PREFACE

During my preparation for competitive examination in mining engineering paper, I noticed a very definite lack of proper book that could thoroughly cover all those strange and unfriendly questions asked in the previous examinations. This frustration continued during my teaching in the Department of Mining Engineering, at BIT Sindri, as the students always come with questions asked in the previous examination which I never had the chance to see during my undergraduate studies. As a teacher, however, I often heard from my students those same frustrations that I had years ago.

This book addresses all the topics related to mining engineering course. Students will be more benefited during their short term preparation for the competitive examinations as this book cover every thing in brief. A sufficient number of objective questions have been given in each topic for practice.

This book consists of 10 chapters in which the basic information and formulas are given :

- Chapter 1 : Mining Methods
- Chapter 2 : Rock Mechanics
- Chapter 3 : Mine Ventilation
- Chapter 4 : Mine Environment
- Chapter 5 : Mine Surveying
- Chapter 6 : Mining Machinery
- Chapter 7 : Mine System Engineering
- Chapter 8 : Mineral Economics
- Chapter 9 : Mineral Dressing
- Chapter 10: Geostatistics

CONTENTS

1.0 Mining Methods..... 1-39

1.1 Underground Coal Mining...1-20

1.1.1 India's Rank in World Coal Production...1

1.1.2 Characteristics of Coal...1

1.1.3 Banded Constituents of Coal...1

1.1.4 Size Gradation of Coal...1

1.1.5 Analysis of Coal...2

1.1.5.1 Ultimate analysis...2

1.1.5.2 Proximate analysis...2

1.1.6 Types of Coal...2

1.1.6.1 Non-Coking coal...2

1.1.6.2 Coking Coal...2

1.1.7 Caking or Agglutinating Index...3

1.1.8 Protodyaknov Index...3

1.1.9 Composition of different Coal Burning Gases...3

1.1.10 Incubation Period of Coal...3

1.1.11 Mining Methods...3-6

1.1.11.1 Bord & Pillar Mining...3

1.1.11.1.1 Dimension of Pillar and Galleries at Different Depths...3-4

1.1.11.1.2 Percentage of Extraction...4

1.1.11.1.3 Tributary Area...4

1.1.11.1.4 Load Acting on Pillar...4

1.1.11.1.5 Strength of Pillar...4-5

1.1.11.1.6 Factor of Safety of Pillar...5

1.1.11.1.7 Line of Extraction...5

1.1.11.2 Hydraulic Mining...5

1.1.11.3 Auger Mining...5-6

1.1.11.4 Contour Mining...6

1.1.11.5 Area Mining...6

1.1.11.6 Strip Mining...6

1.1.11.7 Thick Seam Mining Methods...6

1.1.11.7.1 Slice Mining...6

1.1.11.7.2 Sublevel Caving...6

1.1.11.7.3 Integrated Sublevel Caving or Sautirage Mining...6

1.1.12 Explosives, Drilling and Blasting...7-12
1.1.12.1 Definition of Explosives...7
1.1.12.2 Classification of Explosives...7
1.1.12.3 Permitted Explosives...7
1.1.12.4 Types of Explosives...7
1.1.12.5 Characteristics of Explosives...7-8
1.1.12.6 Powder Factor...8
1.1.12.7 DGMS Stipulations on Maximum Permissible Charge in a Shot Hole...8
1.1.12.8 Detonator...8-9
1.1.12.9 Types of Detonators and their Characteristics...9
1.1.12.10 DGMS Stipulations on Delay Interval...9
1.1.12.11 Drilling Pattern...9-10
1.1.12.12 Drilling Method and their Rock Cutting Principles...10
1.1.12.13 Common Methods of Exploration Drilling...10
1.1.12.14 Standards of Drill Hole Diameters and Core Sizes...10
1.1.12.15 Ground Vibration due to Blasting...11
1.1.12.16 Threshold Value of Ground Vibration...11-12
1.1.13 Subsidence in Mines...12-13
1.1.13.1 Introduction...12
1.1.13.2 Angle of Draw...12-13
1.1.14 Exercise...13-20
1.2 Underground Metalliferous Mining...20-28
1.2.1 Gestation Period...20
1.2.2 Cut-off Grade...20
1.2.3 Break-even cut-off grade...20
1.2.4 Raise or Winze...20
1.2.5 Raising Methods...20-21
1.2.5.1 Open Raising by drilling, blasting, mucking...20
1.2.5.2 Two or three compartment raising...20
1.2.5.3 Raising with Jora raise lift...20
1.2.5.4 Raising by Long Hole Drilling...20
1.2.5.5 Raising with Alimak raise climber...21
1.2.6 Sampling...21
1.2.7 Average grade of an ore block...21
1.2.8 Metal Mining Methods...22
1.2.8.1 Overhand stoping...22
1.2.8.2 Underhand stoping...22
1.2.8.3 Breast stoping...22
1.2.8.4 Sublevel stoping...22
1.2.8.5 Long-Hole stoping or Blast-Hole stoping...23
1.2.8.6 Room and Pillar Mining...23
1.2.8.7 Shrinkage stoping...23
1.2.8.8 Cut and Fill Stoping...23
1.2.8.9 Square Set Stoping...23
1.2.8.10 Resuing 23-24
1.2.8.11 Block Caving...24
1.2.8.12 Sub-Level Caving...24
1.2.9 Qualification and appointment of Competent persons in U/G metal mines...24-25
1.2.9.1 Manager...24
1.2.9.2 Assistant managers/under manager...24-25
1.2.9.3 Winding Engineman...25
1.2.10 MMR regarding Representative Factor...25
1.2.11 MMR regarding Ladders in u/g metal mines...25
1.2.12 Exercise...26-28
1.3 Opencast Mining Methods...28-39
1.3.1 Definition...28
1.3.2 Stripping Ratio...28-29
1.3.3 Break-even Stripping Ratio...29
1.3.4 Determination of Stripping Ratio...29-30
1.3.5 Parameters of the Working Bench...30-31
1.3.6 DGMS norms in Opencast Mining...31-32
1.3.7 Types of Ramps...32
1.3.8 Determination of number of faces required for a target production of 'y'...32-33
1.3.9 Determination of Powder Factor...33
1.3.10 Types of slope failure...33
1.3.11 Determination of Factor of Safety of the Slope...33-35
1.3.12 Pit Limit Optimization...35
1.3.13 Exercise...35-39
2.0 Rock Mechanics.....40-69

2.1.2 Geophysical prospecting...40
2.1.2.1 Seismic Method...40
2.1.2.2 Electrical Method...40
2.1.2.3 Gravity Method...40
2.1.2.4 Magnetic Method...40
2.2 Rock Quality Designation (RQD)...40-41
2.3 Physical and Mechanical Properties of Rocks...41-46
2.3.1 Unit Weight...41
2.3.2 Specific Gravity of Solids...42
2.3.3 Void Ratio...42
2.3.4 Porosity...42
2.3.5 Density...42-43
2.3.5.1 Dry density...42
2.3.5.2 Bulk density...42
2.3.5.3 Saturated density...42-43
2.3.6 Moisture Content...43
2.3.7 Degree of Saturation...43
2.3.8 Permeability...43
2.3.9 Anisotropy and Isotropy...43-44
2.3.10 Stress...44
2.3.11 Strain...44
2.3.12 Strain Energy...44
2.3.13 Modulus of Elasticity of rock...44-45
2.3.14 Poisson Ratio...45
2.3.15 Unit Volume Change...45-46
2.3.16 Deformability...46
2.3.17 Hardness...46
2.4 Classification of Rock Strength...47
2.4.1 Compressive Strength...47
2.4.2 Tensile Strength...47
2.4.3 Shear Strength 47
2.5 Seismic Velocities related to material properties...47-48
2.6 Rock Mass Strength Indices...48
2.6.1 Point Load Strength Index...48
2.6.2 Slake Durability Index...48
2.7 Testing of Rock Samples...49-50
2.7.1 Uni-axial Compressive Strength Test 49
2.7.2 Brazilian Test...49
2.7.3 Bending Test...49-50
2.7.4 Punch Shear Test...50
2.10 Mohr's Circle Analyses...51-53
2.11 Shearing on an Inclined Plane...54
2.12 Generalised Hoek-Brown Strength Criterion...54-55
2.13 Determination of Factor of Safety of the Slope...55-56
2.14 Stress and Displacement Distribution in Circular Excavation...56-57
2.15 Determination of Length of the Roof Bolt...57-58
2.16 Determination of mean shear strength using short anchor tests...58
2.17 Exercise...58-69
3.0 Mine Ventilation..... 70-91
3.1 Composition of Mine Air...70
3.2 Layers of Gases in Mine Atmosphere...70
3.3 Damps in u/g Mine Atmosphere...70-71
3.4 Permissible Concentrations of different Gases in Indian Underground Mines...71
3.5 Degree of Gassiness of a Coal Mine...71
3.6 Gas Monitoring Instruments...71-72
3.7 Coward Diagram...72-73
3.8 Dealing with Firedamp in Mines...73
3.9 Methane Layering Number...73-74
3.10 Graham's Ratio and its Importance...74
3.11 Rescue/ Escape/ Reviving Apparatus...74-76
3.12 Air Flow through the Mine Openings...76-77
3.12.1 Nature of Flow...76
3.12.2 Pressure Drop in Mine Airway...76-77
3.12.3 Shock Loss at Area Changes and Bends...77
3.12.4 Coefficient of Contraction...77
3.13 Equivalent Resistance...77-78
3.14 Air Power (AP)...78
3.15 Equivalent Orifice...78
3.16 Methods of Velocity Measurement...78
3.17 Natural and Mechanical Ventilation...78-84
3.17.1 N.V.P...78-79
3.17.2 Motive Column...79
3.17.3 Head (H_e) Generation by Mine Fan...79-82
3.17.4 Fan Laws...82
3.17.5 Ventilation Survey...82-83
3.17.6 Measurement of Velocity of Air Flow...83-84
3.17.7 Measurement of Quantity of Air Flow...84
3.17.8 Measurement of Relative Humidity...84

4.0 Mine Environment..... 92-114
4.1 Air Pollution...92-99
4.1.1 Definition...92
4.1.2 Air Pollutants Standards...92-93
4.1.3 Sources of Air Pollution in Mining Industries...93
4.1.4 Air Pollution Control Devices...93-98
4.1.5 Miner's Diseases and Dust Hazards...98
4.1.6 Dust Sampling Methods...98-99
4.2 Noise Pollution...99-104
4.2.1 Definition...99
4.2.2 Measurement of Noise...99-100
4.2.3 Equivalent Noise Level...100-101
4.2.4 Frequency Spectrum...101
4.2.5 Frequency Weighting...101-102
4.2.6 Sound Wave Propagation...102-103
4.2.6.1 Point Source...102
4.2.6.2 Line Source...102-103
4.2.7 Noise Pollution Standards...103-104
4.2.7.1 Worksite Standards...103
4.2.7.2 Ambient Standards...103-104
4.3 Water Pollution...104-105
4.3.1 Water Pollution Parameters...104-105
4.3.1.1 pH...104
4.3.1.2 Total Suspended Solids...104
4.3.1.3 Bio-chemical Oxygen Demand (BOD)...104-105
4.3.1.4 Chemical Oxygen Demand (COD)...105
4.3.2 Standards of Effluents Discharge from Coal Mining...105
4.4 Illumination...105-107
4.4.1 Definitions of Important terms of Photometry...105-106
4.4.1.1 Intensity...105
4.4.1.2 Horizontal Illuminance...105
4.4.1.3 Vertical Illuminance...105
4.4.1.4 Mean Spherical Candle Power (m.s.c.p)...105-106
4.4.1.5 Illumination...106
4.4.1.6 Lumen (lm)...106
4.4.1.7 Lux...106
4.4.1.8 Footcandle (fc)...106
4.4.2 Illumination Standards in Mines...106-107
4.4.2.1 Opencast Mining Areas...106-107

5.0 Mine Surveying.....115-155
5.1 Introduction...115
5.2 Representative Factor...115
5.3 Distance Measuring Instruments...115
5.4 Correction Factors in Line Measurement...115-117
5.4.1 Correction Factor Due to Incorrect Chain...115-116
5.4.2 Slope Correction Factor...116
5.4.3 Correction for Absolute Length...116-117
5.4.4 Correction for Temperature...117
5.4.5 Correction for Pull...117
5.4.6 Correction for Sag...117
5.5 Traversing 117-120
5.5.1 Bearing of Lines...118
5.5.2 True Meridians or Geographical Meridians...118
5.5.3 Magnetic Meridian...118
5.5.4 Arbitrary Meridian...118
5.5.5 Magnetic Declination...119
5.5.6 Designation of Bearing...119-120
5.5.6.1 Whole Circle Bearing...119
5.5.6.2 Quadrantal Bearing or Reduced Bearing...119-120
5.5.6.3 Fore and Back Bearing...120
5.6 Theodolite...121-126
5.6.1 Introduction...121-122
5.6.2 Magnification or Magnifying Power...122
5.6.3 Error in Theodolite Traverse...122-123
5.6.3.1 Permissible Linear Error...122-123
5.6.3.2 Permissible Angular Error...123
5.6.4 Relation between Angular and Linear Measurement...123
5.6.5 Latitude and Longitude of a Line...123-124
5.6.6 Close Traverse Survey...124
5.6.7 Computation of Areas of Irregular Boundaries ...124-126
5.6.7.1 Mid-Ordinate Rule 124-125
5.6.7.2 Average Ordinate Rule 125
5.6.7.3 Trapezoidal Rule 125-126
5.6.7.4 Simpson's Rule 126
5.7 Levelling...126-130
5.7.1 Introduction...126-127
5.7.2 Classification of Levelling...127-128
5.7.2.1 Differential Levelling...127

5.7.2.4 Cross Sectioning...127
5.7.2.5 Reciprocal Levelling...127-128
5.7.2.6 Barometric Levelling...128
5.7.2.7 Hypsometry...128
5.7.2.8 Trigonometric Levelling...128
5.7.3 Determination of Reduced Level (R.L)...128-129
5.7.4 Correction Factor in Levelling...129
5.7.4.1 Correction for Curvature...129
5.7.4.2 Correction for Refraction...129-130
5.8 Tacheometric Survey...130-132
5.8.1 Introduction...130
5.8.2 Method of Tacheometry...130-131
5.8.3 Determination of Horizontal and Vertical Distance in Different Conditions...131-132
5.9 Triangulation Adjustments...132-135
5.9.1 Definition of Various Observation and error Parameters...132
5.9.2 The Most Probable Values of Quantities...133
5.9.2.1 Direct Observation of Equal Weight...133
5.9.2.2 Direct Observation of Unequal Weight...133
5.9.2.3 Indirect Observation of Equal Weight...133
5.9.2.4 Indirect Observation of Unequal Weight...133
5.9.3 The Probable Error...133-134
5.9.3.1 Direct Observation of Equal Weight...133-134
5.9.3.2 Direct Observation of Unequal Weight...134
5.9.4 Laws of Weights...134-135
5.9.5 Computed Quantities...135
5.10 Correlation...136-137
5.10.1 Introduction...136
5.10.2 Weisbach Triangle Method...136-137
5.11 GPS...137-140
5.11.1 Introduction...137
5.11.2 How it works?...137
5.11.3 How accurate is GPS?...137
5.11.4 GPS Applications in Coal Industry...138-139
5.11.5 The GPS Satellite System...139-140
5.12 Photogrammetry...140-143
5.12.1 Introduction...140
5.12.2 Aerial Photogrammetry...140
5.12.2.1 Distortion Due to Height or Relief...141
5.12.2.2 Flight Planning...141
5.12.2.4 Intervals between Exposures...143
5.13 Exercise...143-155
6.0 Mining Machinery...156-182
6.1 Types of Clutches...156
6.2 Types of Gears...156-157
6.3 Wire Ropes...157-159
6.3.1 Introduction...157
6.3.2 Classification of Wire Ropes...157-158
6.3.2.1 On the Basis of Operation...157
6.3.2.2 On the Basis of Construction...157-158
6.3.3 Application of Wire Ropes in Mines...158
6.3.4 Space Factor...158
6.3.5 Mass of Rope...158-159
6.3.6 Strength of Rope...159
6.3.7 Rope Splicing...159
6.3.8 Capping of Rope...159
6.4 Winding System...159-163
6.4.1 Types of Winding System in U/G Mines...159-163
6.4.1.1 Drum Winding...159-162
6.4.1.2 Friction Winding or Koepe Winding...162-163
6.4.2 Pit Top Track Arrangement in Winding System...163
6.5 Underground Haulage System...163-166
6.5.1 Types of Haulage System...163-164
6.5.1.1 Direct Rope Haulage...163
6.5.1.2 Endless Haulage...163-164
6.5.1.3 Main and Tail Rope Haulage...164
6.5.1.4 Tail Rope Haulage...164
6.5.1.5 Gravity Haulage...164
6.5.2 Super-Elevation...164
6.5.3 Safety Devices Used in Haulage Arrangement...164-165
6.5.4 Power Requirement of the Motor to Run Haulage...165-166
6.6 Locomotive Transport...166-167
6.6.1 Introduction...166
6.6.2 Types of Locomotive...166-167
6.7 Conveyors Transport...167-168
6.7.1 Belt Conveyor...167
6.7.2 Cable Belt Conveyor...167
6.7.3 Disc Conveyors...167
6.7.4 High Angle Conveyor...168

7.0 Mine System Engineering..... 183-209
7.1 Linear Programming...183-185
7.2 Transportation Model...186-192
7.2.1 Types of Transportation Problem...186
7.2.2 Initial Basic Feasible Solution of a Transportation Model...186-189
7.2.3 Optimal Solution of a Transportation Model 189-192
7.3 Assignment Problem...192-195
7.3.1 Assignment Problem Solutions...192
7.3.2 The Hungarian Method...192-195
7.4 Network Analysis in Project Planning...195-198
7.4.1 Introduction...195-196
7.4.2 Methods of Determining the Duration of a Particular Project...196-198
7.4.2.1 Critical Path Method (CPM) or Critical Path Analysis (CPA)...196
7.4.2.2 Programme Evaluation and Review Technique (PERT)...196-198
7.4.3 Network Crashing...198
7.5 Reliability...198-201
7.5.1 Introduction...198-199
7.5.2 Statistical Concepts of Reliability...199
7.5.3 Reliability of a System...199-200
7.5.4 Mean Time Between Failure (MTBF)...200-201
7.6 Waiting Line Problem...201-202
7.6.1 Introduction...201
7.6.2 Characteristics of Queuing System...202
7.7 Work Study...203-204
7.7.1 Introduction...203
7.7.2 Basic Techniques of Work Study...203-204
7.7.2.1 Method Study...203
7.7.2.2 Work Measurement...203-204
7.7 Exercise...204-209
8.0 Mine Economics..... 210-218
8.1 Appraisal Techniques...210
8.2 Determination of Future Worth...210
8.3 Determination of Present Value...210-211
8.4 Payback Period...211
8.5 Rate of Return on an Investment...211
8.8 Profitability Index...212
8.9 Depreciation...212
8.10 Redemption...212
8.11 Depletion...212-213
8.12 Exercise...213-218
9.0 Mineral Dressing..... 219-226
9.1 Classification...219
9.2 Crushers...219-220
9.2.1 Introduction...219
9.2.2 Types of Crushers...219-220
9.2.2.1 Primary Crushers...219-220
9.2.2.2 Secondary Crushers...220
9.2.2.3 Tertiary Crushers or Impact Crushers...220
9.3 Grinding Mills...220
9.3.1 Tumbling Mills...221
9.3.1.1 Rod Mill and Ball Mill...221-222
9.3.1.2 Autogenous Mills...222
9.3.2 Stirred Mills...222
9.4 Principles of Classification in Liquid Medium ...222-223
9.4.1 Free Settling...222-223
9.4.2 Hindered Settling...223
9.5 Jigging...223-224
9.6 Dense Media Separation/ Heavy Media Separation...224
9.7 Froth Floatation...224
9.8 Washing...224
9.9 Exercise...225-226
10.0 Geostatistics..... 227-247
10.1 Arithmetic Mean...227
10.2 Median...227
10.3 Mode...227-228
10.4 Standard Deviation...228
10.5 Mean Deviation or the Average Deviation...228
10.6 Coefficient of Variation...228
10.7 Probability Distribution Function...229-230
10.8 Normal Distribution...230-231
10.9 Poisson Distribution...231
10.10 Exponential Distribution...231
10.11 Expectation of a Variate...231-232
10.12 Skewness...232

10.15.1 Additional Theory of Probability...233-234
10.15.2 Theorem of Total Probability for Compound Events...234
10.16 Bayes's Theorem...234
10.17 Determination of Average Grade of Ore in a Vertical Section...234-235
10.18 Tonnage Factor...235
10.19 Determination of Average Grade of Ore in a Horizontal Section...236-237
10.19.1 Triangular Method...236
10.19.2 Polygons...236
10.19.3 Constant Distance Weighting Technique...236-237
10.19.4 Inverse Distance Weighting Technique...237
10.20 Applications of Statistics on the Orebody...238
10.21 Trend of the Curve of Geostatistical Variance vs. Lag Distance...238-239
10.22 Quantification of Deposits through Variogram...239
10.23 Exercise...239-247

11.0 Selected Questions and Solutions from GATE 2008 and GATE 2009 248-290

Bibliography 291

Index 292-295

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1.0 Mining Methods

1.1 Underground Coal Mining

1.1.1 India's Rank in World Coal Production

According to the 2007 BP Statistical Energy Survey, world 2006 coal production was 6.2 billion tonnes. The world's largest coal producers were, in order, China, the USA, India, Australia, Russia, South Africa, Germany, Indonesia and Poland. Indian coal generally has a high ash content and low calorific value, so most coking coal must be imported. Major Indian coal fields are found in Bihar, West Bengal, and Madhya Pradesh.

1.1.2 Characteristics of Coal

The characteristics of different types of coal is given below in Table 1.1 :

Table 1.1 : Characteristics of Coal

Type of Coal	Rank	Depth of Burial	Maximum temperature during burial	Moisture Content	Fixed Carbon Content
Lignite	1	0.2-1.5 km	25-45°C	30-50%	20-35%
Sub bituminous	2	1.5-2.5 Km	45-75°C	10-30%	35-45%
Bituminous	3	2.5-6 km	75-180°C	5-10%	45-80%
Anthracite	4	> 6 km	>180°C	< 5%	80-96%

Lignite — Low Grade [Low calorific value, High V.M.]
Sub bituminous
Bituminous
Anthracite — High Grade [High calorific value, low V.M.]

1.1.3 Banded Constituents of Coal

- Vitrain
- Durain
- Clarain
- Fusain

1.1.4 Size Gradation of Coal

- Steam coal : 50 mm to 200 mm
- Rubble coal : 25 mm to 50 mm
- Slack coal : 0 mm to 25 mm

1.1.5 Analysis of Coal

1.1.5.1 Ultimate Analysis : Ultimate analysis of coal and coke is defined as the determination of the carbon and hydrogen in the material, as found in the gaseous products of its complete combustion, the determination of sulphur, nitrogen, and ash in the material as a whole, and the estimation of oxygen by difference.

1.1.5.2 Proximate Analysis : Proximate analysis covers the determination of moisture, volatile matter, fixed carbon, and ash in coals and cokes, and is used to establish the rank of coals, to show the ratio of combustible to incombustible constituents, or to provide the basis for buying/selling, and otherwise evaluating coal for various purposes. Thus, it is important to both the suppliers and users of coal to have a rapid, accurate, and reliable procedure for obtaining the proximate analysis.

1.1.6 Types of Coal

1.1.6.1 Non-Coking coal : The categorisation of non-coking coal is done on the basis of heating value of coal and is given below in Table 1.2.

Table 1.2 : Categorisation of non-coking coal

Grades	UHV Range in Kcals/kg
A	> 6200
B	5601 - 6200
C	4941 - 5600
D	4201 - 4940
E	3361 - 4200
F	2401 - 3360
G	1301 - 2400

[Source : D. J. Deshmukh, Vol. 1]

1.1.6.2 Coking Coal : The categorisation of coking coal is done on the basis of ash percentage and is given below in Table 1.3

Table 1.3 : Categorisation of coking coal

Grades	Range of Ash Percentage
Steel 1	<15
Steel II	15-18
Washery I	18-21
Washery - II	21 24
Washery - III	24-28

1.1.7 Caking or Agglutinating Index

The maximum ratio of sand to coal in a mixture which, after carbonization, gives a coherent mass capable of supporting a 500 g weight.

1.1.8 Protodyaknov Index

It is a simple empirical test for determining the strength of coal. The test consists basically of the crushing of a sample of 25 -75 gms, of fragments of coal larger than 10 mm contained in a hollow cylinder by five successive drop hammer blows. The index used for cuttability, drillability, amenability to hydraulic mining of coal seams or for design of the pillar. The Protodyaknov index of Indian coals varies from 1-4.

1.1.9 Composition of different Coal Burning Gases

The composition of various coal burning gases and its corresponding heating value is given in Table 1.4.

Table 1.4 : Composition of different Coal Burning Gases

Gas	Main Composition	Heating Value
Coal gas or coke oven gas	CO - 5.8; H_2 - 57.6; CH_4 - 29.6	4000-5000 kcal/m^3
Producer gas	CO - 20-30; N_2 - 50-60; H_2 - 8-20	1250-1600 kcal/m^3
Water gas	CO - 40.2; H_2 - 50	2600 kcal/m^3

1.1.10 Incubation Period of Coal

The time elapses in coal is 1st exposed to air and spontaneous fire of coal called the incubation period of coal.

1.1.11 Mining Methods

1.1.11.1 Bord & Pillar Mining

1.1.11.1.1 Dimension of Pillar and Galleries at different Depths : Dimension of pillar and galleries in u/g Bord & Pillar mining as per Regulation No. 99 of CMR 1957 is given below in Table 1.5.

Table 1.5 : Dimension of Pillar and Galleries

Depth of seam from the surface (m)	Width of galleries			
	< 3	< 3.6	< 4.2	< 4.8
	The distance between centres of adjacent pillars shall not be less than (in m)			

60 < D < 90	13.5	16.5	19.5	21.0
90 < D < 150	16.5	19.5	22.5	25.5
150 < D < 240	22.5	25.5	30.5	34.5
240 < D < 360	28.5	34.5	39.5	45.0
D > 360	39.0	42.0	45.0	48.0

1.1.11.1.2 Percentage of Extraction : Percentage of extraction is the ratio of volume coal extraction during development to total in-situ volume of coal. In Figure 1.1, the size of the pillar is a_1 (length) × a_2 (breadth) and the width of the galleries in dip and rise directions are b_1 and b_2 respectively. The percentage of extraction can be determined by the following formula.

$$R = \frac{(a_1+b_1)*(a_2+b_2)-a_1*a_2}{(a_1+b_1)*(a_2+b_2)}$$

Where, a_1 & a_2 are length and breadth of the pillar, b_1 & b_2 are the size of galleries in dip and rise direction

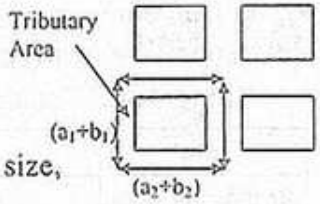


Fig. 1.1 B & P Development Panel

For a square pillar and same gallery size,

$a_1 = a_2 = a$ & $b_1 = b_2 = b$

$$R = \left[1 - \frac{a*a}{(a+b)*(a+b)}\right]$$

1.1.11.1.3 Tributary Area : It is the area of development gallery shown in Figure 1.1.

Tributary Area (TA) = $(a_1 + b_1) * (a_2 + b_2) - a_1*a_2$

1.1.11.1.4 Load Acting on Pillar : Load acting onthe pillar can be determined by

$$P = \gamma * D * \frac{1}{(1-R)} = \gamma * D * \frac{(a+b)^2}{a^2}$$

where, R = percentage of extraction,
D = depth of the pillar from surface
a = length and breadth of the pillar
b = width of the galleries
γ = weight per unit volume of the super-incumbent rock

1.1.11.1.5 Strength of Pillar : The strength of pillar can be determined by the empirical relationship given by Salamon and munro, 1967 as

$$S = K\frac{W^{\alpha}}{L^{\beta}} \text{ [Salamon and Munro, 1967]}$$

Where, S = Strength of the pillar in lb/in^2
K = constant (1320 lb/in^2)
W = width of the pillar, in ft.
L = height of the pillar, in ft.
α = 0.46 and β = 0.66

1.1.11.1.6 Factor of Safety of Pillar : Factor of Safety of Pillar is given by

F.O.S. = Strength of pillars (S) / Load actingon pillars (P)

$$= \frac{K*W^{\alpha}}{L^{\beta}} * \frac{(a)^2}{\gamma*D*(a+b)^2}$$

1.1.11.1.7 Line of Extraction :

- Diagonal line of extraction => Generally preferred as it facilitates good roof control
- Step diagonal line of extraction => It is preferred in panels worked in conjunction with hydraulic stowing as it facilitates water drainage
- Steep diagonal line of extraction
- Straight line of extraction

1.1.11.2 Hydraulic Mining : Hydraulic mining of coal comprises two operations viz hydraulic cutting of coal from the seam and the hydraulic transport of the coal and water mixture from the mine. The choice of the mining method depends on the following suitability conditions.

- Can be applicable in any seam thickness but moderate to high desirable for better economy.
- Gradient of 70° required for transport of materials in steel flumes but up to 40° material transportation possible with ultra high molecular weight plastics.
- Coal seam should be soft or ploughable.
- Water pressure of the monitor jet at the nozzle required for breaking of coal is in the range of 10-20 MN/m^2 (101-203 kg/cm^2).
- The cost of hydraulic transport is 10 times less than conventional transport.
- Air pollution problem is less.
- Productivity is higher.
- For pipe line transportations of coal the ratio of coal to pipe diameter should not be more than 1:3.

1.1.11.3 Auger Mining : A mining technique often used by surface mine operations when the overburden becomes too thick

mining methods. Large-diameter, spaced holes are drilled up to 200 ft (61 m) into the coal-bed by an auger. Like a bit used for boring holes in wood, this consists of a cutting head with screw like extensions. As the auger turns, the head breaks the coal and the screw carries it back into the open and dumps it on an elevating conveyor; this, in turn, carries the coal to an overhead bin or loads it directly into a truck. Auger mining is relatively inexpensive, and it is reported to recover 60% to 65% of the coal in the part of the bed where it is used.

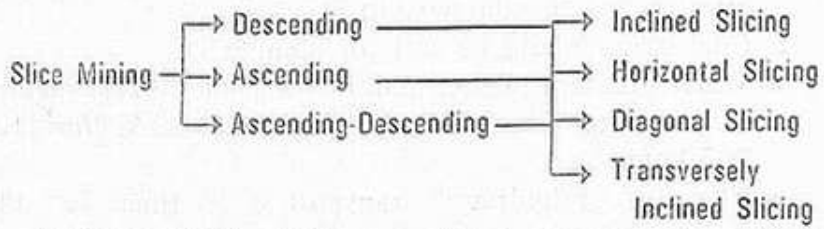
1.1.11.4 Contour Mining : Surface mining that progresses in a narrow zone following the outcrop of a coal seam in mountainous terrain, and the overburden, removed to gain access to the mineral commodity, is immediately placed in the previously mined area, such that reclamation is carried out contemporaneously with extraction.

1.1.11.5 Area Mining : A surface mining method that is carried on in level to gently rolling topography or relatively large tracts of land. Active area mine pits may be several miles long.

1.1.11.6 Strip Mining : Strip mining is usually associated with coal, where a seam or multiple seams are located at a certain elevation or elevations through a mountain or hill.

1.1.11.7 Thick Seam Mining Methods

1.1.11.7.1 Slice Mining : In slice mining, a coal seam is divided into slices of suitable thicknesses and individual slices are worked separately. Slices may be taken in both ascending order or in descending order or in mixed order. The different approaches of mining methods can be categorised as



1.1.11.7.2 Sublevel Caving : Sublevel caving is applicable to thick seams with caveable roof and soft coal. It can also be apply for the hard roof and hard coal, where caving is induced by blasting.

1.1.11.7.3 Integrated sublevel caving or Soutirage Mining : Recovering all the coal of the seam in a single operation from face progressing on the floor.

1.1.12 Explosives, Drilling and Blasting

1.1.12.1 Definition of Explosives : Explosive is a chemical compound or mixture, when exploded by action of heat, impact, gives large volume of gases in a very short time at high temperature & pressure.

1.1.12.2 Classification of Explosives

All commercial explosives are broadly divided in two categories :

- Low Explosives : The chemical reaction in low explosives is called deflagration which is a rapid process of combustion without accompanying any shock wave but gives a heaving effect.

  Gun Powder is a type of low explosive and its composition is: Sodium Nitrate - 72% Sulphur - 12% Coal - 16%
- High Explosives : Reaction in High Explosives is characterised by an associated shock wave initiated by a detonator. It basically contains : Oxidisers - Such as Ammonium Nitrate Fuel Oil Sensitizers, Methyle Amino Nitrate, Per chlorate Salt, Physical Sensitizer : Entrapped Air Bubbles

1.1.12.3 Permitted Explosives : The explosives incapable of igniting firedamp or coal dust explosion and can be used in u/g.

1.1.12.4 Types of Explosives : Various types of explosives with their brand name is given below in Table 1.6.

Table 1.6 : Types of explosives

Name of the explosives	Type of explosive	Brand Name
P1	Permitted (Unsheathed explosive)	Ajax G, Viking G, Godyne
P2	Non-Permitted in u/g (Sheathed explosive)	Not manufactured in India
P3	Permitted (Equivalent Sheathed explosive)	Unisax G, Permadyne
P4	Explosives approved for special purposes	Not manufactured in India
P5	Solid blasting explosive	Soligex G, Pentadyne

1.1.12.5 Characteristics of Explosives : The characteristics of various types of commercial explosives with respect to velocity of detonation (VOD), composition, form and size of particle is

Table 1.7 : Characteristics of Explosives

Explosive	Composition	Size	Form	VOD (km/s)
ANFO	Ammonium Nitrate (94 %) + Diesel Oil (6 %)	2 mm	All solid	3.2
Slurry	Oxidiser [$NH_4(NO_3)_2$/$NaNO_3$/ $Ca(NO3)_2$] + Cross linking agents (Potassium or sodium dichromate/ Antimony/ Boron) + Gel (Starch) + Fuel Sensitisers (TNT/ PETN)	0.2 mm	Solid/ Liquid	3.3
Emulsion	Micro droplets of a ammonium nitrate solution surrounded by fuel phase	.001 mm	Liquid	5-6

1.1.2.6 Powder Factor : This is defined as the tonne of coal produced per kg of explosive. For solid blasting it lies in the range of 1.5 to 2.5 te / kg and for cut blasting it is lies in the range of 4 to 6 te/kg.

This can be determined as

$$\text{Power Factor} = \frac{\text{Coal produced (tonne)}}{\text{Explosive required (kg)}}$$

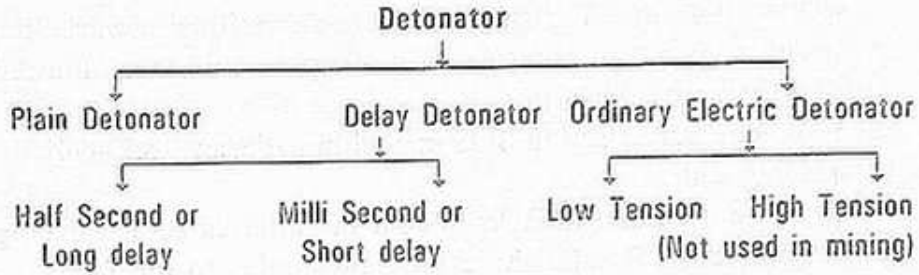
1.1.12.7 Maximum Permissible Charge in a Shot Hole : Maximum permissible charge in a shot hole as per DGMS Stipulations are given in Table 1.8.

Table 1.8 : Maximum Permissible Charge in a Shot Hole

Types of explosives	Degree of Gassiness/type of application	Max. permissible charge per shot hole (gms)
P1	Degree 1 mines, Cut face	800
P3	Degree 1, II & III mines, Cut face	1000
P5	Degree 1 mines, Cut blasting	1000
P5	Degree 1 mines, Solid blasting	565

Note : The coal must be pre-cut to provide a free face and the length of the shot holes must be 15 cm less then the length of the cut.

1.1.12.8 Detonator : Different types of detonators used for blasting in mines are classified as :



1.1.12.9 Types of Detonators and their Characteristics : Types of detonator and their characteristics are given below in Table 1.9.

Table 1.9 : Type of Detonators and their Characteristics

Detonator Characteristics	Plain Detonator	Ordinary Electric Detonator	Delay Detonator
Type	Non-delay (Instantaneous)	Non-delay (Instantaneous)	Delay
Base Charge	P.E.T.N*	P.E.T.N*	P.E.T.N*
Priming Charge	A.S.A**	A.S.A**	A.S.A**
Fired by	Ignited by Safety Fuse	Passing electric current through fuse head	Passing electric current through fuse head
Delay element	Not used	Not used	Copper or Brass sleeve
–	Ordinary	Low tension and High tension type	Low tension

*P.E.T.N : Penta-Erythritol Tetra-Nitrate

**A.S.A. : Lead Azide, Lead Styphnate and Aluminium Powder

1.1.12.10 DGMS Stipulations on Delay Interval :

- For non-incendive delay detonators in BOS (solid blasting), the maximum delay period between the 1st and last shot is 150 ms in degree I & II gassy coal mine and 100 ms in degree III gassy coal mine.
- The delay period between two consecutive shots with different delay numbers will not exceed 60 ms.
- Distance between two adjacent shots with different delay numbers will not come closer than 0.6 m at the explosive charge ends.

1.1.12.11 Drilling Pattern : The drilling pattern for blasting of coal/ rock depends on the hardness of coal/ rock and the types

- Pyramid cut : The shot holes are so inclined towards the centre as to appear to meet at one point and when blasted create a pyramid.
- Wedge cut : Used in BOS in coal in galleries B&P working
- Drag cut :
- Fan cut : Used in BOS in coal in galleries B&P working
- Burn cut : Parallel holes at right angles to the face.
- Coromat cut : Parallel holes at right angles to the face.
- Ring drilling : This is used in blasting galleries in coal mines and sublevel stoping in metal mines.

1.1.12.12 Drilling Method and their Rock Cutting Principles :

- Diamond drilling : Rotary abrasive
- Percussive drilling : Impact action
- Rotary roller drilling : Cutting and ploughing
- Jet piercing : Thermal spalling

1.1.12.13 Common Methods of Exploration Drilling : Various types of drilling with their specific applications are given below in Table 1.10

Table 1.10 : Types of Exploration Drilling

Types of drill	Size of drill hole	Economic depth	Application
Percussive drilling	50–75 mm	300 m	Soft and medium hardness rocks
Churn or Cable drilling	75–500 mm	300 – 600 m	Placer deposits, moderately soft formations
Diamond drilling	30–200 mm	100 -3000 m	Any rock formation except fissured
Calyx or chilled shot	75–1800 mm	< 450 m	Any rock drilling except hard

1.1.12.14 Standards of Drill Hole Diameters and Core Sizes : Standards of drill hole diameters with their core sizes are given below in Table 1.11.

Table 1.11 : Standards of Drill Hole Diameters

Standard	Drill rod outside diameter	Hole diameter	Core diameter
NX/ NW	67	75	54
BX/ BW	54	60	40
AX/ AW	44	47	28
EX/ EW	[illegible]	[illegible]	[illegible]

1.1.12.15 Ground Vibration due to Blasting : Ground vibrations is considered as one of the most undesirable side effects of rock residential buildings in the vicinity of mining operations and thus may lead to conformation and hamper normal production of the mine.

Broadly, ground vibrations may be controlled by using the following safety measures.

- Use Delay Detonators
- Optimization of the firing time by trials
- Proper Blast Design
- Lower Charge Per Delay
- Direction of Initiation

This prediction of ground vibration for long cylindrical charges in which any linear dimension should be scaled with square root of the charge weight and is given by the equation

$$V = K\left(\frac{D}{\sqrt{Q}}\right)^{E}$$

Where, V is the peak particle velocity, D is the distance of the measuring transducer and Q is the maximum charge weight per delay. K and B are site constants to be determined by regression analysis.

1.1.12.16 Threshold Value of Ground Vibration : DGMS (India) have specified permissible standards of peak particle velocity (in millimeter per second) depending on type of structure and dominant excitatory frequency. Permissible peak particle velocity at the foundation level of structure in mining areas is given below in Table 1.12.

Table 1.12 : Permissible peak particle velocity

Type of Structure	PPV in mm/sec at a foundation level of Structure at a frequency		
	<8Hz	8-25 Hz	>25Hz
Building structure not belonging to owner			
1. Domestic House structure kacha brick & cement	5	10	15
2. Industrial building RCC& Framed Structure	10	25	25
3. Objects of historical importance and			

Building belonging to owners with limited span to life			
1. Domestic houses structure Kacha bricks & cement	10	15	25
2. Industrial building RCC& Framed Structure	15	25	50

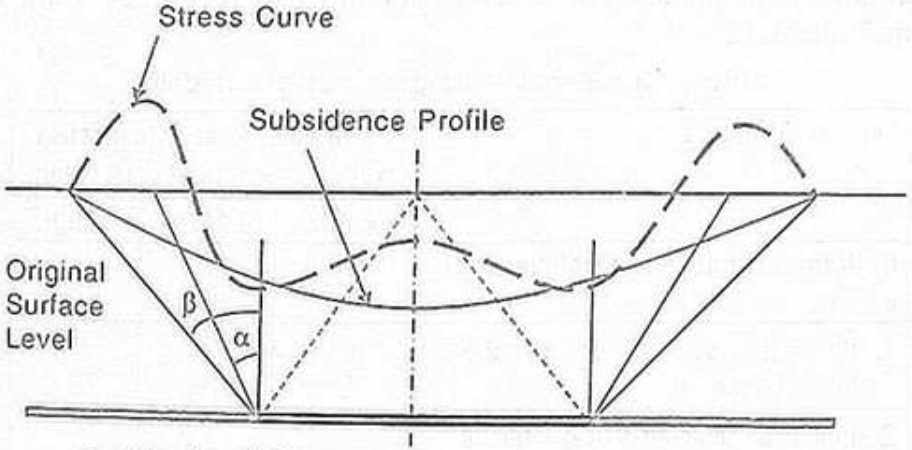
1.1.13 Subsidence in Mines

1.1.13.1 Introduction : The fractures extend vertically to the surface from the boundaries of the underground excavation.

Geometry of subsidence features over Bord & Pillar workings :

- Sinkholes : these are abrupt depressions in the ground surface that occur as a result of collapse of the overburden into an underlying mine void. the diameter of the sinkhole increases with depth.
- Troughs : these are dish shaped depressions at ground surface. In this case the overburden does not break to form a sinkhole.
- Ground Cracks : These are distinct vertical and horizontal displacements in the overburden.

1.1.13.2 Angle of Draw : This indicates the limit up to which subsidence takes place in the surface. The subsidence profile of three different situation viz. Critical Area, Sub-Critical Area, and Super-Critical Area are shown in Figure 1.2, Figure 1.3, and Figure 1.4 respectively.



α : Angle of fracture
β : Angle of Draw

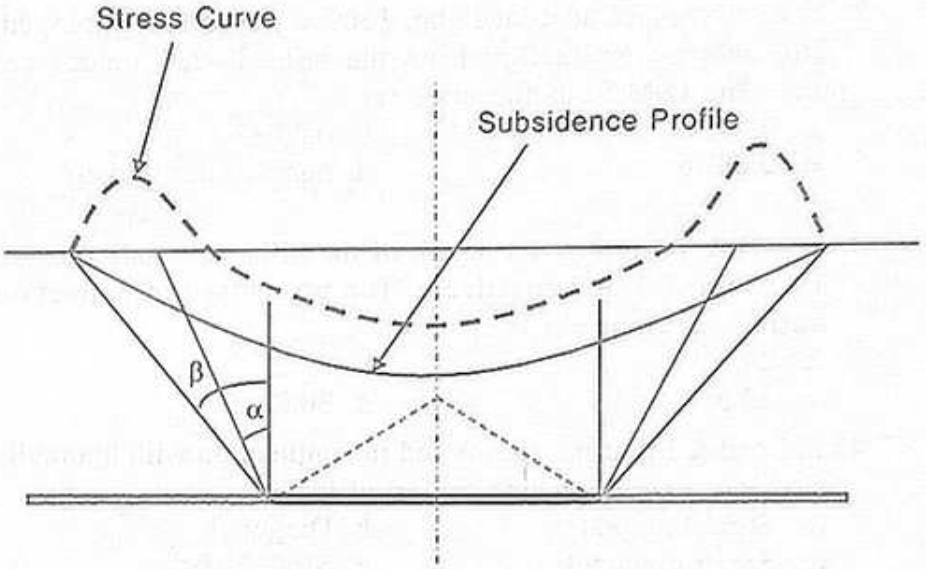


Figure 1.3 : Subsidence Profile of Partial Area or Sub-critical Area

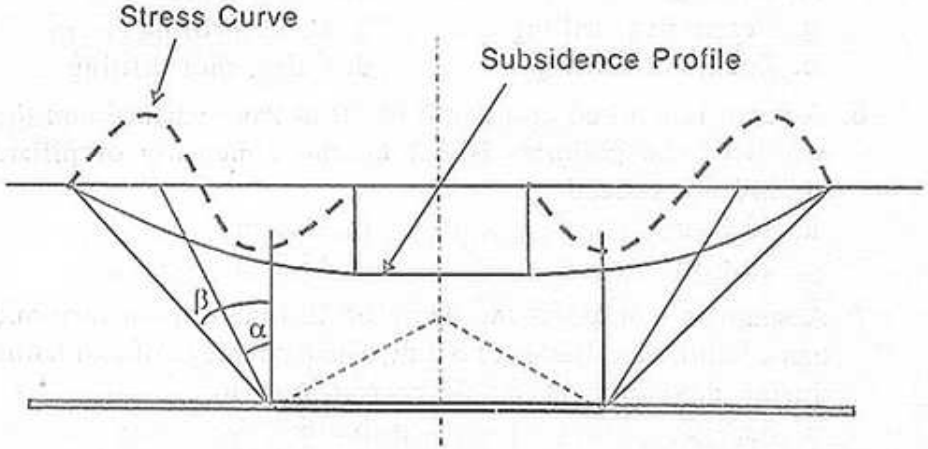


Figure 1.4 : Subsidence Profile of Partial Area or Super-critical Area

EXERCISE

1. A coal heading 4 m wide and 2.5 m high has an advance of 1 m per cycle. The amount of explosive used in blasting is 6 kg. Taking specific gravity of coal as 1.5, the powder factor is

   a. 1.66 te/ kg b. 2.50 te/ kg
   c. 2.99 te/ kg d. 3.32 te/ kg

   [GATE, 99]

2. In an underground coal mine, 1800 workers are employed. The average production from the mine is 900 tonnes per day. The O.M.S. of the mine is
   a. 0.24 te b. 0.60 te
   c. 0.65 te d. none of the above
   [GATE, 99]

3. In a B & P mining, the width of the pillar and galleries are 19.5 m and 3 m respectively. The percentage of extraction during development is
   a. 24.8 b. 25.2
   c. 29.3 d. 30.5

4. In Bord & Pillar panels worked in conjunction with hydraulic stowing, extraction line preferred is
   a. Step diagonal b. Diagonal
   c. Steep diagonal d. Straight line

5. The drilling method which can apply for most wider range of depth is
   a. Percussive drilling b. Cable drilling
   c. Diamond drilling d. Calyx shot drilling

6. A seam is worked at a depth of 70 m from surface and the width of the galleries is 3.2 m, the dimension of pillars should not exceed
   a. 15.0 m b. 16.5 m
   c. 19.5 m d. 13.5 m

7. A seam is worked at the depth of 120 m from surface and has a width of galleries of 3.4 m. The percentage of extraction during development should be restricted to
   a. 33.5 % b. 26.2 %
   c. 39.8 % d. 42.2 %

8. The average vertical stress in a coal pillar of a B & P panel is found to be 8 MPa. The depth of the pillar and the average density of overburden rock are 200 m and 2500 kg/ m^3 respectively. Assuming pillar stress to be the same in all coal pillars of that panel, the extraction ratio of the panel is
   a. 0.387 b. 0.513
   c. 0.613 d. 0.751

9. In a coal mine the depth of working is 200 m from surface, width of galleries 3 m, height of galleries 3 m and distance of square pillars between centres of bords is 22.5 m. What
   a. 128.53 kg/cm^2 b. 138.97 kg/cm^2
   c. 142.05 kg/cm^2 d. 139.97 kg/cm^2

10. In any working below ground there should be provided at least two shaft or incline which should be away from each other by a minimum distance
    a. 12.5 m b. 13.5 m
    c. 14.5 m d. 15.4 m

11. Maximum number of faces available for working in a shift in a 5 heading bord and pillar development district is
    a. 10 b. 12
    c. 13 d. 15
    [GATE, 2006]

12. Coalification has taken place in the order of
    a. Peat, lignite, bituminous, anthracite
    b. lignite, Peat, bituminous, anthracite
    c. Peat, lignite, anthracite, bituminous
    d. bituminous, Peat, lignite, anthracite

13. The method of mining a coal seam 8.5 m thick is by
    a. B & P method in lifts
    b. L/W methods in lifts
    c. A combination of B & P and L/W methods
    d. L/W sublevel caving

14. In horizon system of mining lateral are driven
    a. At right angles to the strike of the seam
    b. Parallel to the strike of the seam
    c. At 45° to the dip of the seam
    d. At any angle to the strike of the seam

15. Which one of the following HEMM is a tyre mounted?
    a. LHD b. SDL
    c. Dozer d. Shovel

16. Wagon drill working at minimum compressed air pressure of
    a. 6 kg/ cm^2 b. 7 kg/ cm^2
    c. 8 kg/ cm^2 d. 9 kg/ cm^2

17. Square pillars in B & P panel are of size 30 m centre to centre and the galleries are of width 4 m. The tributary area for each pillar in m^2 is
    a. 224 b. 256
    c. 676 d. 884

18. An underground panel in a coal mine is being developed on

B & P system. The pillar size is 30 m centre to centre and the adjacent galleries are 4 m wide. The percentage of extraction during development is

a. 17.6 b. 19.5
c. 24.9 d. 26.2

19. A bord and pillar panel will be developed with square pillars of size 25 m and an extraction ratio of 30 %. The width of the gallery in m is

a. 4.2 b. 4.6
c. 4.8 d. 5.3

20. The mining method used in thick and steep coal seam, where the slices are neither horizontal nor parallel to the inclination of the seam, is called

a. transverse slicing b. inclined slicing
c. horizontal slicing d. integral caving

[GATE, 2006]

21. Select the correct order with respect to the core sizes

a. AX > BX > EX > NX
b. NX > BX > AX > EX
c. AX > BX > NX > EX
d. EX > AX > BX > NX

22. Hydraulic mining of coal is normally applicable to mining seams, which are

a. thick and steeply inclined and have soft coal
b. thick and flat gradient and have soft coal
c. thin and flat gradient and have soft coal
d. thin and steeply inclined and have soft coal

23. For hydraulic transportation of solids in pipelines, the ratio of the size of solids to the pipe diameter should not be more than

a. 1 : 2 b. 1 : 3
c. 1 : 4 d. 1 : 7

24. The layout of the pipelines for hydraulic stowing should be such that the hydraulic profile

a. is above the geometric profile
b. is below the geometric profile
c. is on the geometric profile
d. there is no relation with the geometric profile

25. Contiguous seams means the parting between two seams is

a. 10 m b. 9 m
c. 8 m d. 12 m

26. For a delay blasting of 10 circuits, a sequential timer is set on 25 ms between circuit initiations. If the nominal firing time of a detonator is 400 ms, the total add on time in ms for 10 circuits will be

a. 650 b. 400
c. 625 d. 500

[GATE, 2002]

27. A shock tube initiating system, such as nonel

a. doesn't need detonators for initiation
b. It can be used in under-water condition
c. It is not effected by static electricity or strong currents
d. It creates lots of noise

[GATE, 1995]

28. In rock blasting the term "impedance of medium" is the product of

a. wave pressure and poisson's ratio
b. wave pressure and density of medium
c. propagation velocity and poisson's ratio
d. propagation velocity and density of medium

[GATE, 1998]

29. A coal heading 4 m wide and 2.5 m high has an advance of 1 m per blasting round. The amount of explosive used in the blasting round is 6 kg. If the sp. gr. of coal is 1.5, the powder factor is

a. 1.66 te/kg b. 2.50 te/kg
c. 2.99 te/kg d. 3.32 te/kg

[GATE, 1999]

30. The delay interval in short delay detonator is

a. 25 ms b. 30 ms
c. 15 ms d. 35 ms

31. Emulsion explosive is a mixture of

a. Ammonium nitrate, TNT, NG
b. Ammonium nitrate, TNT, guar gum
c. Ammonium nitrate, water, NG
d. Ammonium nitrate, water, hollow micro-balloons

32. To prevent slope failure the blasting system selected should be

a. Pre-split b. Muffled

33. The group of holes which are detonated on the first few delays are called
a. Easers
b. Trimmers
c. Cut holes
d. None of the above

34. The ratio of length of explosive column to the hole diameter which will work as a spherical charge, is
a. L/D < 6
b. L/D > 6
c. L/D < 4
d. L/D > 4

35. In case of an opencast mine blasting, the danger zone comprises of area within the diameter of
a. 400 m
b. 600 m
c. 500 m
d. 300 m

36. The place in mine, where explosive is stored is called
a. Shelter
b. Reserve station
c. Magazine
d. None of the above

37. The basic constituents of an emulsion explosive consist of oxidizers, fuels and surfactant with weight percentages of X, Y, Z respectively. Identify the correct relationship from the followings
a. X > Y > Z
b. Y > X > Z
c. Y > Z > X
d. X > Z > Y

38. In a drift of 2.4 m X 3.0 m size, the pull obtained per blast is 1.8 m. the total explosives consumed per blast is 36 kg. If the tonnage factor is 0.36 m3/te, the powder factor is
a. 0.5 kg/ te
b. 1.0 kg /te
c. 1.5 kg/ te
d. 2.0 kg/ te

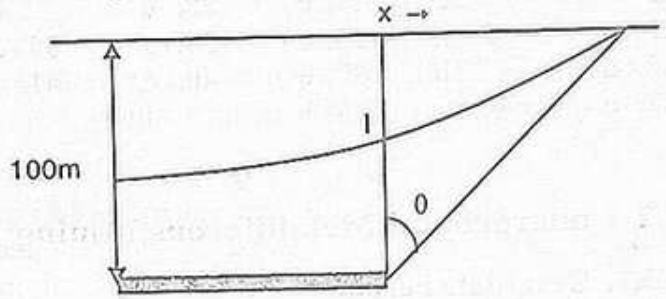
39. Reaction time of seismic detonators is
a. < 5 ms
b. > 10 ms
c. < 1 ms
d. < 1 ms & > 5 ms

40. At zero oxygen balance, the ratio of oxidizers to fuel ratio for ANFO is
a. 92 : 8
b. 93 : 7
c. 94 : 6
d. 95 : 5

41. Deflagration is
a. Rapid explosion
b. Explosion and burning
c. Rapid burning but not an explosion
d. None of the above

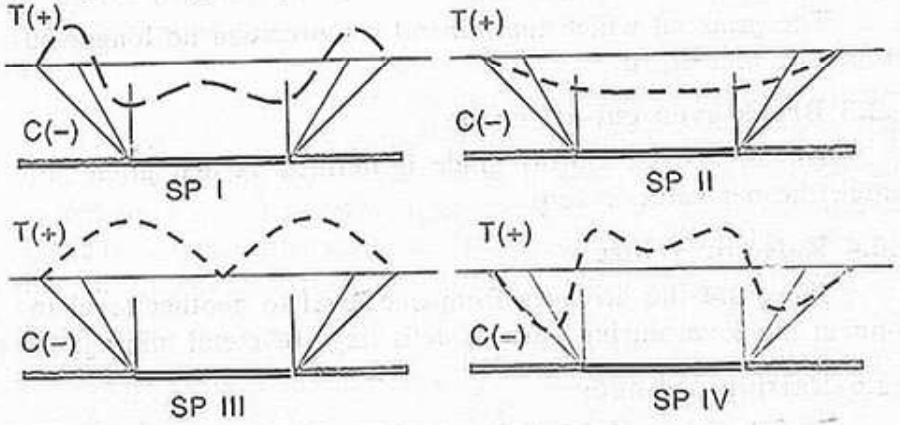
42. The ground in central portion of the subsidence trough is
a. lateral tension
b. vertical compression
c. lateral compression
d. none of the above

43. The subsidence profile over a longwall panel in a perfectly horizontal seam is given as, $s = 0.746 - 0.750 \tanh(0.05\,x)$
Where, s = subsidence at any point on the profile, m
x = horizontal distance measured from point of inflection (I) in the direction of decreasing subsidence, m
If the depth of the panel is 100 m, the angle of draw (θ) in degrees is
a. 15.4
b. 25.4
c. 30.6
d. 59.1



Half Panel Width

44. Lateral surface strain profile (SP) caused due to extraction of a longwall panel can be approximated by



a. SP I
b. SP II
c. SP III
d. SP IV

45. An underground panel extracted in a coal mine has a strike length of 500 m. After subsidence survey, it was found that the length of a straight line between the extreme limits of the subsided area on surface is 593.2 m along the strike direction. The depth of the extracted panel from the surface is 100 m. The angle of draw for the overlying strata assuming identical subsidence on either side of the centre of the panel is

a. 18° b. 25°
c. 35° d. 28°

Answer

1. b	2. d	3. a	4. a	5. c	6. c
7. a	8. a	9. d	10. b	11. c	12. a
13. a	14. a	15. a	16. a	17. b	18. c
19. c	20. a	21. b	22. a	23. b	24. a
25. b	26. c	27. c	28. d	29. b	30. a
31. d	32. a	33. c	34. a	35. c	36. c
37. d	38. b	39. c	40. c	41. c	42. a
43. c	44. a	45. b			

○

1.2 Underground Metalliferous Mining

1.2.1. Gestation Period

The time interval between the mining start and production of ore starts. This is very high in case of metal mines.

