, lime stone, soft coal etc. For a vast majority of surface mines, however, drilling and blasting are pre-requisite to excavation and essential to the production cycle. A drilling (or any penetration) system must perform two separate operations in order to achieve advance into rock:

1) fracture of material in the solid and
2) ejection of the debris formed.

The first phase is, of course, actual penetration, while the second involves removal of cuttings. Both affect drilling and drill performance, but they are distinct and separate phases in the process.

Causing rock to break during drilling is a matter of applying sufficient stress with a tool to exceed the strength of a rock. This resistance to penetration to rock is termed as drilling strength. It is not equivalent to any of the well-known strength parameters. Further, a stress field created must be so directed as to produce penetration in the form of a hole of a desired shape and size. These stresses are dynamic (time dependent) in nature. But, in a drilling process, they are demonstrated to be applied so slowly as to closely simulate static loads. A rate-of-loading effect in rock drilling is demonstrated as negligible.

In absence of a reliable means of complex mechanization that may embrace all ranges of geological and technological conditions, extraction of coal from large opencast mines with a number of seams and overburden partings is associated with use of more efficient equipments. Different coal winning operations are carefully studied and analysed with an OR modeling tool called Markov Chain analysis. A drilling operation has been critically studied with the data obtained from the operating opencast mines and a mathematical model too is developed to assess the standard vector of probability as a criterion of reliability of complicated sub-systems.

The Petri net model of drilling operation has also been developed to analyse various activities of drilling so that automation in this critical operation can be made possible.

4.2 Application of Markov chain in mining operations

Markov process is a way of analyzing a current movement of some variables in an effort to forecast its future movement. Frequently when the behaviour of a system is described by saying it is in a certain state at a specified time, the probability law of its future state of existence depends only upon the state it is in presently, and not on how the system arrives in that state. When this situation occurs, the system behaviour can be described by a process called 'Markov processes'. All the processes or operations of working face are characterized by an influence of random factors that act on and may decrease the reliability of technological operations. The method evaluates reliability during operation or failure state of elements. As an analysis of such process is random in nature, a theory of random process can be used. This operation at a working face approximately can be considered as a Markovian process. Hence, Markovian process is utilized to evaluate all probability states of operation as well as failure. The subsystem of a mine "working face" is considered to be complicated combination of technological elements on which operations are realized. The structure of a sub-system is determined by inter-relationship of the technological elements and by composition and successive functioning of the technological process. A well known drilling and blasting method of coal extraction and OB removal, it seems, has not been studied properly in developing countries with an application of the OR techniques.

The working face consists of the following processes and operations.

i) Drilling of blast holes
ii) Charging of blast holes
iii) Blasting
iv) Loading

As mentioned above, for an analysis of such processes of random nature, the theory of random process can be used. The quantitative evaluation of the functioning of the sub-systems “working face” can then be done in the following manner:

- ☸ A diagram of mutual action and mutual correlation of main technological elements is prepared in the form of a flow chart (fig.-4.1).

- ☸ A mathematical formulation is work out for functioning of process and operation of working face taking into consideration of their random character.

The following probability notations are used:

$p^t(000)$	-	All elements of drilling process are functioning well
$p^t(100)$	-	Drilling process does not function because of seam faults F
$p^t(010)$	-	Drilling process stopped due to the failure of electric drill E
$p^t(110)$	-	Drilling process does not function due to F and E
$p^t(0,ch)$	-	All elements in charging of holes are satisfactory
$p^t(1,ch)$	-	Charging is stopped due to lack of explosive at the face LE
$p^t(0,BL)$	-	All elements in blasting are functioning well
$p^t(1,BL)$	-	Blasting operation does not function due to break-down of exploder B
$p^t(0,L)$	-	All elements in loading are working properly
$p^t(1,L)$	-	Loading process is stopped due to break-down of loading machine.

For determination of the probability of the processes and operations, the system of differential equations can be written in the following form for respective processes:

Equations for drilling process:

Group – A

$$\frac{d\ p^t(000)}{dt} = -(\lambda F + \lambda E)\ p^t(000) + \mu F\ p^t(100) + \mu E\ p^t(010) \quad \ldots(1)$$

$$\frac{d\ p^t(100)}{dt} = -(\mu F + \lambda E)\ p^t(100) + \lambda F\ p^t(000) + \mu E\ p^t(110) \quad \ldots(2)$$

$$\frac{d\ p^t(010)}{dt} = -(\mu E + \mu F)\ p^t\ (010) + \lambda E\ p^t\ (000) + \mu F\ p^t\ (110) \quad \ldots(3)$$

$$\frac{d\ p^t(110)}{dt} = -(\mu E + \mu F)\ p^t\ (110) + \lambda E\ p^t\ (100) + \lambda F\ p^t\ (010) \quad \ldots(4)$$

$$p^t(000) + p^t\ (100) + p^t\ (010) + p^t\ (110) = 1 \quad \ldots(5)$$

Similarly for the charging process:

Group – B:

$$\frac{d\ p^t(0,ch)}{dt} = -\lambda LE\ p^t\ (0,ch) + \mu LE\ p^t\ (1,ch) \quad \ldots(6)$$

$$\frac{d\ p^t(1,ch)}{dt} = -\mu LE\ p^t\ (1,ch) + \lambda LE\ p^t\ (0,ch) \quad \ldots(7)$$

$$p^t(0,ch) + p^t\ (1,ch) = 1 \quad \ldots(8)$$

Group – C:

For the process of blasting:

$$\frac{d\ p^t(0,BL)}{dt} = -\lambda BL\ p^t\ (0,BL) + \mu BL\ p^t\ (1,BL) \quad \ldots(9)$$

$$\frac{d\ p^t(1,BL)}{dt} = -\mu BL\ p^t\ (1,BL) + \lambda BL\ p^t\ (0,BL) \quad \ldots(10)$$

$$p^t\ (0,BL) + p^t\ (1,BL) = 1 \quad \ldots(11)$$

Group – D:

For the process of loading:

$$\frac{d\ p^t(0,L)}{dt} = -\lambda L\ p^t\ (0,L) + \mu L\ p^t\ (1,L) \quad \ldots(12)$$

$$\frac{d\ p^t(1,L)}{dt} = -\mu L\ p^t\ (1,L) + \lambda L\ p^t\ (0,L) \quad \ldots(13)$$

$$p^t\ (0,L) + p^t\ (1,L) = 1 \quad \ldots(14)$$

This system of the differential equations with stationary conditions is considered. The sub-system “working face” takes a form of homogenous algebraic equation equating its derivative to zero.

Where λF, λE, λBL, λLE and λL are intensities of failures and μF, μE, μBL, μLE and μL, are the intensities of restoration correspondingly to F, E, BL, LE and L respectively. It follows from the above that the main parameters for calculation are λi and μi. The values of these parameters for the random factors mostly influence reliability of functioning of the "working face" at a mine. The solution of the system of homogenous algebraic equations can be found out from the value taken for λi and μi considering the specific data given in table-4.1.

Table –4.1
Working data from fields

Sl. No.	**The random factors in the main processes of the sub-system working face**	**Values of parameters**	
		λ_i **(1/hr)**	μ_i**(1/hr)**
1	Drilling of blast holes : due to the seam faults due to the failure of electric drill	 0.001 0.0025	 0.5 0.125
2	Charging of blast holes : Due to lack of explosives at face	0.0001	0.067
3	Blasting : due to the break-down of exploder	0.0001	0.0833
4	Loading : due to break-down of loading machine	0.005	0.1

A solution of the algebraic equations has to be found out by using the parameters as mentioned in the table-4.1 above.

Analysis of Group A:

All the derivatives as shown in equation (1)-(5) have been converted into algebraic equations by equating to zero and the same equations can be represented as below:

- $-(\lambda F + \lambda E)\, p^t(000) + \mu F\, p^t(100) + \mu E\, p^t(010) = 0$ …(1)
- $-(\mu F + \lambda E)\, p^t(100) + \lambda F\, p^t(000) + \mu E\, p^t(110) = 0$ …(2)
- $-(\mu E + \mu F)\, p^t(010) + \lambda E\, p^t(000) + \mu F\, p^t(110) = 0$ …(3)
- $-(\mu E + \mu F)\, p^t(110) + \lambda E\, p^t(100) + \lambda F\, p^t(010) = 0$ …(4)
- $p^t(000) + p^t(100) + p^t(010) + p^t(110) = 1$ …(5)

Multiplying (λF + λE) with equation (5) and adding the resultant equation with equation (1), we get

$$(\mu F + \lambda F + \lambda E)\, p^t(100) + (\mu E + \lambda F + \lambda E)\, p^t(010) + (\lambda F + \lambda E)\, p^t(110) = (\lambda F + \lambda E)$$

Putting the value of λ_i and μ_i as per the values in table-1

$$0.5035\, p^t(100) + 0.1285\, p^t(010) + 0.0035\, p^t(110) = 0.0035 \qquad \ldots (15)$$

Subtracting the product of λF and the equation (5) from the equation (2) the result would be,

$$-(\mu F + \lambda E + \lambda F)\, p^t(100) + \lambda F\, p^t(010) + (\mu E + \lambda F)\, p^t(110) = \lambda F$$

Putting the value of μE, μF and λE and λF from the table 4.1 above, the equation would be,

$$-0.5035\, p^t(100) - 0.001\, p^t(010) + 0.124\, p^t(110) = -0.001 \qquad \ldots (16)$$

Similarly subtracting the product of λE and the equation (5) from the equation (3), the result would be,

$$-\lambda E\, p^t(100) - (\mu E + \lambda F + \lambda E)\, p^t(010) + (\mu F - \lambda E)\, p^t(110) = -\lambda E$$

Putting the value of λE, μE, λF & μE on the above equation, result would be,

$$-0.0025\, p^t(100) - 0.1285\, p^t(010) + 0.4975\, p^t(110) = -0.0025 \qquad \ldots (17)$$

From the above three equations (15),(16)&(17) the values of different parameters have been calculated.

Summing up the equations (15) and (16), the result would be,

$$0.1275\, p^t(010) - 0.1275\, p^t(110) = 0.0025 \qquad \ldots (18)$$

On simplification of equation (16) & (17), we get

$$0.0647\, p^t(010) - 0.2502\, p^t(110) = 0.0012625 \qquad \ldots (19)$$

Again, from the equations (18) and (19), the value of $p^t(110)$ and $p^t(010)$ can be found out

Hence $p^t(110) = 0.00051785392 \cong 0.00052$ … (20)
And $p^t(010) = 0.02150952 \cong 0.02151$ …(21)

Putting both the values in equation (15) the value of $p^t(100)$ is found as $0.0014580834 \cong 0.00146$ and, hence $p^t(000) = 0.97651$

Analysis of Group – B:

All the algebraic equations may be considered by keeping the derivatives to zero.

Hence

$$- \lambda LE\, p^t(0, ch) + \mu LE\, p^t(1, ch) = 0 \quad \ldots (22)$$
$$- \mu LE\, p^t(1, ch) + \lambda LE\, p^t(0,ch) = 0 \quad \ldots (23)$$
$$p^t(0,ch) + p^t(1, ch) = 1 \quad \ldots (24)$$

Adding the equation (22) with the product of λLE and the equation (24), the value would be

$$P^t(1, ch) = \frac{\lambda\ LE}{\mu LE + \lambda LE}$$

Putting the value of . λLE and μLE the result will be $p^t(1, ch)$

$$= \quad 0.00149$$

Hence, putting the value of $p^t(1, ch)$, the value $p^t(0,ch)$ is found to be 0.9985

Analysis of Group C:

The derivative of all equation will be zero

Hence the algebraic equation would be :

$$- \lambda BL\, p^t(0, BL) + \mu BL\, p^t(1, BL) = 0 \quad \ldots (25)$$
$$- \mu BL\, p^t(1, BL) + \lambda BL\, p^t(0, BL) = 0 \quad \ldots (26)$$
$$p^t(0,BL) + p^t(1, BL) = 1 \quad \ldots (27)$$

Adding the equation (25) with the product of λBL and the equation (27) and on simplification, we would get the result as :

$$p^t(1,BL) = \frac{\lambda BL}{\mu BL + \lambda BL} = \frac{0.0001}{(0.0833 + 0.0001)}$$

$$= \quad 0.001186 \cong 0.0012$$

And, hence,
$p^t(0,BL) = 0.9988$

Analysis of Group D:

All the derivatives are converted into algebraic equations by making the derivative to zero and hence the result obtained is :

$$- \lambda L\ p^t(0, L) + \mu L\ p^t(1, L) = 0 \quad \ldots (28)$$
$$- \mu L\ p^t(1, L) + \lambda L\ p^t(0, L) = 0 \quad \ldots (29)$$
$$p^t(0,L) + p^t(1,L) = 1 \quad \ldots (30)$$

Adding the equation (28) with the product of μL and the equation (30) and on simplifying the equation, we may get,

$$P^t(1, L) = \frac{\lambda L}{(\mu L + \lambda L)} = \frac{0.005}{0.1 + 0.005} = 0.0476$$

Hence, $p^t(0, L) = 0.9524$

Now, the values of probability states for the processes and operations in the sub-system "working face" shown in the table 4.2, below :

Table – 4.2
Probability states of different mining processes

Sl. No.	Operation	The values of probability state	
1	Drilling	$p^t(000)$	0.97651
		$p^t(100)$	0.00146
		$p^t(010)$	0.02151
		$p^t(110)$	0.00052
2	Charging	$p^t(0,ch)$	0.9985
		$p^t(1,ch)$	0.00149
3	Blasting	$p^t(0,BL)$	0.9988
		$p^t(1,BL)$	0.0012
4	Loading	$p^t(0,L)$	0.9524
		$p^t(1,L)$	0.0476

Working state probability of "working face" altogether is a complicated function of the probability states of the processes and operations and it may be approximately evaluated according to the following method. As mentioned above, during the functioning of the sub-system "working face", the following serially linked processes take place: drilling, charging, blasting and loading. In such cases, a closed circuit of state transitions (Fig – 4.2) for the sub-system "working face" can be used for calculation of state of probability. The central state O denotes continuous work of sub-system, states 1, 2…. j failures of j-process and operation. Any transition is possible only from state

O to state 1, 2…. j and reverse. Transition probability p from state 0 to state 1, 2…. j by time ⊇t is characterized by the intensity of failures λ and intensity of restoration μ which are calculated as the first derivative of probability.

$$\lambda 0j = \frac{\partial p\,\{0 \rightarrow j\}}{\partial T} \qquad \mu j0 = \frac{\partial p\,(j \rightarrow 0)}{\partial T}$$

In accordance with the principles of the transition circuit, a system of differential equations may be represented in relation to the probabilities of all states of sub-system "working face".

$$\frac{dPo}{dT} = -(\lambda 1 + \lambda 2 + \ldots\ldots + \lambda j)\,Po + \mu_1 P_1 + \mu_2 P_2 \ldots. + \mu_j P_j$$

$$\frac{dP_1}{dT} = -\mu_1 P_1 + \lambda_1 P_o$$

$$\frac{dP_j}{dT} = -\mu_j P_j + \lambda_j P_o$$

Where P_o - Probability of working state of sub-system;

P_j - Probability of non-working state of sub-system due to delay of j-process or operation at working face j = 1, 2…

As a result of closed transitions and absence of non-return states, a system of differential equations possesses a stationary solution. Considering unknown probability as constant value, its derivative may be equalized to zero. In this case, we get the system of homogeneous algebraic components. Then the system is reduced to a single equation of the following type.

$$-\quad (\lambda_1 + \lambda_2 + \lambda_3 + \ldots\ldots + \lambda_j)\,Po + \mu_1 P_1 + \mu_2 P_2 + \ldots\ldots\ldots\ \mu_j P_j = 0$$

This equation possesses infinite number of solutions

$$P_j = \frac{\lambda j}{\mu j}\ Po$$

Where probability Po plays a role of independent variable.

Now, by applying a standard condition we get,

$$\sum Pj + P_o = 1$$

$$P_o + \frac{\lambda 1}{\mu 1} P_o + \frac{\lambda 2}{\mu 2} P_o + \ldots\ldots\ldots\ldots\ldots\ldots + \frac{\lambda j}{\mu j} P_o = 1$$

Thus,

$$P_o = \left(1 + \frac{\lambda 1}{\mu 1} + \frac{\lambda 2}{\mu 2} + \ldots\ldots\ldots + \frac{\lambda j}{\mu j}\right)^{-1}$$

$$= \quad (1 + \sum \frac{\lambda j}{\mu j})^{-1}$$

Where λ_i, and μ_i are the values of intensities of failures and restoration respectively for a whole sub-system like a "working face".

The sequence of operation and their interrelationships can be evaluated as:

$$\lambda_0 = (\lambda_1 + \lambda_2 + \ldots\ldots + \lambda_j)$$

$$\mu_0 = \frac{\lambda 0 P0}{1 - P0}$$

Hence, the value of λ_1, λ_2, λ_3, λ_4 and λ_0 has to be looked out. The summation of probability due to seam faults and due to failure of drill will be 1.

$$\lambda 1 = 0.001 + 0.0025 = 0.0035$$

$$P1 = (1 + \frac{\lambda i}{\mu i} + \frac{\lambda ii}{\mu ii})^{-1} = (1 + \frac{0.001}{0.5} + \frac{0.0025}{0.125})^{-1}$$

$$\mu 1 = \lambda 1 \frac{Po(1)}{1 - Po(1)} = \frac{0.0035 \ x \ 0.978473581}{1 - 0.978473581}$$

$$= 0.15909 \approx 0.16$$

Table – 4.3
Values of different mining parameters

Sl. No.	Process/operation	The value of parameters	
		1/hr	**1/hr**
1	Drilling	0.0035	0.16
2	Charging	0.0001	0.2
3	Blasting	0.0001	0.0833
4	Loading	0.005	0.1

Po of working system has to be found out by using the value shown in the table -4.3.
Intensity of the failure for whole sub-system = 0
Probability of possible status of the sub-system "working face" is:

P_o, P_1, P_2, P_3, P_4

$$\text{Hence } \lambda_0 = (0.0035 + 0.0001 + 0.0001 + 0.005) = 0.0087$$

$$P_o = (1 + \frac{\lambda 1}{\mu 1} + \frac{\lambda 2}{\mu 2} + \frac{\lambda 3}{\mu 3} + \frac{\lambda 4}{\mu 4})^{-1}$$

$$= (1.073575048)^{-1} = 0.931467 = 0.9315$$

$$P_1 = \frac{\lambda 1 Po}{(1-Po)} = \frac{0.0035\ x\ 0.9315}{(1-0.9315)} = 0.0476$$

$$P_2 = (\frac{\lambda 2 Po}{1-Po}) = \frac{0.0001\ x\ 0.9315}{1-0.0315} = 0.00136$$

$$P_3 = (\frac{\lambda 3\ Po}{1-Po}) = (\frac{0.0001\ x\ 0.9315}{1-0.9315}) = 0.00136$$

$$P_4 = (\frac{\lambda 4 Po}{1-Po}) = (\frac{0.005\ x\ 0.9315}{1-0.9315}) = 0.06799$$

Hence, the probability of a possible state of the sub-system "working face" is given in the table-4.4, below:

Table –4.4
Probability of the possible state of the sub-system "working face"

Po	P1	P2	P3	P4
0.9315	0.0467	0.00136	0.00136	0.06799

Real productivity of the sub-system "working face" (QF) can be calculated by the formula QF = QT.F $\frac{dPo}{dT}$

Where QT.F = Theoretical output from a working face.

The result obtained shows a high reliability of technological schemes of the sub-system "working face". The most non-reliable processes that decrease the reliability of this sub-system are drilling and loading. This method allows evaluation more objectively of the reliability in functioning of the sub-system and also the real output from the working faces in drilling, blasting as well as in mechanized method of extraction.

4.3 Petri net modeling of drilling operation

With a view to demonstrating the effectiveness of a proposed methodology, different sub-tasks of drilling operation are decomposed into Petri net modeling form and simulated in a computer. A token is fired and various activity points are seen to be active when a movement of token takes place from one place to another through a transition. The typical interpretation (decomposition) of drilling operation with various places and transitions are shown in the fig.4.3 and depicted in detail in the table no 4.5. The first operation is to check the crawler position, mast and other mechanical parts of drill machine and when found all of them in correct position in all respects; the machine is allowed to be marched to the site of drilling. There are various steps for drilling (as described in the table) that can be automated and the system of controlling from a remote place can be implemented after due validation of the model simulated in a computer. This concept can be further analyzed and due attention and study for automation in drilling operation can be paid so that drilling operation independent of an operator may be made a realty in future years.

Table no: – 4.5

Description of drilling operation and its interpretation

	Place Description		**Transition Description**
P0	Checking of the mast & crawler position is complete and marching of drill is ready to commence the task.	T0	The marching of drill is in progress.
P1	The marching of drill is completed.	T1	The levelling of drilling machine by jacks starts and is in progress.
P2	The levelling of machine is completed.	T2	Lifting of the mast to erect position starts and is in progress.
P3	The positioning of drill mast in erect position is completed.	T3	Lowering of drill rod starts to touch the ground and continues.
P4	The process of drill rod touching the ground is completed.	T4	Compressor starts and is in progress.
P5	Drill compressor has been operating.	T5	Drilling operation starts and is in progress.
P6	Drilling operation is completed (the full rod is drilled).	T6	Lifting of rod from the hole starts & is in progress.
P7	Lifting of rod upto the ground level is completed and compressor operation is stopped.	T7	Drill rod is lifted to its original position and lowering down of mast to its original (horizontal) position starts & is in progress.
P8	Lowering down of mast is completed (to its horizontal position).	T8	Levelling of jacks lifting to its initial position one after another starts & is in progress.
P9	All levelling jacks are lifted up and the drill rests on its crawler chains.	T9	Checking up the mast & crawler positioning starts and is in progress.

4.4 Conclusion

The mining operation involves processes like drilling, charging, blasting and loading. They are modeled with the Markovian analysis techniques indicating various probability rates. Charging and blasting operations are conducted with high probability whereas a preliminary process like drilling and a terminal process like loading are conducted with low probability. In order to maximize probability of the overall operation, a proper mix of operations based on Markov chain that cover the working state and the non-working state, has to be adopted. It is convenient to use a standard vector of probability as a criterion of reliability of complicated sub-systems. This is defined as the determination of the working state probability and non-working state probability of the sub-system and its main processes and operations.

The drilling operation in an opencast mine is very cumbersome. It affects health of an operator and helpers to a great extent because they are exposed to an environment fraught with dust, noise and vibration. The concept of automation in drilling operation has been dealt with in a paper that reviews the application of the Petri net modeling technique so that a tedious job of drilling can be conducted with a system independent of an operator. It, however, is a preliminary work on the subject area and further research and analysis is required to validate the models. The application of Petri nets gets acceptance among mining community and it is now taking long strides, to incorporate automation in the operational sectors of mining industries. Presently the Petri net models deal with the operational parameters of drilling. They are modeled and simulated in a computer by applying the “firing rule”. There is enough scope for further study on the subject of drilling which can be validated and automation of a system can be implemented with greater success. It is, therefore, recommended to undertake an in-depth study and analysis of drilling operation in view of application of the Petri net technique. Further, it needs to carry out simulation work on computers and validate the model for actual implementation.

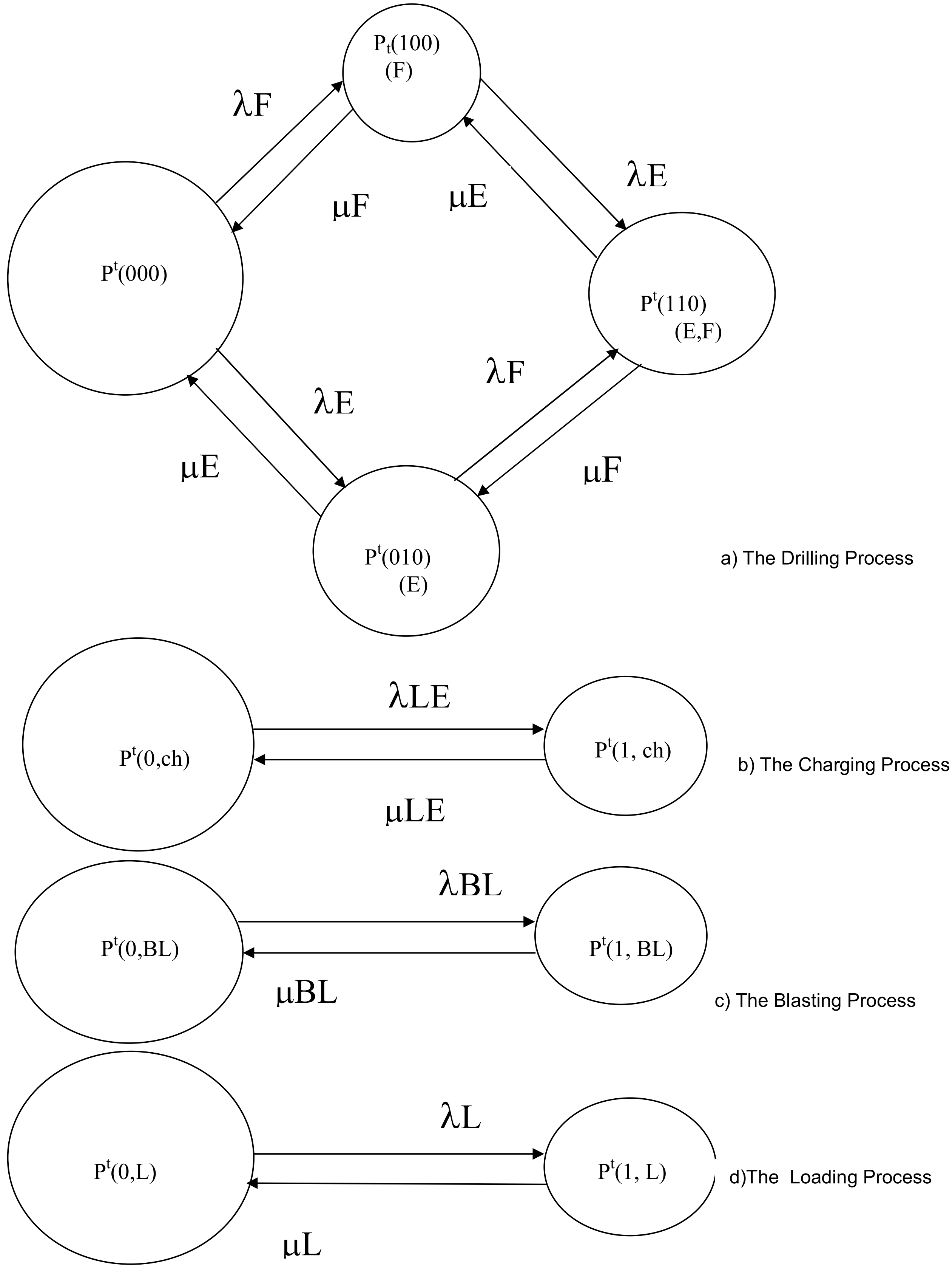


Fig.-4.1: Flow chart of different processes/ operations in the sub system of "WORKING FACE" (states of mutual relation)