1.2.2 Cut-off Grade

The grade at which the mineral resource can no longer be processed at a profit

1.2.3 Break-even cut-off grade

The breakeven cut-off grade is defined as that grade for which the net value is zero

1.2.4 Raise or Winze

These are the drivages from one level to another level to connect the level during development stage of metal mining.

1.2.5 Raising Methods

1.2.5.1 Open Raising by drilling, blasting, mucking : Suitable for moderate length (upto 8 m), inclination of 40° - 60° and hard strata & country rock.

1.2.5.2 Two or three compartment raising : Suitable for vertical and very steep raises only and limited to 15-20 m.

1.2.5.3 Raising with Jora raise lift : This is mechanised and suitable for 100 m long but not used now.

1.2.5.4 Raising by Long Hole Drilling : The practical limit of this raising is 15- 50 m and the inclination is 35° from vertical.

1.2.5.5 Raising with Alimak raise climber : Suitable for greater than 40^0 inclinations from horizontal, length of the raise not more than 150 m.

1.2.5.6 Raise borers : Suitable for limited diameter of the raise (1-3.7 m) as the raise drill bits are available in these sizes. This is a method of drilling a pilot hole of 230 mm to 300 mm diameter and back reaming the hole of suitable sizes.

1.2.5.7 Drop raising method : Suitable for 50 – 60 m long raise and nearly vertical. The charging and blasting of the holes is based on Vertical Crater Retreat. Explosives charges used in crater method are normally spherical. Spherical charges have been defined as having a length: diameter (L: D) ratio of 6 : 1.

1.2.6 Sampling

The process of inferring something about a large group of elements by studying only a part of it is referred as sampling. There are mainly four types of sampling method in mines. These are Channel or Groove Sampling, Chip Sampling, Grab Sampling and Bulk Sampling.

Channel sampling in rock can be a difficult chore depending upon the hardness of the rock. This technique is generally used when sampling underground openings (drifts, tunnels, adits, and cross cuts) and trenches dug in rock. The best approach is to chip a sample of rock that is one half to one inch deep and six inches wide for the full length of the wall being sampled. The rock that is chipped off the wall should be collected as it is removed and stored in sample bags which represent a regular sample interval if the rock type is homogeneous. If there are changes in the rock type then the sample interval should correspond to the changing rock types.

1.2.7 Average grade of an ore block

The average grade of the block can be determined by

$$\text{Average grade of the block} = \frac{\sum L_i * w_i * g_i}{L * W}$$

Where, L = total length of the block

W = average width of the block

g_i = grade of ith section

The average grade determination of the ore block shown in Figure 1.5 is given by

⇒ Average grade of the block

$$= \frac{L_1 * w_1 * g_1 + L_2 * w_2 * g_2 + L_3 * w_3 * g_3}{}$$

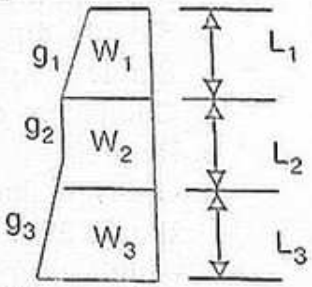


Figure 1.5 :

1.2.8 Metal Mining Methods

1.2.8.1 Overhand stoping

- The stoping operation proceeds from lower main level towards the upper main level.
- Applicable in steeply dipping narrow veins or in bedded deposits of 2-3 m thickness
- Strong ore and stable wall rocks
- The ore drilled by compressed air drills

1.2.8.2 Underhand stoping

- The stoping operation proceeds from upper main level towards the lower main level commencing from a winze.
- Applicable in thin steeply dipping (> 40°) veins.
- Strong wall rocks
- Transport of ore is difficult and the blasting efficiency is poor.

1.2.8.3 Breast stoping

- Applicable to deposits of horizontal or mild dip and of thickness up to 3 m (up to 3 m thickness mining can be possible with single lift but for more than that two lifts are required).
- Ore and hanging wall should be strong
- This method is of low cost, and preferred for low grade of ore
- Method lends itself to large scale mechanization in drilling, blasting, loading and transport.
- The worked out space supported by irregular pillars, timber cribs and props.

1.2.8.4 Sublevel stoping

- Sublevel means a level intermediate between the two main levels.
- The stoping operation proceeds from lower main level towards the upper main level.
- Steeply dipping ore and thick ore body
- Stable hanging wall and foot wall
- Competent ore
- Regular ore boundaries
- Ore is produced by drilling and blasting. Long-hole drilling should be of diameter in the range of 50-200 mm and length up to 90 m in ring-hole pattern or fan pattern.

1.2.8.5 Long-Hole stoping or Blast-Hole stoping

- Applicable to steeply dipping (> 50°) or vertical deposit.
- Involves large diameter (150 -200 mm) and 60-70 m long drill holes from upper level to lower main level.
- Drill holes are blasted in VCR (vertical crater retreat) method

1.2.8.6 Room and Pillar Mining

- Ore and hanging wall should be strong.
- Applicable to flat or mildly dipping (< 30°) ore bodies.
- The thickness may be in the range of 3-20 m.
- In case of thicker deposit excavation starts with the top section and the remainder of the ore is recovered by benching in one or more steps.

1.2.8.7 Shrinkage stoping

- Applicable to steeply dipping deposit.
- Ore should be of free flow characteristics.
- Ore that is not affected by storage in the stope (certain sulfide ores tend to oxidize and decompose when exposed to atmosphere).
- Suitable for thickness of ore body in the range of 3–12 m.
- Comparatively strong hanging wall and footwall.
- 40 % of the broken ore in the stopes has been drawn.
- Recovery of the ore varies from 75-90 %.

1.2.8.8 Cut and Fill Stoping

- Applicable to steeply dipping as well as mildly dipping ore body with firm ore.
- Applicable to small as well as large deposits.
- Offers selectivity
- Dilution of the ore is low
- Stoping operation is cyclic
- This method provides permanent support to surface structures

1.2.8.9 Square Set Stoping

- Square set stoping is a method of mining in which the walls and back of the stope are supported by regular frame work of timber called square sets.
- Applicable for very weak country rock and ore
- For recovery of fracture remnants
- Ore should be of high grade to compensate the costly and labour intensive method

of valuable ore and involves blasting of a sizeable quantity of barren wall rock in contact with the vein

- Steeply inclined deposits

1.2.8.11 Block Caving

- Steeply dipping deposits or massive ore body
- Weak ore to cave under its own weight
- Low value and grade of ore
- Surface is allowed to cave

1.2.8.12 Sub-Level Caving

- Sub-level caving is a caving method where the overburden and part of the ore is induced to cave in.
- Overburden and ore should be of weak in nature
- Steeply dipping deposit
- Fan drilling is suitable for extraction
- Dilution of the ore vary from 10 to 35 %
- Ore loss is in between 5 and 20 %

1.2.9 Qualification and Appointment of Competent Persons in U/G Metal Mines

1.2.9.1 Manager : The appointment of a manager in u/g metal mines done on the basis of no. of persons employed in below ground or in the entire mine. The qualifications required for appointment of manager is given in Table 1.13.

Table 1.13 : Qualification and Appointment of a Manager in u/g Metal Mines

No. of persons employed (X)	Qualification
X > 150 in below ground or X > 400 in all the mine	First Class Manager's Certificate
150 > X > 75 in below ground or 400 > X > 150 in all the mine	First Class/ Second Class
In any other case	First Class/ Second Class/ Manager's Permit

1.2.9.2 Assistant Managers / Under Manager : The number assistant manager required in u/g metal mines decided on the basis of no. of persons employed in below ground or in the entire mine. The number required for appointment of assistant manager given in Table 1.14.

Table 1.14 : Number of Assistant Manager in u/g Metal Mines

No. of persons employed (X)	Number of A.M./ U.M
450 > X > 600 in below ground or 1600 > X > 1200 in all the mine	1
X > 600 in below ground or X > 1600 in all the mine	1 additional A.M. for every additional 300 persons employed in u/g or 800 in all the mine or part thereof

Provided that in a mine where, X > 1200 in below ground or X > 3200 in all the mine, at least 1 A.M. shall hold First Class Managers Certificate.

1.2.9.3 Winding Engineman : The qualification required of a Winding Engineman in u/g metal mine is given below in Table 1.15.

Table 1.15 : Qualification of Winding Engineman in u/g Metal Mine

Type of Engine	Qualification
In case of electric engine of 75 h.p. or more Or Steam or compressed air winding engine which has cylinder exceeding 30 cm in diameter	First Class Engine Driver's Certificate
In any other case	Second Class Engine Driver's Certificate

1.2.10 MMR regarding Representative Factor

This is ratio of actual lease hold area to the area shown in the mine plan.

RF = 200 : 1, in case of mica mines and other mines having small scale working below ground

RF = 2000 : 1, in case of mine having large opencast workings and also in case of surface plans of large leasehold areas

RF = 1000 : 1, in other cases

1.2.11 MMR regarding Ladders in u/g metal mines

A. Project at least 1m above the mouth of the shaft winze, rise or stope

B. Inclination of the ladders should not exceed 80° from

EXERCISE

1. Exploration means
   (a) search of ore (b) estimation of reserve
   (c) opening up of deposit
   (d) shaft sinking to access the deposit

2. Ore grade for which revenue from the recoverable reserve exactly equals the cost of mining, treatment, and marketing is known as
   (a) cut-off grade (b) average grade
   (c) break-even grade (d) liquidation grade
   [GATE, 2000]

3. A Zn deposit is formed by a geological process is called
   (a) Hydrothermal (b) Magmatic separation
   (c) Supergene (d) None of the above

4. An ore block of length 30 m and height 20 m having an average width of 1.5 m with sharp boundary walls is mined with a minimum mining width of 2 m. If the tonnage factors are 3 t/m^3 for ore, 2 t/m^3 for wall rock and average grade of ore is 10 %, the diluted grade of ore mined in percent is
   (a) 7.50 (b) 7.84
   (c) 8.18 (d) 8.52
   [GATE, 2006]

5. A narrow vertical or inclined excavation made in the ore block along the width and the height of a stope to continue stoping is termed as
   (a) trough (b) slot
   (c) undercut (d) ore pass
   [GATE, 2006]

6. Gestation period is
   (a) the time interval between the mining start and production start
   (b) the time interval between the mining start and mining close
   (c) lag on ignition period
   (d) time to reach the breakeven point

7. The suitable method of mining a 2 m thick strong sulphide ore deposit within strong wall rocks dipping at 18° and occurring at 60 m below the surface is
   (a) overhand stoping (b) top slicing
   (c) underhand longwall stoping (d) breast stoping

8. The diameter of holes in mm commonly used for ring drilling by drifters in sublevel stoping is between
   (a) 25 and 32 (b) 33 and 44
   (c) 45 and 64 (d) 65 and 100
   [GATE, 2005]

9. High production rates coupled with large scale and extensive subsidence results from the method of
   (a) top slicing (b) block caving
   (c) sublevel caving (d) VCR mining
   [GATE, 2000]

10. Cut and fill stoping can be operated
    (a) underhand only (b) overhand only
    (c) both the above (d) none of the above

11. In sublevel stoping operation the draw system is generally
    (a) draw point (b) cross drift
    (c) finger raise and chute (d) none of the above

12. Blasting of stope in VCR method consists of
    (a) blasting one row after another
    (b) creating initial slot going for mass blast
    (c) blasting all the holes in slices
    (d) blasting one column after another

13. If the RQD of the ore and wall rock are low, the stoping method suitable is
    (a) shrinkage (b) sublevel
    (c) block caving (d) cut and fill

14. Broken rock of 3684 m^3 volume is heaped into a square pyramid of height 12m. the surface area in m^2 required is
    (a) 921 (b) 307
    (c) 614 (d) 1228

15. Order the followings in decreasing stability of host rock : Block caving (1), Cut & Fill (2), VCR stoping (3), Room & Pillar (4)
    (a) 1, 2, 3, 4 (b) 2, 3, 4, 1
    (c) 4, 3, 2, 1 (d) 4, 2, 3, 1

16. Dilution in sublevel caving is
    (a) 10 – 35 % (b) > 40 %
    (c) < 10 % (d) < 5 %

17. Every ladder way used in a mine shall project above the

(a) 1.0 m (b) 1.5 m
(c) 2.0 m (d) 0.5 m

18. Main difference in VRM and Blast Hole stoping is
(a) Drilling technique (b) Blasting technique
(c) Drilling and Blasting technique
(d) None of the above

19. Best suited mining method for thick vein/ massive deposit with strong ore, strong walls and steep dip is
(a) open stope (b) cut and fill
(c) shrinkage stope (d) block caving

20. A block ore between two levels 30 m apart is 40 m long. The ore body thickness, specific gravity and grade are 2 m, 2.8 and 2 % respectively. The tonnage of copper available in the block of ore is
(a) 134.4 (b) 236.2
(c) 156.8 (d) 290.3

21. In-situ ore of 1.5 % Cu got diluted with waste rock of value 0.2 % Cu. If the resultant extracted ore has 1.2 % Cu. What is the percentage of dilution?
(a) 30 (b) 25
(c) 23 (d) 20

Answer

1. b	2. c	3. a	4. c	5. b	6. a
7. d	8. d	9. b	10. a	11. a	12. c
13. d	14. a	15. a	16. a	17. a	18. b
19. c	20. a	21. a			

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1.3. Opencast Mining

1.3.1 Definition : In this type of mining, all the layers cover the mineral block/ coal extracted to make the access to mineral/ coal deposit. The overburden material extracted dump in the area not bearing the mineral/ coal block. This type of mining method also called open-pit mining, surface mining and strip mining. Selection of the opencast mining method depends on the stripping ratio.

1.3.2 Stripping Ratio : It can be defined as the ratio of volume of overburden removed to weight of recoverable mineral overburden because the depth increases, cost of extraction of the overburden material increases.

$$\text{Stripping Ratioo} = \frac{\text{Volume of overburden in cubic meter}}{\text{Weight of recoverable mineral reserve in tonne}}$$

1.3.3 Break-even Stripping Ratio : The point beyond which the mineral/coal cannot be economically extracted out is called the break-even stripping ratio. It can be also defined as the ratio of excess cost required to produce per tonne mineral/coal by underground mining method in comparison to the opencast mining method to the stripping cost per cubic meter of overburden by the opencast method.

$$\text{Break-even Stripping Ratio} = \frac{\text{Cost of extraction per cubic meter of overburden material}}{\text{(Recoverable price per te of ore) – (O.C. production cost per te of ore + Profit margin per te of ore)}}$$

1.3.4 Determination of Stripping Ratio : The stripping ratio of the block shown in Figure 1.6 and Figure 1.7 can be determined as given below.

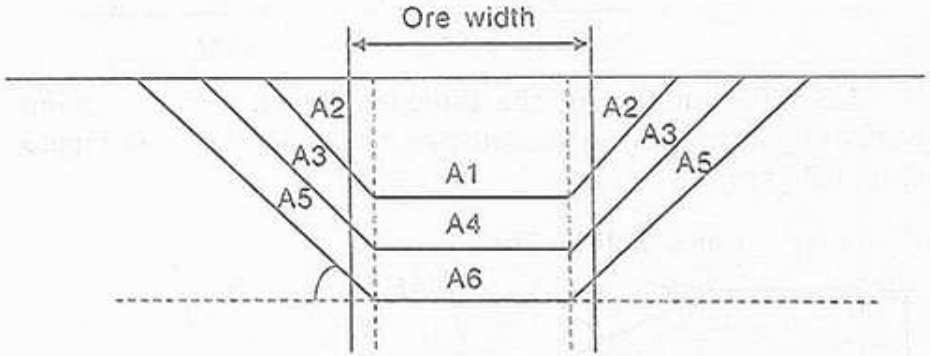


Figure 1.6 : Excavation of Ore Block

$$\text{Overall Stripping Ratio (SR)} = \frac{Vw * \rho w(waste)}{Vo * \rho(ore)}$$

$$= \frac{A_1 + 2A_2 + 2A_3 + 2A_5}{A_4 + A_6}$$

Where, A_1, A_2, A_3, A_4, A_5, A_6 : area of cross section of zone shown in the above figure.

V_w = volume of waste material
ρ_w = density of waste material
V_o = volume of overburden material
ρ_o = density of overburden material

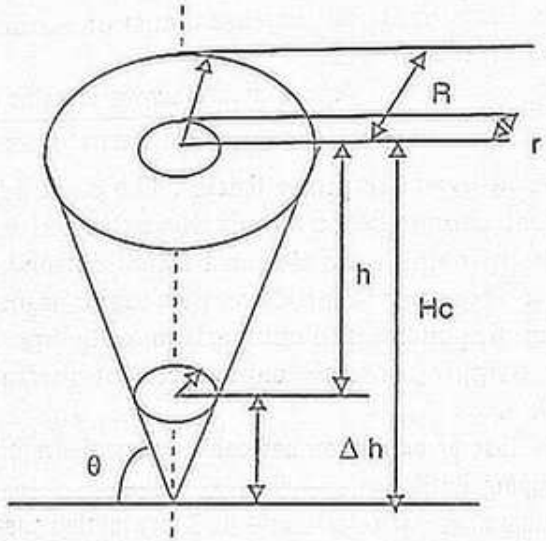
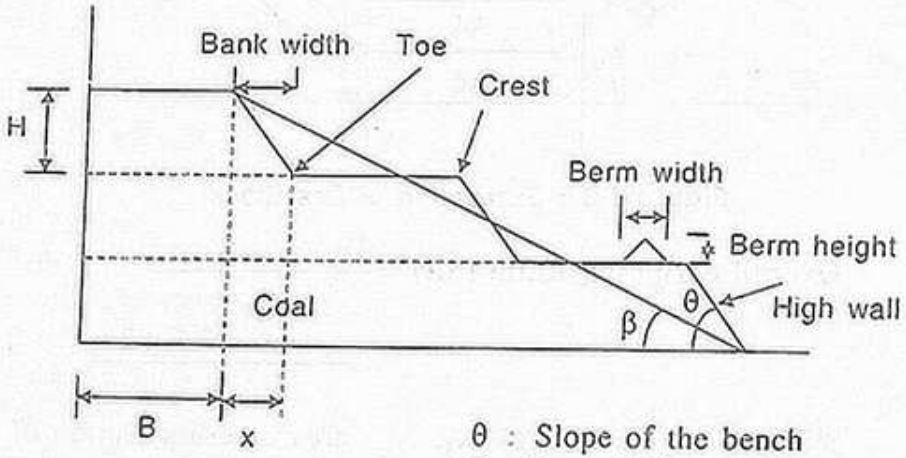


Figure 1.7 : Excavation of Ore Block

Overall Stripping Ratio (SR)

$$= \frac{v_w * \rho_w(waste)}{v_0 * \rho_0(Ore)} = \frac{\left(\frac{1}{3}\pi R^2 H_c - \frac{1}{3}\pi r^2 \Delta h\right) * \rho_w}{\frac{1}{3}\pi r^2 \Delta h * \rho_0}$$

1.3.5 Parameters of the Working Bench : In a working bench of opencast mine, the basic parameters are shown in Figure 1.8 and Figure 1.9



θ : Slope of the bench
β : Overall Slope angle
B : Bench width
H : Bench height
x : Bank width

Figure 1.8 : Working Bench of Opencast Mine

$$\text{Overall Slope Anlge } (\beta) = tan^{-1}\frac{3H}{2B+3x} = \frac{3H}{2B+3\frac{H}{tan\theta}}$$

Definition of the terms

- **Spacing (S) :** Distance between the two holes in the same row called spacing
- **Burden (B) :** Distance between the crest of the bench and 1st row of drill holes or the distance between the two rows of holes called burden.
- **Height of the bench (H) :** Vertical distance between the benches called the bench height.
- **Width of the bench :** This is the horizontal gap between the toe of the upper bench to the crest of the lower bench. This should be always greater than the height of the bench.
- **Safety Berms :** This can be referred to the piles of rock material near the crest of the bench to improve the safety the mine safety.
- **Bank width :** This is the horizontal projection of the slope face.
- **Ramps :** This is the passage to make the access from one level to another level. Maximum permissible gradient of this passage is 1 in 10.

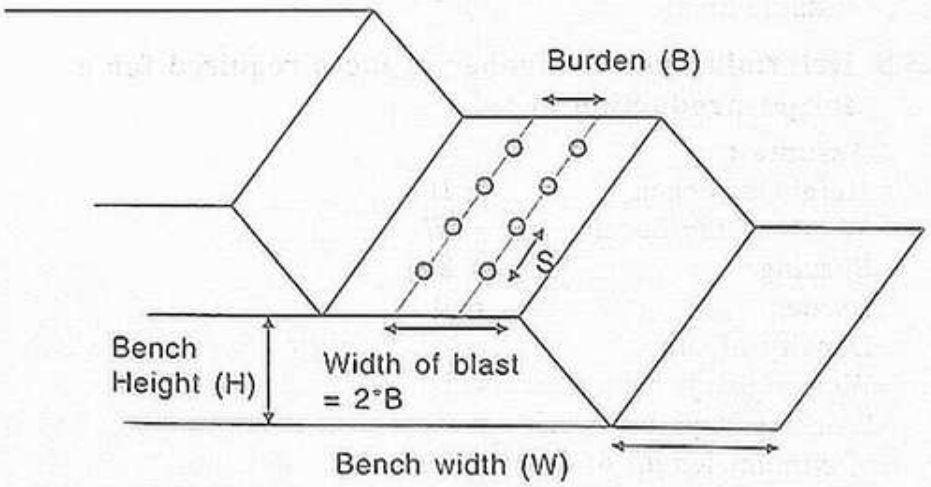


Figure 1.9 : Working Bench of Opencast Mine

1.3.6 DGMS norms in Opencast Mining

- In alluvium soil, morum, gravel, clay, debris or other soft ground slope of the benches shall not exceed 45° from the horizontal or the sides shall be kept benched and the

- In coal, slope of the benches shall not exceed 45° from the horizontal or the sides shall be kept benched and the height of the bench shall not exceed 3 m and the breadth shall not be less than the height.
- In alluvium soil, morum, gravel, clay, debris or other soft ground, if any pillar is left for the purpose of measurement, its height shall not exceed 2.5 m and where the height of such pillar exceeds 1.25 m, the base of the pillar shall not be less than 1.5 m in diameter.
- Maximum gradient of the haul road is 1 in 14.
- The permissible length of the trailing cable in opencast mines is 300 m as per IER.

1.3.7 Types of Ramps

- Spiral : It is not suitable where there is a problem of slope stability. But this layout is preferable when the pit wall is very steep.
- Switch-back : This is not spiral in nature but the road turns 180° at the end of the extent.
- Zigzag : There is not any uniformity in this type of layout. This type of layout changes the path to make the access to the lower level as per the availability of the suitable/ stable area.

1.3.8 Determination of number of faces required for a target production of 'y'

Assume :

Height of bench	= H
Width of the bench	= W
Spacing	= S
Burden	= B
Density of ore	= γ
Working days per year	= D
Working days per week	= 6
Maximum length of face	= L

Two rows of blasting in a round [Total width of blast = 2*B]

Production per hole = H * B * S * γ

$$\text{Production required per week} = \frac{Y*6}{D}$$

Length of face required to accommodate above number of holes

$$= \frac{\left(\frac{Y*6}{D}\right)}{H*B*S*\gamma} * \frac{Spacing\ (S)}{Number\ of\ rows\ (2)}$$

$$= \frac{Y*6*S}{2*H*B*S*\gamma*D} = \frac{Y*3}{H*B*\gamma*D}$$

For production at shovel face another face of the same length is required for drilling at the same time.

$$\text{Length of the face} = \frac{Y*3*2}{H*B*\gamma_1*D}$$

Again for grade control another face length of the same size required

$$\text{Total face length required} = \frac{Y*3*2*2}{H*B*\gamma*D}$$

Number of bench required

$$= \frac{Y*12}{H*B*\gamma*D} * \frac{1}{L} = \frac{Y*12}{H*B*\gamma*D*L}$$

1.3.9 Determination of Powder Factor

$$\text{Powder Factor (PF)} = \frac{\text{Production per hole (te)}}{\text{Explosive consumption per hole (kg)}}$$

$$= \frac{H*b*S*\gamma}{\left(\frac{\pi*d^2}{4}\right)*(H-h)*\gamma_e}$$

Where, γ_e = specific gravity of explosive
h = length of stemming
d = diameter of the blast hole

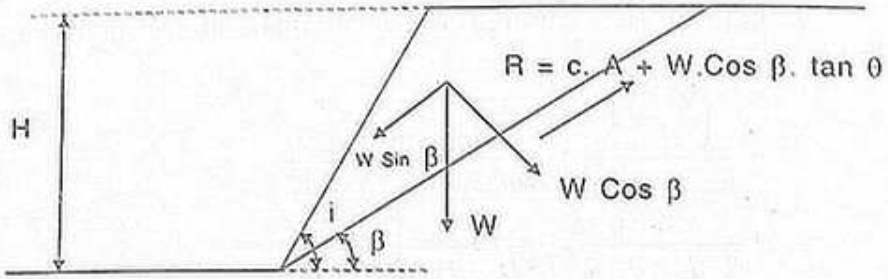
1.3.10 Types of slope failure

There are four types of slope failure occurs in the mining areas. These are as follows :

- Circular failure
- Plane failure
- Wedge failure
- Toppling failure

1.3.11 Determination of Factor of Safety of the Slope

CASE I : When there is no tension crack and water



Assuming

Height of the bench	= H
Overall slope angle from horizontal	= i
Angle of discontinuity from horizontal	= β
Weight of the block	= W
Friction angle	= φ
Cohesion	= c
Area of the failure plane	= A

Total force tending to induce sliding of the block

= W. Sin β

Total force to resist sliding of the block

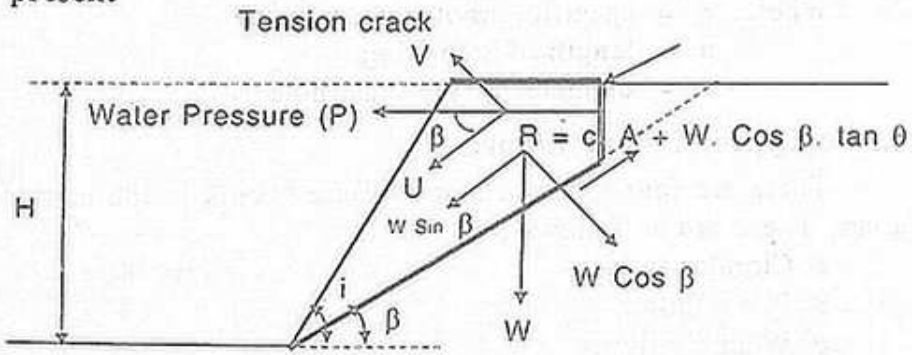
= W. Cos β. tan φ + c. A

[Neglecting the water pressure]

$$\text{Factor of Safety (F.O.S.)} = \frac{\text{Total force to resist sliding}}{\text{Total force tending to induce sliding}}$$

$$\Rightarrow \quad F.O.S. = \frac{c.A + W.Cos\beta.tan\varphi}{W.Sin\beta}$$

CASE II : When there is tension crack and water pressure present



Water pressure along the sliding plan = U = P. Cos β

Water pressure normal to the sliding plane = V = P. Sin β

Total force tending to induce sliding of the block

= W. Sin β + U

Total force to resist sliding of the block

= (W.Cos β − V). tan φ + CA

[Neglecting the water pressure]

$$\text{Factor of Safety (F.O.S.)} = \frac{\text{Total force to resist sliding}}{\text{Total force tending to induce sliding}}$$

$$\Rightarrow \quad F.O.S. = \frac{c.A + (W.Cos\beta - V).tan\varphi}{W.Sin\beta + U}$$

1.3.12 Pit Limit Optimization : Optimization techniques are as follows :

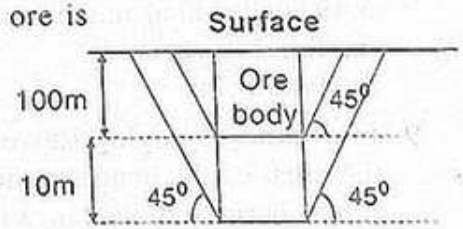
- Heuristic technique
- Dynamic programming
- Linear Programming (Integer Programming)
- Network flow theory
- Graph theory

EXERCISE

1. Which one of the following term is not related to opencast mining?
   a. Boxcut b. Highwall
   c. Crest d. Winze

2. As per MMR 1961, mine plan of a mine having large opencast working shall be on a scale having a representative factor of
   a. 200:1 b. 2000:1
   c. 1000:1 d. 4000:1

3. A vertical ore body having a consistent width of 100 m has been excavated by the open pit method up to a depth of 100 m at a pit slope angle of 45°. The figure shows the cross section of the open pit. If the ore body is further excavated by a depth of 10 m, maintaining the same pit slope angle, the ratio of waste rock to ore is
   a. 1.2
   b. 2.1
   c. 3.0
   d. 3.9



4. An opencast mine has four benches. The height, width and face angle for each bench are 13 m, 30 m and 70° respectively. The overall slope angle of the benches in degrees is
   a. 25.5° b. 20.7°

c. 33.4° d. 19.7°

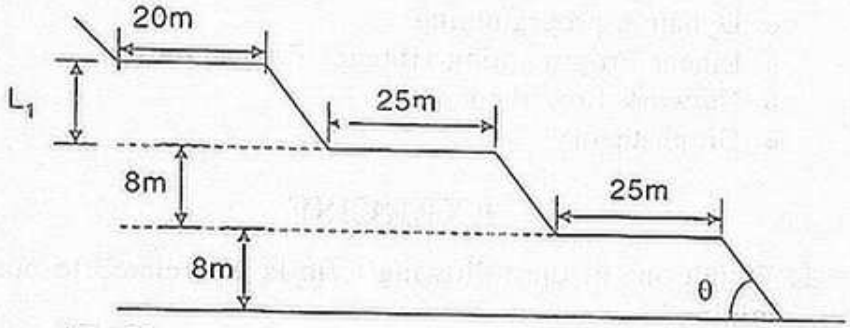
[GATE, 01]

5. Four benches are being worked by the opencast mining system. Height, width and face angle for each bench are 15 m, 50 m and 70° respectively. The overall slope angle of the benches in degrees is

a. 15.45° b. 19.25°
c. 32.65° d. 36.25°

[GATE, 07]

6. What will be the overall slope of the benches shown in the figure?



a. 17.45° b. 18.56°
c. 27.56° d. 43.55°

7. In a mine for one shovel six trucks are assigned. The shovel loading time per truck is 5 min, and truck cycle time 20 min. Calculate the match factor.

a. 0.66 b. 1.00
c. 1.20 d. 1.50

8. For an open pit mining the value of metal is Rs. 210 /kg, and recoverable grade is 1.2 %. Production cost per tonne of ore inclusive of mining and processing but excluding stripping is Rs. 2000. If the break even stripping ratio is 3.49 m^3/te, then the stripping cost is

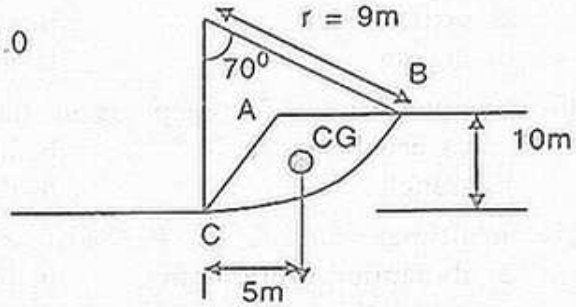
a. Rs. 149 /m^3 b. Rs.72 /m^3
c. Rs.60 /m^3 d. Rs.49 /m^3

9. In a mine deploying shovel and dumper, the distance from the crest of the bench to the centre of the dumper including safety berm and clearance is 7.5 m, the dumping radius and the cutting radii of the shovel are 15 m and 12 m respectively. The width of the working bench of the mine in m is

a. 49.5 b. 46.5

10. For a topsoil bench in an opencast mine as shown in the figure, the weight of the sliding block of unit width, ABCA, is 500 kN and the cohesive strength of the material is 20 kN/m^2. Assuming the surface to be totally frictionless, the factor of safety (FOS) for the stability of the slope is

a. FOS < 0.5
b. 0.5 < FOS < 1.0
c. FOS = 1.0
d. FOS > 1.0



11. A haul ramp of 6 % grade is to be constructed connecting a bench of 15 m height to the surface. If the cost of ramp construction is 1 million rupees per km, the cost of this ramp construction in million rupees will be

a. 0.25 b. 0.10
c. 2.50 d. 0.50

12. Maximum permissible gradient for the haul road in opencast mines is

a. 12° b. 13°
c. 14° d. 15°

13. Dumper arrives at a shovel at a constant rate of 10 per hour. The service rate is constant at 15 per hour. At a time, the expected number of dumpers in the queue is

a. 2 b. 3
c. 5 d. 4

14. A 15 m^3 dragline is deployed in an O.B. bench of an O.C. mine. It works for 40 days at the rate of 6 hrs per shift and 3 shifts a day. The cycle time, bucket fill factor, and operating efficiency of the dragline are respectively 50 s, 0.8, and 75 %. The total volume of O.B. in m^3 handled by the dragline is

a. 356918 b. 365918
c. 532158 d. 466560

15. In case of an opencast blasting the danger zone comprises of area within the radius of

a. 300 meters from the place of firing

c. 500 meters from the place of firing
d. 600 meters from the place of firing

16. In case of an O.C. mine, if the pit wall are too steep to allow suitable bends the layout of haul road geometry selected should be
a. switch back b. spiral
c. zigzag d. none of the above

17. To open a mine for deep seated, the cut used is
a. external box cut b. internal box cut
c. trench d. deep cut

18. In ultimate pit analysis the most widely used method is
a. dynamic programming b. linear programming
c. simulation d. heuristic technique

19. For O.B removal with shovel – dumper combination, the optimum size of the dumper depends on
a. distance of the haul b. size of the shovel
c. distance of the haul and size of the shovel
d. tonnage to be handled

20. In shovel loading of broken iron ore, if loading cycle time of the shovel of 5 m^3 bucket capacity, is 60 seconds, bucket fill ratio is 0.8, rock looseness factor is 1.5 and shovel working time utilization factor is 0.6, then the hourly output of the shovel in m^3/hr will be
a. 92 b. 94
c. 96 d. 97

21. In an overburden bench of an iron ore mine the following data were found:

Height	= 10 m
Burden	= 4 m
Spacing	= 3 m
Sub-grade drilling length	= 1m
Collar stemming length	= 3 m

If the diameter of the explosive column is 150 mm and the specific gravity of explosive and that of rock are 0.9 and 2.4 respectively, the powder factor in te/kg will be
a. 3.29 b. 4.98
c. 2.26 d. 4.53

22. From a hard rock open pit mine, the following data available

Bench height	: 12 m
Sub-drilling	: 1 m
Collar stemming	: 4 m
Diameter of hole	: 150 mm
Density of rock	: 2500 kg/m^3
Density of explosive	: 800 kg/m^3

The powder factor in kg/t is
a. 0.314 b. 0.353
c. 2.833 d. 3.183

[GATE, 06]

23. The feasibility study of an opencast mine revealed the following :

Recoverable value per tonne of ore	= Rs. 900
Production cost per tonne of ore	= Rs. 550
Stripping cost per tonne of waste	= Rs. 45

The break even stripping ratio will be
a. 7.77 b. 32.20
c. 1.78 d. 12.22

24. A tension crack of 5 m depth behind a rock slope crest is completely filled with water. What is the horizontal force exerted on a unit slope length by the water in the tension crack.
a. 122.60 kN b. 245.00 kN
c. 12.26 MPa d. 24.50 Nm

Answer

1. d	2. d	3. a	4. a	5. b	6. a
7. d	8. a	9. d	10. a	11. a	12. c
13. a	14. d	15. c	16. b	17. b	18. d
19. c	20. c	21. c	22. a	23. a	24. a

□

2.0 Rock Mechanics

2.1 Methods of Rock Exploration

2.1.1 By direct penetration

Direct penetration into the rock mass helps in direct observation of the structure and characteristics of the underlying strata. This can be achieved by core boring and recovery.

2.1.2 Geophysical prospecting

By making certain physical measurements from the surface without penetration the characteristics of the underlying strata can be estimated. There are four principle methods of geophysical exploration :

2.1.2.1 Seismic Method : This methods works on the basis of elastic properties and densities of the rocks. Seismic exploration is done by Reflection (for > 600m depth) and Refraction (for shallow depth).

2.1.2.2 Electrical Method : This method works on the principle of variation in electromagnetic field and resistivity of the underlying rocks.

2.1.2.3 Gravity Method : The basic theory involved in this method is the Newton's Law of Gravitation. this method works on the basis of variationin density of rock masses.

2.1.2.4 Magnetic Method : This method works on the basis of variation in magnetic properties of rock masses.

2.2 Rock Quality Designation (RQD)

This is the ratio of the sum of the length of the cores greater than 10 cm to the total length of the core run. This parameter represents the weakness of the rock. The classification of rock sample on the basis of RQD is shown in Table 2.1.

$$RQD\ (\%) = \frac{\text{(Sum of the length of the cores greater than 10 cm)}}{\text{(Total length of the core run)}} \times 100$$

Table 2.1 : Classification of Rock

RQD (%)	Description
0-25	Very poor
25-50	Poor
50-75	Fair
75-90	Good
90-100	Excellent

- RQD could be obtained from discontinuity spacing measurements made on core or an exposure using the equation

  $$RQD = 100e^{-0.1\lambda}(0.1\lambda + 1)$$

  Where λ = the average number of fractures per meter of core length

  For values of λ in the range 6 to 16/m, a good approximation to measured RQD values was found to be given by the linear relation

  $$RQD = -3.68\lambda + 110.4$$

- The *RQD* may be estimated from the number of discontinuities per unit volume. The suggested relationship for clay-free rock masses is:

  $$RQD = 115 - 3.3\ Jv$$

  Where, Jv is the sum of the number of joints per unit length for all joint (discontinuity) sets known as the volumetric joint count.

2.3 Physical and Mechanical Properties of Rocks

2.3.1 Unit Weight

The unit weight of any solid can be expressed as

$$\gamma = \frac{W}{V}$$

Where, W = weight of solid

V = volume of solid

Two types of unit weight, namelydry unit weight γ_d and unit weight of water γ_w are commonly used in case of solids.

$$\gamma_d = \frac{W_s}{V}$$

$$\gamma_w = \frac{W_w}{V}$$

2.3.2 Specific Gravity of Solids

The specific gravity of any material is the ratio of its density to that of water. In the case of soils, it should be computed for solid phase only. It can be expressed as

$$G_s = \frac{M_s}{V_s \rho_w} = \frac{W_s}{V_s \gamma_w}$$

2.3.3 Void Ratio

The relative volume of voids and solids may be expressed using the void ratio, e.

$$e = \frac{V_v}{V_s}$$

2.3.4 Porosity

The porosity of a rock sample is defined as the ratio of the volume of voids to the total volume of the sample.

$$\text{Porosity (P)} = \frac{V_v}{V} * 100\% = \frac{e}{1+e} * 100\%$$

Where, P = porosity in percentage
V_v = volume of void i.e. volume of air and water present in the pore spaces
V = total volume of the rock sample

It can also be defined as water porosity (Pw) and air porosity (Pa)

$$(PW) = \frac{V_w}{V} * 100\%$$

$$(Pa) = \frac{V_a}{V} * 100\%$$

2.3.5 Density

It is defined as mass per unit volume of the rock. There are three types of density viz. dry density, bulk density and saturated density.

2.3.5.1 Dry density : It refers to mass per unit volume when the rock mass is completely dry i.e. void contain only air.

2.3.5.2 Bulk density : It refers to mass per unit volume in normal condition i.e. void contain some liquid and some air.

2.3.5.3 Saturated density : It refers to mass per unit volume

The relationship between the dry density and bulk densityis

$$Y_d = \frac{Y_b}{1+m} = \frac{Y_b}{1+\frac{W_w}{W_s}}$$

Where, Y_d = dry density
Y_b = bulk density
m = the moisture content of the sample
W_w = the weight of water
W_S = the weight of solids

2.3.6 Moisture Content

It is the ratio of the weight of the water present in the sample to the weight of the sample.

$$\text{Moisture content (w)} = \frac{W_w}{W_s} * 100\%$$

Where, W_W = weight of water
W_S = weight of sample

2.3.7 Degree of Saturation

It is the ratio of volume of water in the void to the total volume of voids in the rock sample. The rock mass having higher porosity has higher degree of saturation.

$$S = \frac{V_w}{V_v}$$

Where, V_w = volume of water
V_v = volume of void

2.3.8 Permeability

It is the ability of a porous material to allow a liquid to pass through its pores. Flow of a liquid through a porous mass is given by

Q = K*I*A [Darcy's Law]

where, Q = the flow through the area A (m^3/S)
I = the hydraulic gradient (dimensionless)
K = the coefficient of permeability (m/s)
A = Area of cross section (m^2)

2.3.9 Anisotropy and Isotropy

An isotropy material has the same property in three axes whereas an anisotropy material has different property in different

direction or axis, it always behaves like anisotropy materials.

2.3.10 Stress

It is defined as the load applied on the material per unit area.

$$\text{Stress } (\sigma) = \frac{F}{A}$$

2.3.11 Strain

It is the ratio of change in length of the sample due to applied load to the original length of the sample.

$$\varepsilon = \frac{\Delta l}{l}$$

Where, ε = tensile or compressive strain, m/m
Δl = total elongation in a bar, m
l = original length of the bar, m

2.3.12 Strain Energy

The strain energy of the rock can be determined as

$$U = \frac{P.\Delta l}{2}$$

$$\Delta l = \frac{P.L}{A.E}$$

$$U = \frac{P^2 L}{2.A.E}$$

Where, U = total energy stored in the bar or strain energy
P = tensile load
Δl = total elongation in the bar
L = original length of the bar
A = cross-sectional area of the bar
E = Young's modulus

2.3.13 Modulus of Elasticity of rock

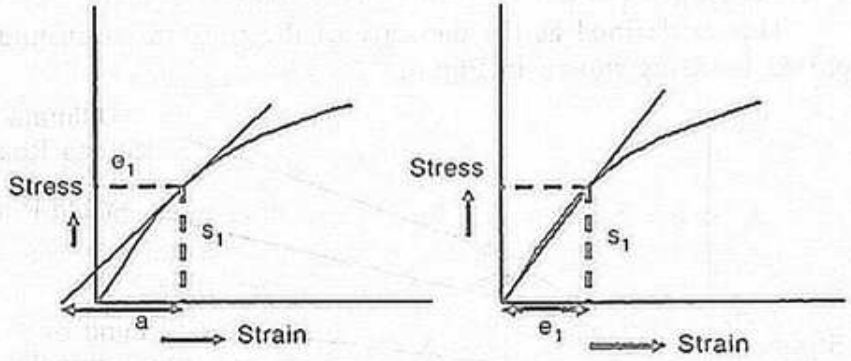
Rate of change of strain as a function of stress. The property of the material to recover the deformation is known as elasticity. Within the elastic limit the stress is proportional to the strain, called Hook's law.

$$E = \frac{\sigma}{\varepsilon}$$

Where, E = elastic constant, called Modulus of Elasticity
ó = the stress applied on the material

There are two types of modulus of elasticity viz. Tangent modulus of elasticity and Secant modulus of elasticity as shown in Figure 2.1.

Tangent modulus of elasticity is the slope of the stress-strain diagram at any point whereas secant modulus of elasticity is stress divided by strain at any given value of stress or strain. It is also called stress-strain ratio.



Tangent modulus of elasticity = s_1/a
Secant Modulus of elasticity = s_1/e_1

Figure 2.1 : Stress-Strain cure of a rock sample

2.3.14 Poisson Ratio

This is the ratio of lateral strain to longitudinal strain. If the material is subject to uni-axial compressive stress (σ_x), then there is a strain in longitudinal (x) direction as well as in the lateral direction (y). If the load applied in a rock sample is P and corresponding change in diameter and length are ΔD and ΔL respectively as shown in Figure 2.2, Poisson ratio can be calculated as

Poisson Ratio =

$$\frac{\text{Lateral Strain } (\varepsilon_{lat})}{\text{Longitudinal Strain } (\varepsilon_{lon})} = \frac{\Delta D/D}{\Delta L/L}$$

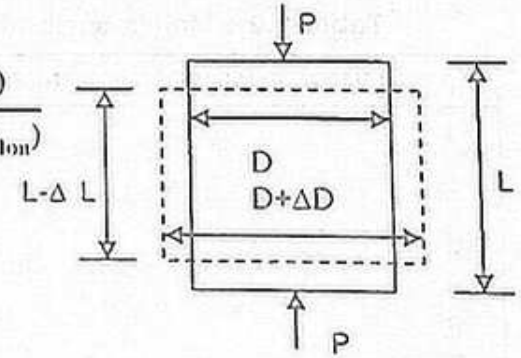


Figure 2.2 : Rock sample with applied load P

2.3.15 Unit Volume Change

The change in unit volume if rock samples is given by

$$\frac{\Delta V}{V} = \varepsilon(1-2\nu)$$

where, ΔV = change in volume
V = original volume
ε = strain
ν = Poisson's ratio

2.3.16 Deformability

This is defined as the capacity of the rock to strain under applied loads as shown in Figure 2.3.

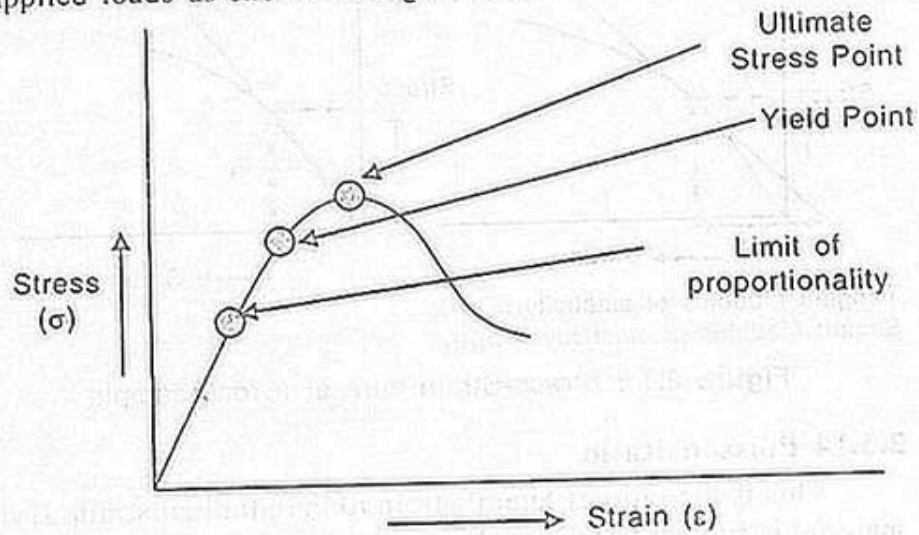


Figure 2.3 : Stress-Strain curve

2.3.17 Hardness

Hardness of rock is defined as its resistance to abrasion. Mohr's scale of hardness of minerals is given below in Table 2.2.

Table 2.2 : Mohr's scale of hardness of minerals

No. of relative hardness scale of rating H	Mineral
1	Talc (Softest)
2	Gypsum
3	Calcite
4	Fluorite
5	Apatite
6	Orthoclase Feldspars
7	Quartz
8	Topaz
9	Corundum

2.4 Classification of Rock Strength

2.4.1 Compressive Strength

- Uni-axial Compressive Strength: It is the strength of the rock that can sustain without failure, when the load applied in one direction (parallel to the axis) only.
- Tri-axial Compression Strength: When a rock mass is subjected to an all round pressure and it is further subjected to an additional vertical pressure, then strength exhibited by the rock mass is known as a tri-axial compression strength.

2.4.2 Tensile Strength

It is defined as the maximum tensile stress which a material is capable of developing. Since it is difficult to prepare rock samples for direct tensile tests, tensile strength of rock samples are determined indirectly by Brazilian tests. Tensile strength can be also determined from a point load test (Reichmath, 1963)

$$\text{Tensile Strength} = \sigma_t = 0.0675 * \frac{P}{D^2} \ [\text{Kg/cm}^2]$$

Where, P = the failure load in Kg
D = the core diameter in cm

2.4.3 Shear Strength

It is defined as the capacity of a rock mass to bear the shear load without failure.

2.5 Seismic Velocities related to material properties

The seismic velocities of material properties are of two types namely Body Waves and Surface Waves.

Body Waves are two types namely P-Wave and S-Wave whereas Surface waves are Rayleigh waves and Love waves.

P-waves : P waves (primary waves) are longitudinal or compressional waves. In solids, these waves generally travel almost twice as fast as S waves and can travel through any type of material.

S-waves : S waves (secondary waves) are transverse or shear waves, which mean that the ground is displaced perpendicularly to the direction of propagation.

Rayleigh waves : Rayleigh waves, also called ground roll, are surface waves that travel as ripples with motions that are similar to those of waves on the surface of water.

Love Waves : Love waves are surface waves that cause horizontal shearing of the ground. They usually travel slightly faster than Rayleigh waves, about 90% of the S wave velocity.

$$V_p \text{ - P-wave (cmpressive wave) velocity} = \sqrt{\frac{\frac{4}{3}\mu + k}{\rho}}$$

$$V_s \text{ - S-wave (shear wave) velocity} = \sqrt{\frac{\mu}{\rho}}$$

Where, μ = shear modulus
k = bulk modulus
ρ = density of rock

2.6 Rock Mass Strength Indices

2.6.1 Point Load Strength Index

In this test, a rock sample (length is at least 1.4 times the diameter) is loaded between two cones of hardened steel. Failure is caused by development of tensile cracks parallel to the loading axis. Point load strength (I_S) is given by

$$I_{s(d)} = \frac{P}{d^2}$$

where, P = the load at rupture
d = the distance between the point load
$I_{S(d)}$ = point load strength of the core sample having the diameter d.

A common correlation between point load strength and uni-axial compressive strength is given by

$$\sigma_U = 24 * I_{S(50)}$$

where, σ_U = uni-axial compressive strength of a core sample having length to diameter ratio is 2 : 1

$I_{S(50)}$ = point load strength of a core sample of diameter 50 mm.

2.6.2 Slake Durability Index

In this test, 500 gm of dry rock is broken into 10 pieces and put in a cylindrical drum of 140 mm diameter and 100 mm length. The cylindrical wall of the drum is made of sieve mesh of 2 mm openings. The drum is turned in a water bath for 10 min at a rate

2.7 Testing of Rock Samples

2.7.1 Uni-axial Compressive Strength Test

For cylindrical Specimen Length to diameter ratio should be lying in the range of 2 to 3. For cubical block L: D is 1.

The relationship between the computed ultimate compressive strength of a sample having L : D is 1 and a sample having length is greater than diameter is given by

$$\sigma_{cu} = \frac{\sigma_{ult}}{0.778 + 0.222\,(D/L)}$$

Where, σ_{cu} = computed ultimate compressive strength of a cubical or cylindrical specimen having L: D is 1
σ_{ult} = ultimate compressive strength of a cylindrical specimen having L: D is greater than 1
D = diameter of a cylindrical sample and the length of a side in case of a cube
L = length or height of a sample

2.7.2 Brazilian Test

For Brazilian test, the specimen should have a length to diameter ratio in the range of 0.5 to 1. The tensile strength is given by a Brazilian test

$$\sigma_t = \frac{2 * F}{\pi * D * L}$$

where, σ_t = tensile stress at failure in Kg/cm2
F = failure load in Kg
D = diameter of the specimen in cm
L = length of the specimen in cm

2.7.3 Bending Test

In this type of test, a beam is cut out of rock sample with rectangular cross section and the specimen is supported at two ends and load applied at the middle of the sample. The tensile strength is obtained by

$$\sigma_t = \frac{3 * P * L * n}{b * d^3}$$

where, 6_t = tensile strength in kg/ cm^2
P = applied load in kg

d = depth of the specimen in cm
n = distance from the neutral axis to the farthermost fibre.

2.7.4 Punch Shear Test

In this test, the shear strength of the specimen can be determined by punching shear. The specimen should be prepared in disc shape for this type of test. The shear strength can be evaluated with the following relation

$$\text{Shear Strength} = S = \frac{P}{A} = \frac{P}{\pi * d * t} \ [\text{kg/cm}^2]$$

Where, P = load applied to punch the specimen in Kg
A = circumferential shear area, cm^2
t = thickness of the disc, cm
d = diameter of the puncher, cm

2.8 Types of Classification of Rock Mass

The basis of rock mass is developed by various scientists and is given in Table 2.3.

Table 2.3 : Classification of Rock Mass

Classification	Basis of classification
Gamble's	Slake durability
Deere's	RQD
Deer's and Miller	Uni-axial compressive strength
Bieniawski's	RMR
Barton's	Q-system
Coates	Various types of classification (based on uni-axial compressive strength, failure characteristics, deformation, homogeneity, continuity)
Hansagi's	C-Factor

2.9 Determination of Shear Strength

In all slope stability analysis it is necessary to use the shear strength properties of either the discontinuities or of the rock mass and there are different procedures for determining the rock mass strength like

- Discontinuity shear strength can be measured in the field and the laboratory.
- Rock mass shear strength is determined by empirical methods involving either back analysis of slopes cut in similar geological conditions, or by calculation involving strength indices.

2.10 Mohr's Circle Analyses

A Mohr's circle describes the two dimensional stresses at a point in a material. It considers the stresses acting on each side of a two dimensional element and plots them on σ vs τ diagram as shown in Figure 2.4. Each point on the circle represents the normal and shear stresses acting on one side of an element oriented at a certain angle. For example, in this Mohr's circle points A and B represent (σ_z, τ_{zx}) and (σ_z, τ_{xz}), which are the stresses acting on an element aligned with the x and z axes.

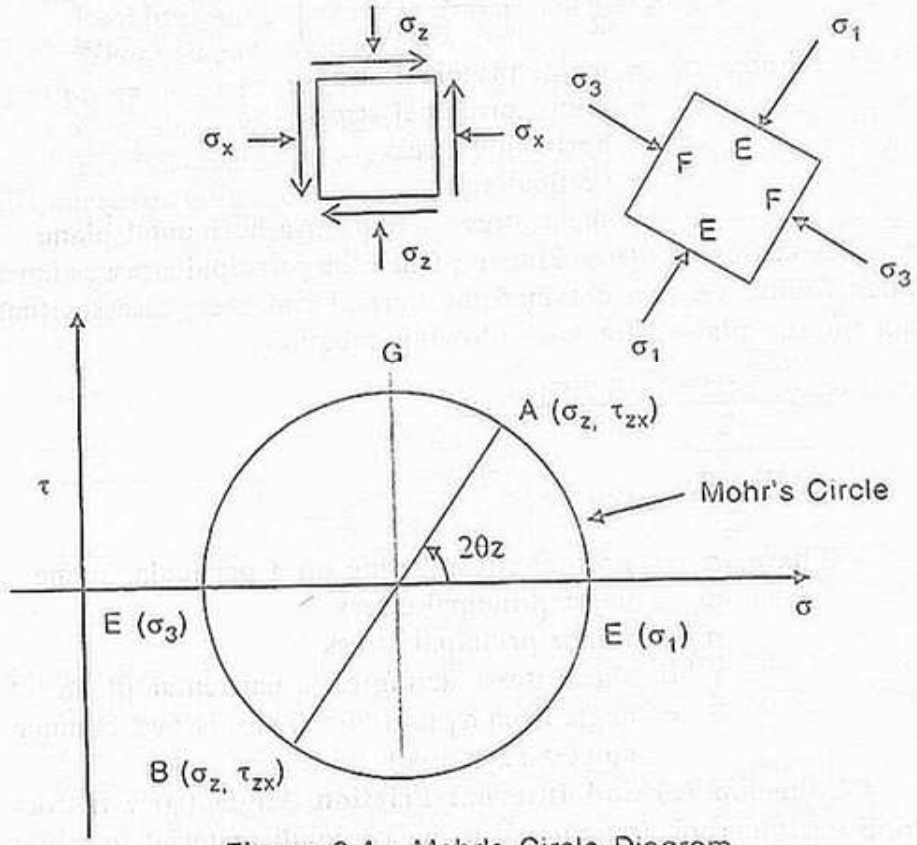


Figure 2.4 : Mohr's Circle Diagram

Principle Stresses : If the soil element is rotated to a certain angle, the shear stresses will be zero on all four sides. The planes on each side of this element are represented by points E and F, and are known as principal planes. The stresses acting on them are known as principal stresses. The major principal stress, σ_1, is the greatest normal stress whereas the minor principal stress, σ_3, is the smallest normal stress.

The magnitudes of σ_1 & σ_3 are given by

$$\sigma_1 = \frac{\sigma_x + \sigma_y}{2} + \sqrt{\left(\frac{\sigma_x - \sigma_y}{2}\right)^2 + \tau_{zx}^2} \quad \text{and}$$

$$\sigma_3 = \frac{\sigma_x + \sigma_y}{2} - \sqrt{\left(\frac{\sigma_x - \sigma_y}{2}\right)^2 + \tau_{zx}^2}$$

The angle between the major principal stress (σ_1) and the vertical stress (σ_z) is

$$\theta_z = \frac{1}{2}\cos^{-1}\left(\frac{2\sigma_z - \sigma_1 - \sigma_3}{\sigma_1 - \sigma_3}\right)$$

Where, σ_1 = major principal stress
σ_3 = minor principal stress
σ_x = horizontal stress
σ_y = vertical stress
τ_{zx} = shear stress acting on a horizontal plane

Stresses on other Planes : Once the principal stresses have been found, we can obtained the normal and shear stresses that act on any plane with the following equations.

$$\sigma = \frac{\sigma_1 + \sigma_3}{2} + \frac{\sigma_1 + \sigma_3}{2}\cos 2\theta$$

$$\tau = \frac{\sigma_1 - \sigma_2}{2}\sin 2\theta$$

Where, σ = normal stress acting on a particular plane
σ_1 = major principal stress
σ_3 = minor principal stress
τ = shear stress acting on a particular plane
θ = angle from σ_1 to σ (clockwise is +ve; counter-clockwise is –ve)

Cohesion (c) and Internal Friction Angle (φ) : In rock slope design, rock is assumed to be coulomb material in which the shear strength of the sliding surface is expressed in terms of cohesion (c) and the friction angle (φ).

For a test carried out at a constant normal stresses, a typical plot of the shear stress against the shear displacement is shown in the Figure 2.5 (a), 2.5 (b) and 2.5 (c). At small displacement, the specimen behaves elastically and the shear stress increases linearly with displacement. As the force resisting movement is overcome, represents the peak shear strength of the discontinuity. Thereafter the stress required to cause displacement decreases and eventually reaches a constant value termed the residual shear strength. If the peak shear strength and residual shear strength values from tests are determined at different normal stress levels and plotted, a linear relationship was established. The plot shows the relationship is called the Mohr's diagram.

The relationships between peak shear strength (τ) and normal stress (σ) is given by

Peak shear strength (τ) = c + σ tan φ_p

The relationships between peak shear strength (τ) and normal stress (σ) is given by

Residual shear strength (τ) = σ tan φ_r

Where, φ_p & φ_r are the peak friction angle and residual friction angle respectively.

C is cohesive strength.

For the residual strength condition, the cohesion is lost once displacement has broken the cementing action.

Again, $\varphi_p > \varphi_r$ [Because the shear displacement grinds the minor irregularities on the rock surface and produces a smoother, lower friction surface]

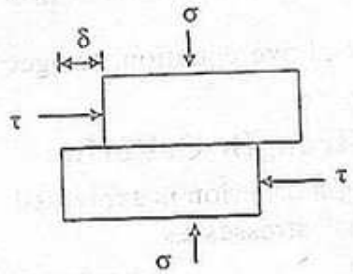


Figure 2.5 (a) : Shearing in plane

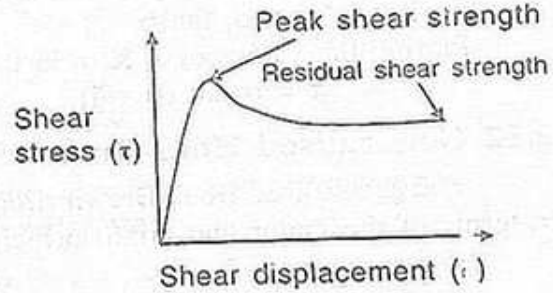


Figure 2.5 (b) Shear disp. Vs. Shear stress

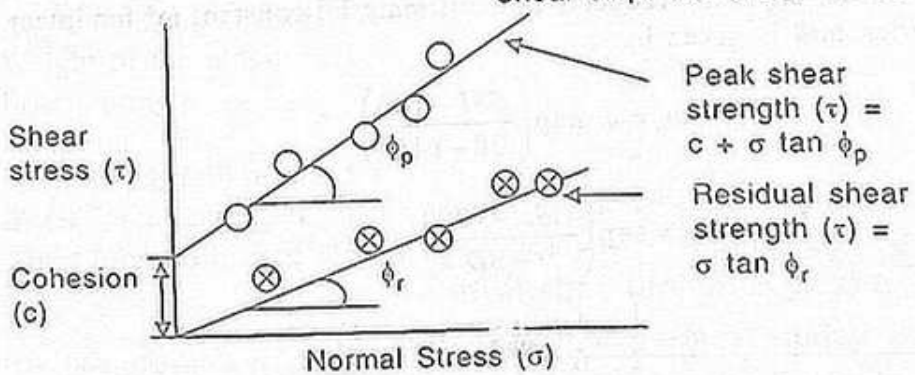


Figure 2.5 (c) : Shear stress Vs Normal Stress Curve

2.11 Shearing on an inclinded plane

The shearing of a sample in an inclined plane as shown in Figure 2.6 is explained below.

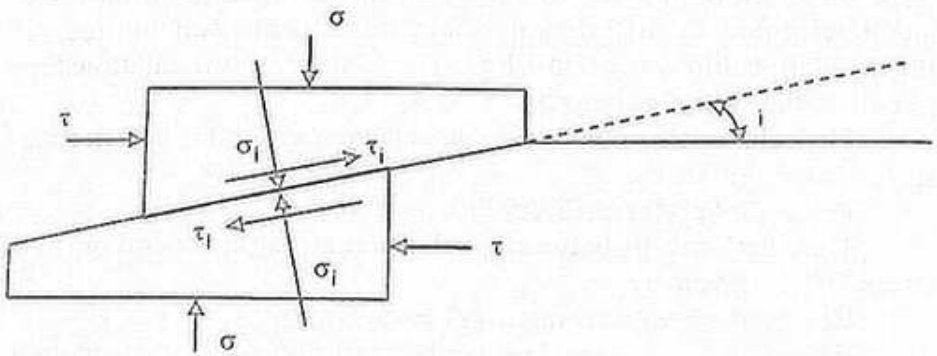


Figure 2.6 : Shearing in an Inclined Plane

In this case shear and normal stresses acting on the sliding surface, τi & σ_i respectively are given by

$$\tau_i = \tau \cos^2 i - \sigma \sin i \cos i$$

$$\sigma_i = \sigma \cos^2 i - \tau \sin i \cos i$$

For the discontinuity surface has zero cohesion, we can get

$$\tau_i = \sigma_i \tan \varphi$$

Putting the values of τ_i & σ_i in the above equation, we get

$$\tau = \sigma \tan (\varphi + i)$$

2.12 Generalised Hoek-Brown Strength Criterion

The generalised Hoek-Brown strength criterion is expressed in terms of the major and minor principal stresses as

$$\sigma_1 = \sigma_3 + \sigma_{ci}\left(m_b \frac{\sigma_3}{\sigma_{ci}} + s\right)^a$$

Where, m_b is the reduced value of material constant mi for intact rock and is given by

$$m_b = m_i \exp\left(\frac{GSI - 100}{28 - 14D}\right)$$

$$s = \exp\left(\frac{GSI - 100}{9 - 3D}\right)$$

$$a = \frac{1}{2} + \frac{1}{6}\left(e^{-GSI/15} - e^{-20/3}\right)$$

[D = 0 for undisturbed in-situ rock masses and 1 for very disturbed rock masses]

GSI is geological strength index.

The uni-axial compressive strength of the rock mass is obtained by setting $\sigma'_3 = 0$ in generalized Hoek-Brown equation

$$\sigma_c = \sigma_{ci} * s^a$$

And, the tensile is obtained by setting $\sigma'_c = \sigma'_3 = \sigma_t$ in generalized Hoek-Brown equation

$$\sigma_t = \frac{s * \sigma_{ci}}{m_b}$$

2.13 Determination of Factor of Safety of the Slope

CASE I : When there is no tension crack and water pressure : The force diagram of the situation is shown in Figure 2.7.

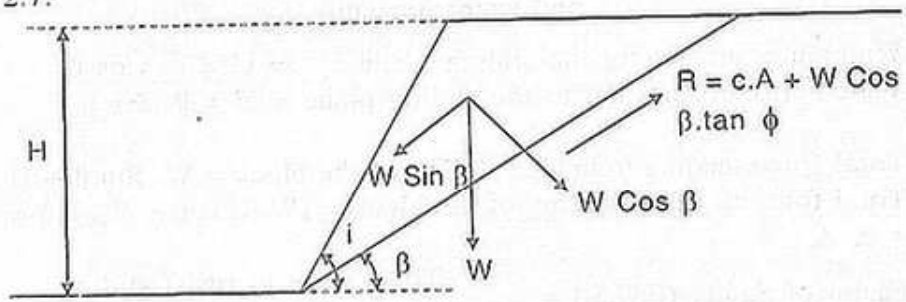


Figure 2.7 : Force diagram of a slope with no tension crack and water pressure

Assuming

Height of the bench	= H
Overall slope angle from horizontal	= i
Angle of discontinuity from horizontal	= β
Weight of the block	= W
Friction angle	= ϕ
Cohesion	= c
Area of the failure plane	= A

Total force tending to induce sliding of the block = W.Sin β

Total force to resist sliding of the block = W.Cos β. tan ϕ + c.A

[Neglecting the water pressure]

$$\text{Factor of safety (F.O.S.)} = \frac{\text{Total force to resist sliding}}{\text{Total force tending to induce sliding}}$$

$$= \frac{c.A + W.Cos\,\beta.\tan\phi}{W.Sin\,\beta}$$

CASE II : When there is tension crack and water pressure present : The force diagram of the situation is shown in Figure 2.8.

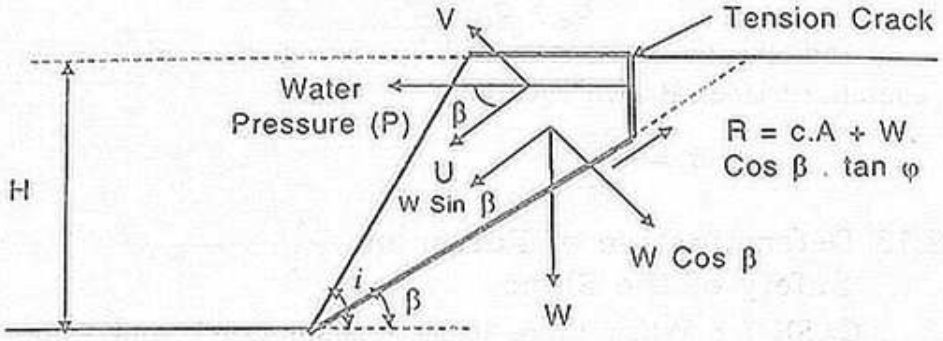


Figure 2.8 : Force diagram of a slope with the tension crack and water pressure

Water pressure along the sliding plane = U = P. Cos β
Water pressure normal to the sliding plane = V = P. Sin β

Total force tending to induce sliding of the block = W. Sin β + U
Total force to resist sliding of the block = (W. Cos β − V) tan φ + c. A

$$\text{Factor of Safety (F.O.S.)} = \frac{\text{Total force to resist sliding}}{\text{Total force tending to induce sliding}}$$

2.14 Stress and Displacement Distribution in Circular Excavation

The stress distribution around a circular excavation as shown in Figure 2.9 can be determined by

$$\sigma_{rr} = \frac{P}{2}\left[(1+K)\left(1-\frac{a^2}{r^2}\right)-(1-K)\left(1-4\frac{a^2}{r^2}+\frac{3a^4}{r^4}\right)\cos 2\theta\right]$$

$$\sigma_{\theta\theta} = \frac{P}{2}\left[(1+K)\left(1+\frac{a^2}{r^2}\right)+(1-K)\left(1+\frac{3a^4}{r^4}\right)\cos 2\theta\right]$$

$$\sigma_{r\theta} = \frac{P}{2}\left[(1-K)\left(1+\frac{2a^2}{r^2}-\frac{3a^4}{r^4}\right)\sin 2\theta\right]$$

$$u_r = -\frac{Pa^2}{4Gr}\left[(1+K)-(1-K)\left\{2(1-2\nu)+\frac{a^2}{r^2}\right\}\cos 2\theta\right]$$

$$u_\theta = -\frac{Pa^2}{4Gr}\left[(1-K)\left\{2(1-2\nu)+\frac{a^2}{r^2}\right\}\sin 2\theta\right]$$

In the above expressions u_r, u_θ are displacements induced by excavation, while σ_{rr}, $\sigma_{\theta\theta}$, $\sigma_{r\theta}$ are total stresses after development of the opening.

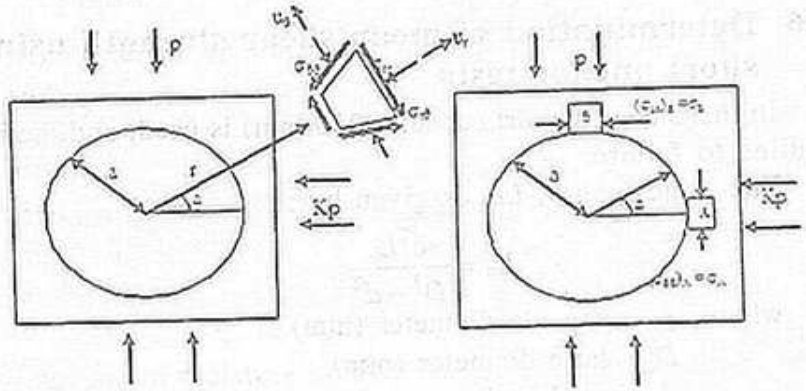


Figure 2.9 : Stress distribution around a circular excavation

By putting r = a in the above equations, the stresses on the excavation boundary are given as

$$\sigma_{rr} = 0$$

$$\sigma_{\theta\theta} = P[(1+K)+(1-K)\,2\cos 2\theta]$$

$$\sigma_{r\theta} = 0$$

Similarly, for $\theta = 0$, and r is very large

$$\sigma_{rr} = KP$$

$$\sigma_{\theta\theta} = P$$

$$\sigma_{r\theta} = 0$$

2.15 Determination of Length of the Roof Bolt

The optimal roof bolt length depends on the support mechanism. Where bolts are merely acting as skin control, they may be as short as 900 mm (Minney and Munsamy, 1998). In the suspension mode, bolts should obtain at least 300 mm of anchorage in the solid strata (Mark, 2000)

Several investigators have also studied the optimal length of

the bolt that should be installed under various conditions. A summary of recommendations is given below.

Lang and Bischoff (1982) : $L = B^{2/3}$

Bieniawski (1987) : $L = B/3$

Unal (1984) : $L = \frac{B}{2}\left(\frac{100 - RMR}{100}\right)$

Where, RMR is the rock mass rating (Bieniawski, 1987) which ranges from 20 to 80 depending on roof rock conditions. B is span (m) and L is bolt length (m).

2.16 Determination of mean shear strength using short anchor tests

In these tests a short capsule (250 mm) is used, and the bolt is pulled to failure.

The bond length, LB, is given by :

$$L_B = \frac{\delta^2 L_c}{D^2 - d^2}$$

where, δ = capsule diameter (mm)

D = hole diameter (mm)

d = tendon diameter (mm)

Lc = capsule length (mm)

The mean shear strength may then be calculated from :

$$\tau = \frac{P_f}{\pi D L_B}$$

where, τ = mean shear strength (Pa)

P_f = yield load of bolt (N)

D = hole diameter (m)

Lc = capsule length (m)

Once τ has been determined by short encapsulated pull tests, the calculation may be reversed in order for the required capsule length to be found through substituting P_f for a design load. A suitable safety factor should also be used.

EXERCISE

1. A sandstone specimen of diameter 54 mm and height 60 mm has moisture content of 20 %, and specific gravity 2.65. The porosity of the sandstone is
   a. 0.346 b. 1.53
   c. 1.732 d. 2.887

2. Porosity of a coarse grain sandstone sample is 15 %. The sp. gr. of sandstone is 2.8. What is the void ratio in the sandstone sample?
   a. 0.150 b. 0.176
   c. 0.850 d. 1.176

3. Porosity of a coarse grain sandstone sample is 15 %. The sp. gr. of sandstone is 2.8. If the sandstone is fully saturated in water, the saturated density of the sample in kg/m^3 is
   a. 1590 b. 2234
   c. 2438 d. 2531

4. Dimension of hydraulic permeability expressed with L, M and T is
   a. L^2 b. LT^{-1}
   c. LT^{-2} d. MLT^{-1}

5. Dimension of modulus of elasticity (E) expressed with L, M and T is
   a. $M L^{-1} T^{-2}$ b. $M L^{-1} T^{-1}$
   c. $M L T^{-1}$ d. $M L T^{-2}$

6. Increase in moisture content of rock, the effect on bearing capacity of the rock is
   a. Increases b. Decreases
   c. No relation with the moisture content
   d. Depends on the rock type

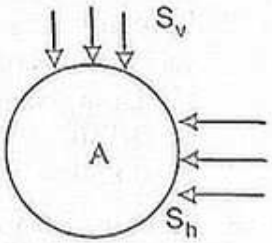
7. A rock specimen had moisture content of 2.5 %. Volume of solids was found to be 75 % and specific gravity of solid is 2.5. Degree of saturation of the rock sample is
   a. 1.975 % b. 19.75 %
   c. 0.1975 % d. 30 %

8. During a laboratory permeability experiment on mill tailings 20 ml of water percolated in 30 min through tailings of 5 cm diameter, and 20 cm length. If the tailing column is subjected to a steady hydraulic gradient of 1.2 m during the experiment, what is the hydraulic conductivity of the tailings?
   a. 94 Darcys b. 172 Darcys
   c. 0.94 cm/ s d. 17.2 m^3/ s

   [GATE, 2000]

9. A circular opening in a biaxial stress field is as shown, with S_v and S_h being the vertical and horizontal compressive stresses respectively. The point A is under tensile stress when

a. $S_h = 0$ b. $S_h = S_v / 3$
c. $S_h = S_v$ d. $S_h = 3\ S_v$



[GATE, 2000]

10. Weak sandstone has modulus of elasticity of 1000 MPa and Poisson's ratio is 0.25. Its shear modulus is
a. 4 GPa b. 400 GPa
c. 500 GPa d. 800 GPa

[GATE, 2000]

11. A core of 10 cm height and 5.4 cm diameter is subjected to point loading. The ram of the hydraulic jack has a cross sectional area of 12.5 cm^2 and the dial gauge reading is 3 MPa. What is the point load strength?
a. 1.63 MPa b. 30 MPa
c. 37.5 MPa d. 68.7 MPa

[GATE, 2000]

12. A resin grouted rock bolt is able to provide yield strength of 200 MPa. If the bolt diameter is 2.5 cm and the grout length 154 cm, what is the adherence (shear strength) between the bolt and the grout?
a. 800 kg/ cm^2 b. 800 kPa
c. 80 MN d. 8000 N/m

13. A tension crack of 5 m depth behind a rock slope crest is completely filled with water. What is the horizontal force exerted on a unit slope length by the water in the tension crack?
a. 122.6 kN b. 245 kN
c. 12.26 MPa d. 24.5 Nm

14. The shear stress on a plane of major principal stress acts is
a. half the major principle stress
b. twice the major principle stress
c. shear stress is absent
d. three times the major principle stress

15. Following is the state of stress in a biaxial stress field :
σ_{XX} = 100 MPa σ_{YY} = 30 Mpa
σ_{XY} = 15 MPa
The major principal stress is equal to
a. 103 MPa b. 130 MPa
c. 145 MPa d. 165 MPa

16. Immediate roof of a mine roadway is supported by 1.5 m long roof bolts placed at 1.2 m centers along and across the roadway. The bolts are of 4 cm^2 cross sectional area are made from steel having tensile strength 300 KN/ m^2. Taking sp. gr. of roof rock is 2.5, the factor of safety of the roof bolt is
a. 1.86 b. 2.26
c. 2.98 d. 4.34

17. The rock specimen obtained from the overlying strata in a coal mine were tested in uni-axial compression and the following results were obtained at 50 % of average compressive strength
Longitudinal strain = 0.06 and Lateral strain = 0.012
The ratio of vertical to horizontal stress in the mine is
a. 3.0 b. 4.0
c. 2.5 d. 0.3

18. A cylinder specimen of sandstone of 50 mm diameter and 125 mm length, tested in uni-axial compression gave the following results :

Load at failure = 100 kN and longitudinal deformation upto failure = 1.5 mm.

Assuming load-deformation curve to be straight line up to failure, find the uni-axial compressive strength and the modulus of elasticity of the sandstone
a. 12.72 MPa; $1.09*10^9$ Pa b. 15.72 MPa; $1.8*10^6$ Pa
c. 1.72 MPa; $1.9*10^9$ Pa d. 12.72 MPa; $1.09*10^{10}$ Pa

19. A rock specimen was subjected to diametrical point load testing. The pressure gauge reading at failure is 12.5 MPa. If the ram area is 13.36 cm^2, and the diameter of the core tested is 54 mm, the point load strength index is
a. 5.7 MPa b. 5.21 MPa
c. 4.85 MPa d. 5.46 MPa

20. For a circular opening in rock under hydrostatic condition at great depth, the maximum tangential tensile boundary stress will occur at
a. 0° with vertical b. 45° with vertical
c. 90° with vertical d. no where along the boundary

21. For a typical hard rock with $E = 10^5$ MPa and Poisson's ratio (υ) = 0.25, the modulus of rigidity in GPa is
a. 40 b. 80 c. 66.67 d. 133.33

22. The length of consecutive cores obtained from 2m section of a borehole (NX size) are 89, 32, 18, 172, 80, 255, 280, 176, 53, 302, 102, 153, 40, 147 and 101 mm. Then the RQD value of rock in that section is
a. 66.9 % b. 84.4 % c. 95.5 % d. 100 %

23. The corrected value (C_o) for the uni-axial compressive strength is the specimen size is not 1 : 1 is given by the relation where C_p is observed uni-axial compressive strength

a. $C_0 = \dfrac{C_p}{0.778 \div 0.222 * (D/L)}$

b. $C_0 = \dfrac{C_p}{0.222 \div 0.778 * (D/L)}$

c. $C_0 = \dfrac{C_p}{0.778 \div 0.222 * (L/D)}$

d. $C_0 = \dfrac{C_p}{0.222 \div 0.778 * (L/LD)}$

24. Barton classified the rock mass on the basis of
a. RMR b. Q-system
c. Slake durability d. RQD

25. Hansagi classified the rock mass on the basis of
a. C-factor b. RMR
c. Q-system d. RQD

26. The term RQD/ Jn expressed in Q system refers to
a. Sizes of joint block
b. Shear strength of block surfaces
c. Environmental conditions influencing the behaviour of the rock mass
d. Roughness of joint surfaces
[GATE, 2006]

27. The instrument Flat jack is used for measuring the
a. In-situ stress in rock b. Load
c. Bed separation resistance d. Roof convergence

28. The instrument Tape extensometer is used for measuring the

29. In a point load test on 50 mm diameter core specimen, rupture was observed at a load of 5000 kg. The point load strength of the specimen was
a. 300 kg/ cm^2 b. 200 kg/ cm^2
c. 400 kg/ cm^2 d. 500 kg/ cm^2

30. In a point load test on 50 mm diameter core specimen, rupture was observed at a load of 5000 kg. The uni-axial compressive strength of the specimen was
a. 3800 kg/ cm^2 b. 4000 kg/ cm^2
c. 4800 kg/ cm^2 d. 5000 kg/ cm^2

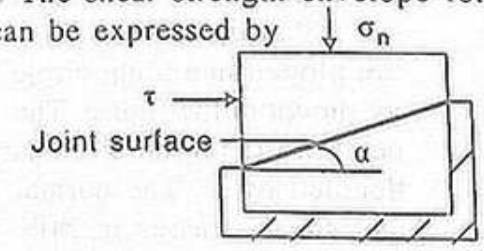
31. During Barzilian test a rock specimen of diameter 5 cm, and height 5 cm, failed at a compression load of 19.6 kN. The indirect tensile strength of the specimen is
a. 0.5 kPa b. 2.5 MPa
c. 5 MPa d. 25 kPa

32. During Barzilian test of a cylindrical rock core of 54 mm diameter and 22 mm thickness, failure occurred at diametrical loading of 11.4 kN. The tensile strength of rock in MPa will be
a. 1.24 b. 2.49
c. 3.05 d. 6.10 [GATE, 2004]

33. For a bulking factor of 1.5, the height of the immediate roof for an extraction thickness of H is
a. H b. 1.5 H
c. 2 H d. 2.5 H

34. The specific gravity of gypsum mineral is 2.4. If the density of a gypsum core is 2100 kg/m^3, the porosity of that core is
a. 0.062 b. 0.125
c. 0.5 d. 0.875
[GATE, 2006]

35. In the direct shear test, the joint surface is inclined $\alpha°$ with the direction of the applied shear load. The bottom half of the specimen is fixed inside the shear box. The friction angle of the joint surface is $\phi°$. The shear strength envelope for the experimental set up can be expressed by



a. $\tau = \sigma_n \tan(\phi \div \alpha)$
b. $\tau = \sigma_n \tan(\phi - \alpha)$
c. $\tau = \sigma_n \tan\phi$
d. $\tau = \sigma_n \tan\alpha$

36. The cohesion and the angle of internal friction of a sandstone rock are 6 MPa and 35° respectively. The magnitudes of theoretical unconfined compressive and tensile strength of sandstone in MPa are

a. 6.24, 0.66 b. 11.53, 3.12
c. 16.14, 4.37 d. 23.05, 6.24

[GATE, 2006]

37. The classification of rock mass on the basis of RQD is given by

a. Gamble b. Deere
c. Bieniawski d. Barton

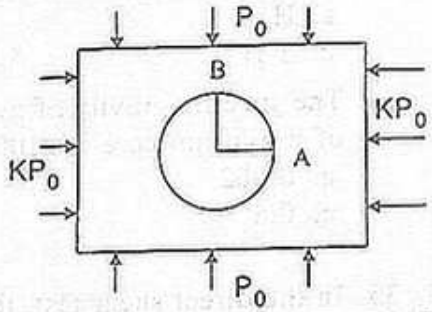
38. A rock specimen of average thickness 0.8 cm failed at an applied load of 1000 kg during determination of shear strength by the punch test. The diameter of the punch is 2.8 cm. The shear strength of the rock specimen in kg/ cm^2 is

a. 50.75 b. 142.10
c. 284.20 d. 446.43

[GATE, 2006]

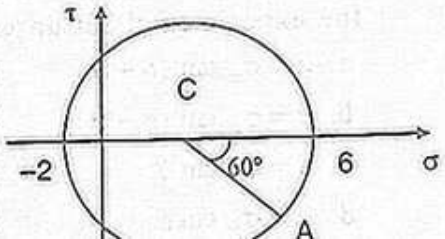
39. A circular tunnel is opened in sandstone rock with far-field stresses of P_O and KP_O as shown in the figure. If the tangential stress at point A is 3 times that of the point B (both points are located on the boundary of the tunnel), the value of K is

a. 0.3
b. 0.6
c. 1.0
d. 3.0



[GATE, 2006]

40. The major and minor principal stresses of a rock specimen are found to be 6 MPa and – 2 MPa respectively. They are plotted in a Mohr circle as shown in the figure. The centre of the Mohr circle is denoted by C. The normal and shear stresses in MPa



a. 2, –2.0 b. 2, –3.46
c. 2, –6.93 d. 2, –3.46

[GATE, 2006]

41. Rock mass parameters for a shale rock, m_b, s, and a of the Hoek and Brown failure criterion as given in the equation are 2.4, 0.01 and 0.52 respectively. The uni-axial compressive strength (σ_{Ci}) of intact shale is 50 MPa.

$$\sigma_1 = \sigma_3 + \sigma_{ci}\left(m_b \frac{\sigma_3}{\sigma_{ci}} + s\right)^a$$

The rock mass strength of shale in MPa is

a. 1.2 b. 10.29
c. 20.83 d. 50.00

42. Rock mass parameters for a shale rock, m_b, s, and a of the Hoek and Brown failure criterion as given in the equation are 2.4, 0.01 and 0.52 respectively. The uni-axial compressive strength (σ_{Ci}) of intact shale is 50 MPa.

$$\sigma_1 = \sigma_3 + \sigma_{ci}\left(m_b \frac{\sigma_3}{\sigma_{ci}} + s\right)^a$$

The uniaxial compressive strength of rock mass is

a. 1.2 b. 4.59
c. 20.83 d. 50.00

43. For a circular opening of radius a, where the far-field stress value is P. The radial and tangential stresses at a distance r from the center of the opening are

a. $\sigma_r = 0; \quad \sigma_t = P\left(1 + \frac{a^2}{r^2}\right)$

b. $\sigma_r = P\left(1 - \frac{a^2}{r^2}\right); \quad \sigma_t = P\left(1 + \frac{a^2}{r^2}\right)$

c. $\sigma_r = 2P\left(1 - \frac{a^2}{r^2}\right); \quad \sigma_t = P\left(1 + \frac{a^2}{r^2}\right)$

d. $\sigma_r = 3P; \quad \sigma_t = 0$

[GATE, 2001]

44. A room and pillar stope at 100 m depth has overlying rock density of 2800 kg/m^3. For a square pillar of side 4 m, height 3 m and for room width of 2.4 m, the mean pillar

a. 3.2 b. 3.8
c. 19.6 d. 7.0

[GATE, 2001]

45. For a cylindrical specimen of cro s sectional area A, length L, Young's modulus E, and axial load P, the strain energy stored is
a. $P^2L/2AE$ b. $PL^2/2AE$
c. P^2L^2/AE d. P^2L/AE

[GATE, 2001]

46. Correlation between the rock mass classification systems of RMR and Q is
a. $RMR = 9 \log_e Q + 44$ b. $Q = 8 \log_e RMR + 34$
c. $RMR = 9 \log_{10} Q + 44$ d. $Q = \log_e RMR + 44$

[GATE, 2001]

47. In a compression strength test the increase of confining pressure leads to a reduction in the rock
a. ductility b. angle of internal friction
c. brittleness d. ultimate bearing load

[GATE, 2001]

48. The ratio of specific weight of an intact rock to that of loose rock is
a. fill factor b. loose density
c. swell factor d. compaction factor

[GATE, 2001]

49. For a wooden prop of modulus of elasticity 60 MPa, critical bearing strength 25 MPa, length 2.5 m and diameter 0.16 m, the slenderness ratio is
a. 15.625 b. 0.064
c. 2.4 d. 0.416

50. During uniaxial compressive testing of a rock specimen of diameter 54 mm and length 120 mm, the axial strain was found to be 6.5×10^{-4} mm/mm. If the Poisson's ratio of the rock is 0.3, the volumetric strain of the rock specimen in mm/mm is
a. $1.95 * 10^{-4}$ b. $2.60 * 10^{-4}$
c. $2.34 * 10^{-2}$ d. $3.12 * 10^{-2}$

51. In a point load strength index test, the samples's length to diameter ratio is
a. 2:1 b. 3:1

52. An immediate roof of RMR 45 will be supported by roof bolts. If the width of the gallery is 4.6 m, the minimum length of the bolt in m will be
a. 1.035 b. 1.265
c. 2.070 d. 2.530

[GATE, 2004]

53. The correct order of hardness of rock in their increasing order
a. Gypsum < Flourite < Quartz < Topaz
b. Flourite < Gypsum < Quartz < Topaz
c. Flourite < Quartz < Gypsum< Topaz
d. Topaz < Quartz < Flourite < Gypsum

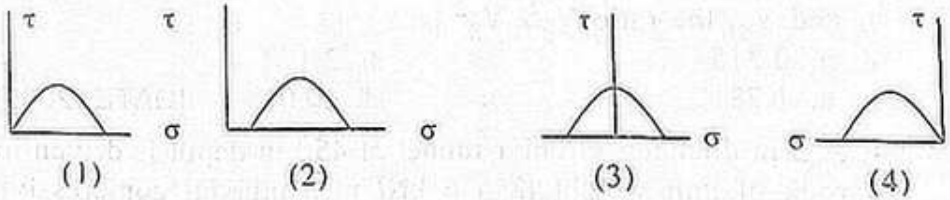
54. A rock specimen of 120 mm length and 54 mm diameter is subjected to uni-axial compressive loading. Two strain gauges, one axially and other radially, are attached to the specimen. At the breaking load, if these two gauges read 0.0082 m/m and 0.0023 m/m respectively, the Poisson's ratio of the specimen is
a. 3.56 b. 0.25
c. 0.28 d. 0.32

[GATE, 2002]

55. A rock specimen was tested in compression. The lateral and longitudinal strains were found 0.6 % and 3 % respectively. The Poisson's ratio of the specimen is
a. 0.2 b. 0.3
c. 0.4 d. 0.18

56. From a core drilling of shale, the average number of fractures per meter of core length is found to be 20. The RQD (%) of the rock is
a. 60 b. 37
c. 49 d. 53

57. Identify the 'Pure shear' state of stress condition from the following four Mohr circle diagrams



a. 3 b. 2
c. 1 d. 4

58. From a series of tri-axial tests conducted on some sandstone specimens, the relationship between the major and the minor principal stresses in MPa is found to be $\sigma_1 = 100 + 5\ \sigma_3$. The angle of internal friction in degrees of that sandstone is

a. 38 b. 79
c. 42 d. 32

[GATE, 2002]

59. From a series of tri-axial tests conducted on some sandstone specimens, the relationship between the major and the minor principal stresses in MPa is found to be $\sigma_1 = 100 + 5\ \sigma_3$. The coefficient of cohesion of that sandstone is

a. 5.3 b. 10.5
c. 22.3 d. 18.5

60. A sample of 3 cm thickness and 8 cm diameter is subjected to punch shear test. The shear failure of the sample occurs at 200 kg load. What is the shear strength of the sample?

a. 2.65 kg/cm^2 b. 3.55 kg/cm^2
c. 8.21 kg/cm^2 d. 12.65 kg/cm^2

61. The values of normal and shear stresses are $\sigma_x = 7.99$ MPa, $\sigma_y = 6.90$ MPa, and $\tau_{xy} = 0.07$ MPa. The values of maximum and minimum principal stresses in MPa are

a. 6.90, 9.00 b. 7.50, 1.09
c. 8.00, 6.90 d. 8.50, 1.55

[GATE, 2005]

62. The measurements of a site investigation are as follows :

Jn = 9, Jr = 1.5, Ja = 2.0, Jw = 1.0, SRF = 2.5. The RQD values taken from five boreholes of the area are 94.2, 95.6, 99.4, 95.3 and 98.0 respectively. The rock mass quality index (Q) is

a. 3.20 b. 5.60
c. 10.14 d. 20.10 [GATE, 2005]

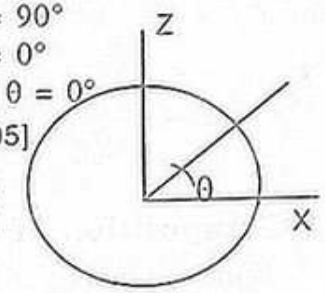
63. The Poisson's ratio (μ) of a rock specimen is 0.27. If the compressive and shear wave velocities are denoted by V_P and V_S, the ratio V_p^2/V_S^2 is

a. 0.315 b. 3.174
c. 1.781 d. 10.07 [GATE, 2005]

64. A 3 m diameter circular tunnel at 450 m depth is driven in rock of unit weight (λ_w) 6 kN/ m^3, uniaxial compressive strength (σ_c) 25 MPa and uniaxial tensile strength (σ_t) 3.0 MPa. The state of stress condition for K = 0.3 is

b. $\sigma_\theta = \sigma_t$, failure in tension at $\theta = 90°$
c. $\sigma_\theta = \sigma_t$, failure in tension at $\theta = 0°$
d. $\sigma_\theta = \sigma_c$, failure in compression at $\theta = 0°$

[GATE, 2005]



65. The number of experimental set ups of flat jack to determine 3 dimensional state of stress in a rock mass is

a. 6 b. 3
c. 2 d. 1 [GATE, 2005]

66. The uni-axial compressive strength and modulus of elasticity of a rock sample are 90 MPa and 30 GPa respectively. The stored strain energy per unit volume in kN.m/ m^3 of the sample at the time of failure is

a. 1.5 b. 140
c. 270 d. 135

67. What is the seismic S-wave velocity in the near surface earth if the density of the earth surface rock is 2500 kg/ m^{3}and the shear modulus is 10^{10} Pa?

a. $2*10^3$ b. $5*10^4$
c. $25*10^4$ d. $5*10^{-4}$

68. What is the seismic P-wave velocity in the near surface earth if the density of the earth surface rock is 2500 kg/m^3, the shear modulus = 10^{10} Pa, and the Poisson's ratio is 0.25?

a. $3.46*10^3$ b. $5.26*10^3$
c. $5.00*10^4$ d. $8.45*10^3$

Answer

1. d	2. b	3. d	4. b	5. a	6. b
7. b	8. c	9. d	10. d	11. b	12. a
13. a	14. c	15. a	16. b	17. b	18. a
19. a	20. b	21. a	22. b	23. a	24. b
25. a	26. a	27. a	28. d	29. b	30. c
31. c	32. d	33. b	34. b	35. a	36. d
37. b	38. b	39. b	40. b	41. b	42. b
43. b	44. d	45. a	46. a	47. d	48. c
49. a	50. b	51. a	52. b	53. a	54. c
55. a	56. b	57. c	58. c	59. c	60. a
61. c	62. a	63. b	64. c	65. b	66. d
67. a	68. a				

3.0 Mine Ventilation

3.1 Composition of Mine Air

Mine air mainly consists of following impurities :

a. Non-toxic but explosive in nature
CH_4, C_2H_2, H_2

b. Toxic gases :
CO_2, Radon and its daughter products

c. Acutely poisonous gases :
CO, NO_2, SO_2, H_2S

3.2 Layers of Gases in Mine Atmosphere

Mine atmosphere consists of following layers of gases from roof to floor :

Roof

H_2	S.G. = 0.070
CH_4	S.G = 0.559
NH_3	S.G = 0.590
N_2	S.G = 0.967
CO	S.G. = 0.972
NO	S.G. = 1.037
O_2	S.G. = 1.105
H_2S	S.G. = 1.175
CO_2	S.G. = 1.529
N_2O	S.G. = 1.53
SO_2	S.G. = 2.264

Floor

3.3 Damps in u/g Mine Atmosphere

(C_2H_6, C_2H_4 etc.) = 5 %; CO_2 = 3-5 %]

b. Blackdamp : [CO_2 = 13 %; N_2 = 87 %]

c. Afterdamp : [Residual air = 30-50 %, N_2 = 44-58 %; CO_2 = 4-8 %; CO = 2-4 %]

d. Stinkdamp : [H_2S]

3.4 Permissible Concentrations of different Gases in Indian Underground Mines

CH_4	→	< 0.75 % in the general body of return air < 1.25 % in any parts of the mine
CO_2	→	< 0.5 % (5000 ppm)
CO	→	< 0.005 % (50 ppm)
Oxides of nitrogen (NO, NO_2, N_2O_4)	→	< 0.0005 % (5 ppm)
O_2	→	> 19%
H_2S	→	< 0.001 % (10 ppm)
SO_2	→	< 0.0005 % (5 ppm)

3.5 Degree of Gassiness of a Coal Mine

The degree of gassiness in coal mine are categorised as given in Table 3.1.