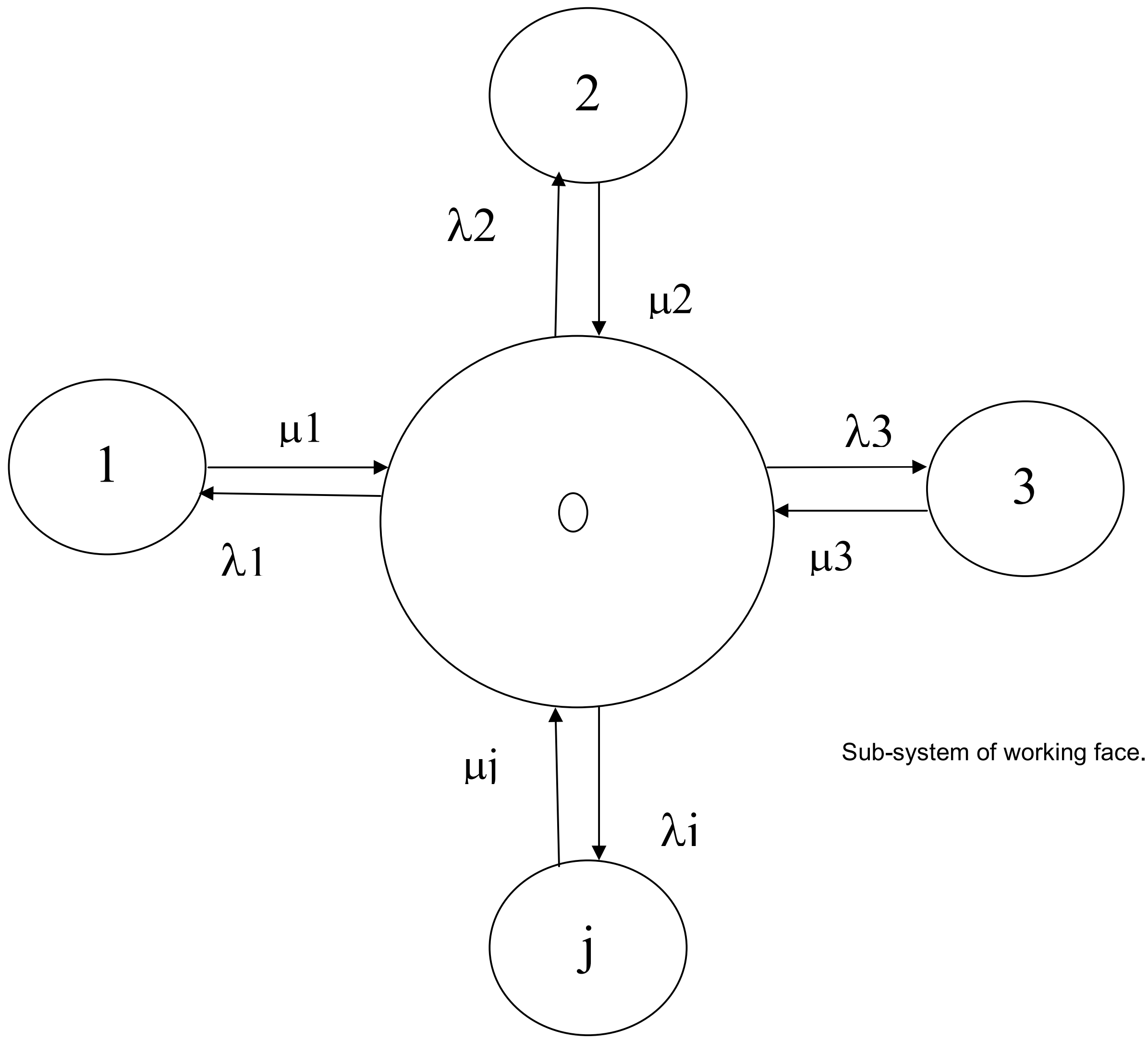


Fig.-4.2: Closed circuit diagram of state transitions for sub-system "working face"

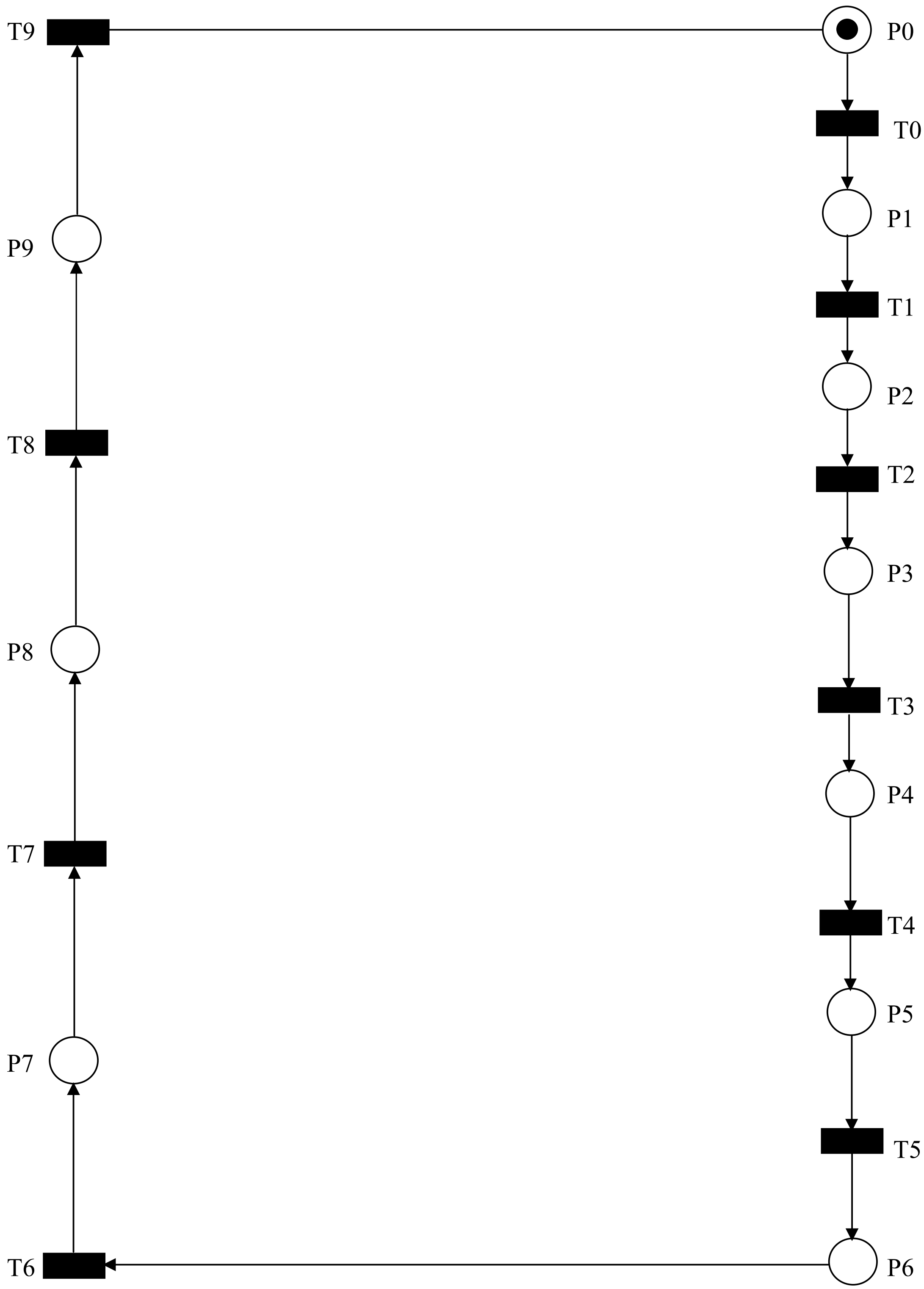


Fig – 4.3: Decomposition of drilling tasks

OPTIMUM BLAST DESIGN MODELS

5.1 Introduction

An ever increasing demand for coal and other minerals in the country and also environmental concerns expressed all over cultivate an interest to achieve higher production targets and also to ensure safety and eco-friendliness. Increasing economic pressures, environmental constraints and safety mandates in recent years call for a precise design of blasting operations at mining industry. An efficient blast design not only reduces an overall mining cost by producing fragmentation of desired size but also ensures eco-friendly mining operations by minimizing ill effects of blasting such as ground vibration, air blast, fly rock etc.

An advent of a large scale mining during the past two decades has brought about a significant change in one's approach to blast design. Computer simulations of blasting processes and prediction of blast results are now standard practices at all major mining operations. In a large opencast mine, a single blast may involve up-to a million tonne of rock. Such a scale precludes a use of trial-and-error method of blast design. Even an occasionally poor blast may significantly affect the economy of an operation. With introduction of computer aided blast design, the recent advances in explosive technology are greatly facilitated. In order to achieve the optimum results, it is now possible to closely match the explosive system, the blast geometry and the rock types.

5.2 Blast Design Patterns

In surface mining, blasting is one among the major operations. It is based on number of parameters namely, the type of rock, hole diameter, the terrain conditions and a desired degree of fragmentation. In order to obtain proper fragmentation with minimum cost, a careful designing of drilling and blasting pattern is essential. A commonly used geometrical design used for most open pit mines is explained as under by Hustrulid :

$$S = K_S B \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots \quad (1)$$
$$B = K_B D \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots \quad (2)$$
$$J = K_J B \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots \quad (3)$$
$$T = K_T B \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots \quad (4)$$
$$H = K_H B \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots \quad (5)$$

Where,

S = Spacing
B = Burden
J = Sub-drill
T = Stemming
H = Bench height
D = Hole diameter

K_S, K_B, K_J, K_T and K_H are constants relating to different parameters (Hustrulid, 1999).

5.3 Modeling of Blasting Process

There are three basic modeling approaches of blasting process:

i) Empirical
ii) Phenomenological
iii) Analytical

An empirical approach relies on the experience factor. It employs a simple criterion based on a ratio of a weight of explosives and a weight or volume of rock to be broken (e.g. powder factor). In case of phenomenological approach, a correlation is sought between certain blast parameters (usually a degree of fragmentation or a volume of rock to be broken) and the energy or a weight of explosives. In this approach, no explicit information is required on failure mechanism of rock explosive energy utilization in various facets of the blasting process or the geology of the rock mass. The following equation represents this approach in a most general form for a given explosive,

$$Q = K_1X + K_2X^2 + K_3 X^3 + K_4 X^4 \ldots\ldots\ldots \quad (6)$$

Where Q is the explosive charge weight,

X is any linear dimension of blasting pattern (usually the burden). The constants K_1 …. K_4 are fitting parameters related to blast geometry and the rock. A main drawback of both these approaches is that they neglect a dynamic process underlying fragmentation and the nature of explosive energy partition.

Analytical approaches, on the other hand, do take into account these factors. They can be realistically incorporated in a blasting model. They have potentials for predicting blast results and provide cost effective blast designs.

Optimum blasting is one of the key areas of economical production of coal in an opencast mine. The primary requisite of any blasting is to get optimum result in existing operating conditions. In general, an optimum blast design provides adequate tonnage of coal or volume of overburden with suitable fragmentation. It ensures smooth loading, transportation and subsequent processing or disposal at a minimum cost. It may also take due care of safety and environmental parameters in accordance with the provisions of statutory requirements prevalent at a time. Very often a mine management faces problem in proper placement and design of the blasting operations. It may be because of the fact that a relationship among various parameters used in designing the geometry of the blast is not properly established. This result is either an over-achievement or an under-achievement of blasted material (i.e. coal or OB). An over-achievement leads to over utilization of the resources (man, machine, money). An under-achievement causes creation of boulders or even cracks in a face. It further causes problems in loading or sometimes in rearrangement of drills to carry out the repeat operation. This results in loss of time, energy and money and also endangers safety. In both the cases, there is financial loss. They affect adversely safety and environment aspects. Thus, neither of the above two situations are desired in the interest of economical production.

In order to avoid the above situation, it is felt to develop mathematical models which can predict accurately a volume of material available after for blasting under different operational conditions and different geometric configurations of blast. In an ongoing work, attempts are made to develop a suitable mathematical model to predict a volume of material for blasting as a function of controllable, operational and geometric variables. A study of vibration is also conducted to evaluate a safe charge in a round and maximum charge/delay at various distances of a blast site from the structures, which are not to be disturbed. Identification of significant decision variables used in the multivariate regression analysis is incorporated in the following line of study beginning with the section 5.4. A formulation of multivariate linear regression model for optimum blast vibration results are then dealt with in the section 5.5. The data for a vibration study is again collected from a field. This point is elaborated in the section 5.6. The Section 5.7 deals with the results of the regression and its analysis for optimum blast design. Different decision variables for fragmentation study are elaborated in the section 5.8. Formulation of regression model is developed in the section 5.9. Analysis of a case study for optimum blast results is briefed in the section 5.10. Investigation of the study is discussed in the section 5.11. And the section 5.12 deals with an analysis of the results of the study and the section 5.13 discuss the limitations of regression models. Concluding remarks are presented in the section 5.14.

5.4 Decision Variables

A classification of variables is absolutely relative. A variable that is considered as an independent variable for one problem may be considered as a dependent variable for the other. The first step in the construction of a mathematical decision model is an identification of controllable independent variables, more commonly known as decision variables. The articulation of decision variables constitutes a basis for the remaining step in the decision model development.

The variables involved in the present study are as follows:

A) Dependent controllable variable:

i) Geometric volume of blasting.

B) Independent controllable variables:

i) Burden

ii) Spacing

iii) Bench height

iv) Borehole diameter

v) Average charge/hole

5.4.1 Burden

The most critical and significant dimension in blasting is that of a burden. There are two requirements to define it properly. To cover all conditions, a burden should be considered as a distance from a charge axis measured perpendicularly to a nearest free face. In this direction displacement is most likely to occur. Its actual value will depend on a combination of variables including the rock characteristics, the explosives used, etc. But when a rock is completely fragmented and displaced a little or not at all, one can assume that critical value has been approached. Usually, most blasters prefer an amount slightly lower than the critical value. A position of burden and spacing (drilling pattern) and breakage pattern with variation of burden in a bench is shown in the figures 5.1 and 5.2 respectively.

A number of empirical relationships are proposed to design a blast and to obtain an approximate value of the burden. Some of the relationships are discussed here:

1) Based on specific gravity of explosives and the rock.

$$B = 3.15De\,(SGe/SGr)^{0.33}$$

Where B is the burden in ft;
De is the diameter of explosives in inches;
SGe is the specific gravity of explosives in gm/cc;
SGr is the specific gravity of rock (in gm/cc);

Later Konya (1983) defined the following relation:

$$B = [(2SGe/SGr) + 1.5]\,De$$

2) Based on the type of explosive loading density (de) and drill hole diameter,

$$B = 1.087\,de^{1/2}$$

Where B is the burden in m,
de is the loading density in kg/m

3) Based on rock strength and type of explosive (Allsman and Speatch, 1960).

$$B = (KD_e/12)\ (P_e/S_t)^{0.5}$$

Where B is the burden in ft;

K is the constant (0.8 for most of rocks);

D_e is the charge diameter in inches,

P_e is the explosion pressure;

S_t is the tensile strength of the rock;

There are many other formulas that provide approximate burden values. But the most required calculations are bothersome or complex to an average man in the field.

A convenient guide that can be used for estimating the burden, however is the K_B ratio (Burden ratio)

$$B = K_B D$$

Where K_B is burden ratio

D is the hole diameter.

Experience shows that when K_B = 30, the blaster can usually expect satisfactory results for average field conditions. To provide a greater throw, the K_B value could be reduced below 30, and subsequently, finer sizing is expected to result. Light density explosives, such as field mixed ANFO mixtures necessarily require the use of lower K_B ratios (20 to 25), while dense explosive, such as slurries and gelatins permit the use of K_B near 40. The final value selected should be the result of adjustments made to suit not only the rock and explosive types and densities but also a degree of fragmentation and displacement desired. To estimate the desired K_B value, one has to know that densities for explosives are rarely greater than 1.6 or less than 0.8 gm/cm^3. Also for rocks requiring blasting, the density in gm/cm^3 rarely exceeds 3.2, nor is it less than 2.2 with 2.7 for the most common value.

Thus, the blaster can, by first approximating the burden at a K_B of 30, make simple estimations for 20 (or 40) to suit a rock and explosive characteristics and densities.

- For light explosives in dense rock , K_B = 20,
- For heavy explosives in light rock , K_B = 40,
- For high explosive in average rock, K_B = 25,
- For heavy explosive in average rock, KB = 35.

5.4.2 Spacing

Spacing can be defined as a distance between two adjacent blast holes measured perpendicular to the burden. It controls the mutual effect between the holes. Spacing is calculated as a function of burden, hole depth, relative primer location between adjacent charges and also depends upon initiation time interval. A calculation of spacing in relation to burden is worked out by many scholars. They may be summarized as follows:

1) Konya (1983)

$S = 1.15 – 1.4B$

2) T.N.Hagan

$S = B$, for adequate results

$S = 1.15B$ for hard, massive rocks

3) Vutukuri and Bhandari (1973)

$S = 0.9B + 0.91$

4) Ash (1963)

$S = K_S B$

K_S = 1 to 2 (Spacing Ratio)

Where spacing (S) and burden (B) are in mts.

Ideal energy balancing between charges is accomplished usually when the spacing dimension is nearly equal to the double that of the burden (K_S=2) and when charges are initiated simultaneously. For long interval delays, the spacing should approximate the burden, or K_S=1. For short periods delay, the K_S value will vary from 1 to 2 depending upon the interval used. However, since structural planes of weakness such as jointing, etc. are not actually perpendicular to one another, the exact value of K_S normally will vary from 1.2 to 1.8, the preferred value of which must be tailored to local conditions.

5.4.3 Bench height

The height of an individual bench depends on the depth and capacity of drilling equipment and also on a degree of fragmentation required for a particular rock. Some bench heights are necessarily determined by the angle stratification of the rock and by a presence of clay seams or planes of weakness. Usually in mines, its value is relatively constant and is set to conform to the working specification (i.e. boom height) of loading equipment. A bench height is related to a degree of keeping and spreading of material broken by blasting. It, thus, directly affects the requirements of displacement to be accomplished by a round design. A position of holes in a bench before and after the blasting is shown in the fig- 5.3.

5.4.4 Borehole diameter

A selection of hole diameter is governed with several factors, such as bench height, critical diameter of charge, drilling cost, production requirement, rock structure (block size), the required degree of fragmentation, environmental constraints and unit cost of production. From the detonation theory of explosives, a hole diameter should be greater than the critical diameter of the charge. Environmental constraints, rock structure and cost of production also decide a hole diameter to be selected for blasting. The height also limits the maximum and minimum charge diameter that should be used. It influences drill selection, which can be expressed as given below:

$d_{min} = 10H$

$d_{max} = 16.66H + 50$

Where d_{min} is the minimum hole diameter (mm)

d_{max} is the maximum hole diameter (mm)

H is the bench height (m)

5.4.5 Average charge/hole

Blast holes are normally maintained at equal depths for uniform floor gradient. But sometimes situations may arise where depth of the holes are not equal, and thereby it may create a variation in charging of holes. In such a case, an average charge/hole can be calculated by summing up all the charges and dividing it by a number of holes in that particular round of blast.

5.4.6 Peak particle velocity

Particle velocity represents the velocity of a particle at any instant of time during the vibration disturbance. So particle velocity is the rate of change of particle displacement with respect to time. This is a speed of excitation of particles in the ground resulting from the velocity of propagation of a rock. Peak particle velocity (ppv) is the maximum velocity from the position of rest. Peak particle velocity is measured in terms of mm/sec by a modern seismograph machine that is highly sensitive electronic instrument designed to measure and record the intensity of ground vibrations with dominating frequency band. This ppv does not represent distances that the ground moves, but rather the speed with which the ground vibrates.

The magnitude of ground vibration depends upon various factors such as geological features of the blasting face, type of explosives, blast designs, maximum charge per delay, total charge per round, strata where the blast is to be performed, and distance of the monitoring point from the blasting face. Vibration can be reduced to safe limit by optimizing a blast design as well as the above parameters. The table-5.1 below shows the effect of vibration on residential structures.

Table – 5.1

Effect of vibration on residential structures

Peak particle velocity (ppv) [mm/sec]	Effects on the structures
250	Cracks in solid concrete slabs or wall may appear
125	Cracks in masonry may begin to appear
75	Cracking may begin in mortar joints, in concrete block foundations
50	Above this level, there is a possibility of structural damage occurring
25	New cracks in dry wall may appear
19	Existing cracks in dry wall may extend
12.5	Cracks in old plaster may appear. Existing cracks in plaster may extend
7.5	Vibrations are easily detectable by people

(Source:CMRI Reports)

Low frequency vibrations are of great concern. The reason is that amplitudes at excitation frequencies that encompass fundamental frequencies of the structures produce the greatest response displacement. The typical structural fundamental frequencies are 4-10Hz for one or two storey structures and 10-15 Hz for wall and floor. Analyses of vibration records in Indian geo-mining conditions indicate that opencast coal mine blasts generate low frequency vibrations whereas non-coal mine and underground coal mine blasts produce high frequency vibrations (Singh et.al. 1996). The table-5.2 depicts the thresh hold value of vibration at a foundation level of structures as suggested by Director General Mines Safety (DGMS) Technical circular 7 of 1997.

Table – 5.2

Thresh-hold values of vibration

Sl. No.	Type of structure	Dominant excitation frequency		
		<8 Hz	**8-25Hz**	**>25Hz**
A	**Building/structure not belonging to the owner**			
1	Domestic houses/structures (Kuccha, brick and cement)	5	10	15
2	Industrial building	10	20	25
3	Objects of historical importance and sensitive structures	2	5	10
B	**Buildings belonging to owner with limited span of life**			
1	Domestic houses/structures	10	15	25
2	Industrial buildings	15	25	50

(Source: DGMS (Circular 7 of 1997)

5.4.7 Maximum charge/delay

Maximum charge/delay is one of the important parameters that affect vibration of a particular round of blast. This factor is judiciously decided before conducting a trial blast, and then, the optimum quantity is calculated mathematically. It complies the requirements of the statute in respect of limits of vibration. Maximum charge/delay varies for coal and OB benches. Now a days, DTH, TLD and, in the hole delay serves a great purpose to reduce this parameter to meet the regulatory needs of a mine.

5.4.8 Total charge in a round

This is a restriction normally imposed by the DGMS for cases like a blasting site is located very close to structures or villages. In accordance with the provisions of the Coal Mines Regulation 1957, the owner, agent or manager needs to take permission from the DGMS to carry out blasting operations at a mine located within a distance of 300m from any residential buildings, structures, roads, railways that do not belong to owner. In such cases, the DGMS advises him / concerned person to carry out trial blasts by any approved scientific body or institute to fix up total charge/round and maximum charge/delay so that the above nearby structures or human settlements are not adversely affected or any life or property is endangered due to the blasting operations at a mine. The total charge/round is a decisive factor to control vibration in a particular blast operation. The blaster has to abide by the restrictions imposed by the DGMS for this total charge/round and maximum charge/delay in order to prevent any damage caused to nearly structures or human life due to blasts.

5.4.9 Distance of the monitoring station from the blast site

A distance of structures from a blast site is a critical parameter from the point of view of vibration. It is inversely proportional to each other. It means that the more is the distance of structures from a blast site, the less would be damage caused to them due to vibration. Normally, this distance is determined, because we have to carry out blasting operations at a place where there is presence of coal seam. As such villages, roads or railway lines are located at fixed locations and they cannot be shifted every now and then. Hence, we have to compromise the maximum charge/delay and total charge/round keeping optimum distance of the structures to a blast for structures site. Very often complaints are received from villagers that are caused to their houses due to blasting operations at mines. This further needs a careful study and investigation of vibration caused by blasts that a recognized agency or mine personnel exclusively trained for the purpose. He should conduct tests using the vibration monitor or any microcomputer based seismograph.

5.5 Formulation of regression model

In the present study, an analysis of blast vibration data is conducted to develop a suitable mathematical model. It helps to predict the future course of action to conduct controlled blasting operations at a mine. The study keeps in view the variation of dependent variables and its effect on the stability of structures. The different parameters used in the formulation of the model are as follows:

Notations:

Vm	=	Peak particle velocity in mm/sec.
Q	=	The maximum charge/delay in kg.
D	=	Distance of the monitoring point from the blast site in mts.
k, m	=	Constants dependent upon the rock, explosive and blast design parameters.
$D/Q^{1}/_{2}$	=	Square root scaled distance or simple scaled distance.

The fundamental predictor equation of ground vibration is represented in the following form:

$$Vm = k\,(D/Q^{1}/_{2})^{m} \qquad \ldots\ldots\ldots\ldots \qquad (1)$$

Taking Log on both sides of the equation (1), we get

$$\text{Log}\,(Vm) = \text{Log}k + m\,\text{Log}\,(D/Q^{1}/_{2}) \qquad \ldots\ldots\ldots\ldots \qquad (2)$$

This can be written in the form of a straight line as

$$Y = mx + c \qquad \ldots\ldots\ldots\ldots \qquad (3)$$

Where

$$Y = \text{Log}\,(Vm) \qquad \ldots\ldots\ldots\ldots \qquad (4)$$

$$x = \text{Log}\,(D/Q^{1}/_{2}) \qquad \ldots\ldots\ldots\ldots \qquad (5)$$

$$c = \text{Log}\,k \qquad \ldots\ldots\ldots\ldots \qquad (6)$$

The x - y relationship in equation (3) is obviously a straight line with the slope 'm' and the Y intercept 'c' in order to plot the line, the values of x and Y are calculated using the values of Vm, D and Q.

Now, a shortcut method through which m and c can be directly obtained is:

$$m = \frac{\sum xY - \sum x.\sum Y}{n\sum x^2 - (\sum x)^2}$$

$$c = \frac{\sum Y}{n} - m\frac{\sum x}{n}$$

$$Y = mx + c$$

5.6 Data Collection

In view of developing a regression model, the required data were collected from the site and also from the available records. The raw data is converted into a usable form in accordance with the requirements. The details on the status of the data collected for the purpose are illustrated in table No.5.3

Table No.5.3

Blasting data collected from the field

Sl. No.	PPV (mm/sec)	Distance	Quantity	$D/A^{1/2}$	y=log (ppv)	x=log $(D/Q^{1/2})$	xy	x^2
1	1.3	300	55	40.45	0.11	1.61	0.18	2.58
2	1.4	290	175	21.92	0.15	1.34	0.20	1.80
3	2	280	65	34.73	0.30	1.54	0.46	2.37
4	1.5	300	190	21.76	0.18	1.34	0.24	1.79
5	1.5	270	170	20.71	0.18	1.32	0.23	1.73
6	2	310	160	24.51	0.30	1.39	0.42	1.93
7	1.5	300	135	25.82	0.18	1.41	0.25	1.99
8	1.7	310	135	26.68	0.23	1.43	0.33	2.03
9	1.5	320	160	25.30	0.18	1.40	0.25	1.97
10	2.1	310	185	22.79	0.32	1.36	0.44	1.84
11	1.6	280	40	44.27	0.20	1.65	0.34	2.71
12	1.5	300	65	37.21	0.18	1.57	0.28	2.47
13	1.4	300	60	38.73	0.15	1.59	0.23	2.52
14	2	325	180	24.22	0.30	1.38	0.42	1.92
15	1.3	300	140	25.35	0.11	1.40	0.16	1.97
16	1.1	315	145	26.16	0.04	1.42	0.06	2.01
17	1.8	325	50	45.96	0.26	1.66	0.42	2.76
18	2	325	55	43.82	0.30	1.64	0.49	2.70
19	2.2	325	80	36.34	0.34	1.56	0.53	2.43
20	2	300	60	38.73	0.30	1.59	0.48	2.52
21	2.1	290	40	45.85	0.32	1.66	0.54	2.76
22	1.8	300	110	28.60	0.26	1.46	0.37	2.12
23	2.3	310	90	32.68	0.36	1.51	0.55	2.29
24	1.5	325	45	48.45	0.18	1.69	0.30	2.84
25	2.2	325	125	29.07	0.34	1.46	0.50	2.14
26	1.2	300	55	40.45	0.08	1.61	0.13	2.58
27	2	300	65	37.21	0.30	1.57	0.47	2.47
28	1.8	290	60	37.44	0.26	1.57	0.40	2.48
29	1.7	305	60	39.38	0.23	1.60	0.37	2.54
30	2.5	320	140	27.04	0.40	1.43	0.57	2.05

(Source: Field data)

5.7 Results and Analysis

Intensity of blast vibrations attenuates with a distance and also depends on the maximum charge per delay. Peak particle velocity is taken as a criterion of blasting damage. The maximum ppv is worked out to be 2.5mm/sec. with the maximum charge/delay as 140kg at a distance of 320m.