Table 3.1 : Degree of Gassiness

Degree of Gassiness	Percentage of inflammable gas in general body of return air	Rate of emission of gas m^3/te of coal output
I	< 0.1	< 1
II	> 0.1	1-10
III		> 10

3.6 Gas Monitoring Instruments

Different gas monitoring instruments used in mines are given in Table 3.2.

Table 3.2 : Gas monitoring instruments

Gas	Name of the instruments	Working principle/ chemical used for detection
CH_4	MSA D-6 Methanometer	Wheatstone bridge circuit
	Flame Safety Lamp	Length of the flame
	MSA Automatic firedamp Detectors (AFD 2)	Wheatstone bridge circuit
	Interference Methanometer	Refractive index of methane
CO	Draeger multigas Detector with P.S. Detector	Silica gel impregnated with light yellow potassium palladium sulphite
	Draeger multigas Detector with Hopcalite	Analysing cell containing MnO_2 + Cu_2O
	Draeger multigas Detector with Hoolamite tube	I_2O_5 + H_2SO_4

3.7 Coward Diagram

Coward diagram indicates the chances of fire damp explosion for different composition of methane air mixture.

The combustion of methane in air can be expressed as

$$CH_4 + 2\ O_2 + 8\ N_2 = CO_2 + 2\ H_2O + 8\ N_2$$

It is clear from the above equation that proper oxygen balance occurs when methane content of air is 1/ 11th or 9.5 % by volume. Thus the methane content of 9.5 % in air forms the most explosive mixture.

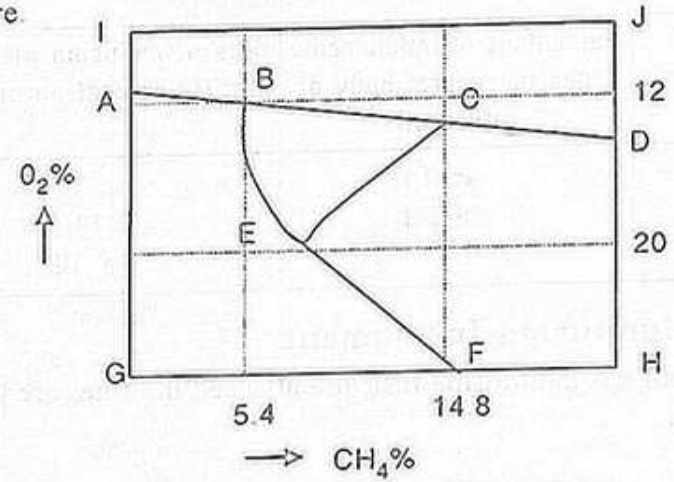


Figure 3.1 : Coward Diagram

There are four zones in the above Coward diagram shown in

ABEFG : Not capable of forming explosive mixture with air

BCE : Highly explosive mixture

CEFHD : Capable of forming explosive mixtures with air contain too much methane to explode

ABCDJI : Mixture which cannot be produced from methane air

The above diagram clearly represents the limits of explosibility of the methane is 5.4 % to 14.8 %. The limits of explosibility expand, if methane is associated with other combustible gases such as C_2H_6, H_2 etc. depending on their relative concentration and individual limits of explosibility.

The limits of explosibility of a mixture of combustible gases can be computed from the Le Chatelier equation as :

Limits of explosibility of a mixture of combustible gases =

$$= \frac{100}{V_1 / x_1 + V_2 / x_2 + \ldots\ldots} \%$$

Where, V_1, V_2 ... are the percent by volume of the different combustible components ($V_1 + V_2 + \ldots\ldots = 100\%$) and $x_1, x_2, \ldots\ldots$ are the limits of explosibility of the corresponding components.

3.8 Dealing with Firedamp in Mines

The gas is diluted by the ventilation current of air. The quantity and velocity of the air should be such that the gas is diluted to a safe limit. The general mechanism of dilution of gas in normal mine roadways or at the face where turbulent flow occurs is by turbulent diffusion. The distance required for complete mixing of methane in air by turbulent diffusion in a duct is given by

$$L = 22 * r / \sqrt{f}$$

Where, L = distance from source of gas

f = dimensionless resistance coefficient

r = radius of the duct

3.9 Methane Layering Number

The stability of methane layers as well as their length is indicated by a dimensionless layering number L and is given by

$$L = \frac{\upsilon}{\left(g * \frac{\rho_a - \rho_m}{} * \frac{Q}{W} \right)^{1/3}}$$

Where v = average velocity of air, m/s
Q = quantity of methane emitted into the airway, m^3/s
W = width of airway, m
g = acceleration due to gravity
ρ_a = density of air
ρ_m = density of methane

$$g * \frac{\rho_a - \rho_m}{\rho_a} = 4.32$$

For L> 2, in a horizontal airway causes turbulent diffusion and breaks up the methane layer.

The index can also be determined by,

$$L = \left(\frac{24 * v^2}{c\sqrt{F}} \right)^{1/3}$$

Where v = average velocity of air, m/s
c = mean methane content of air current (% CH_4)
F = area of cross section at the measuring station, m^2

3.10 Graham's Ratio and its Importance

It is the ratio of CO produced to O_2 consumed in the mine atmosphere. The ratio indicates the spontaneous heating and its stage.

Graham's ratio = CO produced/ Oxygen consumed

→ 0.1 – 0.5 % is normal to coal mine
→ 1 % indicates existence of spontaneous heating
→ 2 % indicates heating in advanced stage approaching active fire
→ 3 % or more indicates active fire

3.11 Rescue/ Escape/ Reviving Apparatus

A person doing normal work breathes at the rate of 15 breath/ min and consumes 1.5 litres of O_2 /min. The rescue and reviving apparatus are also designed in such a way that it can supply 1.5 litres of O_2/min. The apparatus used in Indian mines are given in Table 3.3.

Table 3.3 : Rescue/Escape/Reviving Apparatus

Name of the apparatus	Model	Characteristics
Proto self contained compressed oxygen apparatus	Proto Mark IV	o Closed circuit operation o 300 lit. of O_2 compressed at 150 kg/cm^2 o 2 kg CO_2 absorbent known as PROTOSORB ($Ca(OH)_2$ + NaOH) o 2 hrs. use o Weight 15.6 kg
	Proto Mark V	o Closed circuit operation o 300 lit. of O_2 compressed 150 kg/cm^2 o 1 hr. and 2 hrs. use capacity o Feed rate 2 lit. O_2/min in 2 hrs. apparatus and 2.5 lit. O_2/min in 1 hr. apparatus o Weight (14.5 kg for 1hr and 17.2 kg for 2 hrs.)
Drager self contained compressed oxygen apparatus (S.C.B.A)	BG 174	o Closed circuit operation o Weight 12.8 kg o 4 hrs. use o 400 lit. of O_2 compressed 200 kg/cm^2 o Feed rate 1.5 lit./min
Chemical Oxygen Breathing Apparatus	AUER/SAR - 30	o Closed circuit operation o 30 minutes use o It consists of a Canister where KO_2 is stored o O_2 needed for breathing produced from the chemical reaction as follows $2\ KO_2 + H_2O \rightarrow 2\ KOH + 3/2\ O_2$ $2KO_2 + CO_2 \rightarrow K_2CO_3 + 3/2\ O_2$
Self Rescue Breathing Apparatus	SPIRAL 2C (France)	o Closed circuit operation o Weight is 3 kg o 1 -4 hour use depends on the work performed o It consists of a Canister where KO_2 is stored o Working principle is the same as AUER/ SAR 30
Self Rescuer		o Open circuit operation i. e., O_2 is not supplied by the self rescuer o Weight is very low - 1 kg o It works on the principle of conversion of CO to CO_2 [

		o It is used to escape during fire occurs o It can be used in place where maximum CO is 2% o Maximum 1 hr. use
Gas Mask		o Open circuit operation i. e., O_2 is not supplied by the self rescuer o Weight is very low - 1 kg o It works on the principle of conversion of CO to CO_2 [Hopcalite (MnO_2+Cu_2O) catalyst is used] o It is used to escape during fire occurs o It can be used in place where maximum CO is 2% o Maximum 1 -2 hr. use
Reviving Apparatus	Pulmometer	o It is used to supply pure O_2 to unconscious person

3.12 Air Flow through the Mine Openings

3.12.1 Nature of Flow

There are two types of flow namely streamline & turbulent and the nature of flow is governed by a dimensionless parameter called the Reynolds number (Re).

$$Re = \frac{VD\rho}{\mu} = \frac{VD}{\nu}$$

Where, V = velocity of flow
D = diameter of the duct
ρ = density of the air
μ = viscosity of the air
$\nu = \mu / \rho$ = kinematics viscosity

For square shaped duct, D = 4A/P
A = area of cross section of the duct
P = perimeter of the duct

3.12.2 Pressure Drop in Mine Airway : Frictional pressure drop during the incompressible flow of air is given by

$\Delta P = f L V^2 \rho / 2D = f L \rho V^2 \rho / 8A = k S V^2 / A = k S Q^2 / A^3$, $R = Q^2$

L = length of the flow
V = velocity of the flow
ρ = density of the air
D = diameter of the duct = 4A / P
P = Perimeter of the duct
A = Area of cross section of the duct
k = coefficient of friction = f ρ/8 [Ns^2m^{-4}]
S = Area of the rubbing surface (m^2) = L*P
Q = flow through the airway (m^3/s)
R = resistance of the path = k S / A^3

For the flow to be laminar, Re $\leq$ 2000 and f is given by

$$f = 64/Re = 64*\mu/VD\rho$$

3.12.3 Shock Loss at Area Changes and Bends : The shock loss is given by

$$\Delta P_s = X * P_v = X* \rho V^2 / 2$$

X = shock factor (dimensionless constant)
P_v = velocity pressure

3.12.4 Coefficient of Contraction : The coefficient of contraction during area changes is given by

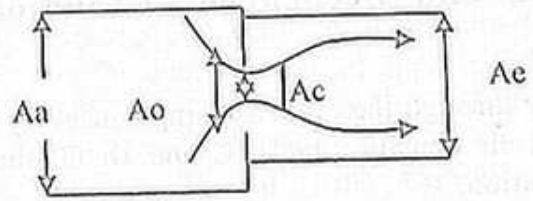
$$\text{Coefficient of contraction (C)} = \sqrt{\frac{1}{Z - Z * N_C^2 + N_C^2}}$$

Z = contraction factor

$N_c = \frac{A_0}{Aa}$ = ratio of contraction

$N_e = \frac{A_0}{Ae}$ = ratio of expansion

C = A_c / A_o = coefficient of contraction



3.13 Equivalent Resistance

- o Equivalent resistance of a number of roadways in series is given by (Rs) = $R_S = R_1 + R_2 + R_3 + \text{———} Rn$
- o Equivalent resistance of a number of roadways in parallel (R_p) is given by

$$\frac{1}{\sqrt{R}} = \frac{1}{\sqrt{R}} + \frac{1}{\sqrt{R}} + \frac{1}{\sqrt{R}} + \ldots\ldots + \frac{1}{\sqrt{R}}$$

Where,

R_S = equivalent resistance for n number of roadways situated in series

R_P = equivalent resistance for n number of roadways situated in parallel

$R_1, R_2 \ldots R_n$ are the resistances of individual roadways

3.14 Air Power (AP)

Power of ventilation or air power is the rate at which work is done to maintain the air flow through a system and is given by

Air Power (AP) = $\Delta P * Q = R\,Q^2 * Q = R\,Q^3$ [Watt]

Where Q = quantity of air flow in m^3/s

ΔP = pressure drop in Pa.

3.15 Equivalent Orifice

The resistance of a mine airway can be expressed in terms of the equivalent orifice and is given by

$$A = \frac{1.2 * Q}{\sqrt{P}} = \frac{1.2}{\sqrt{R}}$$

Where, A = area of equivalent orifice

Q = quantity of air flow, m^3/s

P = pressure of air flow

R = resistance of the roadways

3.16 Methods of Velocity Measurement

- o Single point measurement
- o Continuous traversing
- o Precise traversing

3.17 Natural and Mechanical Ventilation:

3.17.1 N.V.P

Air flow through the mine opening causes by natural means (difference in air densities in U. C and D. C. shaft) called the natural ventilation

$$N.V.P. = D\,(\rho_d - \rho_u)\,g \qquad (Pa)$$

Where, D = depth of the shaft

$$\rho_d = \text{density of air in D.C. Shaft} = \frac{(B_d - 0.378 e_d) * 1000}{287.1 * T_d}\, kgm^{-3}$$

$$(B_u - 0.378 e_d) * 1000$$

Where,

B_d and B_u = mean barometer reading in kPa in the D.C and U.C. shafts respectively

e_d and e_u = average vapour pressures of moisture in kPa in the D.C and U.C. shafts respectively.

T_d and T_u = mean temperatures of the D.C and U.C. shafts respectively in K.

For mine shafts $B_d = B_u = B$ and neglect the effect of moisture, the above equation can be rewritten as

$$N.V.P. = \frac{gDB(T_u - T_d) * 10^3}{287.1 * T_u * T_d} \quad (Pa)$$

3.17.2 Motive Column

Height of air column in the D.C. shaft which causes the N.V.P. termed as motive column.

$$\text{Motive Column (h)} = \frac{N.V.P}{\rho_d * g}$$

Where, h = height of motive column in m

N.V.P in Pa

ρ_d in kg/m^3

g in m/s^2

3.17.3 Head (H_e) Generation by Mine Fan

The velocity triangle at the inlet and outlet of a centrifugal impeller is shown in Figure 3.2. Euler's head (H_e) generated by the mine fan is

$$H_e = \frac{U_2(U_2 - V_{R2}\cot\beta_2)}{g} - \frac{U_1(U_1 - V_{R1}\cot\beta_1)}{g}$$

Where, U = the peripheral velocity of the impeller in m/s

V = the absolute velocity of flow in m/s

W = the relative velocity of flow of air to that of the impeller in m/s

V_R = radial component of the absolute velocity of flow

V_U = tangential component of the absolute velocity of flow

r_1 = radius of the impeller inlet in m

r_2 = radius of the impeller outlet in m

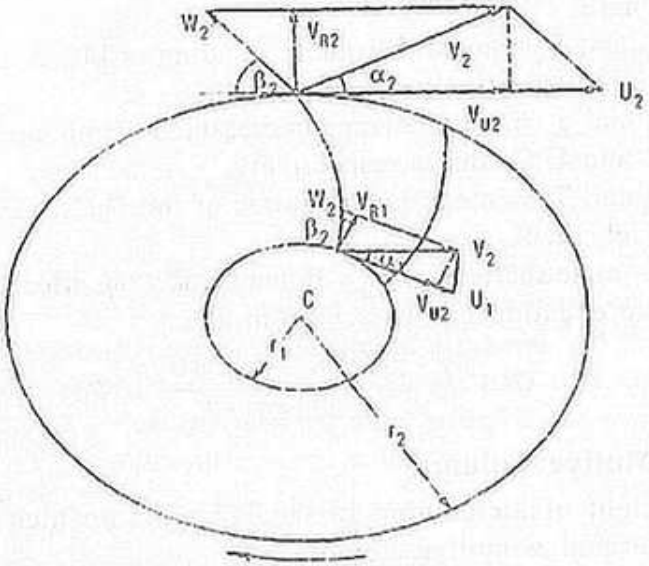


Figure 3.2 : Velocity triangles at the inlet and outlet of centrifugal impeller

For the air has no pre-rotation before entering the impeller,

$$V_{U1} = 0 \text{ and } V_{R1} = V_1$$

Therefore the above equation can be rewritten as

$$H_e = H_{e2} = \frac{U_2(U_2 - V_{R2}\cot\beta_2)}{g}$$

In general, $H_e = \dfrac{U(U - V_R\cot\beta)}{g}$

The head characteristics curve and the Power characteristic curve are shown in Figure 3.3 and Figure 3.4 respectively.

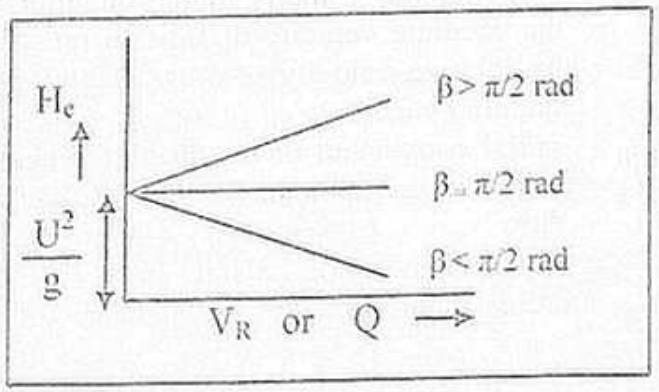


Figure 3.3 : Head vs. Quantity

Where, $U = 2\pi rn = \omega r$ [n = speed of the impeller in r.p.s.]

Also, V_R = Q/area of discharge of impeller = $Q/2\pi rB$

[B = width of impeller in m]

Again, Power (P_e) = He* (ρ*g) * Q = He*K*V_R =

$$K = \frac{\left(UV - UV_R^2\cot\beta\right)}{g}$$

[Where, K is constant = ρg and $V_R \infty Q$ for a particular impeller]

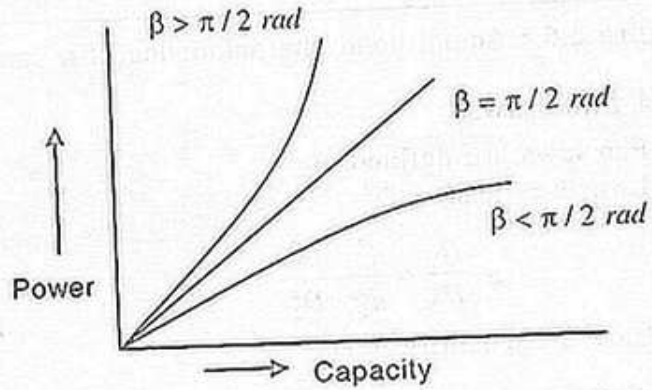


Figure 3.4 : Power vs. Capacity characteristics

The actual head developed by a fan is equal to the theoretical head developed minus the head loss due to friction, diffusion and shock. The various hydraulic losses due to friction, diffusion and shock are shown in Figure 3.5 and the actual head characteristic is shown in Figure 3.6

Actual head developed by the fan =

$H = H_i$ - Friction loss – Diffusion loss - Eddy and separation loss (shock loss)

$= H_i - K_1 Q^2 - K_2 Q^2 - K_3 (Q - Q_S)^2$

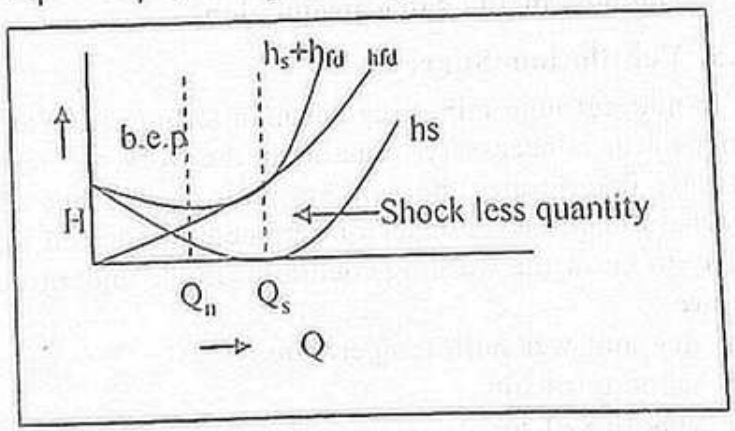


Figure 3.5 : Hydraulic losses in an impeller

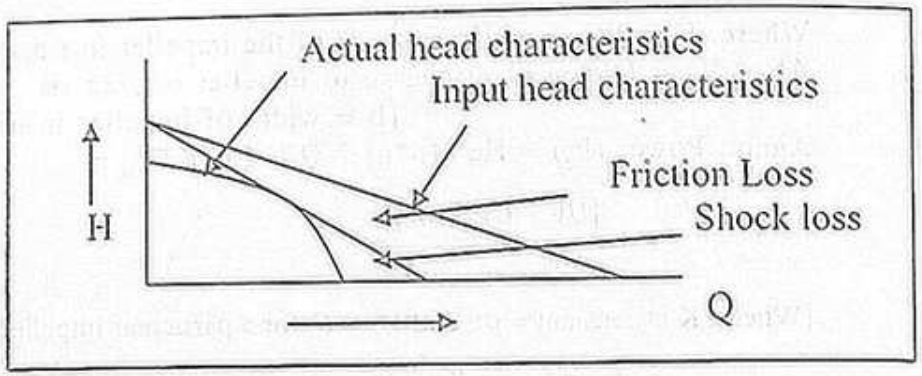


Figure 3.6 : Actual head characteristics of a centrifugal fan

3.17.4 Fan Laws

Fan laws are defined as

Law 1 : Head α D^2

$$\frac{H_1}{H_2} = \frac{n_1^2 * D_1^2}{n_2^2 * D_2^2}$$

Law 2 : Quantity α D^3

$$\frac{Q_1}{Q_2} = \frac{n_1 * D_1^3}{n_2 * D_2^3}$$

Law 1 : Power α D^5

$$\frac{P_1}{P_2} = \frac{n_1^3 * D_1^5}{n_2^3 * D_2^5}$$

Where, H = head; Q = Quantity; P = power; n = speed, D = diameter of the impeller

[The above relations are true for geometrically similar impeller, i.e. the diameter and the width of the impeller changes in the same proportion]

3.17.5 Ventilation Survey

In underground mines regular monitoring of the quality and quantity of air is necessary to maintain the working conditions as it not only deteriorates the workers performance but also effect the workers health. A number of parameters required to monitor regularly to know the working conditions in the underground area. These are

- dry and wet bulb temperature
- velocity of air
- presence of noxious gases in the mine atmosphere
- barometric pressure
- pathogenic dust concentration
- humidity

As per Indian mine regulations, in all mechanically ventilated mines, quantity measurements should be made every 14 days in case of gassy coal mines and every 30 days in case of non-gassy coal mines. The measurements should be made in the main intake and main return of the seam, in the splits near the starting points and in ventilation districts. There is also provision of monitoring temperature, humidity and other environmental conditions in every 30 days.

3.17.6 Measurement of Velocity of Air Flow

- **Smoke Cloud Method** : Time taken to cover a particular distance S, shown in Figure 3.7, by smoke has been recorded and velocity can be calculated by

  V = S / T

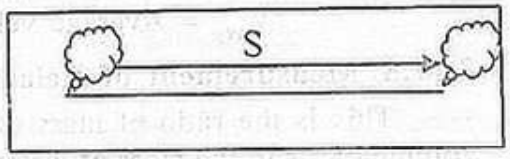


Figure 3.7 : Smoke Cloud Method

- **Hot-wire Anemometer** : This is a very sensitive instrument for measuring low air velocities.
- **Vane Anemometer** : This simply gives the distance travelled by the vanes set in the anemometer at a particular duration of time. The velocity thus calculated indirectly by dividing the distance by time.
- **Velometer** : The velometer consists of a double-pivoted spring-balanced aluminium vane mounted in a jeweled bearing bakelite case. This directly gives the velocity of air movement.
- **The Pitot-static tube** : It comprises of two concentric tubes, the outer of which has a few holes on the sides. The annular opening between the two tubes at the nose end is sealed so that the inner tube records the total pressure and the outer tube records the static pressure respectively. The velocity pressure can be calculated by subtracting the static pressure from the total pressure. The velocity of the air can be calculated by the following formula

$$V = \sqrt{\frac{2P_v}{K\rho}}$$

Where, V = velocity of air in m/s
P_V = velocity pressure in Pa
ρ = density of air in kg/ m^3
K = correction factor [for standard design K = 1]

3.17.7 Measurement of Quantity of Air Flow

This requires the measurement of average air velocity at any point of mine airway and the cross-sectional area of the airway at that point. The quantity of air flow can be calculated by multiplying the above two factors. The average velocity can be calculated by multiplying the air velocity monitored at the centre of the duct with the method factor.

i.e., $Q = A \cdot V_{avg}$,
Where, Q = Quantity of air flow
A = Area of Cross section
V_{avg} = Average velocity

3.17.8 Measurement of Relative Humidity

This is the ratio of mass of water vapour present per unit volume of air to the mass of water vapour required to saturate the unit volume of air. This can be directly measured with the help of Hygrometer. It can also be calculated from the reading of dry bulb and wet bulb temperature.

Relative Humidity =
[100 – 7* (D.B.T. – W.B. T)] [For D.B.T. > 25°C]
[100 – 8* (D.B.T. – W.B. T)] [For 20°C < D.B.T. < 25°C]
[100 – 9* (D.B.T. – W.B. T)] [For D.B.T. < 20°C]

EXERCISE

1. In an underground working area, CH_4 and CO_2 are normally expected to be found respectively
   (a) Near the floor and along the roadway
   (b) Near the floor and near the roof
   (c) Near the roof and near the floor
   (d) Along the roadway and near the floor
2. The oxygen concentration in u/g working should not be less than
   (b) 20 %
3. The maximum wet bulb temperature in u/g mines is
   (a) 30°C (b) 30.5°C
   (c) 33°C (d) 33.5°C
4. The wet bulb temperature of air at a particular place in a mine is 31°C, then the velocity of air in that place should be
   (a) more than 1 m/s (b) less than 1 m/s
   (c) more than 2 m/s (d) less than 2 m/s
5. Quantity of air to be circulated to u/g workings depends on
   (a) Manpower employed in a largest shift
   (b) Manpower employed in a day
   (c) Total production in a day
   (d) Both (a) and (c)
6. Quantity of air to be circulated to u/g workings where 150 worker employed in a largest shift and daily production 500 te will be
   (a) 900 m^3/ min (b) 1250 m^3/min
   (c) 900 m^3/ sec (d) 1250 m^3/sec
7. An air sample produces 8 % CH_4 and 3 % H_2. If the lower flammability of CH_4 is 5 % and H_2 is 4 %, what is the lower flammability of the mixture?
   (a) 9.50 % (b) 7.16 %
   (c) 4.68 % (d) 3.90 %
8. 20 m^3/s of air flow through a longwall panel having 1000 tpd of production. Between the intake and return of the panel, the firedamp increases from 0 to 0.2 %. The rate of firedamp emission is
   (a) 3.45 m^3/day (b) 3.45 m^3/s
   (c) 3.45 m^3/t (d) 3.45 l/g
9. A man exhales 0.002 m^3 of CO_2 per min when working in a confined space in underground. Assuming that the fresh air contains 0.03 % CO_2, the minimum quantity of fresh air to be supplied per person in m^3/min, so that concentration of CO_2 in air never crosses 0.4 %, will be
   (a) $0.5 * 10^{-3}$ (b) $5.4 * 10^{-3}$
   (c) $540 * 10^{-3}$ (d) $66 * 10^{-3}$
10. Air at dry bulb temperature of 27°C enters a 1200 m deep shaft. The specific heat of air-water vapour mixture is 1.03 KJ/kg °C. If the air goes through constant-moisture adiabatic

shaft in °C is
(a) 11.4 (b) 33.1
(c) 35.4 (d) 38.4

11. Dry air enters a stope at a mass flow rate of 20 kg/s. The machines working in the stope add 50 W of sensible heat and 100 kW of latent heat. The average latent heat of evaporation of water is 2500 kJ/kg. The increase in the mixing ratio in kg/kg is
(a) 0.800 (b) 0.003
(c) 0.002 (d) 0.001

12. The mean velocity of air current through an airway of 9 m^2 cross-sectional area is found to be 1.5 m/s. If the mean methane content of air current is 0.6 %, the layering index will be
(a) 3.10 (b) 2.71
(c) 2.15 (d) 1.80

13. Air of 20.8 % O_2 concentration enters a coal mine panel at 20 m^3/s and exits with 20.6 % O_2. The oxygen depletion is due to oxidation of carbon in spontaneous heating process only, assuming the density of oxygen as 1.3 kg/m^3 and the calorific value of carbon as 34 MJ/kg, the heat generated in kW from the panel due to spontaneous oxidation is
(a) 533 (b) 640
(c) 663 (d) 693

14. A sealed of area known to have black damp has 18 % O_2, 2 % CO_2, and 78 % N_2. If fresh air has 21 % O_2, 0.03 % CO_2 and 78.97 % N_2, the percentage of black damp in the area is
(a) 10.31 (b) 12.28
(c) 12.31 (d) 14.28

15. Which one of the following composition of methane in air is most explosive in nature?
(a) 8.5 % by volume (b) 9.5 % by volume
(c) 10.5% by volume (d) 11.5% by volume

16. The percentage of inflammable gas in an u/g coal mine should not exceed
(a) 0.75 % in the general body of the return air of any ventilation district and 1.5 % in any place of the mine
(b) 0.75 % in the general body of the return air of any ventilation district and 1.25 % in any place of the mine
(c) 0.5 % in the general body of the return air of any ventilation district and 1.25 % in any place of the mine
(d) 0.5 % in the general body of the return air of any ventilation district and 1.5 % in any place of the mine [CIL 01, 07]

17. If the CO/ O_2 deficiency ratio in a mine is 2 %, it indicates
a. existence of spontaneous heating
b. active fire
c. heating in advanced stage
d. None of the above [CIL 01]

18. In an underground coal mine area the graham's ratio found 3 % is the indication of
a. existence of spontaneous heating
b. active fire
c. heating in advanced stage
d. normal to the coal mine

19. The early indication of spontaneous heating of coal can be given by
a. Sweating of strata b. Formation of CO_2
c. Rise of temperature d. None of the above [CIL, 01]

20. Ignition temperature of methane is
a. 100 – 1250°C b. 923 – 1023°C
c. 675 – 975°C d. 423 – 1000°C

21. The lower and upper limit of methane – air explosive mixture is
a. 0.75 %, 1.25 % b. 5.4 %, 14.8 %
c. 5.0 %, 15.0 % d. 5.9 %, 12.5 %

22. Geothermic gradient is
a. Increase in temperature with depth
b. Increase in pressure with depth
c. Decrease in temperature with depth
d. Decrease in pressure with depth

23. Which of the following instrument is used to measure the cooling power of the air?
a. Anemometer b. Velometer
c. Manometer d. Kata thermometer

24. Which one of the following is used as a reviving apparatus in u/g coal mine?

a. Draeger BG 172 b. Pulmometer
c. Any open circuit rescue apparatus
d. MSA self rescuer

25. Which one of the following is used as an escape apparatus in u/g coal mine?
a. Draeger BG 172 b. Pulmometer
c. Any open circuit rescue apparatus
d. MSA self rescuer

26. The chemical Hopcalite used in
a. Self contained breathing apparatus
b. Gas mask
c. CO- detector
d. MSA- methanometer

27. The chemical protosorb used in
a. Self contained breathing apparatus
b. Gas mask
c. Reviving apparatus
d. MSA- methanometer

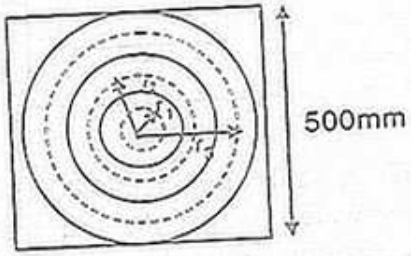
28. Oxygen flow rate in the self contained breathing apparatus Proto Mark IV is
a. 1 L/ min b. 1.5 L/ min
c. 2 L/ min d. 2.5 L/ min

29. Samples of air collected in the intake and return gate roads of an advancing longwall face show 0.25 % and 0.75 % CH_4, respectively. The panel has a production of 400 t in an 8 hour shift when an air quantity of 18 m^3/ s passes through it. Then the methane emission in m^3/ te of coal mined is
a. 0.02 b. 0.81
c. 2.25 d. 6.48

30. For the same roadway roughness, area, length and velocity condition the ratio of resistances of a square roadway to that of a rectangular roadway having width and height ratio, 2 : 1 is
a. $\frac{1}{\sqrt{3}}$ b. $\frac{2\sqrt{2}}{3}$
c. $\frac{3\sqrt{2}}{2}$ d. $\frac{\sqrt{2}}{3}$

mine management decided to reduce the flow rate of air to 20 m^3/s using a regulator without changing the ventilation pressure. The area of regulator expressed in m^2 will be
a. 0.99 b. 0.77
c. 0.71 d. 0.60
[GATE, 04]

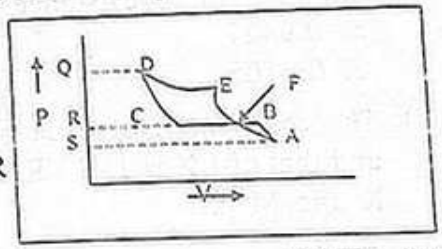
32. While measuring average air velocity of air flowing through a circular duct using precise traversing with equal area method, the radii r1, r2 and r3, in mm, as shown in the figure of the three circles should be
a. 83, 167 and 240
b. 36, 107 and 261
c. 102, 177 and 228
d. 108, 157 and 216



33. When two identical and parallel intake roadways are changed to three identical and parallel intake roadways of same size and length, the ratio of new equivalent resistance to old equivalent resistance will be
a. 0.44 b. 0.67
c. 1.50 d. 0.82

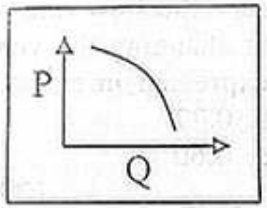
34. A mine fan delivers 50 m^3/s of air at a fan drift of 10 mm of w.g. It is proposed to increase the quantity to 60 m^3/s of air, the required w.g. shall be
a. 12.0 mm b. 14.4 mm
c. 13.6 mm d. 20.2 mm
[CIL 2001]

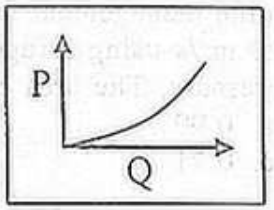
35. The figure shows PV (pressure-volume) diagram of a mine ventilated by both natural means and an exhaust fan. The total ventilation energy is represented by
a. area ABF + area CDEF
b. area ABRS + area CDEF
c. area ABRS + area QDCR
d. area ABF + area QDCR

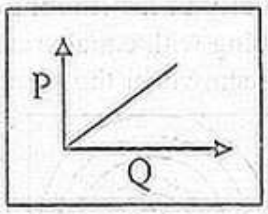


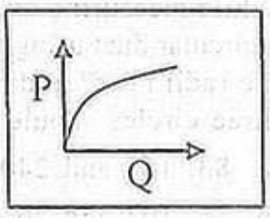
[GATE, 04]

36. The plot of theoretical power consumed against quantity delivered by a forward bladed centrifugal fan is best represented by

a. 

b. 

c. 

d. 

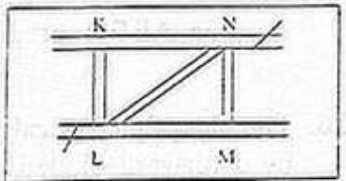
[GATE, 04]

37. A fan running at 200 rpm develops a pressure of 50 mm water gauge; the fan speed is increased to 300 rpm. The water gauge developed will be
a. 75 mm b. 112.5 mm
c. 164 mm d. 97 mm

38. A mine fan produces 30 m^3/s of air at 240 Pa pressure running at 1400 rpm. If the fan speed is increased to 2100 rpm, the pressure in Pa and the quantity in m^3/s generated by the fan will be
a. 540 and 67.5 b. 540 and 45.0
c. 360 and 45.0 d. 540 and 30.0
[GATE, 04]

39. The fan drift pressure readings and fan quantities are measured to be 500 Pa and 65 m^3/s respectively at the time when NVP aids fan pressure. On reversal, the fan delivers a quantity of 50 m^3/s while acting against NVP with fan drift pressure of 600 Pa. Assuming that the NVP remains constant in both cases the total resistance of the mine in Ns^2/m^8 will be
a. 0.9487 b. 0.2750
c. 0.4105 d. 0.1635 [GATE, 04]

40. The resistances of each branches KL, LM, MN, NK is R and that of LN is 1.414 R. The equivalent resistance between K and M is
a. R
b. 0.5 R
c. 0.75 R
d. 2 R



41. The blade tip speed of a backward bladed centrifugal fan running at 300 rpm delivers 120 m^3/s of air. If the tip speed of the blade is 47.12 m/s and width of the fan is 1.3 m then the radial velocity of air at the periphery of the impeller in m/s is
a. 9.79 b. 12.73
c. 30.79 d. 40.22
[GATE, 04]

42. A main fan of a mine circulates 40 m^3/s of air at a pressure of 500 Pa. The mine has two splits of equal resistance. The trunk airways have a total resistance of 0.2 Ns^2/m^8. When a booster fan is installed in one split, the flow of air in the other splits stops. Assuming that the pressure remains unchanged, the new air quantity flowing through the mine in m^3/s is
a. 20 b. 30
c. 40 d. 50
[GATE, 06]

Answer

1. c	2. c	3. d	4. a	5. d	6. b
7. c	8. c	9. c	10. d	11. a	12. a
13. c	14. b	15. b	16. b	17. c	18. b
19. a	20. b	21. b	22. a	23. d	24. b
25. c	26. b	27. a	28. c	29. d	30. b
31. a	32. c	33. a	34. b	35. a	36. b
37. b	38. b	39. d	40. b	41. a	42. c

□

4.0 Mine Environment

4.1 Air Pollution

4.1.1 Definition

Air pollution is the excessive concentration of foreign matter in the air which adversely affects the well being of the individual or causes damage to the property. The (PM_{10}) and ($PM_{2.5}$) Particulate Matter, Sulphur Dioxide (SO_2) and Oxides of Nitrogen (NO_X) concentration in downwind direction considering predominant wind direction, at a distance of 500 m from the following dust generating source (Mines) shall not exceed the standard specified in the table given below.

4.1.2 Air Pollutants Standards

The revised air pollution standards as per CPCB, New Delhi published in November, 2009 are given in Table 4.1.

Table 4.1 : Air Pollutants Standards

Pollutant	Time weighted average	Concentration in Ambient Air	
		Industrial, Residential, Rural and other area	Ecologically Sensitive area (notified by Central Government)
Particulate Matter (Size less than 10 μm) or PM_{10}	Annual*	60 $\mu g/m^3$	60 $\mu g/m^3$
	24 hours**	100 $\mu g/m^3$	100 $\mu g/m^3$
Particulate Matter (Size less than 2.5 μm) or $PM_{2.5}$	Annual*	40 $\mu g/m^3$	40 $\mu g/m^3$
	24 hours**	60 $\mu g/m^3$	60 $\mu g/m^3$
Sulphur Dioxide (SO_2)	Annual*	50 $\mu g/m^3$	20 $\mu g/m^3$
	24 hours**	80 $\mu g/m^3$	80 $\mu g/m^3$
Oxides of Nitrogen (NO_2)	Annual*	40 $\mu g/m^3$	30 $\mu g/m^3$
	24 hours **	80 $\mu g/m^3$	80 $\mu g/m^3$
CO	8 hours**	2 mg/m^3	2 mg/m^3
	1 hour**	4 mg/m^3	4 mg/m^3
Ozone (O_3)	8 hours**	100 $\mu g/m^3$	100 $\mu g/m^3$
	1 hour**	180 $\mu g/m^3$	180 $\mu g/m^3$
Lead (Pb)	Annual*	0.5 $\mu g/m^3$	0.5 $\mu g/m^3$
	24 hours**	1.0 $\mu g/m^3$	1.0 $\mu g/m^3$
Ammonia (NH_3)	Annual*	100 $\mu g/m^3$	100 $\mu g/m^3$
	24 hours**	400 $\mu g/m^3$	400 $\mu g/m^3$

* Annual arithmetic mean of minimum 104 measurements in a year taken twice a week 24 hourly at uniform interval.

**24 hourly or 8 hourly or 1 hourly monitored values, as applicable, shall be complied with 98 % of the time in a year, 2 % of the time, that may exceed the limit but not on two consecutive days of monitoring.

4.1.3 Sources of Air Pollution in Mining Industries

The main sources of dust generation in the mining industries are

- Haul road
- Drilling operation
- Loading plant
- Crushing plant

4.1.4 Air Pollution Control Devices

Gravity Settler : The simplest kind of air cleaning device is the settling chamber, which is shown below in Figure 4.1. It is

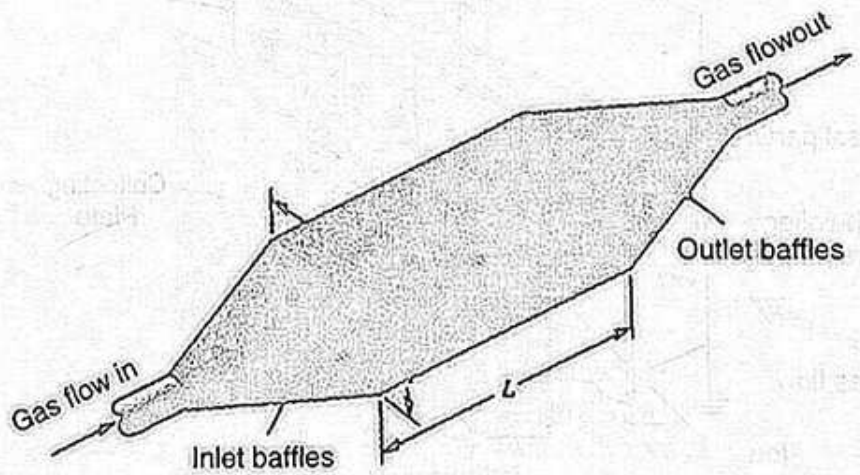


Figure 4.1 : Gravity Settler

basically an enlarged section or compartment in a flue in which the velocity of the carrier gas is reduced. When the gas velocity is slowed sufficiently, coarse particles (> 40 μm in diameter) can settle out by gravity. A settling chamber of this type usually serves as a pre-cleaner, to prevent clogging of the more efficient small particle collectors that must follow it. Baffles are usually added inside the chamber to increase settling and particulate removal efficiency.

To compute the efficiency-particle diameter relationship, we get

$$\eta = \frac{L\left(gD^2\rho_p/18\mu\right)}{HV_{avg}}$$

Where, V_{avg} = Horizontal velocity of the gas in the chamber

$$= \frac{Q}{WH}$$

D = diameter of the particulate matter.
ρ_p = density of the particulate matter
μ = coefficient of viscosity
H = Height of the gravity settling chamber
L = length of the gravity settling chamber
W = width of the gravity settling chamber

➢ **Electrostatic Precipitator :** The electrostatic precipitator (ESP) is like a gravity settler or centrifugal separator, but

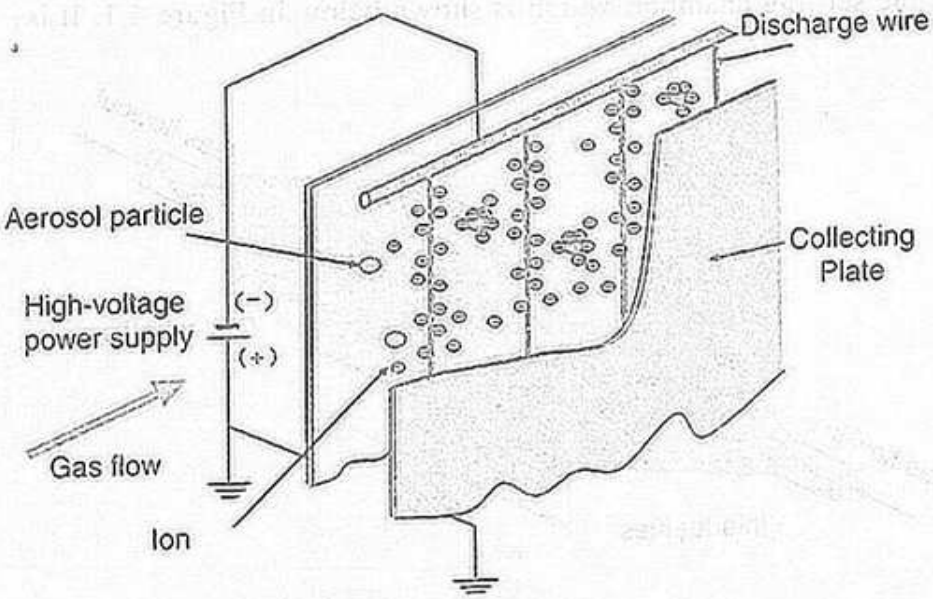


Figure 4.2 : Electrostatic precipitator

electrostatic force drives the particles to the wall. The working mechanism and components of an ESP is shown in Figure 4.2. It is effective on much smaller particles than the gravity settling chamber and centrifugal separator. In all three kinds of devices, the viscous (Stoke's law) resistance of the particle to being driven to the wall is proportional to the particle diameter. In ESPs the resisting force is proportional to the diameter.

To compute the efficiency-particle diameter relationship, we get

$$\eta = 1 - \exp\left(\frac{V_t A}{Q}\right) \rightarrow \textit{Deutsche– Anderson equation}$$

Where, V_t = terminal settling velocity of the particulate matter
A = Area of cross section of the plates
Q = Rate of flow of polluted gas.

➢ **Cyclone Separator :** It is probably the most widely used particle collection device in the world. A cyclone consists of a vertical cylindrical body, with a dust outlet at the conical bottom as shown in Figure 4.3. The gas enters

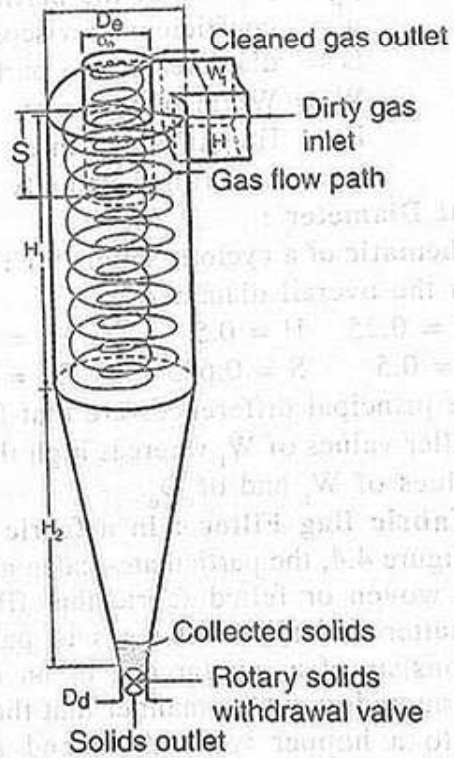


Figure 4.3 : Cyclone Separator

through a rectangular inlet, normally twice as high as it is wide, arranged tangentially to the circular body of the cyclone, so that the entering gas flows around the circumference of the cylindrical body, not radially inward. The gas spirals around the outer part of the cylindrical body with a downward component, then turns and spiral upward, leaving through the outlet at the top of the device. During the outer spiral of the gas the particles are driven to the wall by centrifugal force, where they collect, attach to each other, and form larger agglomerates that slide down the wall by gravity and collect in the dust hopper in the bottom.

Efficiency of particulate matter removal in a cyclone separator can be determined by

$$D_{cut} = \left(\frac{9W_i\mu}{2\pi N V_c \rho_p} \right)^{1/2}$$

Where, N = Effective number of turns gas makes in traversing the cycloe.

ρ_p = density of the particulate matter

μ = coefficient of viscosity

D = diameter of the particulate matter

W_i = Width of the inlet

H = Height of the inlet

V_c = centrifugal velocity

Cut Diameter :

Schematic of a cyclone separator : Dimensions are typically based on the overall diameter D_0.

Wi = 0.25 H = 0.5 H_1 = 2 H_2 = 2

D_e = 0.5 S = 0.625 D_d = 0.25

The principal differences are that high efficiency cyclones have smaller values of W_i whereas high throughout cyclones have larger values of W_i and of D_e.

➢ **Fabric Bag Filter :** In a fabric filter system shown in Figure 4 4, the particulate- laden gas stream passes through a woven or felted fabric that filters out the particulate matter and allows the gas to pass through. It typically consists of a tubular bag or an envelope, suspended or mounted in such a manner that the collected particles fall into a hopper when dislodged from the fabric. Small particles are initially retained on the fabric by direct interception, inertial impaction, diffusion, electrostatic attraction, and gravitational settling. After a dust mat has formed on the fabric, more efficient collection of submicron particles (+99%) is accomplished by sieving.

Filter bags, usually tubular or envelope- shaped, are capable of removing most particles as small as 0.5 μm and will remove substantial quantities of particles as small as 0.1 μm. Filter bags ranging from 1.8 to 9 m long, can be utilized in a bag-house filter arrangement similar to shown below.

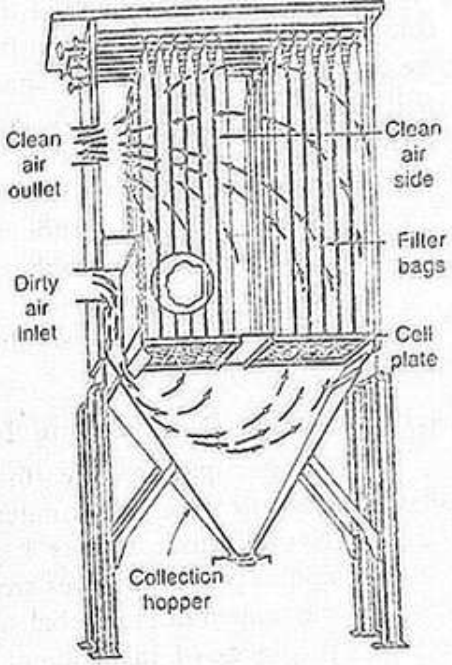


Figure 4.4 : Fabric-filter Baghouse

∘ **Wet Scrubber:** Scruboers effectively divide the flow of particle-laden gas by sending many small drops through it. In air pollution control engineering, the term scrubber originally

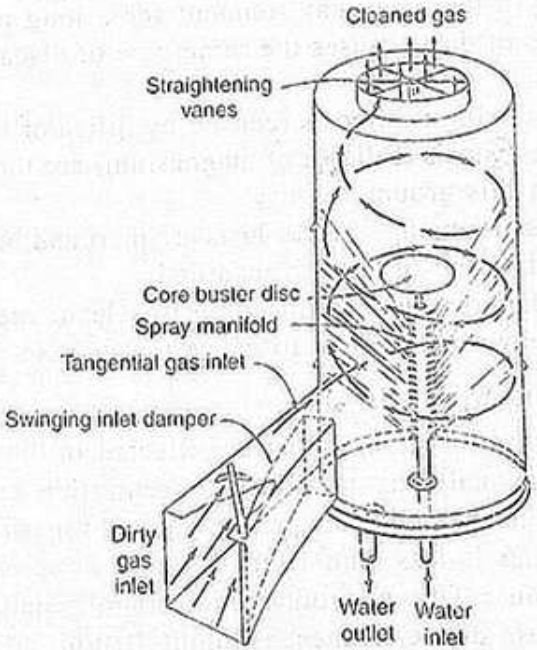


Figure 4.5 : Cyclone spray scrubbers

meant a device for collecting fine particles on liquid drops. Most fine particles will adhere to a liquid drop if they contact it. So if we can make the drop and the particle touch each other, the particle will caught on the drop. A typical Cyclone spray scrubbers is shown in Figure 4.5.

There are several types of wet scrubbers as

- Spray-chamber scrubbers
- Cyclone spray scrubbers
- Orifice scrubbers
- Wet-impingement scrubbers
- Venturi scrubbers

4.1.5 Miner's Diseases and Dust Hazards

Mine occupational health hazards leads to the following diseases due to poor environmental working conditions :

A. **Nystagmus :** This is a type of disease in which muscles and nerves of the eyes are affected and there is an abnormal movement of the eyeballs. This disease is caused by working in places of insufficient light.

B. **Ankylostomiasis :** This refers to the disease like anemia or hookworm disease and is caused by blood sucking worm enters to the body during working in insanitary conditions or standing in dirty water with bare foot.

C. **Pneumoconiosis :** This is a type of lungs diseases caused by working in the dusty environment for a long period.

Different types of dusts causes the same type of disease but can be classified as :

i. Fibrosis producing : Silicosis (caused by Silica or Quartz dust) and Asbestosis (Silicate of magnesium) are the most important in this group.

ii. Non-Fibrosis producing : These dusts are inert and become encapsulated in the tissues or absorbed.

iii. Toxic producing : This group comprises lead, mercury and other heavy metals dust to cause the disease.

4.1.6 Dust Sampling Methods

- **Filtration :** In this method dust is collected in the filter paper and on analyzing this dust concentration can be calculated. This method is adopted in general for particles whose diameter is less than 10μm.
- **Sedimentation :** This method is adopted in general for particles whose diameter exceeds about 10 μm.
- **Impingement Method :** Separation of particulates from an air stream by abrupt collision against a flat surface forms the basis of this method.

4.2 Noise Pollution

4.2.1 Definition

Sound (or noise) is the result of pressure variations, or oscillations, in an elastic medium (e.g., air, water, solids), generated by a vibrating surface, or turbulent fluid flow. Sound propagates in the form of longitudinal (as opposed to transverse) waves, involving a succession of compressions and rarefactions in the elastic medium. When a sound wave propagates in air (which is the medium considered in this document), the oscillations in pressure are above and below the ambient atmospheric pressure. The unit of sound pressure level is Pa. Sound waves which consist of a pure tone only are characterised by the following parameters given in Figure 4.6.

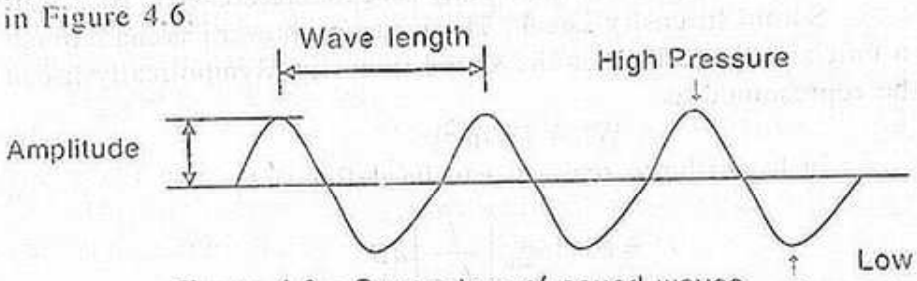


Figure 4.6 : Parameters of sound waves

The speed of a wave is equal to the product of its wavelength and frequency.

$$\upsilon = f * \lambda$$

Where, υ = speed of sound, m/s
f = frequency, Hz
λ = wavelength, m

4.2.2 Measurement of Noise

The human ear is a remarkably sensitive organ. The audible frequency range for the human ear is 20 – 20000 Hz. The average person with normal hearing can detect a sound of pressure amplitude as small as 0.0002 μbar and in the higher range without causing pain up to 1000 μbar. The range of pressure covers a very wide spectrum (from 0.0002 – 1000 μbar). Thus measurement of sound levels with pressure units that can vary over a wide range of values is impractical and inconvenient. To avoid this, a logarithmic

scale called decibel scale is used for the noise measurement. The measurement unit is called decibels (dB).

The magnitude of volume or a sound expressed in decibels is called a sound pressure level (SPL). An SPL is defined mathematically as

$$SPL = 20 * \log_{10}\left(\frac{P}{P_0}\right)$$

Where, SPL = sound pressure level, dB

P = r.m.s sound pressure, Pa

P_0 = reference pressure, Pa ($=2*10^{-5}$ Pa)

Similarly, Sound power level can be represented by

$$L_w = 10 * \log_{10}\left(\frac{W}{W_{ref}}\right) dB$$

Where, W = acoustic power of interest (Watt)

W_{ref} = internationally accepted reference = 10^{-12} watt

Sound Intensity Level : The acoustical power passing through a unit area is defined as the sound intensity. Symbolically it can be represented as

$$I = W/A \ [W/m^2]$$

In logarithmic scale, it can be defined as

$$L_I = 10 * \log_{10}\left(\frac{I}{I_{ref}}\right) dB$$

Where, I = sound intensity of interest (W/m^2)

I_{ref} = internationally reference for sound intensity = 10^{-12} W/m^2

4.2.3 Equivalent Noise Level

The constant sound pressure level which would have produced the same total energy as the actual level over the given time is defined as equivalent noise level. To measure an average sound pressure level the meter averaging time is extended to equal the period of interest, T, which may be an interval in seconds, minutes, or hours. It can be expressed as

$$L_{Aeq,T} = 10 * \log_{10}\left[\sum_{i=1}^{n} t_i * 10^{L_i/10}\right]$$

Where, $L_{aeq,T}$ = equivalent sound pressure level

L_i = sound pressure level for the time duration of t_i

t_i = fraction of time for the sound pressure level L_i

Day-Night Equivalent Noise Levels (Ldn): The day night equivalent noise levels of a community can be expressed as–

$$L_{dn} = 10 \times \log 10[16/24\ (10^{Ld/10}) + 8/24\ (10^{(Ln+10)/10})]\ dB(A)$$

where, L_d = day-equivalent noise levels (from 6AM – 10PM), dB (A)

L_n = night equialent noise levels (from 10 PM – 6AM), dB (A)

The day hours in respect to assessment of noise levels, is fixed from 6 AM – 10 PM (i.e., 16 hrs) and night hours from 10 PM – 6AM (i.e., 8 hrs.). A sound level of 10 dB is added to Ln due to the low ambient sound levels during night for assesing the Ldn values.

4.2.4 Frequency Spectrum

The second parameter, in addition to sound pressure level, required to characterize the noise is frequency. The noise level for the particular source varies over the frequency range. The basic concept here lies in the definition of bandwidth (BW) :

$f_{n+1} - f_n$ = bandwidth

where, f_{n+1} & f_n are successive frequency band limits i.e. lower and upper bands respectively.

Now these bands are geometrically related by the following recursive relationship :

$$\frac{f_{n+1}}{f_n} = 2^k$$

The index K is a positive integer or fraction depends on the octave band. For 1/1 octave band the value of K is 1 and for 1/3 octave band K is 1/3 and so on.

Finally, for all types of octave band analysis, there is a central frequency (f_c) and it is given by

Finally, for all types of octave band analysis, there is a central frequency (f_c) and it is given by

$$f_c = \sqrt{f_n * f_{n+1}}$$

4.2.5 Frequency Weighting

A weighted frequency scale is simply a value of corrections which is applied to sound pressure levels on a power basis as a function of frequency. The frequency weighting network alters the frequency response of the sound level meter. These frequency corrections are designed to match the frequency weighting scales. Therefore, when selected, frequency band level corrections need not be applied when measuring A- and C- weighted values.

Statistically, the A weighting scale follows closely the frequency response of the human ear to sound, that is the ear is less sensitive to frequencies below 1000 Hz. The frequency response characteristics for the various weighting circuits are given below in Figure 4.7.

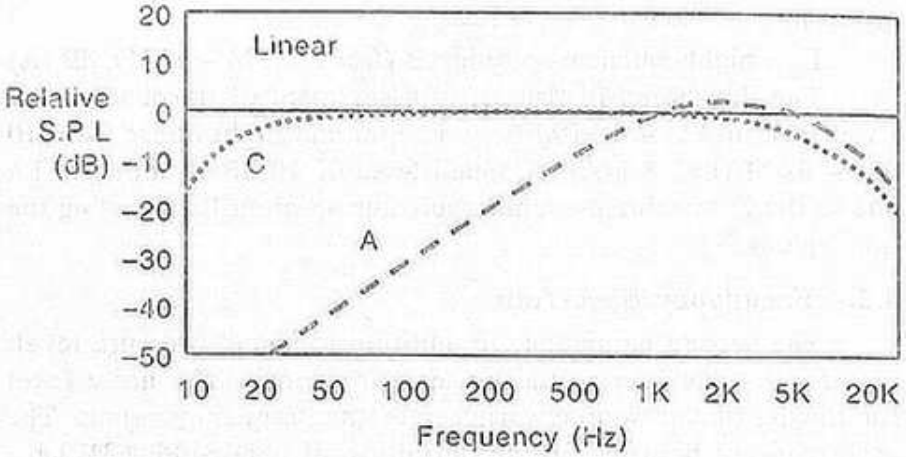


Figure 4.7 : Frequency response characteristics for weigting networks of sound level meters

4.2.6 Sound Wave Propagation

4.2.6.1 Point Source : The relationship between the sound pressure level (Lp) and sound power level (Lw) is given by

$$L_p = L_w - 20^* \log_{10}(r) - 11 \text{ [dB]}$$

Where, L_w = sound power level of the point source

L_p = sound pressure level of the point source at a radial distance of r from the noise source

r = radial distance from source in m

Thus, the relationship between the two sound pressure levels at a radial distance of r_1 and r_2 can be obtained from the above equation is

$$L_{p1} - L_{p2} = [L_w - 20^*\log_{10}(r_1) - 11] - [L_w - 20^*\log10(r_2) - 11]$$

$$\Rightarrow L_{p1} - L_{p2} = 10^*[\log_{10}(r_2) - \log_{10}(r_1)]$$

where, L_{p1} & L_{p2} = sound pressure level at a radial distance of r_1 & r_2 respectively

4.2.6.2 Line Source : Similarly, the relationship between the two sound pressure levels at a radial distance of r_1 and r_2 can be obtained from the above equation is

$$L_{p1} - L_{p2} = 10^* [\log_{10}(r_2) - \log10(r_1)]$$

[Note : The variation is due to the sound propagation characteristics of the point and line source. In case of point source

$L_p \propto 1/r^2$ whereas for line source $L_p \propto \frac{1}{r}$]

4.2.7 Noise Pollution Standards

4.2.7.1 Worksite Standards

Based on the study conducted in the different parts of the world with respect to harmful consequence of noise exposure, it was realized by the Directorate General of Mines Safety (DGMS) to define noise limits for the mining areas in order to achieve the following objectives :

- To prevent a risk of hearing impairment
- To prevent interference of communication essential for safety purposes, and
- To eliminate nervous fatigue with due consideration to the work to be done.

Accordingly following standards and guidelines, based on ILO Code of Practice have been recommended [DGMS Circular No.18 (Tech), 1975]

- A warning limit of 85-dB (A) may be set as the level below which very little risk to an unprotected ear of hearing impairment exists for an eight-hour exposure.
- The danger limit value shall be 90-dB (A) above that the danger of hearing impairment and deafness may result from an unprotected ear.
- A worker should not be allowed to enter, without appropriate ear protection, an area in which the noise level is 115-dB (A) or more.
- Personal protective equipment shall be worn, if there are single isolated outbursts of noise, which can go above 130-dB (A) 'Impulse', or 120-dB (A) 'Fast'.
- No worker shall be allowed to enter an area where noise level exceeds 140-dB (A).

4.2.7.2 Ambient Standards : The ambient noise standards as per CPCB is given in Table 4.2

Table 4.2 : Ambient Air Quality Standards in respect of Noise

Area Code	Category of Area/Zone Day Time	Leq, Limits in dB(A)	
		Night	Time
(A)	Industrial area	75	70
(B)	Commercial area	65	55
(C)	Residential area	55	45
(D)	Silence Zone	50	40

Note :

1. Day time shall mean from 6.00 a.m. to 10.00 p.m.
2. Night time shall mean from 10.00 p.m. to 6.00 a.m.
3. Silence zone is defined as an area comprising not less than 100 m around hospitals, educational institutions, courts, religious places or any other area which is declared as such by the competent authority.
4. Mixed categories of areas may be declared as one of the four above-mentioned categories by the competent authority.

4.3 Water Pollution

4.3.1 Water Pollution Parameters

4.3.1.1 pH : The pH is a dimensionless number that indicates the strength of an acidic or a basic solution. It can be defined as the negative logarithm of the hydrogen ion concentration. The pH scale ranges from 0 – 14. Pure water is neutral (pH = 7) because it contains the same number of hydrogen ions (H^+ ions) as hydroxyl ions (OH^- ions).

4.3.1.2 Total Suspended Solids : Solids occur in water either in solution or in suspension. These two types of solids are distinguished by passing the water sample through a glass fiber filter. By definition, the suspended solids are retained on top of the filter and the dissolved solids pass through the filter with the water.

Total suspended solids (TSS) can be calculated by the following equation

$$TSS = (A - B) / C \ \mu g/m^3$$

Where, A = weight of the filter plus retained solids, μg
B = weight of the clean filter, μg
C = volume of sample filtered, m^3

4.3.1.3 Bio-chemical Oxygen Demand (BOD) : The amount of oxygen required to decompose the biodegradable organics present in the waste water is called the bio-chemical oxygen demand. BOD is used as an indirect measure of the total amount of biodegradable organics in the water. The amount of oxygen required to decompose all the biodegradable organics in a given volume of water is called the ultimate BOD or BOD_L.

BOD at any time of a given solution can be determined by the following equation

$$BOD_t = BOD_L * (1 - 10^{-kt})$$

Where, BOD_t = BOD at any time t, mg/L
BOD_L = ultimate BOD, mg/L
k = a constant representing the rate of the BOD reaction
t = time, day

4.3.1.4 Chemical Oxygen Demand (COD) : There also might be non-biodegradable substances present in the water which cannot be decomposed by the microbes to use as food. In COD test a strong oxidizing agent ($K_2Cr_2O_7$) is used to oxidize the organics rather that relying on microorganisms to do the job. COD test indicates the oxygen required decomposing the organics (biodegradable + non-biodegradable) present in the waste water and thus it is always greater than the BOD. The result of the COD test can be obtained in just 2 hours whereas for BOD test standard time required is 5 days.

4.3.2 Standards of Effluents Discharge from Coal Mining

The standard of effluents discharge from coal mining is given in Table 4.3.

Table 4.3 :
Standards of Effluents Discharge from Coal Mining

Parameters	Maximum permissible value
pH	5.5 - 9.0
Total Suspended Solids	100 mg/L (200 mg/L in case of discharge into Land for Irrigation)
Bio-chemical Oxygen Demand (BOD)	30 mg/L
Chemical Oxygen Demand (COD)	250 mg/L
Oil & Grease (O & G)	10 mg/L

4.4 Illumination

4.4.1 Definitions of Important terms of Photometry

4.4.1.1 Intensity : Intensity of light is the relative amount of luminous energy given by any source and is measured in candles or candle power or in candelas.

4.4.1.2 Horizontal Illuminance : The measure of brightness from a light source, usually measured in foot candles or lumens, which is taken through a light meter's sensor at a horizontal position on a horizontal surface.

4.4.1.3 Vertical Illuminance : The measure of brightness from a light source, usually measured in foot candles or lumens, which is taken through a light meter's sensor at a vertical position on a vertical surface.

4.4.1.4 Mean Spherical Candle Power (m. s. c. p) : It is the average candle power of a lamp in all directions

power of a uniform source giving the same total flux of light. It is directly proportional to the total light given by the lamp and is measured by taking intensity readings in all directions.

4.4.1.5 **Illumination** : It is the total amount of visible light illuminating (incident upon) a point on a surface from all directions above the surface as shown in Figure 4.8. This can be measured in foot candles or in metre candle. One meter candle is the intensity of illumination on a surface of 1 m distant from a source of one candela. Illumination at a surface is inversely proportional to the square of the distance of the surface from the source of light, and directly proportional to Cos(angle between the normal to the surface and the direction of light rays).

Illumination of a surface (meter candle) = (Candela of source)* Cos θ / (distance in m) ^{2}

Where, θ is the angle between the normal to the surface and the direction of light rays

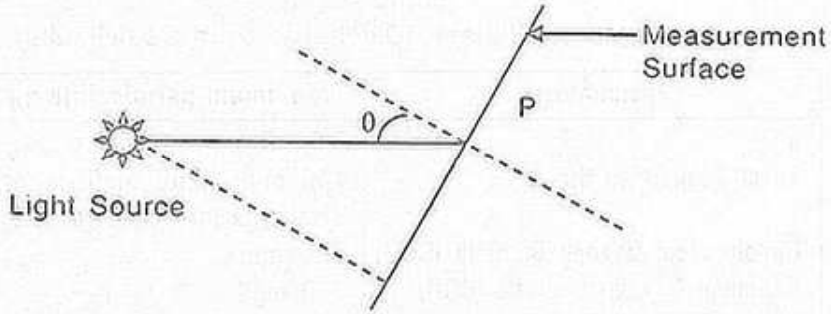


Figure 4.8 : Illumination measurement at a plane

4.4.1.6 **Lumen (lm)** : This is the unit of light (luminous flux) emitted by a light source.

Lumen emitted by a lamp = mean spherical candle power *4π

4.4.1.7 **Lux** : It is the unit of illumination in S.I. units. A lux is an illumination of 1 lumen/ m^2.

4.4.1.8 **Footcandle (fc)** : A unit of measure of illumination. A unit of illumination on a surface that is one foot from a uniform point source of light of one candle and equal to one lumen per square foot. Foot candle values can be measured directly with handheld incident light meters. One foot candle is equal to 1 lumen per sq. ft. of surface.

1 fc = 10.764 lx

4.4.2 Illumination Standards in Mines

4.4.2.1 **Opencast Mining Areas** : The illumination standards

Table 4.4 : Illumination Standards in Opencast Mines

Sl. No.	Location	Minimum Illumination in Lux	Level in which illumination is to be provided
1.	Operational area of draglines and shovels	5	Horizontal
2.	Operational area of drills	10	Vertical
3.	Operator's cabin of Shovel, Dragline, Drill etc.	10	Vertical
4.	Dumper haul road	30	Horizontal
5.	Overburden dumps and coal dumps	0.5 3.0	Horizontal
6.	Roadways and footpaths from bench to bench	3	Horizontal
7.	Coal handling plant, workshop and service buildings	3	As per the BIS specification

4.4.2.2 **Underground Working Areas** : The illumination standards in underground mines are given in Table 4.5.

Table 4.5 : Illumination Standards in Underground Mines

Sl. No.	Place	Recommended minimum average illumination level (in lumens per sq. ft.) for satisfactory light condition
1.	Pit bottom	1.5 3.0
2.	Main junctions	1.25
3.	Roadways	0.4
4.	Haulage engines, control gear and haulage drum	1.5

EXERCISE

1 In ambient air of industrial area, the permissible concentration of suspended particulate matter (PM_{10}) measured on 24 hours average in $\mu g/m^3$ is

a. 200 b. 300
c. 100 d. 400

[GATE, 2005]

2 In ambient air of industrial area, the permissible concentration

of particulate matter (PM_{10}) measured on annual average in $\mu g/m^3$ is

a. 200 b. 60
c. 120 d. 500

3. In ambient air of industrial area, the permissible concentration of PM 2.5 measured on 24 hours average in $\mu g/m^3$ is

a. 60 b. 200
c. 300 d. 350

4. The size of the respirable particulate matter (RPM) considered in the range of

a. < 5 μm b. > 5 μm
c. < 10 μm d. < 1 μm

5. At any particular place the relation between the suspended particulate matter (SPM) and respirable particulate matter (RPM) is

a. SPM > RPM b. SPM < RPM
c. SPM = RPM d. There is no relation

6. In ambient air of an ecologically sensitive, the permissible concentration of respirable particulate matter (PM_{10}) measured on 24 hours average in $\mu g/m^3$ is

a. 250 b. 500
c. 75 d. 100

7. In ambient air of ecologically sensitive, the permissible concentration of respirable particulate matter (PM_{10}) measured on annual average in $\mu g/m^3$ is

a. 60 b. 150
c. 300 d. 500

8. During 24 hours sampling of mass concentration of suspended particulate matter (SPM) by the High Volume Sampler, the following data are obtained :
Air flowing through the filter : 1.7 m^3/ min
Air flowing through filter at the end of the test : 1.5 m^3/ min
Mass of the clean filter : 5.0 g
Mass of the filter after 24 hours of exposure : 5.4 g
Assuming the flow rate through the sampler decreases linearly, the average concentration of suspended particulates in $\mu g/m^3$ is

a. 163.40 b. 173.61

9. To minimize the air pollution problem, residential buildings in the mining areas should be located in

a. Downwind direction of predominant wind direction
b. Upward direction of predominant wind direction
c. Perpendicular to predominant wind direction
d. At any place away from the mines

10. The most wholesome ventilation of buildings is provided by

a. Natural ventilation
b. Plenum system of ventilation
c. A.C. system of ventilation
d. Vacuum system of ventilation

11. The important gaseous pollutants, contributing to acid rains, are

a. SO_2 and NOx b. CO_2 and H_2S
c. NOx and O_3 d. None of these

12. The device, which can be used to control gaseous as well as particulate pollutants, in the industrial emissions, is known as

a. Cyclone separator b. Spray tower
c. Dynamic precipitator d. Fabric filter

13. Electrostatic precipitators remove

a. SO_2 b. NOx
c. Particulate matter d. None of the above

14. Which one of the followings is secondary pollutant

a. SO_2 b. NOx
c. O_3 d. Hydrocarbons

15. Hairs of our nose can remove all particles of size, greater than

a. 1 mm b. 1 micron
c. 10 micron d. 100 micron

16. The sound pressure level is measured as 5×10^{-4} N/m^2. What is the noise level in dB?

a. 27.9 b. 30.7
c. 28.7 d. 25.0

17. The noise levels measured during the operation of three machines are 80 dBA, 75 dBA and 85 dBA respectively. Determine the equivalent noise level for the above three machines.

a. 86.5 b. 87.73
c. 80.00 d. 240.0

18. It is required to find out the day-night equivalent noise levels at a location. The three-hourly day average values in dB are 48, 54, 56, 52, 61 and three-hourly night average values in dB are 36, 42, and 48. What is value of Ldn?

a. 43.9 b. 56.9
c. 55.7 d. 45.8

19. The noise level measured during the operation of a jackhammer are given below :

Noise Level	Duration
70 dBA	30 min
80 dBA	15 min
85 dBA	10 min

The equivalent noise level (Leq) in dBA calculated from these data is

a. 88.50 b. 79.55
c. 71.45 d. 69.55

[GATE, 2005]

20. If the measured sound pressure level at a distance of 10 m from a source is 120 dB, the sound pressure level in dB at a distance of 60 m from the source is

a. 104.44 b. 121.55
c. 127.78 d. 135.56

[GATE, 2006]

21. On a flat terrain noise received from a source 100 m away is 82 dBA. If the receiver is 160 m away from source, what is the noise level in dBA?

a. 72 b. 65
c. 81 d. 78

[GATE, 2001]

22. What is the center frequency of the band number whose lower and upper bands are 1414 and 2828 Hz respectively?

a. 3000 Hz b. 1814 Hz
c. 2000 Hz d. 1500 Hz

23. What is the center frequency for the one-third (1/3) octave band whose lower and upper frequency limits are 7079 and 8913 Hz, respectively?

a. 7000 Hz b. 7928 Hz
c. 8000 Hz d. 7852 Hz

24. What is the sound power level of a sound source radiating energy at a rate of 0.2 W?

a. 110 dB b. 112 dB
c. 113 dB d. 120 dB

25. What is the acoustical power of a 100 hp air compressor whose rated sound power level is 130 dB

a. 13 W b. 9 W
c. 10 W d. 12 W

26. The sound intensity level of a particular mine machine at a radial distance of 10 m is $2.4 * 10^{-7}$ W/ m^2. Determine the sound power level of the noise source in dB. Assume that the sound waves are radiating uniformly in all the directions.

a. 84.4 dB b. 95.4 dB
c. 80.0 dB d. 75.4 dB

27. A small source whose sound power level Lw is 110 dB is hanging freely outdoors. What is the sound pressure level at 20 m from the source?

a. 73 dB b. 75 dB
c. 77 dB d. 76 dB

28. The noise level at 10 m from a long conveyor belt is 95 dBA. What is the noise level at 100 m?

a. 85 dBA b. 90 dBA
c. 95 dBA d. 80 dBA

29. As per the DGMS norm warning level of noise for the worksite area is

a. 85 dBA b. 90 dBA
c. 95 dBA d. 100 dBA

30. A jackhammer operates at a corner of a square field of side 50 m. At the diagonally opposite corner, the SPL sensed is 82.3 dB. The SPL at any of the other two corners of the field in dB is

a. 86.3 b. 85.3
c. 83.6 d. 81.2

[GATE, 2007]

31. A sample of sewage from a coal mines is found to have a BOD after 5 d (BOD_5) of 180 mg/L. Estimate the ultimate BOD (BOD_L) of the sewage. Assume that k = 0.1 /d for this waste water.

a. 240 mg/ L b. 350 mg/ L
c. 400 mg/ L d. 450 mg/ L

32. A 6 ml sample of mine water is diluted to 300 mL with distilled water in a standard BOD bottle. The initial DO in

the bottle is determined to be 8.5 mg/ L, and the dissolved after 5 d at 20 °C is found to be 0.5 mg/ L. Determine the BOD_5 of the mine water.

a. 240 mg/ L
b. 350 mg/ L
c. 400 mg/ L
d. 450 mg/ L

33. The BOD_5 of an effluent from a coke oven plant is 25 mg/ L, and the effluent discharge is 4 ML/d. The receiving stream has a BOD_5 of 2 mg/L and the stream flow is 40 ML/d. What will be the BOD_5 in the stream just below the mixing zone?

a. 4.5
b. 5.2
c. 4.1
d. 4.8

34. An effluent sample is diluted with fresh water to make up a solution of 300 ml. The DO of the solution initially is 8 mg/l and the value falls to 3 mg/l after 5 days. If the 5 day BOD of the original effluent is known to be 50 mg/l, the amount of fresh water added in ml to the solution is

a. 270
b. 160
c. 54
d. 30

[GATE, 2007]

35. The value of COD is related to BOD as

a. COD > BOD
b. COD = BOD
c. COD < BOD
d. There is no relation

36. The permissible value of oil and grease discharge from the mine workshop is

a. 10 mg/ L
b. 20 mg/L
c. 25 mg/L
d. 15 mg/L

37. The permissible value of BOD in the effluent discharge from a coal mines is

a. 120 mg/L
b. 240 mg/L
c. 150 mg/L
d. 250 mg/L

38. The ultimate BOD value of a waste

a. Increases with temperature
b. Decreases with temperature
c. Remains the same at all temperatures
d. Doubles with every 10^0C rise in temperature

39. When a mine water is disposed off in a river, the rate of depletion of dissolved oxygen of the river mainly depends on

a. BOD of the mine water
b. COD of the mine water
c. Total organic carbon present in the mine water

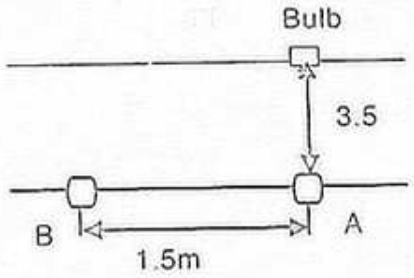
40. Which of the following pairs is correctly matched?

a. $\frac{BOD}{COD} = 0$: Mine water is toxic
b. $\frac{BOD}{COD} \leq 0.2$: Acclimatization of seed is necessary
c. $\frac{BOD}{COD} \geq 0.6$: Mine water is non–biodegradable
d. BOD = COD = 0 : Mine water is devoid of organic matter

41. The correct statement of comparison of ultimate BOD, COD, Theoritical Oxygen Demand and 5-day BOD (BOD_5) is

a. $BOD_u > COD > ThOD > BOD_5$
b. $COD > ThOD > BOD_u > BOD_5$
c. $ThOD > COD > BOD_u > BOD_5$
d. $COD > BOD_u > BOD_5 > ThOD$

42. A filament bulb produces 30 lux illumination at point A as shown in the figure. Assuming that no other source of lighting present in the roadway, the illumination level in lux at point B will be



a. 4.88
b. 1.63
c. 0.35
d. 7.00

[GATE, 2002]

43. Floor illumination at a point directly below a light source in an underground garage of height 4 m is 40 lux. What is the floor illumination in lux at a point 8 m away from the light source?

a. 2
b. 5
c. 10
d. 20

44. The minimum illumination standard at the coal mine pit bottom is

a. 1.5 lux
b. 1.5 lm/ ft^2
c. 0.4 lm/ ft^2
d. 2.4 lux

[GATE, 1997]

45. A single lamp placed centrally at the roof provides 40 lux illumination vertically below, at the floor of an underground workshop. The workshop of dimensions 20 m X 20 m with height 4 m. Assuming uniform spherical dispersion of luminous intensity, the floor level illumination in lux at any corner of the workshop is

a. 23.2 b. 10.9
c. 3.0 d. 0.8

[GATE, 2007]

Answer

1. c	2. b	3. a	4. c	5. a	6. d
7. a	8. b	9. b	10. c	11. a	12. b
13. c	14. c	15. c	16. a	17. a	18. c
19. b	20. a	21. d	22. c	23. b	24. c
25. c	26. a	27. a	28. a	29. a	30. b
31. c	32. d	33. c	34. d	35. a	36. a
37. d	38. c	39. a	40 d	41. c	42. d
43. b	44. a	45 d			

□

5.0 Mine Surveying

5.1 Introduction

Mine surveying is the way of determining the relative positions of points on the surface of the earth and underground.

Surveying primarily classified into two categories :

- **Geodetic Surveying/ Trigonometrical Surveying :** In this type of surveying, it is necessary to take into consideration of curvature of the earth. This type of surveys generally covers large areas.
- **Plane Surveying :** In plane surveying the curvature of the earth is not taken into consideration as the surveys extend covers small areas.

5.2 Representative Factor

It is the ratio of map distance to the corresponding ground distance. That is, if the distance between two points in the ground is 100 m and in the drawing the distance between the points representing the same is 10 cm then the representative factor (R.F.) is given by

$$R.F. = \frac{10\ cm}{10000\ cm} = \frac{1}{1000} = 1:1000$$

5.3 Distance Measuring Instruments

The following instruments can be used for distance measurements.

- Pacing
- Passometer
- Pedometer
- Odometer
- Speedometer
- Perambulator
- Chain

5.4 Correction Factors in Line Measurement

5.4.1 Correction Factor Due to Incorrect Chain

It is often observed that the length of the measuring tape or chain is incorrect due to expansion or contraction and thus the measured length with this incorrect chain should be corrected before plotting

Correction factor due to incorrect chain (C.F.) = $\frac{L \pm \Delta L}{L}$

Where, L = true length of the chain or tape

(L ± ΔL) = incorrect length of the chain or tape

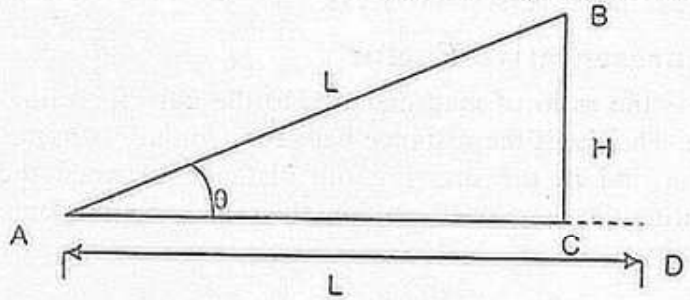
[Note : + sign, if the chain/ tape is shorter than the indication i.e. the chain is too long

– sign, if the chain/ tape is longer than the indication i.e. the chain is too short]

⇒ True Length of Line = $\frac{L \pm \Delta L}{L}$ *Measured Length of Line

5.4.2 Slope Correction Factor

Since the map distance always indicates the horizontal distance between any two points. Thus the measurements along the plane which is not horizontal should be corrected to plot the exact horizontal distance. Angle can be measured by the Clinometers.



Slope Correction Factor (S.C.F) = CD = AB – AC

$$= L - \sqrt{L^2 - H^2}$$

$= L - L\,[1 - H^2/2L^2 - H^4/4L^4 - \ldots\ldots\ldots\ldots]$

$= H^2/2L$ [Neglecting the higher powers as H/L is very small]

If 'θ' is known then the correction factor is given by

S.C.F. = CD = L – L * Cos θ = L (1 – Cos θ)

5.4.3 Correction for Abso.lute Length

The measured length of any line can be corrected by the factor given by

$$Ca = \frac{L * c}{l}$$

Where, Ca = correction for absolute length

L = measured length of a line

c = correction to a tape

l = nominal length of a tape

⇒ Corrected length of measured distance D can be give by

Actual length = D ± Ca

[The sign of the correction will be the same as Cal

5.4.4 Correction for Temperature

Since the length of the tape is increased as the temperature is raised, and consequently, the measured distance is too small. It is given by

$$C_t = \alpha\,(T_m - T_0)\,L$$

Where, C_t = the correction for temperature, in m

α = the coefficient of thermal expansion

Tm = the mean temperature during measurement

To = the temperature at which the tape is standardised

L = the measured length in m.

5.4.5 Correction for Pull

When the pull used during measurement is different from that at which the tape is standardised, this correction should be apply to obtain the actual distance. It is given by

$$C_p = \frac{(P - P_0) * L}{A * E}$$

Where, Cp = the correction for pull, in m

P = the pull applied during measurement, in N

Po = the pull under which the tape is standardised, in N

L = the measured length in m

A = the cross-sectional area of the tape, in sq. cm

E = the modulus of elasticity of steel, N/cm^2

5.4.6 Correction for Sag

The correction for sag is the difference in length between the arc length and the subtending chord. It requires only when the tape is suspended during measurement and the correction is always negative. It can be given by

$$C_s = \frac{L * (M * g)^2}{24 * P^2}$$

Where, Cs = the sag correction for a single span, in m

L = the distance between supports, in m

M = the mass of the tape between the two consecutive span, in kg

P = the applied pull, in N

g = acceleration due to gravity, m/s^2

5.5 Traversing

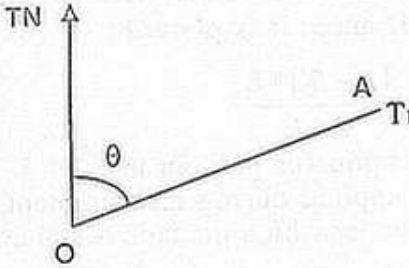
In traverse surveying the directions of the survey lines are set by angular measurements. There are two types of

5.5.1 Bearing of Lines

It is the horizontal angle which the line makes with some reference directions or meridians. There are three types of meridian generally used for surveying like true meridian or geographical meridian, magnetic meridian and assumed meridian.

5.5.2 True Meridians or Geographical Meridians

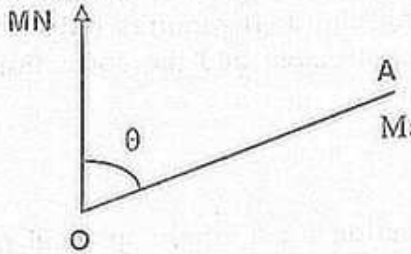
The true or geographical meridian passing through a point on the earth's surface is the line in which the plane passing through the given point and the North Pole and South Pole intersects the surface of the earth. The point of intersections of the earth's axis and the surface of the earth are known as the north geographical pole and south geographical pole. The angle which a line makes with the true north is called geographical bearing/ true bearing.



True Bearing of Line OA = θ

5.5.3 Magnetic Meridian

The direction indicated by a freely suspended magnetic needle at a place free from local attraction is called the magnetic meridian. The angle which a line makes with the magnetic north is called the magnetic bearing.



Magnetic Bearing of Line OA = θ

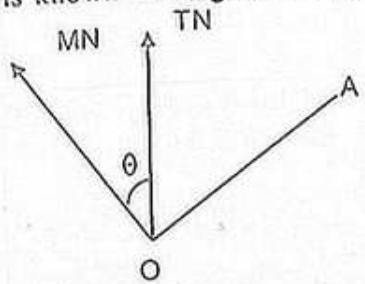
5.5.4 Arbitrary Meridian

The angle between any assumed meridian and a line is known as assumed bearing of the line.

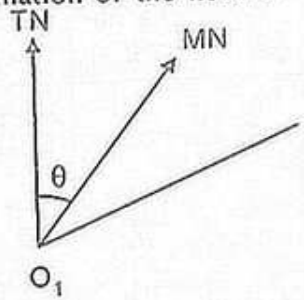
Assumed Bearing of Line OA = θ

5.5.5 Magnetic Declination

Since the magnetic north and true north at a place does not coincide except in few places, the horizontal angle between the two is known as magnetic declination or the declination of the needle.



CASE I



CASE II

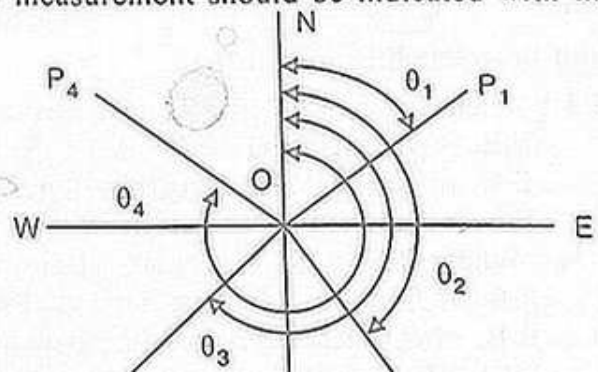
CASE I : Magnetic north is situated in θ^0 W of truth north.
True Bearing of line OA = Magnetic Bearing of line OA - θ°
CASE II : Magnetic north is situated in θ° E of true north
True Bearing of line OA = Magnetic Bearing of line OA + θ°
['θ' is called the magnetic declination]

5.5.6 Designation of Bearing

Bearing of a line can be represented by two types of system.

5.5.6.1 Whole Circle Bearing : In this system the bearing of a line is always measured clockwise from the north direction of the reference meridian. The angle thus measured varying from 0° to 360°.

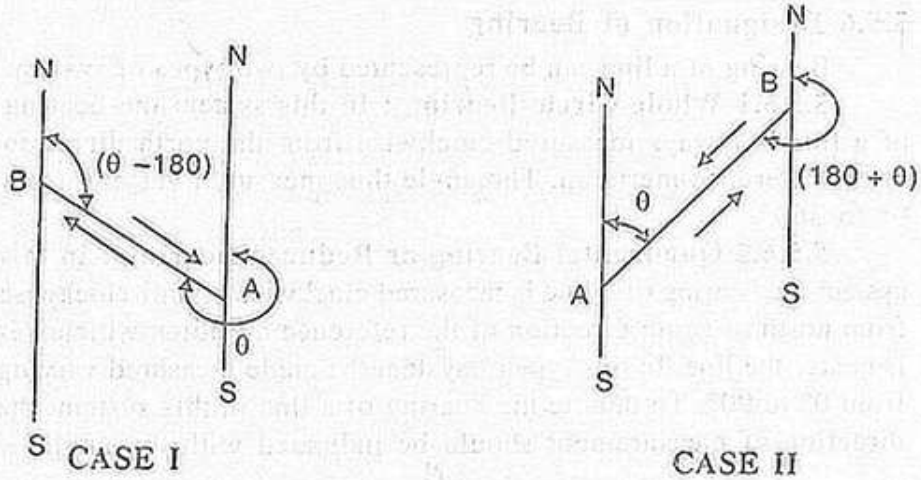
5.5.6.2 Quadrantal Bearing or Reduced Bearing : In this system the bearing of a line is measured clockwise or anti-clockwise from north or south direction of the reference meridian whichever is nearer the line. In this type of system the angle measured varying from 0° to 90°. To denote the bearing of a line in this system, the direction of measurement should be indicated with the angle.



Bearing of lines OP_1, OP_2, OP_3 and OP_4 can be represented by :

Line	Whole Circle Bearing (W.C.B)	Quadrant Bearing or Reduced Bearing (R.B)	Relation between W.C.B and R.B for different cases
OP_1	θ_1	N θ_1 E	R.B = W.C.B [0° < θ < 90°]
OP_2	θ_2	S (180° − θ_2) E	R.B = 180° − W.C.B [90° < θ < 180°]
OP_3	θ_3	S (θ_3 − 180°) W	R.B = W.C.B − 180° [180° < θ < 270°]
OP_4	θ_4	N (360° − θ_4) W	R.B = 360° − W.C.B [270° < θ < 360°]

5.5.6.3 Fore and Back Bearing : The bearing of a line in the direction of the progress of survey is called the fore bearing (F.B) while the bearing of the same line in the reverse direction known as back bearing (B.B). Thus there will be always two bearings of any line on the basis of the direction of the line.



CASE I CASE II

Relation between F.B. and B.B :

CASE I : When F.B > 180°

F.B. of line AB = θ

B.B. of line AB = F.B. of line BA = θ − 180°

Thus, F.B = B.B + 180°

CASE II : When F.B < 180°

F.B. of line AB = θ

B.B. of line AB = F.B. of line BA = θ + 180°

Thus, F.B = B.B − 180°

5.6 Theodolite

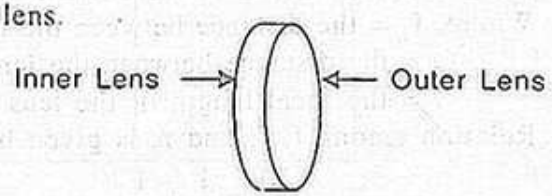
5.6.1 Introduction

Theodolite can be classified in two ways

- **Transit Theodolite :** The telescope of the theodolite can be revolved through a complete revolution about its horizontal axis in a vertical plane.

Parameters of Transit Theodolite :

- **Alidade :** The vernier plate completes with the standards and telescope is called the alidade of the theodolite.
- **Face Left :** When the vertical circle of the instrument is on the left of the observer when taking a reading, the position is called face left and the observation made called face left observation.
- **Face Right :** When the vertical circle of the instrument is on the right of the observer when taking a reading, the position is called face right and the observation made called face right observation.
- **Line of Collimation :** The line of collimation is the imaginary line joining the intersection of the cross-hairs of the diaphragm to the optical center of the object glass and its continuation.
- **Object Glass :** It is compound lens consisting of an outer double convex lens of hard crown glass and an inner concave-convex lens of dense flint glass. This lens is called achromatic lens and the two serious defects namely spherical aberration and chromatic aberration can be eliminated by this type of lens.



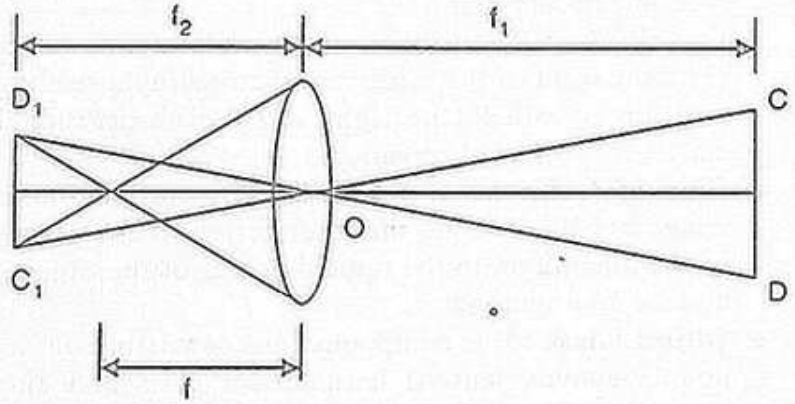
Achromatic Lens

- **Spherical Aberration :** When the rays of light passing through a lens whose surfaces are truly spherical will not converge exactly to one point, the defect is known as spherical aberration.
- **Chromatic Aberration :** The dispersion of white light by a lens leads to number of blurred and colour images instead of single image, the defect is called chromatic aberration.

- **Level Tube or Bubble Tube :** It consists of a sealed glass tube set in a brass tube with Plaster of Paris and filled with alcohol or ether, or a mixture of both.
- **Non-transit theodolite :** When the telescope of the theodolite cannot be revolved through a complete revolution about its horizontal axis in a vertical plane. This is of two types namely Y type and Everest type.

5.6.2 Magnification or Magnifying Power

Magnifying power of a telescope is the ratio of the angle subtended at the eye of the virtual image of the object and the real object. It can also be defined as the ratio of the focal length of the objective to the focal length of the eye piece.



$$\text{Magnifying Power (P)} = \frac{\text{Size of the object}}{\text{Size of the image}} = \frac{CD}{C_1D_1} = \frac{f_1}{f_2}$$

Where, f_1 = the distance between the lens and object (CD)

f_2 = the distance between the lens and image (C_1D_1)

f = the focal length of the lens

Relation among f, f_1 and f_2 is given by

$$\frac{1}{f} = \frac{1}{f_1} + \frac{1}{f_2}$$

5.6.3 Error in Theodolite Traverse

5.6.3.1 Permissible Linear Error : Permissible linear error of closure in theodolite traverse survey can be obtained by

$$Le = \frac{P}{1000} * \sqrt{1 + \frac{e^2 N}{12}}$$

Where,

P = the perimeter of a traverse in m

N = the number of sides of a traverse

e = permissible error per angle depends on the precision of the theodolite (commonly 1')

5.6.3.2 Permissible Angular Error : Permissible angular error of closure in theodolite traverse survey can be obtained by

$$Ae = L * \sqrt{N}$$

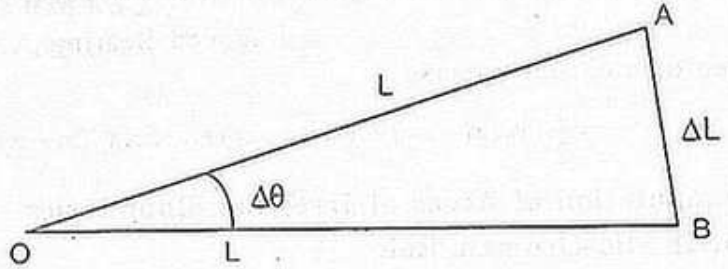
where,

Ae = the permissible angular error of closure

L = least count of the theodolite (generally 20'' or 1')

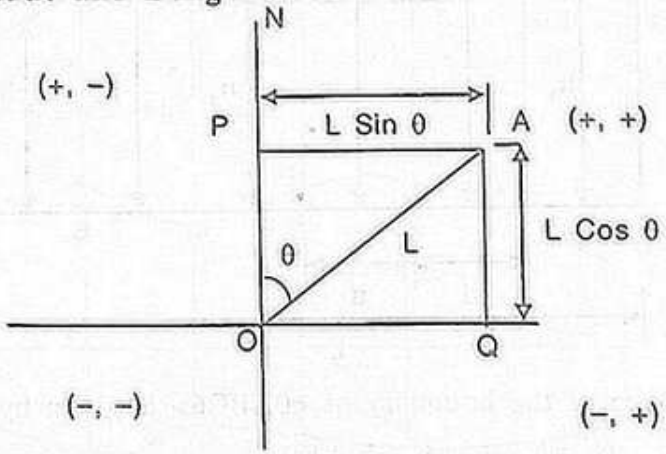
N = number of sides

5.6.4 Relation between Angular and Linear Measurement



$$\Delta L = L * \tan \Delta \theta$$

5.6.5 Latitude and Longitude of a line



The projection of any line in north-south line defined as latitude of the line. The sign is positive, if the direction is in the north side and negative for south side.

Similarly, the projection of any line in east-west line is defined as longitude of the line. The sign is positive in eastward direction and negative for westward direction.