The raw data is analysed and the results thereof is represented below by using the regression model technique.

$$Y = mx + c$$
$$y = (-)\ 27.36x + 41.41$$

Accordingly, a graph is plotted below as shown in fig.5.4. The ppv and distance/sqrt root charge are shown. The relationship indicates that ppv decreases when charge in the holes decreases or else, if the distance is increased. In order to find maximum charge/hole for a blast with respect to the structures at fixed distances, the stipulated ppv has to be maintained as per the requirements of the statute. The above regression model can be adopted in thesame opencast mine. This case analysis is site specific and can not be adopted for general use at other mines. The project specific criterion may be different for different opencast mines. That is why, a study of blasting parameters and their effects with varying charges & distances need to be conducted and then analyzed to develop a model for a specified project.

5.8 Decision Variables for optimum fragmentation

The variables are classified relatively and they are considered as independent variables for one problem may be considered as a dependent variable for the other. The decision variables are controllable independent variables being an useful tool for formulation of mathematical decision models.

The different variables used in the present study are as follows:

A) A dependent controllable variable is

i) Y = Geometric volume of blasting.

B) Independent controllable variables are:

i) X_1 = Borehole diameter (BH Dia)
ii) X_2 = Borehole Depth (BH Depth)
iii) X_3 = Burden (B)
iv) X_4 = Spacing (S)
v) X_5 = Avg. charge/Hole (ACH)

The notations are:

x_i	=	Space variables/i^{th} decision of independent variables
b_i	=	regression co-efficient of the i^{th} variables
a	=	intercept of dependent variable
		Y at origin is also known as regression constant
r^2	=	Co-efficient of regression
ej	=	j^{th} element of the set of error functions
Xij	=	j^{th} observation of variable Xi.

5.9 Formulation of Multivariate Linear Regression Model

In many problems, there are two or more variables that are inherently related. There is a need to explore the relationship among the variables. A single dependent variable or response Y depends on K independent variables, e.g.

$$Y = f(X_1, X_2, X_3 \ldots\ldots X_k) \quad \ldots\ldots\ldots\ldots\ldots\ldots \quad (7)$$

If it is assumed that there is a linear dependency of Y on X_1, X_2, X_3….. X_k, the functional relationship for fitting the model can be expressed as,

$$Y = a_o + a_1X_1 + a_2X_2 \ldots\ldots\ldots\ldots a_kX_k + e \quad \ldots\ldots \quad (8)$$

A general problem of fitting the above model is called multivariate or multiple linear regression problems.

The model describes a hyper plane in a K-dimensioned space of the independent variables (X_i). The unknown parameters (a_i) are called regression co-efficient. The equation (8) can be simplified as:

$$Yj = ao + \sum_{i=1}^{k} bi \; X\,ij + ej \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots \quad (9)$$

For j = 1,2,3 ……………………….. n

The intercept is defined as:

$$\acute{a}o = ao + b1\,\overleftrightarrow{X}_1 + b2\,\overleftrightarrow{X}_2 + \ldots\ldots\ldots\, b_k\,\overleftrightarrow{X}_k \ldots\ldots \quad (10)$$

Where

$$\overleftrightarrow{X}_i = \frac{1}{n} \sum_{j=1}^{n} X_{ij} \text{ is the average level for its } i^{th} \text{ variable}$$

Now $ao = áo – b_1 \overleftrightarrow{X}_1 – b_2 \overleftrightarrow{X}_2 - \ldots\ldots\ldots\ldots\ b_k \overleftrightarrow{X}_k$

$= áo\ \ b_i \overleftrightarrow{X}_I \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots \quad (11)$

The model now becomes

$$Yj = áo - \sum_{i=1}^{k} b_i \overleftrightarrow{X}i + \sum_{i=1}^{k} bi\, Xij + ej$$

$$= áo + \sum_{i=1}^{k} bi\,(Xij - \overleftrightarrow{X}i) + ej \quad \ldots\ldots\ \ldots\ldots \quad (12)$$

for $J = 1,2,3, \ldots\ldots\ldots\ldots\ldots n$

$$ej = Yj - áo - \sum_{i=1}^{k} bi\,(Xij - \overleftrightarrow{X}i) \ldots\ldots\ldots\ldots\ldots \quad (13)$$

Now, apply the method of least squares to the differences between the response Yj and the predicted value of the response at Xi

$$ej^2 = [Yi - áo - \sum_{i=1}^{k} b_i\,(X_{ij} – X_i)]^2 \ldots\ldots\ldots\ldots\ldots\ldots\ldots(14)$$

$$\sum_{j=1}^{n} ej^2 = \sum_{j=1}^{n} [Yi - áo - \sum_{i=1}^{k} bi\,(Xij – Xi)]^2 \ldots(15)$$

In order to find out the values of áo, bi, we have to minimize $\sum_{j=1}^{n} ej^2$ by partial derivatives j=1

with respect to áo, bi, b2, b3 ………..bx

$$\frac{\partial \sum_{j=1}^{n} ej^2}{\partial\, áo} = \frac{\partial \sum_{j=1}^{n}}{\partial\, áo} [Yi - áo - \sum_{i=1}^{k} bi\,(Xij – X)]^2 \quad \ldots (16)$$

$$\frac{\partial \sum_{j=1}^{n} ej^2}{\partial\, (bi)} = \frac{\partial}{\partial\, bi} \sum_{i=1}^{k} [Yi - áo - \sum_{i=1}^{k} bi\,(Xij – Xi)]^2 \ldots (17)$$

On simplifying the equations (16) & (17), we can get the least square normal equations. Using Gauss-Jordan method for solving a system of linear equations, the value of áo, b_1,b_2,b_3 ………b_k can be found out.

On substitution of these values to equation (8), we can get the expression of "Y" a linear function of X_1, X_2, X_3 ………..X_n.

5.10 Analysis of Case Study

The present research is concerned with an establishment and development of geometric volume of blasting as a function of various controllable geometric and operating variables. The blasting volume depends on a number of variables besides the geometric and operating variables. For a particular set of strata conditions and explosive characteristics, optimum blast design parameters can be modeled for a specified mine. A thorough discussion was held with the highly experienced persons who were responsible for the blast design. Accordingly, the data were collected for the purpose of developing a model of an operating mine. The opencast project chosen for the present study belongs to the Public Sector organization, "Mahanadi Coalfields Limited" which is a subsidiary of "Coal India Limited". The data were collected both from the spots and from records available at the company's offices.

To be specific, the field data were collected from opencast quarries that have the same rock characteristics and that use the similar types of explosives for blasting. The data is arranged and presented as shown in the table: 5.4 below:

Table No.5.4

Field data for optimum blast design

Sl. No.	Borehole dia (mm)	Borehole depth (m)	Burden (m)	Spacing (m)	Avg. charge/ hole (kg)	Geometric volume of blast (cum)
1	250	9.0	6	6	133.54	14580
2	160	7.0	4	5	54.16	4480
3	250	8.0	6	6	150.30	4032
4	160	7	4	5	50.10	4200
5	250	8	6	6	173.42	4608
6	160	7	4	5	50.10	3780
7	160	7	4	5	50.10	3780
8	250	9	6	6	188.44	11016
9	160	7	4	5	50.10	4900
10	160	7	4	5	50.10	9800
11	160	7	4	5	52.82	7700
12	250	8	6	6	165.56	5472
13	160	7	4	5	48.88	11480
14	160	7	4	5	50.10	9100
15	160	7	4	5	50.10	8120
16	160	7	4	5	47.15	4760
17	160	7	4	5	51.52	4900

18	160	7	4	5	50.10	7980
19	160	7	4	5	52.18	3360
20	160	7	4	5	50.10	5600
21	250	9	6	6	150.25	5184
22	160	7	4	5	54.26	1680
23	160	7	4	5	51.95	3780
24	160	7	4	5	62.50	11200
25	160	7	4	5	60.10	6300
26	160	7	4	5	58.67	4900
27	250	8	6	6	120.20	5760
28	250	9	6	6	120.20	6480
29	250	8	5	6	122.75	7440
30	250	9	6	6	132.60	17696
31	250	8	6	6	91.10	3168
32	250	8	6	6	128.60	6624
33	250	7	4.5	5	64.88	3623
34	160	7	4	5	50.10	2800
35	250	8	6	6	109.72	6048

(Source: Field data)

The normal equations were formed from the above data table:

$$226331 = 35b_0 + 6860b_1 + 263b_2 + 165.5b_3 + 188b_4 + 2946.75b_5 \quad \ldots (18)$$

$$45368750 = 6860b_0 + 1412600b_1 + 52520b_2 + 35815b_3 + 37550b_4 + 638120.4b_5 \quad \ldots.(19)$$

$$1737381 = 263b_0 + 52520b_1 + 1995b_2 + 1265.5b_3 + 1424b_4 + 23138.96b_5 \quad \ldots. (20)$$

$$1095912 = 165.5b_0 + 33815b_1 + 1265.5b_2 + 813.5b_3 + 904.5b_4 + 15270.05b_5 \quad \ldots. (21)$$

$$1229763 = 188b_0 + 37550b_1 + 1424b_2 + 904.5b_3 + 1018b_4 + 16520.43b_5 \quad \ldots. (22)$$

$$20408280 = 2946.75b_0 + 638120.4b_1 + 23138.96b_2 + 15270.05b_3 + 16520.43b_4 + 316.40b_5 \quad \ldots .(23)$$

The values of b_0, b_1, b_2, b_3, b_4, b_5 need to be found out from the above equations and substituted in the equation given below :.

$$Y = b_0 + b_1x_1 + b_2x_2 + b_3x_3 + b_4x_4 + b_5x_5$$

5.11 Investigation of the results

The linear multiple regression model is developed as shown below using the data and normal equations:

$$Y = 477.85-0.8120(X_1) + 38.779(X_2) - 6.575(X_3) + 68.937(X_4) + 0.0647(X_5)$$

Geometric volume of blasting:

$$Y = 477.8458 - 0.812014\ (BHDia) + 38.7790\ (BH\ Depth) - 6.5751(B) + 68.9371(S) + 0.064\ (ACH)$$

Altogether the five decision/independent variables such as Borehole dia, borehole depth, burden, spacing and avg. charge/hole are considered in view of the effect they exert on the blast volume.

The effect of increasing the borehole diameter is negative on geometric volume of blasting. This indicates that if we increase the borehole diameter keeping the other variables unchanged, the volume decreases, as the charge becomes insufficient to blast the entire volume of rock. The resultant volume is, thus, less than a calculated geometric volume and as a result, the fragmentation is poorer.

The borehole depth bears a positive relation with a geometric volume of blasting. It shows that an increase in the borehole depth from 7m to 8m results in increase of the geometric volume of blasting. Normally, the borehole depth is determined on the basis of bench height that is available. In the present study the bench height of OB is 8-9m and 250mm drill is engaged exclusively for drilling in OB faces. And 160mm drill is used for 6-7m bench height for both coal and OB drilling as per the requirement of the faces in a mine.

An important decision variable is the burden. It has negative value in the present analysis. This negative value indicates that as the burden increases the volume will reduce. The reason is that the shock wave that passes through blasted material will not be optimum. Thus, the objective of smooth loading by shovel to the dumpers can not be carried out productively. This is crucial in decision making process of optimum fragmentation size to cater to the need of a bucket size/capacity of the loading machine.

In case of increase of spacing, the impact on the geometric volume of blasting is positive. It shows an increase of volume of blasting. This is revealed in the present study results. But there is always a limit to increase the spacing. The usual limit of spacing is 1 to 1.2 times the burden. But under no circumstances, it should be less than the burden.

The average charge per hole is a determining factor for production of fines and fragments in a blasting process. In cases where average charge per hole increases the volume of blasting increases nominally. The result of the present study too indicates positive value in the regression equation. The more is the average charge per hole the more would be the fines produced.

So, a limit is always determined for mines and, in that view, the statutory bodies impose restrictions over an average charge per delay and total charge in a round to limit the vibration in accordance with the provisions of the Regulations. These restrictions are based on the studies conducted by scientific bodies approved by the government. The studies specify the limits of the above two factors so that the structures are not affected adversely or damaged due to the vibration impact of blasting, and the peak particle velocity is maintained, on the other hand, within the statutory limits.

5.12 Analysis of the results:

i) The blast volume drops because of the reasons like the fragmentation of a rock does not get free surface and ultimately it remains in the bench as an integral part of a broken rock.

ii) The variation of spacing with blast volume is observed from the prediction equation. An increase of one unit of spacing causes an increase of volume by 68.9371 units. It is because of the geometry of blast design. An increase in spacing increases a width of blast geometry. Hence, it is logical and consistent.

iii) Any increase in burden as explained by regression equation tends to decrease the volume of blast. With increase of one unit of burden there is a decrease of 6.5751 units of volume. This happens because of the fact that the volume of blast is maximum at optimum burden and if further the burden is increased, it causes poor fragmentation resulting in huge lumps and back breaks. Hence, our result is logical and consistent. It is also verified with the field experience and also with people having an experience of blast design.

iv) The variation of blast volume is observed with variation in the average charge per hole. It is found from the regression analysis that when there is an increase of one unit in the average charge per hole, it increases the volume of blasting by 0.064 units. The data taken into consideration for the present research are related to the cases in which the same type of explosives is used. This is also verified from the site and discussed with the blasting experts. It is logically consistent.

v) From the prediction equation, it is found that there is little decrease (i.e. 0.8120 units) of blast volume with an increase of one unit of the blast hole diameter. This happens because the parameters like burden, spacing and depth of hole are fixed. This is quite logical and consistent. On discussing the issue with the experts of blasting and on observation of the actual blast operation, it was found that the above analysis holds relevance in majority of cases.

vi) The effect of borehole depth on blast volume as obtained from the regression analysis shows a rise of 38.7790 units of the latter with unit increase of the former. This is because of the geometry of a blast as the blast volume is a product of bench height, burden and spacing. This seems quite logical.

vii) The average charge/hole for 250mm and 160mm dia hole varies from 115kg to 190kg and 50 kg to 62kg respectively as per the representation on graph (fig.5.5).

viii) The variation of geometric volume of blast and average charge/hole is shown in the fig.5.6.

5.13 Limitation

The multiple linear regression has its limitations. It works on the principle of least squares method of best fit. The results obtained by an application of least squares method get affected by extreme cases, as the outlines get undue weightage. The effect of additions or deletions of decision variables can be examined with difficulty in a regression method, because sensitivity analysis involves intricate computations.

Hence, our results are also affected with the limitations, typical to any regression analysis method. The accuracy of data is of vital importance in order to arrive at an efficient model. The data are collected here for the prediction of blast volume and the subsequent results obtained from the regression analysis highlight the following salient points in respect of rate of blast volume performance.

i) In order to break out selected sections of the rock/coal, in explosive charge is placed within a rock and at a suitable distance.

For this purpose, an opening is made into a rock. The rock mass must also have one or more free face, i.e. it must be exposed or open on one or more planes more or less at right angles to that from which the drilling is done. The rock is blasted in the direction of the free face. This is necessary because a broken rock occupies a much greater space as compared to the space that a solid mass of rock occupies. Hence, the true volume of broken solid needs high accuracy of measurement.

ii) Performance of blasting depends on the intimacy of the mixing process of the main components i.e. slurry. Other important parameters such as the density of charge, the degree of confinement qof the charge, the absence of excess moisture variables, strength, shape and position of booster are also important for better blast performance. The present research, however, does not include the data to that effect, as it seems to fall out of its present scopes. It, however, leaves a scope for further research and analysis that may be conducted in incorporating the data of that kind.

iii) Reliability of data directly affects the linear multiple regression model. Sometimes, they may not be so accurate because of various unseen parameters like shallower holes or uneven distribution of the explosive charge over the length of the hole.

5.14 Conclusion

In view of the parameters of the regression model and its resultant the understanding of fundamentals of ground vibration is much required. Any blasting engineer gets the optimum blast results with understanding of this kind. Human beings notice and reject blast induced vibrations that are at levels lower than the damaged thresholds. A study made is an application of regression technique in blasting considering the two variables like distance and charge weight per delay. It finds out the maximum ppv that needs to remain well within the stipulations laid down by the Director General of Mines Safety, Govt. of India. There is enough scope of further research in this area by which models may be developed using other OR techniques and considering other variables like frequency, duration etc. The effect of control measures may be created between the blast & receiver to minimize ground vibrations such as pits, trenches, initiation etc. The initiation sequencing away from critical structures and a use of longer delays at low frequency sites needs to be studied and assessed.

The empirical method continues to be most common method for calculation of design parameters. The computer simulation is a promising method and needs to be incorporated in the blast design process. Thus, an integration of empirical, computer modeling and instrumented field trials appears to be the state of the art of blast design.

Cross section of quarry (open cut) bench system showing typical drilling pattern geometry of two rows of holes

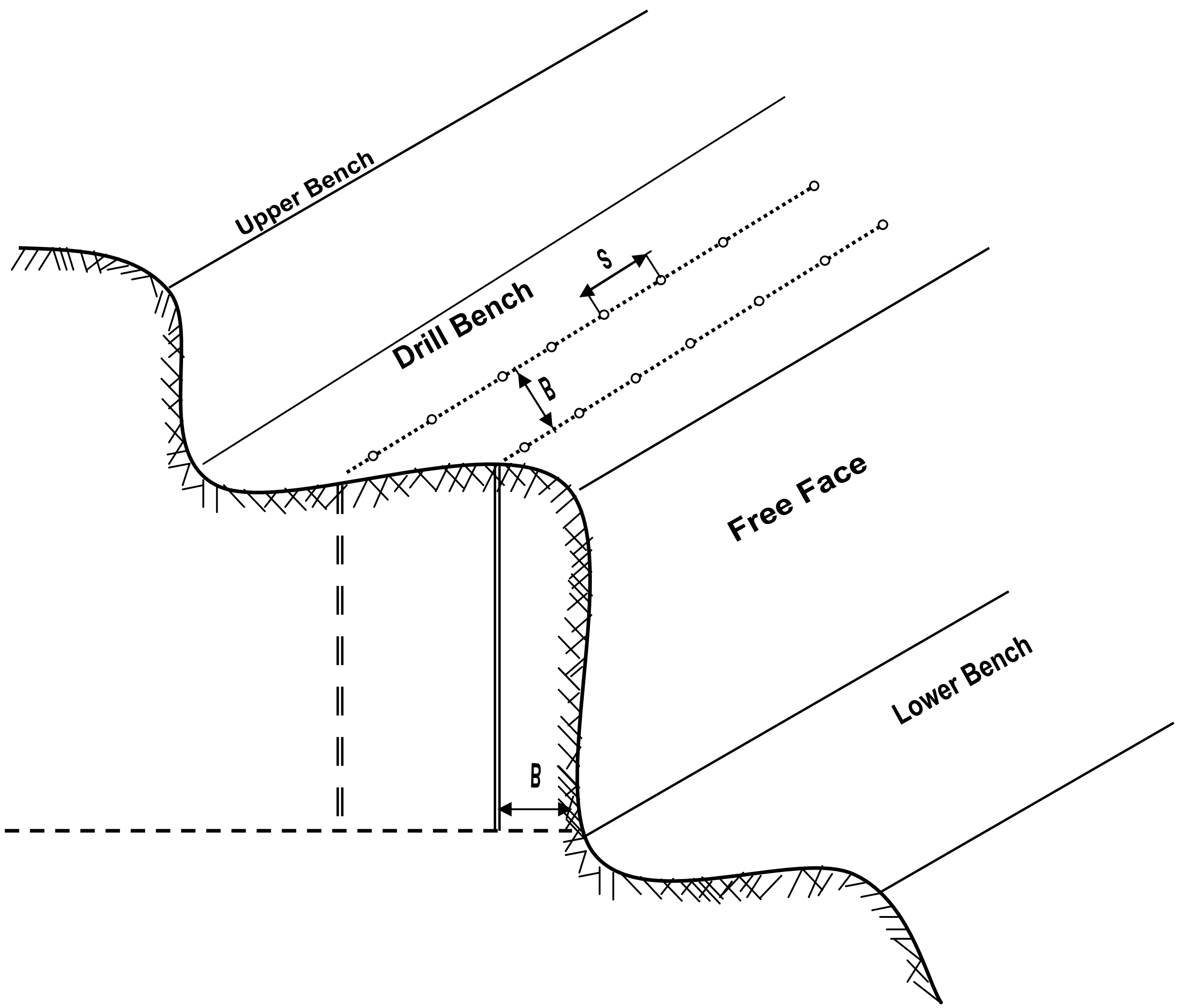
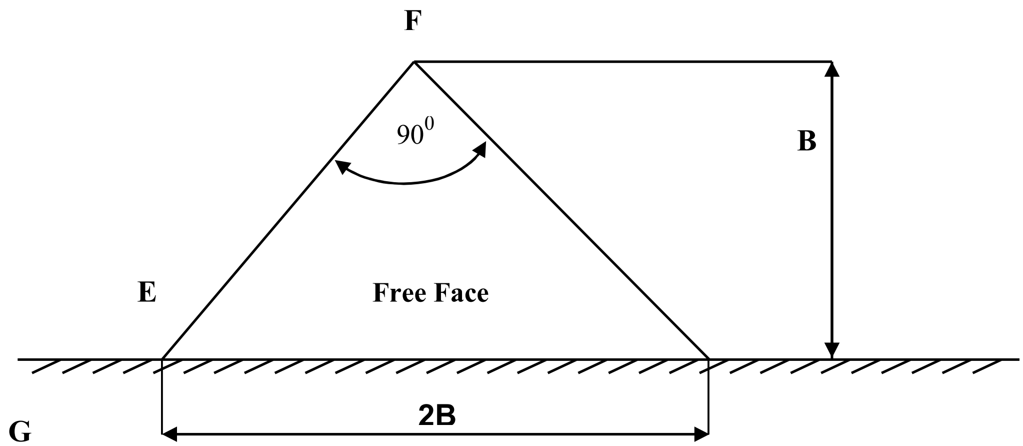
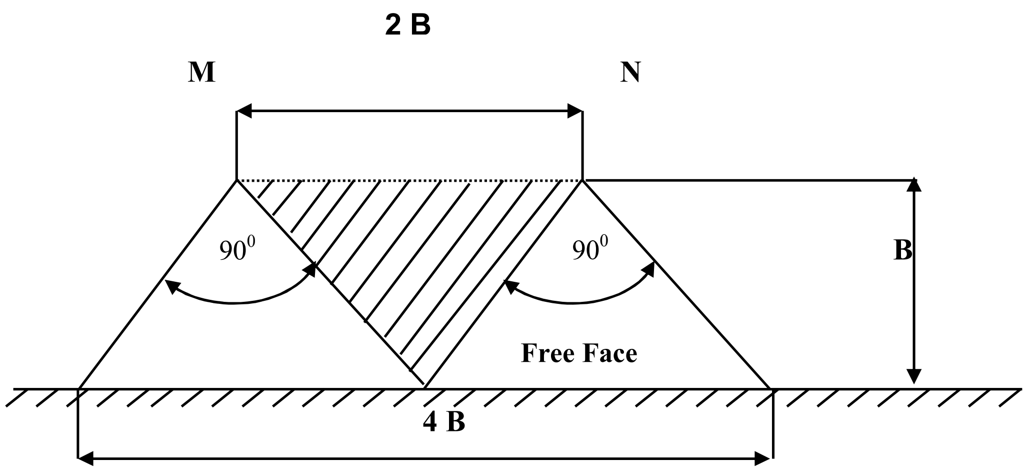


Fig – 5.1: Position of burden & spacing (drilling pattern) in a bench

Volume of rock broken to the free face by a single vertical hole at F.



Breakage pattern for two holes spaced at twice the burden distance



Breakage pattern for two holes spaced at the burden distance

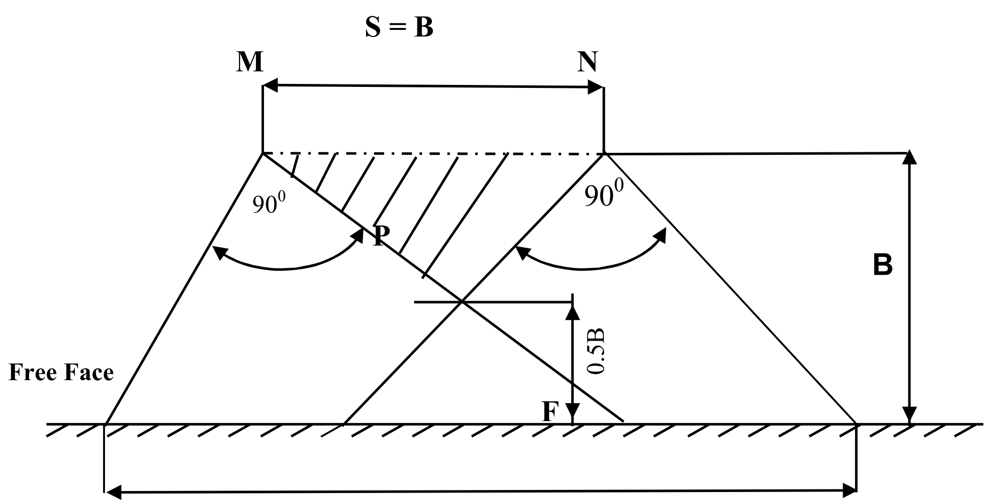


Fig. – 5.2: Breakage pattern with variation of burden

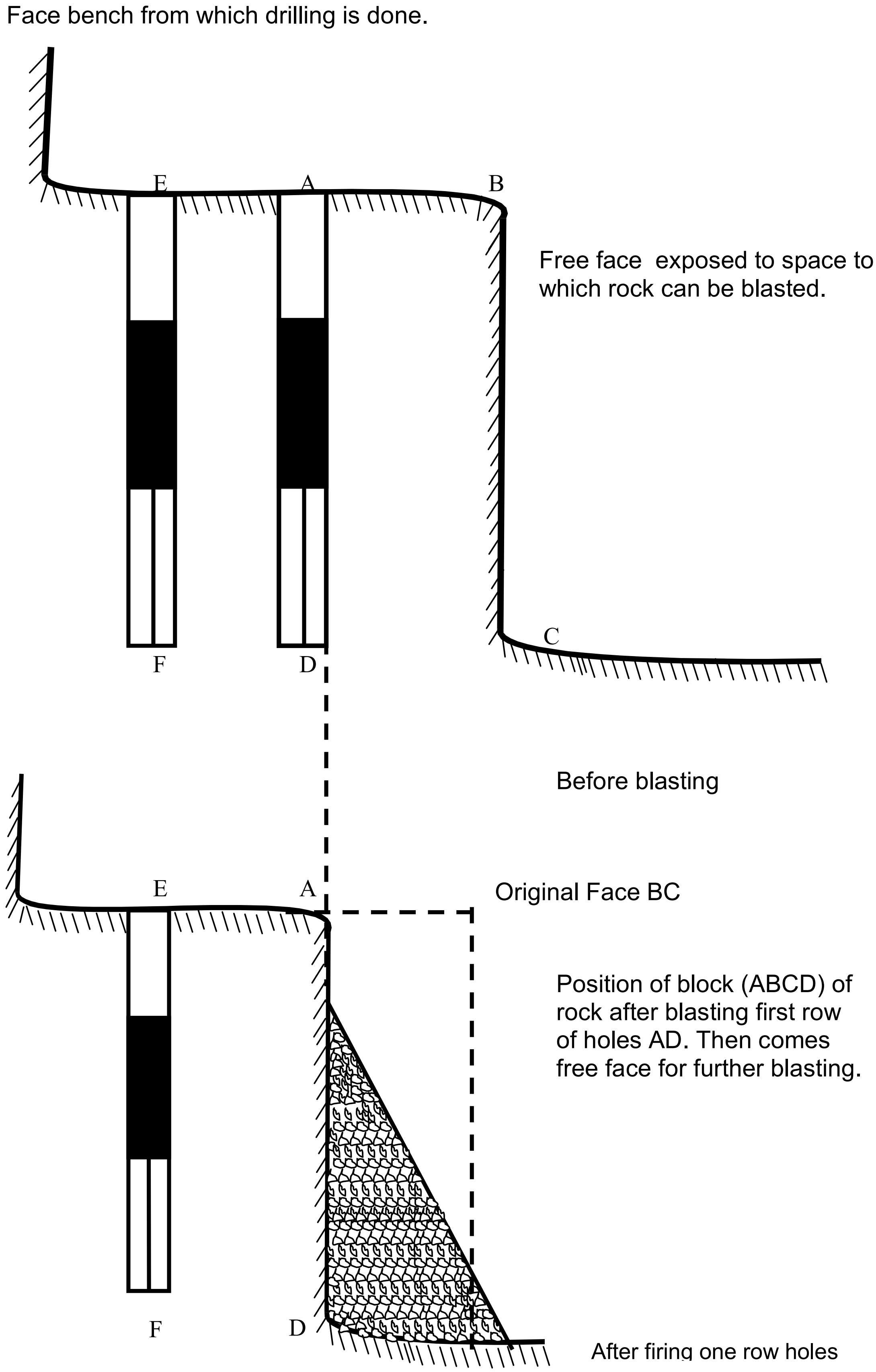


Fig-5.3: Position of holes before and after blasting

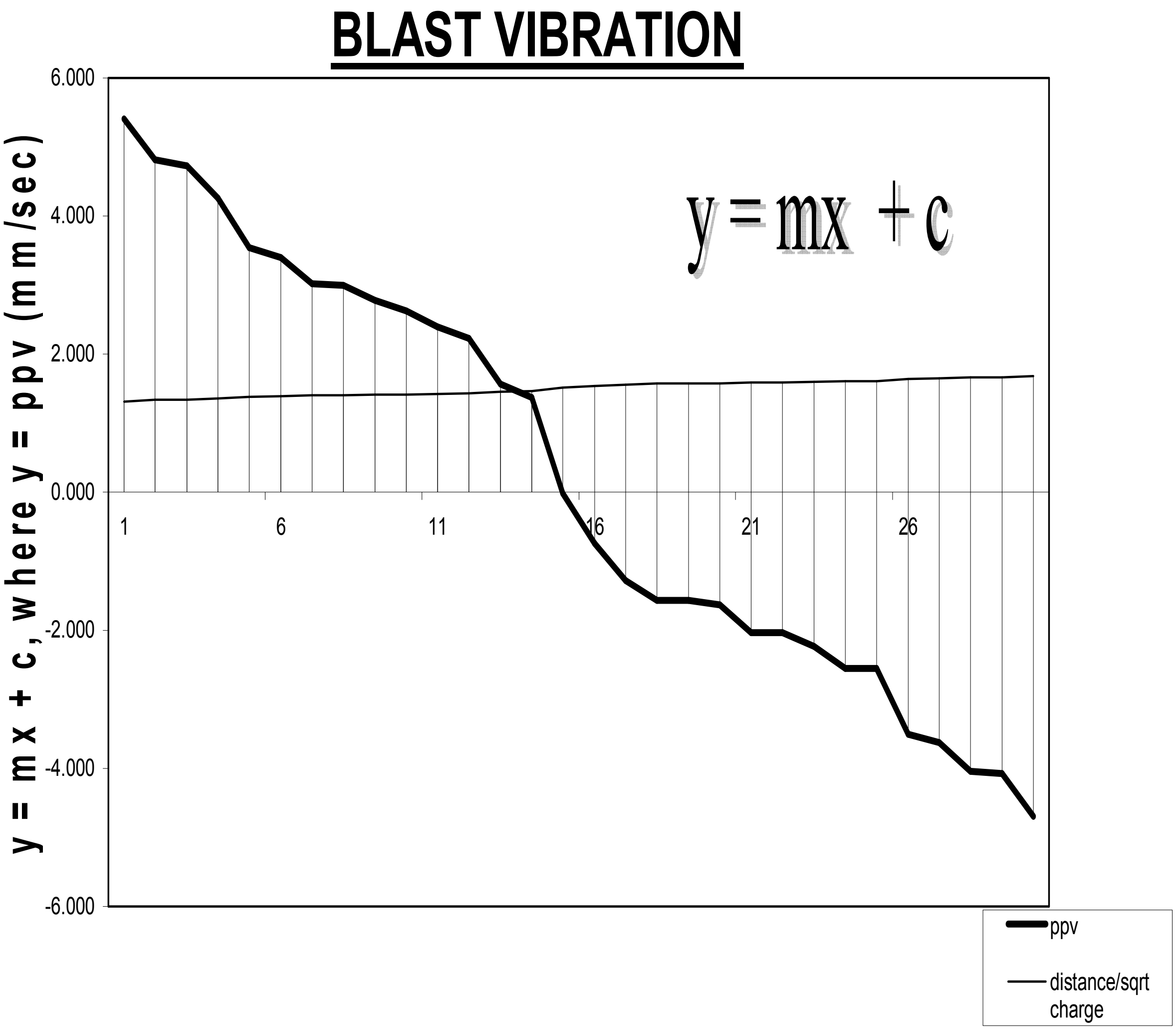


Fig.5.4-Graph showing the scaled distance vrs ppv

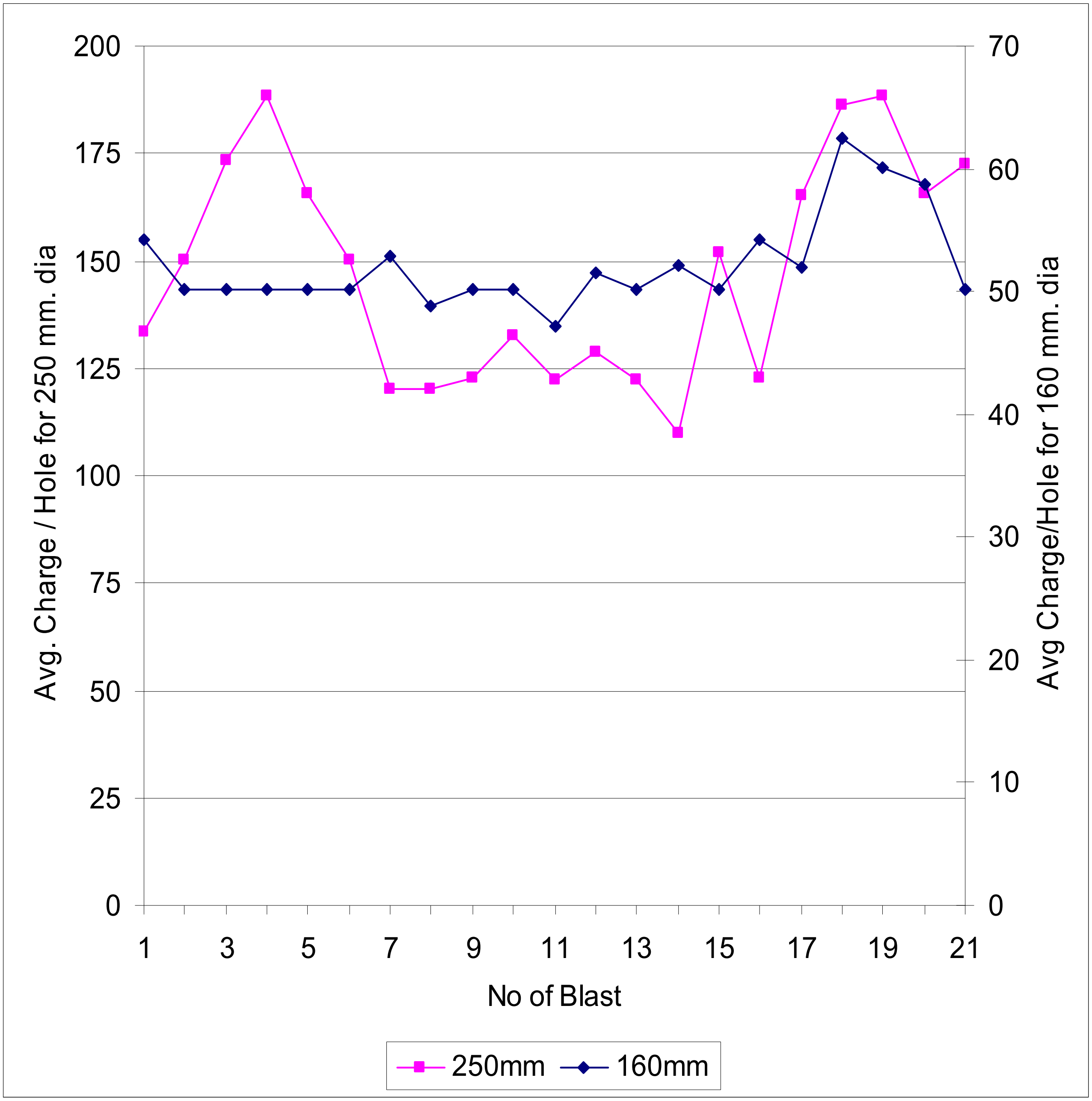


Fig – 5.5: Average Charge /Hole for 250mm and 160mm bore-hole

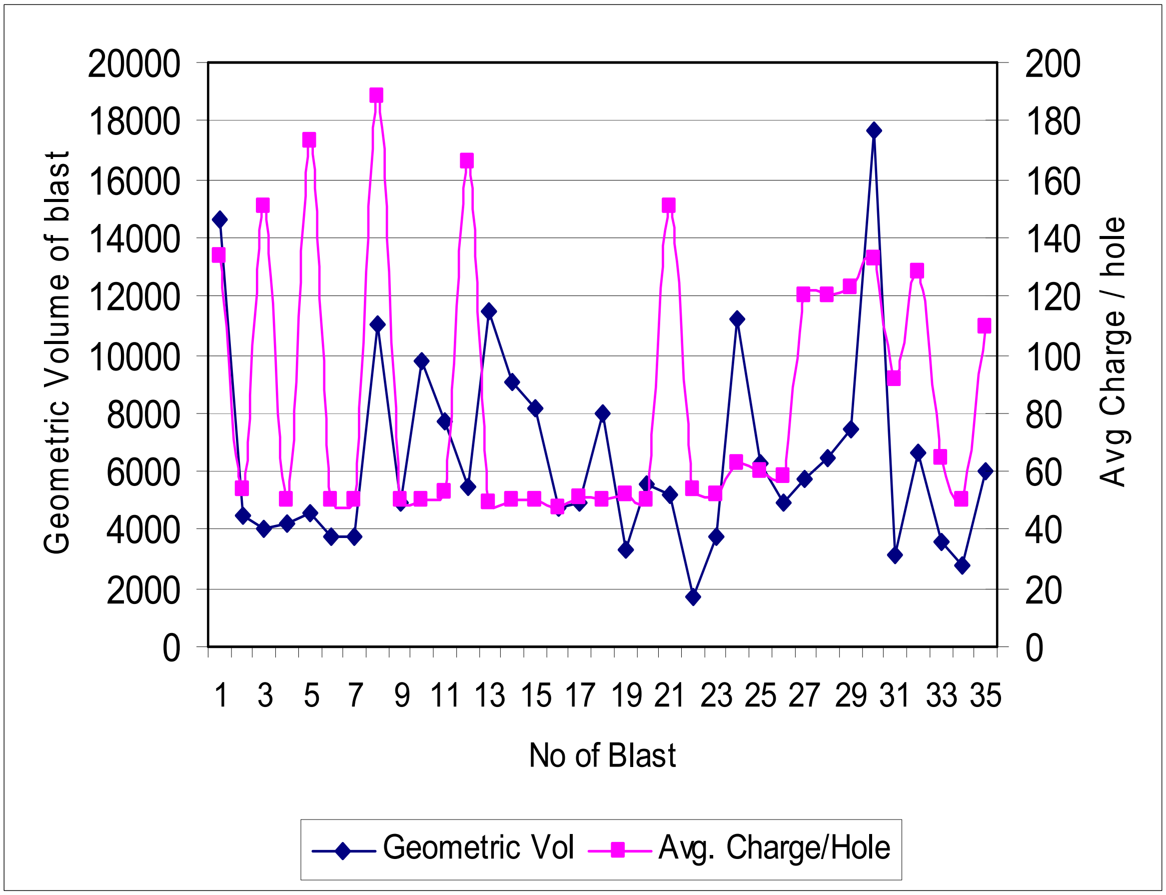


Fig – 5.6: Geometric volume of blast vrs average charge/hole

SHOVEL DUMPER COMBINATION MODEL

6.1 Introduction

As the winds of liberalization blew all over the world, a need aroused to restructure the mining industry. It was felt more as practical necessity than as doctrinaire. In last few years, new developments are witnessed in the Indian mining industry and particularly in the field of mining equipments as opencast mining activities increase at a rapid rate. The modern surface mining system grows so complex with multiple operations such as production, process control, communication, combination of various HEMM (Heavy Earth Moving Machineries) etc. that it creates numerous problems for their developers. At a planning stage, one gets to realize increased capabilities of the systems with unique combination of hardware and software. The system operates under a number of constraints that arise from limited system resources. In view of the capital intensive and complex nature of modern mining systems, the design and operation of these systems require modeling and analysis in order to select the optimal design alternative and operational policy. It is well understood that the flaws in a modeling process can substantially contribute to the development time and cost. The operational efficiency may be affected as well. Therefore, special attention should be paid to the correctness of the models that are used at planning levels.

In opencast mines, once a new face is exposed after blasting, the shovels are required for excavating the minerals and are often kept at more or less fixed locations. Dumpers too are required to move to the shovels in order to collect their loads and transfer them to the dumping stations. The Shovel dumper combination system is a widely used load-haul-technique in surface mining because of the flexibility of its operation and versatility of its application. Number of studies are conducted on the subject with applications of various OR techniques such as queueing theory, markov chains and linear programming models to work out optimum number of shovels and dumpers in view of achieving a targeted amount of production. The advances in computer technology and the associated demand for mineral resources compel the Mining Engineer to apply various OR models in mining industry. With it, optimum utilization of the equipments is achieved with an objective of maintaining the cost of production at minimum. Fortunately, many problems of mining environment can be resolved with an application of OR techniques. On a given project involving overburden removal, the most economical number of dumpers is the number that gives the lowest cost per cubic meter of waste, considering the combined cost of the shovel and the dumpers. Additionally, effective and efficient haulage systems can only be developed through a detailed consideration of inventory, waiting line, allocation and replacement processes in a system. Otherwise, overloads and production bottlenecks may result at unexpected points in the mining system. Large numbers of heavy earth-moving equipment too are analyzed so that the required production targets would be achieved. Because investment is involved in mining industry, no mine design engineer would afford to allow the equipment to remain idle unnecessarily. For this reason, it is unavoidable that the proper setting / adjustment/sequencing of equipments have to be considered while working on the selection process.

The shovel-dumper combination model seeks to combine two basic equipments in view of achieving efficiency and productivity. If the production rate of a shovel remains constant, and if the loads and cycle times of the dumpers to remain constant, it would be fairly simple to determine the most economical number of dumpers to employ on a specified project. However, it is well known that dumper cycle times do not remain constant even though the haul road profiles and the number of dumpers operating remain constant. There may be times when several dumpers are waiting in a queue to be loaded; then later, for no apparent reason, the shovel may have to wait for a dumper. This results in a loss in production. If more dumpers are employed, there will be an excess of dumper capacity. But often there is not enough benefit to compensate for an increased cost of extra dumper(s). This loss of productivity occurs because of a mismatch which causes bunching of the hauling units. The improvement of mine productivity, by minimizing both the idle time of equipment and handling cost per cubic meter of material, makes if necessary to analyze complex problems associated with the mining operations. In this case, the theory of queues can be applied to the situation that involves shovels and dumpers. It analyzes statistically the cost of shovel and dumpers when using various numbers of dumpers. From this, an optimum number of equipments can be obtained.

The present study deals with an analysis of the theory-of-queues taking into consideration of the field data to determine the most economical number of dumpers that may match a shovel for removal of overburden. It is in the context of a large coal mine located in Mahanadi Coalfields Limited.

6.2 Dumper Requirements

The number of dumpers required for the shovel-dumper system is determined by comparing various fleet production capabilities and costs with production requirements and selecting the lowest cost fleet with adequate production capability. In other words, dumper fleet requirements are affected by many factors; mine plan, haulage roads, mine production requirements, loading equipment, equipment performance and cycle time, operating methods and practices, matching of loading equipment and dumpers, and equipment availability and utilization. The suitability of a loading shovel to a hauling dumper and the selection process of a dumper fleet consist of the following points in table-6.1 (Kesimal, 1998)

Table: 6.1

Selection process of dumper fleet and the suitability of the system

Selection Process of Dumper Fleet		The Suitability of Loading Shovel/Hauling Dumper	
*	The material characteristics such as density swell factor, size of fragmentation, etc. as well as the climatic conditions like altitude and rainfall should be considered.	*	The maximum capacity of a dumper depends upon the mine conditions. This differs from mine to mine, but generally, it is about eight times the shovel bucket capacity.
*	The capital and operating costs should be the lowest	*	The minimum capacity of the dumper should be approximately four times larger than the shovel bucket capacity
*	Reasonably high availability ratios		
*	The dumper payload capacity should match the capacity of the shovel		

*	The physical features of the dumper such as ruggedness, horse-power, gradeability, etc. should suit the job conditions.		
*	Consideration of the haul road characteristics (length, gradient, surface and type)		

(Source: Kesimal, 1998)

6.3 Matching dumpers and loading equipments

An efficient mining operation can be defined as moving of the maximum amount of overburden in shortest period of time at the lowest possible cost. Conversely, the primary cause of inefficiency may be the equipment mismatch and the bunching of dumpers at the loading point. The match factor is generally more applicable to a discontinuous mining system (shovels-dumpers) and defined as:

$$\text{Match factor} = \frac{numberofhaulersxloadercycletime}{numberofloadersxhaulercycletime}$$

The perfect match point from the theoretical standpoint is 100 percent, the dumper-shovel fleet efficiency, which occurs when the match factor equals to 1. If a fewer dumpers are used, there will be an excess of loader capacity and the loader will have unnecessarily high idle times. If more dumpers are employed, then there will be an excess of dumper capacity, which may cause shutdown with one or more dumpers. An unutilized loader/dumper is due to what is called a mismatch. On the other hand, irregular arrival of dumpers at the loading point is known as bunching. It results in reduction of the operating efficiency and higher idle time for the dumper fleet.

Some of the factors that affect performance are different capacities of dumpers, poor fragmentation, rain, poor visibility, etc.

6.4 Queueing Theory Approach

The subject of queueing theory had its origin in the pioneering work contributed by Agner Krarup Erlang. He was an engineer at the Copenhagen Telephone Exchange around the beginning of 20th century. He worked on the application of probability theory to telephone traffic problems. It soon drew attention of many other probability theorists and had remained a popular field of research almost throughout eighty years since then. There may be many situations in real life where queueing theory can be applied. They are like broken-down machines waiting for repair at a repair shop, or such dangerous queues as the one formed by planes circling above an airport waiting to land, etc.

Notation and symbols

Kendall introduced a set of notations which have become standard in the literature of waiting line problems. A general queueing system is denoted by (a/b/c): (d/e) where

a = Probability distribution of the inter-arrival time.
b = Probability distribution of service time.
c = Number of servers in the system.
d = Maximum number of customers allowed in the system
e = Queue discipline.

In addition, the size of the population as mentioned in the previous section is important for certain types of queueing problems although they are not explicitly mentioned in Kendall's notation.

Traditionally, exponential distribution in waiting line problems is denoted by M. Thus a system (M/M/1): (∞/FIFO) indicates a waiting line situation when the inter-arrival times and service times are exponentially distributed having one server in the system with the first-in-first-out discipline, when the number of customers allowed in the system can be infinite. 'M' stands for Markovian or the negative exponential distribution in queueing literature.

Sometimes, the queueing system is denoted by a triad $\bullet/\bullet\ /\bullet$, in which the first two members shall be letters that stand for the forms of the input and the service time distributions respectively, and the third member is a number specifying the number of servers.

In the mining scenario, the queueing model can be defined as (M/M/1): (FCFS/∞/M). In this model, the first M means that the dumpers arrival rate that follows the Poisson's distribution. The second "M" means that the random arrival of dumpers and the server (shovels) service rate which is exponentially distributed. Here "1" means that the system has only one server (shovel). The "FCFS" means that service discipline is the "first-come-first-serve" and "∞" means the capacity of the system is infinite. And last "M" means that the number of potential vehicles in the system is not more than M.

In order to apply this concept, we have to consider the peculiar characteristics of mining activities related to shovel dumper combination system to fit this situation.

6.5 Formulations of Queuing theory model:

It is viewed for a particular position on a particular bench, the determination of optional number of dumpers, in the context of shovel dumper combination system, are calculated by applying the queueing theory techniques which is discussed below:

Let

M	=	total number of dumpers in calling population.
Ts	=	shovel loading time in minutes per dumper
Ta	=	dumper travel time outside the systems in minutes per cycle.
Pn	=	Probability that there are 'n' dumpers in the system
Lq	=	Average number of non-productive dumpers waiting in the queue to be served by the shovel in minutes per cycle.
L	=	average number of dumpers waiting in the system per cycle i.e. the sum of dumpers waiting in the queue and the one being served.
Wq	=	average waiting time of dumpers in the queue per cycle.
W	=	average waiting time of dumpers in system per cycle.
M-Lq	=	average number of productive dumpers per cycle.
Qij	=	Production of one cum of OB from j^{th} position of the i^{th} bench by M dumpers in population with one shovel serving.
CUM	=	average load carried by one dumper in cum per cycle.

DOC = dumper operating cost in Rs. / hour

SOC = shovel operating cost in Rs. / hour.

CQij = cost per cum of production from the j^{th} position of i^{th} bench with M dumpers in population and one shovel serving.

λ = mean frequency of arrival per dumper.

Pij = optimal production in cum

Cij = queueing co-efficient

cum = cubic meter

A) Derivation of Qij

Now,

Load carried by a dumper per cycle = CUM

Number of productive dumpers per cycle = (M – Lq)

Cycle time in minutes = $(T_s+T_a+W_q)$

Production per cycle = (M – Lq) * CUM

Production per minute (Qij) = (M – Lq) * CUM/$(T_s+T_a+W_q)$

B) Derivation of CQij

Total material handling cost/cum for the j^{th} position of i^{th} bench = Dumper material handling cost per cum for j^{th} position of i^{th} bench + shovel material handling cost per cum for the j^{th} position of its bench.

i) Shovel Operating Cost/cum

Cost of shovel operating per minute = SOC/60

Cost of shovel operating per cycle = SOC* $(T_s+T_a+W_q)$/60

Production per cycle = (M – Lq)* CUM

Thus, shovel operating cost per cum of production

$$= \frac{SOC*(Ts+Ta+Wq)}{(60*(M-Lq)*CUM)}$$

ii) Dumper Material handling cost/cum from the jth position of ith bench.

Cost of dumper running per minute = DOC/60

Cost of dumper running per cycle = Ta * DOC/60

Production per cycle = (M-Lq)* CUM

Hence,

Cost of running dumper per cum of production

= [(Ta * DOC)/60*(M-Lq)* CUM]

The optimal production can be obtained from queueing model.

C) Number of shovels required

Service time of shovel = Ts min.

Availability of shovel per year = TLS hours

Load carried by dumpers served for the Ts minutes = CUM

Total material handling time for the j^{th} position of the i^{th} bench

= Pij * Cij = X min.

Cycle time = (Ts + Ta + Wq) min.

Production per min = $\frac{(M - Lq) * CUM}{(Ts + Ta + Wq)}$

Production in Xmin from j^{th} position of i^{th} bench

= $\frac{(M - Lq) * CUM}{(Ts + Tq + Wq)}$ * X

This is the maximum material handling capacity of shovel.

So, the number of shovels required for the j^{th} position of the i^{th} bench.

= $\frac{M - Lq) * CUM * X}{(Ts + Ta + Wq)}$ /(TLS) * CUM

= $\frac{(M - Lq) * CUM * X * Ts}{(Ts + Ta + Wq) * CUM * TLS}$

= $\frac{(M - Lq) * X * Ts}{(Ts + Ta + Wq) * TLS}$

6.6 Case studies and analysis

When a case study is carried out at any mine and an analysis is undertaken quite a large number of observations need to be considered. Each activity such as working, breakdown (B/d), idle, maintenance etc. are recorded in different registers and files maintained at mine. Since there is no regular productivity analysis, each observation should be evaluated one-by-one considering the relevant one. The table No.6.2 to No. 6.5 below present the shovel and dumper actual time distributions obtained from last year records, (i.e. from Jan., 2005 to Dec., 2005) respectively.