Latitude of OA = L Cos θ
Departure of OA = L Sin θ
Co-ordinate of A is (L Cos θ, L Sin θ)

5.6.6 Close Traverse Survey

Sum of the latitudesofthe lines = $\sum LCos\ \theta = 0$

Sum of the departures of the lines = $\sum LSin\ \theta = 0$

If the above two equation not holds, there is some error, called closing error and this is given by

Closing Error (E) = $\sqrt{(\sum L.Cos\theta)^2 + (\sum L.Sin\theta)^2}$

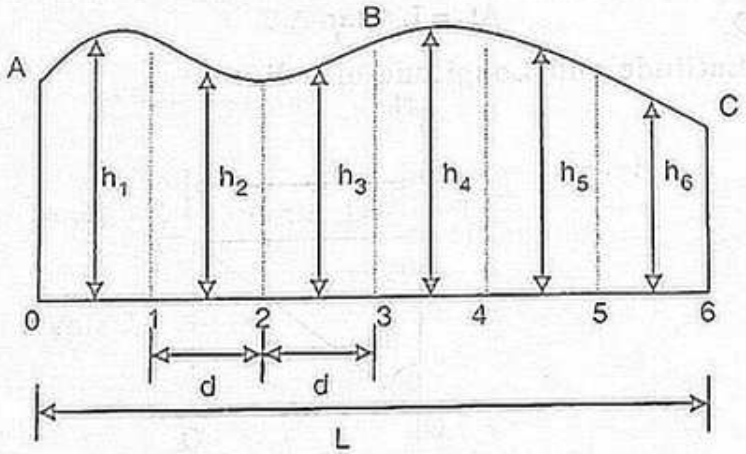
Direction of the closing error = $\beta = \tan^{-1}\left(\frac{\sum L.Sin\theta}{\sum L.Cos\theta}\right)$
= Reduced Bearing

Area of a closed traverse

$$= \frac{1}{2}[y_1(x_1 - x_2) + y_2(x_2 - x_3) + \ldots\ldots y_n(x_1 - x_{n-1})]$$

5.6.7 Computation of Areas of Irregular Boundaries

5.6.7.1 Mid-Ordinate Rule



Area of the boundary of <0ABC6> is given by

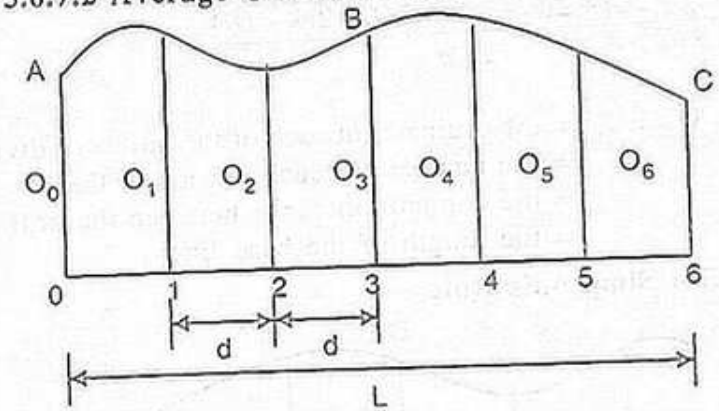
$$\Delta = \frac{(h_1 + h_2 + h_3 + h_4 + h_5 + h_6)}{n} * d * n$$

$$= \frac{(h_1 + h_2 + h_3 + h_4 + h_5 + h_6)}{n} * L$$

where,

d = thecommon distance between the ordinates
L = the length of the base line

5.6.7.2 Average Ordinate Rule



Area of the boundary of <0ABC6> is given by

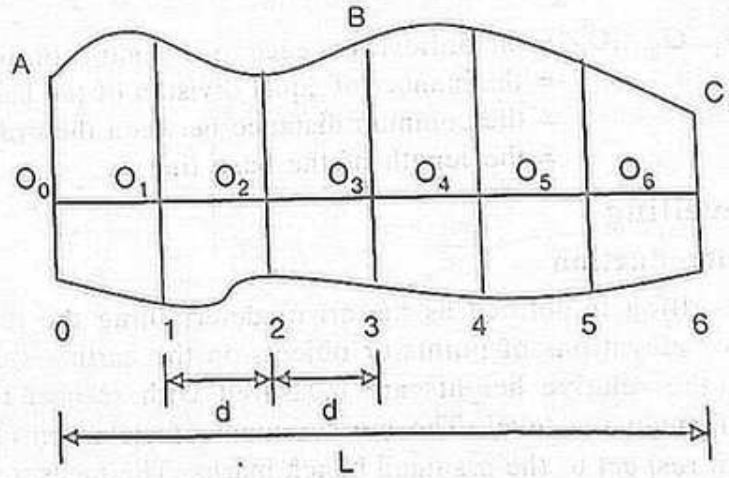
$$\Delta = \frac{(O_0 + O_1 + O_2 + O_3 + O_4 + O_5 + O_6)}{n+1} * d * n$$

$$= \frac{(O_0 + O_1 + O_2 + O_3 + O_4 + O_5 + O_6)}{n+1} * L$$

Where,

$O_1, O_2 \ldots O_6$ = the ordinates at each of the points of division
n = the number of equal division of the base line
d = the common distance between the ordinates
L = the length of the base line

5.6.7.3 Trapezoidal Rule



Area of the boundary of <0ABC6> is given by

Area (Δ) =

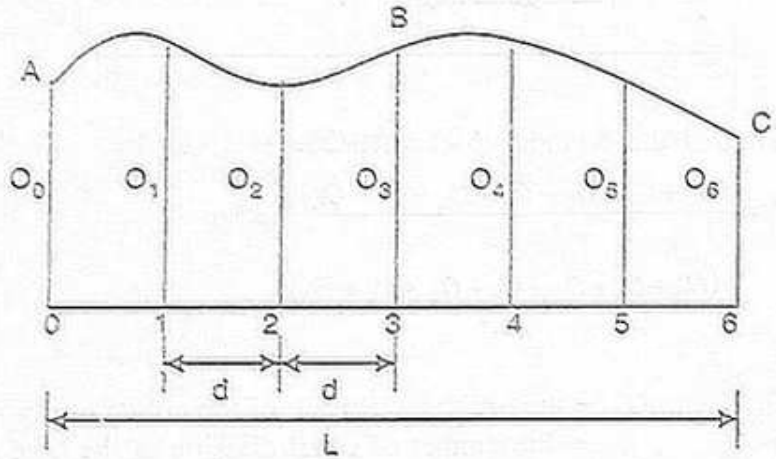
$$\frac{(O_0+2O_1+2O_2+2O_3+2O_4+2O_5+O_6)*d*n}{2*n}$$

$$=\frac{(O_0+2O_1+2O_2+2O_3+2O_4+2O_5+O_6)*L}{2*n}$$

Where,

$O_1, O_2 \ldots O_6$ = the ordinates at each of the points of division
n = the number of equal division of the base line
d = the common distance between the ordinates
L = the length of the base line

5.6.7.4 Simpson's Rule



Area of the boundary of <0ABC6> is given by

$$\text{Area } (\Delta) = \frac{(O_0+4O_1+2O_2+4O_3+2O_4+4O_5+O_6)*d}{3}$$

Where,

$O_1, O_2 \ldots O_6$ = the ordinates at each of the points of division
n = the number of equal division of the base line
d = the common distance between the ordinates
L = the length of the base line

5.7 Levelling

5.1.1 Introduction

Levelling is defined as the art of determining the relative heights or elevations of points or objects on the earth's surface. In India the relative heights are measured with respect to the datum of mean sea level. The measurement of elevation also is done with respect to the assumed bench marks. The measurement

- **Reduced Level (R.L.) :** The elevation of a point is its vertical distance above or below the datum fixed for the reference and it is known as reduced level (R.L.)
- **Back Sight (B.S.) :** It is staff reading on the point of known elevation and also the first reading after the level is set up and levelled.
- **Fore Sight (F.S.) :** It is staff reading on the point whose elevation is to be determined as on a change point.
- **Intermediate Sight (I.S.) :** All the staff readings taken between the back sight and foresight are called intermediate sight.
- **Height of the Instruments (H.I) :** It is the elevation (R.L) of the plane of collimation or line of sight when the instrument is correctly leveled.

5.7.1 Classification of Levelling

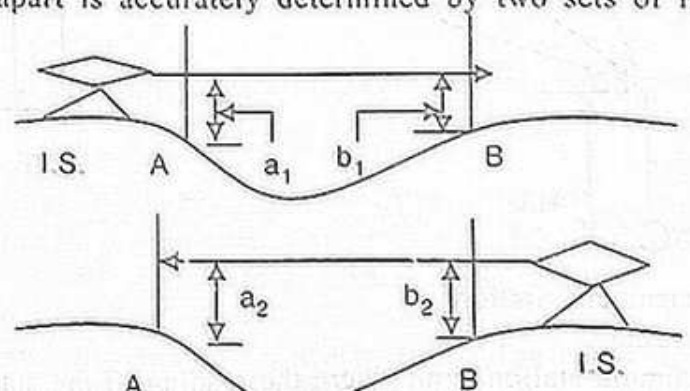
5.7.2.1 Differential Levelling : It is the operation of levelling to determine the elevations of points or to establish bench marks with respect to the known bench marks. It is also known as "taking flying levels"

5.7.2.2 Check Levelling : It is the operation of running levels for the purpose of checking a series of levels, which have been previously fixed.

5.7.2.3 Profile Levelling : It is the operation in which the object is to determine the elevations of points at known distance apart along a given line. It is also called longitudinal levelling or sectioning.

5.7.2.4 Cross Sectioning : It is the operation of levelling to determine the surface undulations to the give line.

5.7.2.5 Reciprocal Levelling : It is the method of levelling in which the difference between the two points situated at a large distance apart is accurately determined by two sets of reading



taking from the two points. The advantage of this method is that it eliminates the errors due to curvature and refraction as well as due to the line of collimation not being exactly parallel to the bubble line.

If d and e are the difference between the levels between two points, these can be obtained as

$$d = \frac{(b_1 - a_1) + (b_2 - a_2)}{2}$$

$$e = \frac{(b_1 - a_1) - (b_2 - a_2)}{2}$$

Where,

a_1 and a_2 are the staff reading at A measuring from two stations respectively

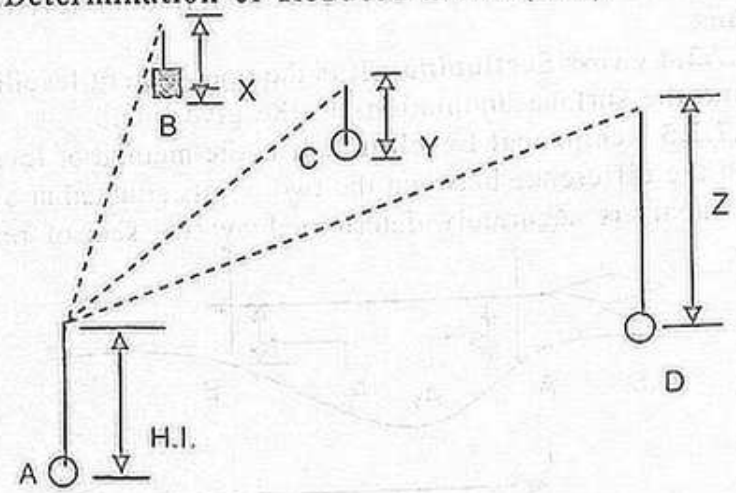
b_1 and b_2 are the staff reading at B measuring from two stations respectively

5.7.2.6 Barometric Leveling : It is the method of leveling in which the altitudes of points are determined by means of a barometric pressure.

5.7.2.7 Hypsometry : It is the method of leveling in which the altitudes of points in the hill are determined by means of observing the temperature.

5.7.2.8 Trigonometric Leveling : It is the process of leveling in which the elevations of points are determined from the vertical angles and horizontal distance measured in the field.

5.7.3 Determination of Reduced Level (R.L)



Instrument Station

Assume :

A : Instrument station from where the reading of the staff height

B : Bench mark station (Reduced level of the B.M. = 100 m)

C : Intermediate station

D : Station at which the R.L has to be determined

Instrument is set at 'A' and the staff reading of the points 'B', 'C' and 'D' has been taken and these are X ,Y and Z respectively.

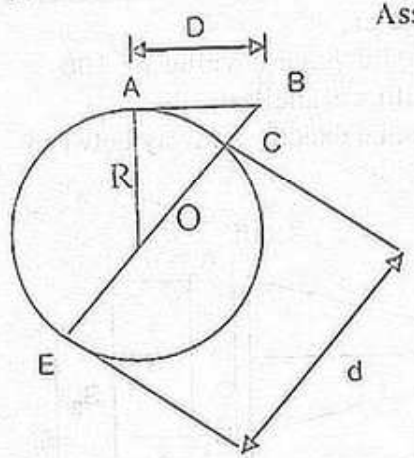
H. I. = B.M. + X

R.L. of C = H.I. – Y = B.M. – X

R.L. of D = R.L. of C + Y – Z = B.M. – X + Y – Z

5.7.4 Correction Factor in Levelling

5.7.4.1 Correction for Curvature : Since the earth surface is spherical in nature, the precise levelling needs some correction factor to obtain the exact reduce level. Since the effect of curvature increases the staff reading, the correction is subtractive. The true staff reading may, therefore, be obtained by subtracting the correction for curvature from the observed staff reading.



Assume :

AC = level line through the instrument axis

O = centre of the earth

R = radius of the earth = OA

d = diameter of the earth = CE

BC = correction for curvature

AB = horizontal line (line of collimation)

D = distance between the inst. station and staff point

From tangent rule, we can get

$AB^2 = BC * BE = BC * (BC + CE) = BC^2 + BC * CE$

$\Rightarrow AB^2 = BC * CE$

[Since BC is very small as compared to diameter (CE) of the earth, the term BC^2 can be neglected]

$\Rightarrow D^2 = BC * d$

$\Rightarrow$ Correction for curvature of the earth = $BC = D^2/d = D^2/2*R$

$\Rightarrow$ Correction for curvature of the earth = $BC = 0.0785\ D^2$

[Taking diameter of the earth = 12742 km]

True Staff reading = Observed staff reading – $0.0785 * D^2$

5.7.4.2 Correction for Refraction : The ray of light from the staff to the instrument is not straight rather it refracted or

refraction is to diminish the staff reading and hence the correction is additive. On an average it may be taken as one seventh of that of curvature and is of opposite effect.

Therefore,

Correction for refraction = BC/ 7 = D^2/ 7*d = D^2/ 14*R

True Staff reading = Observed staff reading + 0 0112 * D^2

Combined Correction = Correction for curvature – Correction for refraction

$$=\left(BC-\frac{BC}{7}\right)=\frac{6}{7}*BC=\frac{6}{7}*\frac{D^2}{d}=\frac{6}{7}*\frac{D^2}{2R}$$

5.8 Tacheometric Survey

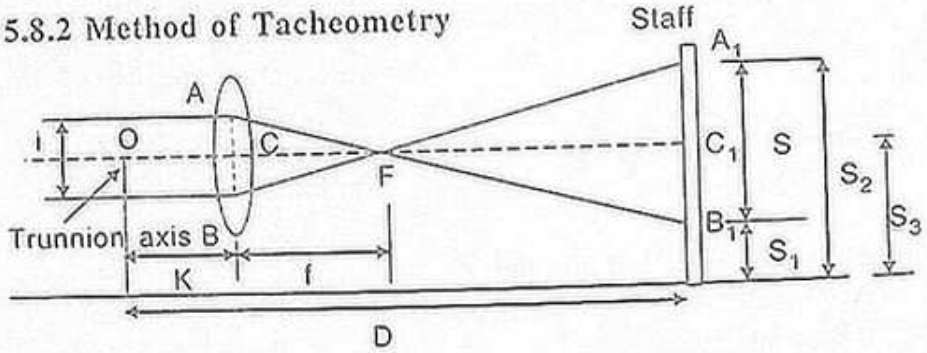
5.8.1 Introduction

This is a branch of angular surveying in which the horizontal and vertical positions of points are determined by instrumental observations without any linear measurements.

Essential features of a tacheometer

- multiplying constant (f/i) should have a value of 100
- telescope should be fitted with an anallatic lens
- the central horizontal wire must be exactly midway between the two lines

5.8.2 Method of Tacheometry



Let,

f = focal length of the object glass

i = distance between the upper and lower stadia hairs

S = $S_2 - S_1$ = staff intercept

D = horizontal distance from the instrument station to staff station

K = distance of trunnion axis from the object glass

S_3 = axial hair reading

⇒ i / S = f / (D – f – K)

⇒ D = f / i * S + (f + K)

f / i = known as multiplying constant = 100

(f + K) = known an additive constant = 0 for anallatic lens

Height of the staff station (H) = elevation of the instrument axis – S_3

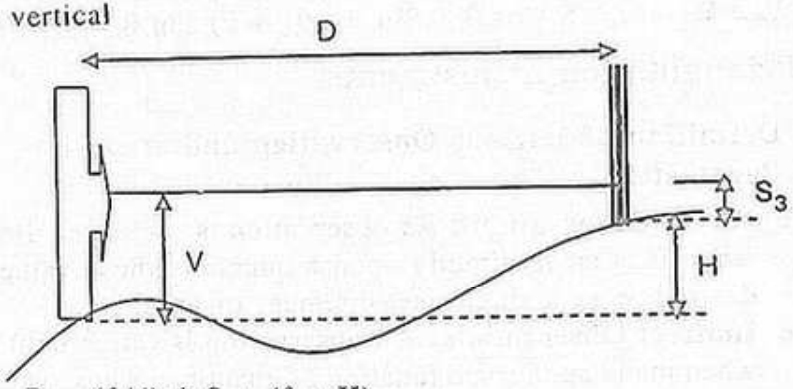
Therefore,

Horizontal distance = D = f / i * S + (f + K) = f / i * S + C (assume)

Vertical distance = Height of instrument – S_3

5.8.3 Determination of Horizontal and Vertical Distance in Different Conditions:

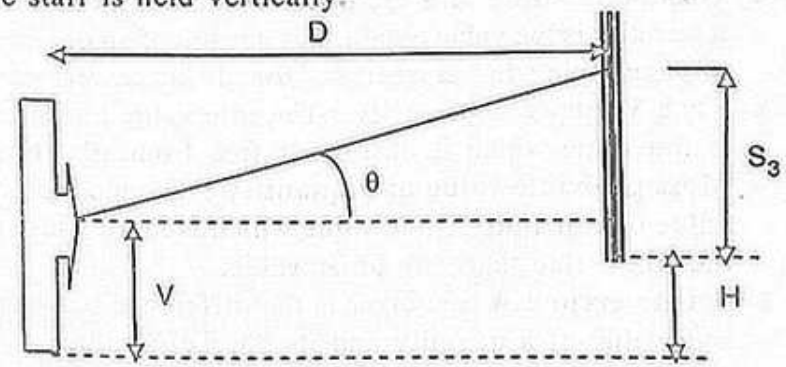
CASE I : when the line of sight is horizontal and the staff held vertical



D = (f / i) * S + (f + K)

V = H – S_3

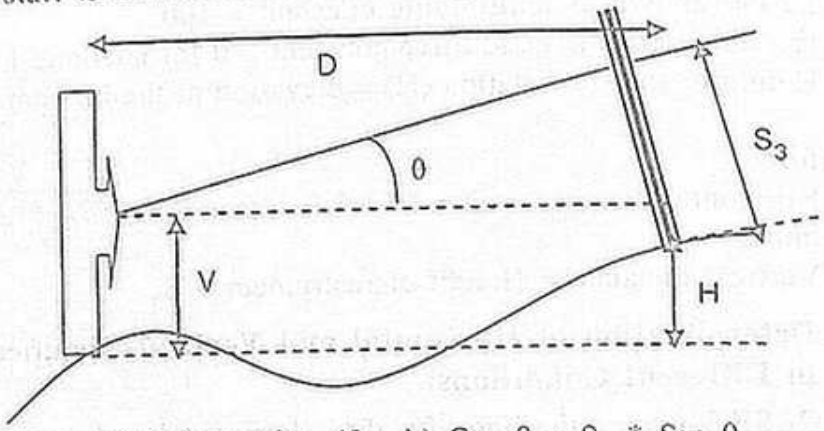
CASE II : when the line of sight is inclined to the horizontal and the staff is held vertically.



D = f/i * S $\cos^2\theta$ + (f + k) Cos θ

V = H + f/i * S Cos θ * Sin θ + (f + k) Sin θ – S_3

CASE III : when the line of sight is inclined to the horizontal and staff is normal to the line of sight



$D = f/i * S \cos\theta + (f + k) \cos\theta + S_3 * \sin\theta$

$V = H + f/i * S \cos\theta * \sin\theta + (f + k) \sin\theta - S_3 * \sin\theta$

5.9 Triangulation Adjustments

5.9.1 Definition of Various Observation and error Parameters

- **Direct Observation :** An observation is said to be direct when it is made directly upon a quantity whose value is desired. e.g., a single measurement of an angle.
- **Indirect Observation :** An observation is called indirect when made upon some function of quantities whose values are to be determined, e.g. measurement of a summation angle for total sum of the individual angles, measurement of an angle by repetition.
- **Observed Value of a Quantity :** The observed value of a quantity is the value obtained as a result of an observation, after applying the corrections for all known errors.
- **True Value of a Quantity :** The true value of a quantity is the value which is absolutely free from all errors.
- **Most probable value of a quantity :** The most probable value of a quantity is the value which is more likely to be the true value than any other value.
- **A true error :** A true error is the difference between the true value of a quantity and its observed value.
- **A residual error :** A residual error is the difference between the most probable value of a quantity and its observed value.

5.9.2 The Most Probable Values of Quantities

5.9.2.1 Direct Observation of Equal Weight

$$\text{Most Probable Value } (z) = \frac{A_2 + A_2 + \ldots A_n}{n}$$

Where, A_1, $A_2 \ldots A_n$ are the angles for observation 1, 2...n respectively.

5.9.2.2 Direct Observation of Unequal Weight

$$\text{Most Probale value } (z) = \frac{w_1 A_1 + w_2 A_2 + \ldots w_n A_n}{w_1 + w_2 \ldots + w_n}$$

Where, A_1, $A_2 \ldots A_n$ are the angles for observation 1, 2...n respectively; w_1, $w_2 \ldots w_n$ are the weights of observation 1, 2,...n respectively.

5.9.2.3 Indirect Observation of Equal Weight : To form a normal equation for each of the unknown quantities, multiply each observation equation by the algebraic coefficient of that unknown quantity in that equation, and add the results.

e.g. $A = \theta_1$, $2A = \theta_2$; $5A = \theta_3$;

Normal equations for the above observations are

$1*A = 1*\theta_1$;

$2*2A = 2*\theta_2$;

$5*5A = 5*\theta_3$;

$$\text{Most Probable Value } (A) = \frac{\theta_1 + 2\theta_2 + 5\theta_3}{1 + 4 + 25}$$

5.9.2.4 Indirect Observation of Unequal Weight : To form a normal equation for each of the unknown quantities, multiply each observation equation by the product of algebraic coefficient of that unknown quantity in that equation and the weight of that observation, and add the results.

e.g. $A = \theta_1$ weight 2; $2A = \theta_2$ weight 5

Normal equations for the above observations are

$1*2*A = 2\theta_1$;

$2*5*A = 10\theta_2$,

$$\text{Most Probable Value } (A) = \frac{2\theta_1 + 10\theta_2}{2 + 10}$$

5.9.3 The Probable Error

5.9.3.1 Direct Observation of Equal Weight

$$\text{The probable error of a single observation} = 0.6745\sqrt{\frac{\sum v^2}{(n-1)}}$$

The probable error of the arithmetic mean = $0.6745\sqrt{\frac{\sum v^2}{n(n-1)}}$

5.9.3.2 Direct Observation of Unequal Weight

The probable error of a single observation of unit weight =

$$0.6745\sqrt{\frac{\sum wv^2}{(n-1)}}$$

The probable error of the weighted arithmetic mean =

$$0.6745\sqrt{\frac{\sum wv^2}{\sum w(n-1)}}$$

Where, v = the difference between the measured value and the arithmetic mean

n = the number of measurements.

w = the weight of measurement.

5.9.4 Laws of Weights

The following laws of weights are established by the method of least squares.

1. The weight of the arithmetic mean of observations of unit weight is equal to the number of observations.
2. The weight of the weighted arithmetic mean is equal to the sum of the individual weights.
3. If two or more quantities are added or subtracted algebraically, the weight of the result is equal to the reciprocal of the sum of the reciprocals of the individual weights.

   e.g. $\alpha = \theta_1$ weight a_1

   $\beta = \theta_2$ weight a_2

   $$\text{Weight of } \alpha + \beta = \text{Weight of } \alpha - \beta = \frac{1}{\left(\frac{1}{a_1} + \frac{1}{a_2}\right)} = \frac{a_1 a_2}{a_1 + a_2}$$

4. If a quantity is multiplied by a factor, the weight of the product is equal to the weight of that quantity divided by the square of that factor.

   e.g. $\alpha = \theta_1$ weight a

   $3\alpha = 3\theta_1$ weight $a/3^2$

5. If a quantity is divided by a factor, the weight of the product is equal to the weight of that quantity multiplied

   e.g. $\alpha = \theta_1$ weight a

   $3\alpha = 3\theta_1$ weight $a*3^2$

6. Weights vary directly as the number of observations in case of angles, and inversely as the lengths of the various routes in the case of lines of levels.
7. Weights are inversely proportional to the squares of the corresponding probable errors, if the angles are measured a large number of times.

5.9.5 Computed Quantities

Case I : The computed quantity is the sum or difference of an observed quantity and a constant.

Let x = the computed quantity

e_x = its probable error

a = the observed quantity

ea = its probable error

k = a constant

then $x = \pm a \pm k$

and $e_x = ea$

Case II : The computed quantity is obtained by the product of an observed quantity and a constant factor.

Then $x = k.a$

$ex = k.e_a$

Case III : The computed quantity is the algebraic sum of two or more independently observed quantities.

$a_1, a_2, a_3 \ldots a_n$ are the independently observed quantities.

$e_1, e_2, e_3 \ldots e_n$ are the probable errors of $a_1, a_2, a_3 \ldots a_n$ respectively

Then $x = \pm a_1 \pm a_2 \pm a_3 \pm \ldots a_n$

and $e_x = \sqrt{e_1^2 + e_2^2 + \ldots e_n^2}$

Case IV : The computed quantity is any fraction of a single observed quantity.

$x = \phi(a)$

Then, $e_x = e_a \frac{dx}{da}$

Case V : The computed quantity is any fraction of two or more independentlyobserved quantities.

$x = \phi(a, b)$

Then, $e_x = \sqrt{\left(e_a \frac{dx}{da}\right)^2 + \left(e_b \frac{dx}{db}\right)^2}$

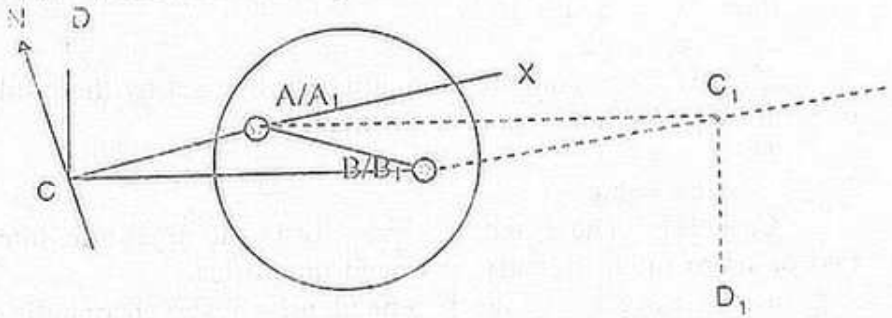
5.10 Correlation

5.10.1 Introduction

Correlation is the type of survey, where the relative position of the underground base line is determined with reference to the surface base line. There are number of correlation methods such as

- Direct traversing through mines entry
- Precise magnetic methods
- Gyro theodolite method
- Optical transfer
- Shaft plumbing : There are a number of shaft plumbing methods :
  - two shafts plumbing
  - single shaft plumbing – Co-planning or exact alignment; Weissbach triangle method; Weiss quadrilateral method

5.10.2 Weisbach Triangle Method



CD = Surface base line whose bearing is known

C_1D_1 = Underground base line whose bearing has to be determined with respect to CD

ABC and ABC_1, known as weissbach triangle

Theodolite is set at the point C and the angles <DCA & <ACB has determined

Assuming, <DCA = α and <ACB = β

In triangle ABC,

AB/ Sin C = AC/ Sin B

⇒ Sin B = b*Sin C/ c

Since angles B and C are very small, Sin B = <B and Sin C = <C

Therefore, <B = (b* angle C)/ c = γ (Assume)

azimuth of AB = azimuth of CD + <DCA + <XAB

= azimuth of CD + <DCA + <ACB + <ABC

Now the theodolite is fixed in underground station C_1 and the angles $B_1C_1D_1$ and $A_1C_1B_1$ has been measured

Again from sine rule, we can get

Angle A_1 = (B_1C_1/ A_1B_1) * angle C_1 = a_1 *angle C_1/c_1

Azimuth of line B_1C_1 = azimuth of AB + angle A_1 + angle C_1

Azimuth of line C_1D_1 = azimuth of line B_1C_1 + angle (180 – <$B_1C_1D_1$)

5.11 GPS

5.11.1 Introduction

The Global Positioning System (GPS) is a satellite-based navigation system made up of a network of 24 satellites placed into orbit by the U.S. Department of Defence. GPS was originally intended for military applications, but in the 1980s, the government made the system available for civilian use. GPS works in any weather conditions, anywhere in the world, 24 hours a day. There are no subscription fees or setup charges to use GPS.

5.11.2 How it Works?

GPS satellites circle the earth twice a day in a very precise orbit and transmit signal information to earth. GPS receivers take this information and use triangulation to calculate the user's exact location. Essentially, the GPS receiver compares the time a signal was transmitted by a satellite with the time it was received. The time difference tells the GPS receiver how far away the satellite is. Now, with distance measurements from a few more satellites, the receiver can determine the user's position and display it on the unit's electronic map.

A GPS receiver must be locked on to the signal of at least three satellites to calculate a 2D position (latitude and longitude) and track movement. With four or more satellites in view, the receiver can determine the user's 3D position (latitude, longitude and altitude). Once the user's position has been determined, the GPS unit can calculate other information, such as speed, bearing, track, trip distance, distance to destination, sunrise and sunset time and more.

5.11.3 How Accurate is GPS?

Today's GPS receivers are extremely accurate, thanks to their parallel multi-channel design. Garmin's 12 parallel channel receivers are quick to lock onto satellites when first turned on and they maintain strong locks, even in dense foliage or urban settings with tall buildings. Certain atmospheric factors and other sources of error can affect the accuracy of GPS receivers. GPS receivers are accurate to within 15 meters on average.

5.11.4 GPS Applications in Coal Industry

GPS entered the Mining Industry as a fast and cost-effective instrument for survey. A shifting landscape is the very nature of mining operations; as shovels and dozers remove coal and ore, they reshape the mine's surface. Real-time GPS allows mining operations to keep on top of these constant changes and provide updated operating instructions to heavy equipment operators. In addition, GPS systems provide a fast and accurate solution for replacing and maintaining control points and calculating the volume of material moved.

Moving mining assets, including dozers, shovels, graders and draglines, are managed and guided using advanced GPS technology. Advanced GPS systems also track and monitor the status and location of dump trucks, providing reports to their heading and velocity as well as the size of the truck's load. Live GPS is becoming commonplace for monitoring and dispatching haul trucks or drills and for providing grade control on shovels. These data can also be tied to a GIS to monitor the location of all equipment, in real time.

GPS being an all-weather real time, continuously available, economic and very precise positioning technique, would have wide range of applications in Indian coal industry. The potential areas for usage of GPS in Indian coal industry includes –

- **Surveying**

Several modern surveying techniques like Satellite Remote Sensing, Photogrammetric, Field surveying procedures using digital theodolites, short and long range EDM instruments like Total Station are available today. However, the advantages of using the satellite based GPS techniques for surveying are :

- GPS measurements do not require inter-visibility between points whereas the conventional surveying tools require line of sight for measurements.
- GPS technique provides a three dimensional position for the point. That is in one go, we get the horizontal and vertical position of the point, unlike in conventional surveying where we need two operations viz., horizontal traverse for planimetric control and a level loop for height control.
- A very high accuracy measurement can be made in a relatively short time for baseline lengths of a few hundred meters to few hundred kilometers and can provide the same accuracy anywhere on earth, in almost any weather condition and at any time of the day.

GPS offers many advantages compared with conventional survey methods. Because there is no need for a rod person, each surveyor can work alone when necessary. GPS also requires much less setup time than did traditional surveying equipment, so the crew can use its time more efficiently. It can also keep a much more flexible schedule and move from one area to the next or one pit to another as needed.

- **Opencast Mine Operations**

The entire opencast mine earthmoving process can be monitored and enhanced by GPS equipped heavy earth moving mining machinery (HEMM). Computerized mine operations management system would include tracking of mining equipment, maintenance, monitoring and diagnostic systems, transmission of loading instructions to dumpers etc for better control.

5.11.5 The GPS Satellite System

The 24 satellites that make up the GPS space segment are orbiting the earth about 12,000 miles above us. They are constantly moving, making two complete orbits in less than 24 hours. These satellites are travelling at speeds of roughly 7,000 miles an hour.

GPS satellites are powered by solar energy. They have backup batteries onboard to keep them running in the event of a solar eclipse, when there's no solar power. Small rocket boosters on each satellite keep them flying in the correct path.

Here are some other interesting facts about the GPS satellites (also called NAVSTAR, the official U.S. Department of Defence name for GPS) :

- The first GPS satellite was launched in 1978.
- A full constellation of 24 satellites was achieved in 1994.
- Each satellite is built to last about 10 years. Replacements are constantly being built and launched into orbit.
- A GPS satellite weighs approximately 2,000 pounds and is about 17 feet across with the solar panels extended.
- Transmitter power is only 50 watts or less.

For better understanding of GPS, we normally consider three segments viz. space segment, control segment and user segment. Space segment deals with GPS satellite system, control segment describes ground based time and orbit control prediction and in user segment various types of existing GPS receiver and its application is dealt.

Space segment consists of 21 GPS satellite with an addition of 3 active spares. These satellites are placed in almost six circular orbits with an inclination of 55°. An orbital height of these satellites is about 20,200 km corresponding to about 26,600 km from the semi major axis. Final arrangement of 21 satellite constellation known as Primary Satellite Constellation. There are 6 orbital planes A to F with a separation of 60^0 at right ascension. GPS satellites are mainly divided into three blocks.

5.12 Photogrammetry

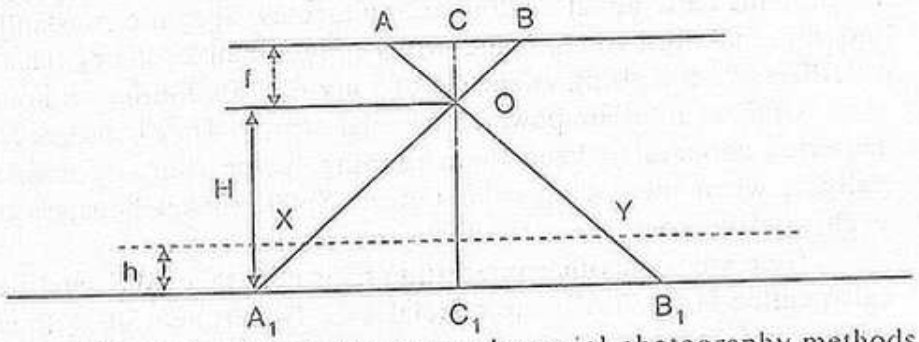
5.12.1 Introduction

This is a type of surveying in which plans or maps are prepared from photographs taken at suitable camera stations. In photo-theodolite, camera is a fixed focus type. The focal length of the lens should be 15 cm or more.

There are two types of photogrammetry survey

- Terrestrial or ground photogrammetry
- Aerial photogrammetry

5.12.2 Aerial Photogrammetry



The above figure represents the aerial photography methods where

A_1B_1 = ground surface covered by the camera

C_1OC = principal axis of the camera

CO = principal distance and should be equal to the focal length (f) of the lens

Scale of the photograph or R.F. along the ground level A_1B_1 = f / H

Scale of the photograph or R.F. along the line XY = f / (H-h)

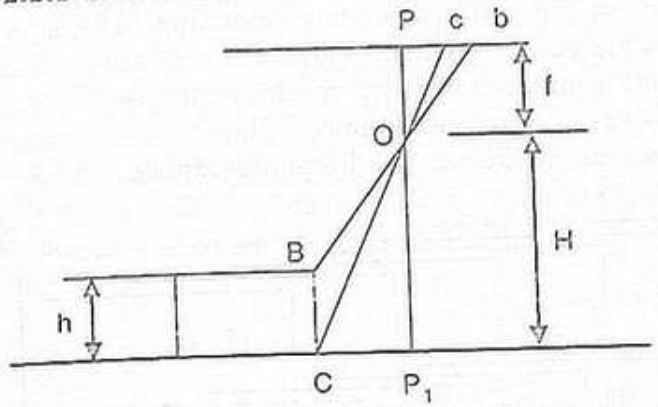
Average photo scale can be obtained by

Average photo scale = f / ($H-h_{av}$)

Where, h_{av} = average height of ground = $(h_1+h_2+\ldots h_n)/n$

$h_1, h_2 \ldots h_n$ = are the ground height at n number of points.

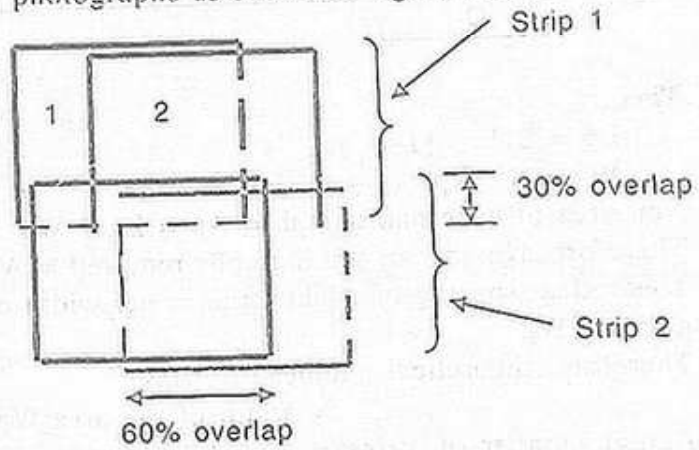
5.12.2.1 Distortion Due to Height or Relief



Though, B is exactly above the point C, it appears as cb in photo-print. bc is called the distortion due to the height BC.

$$\text{Distortion} = \frac{h}{H-h} * Pc = \frac{h}{H} * Pb$$

5.12.2.2 Flight Planning : In order to obtain stereo pairs, every parts of the ground to be surveyed must be photographed at least twice. To achieve this, aeroplane flies in strips and takes photograph with some portion overlapped (generally 60 %) with the adjacent photographs as shown in figure below.



5.12.2.3 Determination of Number of Photographs for a Particular Area

Assuming,

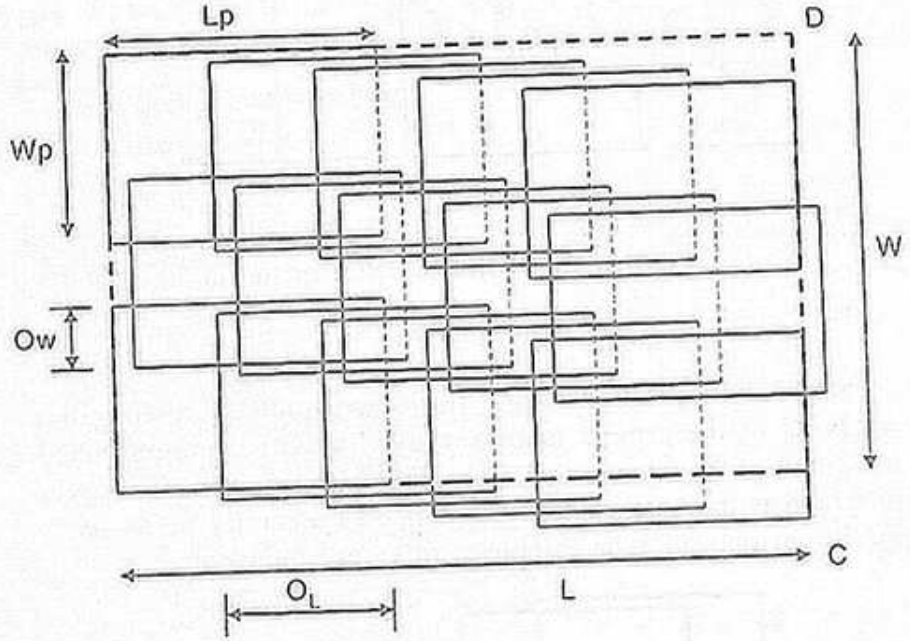
Lp = length of photograph in the direction of flight.

Wp = width of photograph perpendicular to the direction of flight.

O_L = the percentage of the longitudinal overlap.

O_W = the percentage of the side overlap.

Lg = net ground distance corresponding to Lp.
Wg = net ground distance corresponding to Wp.
S = the scale of photograph (1cm = S m).
N = the number of photographs required.
Ap = net area of each photograph.
Ag = area of the tract to be photographed.



Then,

$Lg = S*Lp*(1-O_L)$

$Wg = S*Wp*(1-O_W)$

Net Area of each photograph = Ap = Lg * Wg

Therefore, Number of photographs required = Ag/Ap

Theoretical spacing of flight strips = net width of a single photograph = Wg

Therefore, theoretical number of strips = width of the area/Wg = K

Actual number of strips = K ÷ 1 [one strips required to cover the sides]

Theoretical number of photographs per strip = Length of the area/Lg = M

Actual number of photographs per strip = M ÷ 1 [one photograph being added to cover the end of the area]

Thus, in practice there would be (K ÷ 1) strips with (M ÷ 1) photographs in each.

Actual number of photographs for complete coverage of the area = (K+1) * (M+1)

5.12.2.4 Intervals between Exposures : The time interval between exposures depends upon the speed of the aeroplane and the distance it travels between exposures.

Time interval = 3600*Lg/V [Second]

Where, V is the speed of the aeroplane in km/hr. and

Lg, the distance the aeroplane travels between exposures in km.

EXERCISE

1. The ground distance between the mine managers office and pit top is 300 m and it is represented by 3 cm in the mine plan. What is the representative factor?
   a. 1: 100 b. 1: 1000
   c. 1: 10,000 d. 1: 300
2. The length of a line measured with a 20 m chain was found to be 536 m. It was afterwards found that the chain was 0.05 m too long. What will be the true length of the line?
   a. 568.5 b. 537.3
   c. 587.5 d. 512.3
3. The length of a line measured with a 20 m chain was found to be 536 m. It was afterwards found that the chain was 0.05 m too short. What will be the true length of the line?
   a. 568.1 b. 534.7
   c. 523.5 d. 512.3
4. The length of a line measured by means of a 20 m chain was found to be 610.2 m known to be 608.0. What was the actual length of the chain?
   a. 20.066 m b. 20.012 m
   c. 19.906 m d. 19.928 m
5. The length of a line measured by means of a 20 m chain was found to be 610.2 m known to be 612.0. What was the actual length of the chain?
   a. 20.066 m b. 20.059 m
   c. 19.059 m d. 19.012 m
6. A certain field was measured with a 30 m chain and found to contain 45 km^2. It was afterwards found that the chain was 0.1 m too short. What is the true area of the field?

a. 44.70 km^2 b. 45.76 km^2
c. 70.52 km^2 d. 65.20 km^2

7. If in a length of one 100 m chain along a slope, the ground rises 5 m, find the correction factor for 1 chain length.
a. 0.125 b. 0.325
c. 0.250 d. 0.050

8. The length measured along slope with a 20 m chain is 100 m and the ground rises is 15 m. What is the actual horizontal distance between the two points?
a. 96.857 b. 96.825
c. 100.85 d. 98.875

9. Find the correction factor per 20 m chain for a line measured along a slope of 15°.
a. 0.681 m b. 0.678 m
c. 0.568 m d. 0.786 m

10. The distance between two points A and B measured along a slope of 1 in 2.5 is 275 m. What will be the horizontal distance between two points?
a. 265.83 m b. 216.56 m
c. 287.67 m d. 255.33 m

11. A line was measured with a steel tape which was exactly 30 m long at 18° C and found to be 550 m. The temperature during measurement was 32°C. Find the true length of line, if the coefficient of expansion of the tape is $1.18*10^{-5}$/ °C.
a. 550.09 m b. 551.10 m
c. 549.01 m d. 549.09 m

12. A steel tape is exactly 30 m long at 20°C and coefficient of linear expansion is $11*10^{-6}$ per °C. The mean temperature during measurement is 15°C. The correction for temperature in mm is
a. +1.65 b. -5.50
c. -1.65 d. +7.50
[GATE, 2002]

13. A 30 m tape is suspended between the ends under a pull of 150 N. The mass of the tape is 1.5 kg. What is the correct length of the tape?
a. 29.988 m b. 29.908 m
c. 28.980 m d. 30.008 m

14. Odometer is used for measurement of
a. Distance b. Pressure
c. Temperature d. Area

15. What will be Quadrantal bearing of a line whose whole circle bearing is 236°25'?
a. S 56°25' W b. N 56°25' W
c. S 56°25' E d. S 43°45' E

16. The reduced bearing of a line is S 57°36' W. What will be the whole circle bearing of the line?
a. 237°36' b. 327°36'
c. 137°36' d. 212°24'

17. The fore bearing of a line AB is N 36° W. What will be the back bearing of the line AB?
a. 144° b. 54°
c. 324° d. 36°

18. The fore bearing of a line AB is N 36° W. What will be the back bearing of the line AB?
a. N 36° W b. S 36° E
c. N 54° W d. S 54° E

19. The fore bearing of a line AB is 136°. What will be the back bearing of the line AB?
a. 144° b. 54°
c. 316° d. 44°

20. For two survey points the coordinates are given. The line CD has length and bearing

Point	Southing (m)	Westing (m)
C	280.4	310.8
D	620.9	635.9

a. 1307.18 m, N 43°40'36'' E
b. 470.81 m, S 43°40'36''W
c. 470.81 m, 226°24'
d. 1307.18 m, S 43°40'44''W

21. What will be the angle between the two line OA and OB whose bearings are 56° and 154° respectively?
a. 98° b. 82°
c. 210° d. 18°

22. What will be the angle between the two lines OA and OB whose bearings are S56°E and N 54°E respectively?

a. 98° b. 70°
c. 2° d. 110°

23. The magnetic bearing of a line AB is 136°. Find its true bearing, if the magnetic declination is 8° W.
a. 128° b. 144°
c. 38° d. 54°

24. The magnetic bearing of a line AB is S 32° E and the magnetic declination is 4°E. What is the true bearing of the line AB?
a. 126° b. 152°
c. 36° d. 28°

25. The magnetic bearing of a line AB is S 32° E and the magnetic declination is 4°E. What is the true bearing of the line AB?
a. S 36° E b. N 36° E
c. S 28° E d. N 28° E

26. The true bearing of a line is 326° and the magnetic declination is 8° E. Find the magnetic bearing of the line.
a. 318° b. 334°
c. 138° d. 154°

27. The true bearing of a line is 349° and the magnetic declination is 15° W. Find the magnetic bearing of the line.
a. 274° b. 4°
c. 334° d. 154°

28. Find the magnetic declination, if the magnetic bearing of the sun at noon is 185°
a. 5° E b. 5° W
c. 10° E d. 10° W

29. Find the magnetic declination, if the magnetic bearing of the sun at noon is 358°
a. 7° E b. 7° W
c. 2° E d. 2° W

30. The magnetic bearing and declination of a line were recorded in the year 1906 as S 43°30'E and 2° E respectively. If the declination in the year 2006 is 3° W, the magnetic bearing of the line is
a. S 48°30' E b. S 45°30' E
c. S 41°30' E d. S 38°30' E
[GATE, 07]

31. The results of theodolite survey are given below. The line

Point	Northing (m)	Easting (m)
A	400.5	620.2
B	750.5	320.5

a. 460.78 m b. 349.70 m
c. 106.6 m d. 50.30 m
[GATE, 07]

32. The results of theodolite survey are given below. The line AB has bearing

Point	Northing (m)	Easting (m)
A	400.5	620.2
B	750.5	320.5

a. 23.17 NE b. 23.17 NW
c. 40.57 NW d. 40.57 NE
[GATE, 07]

33. The maximum permissible angular error in seconds in a closed underground traverse of 16 stations measured with a 20 seconds theodolite is
a. 200 b. 120
c. 80 d. 160

34. In tacheometric observations, the staff readings are found to be 2.4 m, 3.0 m and 3.6 m. If the angle of elevation is 45, the multiplying factor is 100 and the additive constant is 0.3 m then the horizontal distance between the staff and the instrument station, expressed in m is
a. 30.20 b. 59.76
c. 60.19 d. 73.21

35. The following consecutive staff readings were taken on continuously sloping ground at a common interval of 30 m:

0.5 0.9 1.2 2.6 3.8

A ———————————————— B

If the elevation of A is 500 m, what will be the elevation of point B?
a. 496.7 m b. 500.3 m
c. 503.3 m d. 503.8 m

36. The following consecutive staff readings were taken with a theodolite on continuously sloping ground at a common interval of 30 m :
0.5 on A, 1.3, 2.2, 1.6, 2.8 on B

If the elevation of A is 510 m, what will be the elevation of point B?

a. 496.7 m
b. 504.2 m
c. 507.7 m
d. 503.8 m

37. The following consecutive readings were taken with a theodolite on continuously sloping ground at a common interval of 30 m:

0.5 on A, 1.3, 2.2, 1.6, 2.8 on B

If the elevation of A is 500 m, what will be the gradient of line AB?

a. 5.5°
b. 6.5°
c. 10.5°
d. 2.5°

38. In levelling between two points A & B on opposite side of an opencast mine, the level was set up near A and the staff readings on A and B were 2.5 m and 3.7 m respectively. The level was then moved and set up near B, and the respective staff readings on A and B were 2.0 m and 3.0 m. Find the true difference of level A and B.

a. 1.0 m
b. 1.1 m
c. 0.6 m
d. 1.3 m

39. Two points A and B were 1000 m apart across a river. The following readings were taken with one level :

Level near station	Reading on A	Reading on B
A	1.5	2.8
B	1.4	2.6

The error in the collimation adjustment of the level was + 0.002 m per 100 m. Determine the true difference of level between A and B.

a. 1.25
b. 0.05
c. 2.5
d. 0.1

40. Two points A and B were 1000 m apart across a river. the following readings were taken with one level:

Level near station	Reading on A	Reading on B
A	1.5	2.8
B	1.4	2.6

The error in the collimation adjustment of the level was +

a. 0.0450
b. 0.0485
c. 0.4850
d. 0.1850

41. The effect of curvature in levelling is an error which is

a. additive
b. subtractive
c. multiplicative
d. divisive

42. The effect of refraction in levelling is an error which is

a. additive
b. subtractive
c. multiplicative
d. divisive

43. The combined effect of curvature and refraction in levelling is an error which is

a. additive
b. subtractive
c. multiplicative
d. divisive

44. What is the correction for curvature for a distance of 1200 m?

a. – 0.0113 m
b. + 0.0113 m
c. – 0.113 m
d. + 0.113 m

45. What will be the correction for refraction for a distance of 800 m, if the diameter of the earth is 12742 km?

a. – 0.007 m
b. + 0.007 m
c. – 0.7 m
d. + 0.7 m

46. What is the combined correction for curvature and refraction for a distance of 1200 m, if the diameter of the earth is 12000 km?

a. – 0.103 m
b. + 0.103 m
c. – 1.030 m
d. + 1.030 m

47. A level is set up at O on a line AB 100 m from A and 1500 m from B. The back sight on A is 0.548 m and the fore sight on B is 5.263 m. What will be the true difference of level A and B?

a. 4.563 m
b. 4.356 m
c. 4.715 m
d. 5.801 m

48. A level is set up at O on a line AB 600 m from A and 800 m from B. The back sight on A is 0.348 m and the fore sight on B is 1.263 m. What will be the true difference of level A and B?

a. 0.896 m
b. 0.915 m
c. 1.544 m
d. 1.611 m

49. From the following page of a levelling field book, the missing values in F.S. and B.S respectively are

Station	B.S.	I.S.	F.S.	Rise	Fall	Remarks
1	4.550					Starting Point
2	2.125		?		0.750	Change point
3		2.225				
4	?		1.975			Change point
5		2.445		1500		

a. 3.804, 0.945 b. 3.804, 3.945
c. 5.300, 0.945 d. 5.300, 3.945

[GATE, 2007]

50. Identify the correct statement for the rise and fall method of RL computation
a. Σ F.S – Σ B.S = Σ Rise – Σ Fall = Σ R.L. of first point – Σ R.L of last point.
b. Σ B.S. – Σ F.S. = Σ Rise – Σ Fall = Σ R.L. of last point – Σ R.L. of first point.
c. Σ F.S – Σ B.S. = Σ R.L. of last point – Σ R.L. of first point.
d. Σ B.S – Σ F.S. = Σ Fall – Σ Rise

51. In tacheometric observations, the staff readings are found to be 2.4 m, 3.0 m and 3.6 m. If the angle of elevation is 45°, the multiplying factor is 100 and the additive constant is 0.3 m then the horizontal distance between the staff and the instrument station, expressed in m is
a. 30.20 b. 59.76
c. 60.19 d. 73.21

52. In tacheometric observations, the staff readings are found to be 1.2 m, 2.0 m and 3.0 m. If the angle of elevation is 30°, the multiplying factor is 100 and the additive constant is 0, then the horizontal distance between the staff and the instrument station, expressed in m is
a. 30.20 b. 59.76
c. 60.19 d. 77.94

53. The most probable value of the angle A from the following observation equations.
a. $10^0 10'$ b. $10^0 3.8'$
c. $9^0 50'$ d. $10^0 2.6'$

54. The most probable value of the Angle A from the following observation equations :
$A = 15^0$ weight 2; $2A = 30^0 10'$ weight 4
a. $15^0 4.4'$ b. $15^0 6.8'$
c. $15^0 4'$ d. $15^0 4.9'$

55. The most probable value of 'A' for the following data :
$A = 20^0$, $B = 50^0$, $A + B = 70^0 10'$
a. $20^0 5.5'$ b. $20^0 3.3'$
c. $20^0 4.3'$ d. $20^0 5'$

56. Given A = 30° ± 1.2; B = 40° ± 1.4;
The most probable value of A+B is
a. 70° ± 1.3 b. 70° ± 1.5
c. 70° ± 1.6 d. 70° ± 1.8

57. Given A = 30° ± 1.4; B = 40° ± 1.6;
The most probable error of A+B is
a. 1.45 b. 2.56
c. 2.13 d. 3.66

58. The most probable value of the area of a circle whose radius is 10° ± 0.04
a. 314 ± 2.5 b. 314 ± 2.6
c. 314 ± 2.7 d. 314 ± 2.8

59. The most probable value and the probable error of the area of a rectangle whose sides are 50 ± 0.04 m and 150 ± 0.06 m
a. 7500 ± 2.7 b. 7500 ± 4.7
c. 7500 ± 5.7 d. 7500 ± 6.7

60. The most probable value of the volume of a cylinder whose height and radius are 10 ± 0.02 and 5 ± 0.01
a. 785 ± 3.87 m^3 b. 785 ± 2.87 m^3
c. 785 ± 5.34 m^3 d. 785 ± 0.53 m^3

61. In shaft plumbing, the plumb bobs are immersed in a bucket of water to prevent
a. oscillation of wires b. effect of ventilation current
c. kinking of the wires d. none of the above

62. In weissbach triangle method of correlation, number of plumb bobs requires for suspensions
a. one b. two

63. In co-planing or exact alignment method of correlation, number of plumb bobs requires for suspensions
a. one b. two
c. three d. four

64. In weissbach quadrilateral method of correlation, number of plumb bobs requires for suspensions
a. one b. two
c. three d. four

65. In precise magnetic method of correlation, number of plumb bobs requires for suspensions
a. one b. two
c. three d. four

66. In weiss quadrilateral method of correlation, number of shaft used for correlation
a. one b. two
c. three d. four

67. In precise magnetic method of correlation, number of shaft used for correlation
a. one b. two
c. three d. four

68. In weissbach triangle method of correlation, number of shaft used for correlation
a. one b. two
c. three d. four

69. The basic concept behind GPS is
a. Triangulation b. Trilateration
c. Traverse d. Tacheometry

70. The number of satellites and orbital planes in a GPS constellation (US system) are
a. 21 and 3 b. 24 and 4
c. 24 and 6 d. 25 and 5

71. The GPS satellites orbital planes are inclined to the equator and spaced in the longitude. The inclination and the spacing of the longitude in degrees are
a. 45° and 55° b. 55° and 60°
c. 60° and 55° d. 60° and 60°

72. The minimum number of satellites required to locate the position of any object on the earth surface
a. 2 b. 3

73. The minimum number of satellites required to locate the position of any object on the air
a. 2 b. 3
c. 4 d. 5

74. GPS gives height from
a. WGS83 b. WGS84
c. WGS85 d. None of the above

75. What is the name of navigation and positioning system proposed by India?
a. GLONASS b. GSTB
c. IRNSS d. EGNOS

76. C/A stands for
a. coarse acquisition b. conditional access
c. code access d. code acquisition

77. Satellite orbits are inclined at an angle of 55^0 from
a. Greenwich meridian b. Equator
c. Specific latitude d. Specific longitude

78. Orbital altitude of GPS satellite from earth surface
a. 15000 km b. 20000 km
c. 25000 km d. 30000 km

79. GPS is a satellite based radio navigation system developed and operated by
a. Department of Commerce b. Department of Defence
c. Department of Mines d. Geological Survey

80. DGPS stands for
a. Different Global Positioning System
b. Direct Global Positioning System
c. Differential Global Positioning System
d. Dynamic Global Positioning System

81. GIS stands for
a. Global Information System
b. Geological Information System
c. Geographical Information System
d. Group Information System

82. Determine the airplane flying height to obtain the average scale of the photograph equal 1/7200. Ground surface elevations vary from 160 to 430 m. Focal length of the camera lens is 153 mm.
a. 1397 b. 1497

83. The scale of the photograph is 1:100. The photograph size is 23 cm X 23 cm. Determine the number of photographs required to cover an area of 150 km^2 if the longitudinal overlap is 60 % and the side overlap is 30 %.

a. 102 b. 101
c. 103 d. 106

84. Determine the number of photographs required to cover an area of 10 × 20 km^2, if the longitudinal overlap is 70 % and the side overlap is 30 %. The scale of the photograph is 1: 100 and the photograph size is 25 cm × 25 cm.

a. 145 b. 135
c. 213 d. 195

85. An image of the top of a building is 90 mm from the principal point of the photograph. The elevation of the top of the building is 500 m and the flying height is 5000 m above the datum level. Determine the relief displacement.

a. 9 mm b. 11 mm
c. 13 mm d. 7 mm

86. The flight map shows the best position of 1000 m high mountain peak at 8 cm from the flight line at the anticipated scale of photography. If the flying altitude is 6000 m above the datum, estimate the relief displacement of the image of the peak and its distance from the edge of the print having a size of 23 × 23 cm.

a. 1.6 cm & 1.9 cm b. 1.8 cm & 1.9 cm
c. 2.1 cm & 1.9 cm d. 1.4 cm & 1.9 cm

87. A line 2500 m long lying at an elevation of 450 m measures 10 cm on a vertical photograph. The focal length of the camera is 20 cm. Determine the scale of the photograph for the area having an elevation of 1000 m.

a. 1: 24890 b. 1: 22980
c. 1: 26990 d. 1: 21990

88. A line measures 10 cm on a photograph taken with a camera having a focal length of 20 cm. The same line measures 3 cm on a map drawn to a scale of 1: 50000. Calculate the flying height of the aircraft if the average altitude of the ground is 320 m.

a. 3020 m b. 4050 m
c. 5000 m d. 3320 m

3000 m above mean sea level (MSL) of a terrain of average elevation of 750 m. If the focal length of the camera is 150 mm, then the scale of the photograph is

a. 1: 16000 b. 1: 18000
c. 1: 15000 d. 1: 14000

[GATE, 02]

Answer

1. c	2. b	3. b	4. d	5. b	6. a
7. a	8. d	9. a	10. d	11. d	12. a
13. a	14. a	15. a	16. a	17. a	18. b
19. c	20. b	21. a	22. b	23. a	24. b
25. c	26. a	27. b	28. b	29. c	30. a
31. a	32. d	33. c	34. c	35. a	36. c
37. a	38. b	39. a	40. b	41. b	42. a
43. b	44. c	45. b	46. a	47. a	48. a
49. d	50. b	51. c	52. d	53. d	54. a
55. b	56. d	57. c	58. a	59. d	60. a
61. a	62. b	63. b	64. b	65. a	66. a
67. a	68. a	69. a	70. c	71. b	72. b
73. c	74. b	75. c	76. a	77. b	78. b
79. b	80. c	81. c	82. a	83. b	84. d
85. a	86. a	87. d	88. d	89. c	

□

6.0 Mining Machinery

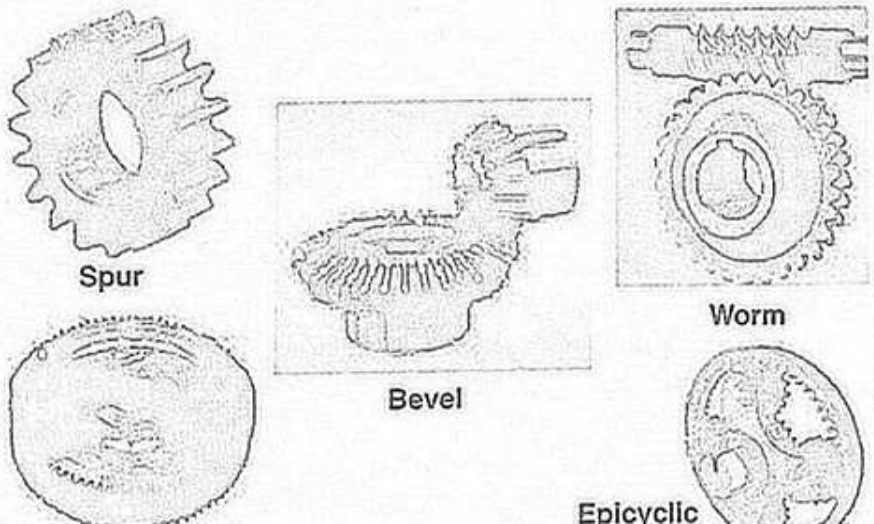
6.1 Types of Clutches

- **Friction Clutch :** Friction clutches can be engaged or disengaged without shock when the driving shaft is revolving at full speed, and the load on the driving engine or motor can be increased gradually from zero to full load. It may also be so arrange that the clutch slips if the load becomes excessive avoiding breakage of some important part of the plant. It is used on endless rope hauling engines.
- **Claw or Dog or Jaw Clutch :** This type of clutches can be engaged or disengaged at very low speed or nearly stationary. This type of clutches provides positive drive that cannot slip. Winding drum and large direct haulages require positive drive that cannot slip and hence use this type of clutches.

6.2 Types of Gears

Various types of gears as shown in Figure 6.1 are explained below.

- **Spur Gears :** Spur gears are the most common type used. Tooth contact is primarily rolling, with sliding occurring during engagement and disengagement.



- **Bevel Gears :** These are used for transmission of power between two shafts whose axes, if extended, would meet at a point.
- **Worm Gears :** This form of gearing is convenient to large speed reduction.
- **Helical Gear :** These are preferred to heavy drives.
- **Epicyclic Gear or Sun and Planetary Gear :** It is used for compactness and high reduction ratio between the speeds of the driver and driven shafts.

6.3 Wire Ropes

6.3.1 Introduction

Construction and composition of wire ropes is very important in order to maintain the safety in the mines. The wire is subjected to various tests like tensile, torsion, bending, wrapping, looping before use in the mines. The ropes used in the wet area should be galvanized i.e. coated with molten zinc.

6.3.2 Classification of Wire Ropes

6.3.2.1 On the Basis of Operation :

- ❑ **Standing Ropes :** The ropes are required more or less stationary to carry the load, e.g. guide ropes in shafts, bucket supporting ropes in bi-cable ropeways
- ❑ **Running Ropes :** The ropes are have to undergo frequent movement to carry the load e.g. ropes used in winding, cranes, coal cutting machines, excavators etc.

6.3.2.2 On the Basis of Construction

- ❑ **Stranded Ropes :** A stranded wire is built up of strands and each strand consists of number of concentrically twisted wires. The central wire is called the king core. It may be of fibre or steel. Wire ropes are designated by stating the number of strand followed by the number of wires in each strand. The flexibility of a strand depends upon the following factors :
  - **Type of Core :** Flexibility of the core is increases, flexibility of the strand also increases.
  - **Thickness of Individual Wires :** Thinner the wires, more is the flexibility.
  - **Number of Wires :** Larger the number of wires more is the flexibility

Stranded ropes are again of two types according to their laying

- Lang's Lay : If the wires in the strand are laid in the same direction as the strands are laid in the rope is called Lang's lay. This type of construction causes the rope to spin. These are favoured for winding and haulage purposes.
- Ordinary Lay or Regular Lay : If the wires in the strand are laid in the opposite direction as the strands are laid in the rope is called ordinary lay.

❑ Non-Stranded ropes : These types of ropes are constructed with number of wires placed in layer wise. Locked coil rope is of non-stranded type.

6.3.3 Application of Wire Ropes in Mines

- Winding Ropes : Flattened stranded rope of Lang's lay type
- Winding Ropes for Shaft Sinking : Non-rotating type (stranded regular lay) or locked coil type
- Haulage Ropes : Stranded type of Lang's lay or regular lay but preferably Lang's lay
- Guide Rope : Half locked coil type
- Coal Cutting Machines (CCM) : Regular lay-type

6.3.4 Space Factor

The space factor of a rope is the ratio of aggregate cross sectional area of the wires to the area of circle drawn around the rope. Strength of the ropes depends on the space factor. The space factor is high, strength of the rope is also high and vice-versa.

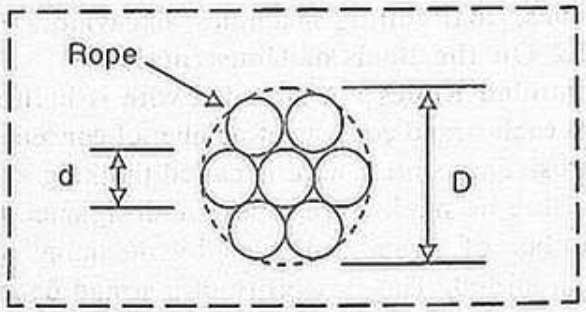


Figure 6.1 : Space Factor in Rope

For the rope shown in the above Figure 6.2,

Space Factor (SF) = $[7*\pi * (d/2)^2]/ [\pi* (D/2)^2] = 7*d^2 / D^2$

Where, d = diameter of the individual wire

D = diameter of the rope

6.3.5 Mass of Rope

The mass of rope depends upon the construction materials

Mass of rope (m) = $k* D^2$ [kg/m]

Where, k = constant depending on the design of the rope and type of material

D = diameter of the rope in cm

6.3.6 Strength of Rope

Strength of rope is also depends on the type of construction and the material used. It is given by

Breaking strength of rope = $s*D^2$ [kN]

Where, s = constant depending on the design of rope and type of material

D = diameter of the rope in cm

6.3.7 Rope Splicing

Joining of two wire ropes permanently without special attachments called rope splicing

6.3.8 Capping of Rope

It is a method to make the rope-end suitable for attachment of load. These are mainly of three types :

- Split capel with rivets
- Coned-socket type capel
- Interlocking type capel (Reliance capel)

6.4 Winding System

6.4.1 Types of Winding System in U/G Mines

6.4.1.1 Drum Winding

❑ Standards of Drum Winding System

- The diameter of the headgear pulley should be at least 100 times the rope diameter
- No spliced rope shall be in use for winding
- Rope is made of cold drawn steel wire
- Breaking load of the winding rope should be at least 10 times the maximum static load in case of a drum winding but it can be 7 times in case of Koepe winding
- Life of the winding rope should not be exceed 3.5 years
- The length of the tapered portion of the socket in rope capping should not be less than 12 times the rope diameter.
- Recapping of winding rope should be done once in 6 months except in Koepe winding

- The angle of fleet which is the angle between the vertical plane of the pulley and the rope when the cage is at the pit-top or pit-bottom, should not exceed 1.5°.

❑ **Various Fittings in Drum Winding Arrangement**

- **Arrangement of cage attachment**

Rope => Cappel => D-Link => Detaching Hook => D-Link => Bull Chain => D-Link => Triangular Plate => Cage Chain => Cage

- **Detaching Hook :** This is a safety device which acts when overwind takes place and used only in drum winding not in koepe winding. Two types are King and Ormerod detaching safety hook.
- **Safety Catches :** This is a safety device which helps to prevent the cage falling back down the shaft during overwind and in case of failure of detaching plate.
- **Keps :** Keps are retractable support used in pit top for landing of the cage only in case of drum winding arrangement not in koepe winding system
- **Tilted or Hinged Platform :** In koepe winding system, it is used at the pit top for landing of the cage.

❑ **Torque -Time diagram at Various Stages of the Drum Winding Operation :** The torque-time diagrams in drum winding are shown in Figure 6.3 and Figure 6.4.

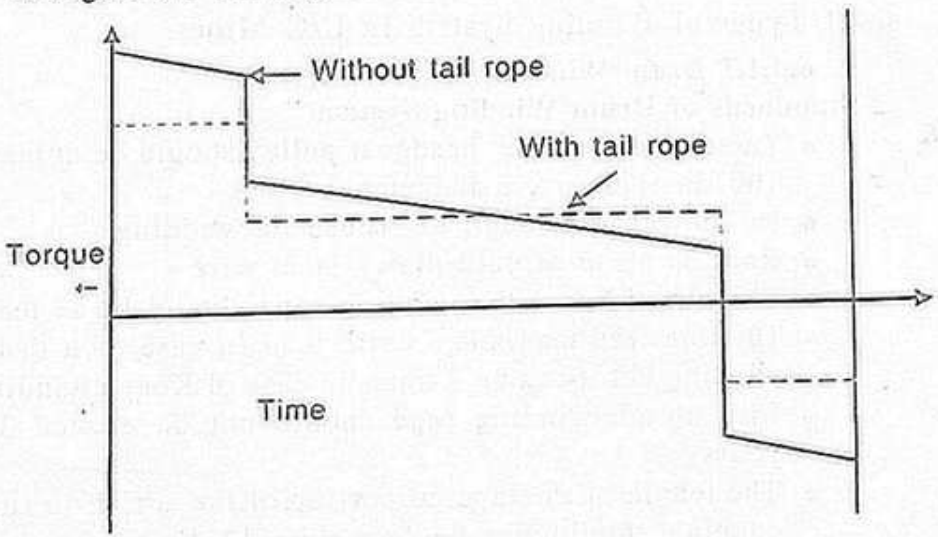


Figure 6.3 :
Torque-time diagram for balanced cylindrical drum

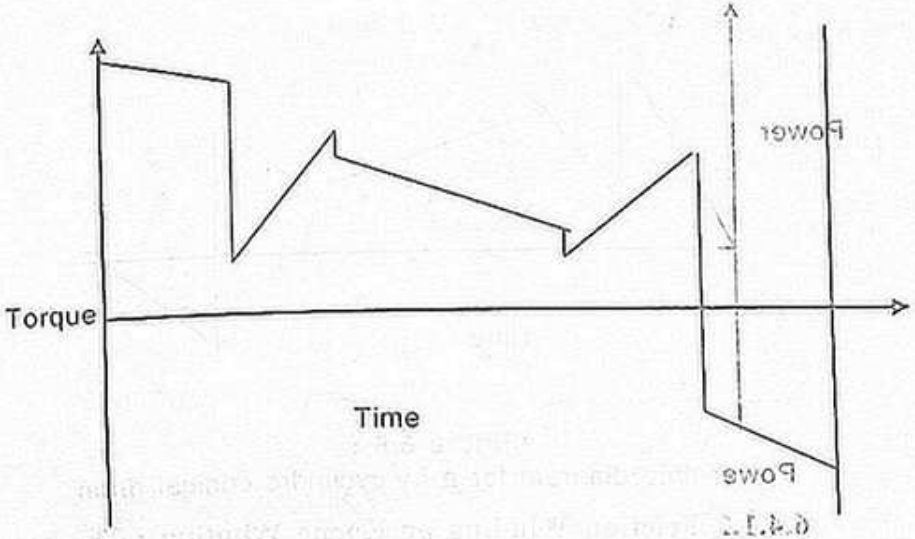


Figure 6.4 :
Torque-time diagram for a by-cylindro-conical drum

Power-Time diagram at Various States of the Drum Winding Operation : The power-time diagrams in drum winding are shown in Figures 6.5 and Figure 6.6

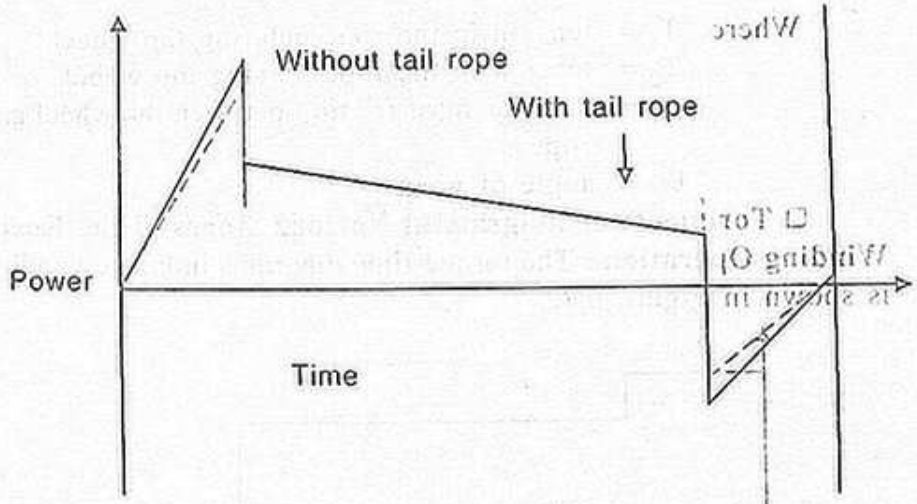


Figure 6.5 :
Power-time diagram for balanced cylindrical drum

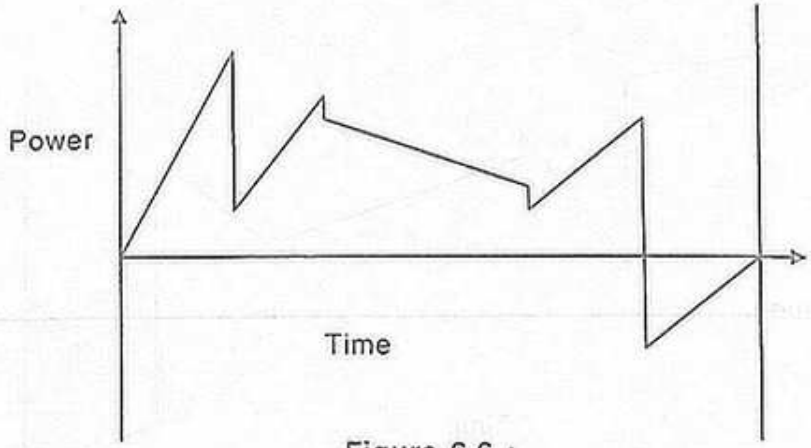


Figure 6.6 :
Power-time diagram for a by-cyclindro-conical drum

6.4.1.2 Friction Winding or Koepe Winding : The basic principle underlying the working of a friction winder is the fact that a belt or hoist rope passing over a wheel driven by a prime mover, has different tensions in the rope on two sides of the wheel. The relationship of these tensions is given by

$$\frac{T_1}{T_2} = e^{\mu\theta}$$

Where, T_1 = tension in the rope entering the wheel
T_2 = tension in the rope leaving the wheel
μ = coefficient of friction between the wheel and rope
θ = angle of wrap

❑ **Torque-Time diagram at Various Stages of the Koepe Winding Operation :** The torque-time diagrams in koepe winding is shown in Figure 6.7.

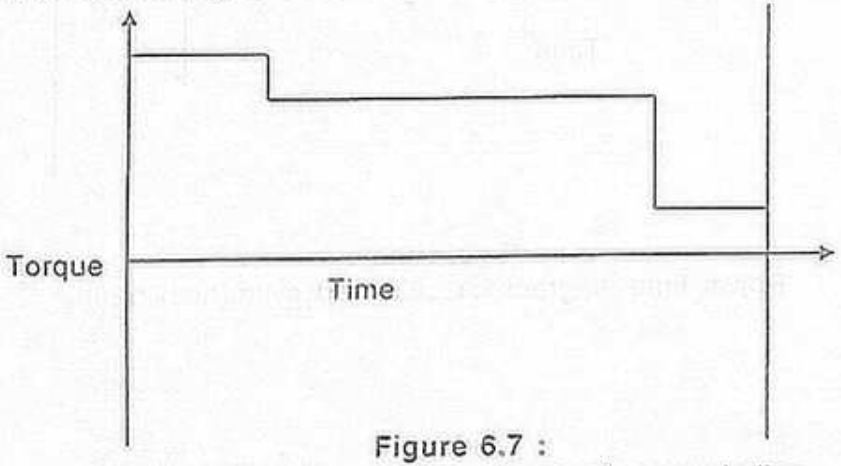


Figure 6.7 :
Torque-time diagram for a friction/koepe winding

❑ **Power-Time diagram at Various Stages of the Koepe Winding Operation :** The power-time diagram inkoepe winding is shown in Fig. 6.8.

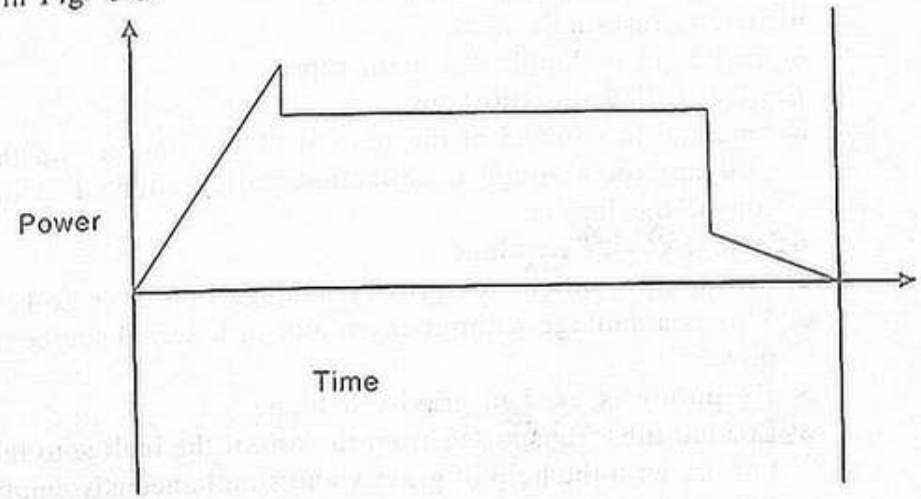


Figure 6.8 :
Power-time diagram for a friction/koepe winding

6.4.2 Pit Top Track Arrangement in Winding System

- Lofco system
- Run round arrangement
- Shunt back system
- Turn table circuit
- Traverse circuit

6.5 Underground Haulage System

6.5.1 Types of Haulage System

6.5.1.1 Direct Rope Haulage :

- Minimum gradient for applicability is 12°
- The rope speed is generally 8 – 12 km/ hr.
- Slip-ring motor with drum controller is used

6.5.1.2 Endless Haulage

- Suitable gradient for applicability is 1 in 6
- The rope speed is generally 3 -7 km/ hr.
- Rope moves uni-directionally
- Clifton pulley is used as a driving pulley
- A squirrel cage motor generally used
- This may over rope or under rope arrangement
- Tubs can be attached with the rope individually or as in a set with the help of three types of clips (Screw clip, Smallman clip or Cam clip)

- It requires double track

6.5.1.3 Main and Tail Rope Haulage

- Suitable for undulating roadways
- It requires single track
- Tail rope is double the main rope

6.5.1.4 Tail Rope Haulage

- Haulage is situated at the bottom of the incline and the pull applied through a deflection pulley situated at the top of the incline

6.5.1.5 Gravity Haulage

- Minimum gradient for gravity haulage operation is 14°
- This is a haulage without any motor or external source of power
- Jig pulley is used in gravity haulage
- Loaded tubs transported from the top of the incline to the bottom with the help of gravity and simultaneously empty tubs moves from bottom to top of the incline.

6.5.2 Super-Elevation

It depends on the radius of the curve, the speed of the train and the gauge of the track as shown in Figure 6.9 and can be calculated by the following formula

$$\text{Super-elevation} = \frac{A * V^2}{g * R}$$

Where, A = gauge of track in meter
V = velocity of train
R = radius of the curve in meter
g = acceleration due to gravity

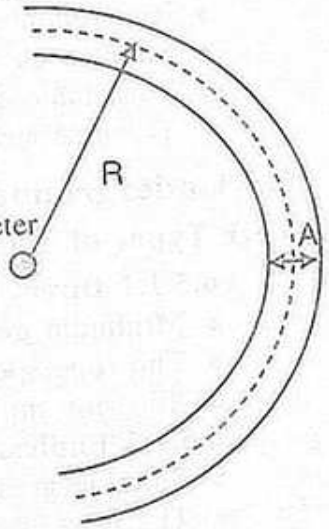


Figure 6.9 : Track gauge and radius

6.5.3 Safety Devices Used in Haulage Arrangement

- Monkey or Back Catch : It is used in the endless haulage of rope breaks
- Stop-block : It is wooden block placed across the track to stop the rail in case of emergency
- Backstay : It is used behind the ascending set of tubs on direct haulage or endless haulage
- Drop Warrick : This is used to prevent the forward movement of tubs and is placed below the brow of the incline and also near the bottom.
- A combined stop-block and runaway switch : This is used at the brow of the direct haulage and is so constructed that at one time only one of them is active. The minimum distance between the two is length of the set of the tubs used (L) plus 4.5 m

6.5.4 Power Requirement of the Motor to Run Haulage

The force diagram of an ascending tub in a haulage system is shown in Figure 6.10.

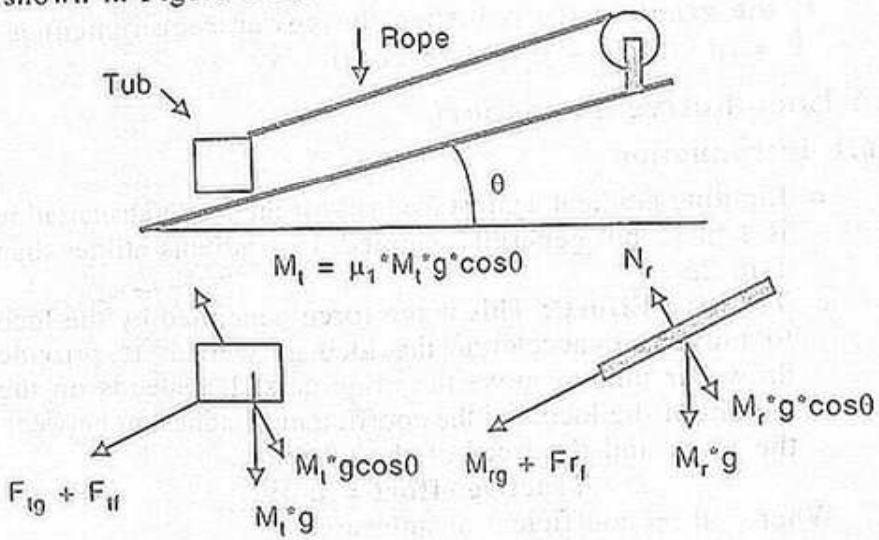


Figure 6.10 : Force diagram of the ascending tubs and rope

Where, F_{tg} = force due to gravity of the loaded tubs along the incline plane
F_{tf} = force due to friction of the loaded tubs along the incline plane
F_{rg} = force due to gravity of the rope along the incline plane
F_{rf} = force due to friction of the rope along the incline plane
θ = angle of inclination

M_t = total mass of the tubs
M_r = total mass of the rope
μ_t = coefficient of friction between the tub and incline surface
μ_r = coefficient of friction between the rope and incline surface

Total force acting along the incline plane (F) = $F_{tg} + F_{tf} + F_{rg} + F_{rf}$

$\Rightarrow F = M_t*g*\sin\theta + \mu_t* M_t*g*\cos\theta + M_r*g*\sin\theta + \mu_r* M_r*g*\cos\theta$

Power requirement to run the haulage (P) = F * V

$= (M_t*g*\sin\theta + \mu_t* M_t*g*\cos\mu + M_r*g*\sin\theta + \mu_r* M_r*g*\cos\theta) * V$

Where, V = velocity of rope

If the gradient (θ) is negative then the power requirement is

$P = (M_t*g*\sin\theta - \mu_t* M_t*g*\cos\theta + M_r*g*\sin\theta - \mu_r* M_r*g*\cos\theta) * V$

If the gradient (θ) is 0 then the power requirement is

$P = (\mu_t* M_t*g + \mu_r* M_r*g*\cos\mu) * V$

6.6 Locomotive Transport

6.6.1 Introduction

- Limiting gradient against the load for the loco transportation is 1 in 15 but generally adopted on gradients milder than 1 in 25
- **Tractive Effort :** This is the force generated by the loco to move and accelerate the loco as well as to provide draw bar pull to move the mine cars. It depends on the weight of the loco and the coefficient of adhesion between the track and the tread of the wheel.

Tractive effort = μ W

Where, μ = coefficient of adhesion
W = weight of the Loco

- **Draw-bar pull :** This is the part of the tractive effort which remains available for pulling the connected load of mine cars after deducting the force required to move the loco itself from the total tractive force of the loco

Draw-bar pull = Tractive Effort – Force required to move the loco

6.6.2 Types of Locomotive

There are three types of locomotives as follows

- **Diesel Locomotives**
  - Locos used in u/g coal mines should have the power unit
  - Weight ranges from 3-15 te
  - Power ranges from 15 -75 kW
  - Maximum permissible CO in exhaust gases is 0.2 %
  - Minimum flash point of diesel oil should be 65°C
- **Electric Battery Locomotive**
  - Less powerful than diesel and trolley wire locomotives
  - Power ranges from 4 -70 kW
  - A fully charged battery gives service for nearly 8 hours
- **Overhead Wire Locomotive/ Trolley Wire Locomotive**
  - D.C current is supplied through the overhead wire to supply power
  - D.C. supply is at 250V
  - Maximum efficiency comparison to the others

6.7 Conveyors Transport

6.7.1 Belt Conveyor

- Belts covered by a layer of PVC instead of rubber are fire proof and only such PVC coated belts is permissible to use in underground coal mines.
- The maximum angle of inclination for conveying coal on PVC belting is 16°
- The belt speed generally preferred 45 m/min to 60 m/min.
- Belt width varies from 0.6 to 1 m.
- Thrustor type brake is used to stop the belt
- Holdbacks are used to prevent the backward movement of the belt in case of a breakdown. This arrangement is provided by ratchet and pawl arrangement

Carrying capacity of a troughed belt conveyor is given by

$T = a * b* V$ (te/sec)

Where, T = carrying capacity

a = average cross sectional area of material in $m^2 = \frac{w^2}{10}$ to $\frac{w^2}{12}$

b = bulk density in te/m^3

V = speed of conveyor belt in m/s

w = belt width

6.7.2 Cable Belt Conveyor

- This type belt only carries the material and rest on the carrier rope situated on both side of the belt
- Suitable for long lengths and in inclination

6.7.3 Disc Conveyors

- These are designed for steep gradient

6.7.4 High Angle Conveyor

- These are designed for steep gradient up to 60°

6.8 Mine Pumps

Before a pump can be selected or a prototype designed, the application must be clearly defined. The common requirement of all applications is to move liquid from one point to another. As pump requirements must match system characteristics, analysis of the overall system is necessary to establish pump conditions. The following will ultimately determine pump selection.

- Capacity range of liquid to be moved
- Differential head required
- Net positive suction head available (NPSHA)
- Shape of head-capacity curve
- Pump speed
- Liquid characteristics
- Construction

The following parameters or terms are frequently used in pump :

- Capacity : Capacity means the flow rate with which liquid is moved or pushed by the pump to the desired point in the process.

The capacity depends on a number of factors like :

- Process liquid characteristics i.e. density, viscosity
- Size of the pump and its inlet and outlet sections
- Impeller size
- Impeller rotational speed RPM
- Size and shape of cavities between the vanes
- Pump suction and discharge temperature and pressure conditions

As liquids are essentially incompressible, the capacity is directly related with the velocity of flow in the suction pipe. This relationship is as follows:

$$Q = A * V$$

Where, Q = capacity in m^3/s

A = area of cross section in m^2

V = velocity in m/s

- Head : The *head* term is used to measure the kinetic energy created by the pump. In other words, head is a measurement of the height of a liquid column that the pump could create from the kinetic energy imparted to the liquid. Imagine a pipe shooting a jet of water straight head. The main reason for using head to measure a centrifugal pump's energy is that the pressure from a pump will change if the specific gravity (weight) of the liquid changes, but the head will not change.

Pressure to Head Conversion formula

The static head corresponding to any specific pressure is dependent upon the weight of the liquid according to the following formula :

$$\text{Head} = \frac{\text{Pressure } (PSi) * 2.31}{\text{Specific Gravity}}$$

The various head terms are discussed below.

- *Static Suction Head* (h_S) : Head resulting from elevation of the liquid relative to the pump centre line. If the liquid level is above pump centreline, h_S is positive. If the liquid level is below pump centreline, h_S is negative. Negative h_S condition is commonly denoted as a "suction lift" condition.
- *Static Discharge Head* (h_d) : It is the vertical distance in feet between the pump centreline and the point of free discharge or the surface of the liquid in the discharge tank.
- *Friction Head* (hf) : The head required to overcome the resistance to flow in the pipe and fittings. It is dependent upon the size, condition and type of pipe, number and type of pipe fittings, flow rate, and nature of the liquid.
- *Vapour Pressure Head* (hvp) : Vapour pressure is the pressure at which a liquid and its vapour co-exist in equilibrium at a given temperature. When the vapour pressure is converted to head, it is referred to as vapour pressure head, hvp. The value of hvp of a liquid increases with the rising temperature and in effect, opposes the pressure on the liquid surface, the positive force that tends to cause liquid flow into the pump suction i.e. it reduces the suction pressure head.
- *Pressure Head* (hp) : Pressure Head must be considered when a pumping system either begins or terminates in a tank which is under some pressure other than atmospheric. The pressure in such a tank must first be converted to feet of liquid. Denoted as hp, pressure head refers to absolute pressure on the surface of the liquid reservoir

supplying the pump suction, converted to feet of head. If the system is open, hp equals atmospheric pressure head.

- *Velocity Head* (hv) : Refers to the energy of a liquid as a result of its motion at some velocity 'v'. It is the equivalent head in feet through which the water would have to fall to acquire the same velocity, or in other words, the head necessary to accelerate the water. The velocity head is usually insignificant and can be ignored in most high head systems. However, it can be a large factor and must be considered in low head systems.
- *Total Suction Head* (H_S) : The suction reservoir pressure head (hp_S) plus the static suction head (h_S) plus the velocity head at the pump suction flange (h_{VS}) minus the friction head in the suction line (hf_S).

$$H_S = hp_S + h_S + hv_S - hf_S$$

The total suction head is the reading of the gauge on the suction flange, converted to feet of liquid.

- *Total Discharge Head* (H_d) : The discharge reservoir pressure head (hp_d) plus static discharge head (h_d) plus the velocity head at the pump discharge flange (hv_d) plus the total friction head in the discharge line (hf_d).

$$H_d = hp_d + h_d + hv_d + hf_d$$

The total discharge head is the reading of a gauge at the discharge flange, converted to feet of liquid.

- **Total Differential Head (H_T)** : It is the total discharge head minus the total suction head or

$$H_T = H_d + H_S \text{ (with a suction lift)}$$
$$H_T = H_d - H_S \text{ (with a suction head)}$$

❖ **Power and Efficiency** : The work performed by a pump is a function of the total head and the weight of the liquid pumped in a given time period.

Pump input or brake horsepower (BHP) is the actual horsepower delivered to the pump shaft.

This can be determined by the following formulas.

$$B.H.P. = \frac{Q * H_T * Sp.Gr.}{3960 * Efficiency}$$

Where, Q = capacity in gallons per minute (gpm)

H_T = total differential head, ft.

Sp. Gr. = Specific gravity of the liquid.

Efficiency = Pump efficiency in %

Pump output or hydraulic or water horsepower (WHP) is

the liquid horsepower delivered by the pump. This can be determined by the following formulas.

$$W.H.P. = \frac{Q * HT * Sp.Gr}{3960}$$

Where, Q = capacity in gallons per minute (gpm)

H_T = total differential head, ft.

Sp. Gr. = Specific gravity of the liquid.

The constant 3960 is obtained by dividing the number or foot-pounds for one horsepower (33,000) by the weight of one gallon of water (8.33 pounds).

❖ **Specific Speed** : Specific speed (N_s) is a non-dimensional design index that identifies the geometric similarity of pumps. It is used to classify pump impellers as to their type and proportions. Pumps of the same Ns but of different size are considered to be geometrically similar, one pump being a size-factor of the other.

The following formula is used to determine specific speed :

$$Ns = \frac{N * Q^{0.5}}{H^{0.75}}$$

Where, Q = capacity at best efficiency point at maximum impeller diameter, gpm.

H = head per stage at BEP at maximum impeller diameter, ft.

N = pump speed, rpm.

❖ **The Affinity Laws** : The Affinity Laws are mathematical expressions that define changes in pump capacity, head, and BHP when a change is made to pump speed, impeller diameter, or both.

According to Affinity Laws :

- Capacity, Q changes in direct proportion to impeller diameter D ratio, or to speed N ratio :

$$\frac{Q_2}{Q_1} = \frac{D_2}{D_1}$$

- Head, H changes in direct proportion to the square of impeller diameter D ratio, or the square of speed N ratio :

$$\frac{H_2}{H_1} = \frac{D_2^2}{D_1^2}$$

$$\frac{H_2}{H_1} = \frac{N_2^2}{N_1^2}$$

- BHP changes in direct proportion to the cube of impeller diameter ratio, or the cube of speed ratio :

$$\frac{BHP_2}{BHP_1} = \frac{D_2^3}{D_1^3}$$

$$\frac{BHP_2}{BHP_1} = \frac{N_2^3}{N_1^3}$$

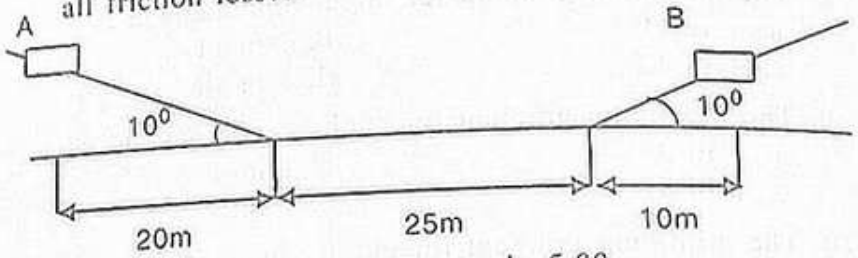
Where the subscript : 1 refers to initial condition, 2 refer to new condition

EXERCISE

1. A spur gear system has two gears of 24 and 33 teeth of module 4. The centre distance in mm between the driving and the driven shafts of the spur gear is
   a. 28.5 b. 57.0
   c. 114.0 d. 228.0
2. The correct sequence of increasing the ductility of the material is
   a. Au- Ag- Fe- Ni-Al-Cu b. Au- Fe-Ag- Al-Ni- Cu
   c. Al- Cu- Ni- Fe- Ag- Au d. Au- Ni- Ag-Cu - Fe- Al
3. Rope used for winding purposes is
   a. Lang's lay type b. Ordinary lay type
   c. Full locked coil rope d. Half locked coil rope
4. The lowest ratio of metallic cross-sectional area to rope cross sectional area occurs in
   a. Round strand rope b. Flattened strand rope
   c. Half locked coil rope d. Full locked coil rope
5. Rope used for haulage transport in underground mine purposes is
   a. Lang's lay type b. Ordinary lay type
   c. Full locked coil rope d. Half locked coil rope
6. A wire rope, round stranded with fibre core, has a diameter of 2.54 cm. What is the mass of the rope per m length, if the value of the constant k is 0.36?
   a. 2.35 b. 2.36
   c. 2.87 d. 3.21
7. A wire rope, round stranded with fibre core, has a diameter

   a. 56 b. 52
   c. 57 d. 58
8. The minimum gradient for direct rope haulage is
   a. 1 in 10 b. 1 in 11
   c. 1 in 12 d. 1 in 14
9. The maximum gradient for endless rope haulage is
   a. 1 in 3 b. 1 in 4
   c. 1 in 5 d. 1 in 6
10. The minimum gradient for gravity haulage is
   a. 1 in 10 b. 1 in 11
   c. 1 in 12 d. 1 in 14
11. The gradient suitable for main and tail rope haulage is
   a. ± 1 in 4 b. ± 1 in 5
   c. ± 1 in 6 d. ± 1 in 7
12. The type of rope haulage suitable for undulating roadways is
   a. direct rope haulage b. endless rope haulage
   c. gravity haulage d. main and tail rope haulage
13. Clifton pulley used in
   a. direct rope haulage b. endless rope haulage
   c. gravity haulage d. main and tail rope haulage
14. Jig pulley used in
   a. direct rope haulage b. endless rope haulage
   c. gravity haulage d. main and tail rope haulage
15. Slip ring motor used in
   a. direct rope haulage b. endless rope haulage
   c. gravity haulage d. main and tail rope haulage
16. Squirrel cage motor used in
   a. direct rope haulage b. endless rope haulage
   c. gravity haulage d. main and tail rope haulage
17. The type of rope suitable for haulage system is
   a. half locked coil b. full locked coil
   c. lang's lay d. ordinary lay
18. In direct rope haulage, the total breaking force of all wires of the rope is 342500 N and the maximum resistive force to motion from all causes is 28406 N. Then the factor of safety against rope breakage will be
   a. 12.06 b. 1.21
   c. 0.08 d. 12.06

19. A mine car is released from rest from the point A. What is the velocity in m/s when the car is at the point B? Neglect all friction losses.

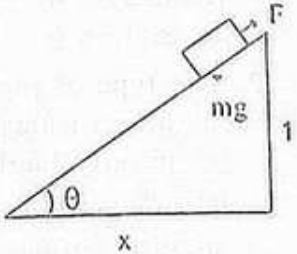


a. 1.76 b. 5.88
c. 1.88 d. 10.88

20. Neglecting the frictional forces, the force F required to prevent the mine car moving down (as shown in the figure) is

a. $mg\left(\frac{1}{1+x^2}\right)$ b. $mg\left(\frac{1}{1+x^2}\right)$

c. $mg\left(\frac{x}{1+x^2}\right)$ d. $mg\left(\frac{x^2}{1+x^2}\right)$



21. For a direct rope haulage installation in a mine drift if the total rope tension is 115 kN, the moment of inertia of the 3 m diameter drum is 9000 kg/m^2, and the speed of the rope is 12 m/s, the total torque of the drum in kNm is

a. 172.5 b. 175.2
c. 176.1 d. 179.1

22. Maximum damage to a haulage rope is caused by

a. fatigue b. wear and tear
c. corrosion d. mine climate

23. The type of rope suitable for haulage system is

a. half locked coil b. full locked coil
c. lang's lay d. ordinary lay

24. The sequence of attachments between the winding rope and the cage is

a. bull chains, triangular plate, capel, detaching hook
b. detaching hook, bull chains, triangular plate, capel
c. triangular plate, bull chains, capel, detaching hook
d. capel, detaching hook, bull chains, triangular plate

[GATE, 2000]

109 kNm. If the rope speed is 7.6 m/s, the motor power required to run the winder is

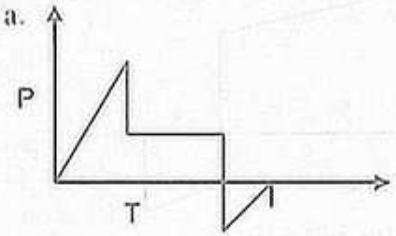
a. 436 kW b. 506 kW
c. 872 kW d. 1574 kW

26. The centre lines of winding drum and winding pulley are parallel and 0.6 m apart. When one cage is at pit bottom, its winding rope is positioned at 1.8 m from the drum centre. The plan distance between the winding drum and the winding pulley axes is 60 m. The fleet angle of installation is

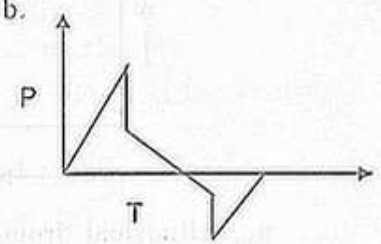
a. 1.15° b. 1.35°
c. 1.42° d. 1.82°

27. Which one of the followings is the power-time diagram for a koepe winder?

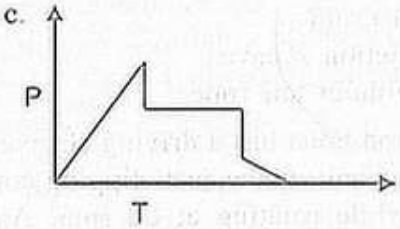
a.



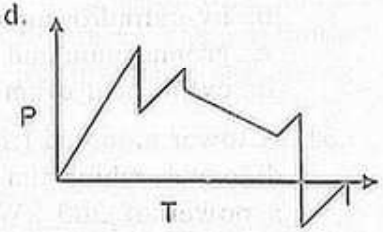
b.



c.



d.