Table-6.2
Shovel-wise different hours of activities

Liebherr SHOVEL 1

Month	Working Hours	B/D Hours	Idle Hours	Maint. Hours	Shift. Hours
Jan-05	328	196	194	26	744
Feb-05	83	598	65	7	744
Mar-05	219	322	118	13	672
Apr-05	359	169	205	11	744
May-05	243	329	132	16	720
Jun-05	362	87	240	55	744
Jul-05	331	57.5	194.5	37	660
Aug-05	306	164.5	214.5	32	660
Sep-05	375	192	151	26	744
Oct-05	398	126	148	48	720
Nov-05	298	365	107	34	744
Dec-05	209	323	168	20	720
	3511	**2929**	**1937**	**325**	**8616**

Liebherr SHOVEL 2

Month	Working Hours	B/D Hours	Idle Hours	Maint. Hours	Shift. Hours
Jan-05	401	73.5	236.5	33	744
Feb-05	375	118.5	205.5	45	744
Mar-05	393	40	210	29	672
Apr-05	314	227.5	190.5	12	744
May-05	386	77	228	29	720
Jun-05	193	404	119	28	744
Jul-05	350	18	310	42	660
Aug-05	269	178	265	32	660
Sep-05	67	620.5	52.5	4	744
Oct-05	322	228	128.5	41.5	720
Nov-05	442	43	199	60	744
Dec-05	365	116	202	37	720
	3877	**2144**	**2346.5**	**392.5**	**8616**

Liebherr SHOVEL 3

Month	Working Hours	B/D Hours	Idle Hours	Maint. Hours	Shift. Hours
Jan-05	372	123	224	25	744
Feb-05	149.5	17	79.5	18	744
Mar-05	363	96	183	30	672
Apr-05	381	147	198	18	744
May-05	374	54	260	32	720
Jun-05	342	143.5	207.5	51	744
Jul-05	268	101	317	34	660
Aug-05	369	17	316	42	660
Sep-05	228	25	478	13	744
Oct-05	122	58	56	14	720
Nov-05	390	92	207	55	744
Dec-05	354	149.5	178.5	38	720
	3712.5	**1023**	**2704.5**	**370**	**8616**

(Source: logbook of equipments)

Table-6.3
Dumper-wise different hours of activities

DUMPER NO 1

Month	Working Hours	B/D Hours	Idle Hours	Maint. Hours	Shift. Hours
Jan-05	376	50	280	38	744
Feb-05	401	22	288	33	744
Mar-05	269	159	218	26	672
Apr-05	353	38	320	33	744
May-05	290	26	370	34	720
Jun-05	12	698	29	5	744
Jul-05	248	66	377	29	660
Aug-05	37	0	32	3	72
Sep-05		0	0	0	0
Oct-05	245	12	289	30	576
Nov-05	346	11	351	36	744
Dec-05	371	5	312	32	720
Total	**2948**	**1087**	**2866**	**299**	**7140**

DUMPER NO 2

Month	Working Hours	B/D Hours	Idle Hours	Maint. Hours	Shift. Hours
Jan-05	424	2	274	44	744
Feb-05	389	3	319	33	744
Mar-05	305	102	235	30	672
Apr-05	312	74	327	31	744
May-05	162	359	178	21	720
Jun-05	192	300	230	22	744
Jul-05	316	0	370	34	660
Aug-05	29	0	40	3	72
Sep-05	0	0	0	0	0
Oct-05	299	0	245	32	676
Nov-05	347	18	346	33	744
Dec-05	272	166	266	27	720
Total	**3047**	**1024**	**2830**	**310**	**7240**

DUMPER NO 3

Month	Working Hours	B/D Hours	Idle Hours	Maint. Hours	Shift. Hours
Jan-05	386	36	274	48	744
Feb-05	425	3	283	33	744
Mar-05	358	11	272	31	672
Apr-05	321	85	304	34	744
May-05	318	9	355	38	720
Jun-05	153	32	338	35	660
Jul-05	6	698	15	1	660
Aug-05	287	55	368	34	660
Sep-05	335	4	366	39	744
Oct-05	360	0	327	33	720
Nov-05	384	0	323	37	744
Dec-05	0	720	0	0	720
Total	**3333**	**1653**	**3225**	**363**	**8532**

DUMPER NO 4

Month	Working Hours	B/D Hours	Idle Hours	Maint. Hours	Shift. Hours
Jan-05	412	54	240	38	744
Feb-05	424	0	287	33	744
Mar-05	365	0	276	31	672
Apr-05	349	12	350	33	744
May-05	295	48	344	33	720
Jun-05	339	32	338	35	744
Jul-05	225	126	336	33	660
Aug-05	264	52	399	29	660
Sep-05	304	30	378	32	744
Oct-05	128	461	114	17	720
Nov-05	258	203	253	30	744
Dec-05	237	235	227	21	720
Total	**3600**	**1253**	**3542**	**365**	**8616**

DUMPER NO 5

Month	Working Hours	B/D Hours	Idle Hours	Maint. Hours	Shift. Hours
Jan-05	360	123	223	38	744
Feb-05	406	10	293	35	744
Mar-05	359	0	283	30	672
Apr-05	334	82	297	31	744
May-05	266	26	396	32	720
Jun-05	3	727	13	1	744
Jul-05	271	71	347	31	660
Aug-05	0	744	0	0	744
Sep-05	0	744	0	0	744
Oct-05	285	69	332	34	720
Nov-05	272	135	305	32	744
Dec-05	270	127	294	29	720
Total	**2826**	**2858**	**2783**	**293**	**8700**

DUMPER NO 6

Month	Working Hours	B/D Hours	Idle Hours	Maint. Hours	Shift. Hours
Jan-05	69	600	69	6	744
Jul-97	364	65	285	30	744
Mar-05	339	10	286	37	672
Apr-05	284	137	295	28	744
May-05	214	231	246	29	720
Jun-05	347	2	356	39	744
Jul-05	315	4	364	37	660
Aug-05	282	68	361	33	660
Sep-05	344	27	338	35	744
Oct-05	335	40	307	38	720
Nov-05	385	8	315	36	744
Dec-05	366	2	318	34	720
Total	**3644**	**1194**	**3540**	**382**	**8616**

DUMPER NO 7

Month	Working Hours	B/D H	Idle Hours	Maint. Hours	Shift. Hours
Jan-05	-	-	-	-	-
Feb-05	83	0	78	7	168
Mar-05	362	2	278	30	672
Apr-05	374	17	320	33	744
May-05	320	52	315	33	720
Jun-05	309	36	360	39	744
Jul-05	324	1	363	32	660
Aug-05	323	3	379	39	660
Sep-05	346	1	362	35	744
Oct-05	338	3	344	35	720
Nov-05	352	14	342	36	744
Dec-05	345	89	257	29	720
Total	**3476**	**218**	**3398**	**348**	**3964**

DUMPER NO 8

Month	Working Hours	B/D H	Idle Hours	Maint.Hours	Shift.Hours
Jan-05	-	-	-	-	-
Feb-05	21	0	26	1	48
Mar-05	319	58	262	33	672
Apr-05	329	54	332	29	744
May-05	323	3	357	37	720
Jun-05	351	1	359	33	744
Jul-05	309	0	381	30	660
Aug-05	283	15	412	34	660
Sep-05	379	0	328	37	744
Oct-05	332	10	341	37	720
Nov-05	74	584	76	10	744
Dec-05	24	600	90	6	720
Total	**2744**	**1325**	**2964**	**287**	**7176**

DUMPER NO 9

Month	Working Hours	B/D Hours	Idle Hours	Maint.Hours	Shift.Hours
Jan-05	-	-	-	-	-
Feb-05	-	-	-	-	-
Mar-05	340	5	273	30	648
Apr-05	361	3	348	32	744
May-05	310	27	347	36	720
Jun-05	338	3	369	34	744
Jul-05	292	25	372	31	660
Aug-05	282	95	339	28	660
Sep-05	371	0	334	39	744
Oct-05	365	0	322	33	720
Nov-05	335	86	291	32	744
Dec-05	350	29	298	43	720
Total	**3344**	**273**	**3293**	**338**	**7104**

DUMPER NO 10

Month	Working Hours	B/D Hours	Idle H	Maint.Hours	Shift.Hours
Jan-05	-	-	-	-	-
Feb-05	-	-	-	-	-
Mar-05	208	9	218	21	456
Apr-05	368	5	337	34	744
May-05	313	34	342	31	720
Jun-05	309	2	388	45	744
Jul-05	314	1	372	33	660
Aug-05	329	4	378	33	660
Sep-05	327	43	337	37	744
Oct-05	108	451	151	10	720
Nov-05	301	102	311	30	744
Dec-05	357	4	328	31	720
Total	**2934**	**655**	**3162**	**305**	**6912**

(Source: Log book of equipments)

Table – 6.4
Abstract of shovel time distribution

Sl. No.	Model	Working hrs.	B/d hrs.	Idle hrs	Maint. hrs	Shift hrs
1	Liebherr-974B	3511	2929	1937	325	8616
2	Liebherr-974B	3877	2144	2346.5	392.5	8616
3	Liebherr-974B	3712.5	1023	2704.5	370	8616

(Source: Log book of equipments)

Table – 6.5
Abstract of dumper time distribution

Sl. No.	Model	Working hrs.	B/d hrs.	Idle hrs	Maint. hrs	Shift hrs
1	BEML-210M	2948	1087	2866	299	7140
2	BEML-210M	3047	1024	2830	310	7240
3	BEML-210M	3333	1653	3225	363	8532
4	BEML-210M	3600	1253	3542	365	8616
5	BEML-210M	2826	2858	2783	293	8700
6	BEML-210M	36.44	1194	3540	382	8616
7	BEML-210M	3476	218	3398	348	3964
8	BEML-210M	2744	1305	2964	287	7176
9	BEML-210M	3344	273	3293	338	7104
10	BEML-210M	2934	655	3162	305	6912

(Source: Log book of equipments)

In addition, the results of field observations (over 100), were statistically calculated to find average period of each equipment. Production is dependent on the number of dumpers as well as the number of trips that they make per shift. The trips depend upon the time to complete one cycle of operation (loading, hauling, dumping and return). This cycle time also continually differs as the face advances because of the change in both hauling and return times. The productivity of shovel-dumper combination system requires time studies to find the time of each segment of cycle time. There are some packages which could be used to calculate matched dumper productivity to shovel productivity, the effect of bunching, finding mismatches, predicting future cycle times etc. Kesimal developed the OPSTSCIM- Open Pit Shovel Dumper and Conveyor Simulations (OPMINE, A produce of France, 1993). Caterpillars VEHSIM, 1984, Singhal and Fytas, 1986; GPSS/PC or GPSS/H – Sturgul, 1991, 1992, 1993, 2000 are among other developments in the theory.

Table – 6.6
Time study of shovel cycle (seconds)

Crowd/Hosting	Swing	Unloading	Swing back	Total time	Avg. time
12	5	2	4	23	27.5
12	4	3	6	25	
10	3	4	8	25	
12	8	4	8	32	
11	6	5	7	29	
12	6	5	8	31	
10	3	4	8	25	29.17
12	6	4	7	29	
12	5	6	8	31	
11	7	5	8	31	
12	8	6	6	32	
10	5	5	7	27	
11	4	5	6	26	24.83
10	4	5	6	25	
10	4	5	4	23	
9	6	6	5	26	
12	5	5	5	27	
8	5	5	4	22	
10	4	5	4	23	25.67
10	4	6	5	25	
11	5	5	5	26	
9	5	4	4	22	
13	5	6	6	30	
11	6	5	6	28	

(Source: Field Study data)

Avg. loading cycle/bucket
= (27.5 + 29.17 + 24.83 + 25.67)/4
= 26.79 seconds

Shovel loading time to
One dumper (considering six buckets)
= 26.79 x 6 = 160.76
= 2.68 min.

Table – 6.7
Time study of dumper cycle (Lead = 1.2km)

Hauling	Unloading	Return	Total time (Ta)	Waiting time
2min 50 sec.	1min 30 sec	3 min	7 min 20 sec.	-
3 min.	1min20 sec.	20min40sec	7 min	-
2min 50 sec.	1min 30 sec	2min 50 sec	7 min 10 sec.	-
3 min	1min 40 sec	2min 45 sec	7 min 25 sec	1 min 10 sec
2 min 55 sec	1min 50 sec	3 min	7 min 45 sec	15 sec
3 min	1min 45 sec	2min 55 sec	7 min 40 sec	30 sec
2 min 50 sec	1min 50 sec	3 min	7 min 40 sec	40 sec

(Source: Field Study data)

Total time = 58.17 min

Average cycle time = 58.17/7 = 8.31

Average loading cycle of dumper (on shovel face) = 2.68m

Total average cycle time of dumper (minute) = 8.31 + 2.68 = 10.99

Table – 6.8
Average loading and dumper cycle time

	Shovel cycle time (sec)	Shovel loading time (min)	Dumper cycle time (min)
Average	26.79	2.68	10.99

The production from dumpers and shovels are calculated considering the amount of work for 50 min. in an hour.

$$\text{Dumper production} = \frac{\text{Payload } (m^3) \times 50\text{min/hr}}{\text{Dumper average cycle time (min)}}$$

$$= \frac{17 \times 50}{10.99} = 77.34 m^3/\text{hr.}$$

$$\text{Shovel production} = \frac{\text{Payload } (m^3) \times 50\text{min/hr}}{\text{Average loading time (min)}}$$

$$= \frac{17 \times 50}{2.68} = 317.16 m^3/\text{hr.}$$

The calculation is made to find out the match factor and total system efficiency (Table-6.9). The loading and hauling systems consists of a electric hydraulic shovel with dipper (bucket) capacity 5.1m^3 and a fleet of off-highway dumpers of 50T payload capacity (17m^3). The system is simulated with 50T capacity dumpers by varying the number of dumpers assigned to a single shovel.

Table – 6.9
Results from the study

Measured parameters			Calculated parameters		
Shovel		Dumper	Number of dumper	Match factor	Total efficiency (%)
Bucket cycle time (sec)	Avg. loading time (min)	Avg. cycle time (min)			
			1	.2436	24.36
			2	.4877	48.77
			3	.7315	73.15
			4	.9754	97.54
26.79	2.68	10.99	4.10	1.00	100
			5	1.219	121.9
			6	1.463	146.3
			7	1.707	170.7
			8	1.950	195

It is evident from the above table that the percentage of efficiency of the system equals to 100% when the match factor and number of dumpers equal to 1 and 4.10 respectively. If we Increase the number of dumpers from 1 to 8 it decreases dumper utilization whereas the shovel utilization increases (upto 5 dumpers). The break-even point corresponding to maximum shovel and dumpers utilization is near the perfect match point (i.e .9754 for 4 dumpers). In future, when the haul distance increases, thereby increasing the haul-return cycle time, a new match factor would be calculated again. Necessary corrections can be made for optimal results based on the above calculations and average cycle time.

In the above calculation made for shovel production, it is assumed that dumpers are always available as required by shovels. This is usually not a situation at all the times. Thus, the production of loading and hauling systems will be less than the production rate for the shovel because of some probability of dumpers not being available at all the times when shovel is ready to load.

The ideal rate of production is 317.16m^3/hr. If a shovel needs to wait for dumpers at times, then the rate will be reduced, as indicated below:

$$Q \quad = (1 - Po)$$

The value of Po varies with the number of dumpers employed. It needs to be determined for calculating the actual production of the shovel. The queueing theory is applied here with the probability of there being no dumper in a queue and a shovel waits until a dumper arrives. This can be represented in the cumulative Poisson expression as given below:

$$Po\ (n,x) = \frac{e^{-x}x^{n}/n!}{\sum_{j=o}^{n}(e^{-x}x^{j}/j!)} = \frac{p(n,x)}{P(n,x)}$$

The values can be found out by utilizing Poisson distribution functions. Here x is the number of dumpers needed; n is the number of dumpers in the fleet.

The value of the functions is required to be evaluated from the Poisson equation with the x replaced by its value 4.10 (table-6.10).

This equation can be rearranged as follows:

$$Po\ (n, 4.1) = \frac{p(n,4.10)}{P(n,4.10)}$$

Table – 6.10

Poisson distribution functions & values of Po with number of dumpers

x	*p*(n,4.10)	*P*(n,4.10)	$\frac{p(n,4.10)}{P(n,4.10)}$ = Po
0	0.01657	0.01657	1
1	0.06794	0.08451	0.80393
2	0.13929	0.22380	0.62239
3	0.19036	0.41416	0.45963
4	0.195131	0.60929	0.32026
5	0.16000	0.76929	0.20798
6	0.10934	0.87861	0.12445
7	0.06404	0.94265	0.067936
8	0.03282	0.97547	0.03365
9	0.01495	0.99042	0.01509
10	0.00613	0.99655	0.00615

The unit cost analysis for 5.1cum elect. hydraulic shovel and 50Te dumper is depicted in the table No.6.11 and 6.12 respectively. The probable production in cubic meters per hour and the variation in the cost per cubic meter based on a varying the number of dumpers is shown in table-6.13. The ratio between the sum of the total cost per hour for the shovel and dumpers and output of the shovel in cum per hour is the cost per cum of overburden removal.

Table – 6.11

Unit Cost Analysis for 5.1cum hydraulic shovel for loading

Sl. No.	Particulars	Nos	Unit Rate	Total Amount
A	CAPITAL COST (Lakh Rs.) BE 1000	1	521.47	521.47
	Annual Capacity in Lakh Cubic Meter (OB)			12.80
B	OPERATIONAL COST			
	Salary & Wages			
	Operators	3		
	Helpers	3		
	Maintenance	4		
	Supervisor/Foreman	1		
1	Total salary and wages/day (Rs.)			17.16
	STORES			
	Running maintenance		20% of Annual depreciation	11.59
	Power		0.65 units/cum @ Rs.3.42	37.79
	Lubricants		10% of power	3.78
2	Sub-total (L.Rs.)			49.38
3	Misc incl. WD		2% capital	10.43
4	Interest on loan capital (50%) @ 10.25%		Debit equity 1:1	26.73
5	Depreciation		11.11%	57.94
6	Total cost (1+2+3+4+5) in L.Rs.			161.63
7	OBR cost/cubic meter (Rs.)			12.63

(Source: CMPDIL reports)

Table-6.12

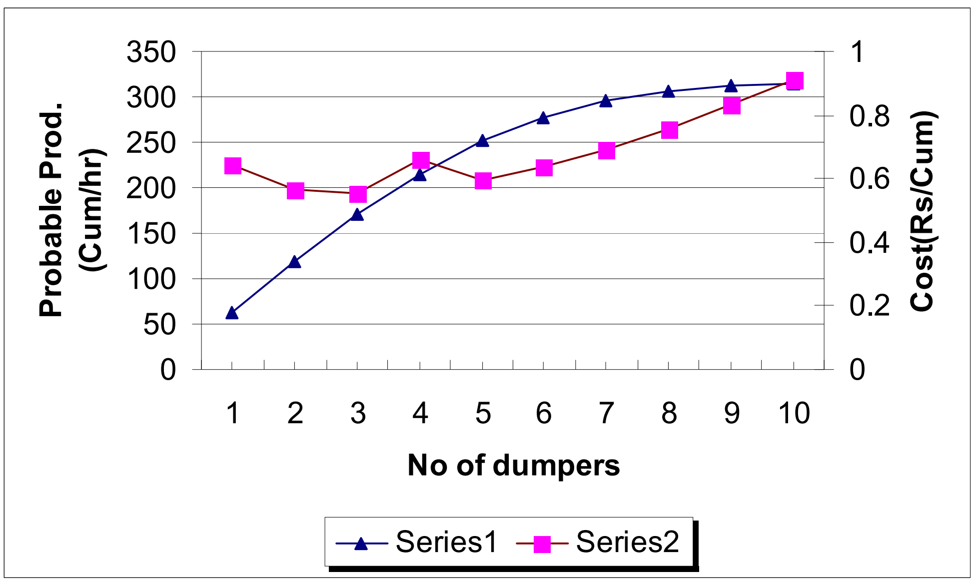
Unit cost analysis for 50T (650HP) dumper for overburden

Parameters	Nos	Unit Rate	Total Amount
CAPITAL COST (Lakh Rs.)	1	141.89	141.89
OPERATIONAL COST			
Salary & Wages			
Operators	3		
Helpers	0		
Maintenance	0.8		
Supervisor/foreman	0.2		
Total salary and wages/day (Rs.)			
Total salary wages/Amount (L.Rs.)			6.24
STORES			
Running maintenance		40% of Annual depreciation	6.31
Diesel		0.06ltr/bhp/hr	37.63
		W.H/Annum : 2800 HSD Rate @ Rs.35.00 per lit	
Lubricants		20% capital	7.53
Sub-Total (L.Rs.)			43.94
Misc. Incl. WD (L.Rs.)		2% capital	2.84
Interest on working capital (50%) L.Rs.		Debit equity 1:1	7.27
Depreciation L.Rs.		11.11%	15.76
Total Cost (1+2+3+4+5) in L.Rs.			76.05
For wages annual earnings of grade-E is considered			
	Overburden		
Lead (km)	Productivity	Cost/cum	
	Lcum	Rs.	
0.5	3.588	21.2	
1	2.77	27.46	
1.5	2.314	32.87	
2	2.044	37.21	
2.5	1.866	40.76	
3	1.686	45.11	
3.5	1.548	49.13	
4	1.44	52.81	
4.5	1.352	56.25	

(Source: CMPDIL reports)

Table – 6.13
Variation in the probable production and cost of overburden removal

Sl. No.	1-Po	Potential production (m3/hr)	Probable production (m^3/hr)	Total cost (Rs./hr)	Cost (Rs./m^3) (3)/(4)
1	0.19607	317.16	62.19	40.09	0.6446
2	0.37761	317.16	119.76	67.55	0.5640
3	0.54037	317.16	171.38	95.01	0.5544
4	0.67974	317.16	215.59	122.47	0.5681
5	0.79202	317.16	252.20	149.93	0.5945
6	0.87555	317.16	277.69	177.39	0.6388
7	0.93206	317.16	295.61	204.85	0.6930
8	0.96636	317.16	306.49	232.31	0.7580
9	0.98491	317.16	312.37	259.77	0.8316
10	0.99385	317.16	315.21	287.23	0.9112



Series 1 indicates probable prod. (cum/hr)
Series 2 indicates cost (Rs/cum)

Fig: 6.1 Graph showing probable production and cost

The match factor 4.10 as calculated earlier is found to be obtained again from intersection of probable production (cum/hr) with cost (Rs. /cum). This indicates that the result is consistent and logical and can be accepted as a optimum measure of dumper requirements with a shovel in the mine under the site specific operating conditions.

6.7 Petri net approach

Petri net modeling technique can be of utmost help to sort out the difficulties and to resolve the complex mining problems.

As discussed earlier in chapter- 3, the Petri nets are multi-focal tools. Petri nets, as graphical and mathematical tools, provide a uniform environment for modeling, formal analysis and design of discrete even systems. Further, the Petri nets as graphical tools provide a powerful communication medium between the user, typically requirements engineer and the customer. Complex requirements specifications, instead of using ambiguous textual descriptions or mathematical notations difficult to understand by the customer, can be represented graphically using the Petri nets. As mathematical tool, the Petri net model can be described by a set of linear algebraic equations, or other mathematical models reflecting behaviour of a system. The ability of the Petri nets to verify the model formally is especially important for real time safety-critical systems such as air-traffic control systems, rail traffic control systems, nuclear reactor control systems, etc. The Petri nets are extensively used currently to model and analysis of communication networks, manufacturing systems, software systems, performance evaluation, etc.

The most mature development involves a use of colored Petri nets. They are demonstrated as a useful language for the design, specifications, simulation, validation and implementation of large software systems. Both deterministic and stochastic performance measures can be evaluated by using a broad class of the Petri net models incorporating in their definitions the deterministic and/or probabilistic time functions. The two basic Petri net based models for handling time are developed:

1) Timed Petri nets
2) Time Petri nets

Ramchandani's "Timed Petri nets" are derived from Petri nets by associating a firing finite duration with each transition of the net. The classical firing rule of PN's is modified first to account for the time it takes to fire a transition and second to express that a transition must fire as soon as it is enabled. These nets and related models have been used mainly for performance evaluation.

Merlin's "Time Petri net" (or TPN's for short) is more general than the timed Petri nets: a timed Petri net can be modeled by using a TPN, but the converse is not true. TPN's are found to be very convenient for expressing most of the temporal constraints while some of these constraints were difficult to express only in terms of firing durations.

Merlin defines "Time Petri nets" with labels: two values of time, two real numbers, a and b, with $a \leq b$, is associated with each transition. Assuming that any transition, e.g. t_i is being continuously enabled after it has been enabled.

- $a(o \leq a)$, is the minimal time that must elapse, starting from the time at which transition t_i is enabled, until this transition can fire, and
- $b(o \leq b \leq \infty)$, denotes the maximum time during which transition t_i can be enabled. Assuming that transition t_i has been enabled at time τ ,then t_i even if it is continuously enabled, can not fire before time. τ +a must fire before or at time τ +b, unless it is disabled before its firing by the firing of another transition.

6.8 Petri net application in mining

The development of the Petri nets, to a large extent motivates a need to model the industrial systems. An application of the Petri nets is established in the fields of electrical & electronics, civil, mechanical and chemical engineering, computer hardware & software developments. Some applications of the Petri nets in medical sciences too find encouraging results. Lately high-level Petri nets are found to be broadening applications to simulate compound systems in decision making, informatics and manufacturing. A complete mining system can be analyzed and sent for reformation from standard components. The Petri nets are applied in cases like long-walling mine, wide conveyor net-work, high–angle conveyor and locomotive based mining. They are used to solve a number of practical problems, such as co-ordination of a cutting and supporting, designing and dispatching of compound transport systems, choice of loading strategy for LHD's, interaction between robot and mining machines. Work on design of mining robots is in progress. Hierarchical decomposition of drills with the help of the Petri net too are tried out for automation in drilling operation at the open cast mining.

6.9 Dynamic Resource modeling of shovel-dumper combination using fusion places

Various resources utilized at an opencast mining project are modeled with tokens that dynamically move from transition to transition in the Petri net based project network. These resources are identified by the "Color" of the tokens. In the classical form of the Petri nets, the modeler is allowed to define only one type of token. This means that in a classical Petri net it will be impossible to depict different objects that the tokens are used to model. In an enhanced form, the modeler is allowed to define more than one type of tokens in a given Petri net by assigning color or type to the token called as colored or typed tokens. An opencast mining project requires resources such as shovels, dumpers, dozers, drills, graders, labor, explosives etc. These resources are utilized on a multitude of work tasks and are dynamically allocated and shared by those work tasks. In the present study, a shovel is allocated with three dumpers in the interest of optimum utilization of both the equipments. In order to realistically schedule the loading and hauling operations, it is essential to model the dynamic allocation and usage of resources. The concept of "fusion places" is applied for modeling the system which does not exist in the classical PNs. A fusion place is a place that has been equated with one or more other places, so that the fused places act as a single place with the same type and number of tokens. The fusion place capability allows places in the Petri Net that exist at different locations in the net work to act functionally as if they are located at the same place. Such places are called fusion places, and the group of such places is called fusion set. Modeling of resources that are shared by a number of work tasks in a mining project is accomplished by using fusion places in conjunction with colored tokens. The fig-6.2 below illustrates a concept of fusion places and its use to model resources in an opencast mining project. In Fig.6.2 (a), shovel loading onto three dumpers are shown. The three dumpers share the common resource depicted by the place "shovel" in the network. The place "shovel" acts as an input place and output place for all the three works tasks. Assuming that there is only one token available in the "shovel" place, the three dumpers can be loaded when the resource is available. Fig.6.2 (b) models this scenario by using fusion places. The fusion set called "shovel" is first defined for the network.

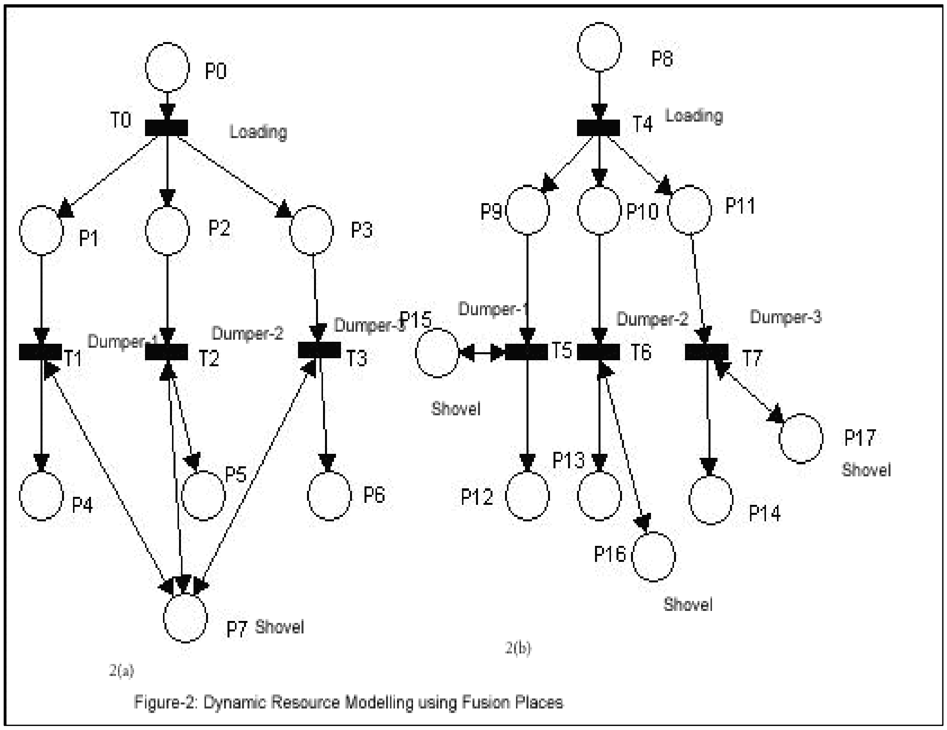


Fig- 6.2: Dynamic Resource modeling using fusion places

Three fusion places representing resource requirements for the three work tasks are then defined. The three work tasks are connected to their respective fusion places as shown in fig.6.2 (b). It is important to note that places belonging to the fusion set "shovel" have the same number and type of tokens. If a mine manager decides to use to two shovels instead of one, appropriate changes can be made in the fusion set. This mechanism simplifies the project network and effectively models resource sharing between various work tasks in an opencast mining project.

6.10 Modeling of shovel dumper combination with Petri nets

In an opencast mine, the overburden (OB) is removed by shovel dumper combination from the working faces to the dump yard. The present study assumes that a shovel is allotted with three dumpers for OB removal from face to dump. Initially, three dumpers are taken to the shovel face and are ready to receive load one by one from the shovel and then move towards the dump yard for unloading the OB. The dumper returns to the shovel face for receiving the load again and the cycle continues. The entire operation is modeled by PNs as shown in Fig.6.3 (a). The loading of OB by the shovel onto the dumper is further decomposed into various sub activities like crowding, lifting, and swinging, unloading and swinging back for resuming the same sequence and is being depicted in Fig.6.3 (b). In this light, the table-6.14 describes different place and transition for the shovel dumper combination system. The table-6.15 too speaks of the sub activities of loading operation of shovel in terms of place and transition of PN model.

Table-6.14

Interpretation of transitions & places of shovel-dumper combination system

	Place Description		Transition Description
P_0	Three dumpers are ready to receive load from shovel	T_0	One dumper moves to the face to receive load
P_1	The dumper is ready to commence positioning	T_1	Positioning of dumper at the face is in progress (2min.)
P_2	Positioning of dumper at shovel face is completed	T_2	Loading starts & is in progress (5 min)
P_3	Loading has been completed	T_3	Dumper moves towards the dump yard (6 min)
P_4	Dumper arrives at the dump yard & is ready to commence unloading	T_4	Unloading at dump yard is in progress (2 min)
P_5	Unloading has been completed & dumper is ready to travel back to the shovel face	T_5	Travelling back of the dumper to the shovel face is in progress (4 min)
P_6	Travelling back of dumper to the face has been completed	T_6	Dumper waits at the shovel face to receive load

Table - 6.15

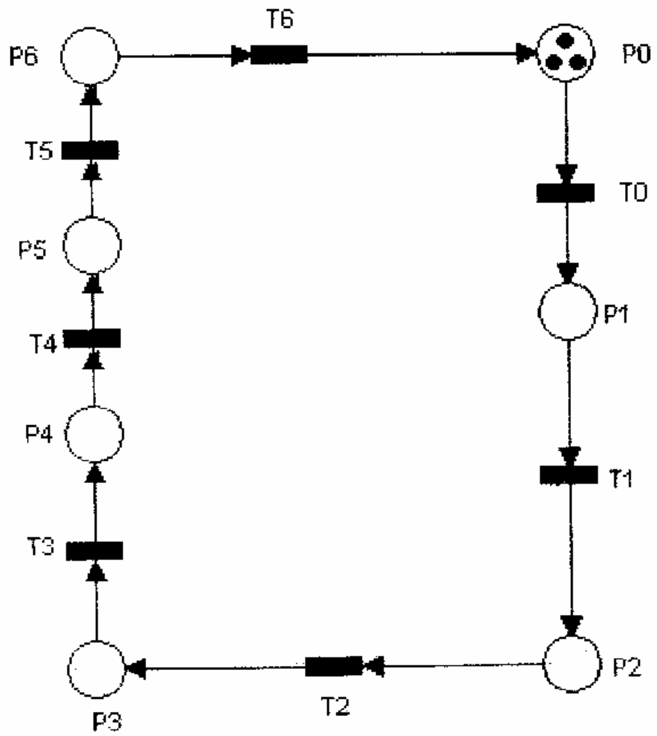
Interpretation of transitions and places of cyclic loading operation of shovel

	Place Description		Transition Description
P_7	Bucket is lowered & crowding starts	T_7	Crowding is in progress
P_8	Crowding has been completed	T_8	Lifting of bucket starts & is in progress
P_9	Lifting is completed	T_9	Swing operation starts & is in progress
P_{10}	Swing operation is completed	T_{10}	Unloading operation starts & is in progress
P_{11}	Unloading is completed	T_{11}	Swing back operation starts & is in progress
P_{12}	Swinging back operation is completed	T_{12}	Lowering of bucket starts and is brought to touch the floor of the bench to resume next cycle of loading operation

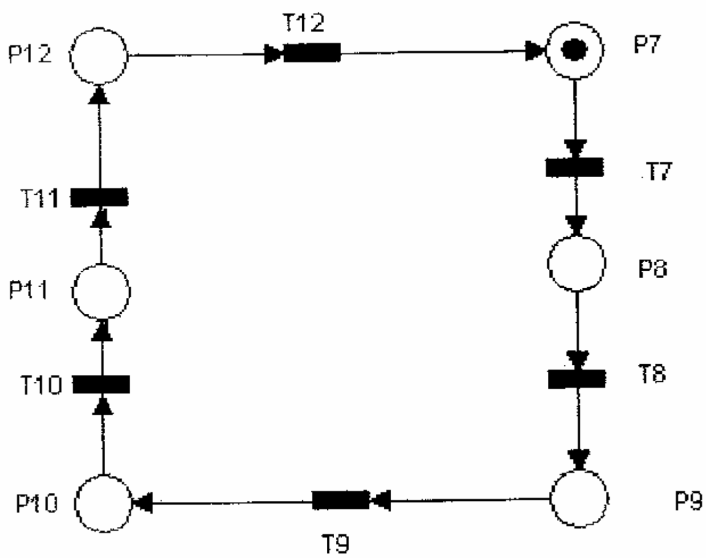
6.11 Conclusion

Optimization of shovel-dumper combination system is still in a development stage even today. This study of the queueing model on allocation of dumpers to shovels indicates that optimal production and productivity can be achieved at minimal cost. From the fig.6.1, it is understood that the number of dumpers matching to a shovel is 4.10 when the match factor is 1. But in practice, the match factor can have an optimum value of 0.9754 with total efficiency 97.54%.

Constructing Petri net models for the opencast mining system requires a great deal of experience and thorough knowledge base in the latest development on PN techniques. The time concept in the PN model is very useful and the application of TPN and SPN is modeling various systems in opencast working. All these can be done with ease and shall be helpful for further study in developing automation in mining. The present study is an attempt to develop a simple PN model that can be simulated in the computer. There is enough scope of research for further study and the development of the mathematical and graphical model of PN with the inclusion of timing constraints in shovel-dumper combination system in an open-cast mine .



(a)PN-based Shovel-Dumper operation model



(b)PN-model of shovel loading cycle

Fig – 6.3: Petri net based shovel dumper combination model

PETRI NET MODELING OF PERT CHART

7.1 Introduction

Survival of mining business unit is at stake in the present era of open ended global markets. It may be due to severe economical conditions. The cost of mining has to be minimized. It requires a review of the present practices and adoption of innovative strategies. In a fast moving mine production scenario, target based planning and scheduling becomes a key word to arrive at a position to achieve the desired output. Scheduling at opencast coal mine project requires: hierarchical decomposition of projects activities as well as risk and uncertainty in the activity time and cost estimates and modeling of dynamically allocated resources. Traditional network techniques that are currently used at the mining industry provide limited modeling versatility and are bit ineffective in modeling a dynamic and stochastic system. The PERT charts have long been used in the planning and scheduling of large projects. A PERT chart is a graphical representation of relationships among various activities which make up a large project. A project consists of a number of activities; some activities must be completed before other activities can start. In addition, time associated with each activity indicates the amount of time it will take. Activities are represented graphically by a node; arcs are used to connect activity nodes to show precedence requirements.

The Petri nets show a similar type of scheduling constraints as the PERT chart does. The PERT chart can be easily converted to the Petri net. Each activity in the PERT chart is represented by a place, while the precedence constraints are represented by transitions.Planning, scheduling and control of the functions, operations and resources of a mining project are among the most challenging tasks faced by the professional mine manager. The outcome of a particular mine facility is a multi-phase task that begins with conceptual planning and continues through detailed design, implementation and regular follow-up phases. The successful completion of the mine facility requires numerous inputs and efforts.

The first requirement for the mining is to acquire land. The procedure to acquire especially forest land is most cumbersome in India. For the purpose, EMP needs to be approved and other formalities are to be completed. In order to succeed in such endeavours, it is employ techniques like planning, scheduling, and co-ordination of important decisions, determining milestones, ensuring flow of resources, and other required inputs. It is important to prepare some form of representation of the designed facility to allow such a planning, scheduling and co-ordination effort to be effective. A common form of representation used at the mining industry is a bar chart or other network based methods such as Critical Path Method (CPM), Program Evaluation and Review Techniques (PERT) and Precedence Diagramming Method (PDM). Short-comings of these methods are numerous and they are reviewed in many research work. A few of them are MacCrimmon et al (1964), Levitt et al (1988), and Prisker et al (1989). The major problem that is highlighted by them is that network techniques are adopted from current practices at the aerospace and manufacturing industries when it is done in disregard to the nature of the mining projects. Three major characteristics of mining projects exert adverse effect by reducing effectiveness of the network based techniques. They include the following:

1. The task of executing a facility can be decomposed into sub-tasks requiring services of specialists like land officer, civil engineer, electrical and mechanical engineers, excavation engineer, surveyor, environment officer, finance officer, personnel officer, construction engineers, general contractors and specialist contractors, etc. These specialists normally belong to different disciplines that are inter-related. A flow of information and the resultant decision making are as such complicated. They render the tasks of planning and scheduling as more complex.
2. Mining projects are operated in a dynamic environment. They are characterized by stochastic phenomenon such as land acquisition constraints, labour productivity and skill fluctuations and variation in geo-mining characteristics.

In this light, the Graphical Evaluation and Review Techniques (GERT) is thought to be reviewed. The reason is its development which extends the modeling capabilities of CPM and PERT. These experiments lead to a use of simulation in the project planning and scheduling. The following paragraph provides a summary of important research conducted in this area. A technique called Probabilistic Network Evaluation (PNET) applies probability theory to reduce number of possible critical paths and evaluates the expected project duration based on representation paths in the network. This research suggests a shift in the scheduling paradigms by adopting the Petri nets. In an opencast mine/project, the Petri nets is, in fact, serve as the backbone of scheduling system

7.2 Planning and scheduling of open pit mine operation

Mining Engineers typically distribute the planning process into three broad segments:

i) Short Range Planning: A day to day planning process is involved in this segment. Its time frame may typically range from one day to one or two months depending on a type of operation and tonnage of mineral to be extracted.

ii) Medium Term Planning: The time frame for this segment may extend from one month up-to two years. It is here that the conceptual pit designs are converted into detailed realistic designs. These designs may be given to short range planners.

iii) Long Term Planning: This segment assesses an overall profitability of a proposed mining operation. Here mines are designed with sufficient detail to provide necessary information as to whether a deposit is of value to consider a more detailed analysis. The time frame may be extended upto the life span of the mine.

In any mining operation, there is considerable overlapping in the above segments. One may merge into another rather than leaving any clear perception of what is universally acceptable. Each mining department nominates a specific time frame to be assigned to each planning process. These time frames of reference may change depending upon the direction of the company and other economic or political influences.

Most recent works in the field of mine planning and production scheduling focus either on computerization of traditional methods or on development of sophisticated mathematical optimization models. Yet, it is clear to all concerned that the optimizers need to be more practical, and the traditional approach needs to be more optimal. The program reviewed in the present research opens a middle ground to strike a balance between these two approaches.

The traditional and optimization approaches that are in current use presume that each one of them finds difficulty in identifying with the other. This may be time especially when we consider the difficulty of understanding the optimization models that one who is not trained in operations research may face. This research seeks to explore a position that the proper role of the operations research renders it as not only a useful and advantageous approach, but also renders it practical, understandable and easy to implement as well.

7.3 Network analysis in project planning

A project in the context of our discussion means one-time operation which has a well defined end-point. As it is indicated in earlier chapters, for an industrial project, the end point would be a date when the plant starts producing things. At the corporate level, the end point would be a date when the company takes over the management of a sick unit. And in the context of marketing, it would be a date on which a new product is sold in the market on commercial basis (Mustafi, 1993). In case of an opencast mine, the end point is a date when a targeted capacity is achieved, that is production of coal is obtained as per a planned schedule.

The Net-work planning consists of arranging the precedence and sequence of project tasks appropriately to provide a road map of execution of the project. It starts with the construction of a diagram, known as 'network diagram'. The diagram reflects the interdependencies and time requirements of the individual tasks that constitute the project. The sequence, interdependencies and time requirements are analyzed further to obtain what may be termed as "planned" duration of the project.

Two most popular forms of this technique that are currently used in many scheduling situations are (1) the Critical Path Methods and (2) Programme Evaluation and Review Techniques. The CPM was developed in 1956 at the E.I. du-Pont Nemours & Co., USA. Its objective was to aid in the scheduling of routine plant overhaul maintenance and construction work. This method differentiates between planning and scheduling. Planning refers to determination of activities that must be accomplished and the order in which such activities should be performed to achieve an objective of a project. Scheduling refers to the introduction of time into a plan thereby creating a time table for various activities to be performed. CPM operates on an assumption that there is a precise known time that each activity in the project will take place (Kothari, 1978).

The PERT was first developed in 1958 for use in defence projects specially in the development of Polaris fleet ballistic missile programme. Thus, these two techniques were contemporary. The PERT allows a manager to calculate the expected total time that the entire project would take to complete. It happens at the stage of formulation and planning of a project. At the same time, it highlights critical or bottleneck activities that are likely to occur in the project. It is in this light that a manager may either allocate more resources or keep a careful watch on such activities as the project progresses (Kothari, 1978). In the PERT chart we usually assume that the time to perform each activity is uncertain and as such three time estimates namely, the optimistic, the pessimistic and the most likely are used. The PERT incorporates statistical analysis in determining the time estimates and enables determination of probabilities concerning the time by which an activity as well as the entire project would be completed. The PERT is a control device. It assists the management in controlling a project once it starts working. It calls attention as a result of constant review to such delays in activities. It might further cause a delay in the completion time of a project.

This network is a graphic representation of various operations of a project. It is composed of activities and events that must be accomplished in order to reach the end objectives of the project. It shows a planning sequence of the accomplishments, their dependence and interrelationships. The basic components of the network are:

The activities can be classified into the following three categories:

1) Predecessor activity:

An activity has to be completed before one or more other activities would start. This activity is known as predecessor activity.

2) Successor activity

An activity may start immediately after one or more of other activities. Such an activity that is completed is known as successor activity.

3) Dummy activity

An activity that does not consume either any resource or time is known as dummy activity. A dummy activity is depicted with a dotted line in the network diagram. A dummy activity in a net-work is added only to represent a given precedence relationships among activities of a project. It is needed when –

a) Two or more parallel activities in a project have same head and tail events.
b) Two or more activities have some (but not all) of their immediate predecessor activities in common.

7.4 Petri net – a graphic tool

The Petri nets are graph-based mathematical models. These models are promising tools for describing and studying manufacturing systems. Moreover, the Petri nets are very much suitable to describe the trade off between flexibility requirements and control policies. They help to improve efficiencies. Usually, this is a major issue to bother open cast mining systems.

The benefits of applying Petri nets to mining system for modeling are as follows:

1. The Petri nets can easily represent concurrent operations.
2. With inhibitor arcs, the Petri nets can represent machine breakdowns, tool failures, and detect occurrences.
3. With mutual exclusion, the Petri nets can model irregular and frequent part-type changes.
4. System modeling can be simplified by dividing the Petri net into several modules. Using these, a study of complex systems may become more convenient.

The Petri nets, being a graphical tool, works on the following twelve criteria. It may be considered, that represent the characteristics of production systems as follows:

1. The ability to represent lead time.
2. The ability to represent a schedule.
3. The ability to represent a logical relationship and parallel asynchronous process.
4. The ability to represent an overlapping and/or a waiting activity.
5. The ability to represent a choice of alternative activities.
6. The ability to represent a resource and its allocation.
7. The ability to represent a hierarchical modeling.
8. The ability to represent a modification of the network.

9. The ability to represent an actual state of system.
10. The communicability between man and graphic model.
11. The simulation capacity.
12. The readability on a graphic display.

Other graphic methods such as GANTT charts , PERT technique, UCLA graph, control graph and GRAI Net. They are unable to satisfy these twelve criteria simultaneously. The criteria may lead to a study of the Petri net based graphical representation. A marked Petri net may describe a model of any discrete system where:

- the Net describes the structure of a system.
- the marking describes the state of a system.
- the evolution of the marking describes the functioning of a system.

7.5 Modeling activities in an opencast project by a PERT network

The success of any large scale project is much dependent upon proper planning, scheduling and controlling of various phases of a project. The PERT network is used on large scale projects as a management tool. It expedites and controls the utilization of personnel, materials, facilities and time so that the critical areas in a project may be pinpointed. Accordingly, necessary corrective action can be taken to meet the scheduled completion date. The PERT chart is a graphical representation of the relationships between various activities. The activities are represented graphically by arrows and events by nodes. The longest path through the PERT network is referred to as the critical path. It is a path that is followed to obtain the TE (earliest expected completion time of the event) value for the final event. Timing interpretation can be added to activities for the purpose of evaluating the completion time of a project. The obtained network is a cyclic graph and that us such a repetitive system which can not be modeled. With the marked graphs, cyclic behaviours can be modeled.

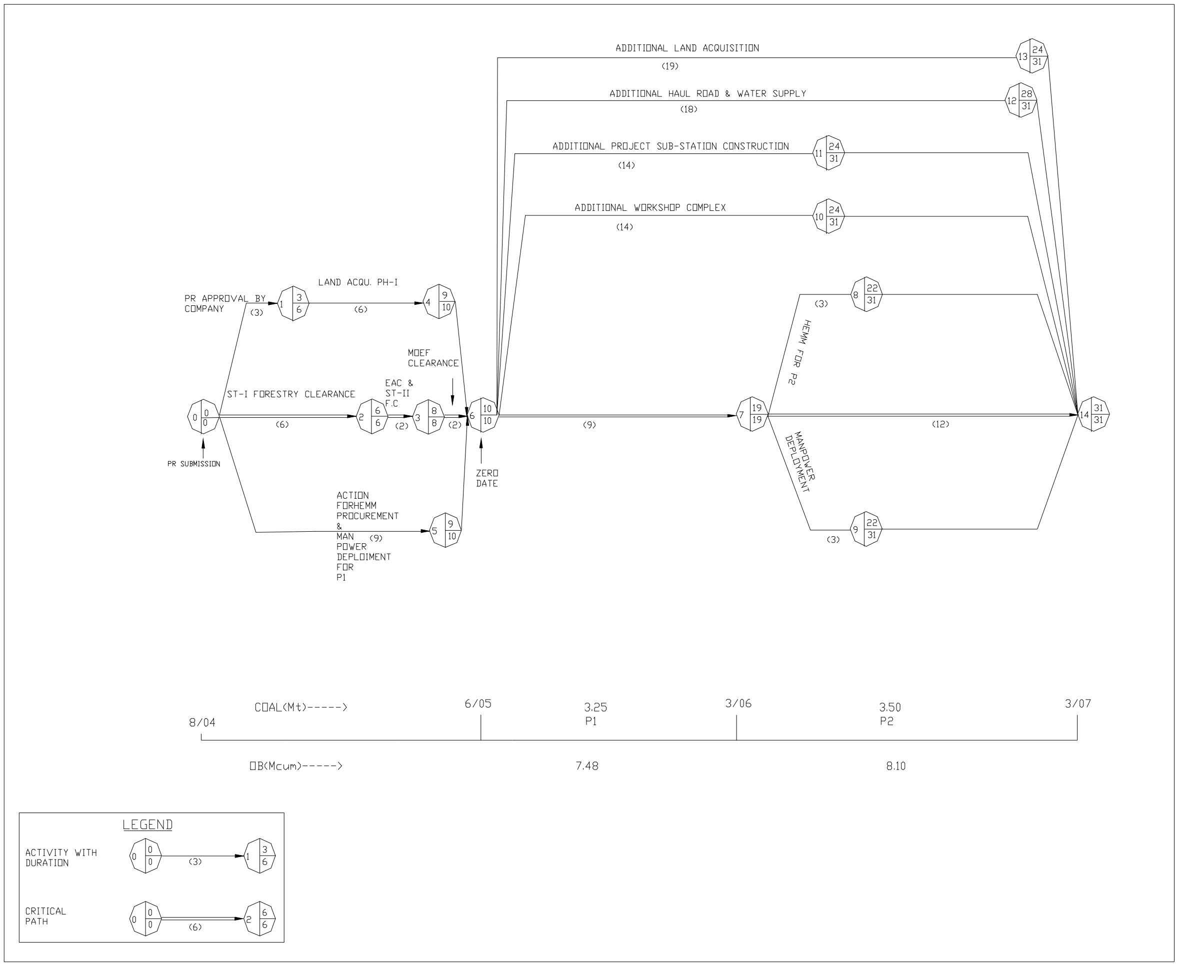


FIG- 7.1: PERT network of an open cast coal mine for expansion

Table 7.1
Activity schedule and duration

		PARTULARS	ACTIVITY MONTHS
1.	0-1	PR Approval By the Company	3
2.	0-2	Stage-I Forestry Clearance	6
3.	0-5	Action for HEMM Procurement & Manpower Deployment for P1	9
4.	1-4	Land Acquisition, Phase-I	6
5.	2-3	EAC & Stage-II Forestry Clearance	2
6.	3-6	MoEF Clearance	2
7.	6-7	Coal Production and OB removal in P1	9
8.	6-11	Addl. Workshop Complex Construction	14
9.	6-10	Addl. Project Substation Construction	14
10.	6-12	Additional Haul Road & Water Supply	18
11.	6-13	Addl. Land Acquisition	19

12.	7-8	HEMM Procurement & Commissioning For P2	3
13.	7-9	Manpower Deployment for P2	
14.	7-14	Coal production and OB removal in P2	12

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The present study reviews the PERT network in the context of an open cast coal mine that produces 2.0 Mt of coal per annum. It has to be enhanced further to 3.5 Mt per annum. The matter is still under considerations. The incremental production of 1.5 Mt is divided into 1.25 Mt and 0.25Mt.for first year and second year respectively. It may be for the implementation of the expansion project. The PERT chart is prepared from the date when the project report (PR) is submitted till an achievement of targeted capacity of. 3.5 Mt/annum. Hierarchical decomposition of different tasks (activities) is carried out to identify the constraints. This may be encountered in the course of implementation of project activities. The activities like PR approval by the company, land acquisition, forestry clearance. MOEF clearance, HEMM procurement, manpower development, additional haul road construction, water supply, substation construction, workshop complex etc are projected in the PERT network with their earliest completion time & latest completion time. The critical path is also marked in the network. (fig.7.1).

7.6 Conversion of PERT chart into equivalent Petri net model

The Petri Nets show the same type of scheduling constraints as the PERT chart does. We can easily convert a PERT network to a Petri Net model. Each activity and event on the PERT chart is represented as transition and place respectively in case of the Petri net model. The Petri Net is an excellent vehicle to represent the concurrency and precedence constraints of the PERT chart. In modeling the PERT chart with the Petri Net, activities that are represented by transitions and places are reserved to model multiple resources of limited amounts. The Petri Nets are more powerful models as compared to the PERT/CPM charts. The following reasons may be given for it:

i) The repeated performance of activities, if necessary, can be modeled by the Petri Net.

ii) Required resources per activity appear explicitly as tokens in the representation.
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iii) Non-deterministic aspects can be dealt with. For example: the order in which a particular resource performs some tasks may not be totally specified.

This simple PERT network is converted to an equivalent Petri net (fig.7.2). The dummy transitions are added as per the need to cater to the requirements of timely completion of the activities.

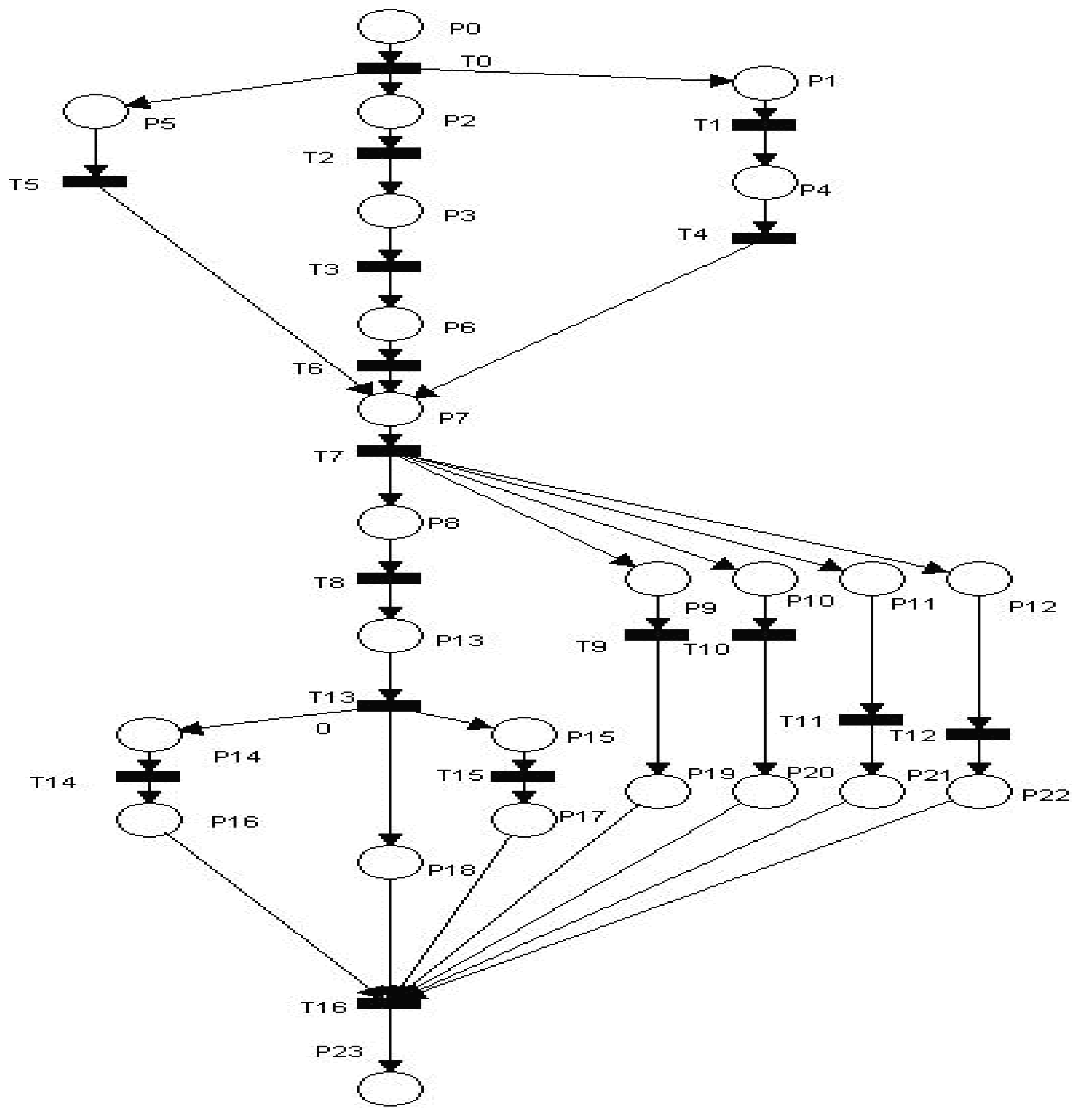

Fig-7.2: Equivalent Peti Net Model of the PERT Chart

Table – 7.2

Description of scheduling operation and its interpretation

Place		Transition	
P0	PR submission starts	T0	PR submission is in progress
P1	PR submission is complete and PR approval process commences	T1	PR approval process is in progress
P2	St.I forest clearance commences	T2	St.I forest clearance is in progress
P3	St.I forest clearance is complete and EAC St.II forest clearance starts	T3	EAC St.II forest clearance is in progress
P4	PR approved is completed and Land acquisition Ph.I is commenced	T4	Land acquisition (Ph-I) is in progress
P5	Action for HEMM procurement and manpower deployment for P1 commences	T5	HEMM procurement and manpower deployment for P1 is in progress
P6	EAC ST.II clearance is complete MoEF clearance is commenced	T6	MoEF clearance is in progress
P7	MoEF clearance is complete and Land acquisition (Ph-I) is complete	T7	Action for infrastructure development is contemplated
P8	Coal production and OB removal in P1 is commenced	T8	Coal production and OB removal in P1 is in progress
P9	Addl. workshop complex construction work begins	T9	Addl. complex construction work is in progress
P10	Addl. project substation construction begins	T10	Addl. project substation construction is in progress
P11	Addl. haul road and water supply work begins	T11	Addl. haul road and water supply work is in progress
P12	Addl. land acquisition is commenced	T12	Addl. land acquisition is in progress
P13	Coal production and OB removal for P1 is complete	T13	Coal production and OB removal for P2 is contemplated.
P14	HEMM procurement and commissioning for P2 is commenced	T14	HEMM procurement and commissioning for P2 is in progress
P15	Manpower deployment for P2 is commenced	T15	Manpower on deployment for P2 is in progress
P16	HEMM procurement for P2 is complete		
P17	Manpower deployment for P2 is complete		
P18	Coal production and OB removal for P2 begins	T16	Coal production and OB removal for P2 is on progress
P19	Addl. workshop complex construction is complete		
P20	Addl. project substation construction is complete		
P21	Addl. haul road and water supply work is complete		

P22	Addl. land acquisition work is complete		
P23	Achievement of additional capacity of project is complete. (2+1.5)=3.5Mte		

7.7 Conclusion

The Petri Net based scheduling of mining projects that are reviewed, in this chapter as a simple and effective method. It may provide mine managers an assistance to develop/work out realistic time and cost estimates for complex mining projects. A practical case study is analyzed. An attempt is made to bring some changes in the usual and traditional system of PERT network incorporation in the Project Report. A thorough study and analysis of the Petri net model is necessary for validation of data and results. There is enough scope of research in the field of application of the Petri net in the mining. Various advantages such as: -

1) A hierarchical and modular decomposition of complex mining projects can reduce the complexity of the project network.
2) Cyclic and probabilistic arcs can be used to model uncertainty and risk,
3) Uncertainty in the time and cost estimates can be modeled by using appropriate statistical distribution. In order to model the shared resources and the dynamic allocation of resources, coloured Petri net can be used by the mine managers to predict resource availability problems.

CONCLUSION

8.1 Summary

The present day market scenario is highly competitive and dynamic. Capital investment of any kind needs to have presumption and production based on efficient engineering as well as financial analysis. It helps to establish profitable proposition. In terms of coal markets, a high demand of coal focuses concerned attention on the coal mining industry. Consequently, a close attention is paid to the aspects of mechanization of the opencast mining. Now a days, the shovels of varying sizes are commonly used at large opencast mines. Their usual capacity combinations are 10 cum rope shovels with 85T rear dumpers. At the Northern Coalfields Limited, a subsidiary of Coal India Limited, 20cum shovels along with 170t dumpers are deployed in view of a need of excavating high volume of overburden in high stripping ratio mines. A usual method of calculating productivity is expressed in terms of output of coal per man-shift. But it does not reflect a correct picture of the opencast mines. It is because of the fact that it does not take into consideration the stripping ratio. It is, therefore, thought more appropriate to express the productivity in terms of cubic meters of volume of excavation (coal and OB) per man-shift in place of the conventional output in tones per man-shift. As far as the profitability of an opencast mine is concerned, it is important to adopt a broad and preferably long term outlook that may be based on mathematical models.

The present study seeks to evolve a method to minimize cost of production right since the stage of planning and designing of opencast mining projects. The operations of the opencast mines involve mainly

(i) drilling and blasting

(ii) loading and transportation.

For the former, an optimal design is needed, while for the latter optimal allocation of shovel dumper combination is required. Monitoring of various parameters of an opencast mining project may be affected on the basis of network analysis. It is essential to avoid any cost overrun or time overrun. In this study, investigations are carried out on the above aspects keeping in view the maximization of productivity and profitability of opencast mining projects. It may result in significant benefits in the long run provided it is followed up properly.

Planning at highly mechanized opencast mines is worked out with an objective of high production of coal and, thus achieving high productivity. The exploitation method essentially requires a matching development system, because any slippage in a progress schedule may adversely affect the main method of extraction. With a very high capital cost requirements for mechanized opencast projects, the coal industry can no longer afford to make decisions in respect of strategic as well as tactical planning solely on the basis of experience and subjectivity no matter whatever may be the reasons for mechanization. The management is ever confronted with crucial decisions as to how much amount of capital needs to be spent on each sub-system towards its mechanization and how to achieve a pertinent target. The management has to seek to justify the investments on both on short term and long term grounds.

The investigations that are carried out in the present research work impart a few insights as to how these decision problems can be addressed with an ultimate goal of strategic / optimal planning. As a part of the investigative research being carried out here, a few mathematical models are developed that can help the management in making objective decisions. To a great extent, it may exhibit potentiality to eliminate inherent risks involved in the subjective mode of decisions that is based entirely on experience and intuition or by a thumb rule approach. It is particularly true in cases of allocation of Heavy Earth Moving Machineries (Shovel-Dumper) to different faces of opencast mining projects.

The present research work seeks to verify various parameters that are responsible for enhancing production and productivity and minimizing cost of production. These parameters monitor major activities. The opencast mining operations involve processes like drilling, charging, blasting, loading and transportation. The present research reviews how these processes are being modeled through suitable OR techniques.

The primary operation of excavation starts with drilling. The basic equipment required for it is a drill machine. Drilling activities are usually hazardous. An operator is exposed to an environment which is prone to the hazards of dust, noise, temperature, vibration etc. In order to safe-guard environment and human health against such hazards, suitable means is searched about to automate drilling operation. The Petri net model has been tried out and implementation of the principle of Markov Chain is formulated and reviewed to find out the working state probability. Blasting in coal and overburden using required quantity of explosives to produce optimum fragmentation too has been reviewed. A study of blast vibration is also conducted applying the multivariate linear regression model to different decision variables. Further, the queueing theory is applied to find out the optimum number of dumpers in the operation at an opencast mine. Also, the Petri net model is worked out for shovel-dumper allocation. The initial activities of opencast mine operations are normally represented in a PERT network in a Project Report. An attempt is made to convert these initial activities from the usual PERT chart to the Petri net model to allow better analysis and simulation. It helps in effective implementation and monitoring of various activities in time so that the target of a project is achieved in due time without allowing any time overrun or cost overrun to occur. In the present study, the application of the strategic planning models is assessed in the interest of achieving the target capacity. Further, various Operations Research techniques are applied to model mine operations so that the analysis of the present method is done mathematically and an optimal solution is worked out in terms of obtaining the desired goal. In this way, entire production planning is worked out with application of the single criterion and multi-criteria decision of mathematical models. Statistical methods are used to establish the prediction equations of blasting and it is optimized using a single criterion decision model. A multi objective (criteria)decision taken in relationship with the availability of shovel-dumper and their dynamic allocation, quantity of OB or coal produced and transported, investment required to achieve the same, manpower requirement etc. provide a decision maker an efficient tool to plan and design in a more realistic manner.

8.2 Conclusion

The outcome of the research work presented in this dissertation may lead us to draw certain conclusions. They may be summarized as follows:

i) The planning of an opencast mining project is preceded by collection of required information and data as regards to geological, financial, manpower, HEMM and other related matters as they prevail under the existing conditions. A careful attention has to be paid while collecting and compiling the information in an usable form. Accuracy of the data is the first requisite for realistic planning. Hence, they need to be collected and verified with personal verification on the site as well as with available records.

ii) The operations that involve charging and blasting occur with high probability, whereas other preliminary processes such as drilling and loading occur with low probability. In order to maximize the probability of the overall operation a proper mix of operations based on Markov Chain, covering working state and non-working state need to be adopted.

iii) The charging and blasting times need to be reduced to bring down the idle time that may occur with the costly HEMM when the method is employed at mining operations. A problem of this kind may be resolved with mechanized loading of SMS.

iv) There occur several associated problems with multifold increase in explosive consumption. Significant technological developments may be adopted in the direction of the usage of explosives. It may help to overcome them to some extent. Optimum and proper design of blast hole geometry, viz. bench height, burden, spacing, borehole dia, average charge/hole and total charge in a round are the remedies suggested to apply with proper use of explosives. It helps to reduce associated problems and results in suitable fragmentation of the blasted muck.

v) As operations of mechanized opencast mines increase, the problem of safety of surface structures in close vicinity against ground vibrations and fly rock fragments may cause serious constraints for the progress of mining operations. The problem sometimes takes such a serious turn that it may suspend or hamper the opencast mining operations for a considerable period of time. A technique of controlled blasting with proper selection of explosive parameters, maximum charge/delay, total charge/round and distance of structures may help to resolve this problem. It is modeled in this study as to how it may be done. The relation among these parameters is significant to decide over the peak particle velocity (ppv) which specifies the safety of the structures. In this regard, a study was undertaken and a model is formulated to work out a safe limit of structures. The maximum charge/delay and total charge/round may be restricted in accordance with the distance of the structures which has to be maintained in view of a blasting operation. The regression model specifies the limiting factors of different decision variables to enable a blasting officer to take decision on the basis of this mathematical model so developed and validated.

vi) The queueing model is applied to find out the optimal number of dumpers to be deployed to a shovel. Further, the dynamic resource allocation model for shovel-dumper combination is worked out with an application of the Petri net techniques. It renders encouraging results at the opencast mining projects. The planner needs to go for a thorough study before he assigns a set of machines at a specific opencast project.

vii) While allocating dumpers to shovels in an opencast project, techno-economical analysis needs to be made to suit the existing mining condition, pit design and other operating parameters.

viii) The PERT network of the opencast mines is normally prepared and incorporated in the PR. The Petri nets, being more powerful models than PERT charts, are adopted by converting the traditional PERT chart. Various places and transitions conversions are incorporated. This Petri net based scheduling of initial activities is a simple and effective tool which provides a mine manager a guidance to develop a realistic time and cost estimates for complex mining projects.

ix) Complex mining projects are put to hierarchical and modular decomposition. It reduces the complexity of the project network. Cyclic and probabilistic arcs are used to model uncertainty and risk. Uncertainty in the time and cost estimates may be modeled by using appropriate statistical distribution. In order to model the shared resources and the dynamic allocation of resources, coloured allocation of resources and coloured Petri nets may be used by the mine managers to foresee resource availability problems.

8.3 Scope of Future Research

The present research work has basic concern with the applications of Strategic Planning Models and Operations Research Techniques in production planning and optimal resource allocation at opencast mines. It is among crucial and critical requirements at an initial point of new opencast mine or at a stage of modifying an existing mine. The following recommendations are worked out to indicate scopes of future research in this direction:

i) The present work is based on the data obtained from the records and mining sites of five different opencast projects. But once the underlying techniques for model building are laid down, the study may be extended to more number of opencast mining projects with varied working conditions and environment.

ii) There is a scope to attempt a similar kind of problem with modified Markov Chain in activities like “Working face” simultaneously. It may create multi-channel service facilities for loading and unloading operations. It may be attempted through a queueing model.

iii) The automation is greatly felt in drilling operation. It is the prime requisite in the present era of technology revolution. The condition of a drill operator inside the cabin is really horrible and unhygienic. An attempt has been made to model the activities of drilling operation by using the Petri nets. The model is simulated in a computer. Further research is required to validate the results in various opencast mines.

At underground mine, it has already been applied in short-wall as well as continuous miner technology faces.

iv) The dissertation considers only the geometrical parameters of blasting at open cast mines. But the explosive characteristics, rock characteristics may be taken into account to carry out further research in the area. As such, the rock characteristics of each area under study are assumed to be same for each blasting and it may not be the case always. The explosive characteristics may vary from company to company and also types of explosives vary in the same company. It may be assumed that fragmentation of rock is suitable for loading by shovel into the dumpers. During the period of data collection, it was observed that there was no such boulder formation. Fragmentation was of desired size. However, fragmentation may not always be the same. It may depend on the strength of rock or a type of explosives used for blasting. It would be more realistic if fragmentation is considered in future studies with varying blast parameters that determine the geometric volume of blast.

It has been revealed from the present study that there was no toe formation in almost all blasting operations. However, the possibility of such occurrences can't be ruled out. Toe requires secondary blasting which involves high cost and unsafe and may further deteriorate the floor of the bench. Hence, in view of economical production and prediction of quantity of volume of blast, toes demand attention as one of the variables.

v) A study was conducted for blast vibration with a few variables like maximum charge/delay, total charge in a round and the distance of the structures from sites of blasting. Through it, the peak particle velocity is determined. The concept of frequency is yet to be experimented with the purpose of analysis in the future scope of research in this vibration study and analysis. It needs to be done with frequency and ppv keeping in view the other variables. The frequency is one of the parameters that relate to vibration of ground that may cause damage to the nearby structures. This needs further study and analysis, with a view to arriving at concrete remedies in such sensitive issues.

vi) The man machine relationship or the ergonomics has not attracted much attention. These conditions are assumed to be invariant in the area under study. Similar assumptions are made for work environment and labour management relationship. Such assumptions are, however, not tenable when the problem is extended to cover larger areas of operation. Consequently, these factors must be suitably scaled or quantified and incorporated in the models.

vii) Standardization is essential for the shovel dumper combination systems and other Heavy Earth Moving Machineries when they are assigned at work places. Attempts are made to dynamically allocate the dumpers to shovels using the concept of Petri nets. The further study may be carried out by including the timing concept and analyzing the model mathematically or graphically by reachability tree techniques. The Petri net model may also be applied to dragline automation or surface miner application. For economical and efficient planning & implementation of the project operation, all types of HEMMs need to be modeled and analyzed for optimal utilization and achieving maximum efficiency.

viii) The queueing theory has been used to model the shovel dumper combination system. The factors like availability and utilization along with the match factor may be adopted for further study and analysis.

ix) More research work may be called for in view of design of shovel and dragline benches to accommodate dumpers, dozers, drills and other equipment for their proper manoeuverability.

x) Initial operations of an opencast mine project are modeled by the Petri net and various activities can be monitored regularly. This Petri net concept can also be applied to short term and long term planning of the mine operations. There is enough scope of research for applying the Petri net model for strategic planning of opencast projects. It is because of the advantages of the model in terms of their analysis capability.

xi) The total mine system planning that is developed with the Strategic Planning Models and Operations Research techniques may be of immense help to a decision maker. For this, research needs to be conducted to develop decision models through an application of suitable quantitative techniques for individual subsystems. The possible interactions need to be cross-checked to achieve the objective of the modeling.

The Petri net model developed under this study is essentially analytical in nature. It can be applied to develop automation and robotics in the field of opencast mining technology. An application of stochastic Petri net modeling and Operations Research techniques to various mine sub-activities needs to be addressed to find out a suitable method for optimal production of coal and OB at opencast mines.

At the end, it is opined that in the present time of globalization and economic liberalization, the mine planning as well as its operations of opencast or underground mining need to be taken up judiciously and objectively. It has to be based on a rate of growth and development at par with the international standards. The techniques and tools of the Strategic Planning Models and Operations Research concepts may be availed, or else, the very survival of the mining industries will be at stake.

REFERENCES

1 Achoff, R.L. and Sasieni, M.W.(1968): “Fundamental of Operation Research”, John Wiley and Sons., Inc, N.Y, pp.455

2 Adhikari, G.R. (1994): “State of the art in blasthole stemming”, Minetech, vol.15, no.2, March-April, pp.56-59.

3 Adhikari G.R., Balachandar, R. and Gupta, R.N. (2003): “Analysis of blast vibrations for two construction projects”, Indian Mining and Engineering Journal, vol. 42, no.06, pp.38-40.

4 Ahuja,N.T.H. and Nandakumar, V.(1985): “Simulation model to forecast project completion time.” Journal of Construction Engineering and Management, ASCE, 111(4), pp.325-342.

5 Ang,A.H.S, Abdelnom, J. and Chaker, A.A. (1975): “Analysis of Activity Networks under Uncertainty”, Journal of Engineering Mechanics, ASCE, vol.101, no.EM4, pp. 373-387.

6 Ash, R.L. (1968): “The Mechanics of Rock Breakage”, Pit and Quarry, vol.56, nos.2,3,4,5.

7 Ash, R.L. (1968): “The design of blasting rounds”, Chapter 7.3 of Surface Mining, ed. E.P. Pflider, Seeley W.Mudd Series, New York, pp.373-397.

8 Balbas, Anton & Garcia, J.I. Diaz (1995): “Spatial relation between laws of vibration from blasting”, Int. Journal of Surface Mining and Reclamation” Balkema, Rotterdam, Netherlands, pp. 161-164.

9 Berthomieu, B. and Diaz, M. (1991): “Modeling and Verification of time dependent systems using time Petri nets”, IEEE transactions on software engineering, vol.17, no.3,March, pp.259-273.

10 Bhattacharya, J. (2003): “Determination of Technical factors and optimization techniques in mine planning”, “Principles of Mine Planning”, published by Allied Publishers Pvt. Ltd., pp.37-105.

11 Biran K.K and Ramaswamy, P. (2000): “Surface Miner – eco-friendly equipment for opencast mines”, Mining Engineers’ Journal, pp.9-17.

12 Bordia, S.K. (1978): “Application of computers in mine planning and design”, Proc. of the Seminar on Applications of Computers in Mineral and Allied Industries, I.E. (Ind.), Dhanbad sub-center, Sept., 15, pp.6-10.

13 Buffa, E.S. and Dyer, J.S. (1981): “Management Science, Model Formulation, Operations Research and Solution Methods”, 2nd ed., John Wiley and Sons, New York, USA, pp.546-556.

14 Campos, Javier., Colom, M. Jose. and Silva, Manuel. (1990): “Performance evaluation of repetitive automated manufacturing system”. Proceedings of

Rensselaers second international conference on computer integrated manufacturing", IEEE transaction, pp.74-81.

15 Chaudhury, S.K. (2005): "Indian Coal Sector At Cross Roads ", The Coal Summit 2005, Organised by India Energy Forum, The Mining Geological and Metallurgical Institute of India, Delhi Chapter and Indian Coal Forum, Edited By Dr.M.M.Seam and N.N.Gautam, pp.59-70.

16 Chen, Chao-Shun., Ke. Yu-Lung, Wu. Jaw Shyang and Kang, Meei-Song. (2002): "Application of Petri nets to solve distribution system contingency by considering customer load patterns", IEEE Transactions of Power Systems, vol.17, no.2,May, pp.417-423.

17 Chhipa, V.K and Sengupta, M. (1995): "Artificial intelligence in surface coal mine planning", International Journal of Surface Mining & Reclamation, pp.47-51.

18 C.M.Robert (1987): "Surface Blast Design" Explosives and Rock Blasting Field Technical Operations, Atlas Powder Company, Subsidiary of the Tyler Corporation, Dallas, Texas, USA, pp.295-316

19 CMPDIL Publication (1984): Coal Mining in India.

20 Cook, S.L. (1956): "Operations Research in Underground Mining", Operations Research for Management, vol.2, Ed. J.F.Mcclosky, Baltimore, pp.211-232.

21 Cummins, A.B. (1973): "SME Mining Engineers Hand Book", Vol..I, Vol..II,Society of Mining Engineers, The A.I.M.P.E, New York.

22 Dantzig, G.B. (1963): "Linear Programming and extensions", Princeton, N.J. Princeton University Press.

23 Dantzig, G.B. (1951): "Application of the Simplex method to a Transportation problem", Chapter III, Activity Analysis of Production and allocation, New York.

24 Debnath, A.K. (1991): "Planning and Design Considerations in Coal Mine Mechanisation", Minetech, May-June, vol.12, no.3, pp.22-28.

25 Dhar, B.B. (1993): "Problems and prospects of Indian mining Industry-Current status and future scenario", National Seminar at R.E.C, Rourkela, Dec., 19-20, pp.1-14.

26 Dhar, B.B. and Sharma, D.K. (1978): "Optimization of Mine Transport System", Ind. Mining and Engg. Jr., vol.26, no.5, May, pp.12-16.

27 Diwedi, Ravindra and Singh, T.N (2003): "A mixed empirical model approach for prediction of ground vibration due to rock blasting". Mining Engineers Journal, vol.5, no.5, pp.15-21.

28 Dzharlkaganov, Bukeikhanov. D.G. and Zhanasov, M.Zh. (2004): "Designing and planning of mining operations development at opencast on the basis of integration of optimization methods and non-formal procedures of decision making", Mine Planning and Equipment Selection – Hardygora, Paszkowska and Sikora (eds), Taylor and Francis Group, London, pp.27-30.

29 Douglas, W. (1981): "Minicomputer, Software for the mineral industry", Mining

Engineering, vol.33, no.11, pp.1588-1596.

30 Edmiston, Phil. (1988): “Short range scheduling for open pit mining”. Proceedings of the First Canadian Conference on Computer applications in the mineral industry/Quebec 7-9 March, Ed. By Kostas Fytas Jean-Luc Collins & Raj K. Singhal, A.A.Balkema publishers Rotterdam,pp.63-65

31 Ehrenberger, V. and Fajkos, A. (1990): “Modeling situations in opencast coal mining” Advances in Mining Science and Technology’s “Mining Modeling” published by Elsevier, Amsterdam Oxford, New York , Tokyo, pp.170-190.

32 Elbrond, J. (1988): “Evolution of Operations Research Techniques in the mining industry” Computer Application in the Mineral Industry, Fytas, Collins & Singhal (eds), Balkema, Rotterdam, pp.3-7.

33 Elevli, Birol (1995): “Open pit mine design and extraction sequencing by use of OR and AI concepts”. International Journal of Surface Mining & Reclamation, pp.149-153.

34 Fabricius, Stefan, M.O, (2001): “Extensions to the Petri net Library in Modelica” Swiss Federal Institute of Technology Zurich (ETHZ), Swizerland, Laboratory for safety analysis.

35 Fytas, K., Hadjigeorgious, J. & Collins, J.L. (1993): “Production scheduling Optimization in open pit mines” International Journal of Surface Mining and Reclamation, pp. 1-9.

36 Gardener, D.C. (1984): “Managing through information”, The Mining Engineer, vol.143, no.9, Sept.

37 Gershon, Mark. (1988): “An open pit planning and scheduling system” Proceedings of the First Canadian Conference on Computer applications in the mineral industry/Quebec 7-9 March. Ed. By Kostas Fytas Jean-Luc Collins & Raj K. Singhal, A.A.Balkema publishers ,Rotterdam, pp.35-40.

38 Ghose, A.K (2004): “Innovative technology in surface mining – where do we go from here?” Journal of Mines, Metals and Fuels, vol.52, no.11, pp.303-304.

39 Gillett, B.E. (1979): Introduction to Operations Research – a computer oriented algorithmic approach, Tata Mc Graw-Hill publishing Co.Ltd., New Delhi.

40 Grayson, R.L. (1989): “Optimization of Longwall panel dimensions using micro computers”, Proceedings of 21st APCOM, SME, Inc., Colorado, pp.465-470.

41 Grayson, R.L., Geogina, M. and Dean, M.J. (1989): “Computer and software use in the U.S. Coal Industries”, 21st APCOM, SME, Inc., Las Vegas, Feb.-March, USA.

42 Gregony, C.E. (2001) “The Coal Age (from AD 1600)” A concise history of Mining revised edition, published by A.A.Balkema/LISSE/ABINGDON/EXTON (PA)/TOKYO, pp.55-62.

43 Gregory, E.Cedric. (1984): “Explosives for North American Engineers”, Trans Tech Publication, pp. 180-203.

44 Hadley, G. (1962), “Linear Programming”, Addison Wesley publishing Co. In., Reading, Mass., USA., pp.520.

45 Hale, R.D. Rokonuzzaman, M. and Gosine, R.G., (1999): “Control of mobile robots in unstructured environments using discrete event modeling”. SPIE International Symposium on Intelligent Systems and Advanced Manufacturing. Boston.

46 Harries, G (1977): “The calculation of the fragmentation of rock from cratering” 15th APCOM Symposium, Brisbane, Australia, pp.325-334.

47 Harris, G. and Hengst, B. (1977): “The use of a computer to describe blasting”. 15th APCOM symposium, Brisbane, Australia, pp.317-324.

48 Hartley, H. and Spence,A.C. (1985): “Management Information Systems”, The Mining Engineering, vol.144, no.2, pp.452-454.

49 Hartmann, B.G and Tuck, M.A. (2001): “Mine automation: A review and study of automation to determine the feasibility of automation at B.H.P World Minerals Cannington” Proc. Of thirteenth international symposium, Mine Planning and Equipment Selection,2001, Ed. Raj K.Singhal and B.P.Singh, New Delhi, India, pp. 191-199.

50 Hillier, F.S. and Lieberman, G.J. (1987): “Operations Research”, Holden-Day, Inc., USA. CBS Publisher (India), pp.214.

51 Hillion, H.P (1989): “Timed Petri nets and applications to multi-stage production system: Advances in Petri nets” Lecturer Notes in Computer Science 424, New York, NY: Springer-Verlag.

52 Hino. K. (1956):”Fragmentation of Rock Through Blasting”, Journal of Ind. Explosive Soc., Japan, vol.17, no.1, pp.1-11.

53 Huang, Yi. and Kumar, Uday. (1994): “Optimising the number of load-haul-dump machines in a Swedish mine by using queuing theory: A case study”. International Journal of Surface Mining Reclamation and Environment, 8, Balkema Rotterdam, Netherlands, pp.171-174.

54 Hustrulid, William, (1999), “Preliminary guidelines for blast layout”, Blasting Principles in Open pit Mining, A Balkema/Rotterdam/Borrok field, pp.73-94.

55 Jackson, I.F. (1985): “The analysis of a Mining System using structural regression equations”, Ph.D. Dissertation, P.S.U., University park, Penn. State.

56 Jardine, Gordon, H. and Evans, Bryan, J. (1988): “Mine scheduling using interactive graphics” Proc. of the First Canadian Conference on Computer Applications in Mineral Industry, Quebac 7-8, March, Ed. by Kostas Fytas, J.L. Collines and Raj K.Singhal, A.A. Balkema/Rotterdam/Brookfield, pp.19-22.

57 Jawed, M. and Sinha, D.K. (1989a): “Computer aided selection of most economic longwall mining system”, Indian Conference on Computer Application, pp.87-99.

58 Jawed, M. and Sinha, D.K. (1985): “System approach to production planning and scheduling with special reference to coal mines-A case study”, Journal of Mines,

Metals and Fuels, February, pp.37-41.

59 Johnson, Richard A. (2000): "Curve fitting" Millen & Freund's Probability and Statistics for Engineers, published by Pearson Education (Singapore), pp.330-336.

60 Jog, S.N., Kumar, R., and Roy, P.R. (1980) "Mathematical Model for Controlled blasting parameters" Minetech, vol.4, no.3&4, pp.33-36.

61 Kahlon, S.S. (1960): "Performance and power consumption of two continuous-type Mining machines", M.S. Dissertation, P.S.U., University part, Penn. State.

62 Kalam, Abdul. (2005): "Key issues and options that shape the choice of technology in coal mining", The Coal Summit 2005, organised by India Energy Forum, the Mining Geological and Metallurgical Institute of India, Delhi Chapter and Indian Coal Forum, Edited by Dr.M.M.Seam and N.N.Gautam, pp.189-205 (C-4).

63 Karmarkar, N. (1984): "A new polynomial – time algorithm for linear programming" combinatorica vol.IV, pp. 373-395.

64 Kate, S.R. (1996) "Improving drill productivity by using compatible energy explosives", Journal of Mines, Metals and Fuels, vol.XLIV, no.1, pp. 52-56.

65 Kesimal, Ayhan. (1998): "Applying the queueing theory approach to determine the most economical number of trucks matching shovel for overburden removal in a coal mine " Mineral Resource Engineering, vol.7, no.1, pp.29-38.

66 Kharchenko, A.Victor. (1979): "Long term planning of mining operation at existing mines". 16th APCOM Symposium, pp.493-498.

67 Konyukh, V. (2003): "Automation and Robotics. The Different Approaches". 19th World Mining Congress, New Delhi: pp 1741-1748.

68 Konyukh, V. (2002): "Robotics for Mining". Mineral Resource Engineering, vol.11, no.1, Imperial College Press, pp 73-88.

69 Koopmans, T.C. (1951): "Activity Analysis of production and Allocation", New York, John Wiley and Sons, Inc. New York.

70 Koopmans, T.C. (1941): "Optimum Utilization of the Transportation System", Economica, vol.17 (Supplement).

71 Kothari, C.R. (1978): "Network analysis CPM and PERT", Quantitative Techniques, Vikas Publishing House Pvt. Ltd., pp.427-458.

72 Kumar, A. (1996): "Production planning through optimal resource allocation in opencast mines", Ph.D thesis, B.I.T. Sindri.

73 Kumar, Manglam. (1973): "Coal Industries in India", pp.19-37.

74 Kumar, Shailendra. (2002): "Latest trends in surface drilling and blasting", Mining Engineer's Journal, vol.3, no.10, pp.19-22.

75 Kumar Shashi (2005): "CIL Road Map for Meeting Energy Challenges", The Coal Summit 2005, organised by India Energy Forum, the Mining Geological and Metallurgical Institute of India, Delhi Chapter and Indian Coal Forum, Edited by Dr.M.M.Seam and N.N.Gautam, pp.1-21 (A-1).

76 Kuhn, H.W. and Tucker, A.W. (1951):" Non-linear programming", Proceedings Second Berkeley Symposium on Mathematical Statistics and probability, pp.481-492.

77 Ladds, C.G.Davey., O.L., Rorke., A.J., Jordan., D.J.P. and Cunningham, C.V.B. (1993): "Blast monitoring at ATCOM for improved blasting efficiency", The Journal of Explosives Engineering, vol.II, no.3, pp.10-12, 43-45.

78 Lang, L.C. and Favreau, R.F. (1972): "A modern approach to open pit blast design and analysis", CIM, Bulletin, June.

79 Lee, S.M. and Moore, L.J. (1977):"Optimization of Transportation problem with multi objective", AIIE, Transaction, 5, pp.333-338.

80 Levitt, R.E., Kartam, N.A. and Kunz, J.C. (1988): "Artificial intelligence techniques for generating construction project plans" Journal of Construction Engineering & Management, ASCE, vol.114, no.3.

81 Lippincott, S. (1992): "Questions & Answers", The Journal of Explosives Engineers, vol.10, no.3, pp.42.

82 Li, Z. (1990): "A methodology for the optimum control of shovel and truck operations in open pit mining" Mining Science and Techno.logy, 10, Elsevier Science Publisher, B.V.Amsterdam, Netherlands, pp.337-340.

83 Lizottee, Yves. and Bonates Eduardo (1987): "Truck and shovel dispatching rules assessment using simulations" Mining Science and Technology, 5, Elsevier Science Publishers, Netherlands, pp.45-58.

84 Lizottee Yves and Bonates Eduardo (1988): "A combined approach to solve truck dispatching problems" Computer Applications in the Mineral Industry, Fytas, Collins and Singhal (eds), Balkema Rotterdam, pp.403-412.

85 Loomba, N.P. (1964), "Linear programming", New York, Mc Graw Hill.

86 MacCrimmon, K.R. and Ryavec, C.A. (1964): "An Analytical Study of the PERT Assumptions", Operations Research, vol.12, pp. 16-38.

87 Manula, C.B. and Nelson, F.J. (1964): "Evaluating safety with computers", Mining Congress Jr., vol.50, December, pp.55-59.

88 Manula, C.B. and Sanford, R.L. (1969):" A complete coal Mining Simulator," Special Research report no.SR75, PSU, University park, Penn's, State.

89 Markland, R.E. (1979): "Topics in Management Science", John Wiley and Sons, Inc. New York, pp.694.

90 Martens, P.N and Inderst R. (1997): "Representing model based knowledge in mine planning",Mine Planning and Equipment Selection, Balkema, Rotterdam, pp.699-703.

91 Mathur, R.B. (1993):" CMPDI's Role in Locating New Projects and Upgrading coal inventory in India", Minetech, vol.14, no.2, pp.11-15.

92 Maulik, U., De, P. and Bandopadhyay, S. (2001): "Efficient vehicle allocation in

opencast mines: Integration with Genetic Algorithms", Institution of Engineers (India) Journal (Mining), vol.81, February, pp.23-26.

93 Miroslaw and Hajdasinski, M. (1995): "Optimizing mine life and design capacity", International Journal of Surface Mining and Reclamation" pp.23-30.

94 Mishra, A.K.(2004): "Blasting technology in the 21st Century",Minetech vol.25,no.1,pp.44-51.

95 Mishra, M.P (2000): "Continuous Surface Miner – an appropriate technology of MCL", Journal of Mines, Metals and Fuels, MCL number, pp.338-349.

96 Mishra, M.P. and Mahapatra, R.K. (1999): "Improvement in capacity utilization in underground mines – A suggestive approach" National Seminar on System capacity utilization in Mines, MCL, Sambalpur, pp.15-32.

97 Mishra, G.B. (1984): "Surface Mining", pp.30-31.

98 Mohanty, B., Tidman, J.P. and Jogerson, G.K (1988): "Advanced Computer Simulations – the key to effective blast designs in open-pit and underground mines", Computer Applications in Mineral Industry, Fytas, Collins and Singhal(eds), Balkema, Rotterdam, 41-48.

99 Mookherjee, A. (1963): "A study of power consumption of stripping shovels", M.S. Dissertation, PSU University Park, Penn. State.

100 Mukherjee, K. and Prasad, A. (1992): "Optimal planning for coal transportation – A case study", October, IIIE Journal, vol.XXI, no.10, pp.5-10.

101 Munirathinam, Mohan. and Yingling, John C. (1994): "A review of computer based truck dispatching strategies for surface mining operations", International Journal of Surface Mining ,Reclamation and Environment, Balkema, Netherlands, pp.1-15.

102 Murata, T. (1989): "Petri nets: Properties, Analysis and Application" Proc. of IEEE, vol.77, no.4, pp.541-580.

103 Murthy, P.A.N. (1989): "An application of transportation technique (OR) for coal dispatches through various sidings from different collieries in an area", Minetech, vol.10, no.2, March-April, pp.55-58.

104 Mustafi, C.K. (1993): "Operations Research-Methods and Practice", 2nd ed., Wiley Eastern Limited, New Delhi, pp.152.

105 Mutmansky, J.M. (1973): "The scope of system engineering", SME Mining Engineering Handbook, vol.2, Given, I.A. ed., New York, pp.30.1-30.5.

106 Narayanan, M.P. and Mishra, B.C. (1989): "Technological Upgradation, New Technology and R&D in Coal India", Journal of Mines, Metals and Fuels, Dec., pp.543-552.

107 Nelson, N.G. and Machina, M.(2003): "Rock fragmentation by optimized projectiles", 2003 SME Annual Meeting, Feb., 24-26, Cincinnati Ohio, pp. 1-5.

108 Norbert, L.E. (1965): "Management Operations Research", Holt, Rinehart and Winston, Inc., New York, pp.320.

109 Ogbonlowo, D.B. and Wang, Y.J. (1988): “An application of queuing analysis to the production of aggregates in quarry operations”. Computer application in the Mineral Industry, Fytas, Collins & Singhal (eds), Balkema, Rotterdam, pp.423-429

110 Pal Roy, P. and Mondal, S.K.(1999) :“Techno-economic assessment of drilling and blasting parameters some in-depth studies”, Journal of Mines, Metals and Fuels, vol.XLVII, no.12, pp. 381-389.

111 Pal, U.K. and Ghosh, Nilay. (2002): “Optimisation of Blast design parameters at Sonepur Bazari Opencast Project”, The Indian Engineering and Mining Journal, vol.41, no.08-09, pp.36-41.

112 Pattanaik, Ashok Kumar. (1994): “Application of OR models to opencast coal mining”, M.I.M thesis, Sambalpur University.

113 Paul, Biswajit.(2002): “Prospective plan of the coal industry-2020 and the Role of Information Technology”, National Seminar on Prospects of Mineral Industry-2020, (13-14,July) organised by Indian Mine Managers Association, Kalinga Branch, Bhubaneswar.

114 Paul, Jyoti. P. and Sengupta, M (1987): “Computer Aided Blasting Round Design in an Arctic Coal Mine”, Mining Science and Technology, Elsevier Science Publishers B.V., Amsterdam, Netherlands, pp.299-308.

115 Peterson, J.L. (1981): “Petri net theory and modeling of systems” Englewood Cliffs, NJ:Prentice Hall Inc.

116 Pfleider, E.P. (1968): “Surface Mining”, published by the American Institute of Mining, Metallurgical and Petroleum Engineers, Inc New York.

117 Pearse, G.E. (1955): “Rock Blasting”, Mine Quarry Engineering, January.

118 Prisker, A. Sigal, C. and Hammesfahr, R. (1989): “SLAM II network models for decision support”, Prentice Hall Inc., Englewood Cliffs, N.J.

129 Pugliese, J.M. (1972): “Designing blast patterns using empirical formulas”, US Bureau of Mines, I.C. 8550.

120 Raghavendra Rao, Y. and Teggi, Vilas (1997): “Environmental-friendly Mining at Cement Ambuja” Journal of Mines, Metals & Fuels, vol.XLV, pp.203-208.

121 Ramani, R.V. (1974): “Planning in the Mineral Industry present and the future”, Journal of Mines, Metals and Fuels, vol.22, no.10, October, pp.340-342.

122 Ramlu, M., Chakraborty, A.K. and Reddy, A.H. (2002): “Influence of burden on ground vibrations in a lime stone quarry” Indian Engineering and Mining Journal, vol.41, no.8 & 9, pp.24-30.

123 Rai, P. (2001): “Some aspects of material transportation by rear dump trucks in opencast mines”, Minetech, vol.22, no.3, pp.3-10.

124 Rath, S K. (1994): “Studies on some aspects of cellular manufacturing systems” Ph.D thesis (unpublished.) submitted to IIT, Kharagpur.

125 Ray, S.C. (1984): “A decision model for working small deposits”, Proceedings of

National Science Academy, 50 a (5).

126 Ray, S.C. and Mazumdar, S. (1980): "Application of Linear programming to investment allocation over the life of a mining project", Journal of Mines, Metals and Fuels, Dec., pp.279-282.

127 Rivettee, B.H.P. (1956): "Operations Research in British Coal Industry", Operations Research for Management, vol.2, ed. Mcclosky, Baltimore, pp.195-210.

128 Rzhevsky, V.V. (1985): "Opencast Mining – Unit operations, MIR Publishers, Moscow.

129 Samanta, Bimal., Sarkar, Bijan. and Mukheree, S.K. (2005):"Modeling and Analysis of HEMM engine failure with Petri nets". Industrial Engineering Journal vol.XXXIV, no.4, pp. 9-20.

130 Sastry, S.S. (1989): "Matrices and Linear Systems of Equations: Introductory Methods of Numerical Analysis", published by Prentice Hall of India, New Delhi, pp.161-193.

131 Sastry, V.R., Teggi, V. and Chandar, K.R. (2003): "Shock tube initiation for eco-friendly blasting, a few case studies", IE(I) Journal-MN, vol.83, pp.40-46.

132 Sawhney, Anil. (1997) "Petri net Based Simulation of Construction Schedules" Proc. of the 1997 Winten Simulation Conference, ed S.Andradottir, K.J.Healy, D.H.Withers and B.L.Nelson, pp.1111-1118.

133 Schimm, Bernhard. (2004): "Surface Mining applications worldwide for coal and limestone mining with particular reference to India", Journal of Mines Metals and Fuels, vol.52, no.11, pp. 236-239.

134 Seam, M.M. and Dutta, N.L. (1989): "Optimal fragmentation – parameters of drilling and blasting", The Indian Mining and Engineering Journal, vol.XXVIII, no.10, pp.45-53.

135 Seegmiller, B.L. (1973): "Computers and Operations Research, Mine usage", SME Mining Engineers Hand Book, Ed. I.A. Given, vol.2, Sec.30, AIME, New York.

136 Sen, Sumit (2003): "Blasting Optimization and Simulation : 'Computing Benefits' to Mining Operations" Proceedings of Fourth Indian Conference on Computer Applications in Mineral Industry (ICCAMI,2003) during Nov.,7-8 at RRL (CSIR), Bhubaneswar, ed by K.K.Rao, PSR Reddy & V.N.Mishra, pp. 14-24.

137 Sharma, D.K. (1977): "Mine System Engineering". The Indian Mining and Engineering Journal, vol.16, no.12, Dec., pp.15-17.

138 Shenoy, G.V. and Pant, Madan. (1994): "Regression Analysis and Multivariate Statistical Analysis", Statistical Methods in Business and Social Sciences", pp.207-274.

139 Shulman, M.J. (1989): "Optimal allocation with limited resources", Proceedings of 21st APCOM, SME, Inc., Colorado, pp.116-131.

140 Singh, G.P. (1999): "Optimisation of shovel dumper combination by using queueing

theory" M.Tech(Mining) thesis, IIT, Kharagpur.

141 Singh, S.P and Lamond, Rodney. D. (1993): "Prediction and Measurement of Blast Vibration" International Journal of Surface Mining and Reclamation 7 A.A-Balkema publishers Rotterdam, Netherland, pp.149-154

142 Singh, P.K.and Roy, M.P (2003): "Ground Vibration from Blasting Vis-à-vis prediction and control measures", Mining Engineers Journal, vol.5, no.5, pp.7-14.

143 Singh, R.D. (1997): "Principles and practices of Modern Coal Mining" New Age International (P) Ltd. Publishers New Delhi.

144 Singh, S.R. (1988): "Productivity growth in Coal Mining Industry in India", Minetech, vol.9, no.5, November-December, pp.73-87.

145 Singh, S.R. (1989): "Elasticity of substitution and productivity change – Coal Indian Experience", Minetech, vol.10, no.2, pp.73-78.

146 Singh and Vanoudhensden (1992): "Joint Pricing and Production decisions under varying and constant price", Journal of Engineering Production.

147 Singh, V.K. (2004): "Surface Mining Technology at Northern Coalfields Ltd.", Journal of Mines Metals and Fuels, vol.52, no.11, pp.227-235.

148 Singhal, Raj.K. and Fytas, Kostas (1986): "Optimization of loading and hauling operations in surface mining" proceedings of 3rd conference on the use of computers in coal industry/Morgantown, pp.119-129.

149 Sinha, D.K. (1972): "System analysis with application of economy-mathematical modeling in computerized planning and control of mines", Proceedings of the symposium on computer applications and operations research to mining problems, TISCO, March 10-11, pp.397-403.

150 Sinha, D.K. (1984): "Optimal planning process for exploitation of mineral resources with reference to developing nations", World Mining Congress, New Delhi, November.

151 Sinha, D.K. and Sharma, R.N. (1976): "Total System Planning for exploitation of coal in India" 14th APCOM symposium, pp.726-739.

152 Sinha, D.K. and Mukherjee, K. (1983): "Optimization of long term plan in opencast project through Operations Research", Proceedings of National symposium on surface mining, ISM, Dhanbad.

153 Sinha, D.K. (1972) and Ray S.C. (1978): "Capacity Analysis of Mine Systems – a prerequisite for production planning in underground mines", Indian Mining and Engineering Journal, May, pp.7-11.

154 Sinha, T., Rao, V.K. and Choubey, S.W .(2000): "Practical aspects of drilling and blasting in NCL" Journal of Mines Metals and Fuels, vol.XLVIII, pp. 94-98.

155 Sko Dack, T. (1982): "Using care to predict equipment needs, cost and productivity", Journal of Mining Engineering, November.

156 Sprott, David (1988): "Computer application in blast vibration monitoring", Computer

Application in Mineral Industry, Fytas, Collins & Singhal (eds) Balkema, Rotterdam, pp.489-494.

157 S.Ramazan, K.Dagdelen and T.B.Johnson(2005): "Fundamental tree algorithm in optimizing production scheduling for open pit mine design". Mining Technology, March, vol.114, no.1, pp. A45 - A54.

158 Srinivas, P. and Venkateswara ,Rao. Y. (1989): "Cyclic queuing models for open-pit productivity analysis" Proceedings of XXII, Annual Convention of Operations Research Society of India, Dec. 7-10, pp.113-115.

159 Sturgul, John, R. (2000): "GPSS/H and Simulation in Mining" "Mine Design Examples using Simulation" published by the Society for Mining, Metallurgy and Exploration Inc., pp.1-19.

160 Swarup, Kanti., Gupta, P.K. and Mohan, Man. (2001): "Operations Research", Ninth Revised Edition, Sultan Chand & Sons, New Delhi.

161 Ta, C.H., Kresta, J.V., Forbes, J.F. and Marquez, H.J. (2005): "A Stochastic Optimization approach to mine truck allocation" International Journal of Surface Mining, Reclamation and Environment, vol.19, no.3, September, pp.162-175.

162 Taffler, R. (1979): "Using operations research", Prentice Hall International, Inc. London, U.K., pp.240.

163 Taha, H.A. (1997): "Operations Research: an Introduction", 6th ed., Prentice-Hall of India ,New Delhi, pp.215.

164 Tatiya, R.R. and Ajmi, A.A .(2000): "A mathematical model to choose burden during bench blasting based on rock strength – a case study", Journal of Institution of Engineers (India), vol.81, pp. 14-17.

165 Temeng, Victor, A., Otuonye, Francis, O. and Frendewey, James, O. (1998): "A non-pre-emptive goal programming approach to truck dispatching in open pit mines". Mineral Resource Engineering, vol.7, pp.59-67.

166 Theil, H. and Van De Panne(1960):"Quadratic programming as an extension of classical Quadratic Maximization", Management Science, vol.7, no.1, October, pp.1-20.

167 Udupa, K.R.D. (1977): "Statistics and its uses in mining", Minetech, vol.1, no.4, March, pp.45-46.

168 Verma, H.L. and Gross, C.W. (1978): "Introduction to quantitative methods – a managerial emphasis", John Wiley and Sons, Inc., New York, pp.694.

169 Volovoi, Vitali. (2003): "Modeling of System Reliability Using Petri nets with aging tokens", Reliability Engineering and System Safety (1-9).

170 Wagner, H.M. (1975): "Principles of Operations Research with application to Managerial Decisions", 2nd ed., Prentice Hall, Inc., Englewood Cliffs. N.J., pp.223.

171 Wang, Jiacun. and Deng, Yi. (2000): "Reachability Analysis of Real Time Systems Using Time Petri Nets" IEEE Transactions on Systems, man and cybernetics-part B:

cybernetics: vol.30, no.5, pp.725-736.

172 Watson, W.D., T. Medlin, K. Krohn, D.S. Brookshire and R.L. Berknope. (1988): "Economic Effects of Western Federal Land use Restrictions on United States Coal Markets" "Operations Research In Review".

173 Wilkinson, W.A and Kecojevic (2004): "Elements of Drill and Blast Design and 3D visualization in surface coal mines" Society of Mining Engineers Annual Meeting, Colorado, 1-6.

174 Woof Mike (2004): "Drilling deep", World Mining Equipment, vol.28, no.2, pp.42-48.

175 Woof Mike, (2003): "New Worlds Robot loading and haulage are coming to material moving". World Mining Equipment. vol.27, no.9, pp. 20-23.

176 Wooley, J.C. and Crandall K.C. (1983): "Stochastic network model for planning and scheduling", Journal of Construction Engineering and Management, ASCE, vol. 109, no.3, pp. 342-354.

177 Zurawski, Richard. and Zhou, MengChu. (1994): "Petri nets and Industrial Applications: A Tutorial", IEEE trans on industrial electronics, vol.41, no.6, December, pp.567-583.

CURRICULUM VITAE

1. Name : KSHIROD CHANDRA BRAHMA
2. Grade/Designation : E7/ Additional General Manager (Mines)
3. Present place of posting : Vastan Lignite Opencast Mines, Gujarat Industries Power Co.Ltd. (GIPCL)
4. Date of Birth : 23.02.1964
5. Date of joining in Coal India Ltd.: 25.09.1987
6. Date of joining in GIPCL : 23.12.2006
7. Age : 44 years
8. Qualification :

a) Educational

Sl. No.	Course	Institute/ University	% of marks	Year
1	Post Graduate Diploma in Information Technology (PGDIT)	Sambalpur University, Sambalpur ,Orissa,India.	73	2003
2	Post Graduate Diploma in Labour Laws & Personal Management (PGDLL&PM)	L.R. Law College, Sambalpur University.	53	1999
3	Bachelor in Laws (LLB)	L.R. Law College, Sambalpur University	55	1997
4	Post Graduate Diploma in Environment & Ecology (PGDEE) (Correspondence course)	Indian Institute of Ecology and Environment, New Delhi	65	1997
5	Master in Industrial Management (MIM)	University College of Engineering, Burla, Sambalpur University	73	1994
6	Degree in Mining Engineering	Regional Engineering College, Rourkela (Now NIT)	80*	1987
7	H.S.C	Board of Secondary Education, Orissa.	69	1980

* Topper of the batch holding the only Honours in the Mining Discipline.

b) Professional : First Class Mine Manager Certificate of Competency (Coal) – 1992

8. Life member of Professional bodies :

Sl. No.	Organization/professional body	Life membership No.
1	Institution of Engineers (India) (IE(I))	FIE - No.F/109699/7
2	Mining, Geological and Metallurgical Institute of India (MGMI)	LM/6054
3	Indian Institute of Industrial Engineering (IIIE)	LM/8678
4	Operations Research Society of India (ORSI)	0342/K/098/ML
5	Institute of Scientific & Technical Education (MISTE)	LM/25889
6	All India Management Association (AIMA)	LM-2021210
7	Mining Engineers Association of India (MEAI)	LM-1285
8	Indian Mine Manager's Association (IMMA)	LM-921
9	National Institute of Personnel Management (NIPM)	LM-22545
10	Indian Society for Training and Development (ISTD)	B-645/2001
11	Indian Association of Environmental Management (IAEM	LM-1444
12	Biomedical Society of India (BSI)	LM-1322841
13	Society of Geo-scientists & Allied Technologists (SGAT)	LM-432
14	Institution of Valuers (MIV)	F-16616 (L.M)

9. Experience :

Sl. No.	From	To	Designation	Working Experience	Place of posting
1.	Dec.' 06	Till date	Head of Management (Mines)	Overall In-charge of all the Mining Projects of GIPCL.	Vastan Lignite Mine, SLPP, GIPCL.
2.	Sept.' 01	Dec.'06	Supdt. Of Mines	Planning & designing of large opencast project of MCL: Lingaraj OCP(10 Mty), Lakhanpur OCP(10 Mty), Samaleswari OCP (5 Mty),Bhubaneswari OCP(20Mty).	CMPDI, RI-VII, Bhubaneswar, Orissa,India.
3.	July'99	Sept.'01	Supdt. Of Mines/ Manager	Sole control and administration of the project as Project Manager. Shifting of village Ghanamal & acquisition of land for project expansion. Liasioning with local authority and villagers for smooth operation of mine. Achieved a target of 4.7Mte/yr coal with profit margin of 100 crores +. First introduction of surface miner in coal mines in this project was grand success in the history of coal industry in India.	Lakhanpur OCP, MCL, Sambalpur, Orissa,India.

Sl. No.	From	To	Designation	Working Experience	Place of posting
4.	Nov.'92	July'99	Quarry Incharge and Production Manager	Looking after management control and direction for the quarry workings with due regards to safety, conservation and quality and producing the targeted Coal and OB with maximum productivity and profitability.	-do-
5.	Aug.'91	Nov.'92	Manager	As Manager of Lakhanpur OCP opened up the mine with access trench and box cut drivage dealing with the local problems of the villagers, local politicians and obtaining permissions and exemptions from statutory bodies.	-do-
6.	May'88	Aug.'91	JET, JME, Under Manager	Shift Incharge & General shift incharge for production of Coal of underground mine	Orient Colliery Mine No.3, Ib-valley Area, Brajrajnagar, Orissa.SECL.
7.	Sept.' 87	May'88	JET (Mining)	Shift Incharge and Blasting Officer in opencast mines	Lajkura Opencast Project ,SECL.

10. Publication of papers : Published 23 papers in National and International Conferences, Seminars, Journals, etc. A few publications are mentioned below:

a. Brahma, K.C. and Kumar, Ashok. (1999):"Multi Variate Linear Regression Model - A Case Study in Opencast Mine Blasting" [Proc. of the Second International Conference on Operations and Quantitative Management (ICOQM), Jan. 3-6, Ahmedabad, India, Tata Mc Graw Hill Publishing Company, New Delhi] pp.472-477.

b. Kumar, Ashok and Brahma, K.C. (1999): "Finite Source and Multiple Server - An application in Mining", Proc. of the Second International Conference on Operations and Quantitative Management (ICOQM), Jan. 3-6, Ahmedabad, India, Tata Mc Graw Hill Publishing Company, New Delhi, pp.451-455.

c. Bandopadhyaya, A.K., Brahma, K.C. and Kumar, Ashok. (1999): "Operation Research Techniques for Optimal Planning and Allocation of Coal - A case study in Mining" National Seminar by MGMI, on SCUIM'99 at MCL, Sambalpur, 6th February,1999, pp. 71-75.

d. Bandopadhyaya, A.K. and Brahma, K.C (2000): "A new horizon in Coal Mining Industry", Proc. of International Seminar on Quality, Productivity & Environmental Concern of the Indian Coal Industry in the new Millennium organised by Indian Mine Managers Association, Bhubaneswar, 22-23rd Jan.'2000,pp.76-80.

e. Bandopadhyaya, A.K and Brahma K.C (2001): "Environment friendly mining of coal at Lakhanpur OCP, MCL, National Seminar on Environmental Issues and Waste Management in Mining and Allied Industries, Feb.,23&24, REC, Rourkela, pp.110-115.

f. Brahma, K.C and Bandopadhyaya, A.K .(2001): "Blast free mining of coal at Lakhanpur Opencast Project of MCL" IMMA National Seminar Mining Vision-2010, 7-8 July, MCL(HQ), Burla, Sambalpur, pp.9-11.

g. Bandopadhyaya, A.K and Brahma, K.C (2001): "Surface miners at Lakhanpur Opencast Project - A revolution in opencast coal mining technology", Proc. of the Tenth International Symposium on Mine Planning and Equipment Selection, New Delhi, Nov.,19-21, 2001 Oxford & IBH Publishing Co. Pvt. Ltd., pp.287-293.

h. Venukumar, N and Brahma, K.C (2003): "Some aspects of Petri nets and its application in Mining" MEAI-Seminar on Recent Trends in Mine Mechanisation Exploration to Mine Closure, 21-22, November, Puri.

i. Venukumar, N and Brahma, K.C. (2004): Petri nets and its application in Mine Automation" National Seminar on Policy Formulation and Strategic Planning for Mineral Industry-2012 organised by IMMA, Bhubaneswar 10-11 January, pp.60-69.

j. Singh, S.R., Venukumar, N. and Brahma, K.C. (2004): "Industrial application of Petri nets - An overview" Geominetech Symposium Proceedings on New Equipment - New Technology Management and Safety in Mining and Mineral based Industries, 11-12, May, Bhubaneswar, pp.176-180.

k. Singh, S.R., Brahma, K.C. and Pal, B.K. (2005): “An approach to a Strategic Planning for Mine closure of opencast mines”. Proc. of the Conference on Technological Advancements and Environmental Challenges in Mining and Allied Industries in the 21st Century (TECMAC-2005), 5-6, February, National Institute of Technology, Rourkela, pp.683-690.

I. Singh S.R., Brahma, K.C. and Mishra, P.C(2004): "Environmental Impact and Monitoring of Air Quality in an Opencast Mine of Mahanadi Coalfields Limited", Proceedings of the All India Seminar on Emerging Technology for sustainable environment in chemical and allied industries (ETSE-2004), organised by Deptt. Of Chemical Engineering, National Institute of Technology, Rourkela during 2nd and 3rd October,2004, pp.87-99

11. Present Address : **Residence**

Quarter No.F4,
GIPCL Township,
At & P.O.Nani Naroli,
Tal.Mangrol, Dist.Surat-394 110,
Gujarat.

Office

Gujarat Industries Power Company Limited,
Vastan Lignite Opencast Mine,
Surat Lignite Power Plant,
At & P.O.Nani Naroli,
Tal.Mangrol, Dist.Surat-394 110,
Gujarat.

12. Permanent Address : At: Nisankhapur,
P.O.: Hatadihi,
P.S: Nandipada,
Dist.: Keonjhar (Orissa),
PIN-756141.

13. Telephone Nos. : 02629-261104 (R), 02629-261103 (O),
09909925303(M), 09909925304(O),

14. Email address : kcbrahma2003 @ yahoo.co.in

15. Language :

English	Hindi	Oriya
Good	Good	Good

16. Objective : I wish to excel in the field of research and development considering the practical bottlenecks and difficulties in production, planning and implementation where my conceptual knowledge would be supportive.

17. Highlights : Good interpersonal & communication skill, Eagerness to learn & implement

18. Strength : Confident, task oriented and having positive attitude.