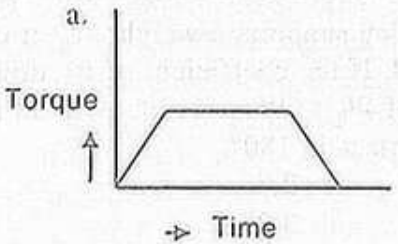


28. For tower mounted friction hoist having a coefficient of friction (μ) between the rope and the sheave of 0.30, the wrap factor $C^{\mu\theta}$ is

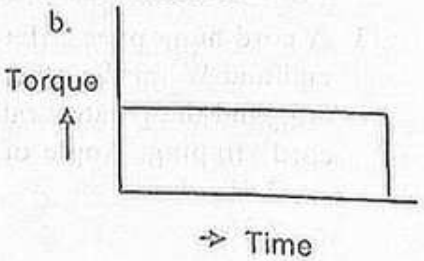
a. 1.60 b. 2.57
c. 4.11 d. None of the above

29. For a standard 3-period torque–time diagram of a mine hoist with constant acceleration and deceleration is

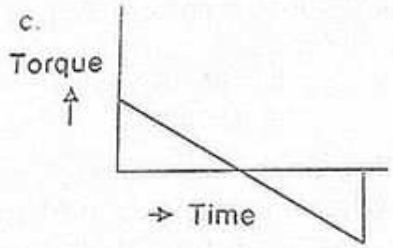
a.



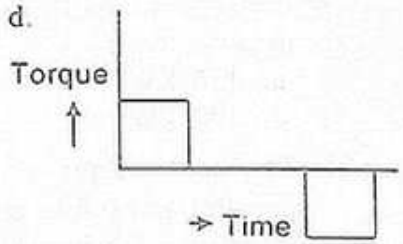
b.



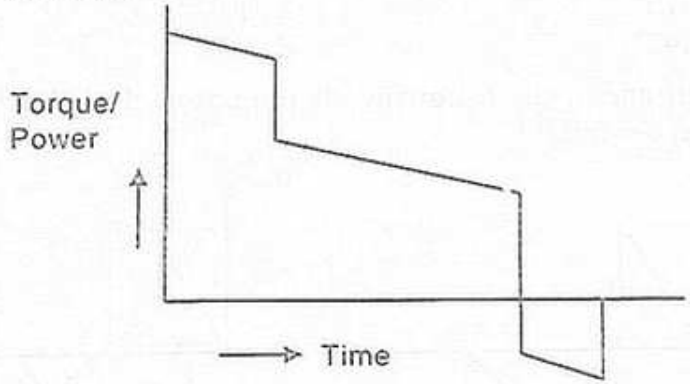
c.



d.



30. The given figure shows the duty cycle of hoist system. The system is



a. cylindrical drum with tail rope
b. by-cylindro conical drum
c. ground mounted friction sheave
d. cylindrical drum without tail rope

31. A tower mounted friction hoist has a driving sheave of 4 m diameter, which can transmit, under just slipping condition, a power of 200 kW while rotating at 60 rpm. Assuming coefficient of friction between rope and rope tread is 0.4, the maximum possible rope tension in kN on the heavier side of the sheave will be

a. 22.3 b. 10.5
c. 1.17 d. 44.5

32. In koepe winding the over wind is prevented by

a. safety hook b. breakage of rope
c. convergence of guides d. thickening of guides

33. A cord hung over a flat pulley supports a weight W_1 at one end and W_2 at the other end. If the coefficient of friction is 0.3, find the greatest ratio of W_1 to W_2 possible without the cord slipping. Angle of contact is 180°.

34. If the r.m.s torque for a winder is 109 kNm, the cylindrical winding drum diameter is 4.4 m and the maximum rope speed is 7 m/s. What will be the minimum power requirement of the motor?

a. 347 kW b. 375 kW
c. 365 kW d. 247 kW

35. A round stranded fibre core winding rope of diameter 2.54 cm has a space factor of 0.5. The tensile strength of the rope steel is 1500 MPa. At a safety factor of 10, what is the maximum load the rope can take?

a. 38 kN b. 76 kN
c. 3.8 MPa d. 7.6 kg/ mm^2

36. The maximum permissible gradient for locomotive is

a. 1 in 12 b. 1 in 14
c. 1 in 16 d. 1 in 15

37. The maximum allowable gradient for adhesion locomotives transportation is

a. 1 in 9 b. 1 in 10
c. 1 in 12 d. 1 in 15

38. An 8t locomotive has a coefficient of adhesion 0.18 and a coefficient of rolling resistance 100 N/te mass. The maximum tractive force available to haul the trailing load in kN along a straight track is

a. 0.64 b. 13.33
c. 14.13 d. 14.93

39. A locomotive haulage train transports 50 te of coal in its out by journey. In the return journey it takes back 50 te of material in by. The ideal gradient for this locomotive haulage system is

a. 1 in 50 in favour of coal movement
b. 1 in 50 in favour of material movement
c. zero
d. none of the above

40. A 10,000 kg locomotive runs at a speed of 3 m/s along a level track. The coefficient of adhesion at the track is 0.23, and the rolling resistance of the locomotive is 90 N per 1000 kg. The maximum power available for hauling the train in kW is

a. 40.7 b. 65.0

41. Maximum permissible gradient for PVC belt conveyor transportation is
a. 1 in 14 b. 1 in 15
c. 1 in 16 d. 1 in 17

42. Maximum permissible gradient for high angle conveyor transportation is
a. 50° b. 60°
c. 70° d. 65°

43. An AFC of length 114 m runs at a speed of 0.535 m/s. The mass of chains and flights is 54 kg/m, and the coefficient of friction between the chain and trough is 0.33. What is the power needed to drive the empty conveyor?
a. 6.94 kW b. 10.68 kW
c. 21.36 kW d. 25.74 kW

44. A conveyor transported the coal at the rate of 220 te/hr is 600 m long moving up a gradient of 1 in 60. If the belt width (W) is 0.75 m, and cross-sectional area of the material is 0.1 W^2, determine the belt speed. The bulk density of coal 0.8 te/m^3.
a. 0.04 m/s b. 0.96 m/s
c. 1.36 m/s d. 1.80 m/s

45. A cable belt conveyor is required to carry coal of bulk density 0.8 t/m^3 at a rate 600 t/hr up an inclined length of 1200 m. The total vertical lift is 300 m. The power required to raise the material against gravity in kW will be
a. 490.5 b. 392.4
c. 120.0 d. 1765.8

46. A locomotive is negotiating a level curved path with radius of curvature 30m. The gauge of the rails is 0.75 m. If the maximum elevation of the track is 1/8 of the track gauge, the maximum turning speed in m/s of the locomotive is
a. 5.1 b. 5.8
c. 6.1 d. 6.5

47. A 12 tonne diesel locomotive of 60 kW is plying in an underground haulage roadway. The coefficient of adhesion is 0.25 and the maximum gear efficiency is 80 %. The speed in m/s at which it will haul a train at its full power is
a. 2.548 b. 2.448
c. 2.038 d. 1.630

48. A 24 t locomotive pulls 300 t of trailing load from a level siding. If antifriction bearings having specific rolling resistance of 100 N/t is fitted on all equipment and the coefficient of adhesion is 0.3, the maximum possible acceleration of the train in m/s^2 will be
a. 0.12 b. 1.30
c. 0.22 d. 1.20

49. In a belt conveyor, the cross-sectional area of material in m^2 can be expresses as $B^2/11$, where B is the width of the belt in m. The belt transport coal of bulk density 1200 kg/m^3 at a rate of 188 kg/s. The speed of the belt is 1.5 m/s and the fill factor is 0.8. The required width of the belt in m is
a. 1.07 b. 1.12
c. 1.20 d. 1.44

50. For a 150 m long chain conveyor, the component of power required to move coal against gravity along an inclination 15° is 5 kW. Hourly capacity of the chain conveyor will be
a. 47.2 te b. 37.8 te
c. 13.1 te d. 3.4 te

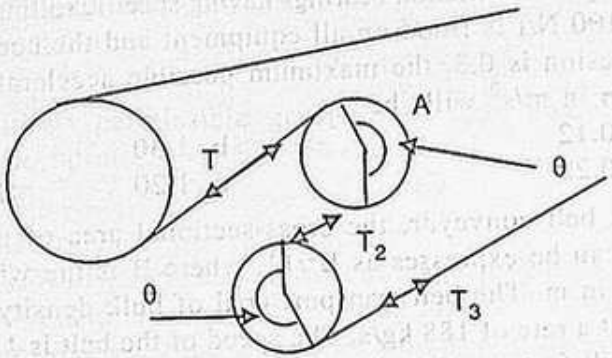
51. A flat steel plate conveyor of width 0.72 m runs at a speed of 0.9 m/s. If the dynamic angle of repose of the material conveyed is 20°, the conveyor theoretical capacity in m^3/ hr is
a. 152 b. 212.3
c. 394 d. 1217

52. For AFC, open-bottom pans are preferable to closed-bottom pans because
a. these don't sink into soft floor being of lighter weight
b. the return chain can not get jammed due to accumulation of coal particles
c. these can be easily inspected and maintained
d. these make less noise

53. In a conveyor belt drive the tension on the tight side is double that on the slack side. If the value of μ is 0.3, find the angle of lap required if the belt is not to slip on its driving drum.
a. 132° b. 135°
c. 136° d. 150°

54. A conveyor belt has two drive drums A and B as shown in the figure. If each angle of wrap 'θ', in both the drive drums

drum is 0.35, the ratio of the belt tensions T_1 and T_3 is



a. 4.1 b. 12.6
c. 8.2 d. 16.6

55. In a cable belt conveyor the function of the cable is to
a. increase the lateral stiffness of the belt
b. increase the tensile strength of the belt
c. support and provide motion to the belt
d. minimize elongation of the belt under tension

56. Function of snub pulley in the belt conveyor system is
a. cleaning device b. safety device
c. braking device d. increasing the angle of wrap

57. Function of rotary brush in the belt conveyor system is
a. cleaning device b. safety device
c. braking device
d. increasing the angle of wrap

58. Function of pull cord in the belt conveyor system is
a. cleaning device
b. safety stopping device
c. material discharging on the side of the belt
d. increasing the angle of wrap

59. Axial thrust in a turbine pump acts
a. from suction side to delivery side
b. from delivery side to suction side
c. perpendicular to pump shaft
d. in the delivery pipe

60. The approximate head generated by a single-stage centrifugal pump of 250 mm diameter impeller running at 1440 rev/min with manometric efficiency of 0.7 is
a. 25 m b. 36 m

61. If a pump develops 10 bars of pressure and has a flow rate of 1500 l/min, what will be the pump motor power at 90 % efficiency?
a. 25 kW b. 28 kW
c. 30 kW d. 45 kW

62. The inflation pressure in truck tyre as recommended by the tyre manufacturer is checked
a. when the truck has worked for 4 hours
b. when it fully loaded by the shovel
c. when it returns to the garage at the end of the shift
d. when it leaves the garage at the start of the shift

63. Lemniscate bar is the component of
a. Road header b. Shield support
c. DERDS d. AFC

64. Cowl is the component of
a. Road header b. Shield support
c. DERDS d. AFC

65. Flight bar is the component of
a. Road header b. Shield support
c. DERDS d. AFC

66. A twin cable ropeway consists of
a. two track ropes and one traction rope
b. one track rope and two traction ropes
c. two track ropes and two traction rope
d. three track ropes and one traction rope

67. The resistance to motion of a dumper on a horizontal track is $(35 + 0.1\ v^2)$ N per tonne, where v is the velocity in m/s. Its tractive effort is 325 N/te. The acceleration in m/s^2 at a velocity of 10 m/s is
a. 0.14 b. 7.14
c. 0.28 d. 28.00

68. The production rate of a bucket wheel excavator is 400 m^3/hr in a material of digging resistance 25 N/ mm. Assuming digging resistance inversely varies with square root of production capacity of the excavator, the production rate in m^3/hr in another material of digging resistance of 45 N/mm will be
a. 1296.0 b. 298.1
c. 222.2 d. 123.5

Answer

1. c	2. c	3. a	4. d	5. a	6. a
7. a	8. c	9. d	10. d	11. c	12. d
13. b	14. c	15. a	16. b	17. c	18. a
19. b	20. b	21. a	22. b	23. c	24. d
25. a	26. a	27. c	28. b	29. b	30. d
31. b	32. c	33. a	34. a	35. a	36. d
37. d	38. b	39. c	40. b	41. c	42. b
43. b	44. c	45. a	46. c	47. a	48. a
49. d	50. a	51. a	52. d	53. a	54. d
55. c	56. d	57. a	58. b	59. b	60. a
61. b	62. d	63. b	64. c	65. d	66. a
67. d	68. d				

□

7.0 Mine System Engineering

7.1 Linear Programming

Linear programming may be classified as the most popular optimization technique used ever in mine system engineering. Its popularity is partly because of the readily available software packages for problem solution apart from its potential for detailed modelling and analysis. There are various methods to solve the problem in linear programming such as graphical method and simplex method but the most commonly used algorithm is the simplex method.

The only significant disadvantage of LP is that, it can only be used for linear optimization models. A linear mathematical function $f(X_1, X_2 \ldots X_n) = a_1X_1 + a_2X_2 + a_3X_3 + \ldots\ldots a_nX_n$

Where, $X_1, X_2 \ldots X_n$, are variables and $a_1, a_2 \ldots a_n$ are constants.

Graphical Method : This is the method to optimize (maximum/ minimum) the solution graphically without any complex calculation.

Example :

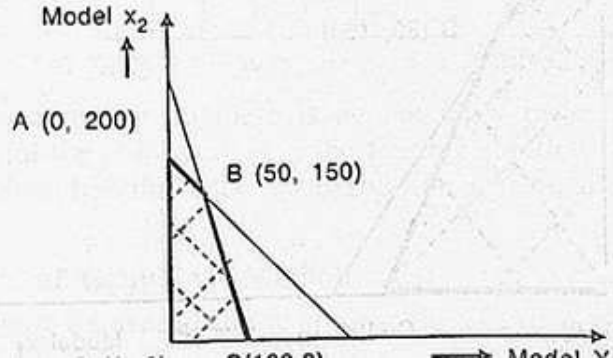
Maximise $Z = 2x_1 + x_2$

Subject to, $3x_1 + x_2 \leqslant 300$

$x_1 + x_2 \leqslant 200$

$x_1, x_2 \geqslant 0$

Let the horizontal axis represents model x_1, and the vertical axis model x_2. Plot all the constraint lines and mark the common segments to determine the feasibility region as has been done in the graph shown.



Any point on the thick line or inside the shaded portion of the above graph will satisfy all the restrictions of the problem. Thus ABCO is the feasibility region carried out all the constraints operating on the objective function.

Point	Feasible solution of variables		Value of objective function z
	x_1	x_2	
A	0	200	200
B	50	150	250
C	100	0	200
O	0	0	0

From the above table we find that the value of objective function z is maximum at B (50, 150), and this is equal to 250. Thus, this is the optimum solution out of the several feasible solutions.

Multiple Solutions : Multiple solutions are alternate solutions to a problem yielding the same optimal value of the objective function.

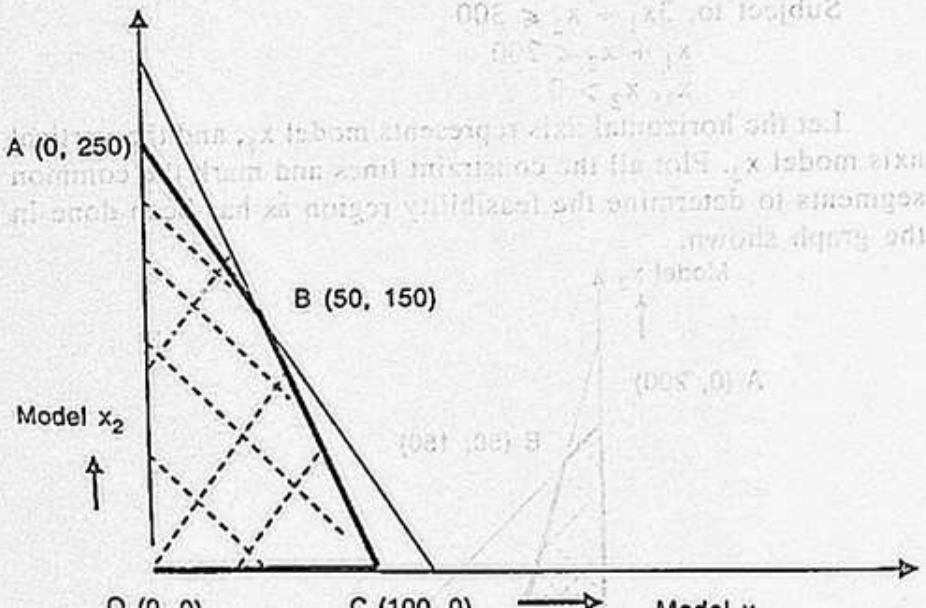
Example :

Maximise $Z = 2x_1 + x_2$

Subject to $3x_1 + x_2 \leqslant 300$

$4x_1 + 2x_2 \leqslant 500$

$x_1, x_2 \geqslant 0$



Point	Feasible solution of variables		Value of objective function z
	x_1	x_2	
A	0	250	250
B	50	150	250
C	100	0	200
O	0	0	0

From the above table we find that the value of objective function z is maximum at two points A (0, 250) & B (50, 150), and this is equal to 250. Thus there are multiple solutions which give optimum value out of the several feasible solutions.

Unbounded Solution

An unbounded solution is one in which the objective function value can be arbitrarily increased without violating the constraints.

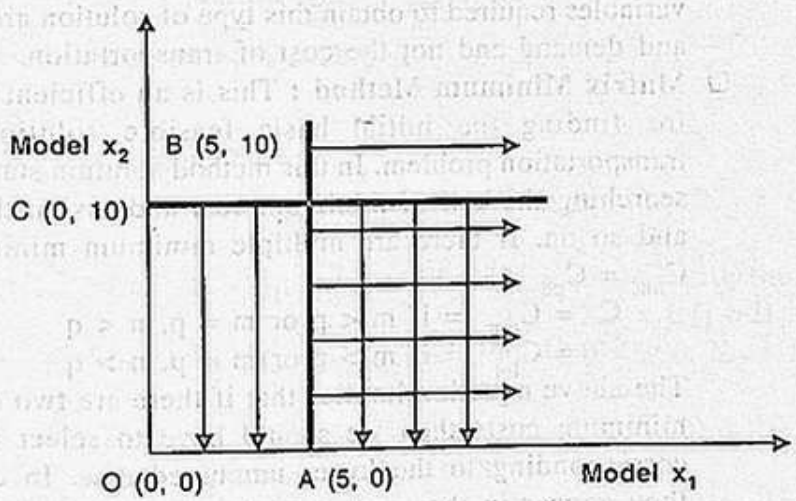
Example :

Maximise $z = x_1 + x_2$

Subject to $x_1 \geqslant 5$

$x_2 \leqslant 10$

$x_1, x_2 \geqslant 0$



In the above graph there is no boundary or intersection points beyond the point B to give the feasible solution. Thus the above problem has the unbounded solution instead of optimal solution.

Limitations of Graphical Method

It cannot be used to solve the problem having more than

7.2 Transportation Model

Transportation model is perhaps the earliest application of linear programming in real life problem.

7.2.1 Types of Transportation Problem

There are two types transportation model namely Balanced Transportation Problem (Supply and Demand are equal) and Unbalanced Transportation Problem (Supply and Demand are unequal).

There are three methods (The North-West Corner Rule, Matrix Minimum Method, and Vogel Approximation Method) for initial basic feasible solution and one optimum solution method (UV method).

7.2.2 Initial Basic Feasible Solution of a Transportation Model

- **The North-West Corner Rule :** This rule is a simple method of obtaining a basic feasible solution of the transportation problem. In this method the procedure for the solution is started from the north-west corner and approaches towards the south-east corner. The only variables required to obtain this type of solution are supply and demand and not the cost of transportation.
- **Matrix Minimum Method :** This is an efficient method for finding the initial basic feasible solution of a transportation problem. In this method solution starts with, searching the cell of minimum cost and assign the units and so on. If there are multiple minimum minima, say $C_{mn} = C_{pq}$

  $C = C_{mn}$ = if $m < p$ or $m = p$, $n < q$

  $= C_{pq}$ = if $m > p$ or $m = p$, $n > q$

  The above equation implies that if there are two or more minimum costs then we should have to select the cell corresponding to the lower numbered row. In case of, they appear in the same row, we should select the cell corresponding to the lower numbered column.
- **Vogel Approximation Method (VAM) :** This method requires a good deal of calculations for finding the initial basic feasible solution of the transportation problem. This method however, usually yields a starting solution which is very close, if not equal, to the optimum solution.

to be transported to three power plants P_1, P_2 and P_3. The cost of transportation of each tonne from mines to the plants along with normal capacities of mines and plants are indicated below :

Mines	Plants			
	P_1	P_2	P_3	a (Capacity of mines)
M_1	25 (X_{11})	17 (X_{12})	25 (X_{11})	400 (a_1)
M_2	15 (X_{21})	10 (X_{22})	18 (X_{23})	300 (a_2)
M_3	16 (X_{31})	13 (X_{32})	16 (X_{33})	400 (a_3)
b (Capacity of plants)	250 (b_1)	350 (b_2)	500 (b_3)	1100

[Note : The above problem is a balanced transportation problem as the sum of the capacity of supply by the mines is equal to the capacity sum of the consumed by the plants.]

Thus, if the equation $\sum_{i=1}^{n} a_i = \sum_{j=1}^{m} b_j$ is holds, problem is balanced transportation problem

And if the equation $\sum_{i=1}^{n} a_i \neq \sum_{j=1}^{m} b_j$ is holds, problem is unbalanced transportation problem]

Solution :

North-West Corner Rule :

Step I : $X_{11} = \min(a_1, b_1) = b_1 = 250$; i.e. if $b_1 = a_1$; $X_{i1} = 0$ [i >1]

Step II : $X_{12} = \min(a_1 - b_1, b_2) = (150, 350) = a_1 - b_1 = 150$; thus $X_{1j} = 0$ [j >2]

Step III : $X_{22} = \min[b_2 - (a_1 - b_1), a_2)] = (200, 300) = 200$; $X_{i2} = 0$ [i > 2]

Step IV : $X_{23} = \min[a_2 - (b_2 - a_1 - b_1), b_3] = (100, 500) = 100$

Step V : $X_{33} = b_3 - a_2 - b_2 - a_1 - b_1 = 400$

Mines	Plants			
	P_1	P_2	P_3	a (Capacity of mines)
M_1	25 (250)	17 (150)	25 (0)	400 (a_1)
M_2	15 (0)	10 (200)	18 (100)	300 (a_2)
M_3	16 (0)	13 (0)	16 (400)	400 (a_3)
b (Capacity	250 (b_1)	350 (b_2)	500 (b_3)	1100

Therefore, Cost of transportation
= 25*250 + 17*150 + 10*200 + 18*100 + 16*400
= 19,000

Matrix Minimum Method :

Minimum cost cell = X_{22} (10)=min (a_2, b_2) = 300; X_{2j} = 0 [j ≠ 2]

Next minimum cost cell = X_{32} (13) = min ($b_2 - a_2$, a_3) = 50; X_{i2} = 0 [i ≠ 3]

The solution will proceed in the same way

Mines	Plants			
	P_1	P_2	P_3	a (Capacity of mines)
M_1	25 (0)	17 (0)	25 (400)	400 (a_1)
M_2	15 (0)	10 (300)	18 (0)	300 (a_2)
M_3	16 (250)	13 (50)	16 (100)	400 (a_3)
b (Capacity of plants)	250 (b_1)	350 (b_2)	500 (b_3)	1100

Therefore, Cost of transportation = 16*250 + 10*300 +13*50 + 25*400 + 16*100
= 19250

Vogel Approximation Method :

Step I :

Mines	Plants				Penalty
	P_1	P_2	P_3	a (Capacity of mines)	
M_1	25 ( )	17 (350)	25 ( )	400 (a_1)	8
M_2	15 ( )	10 (0)	18 ( )	300 (a_2)	5
M_3	16 ( )	13 (0)	16 ( )	400 (a_3)	3
b (Capacity of plants)	250 (b_1)	350 (b_2)	500 (b_3)	1100	
Penalty	1	3	2		

Penalty = Min. cost of any row – Next minimum cost of any row
= Min. cost of any column – Next minimum cost of any column

The maximum penalty found in row 1, i.e. C_{11} - C_{12} and thus assigned the value. In the next step column 2 has been

Step II :

Mines	Plants			Penalty
	P_1	P_3	a (Capacity of mines)	
M_1	25 (0)	25 ( )	50 (a_1)	0
M_2	15 (250)	18 ( )	300 (a_2)	3
M_3	16 (0)	16 ( )	400 (a_3)	0
b (Capacity of plants)	250 (b_1)	500 (b_3)	1100	
Penalty	1	2		

Step III :

Mines	Plants		Penalty
	P_3	a (Capacity of mines)	
M_1	25 (50)	50 (a_1)	
M_2	18 (50)	50 (a_2)	
M_3	16 (400)	400 (a_3)	
b (Capacity of plants)	500 (b_3)	1100	
Penalty	2		

Final Solution :

Mines	Plants			
	P_1	P_2	P_3	a (Capacity of mines)
M_1	25 (0)	17 (350)	25 (50)	400 (a_1)
M_2	15 (250)	10 (0)	18 (50)	300 (a_2)
M_3	16 (0)	13 (0)	16 (400)	400 (a_3)
b (Capacity of plants)	250 (b_1)	350 (b_2)	500 (b_3)	1100

Therefore, Cost of transportation =
15*250 + 17*350 +25*50 + 18*50 + 16*400
= 18250

7.2.3 Optimal Solution of a Transportation Model

UV method :

The ... solution has been derived from

Mines	Plants			
	P_1	P_2	P_3	a (Capacity of mines)
M_1	$X_{11} = 250$ (B.V.)	$X_{12} = 150$ (B.V.)	$X_{13} = 0$ (N.B.V.)	400 (a_1)
M_2	$X_{21} = 0$ (N.B.V.)	$X_{22} = 200$ (B.V.)	$X_{23} = 100$ (B.V.)	300 (a_2)
M_3	$X_{31} = 0$ (N.B.V.)	$X_{32} = 0$ (N.B.V.)	$X_{33} = 400$ (B.V.)	400 (a_3)
b (Capacity of plants)	250 (b_1)	350 (b_2)	500 (b_3)	1100

[B.V. = Basic Variable; N.B.V. = Non-basic Variable]

Therefore, Cost of transportation

$= 25*250 + 17*150 + 10*200 + 18*100 + 16*400$

$= 19{,}000$

Optimal Solution :

Mines	Plants			
	P_1	P_2	P_3	
M_1	$X_{11} = 250$ ($C_{11} = 25$)	$X_{12} = 150$ ($C_{12} = 17$)	$X_{13} = 0$ ($C_{13} = 15$)	U_1
M_2	$X_{21} = 0$ ($C_{21} = 15$)	$X_{22} = 200$ ($C_{22} = 10$)	$X_{23} = 100$ ($C_{23} = 18$)	U_2
M_3	$X_{31} = 0$ ($C_{31} = 16$)	$X_{32} = 0$ ($C_{32} = 13$)	$X_{33} = 400$ ($C_{33} = 16$)	U_3
	V_1	V_2	V_3	

The determination of the entering variable from among the current non-basic variables (those that are not part of the starting basic solution, i.e., value is 0) is done by computing the non-basic coefficients in the objective function, using the method of multiplier.

In this method we associate the multipliers U_i and V_j with row i and column j of the transportation tableau. For each current basic variable x_{ij}, these multipliers satisfy the following equations:

$U_i + V_j = c_{ij}$ for each basic variable x_{ij}

Basic Variable (U, V)	Equation	Solution
X_{11}	$U_1 + V_1 = 25$	Set $U_1 = 0$, $V_1 = 25$
X_{22}	$U_2 + V_2 = 10$	$U_3 = -9$, $V_3 = 25$
X_{23}	$U_2 + V_3 = 18$	
X_{33}	$U_3 + V_3 = 16$	

Next we use U_i and V_j to evaluate the non-basic variables by computing $U_i + V_j - C_{ij}$ for each non-basic x_{ij}

Non-basic Variable	$U_i + V_j - C_{ij}$
X_{13}	$U_1 + V_3 - C_{13} = 10$ [E.V. for Maximum Positive]
X_{21}	$U_2 + V_1 - C_{21} = 3$
X_{31}	$U_3 + V_1 - C_{31} = 0$
X_{32}	$U_3 + V_2 - C_{32} = -5$

$X_{11} = 250$	$X_{12} = 150 - \theta$	$X_{13} = 0 + \theta$
$X_{21} = 0$	$X_{22} = 200 + \theta$	$X_{23} = 100 - \theta$
$X_{31} = 0$	$X_{32} = 0$	$X_{33} = 400$

'θ' can be determined from the constrained derived from the above tableau

$150 - \theta \geq 0$

$100 - \theta \geq 0$

Thus, $\theta = 100$

The new solution after assigning value to entering cell and shifting the same corresponding to that value for balancing the demand and supply as follows :

Mines	Plants			
	P_1	P_2	P_3	
M_1	$X_{11} = 250$ ($C_{11} = 25$)	$X_{12} = 50$ ($C_{12} = 17$)	$X_{13} = 100$ ($C_{13} = 15$)	U_1
M_2	$X_{21} = 0$ ($C_{21} = 15$)	$X_{22} = 300$ ($C_{22} = 10$)	$X_{23} = 0$ ($C_{23} = 18$)	U_2
M_3	$X_{31} = 0$ ($C_{31} = 16$)	$X_{32} = 0$ ($C_{32} = 13$)	$X_{33} = 400$ ($C_{33} = 16$)	U_3
	V_1	V_2	V_3	

The leaving variable is x_{23} as the assigned value becomes zero in the new solution. Thus x_{23} becomes non-basic variable and x_{13} is basic variable.

The new cost of transportation is

$= 250*25 + 50*17 + 100*15 + 300*10 + 400*16 = 18000$

The above iteration continues till the value of all non-basic variables $(U_i + V_j - C_{ij})$ becomes negative.

7.3 Assignment Problem

An assignment problem in the most general form may be stated as follows; given n individual and n jobs, we wish to assign each individual to exactly one job so that each job is covered by exactly one individual, in such a way that the total cost is minimized.

7.3.1 Assignment Problem Solutions

- Transportation model
- Simplex method
- Branch and Bound method
- Complete Enumeration method
- Hungarian assignment method

7.3.2 The Hungarian Method

The assignment problem can be solved with this method as follows :

Step I : For the original cost matrix, identify each row's minimum, and subtract it from all the entries of the row.

Step II : For the matrix resulting from step I, identify each column's minimum, and subtract it from all the entries of the column.

Step III : Identify the optimal solution as the feasible assignment associated with the zero elements of the matrix obtained in step II.

Example : There are three jobs A, B, and C can be assigned to P, Q and R. A single person can finish a single job. The cost of completion of different people for different job is different and is given below.

Job ⇨ / Person ⇩	A	B	C
P	15	10	9
Q	9	15	13
R	10	12	8

Determine the minimum cost to finish all the jobs.

Solution : Let p and q be the minimum costs associated

The row minimums for step I are computed from the original cost matrix as shown below :

Job ⇨ / Person ⇩	A	B	C	Row Minimum
P	15	10	9	$p_1 = 9$
Q	9	15	13	$p_2 = 9$
R	10	12	8	$p_3 = 8$

Next, subtract the row minimum from each respective row to obtain the reduced matrix is given below :

Job ⇨ / Person ⇩	A	B	C
P	6	1	0
Q	0	6	4
R	2	4	0
Column Minimum	$q_1 = 0$	$q_2 = 1$	$q_3 = 0$

The application of step II yields the column minimum. Subtracting these values from the respective columns, we get the reduced matrix as given below :

Job ⇨ / Person ⇩	A	B	C
P	6	$\underline{0}$	0
Q	$\underline{0}$	5	4
R	2	3	$\underline{0}$

The cells with underscored zero entries provide the optimum solution.

Thus the total cost to finish the jobs = 9 + 10 + 8 = 27

Example 2 :

Person \ Job	A	B	C	D
P	1	4	6	3
Q	9	7	10	9
R	4	5	11	7
S	8	7	8	5

Step I :

Person \ Job	A	B	C	D	Row Minimum
P	1	4	6	3	$p_1 = 1$
Q	9	7	10	9	$p_2 = 7$
R	4	5	11	7	$p_3 = 4$
S	8	7	8	5	$p_4 = 5$

Step II :

Person \ Job	A	B	C	D
P	0	3	5	2
Q	2	0	3	2
R	0	1	7	3
S	3	2	3	0
Column Minimum	$q_1 = 0$	$q_2 = 0$	$q_3 = 3$	$q_4 = 0$

Step III :

Person \ Job	A	B	C	D
P	0	3	2	2
Q	2	0	0	2
R	0	1	4	3

In the above tableau, the locations of the zero entries do not allow assigning unique chores to all the persons.

Step IV : Draw the minimum number of horizontal and vertical lines in the last reduced matrix that will cover all the zero entries.

Person \ Job	A	B	C	D
P	0	3	2	2
Q	2	0	0	2
R	0	1	4	3
S	3	2	0	0

Step V : Select the smallest uncovered entry, subtract it from every uncovered entry, and then add it to every entry at the intersection of two lines.

Person \ Job	A	B	C	D
P	0	2	1	1
Q	3	0	0	2
R	0	0	3	2
S	4	2	0	0

Thus the optimal cost of finishing the job = 1 + 5 + 10 + 5 = 21

Step VI : Still, if no feasible assignment can be found among the resulting zero entries, repeat step IV.

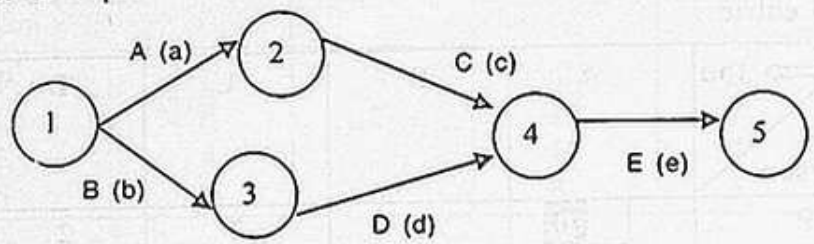
7.4 Network Analysis in Project Planning

7.4.1 Introduction

Network planning consists of arranging the precedence and sequence of project tasks appropriately to provide the road map of the project execution. It starts with the construction of a diagram, known as network diagram that reflects the interdependencies and time requirements of the individual tasks that constitute the project. The individual activities or tasks have the two fundamental

- An event only occurs if and only if all the activities leading into it are complete
- No activity can start until its tail event has occurred

For example



In the above network,

1, 2, 3, 4 and 5 represents the nodes

A, B, C, D and E represents the 5 activities or tasks

a, b, c, d and e represents the time to complete the individuals tasks A, B, C, D and E respectively

Thus here the activity E cannot start until both the activities (C and D) leading into it complete.

7.4.2 Methods of Determining the Duration of a Particular Project

7.4.2.1 Critical Path Method (CPM) or Critical Path Analysis (CPA) : Since all the activities directed to the events must be completed before it occurs, the time at which an event occurs is the maximum duration of the paths directed to the event. Thus, in a complete network of the project, the path having the maximum of the elapsed times from the origin to the terminal node called the critical path and the activities on it are called critical activities. It is possible to have a multiple critical paths if there is more than one path with the longest duration.

CPM is based on the deterministic approach where the durations of the various activities are assumed to be known with certainty.

In the above figure, if a = 5 months; b = 6 months; c = 8 months; d = 4 months and e = 5 months then

Duration of the project along the path 1245 = a + c + e = 18 months

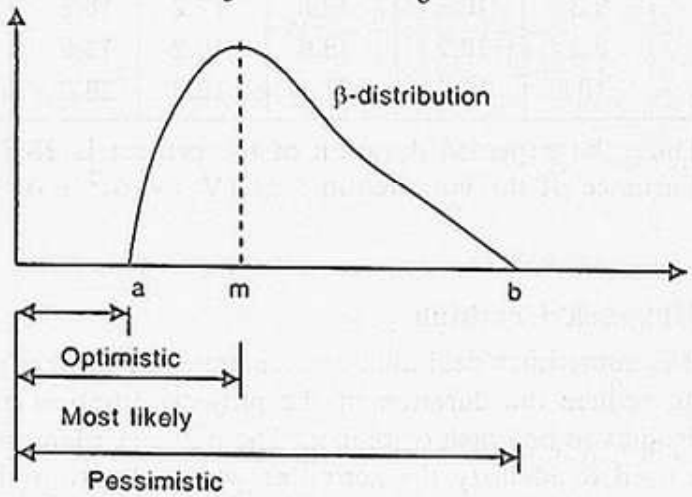
And, duration of the project along the path 1345 = b + d + e = 15 months

Therefore, the longest duration path is 1245 and this path is called the critical path.

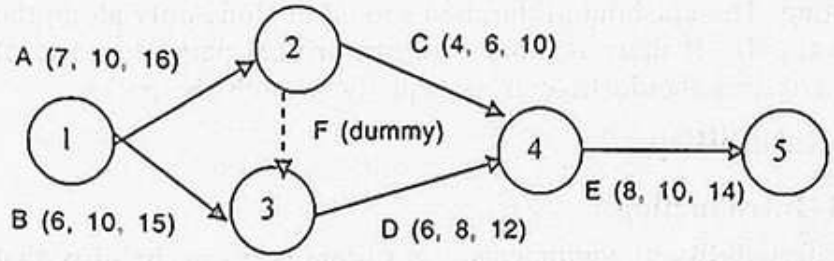
: For a variety of reasons, durations of some of the activities are non-deterministic. It is thus, necessary to use a probability distribution to represent the stochastic nature of the activity duration. It is assumed that this probability distribution is continuous, uni-modal with two non-negative abscissa intercepts. These intercepts denoted by 'a' and 'b' (a < b) are the minimum and maximum time necessary to perform the job. In the context of PERT calculation 'a' is referred to as the optimistic time and 'b' is known as the pessimistic time. In addition, the modal value m is commonly referred to as the most likely time also used in PERT calculation.

The statistical distribution which ideally suits these assumptions is known as a **β-distribution**. The expected duration (t_e) of an activity and the standard deviation of the duration of an activity σ can be approximated by

$$t_e = \frac{a+4m+b}{6}; \quad \sigma = \frac{b-a}{6}$$



For example



Computation of expected duration and standard deviation :

Activity	$t_e = (a + 4m + b)/6$	$\sigma = (b - a)/6$
A*	10.5	1.5
B	10.2	1.5
C	6.3	1.0
D*	8.3	1.0
E*	10.3	1.5

Computation of expected duration of the project :

Activity	t_e	Earliest start	Earliest completion time	Latest completion	Latest start time	Slack
A*	10.5	0	10.5	0	10.5	0
F*	0	10.5	10.5	10.5	10.5	0
B	10.2	0	10.2	0.3	10.5	0.3
C	6.3	10.5	16.8	12.2	18.5	1.7
D*	8.3	10.2	18.5	10.2	18.5	0
E*	10.3	16.8	27.1	18.5	28.8	0

Thus, the expected duration of the project is 28.8 days

Variance of the completion time $(V_T) = \sigma_1^2 + \sigma_2^2 + \sigma_3^2 + \ldots \sigma_n^2$

$= 8.75$

7.4.3 Network Crashing

It is sometimes desirable to accelerate the project progress. Thus, to reduce the duration of the project, duration of certain activities has to be crash or reduce. The network planning device can be used to identify the activities whose duration should be shortened so that the completion time of the project can be shortened in the most economic manner. The process is known as network crashing. The crashing of duration should be done only along the critical path. If there is more than one critical path, duration of the activities should be crashed equally in both the paths.

7.5 Reliability

7.5.1 Introduction

Reliability of equipments or a system is the probability that it will function properly without failure for a specified length of respect to time horizon and combines the time dimension with the performance level of equipment. It is defined formally as the probability of a device performing its purpose adequately for the period of time intended under the operating conditions encountered.

7.5.2 Statistical Concepts of Reliability

Let the life of a component be denoted by the random variable t with distribution function F(t) and density function f(t). In the context of reliability, F(t) and f(t) are also referred to as failure distribution.

The reliability R(t) is given by

$$R(t) = 1 - F(t)$$

Clearly, R(t) indicates the probability that the component will work at least up to time 't' thereby adding the time dimension to the definition.

Probability that a component of age t will fail in the interval t and (t + dt)

$$= h(t) = \frac{f(t)}{R(t)} = \frac{f(t)}{1 - F(t)}$$

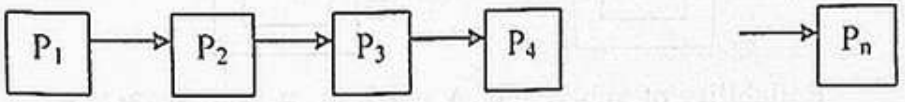
If the failure distribution is exponential with parameter λ, then

$$h(t) = \frac{\lambda e^{-\mu}}{e^{-\mu}} = \lambda$$

7.5.3 Reliability of a System

Every system consists of a number of components. The reliability of the entire system is, obviously, a function of the reliability of its components.

- **Reliability of a System where Components are in Series Arrangement**

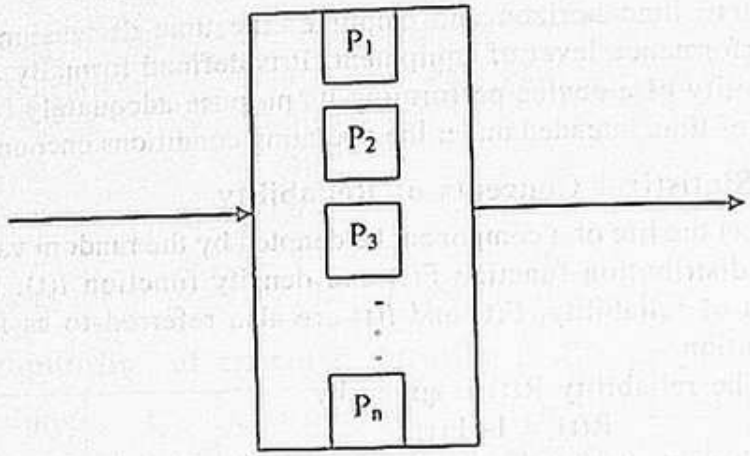


If $P_1, P_2, P_3, P_4 \ldots P_n$ are the reliabilities (probabilities of successful operation) of the n number of components in the system

Then, the Reliability of the system can be obtained by

$$R = P_1 * P_2 * P_3 * P_4 * \ldots P_n$$

- **Reliability of a System where Components are in Parallel Arrangement :**



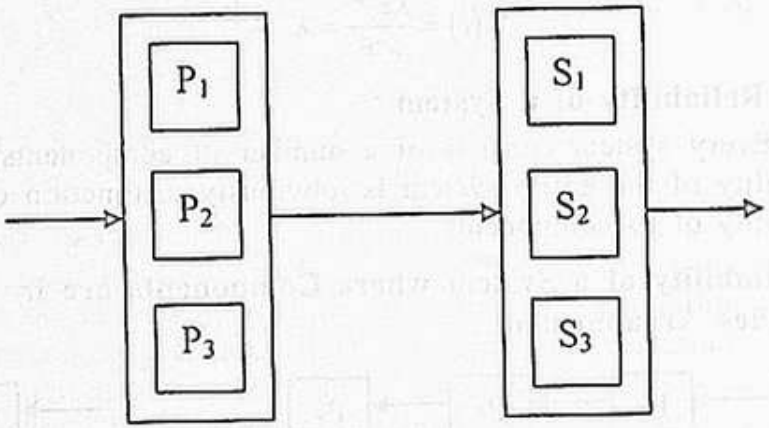
If P_1, P_2, P_3 ...P_n are the reliabilities (probabilities of successful operation) of the n number of components in the system

Then, the Reliability of the system can be obtained by

$R = 1 - (1 - P_1)^* (1 - P_2)^* (1 - P_3)^* \ldots\ldots (1 - P_n)$

❑ Reliability of a System where Components are in Combined Series and Parallel Arrangement

In a system there are two sub-systems A and B. There are three components in both the sub-systems whose individual reliability are shown in the figure below.



Reliability of sub-system A = $1 - (1-P_1)^*(1-P_2)^*(1-P_3)$

Reliability of sub-system B = $1 - (1-S_1)^*(1-S_2)^*(1-S_3)$

Reliability of system = $[1 - (1-P_1)^*(1-P_2)^*(1-P_3)] * [1 - (1-S_1)^*(1-S_2)^*(1-S_3)]$

7.5.4 Mean Time Between Failure (MTBF)

Suppose that, an equipment or a system is in operation. fails it is repaired and the repair time is negligible. Then the average failure time E(t) is identical with the average time between successive failures. This is referred to Mean Time Between Failure (MTBF).

$$MTBF = E(t) = \int_0^\infty [1 - F(t)]dt = \int_0^\infty R(t)dt$$

When the failure distribution is exponential, MTBF = $1/\lambda$

Thus, the reliability or the probability that the equipment run up to time t for exponential distribution failure rate = $e^{-\lambda t}$ = $e^{-t/m}$

Where, m = $1/\lambda$ = mean time between failure

If a Number of Components are in Series

Thus, the reliability or the probability that the system run up to time t for exponential distribution rate

$$= e^{-\lambda_1 t} * e^{-\lambda_2 t} * \ldots\ldots e^{-\lambda_n t} = e^{-t/m} = e^{-(\lambda_1 + \lambda_2 + \ldots \lambda n)t}$$

Then, the system MTBF = $m = \dfrac{1}{\lambda_1 + \lambda_2 + \ldots \lambda_n}$

If a Number of Components are in Parallel

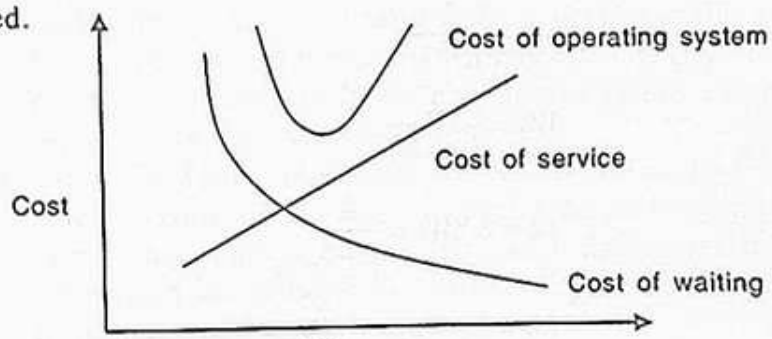
Thus, the reliability or the probability that the system run up to time t for exponential distribution failure rate

$$= 1 - \left(1 - e^{-\lambda_1 t}\right) * \left(1 - e^{-\lambda_2 t}\right) * \ldots\ldots \left(1 - e^{-\lambda_n t}\right)$$

7.6 Waiting Line Problem

7.6.1 Introduction

This problem is applicable to optimize the combination of various machineries in a particular operation like number of dumper required for a single shovel so that the maximum utilization factor of the machineries can be achieved and the overall operating cost is reduced.



7.6.2 Characteristics of Queuing System

The process is generally stochastic and there are three components in a queuing system

- Arrival : Nature of arrival of the machineries
- Service : Nature or capacity of service provided by the machine
- Server : Nature of service facility

The following symbols will be used while studying waiting line problem :

n = number of units in the system who are waiting for service and who are being served.

Pn (t) = transient state probabilities of exactly n customers in the system at time t assuming that the system has started its operation at time zero.

Pn = steady state probability of having exactly n customers in the system.

λ = mean effective arrival rate (number of customers arriving per unit time)

μ = mean service rate per busy server (number of customers served per unit time)

c = number of parallel servers

Ws = expected waiting time per customer in the system

Wq = expected waiting time per customer in the queue

Ls = expected number of customers in the system

Lq = expected number of customers in the queue

ρ = proportion of time that the server is busy or traffic density

Under general conditions of arrival, departure and service discipline

$$\rho = \frac{\lambda}{\mu}$$

$$Ws = \frac{1}{\mu - \lambda}$$

$$Wq = \frac{\lambda}{(\mu - \lambda)}$$

$$Ls = \lambda * Ws = \frac{\lambda}{\mu - \lambda}$$

$$Lq = \lambda * Wq = \frac{\lambda^2}{}$$

7.7 Work Study

7.7.1 Introduction

Work study is a tool or technique of management involving analytical study of a job with a view to determine what exactly has to be done and the measurement of the work content of the job. Work study leads to the elimination of all wasteful, unnecessary and ineffective efforts; it establishes the best way of doing the given job by removing all unsatisfactory working conditions and standardizes the work method.

7.7.2 Basic Techniques of Work Study

7.7.2.1 Method Study : Method study is concerned with the way in which the work is performed. It may be defined as the systematic recording, analysis and critical examination of existing and proposed ways of doing work and is generally concerned with the development of easier and more effective methods.

Objectives of Method Study

- The improvement of procedures so that the time involved in doing a work may be reduced
- The improvement of work place layout
- The reduction of work fatigue
- The development of improved physical working environment

7.7.2.2 Work Measurement : Work measurement is simply a means of establishing the standard time for doing a given job. Work measurement becomes necessary to measure the performance of persons in order to compare output per person. This is important especially when a system of incentives payment has to be worked out. Work measurement is generally attempted through the following two types of studies:

- ❑ **Motion Study** : Motion study is the study of movements in parts which are involved in doing a job. This type of study eliminates the wasteful movements and ensures only the performance of necessary movements. The significance of motion study lies in the fact that it raises employee productivity and reduces fatigue.
- ❑ **Time Study** : Time study refers to the analysis and determination of the time necessary to perform a given task. Time study is a scientific and reliable method and provides a sound basis for introducing incentive scheme in mining industry.

- **Job time study** : This study is applied to set an allowed time for each piece of work turned out at a given standardized operation wherein the parts made are identical.
- **Operation time study** : This is one that applied to establish allowed time for an operation wherein the parts made vary in size and composition.
- **Production time study** : It is an all day study of a given task for the purpose of acquiring detailed job information.

EXERCISE

1. A workshop has four machines for assigning four jobs. Each job can be assigned to only one machine. Estimated processing cost (in Rs.) at each machine to complete each task is given in the table below

		Machines			
		M1	M2	M3	M4
Tasks	T1	61	92	52	72
	T2	42	49	69	85
	T3	47	59	80	71
	T4	65	70	68	72

The minimum value of total processing cost based on an optimal assignment schedules is

a. 214 b. 220
c. 215 d. 262

2. A company has three mines at locations A, B and C, which supply coal to power plants at D, E, F and G. Mines A, B, and C produce 1800 t/day, 1700 t/day, and 2900 t/day respectively. Daily raw coal requirements at the power plants D, E, F, and G from these mines are 2050 t, 1600 t, 1300 t, and 1450 t respectively. Unit transportation costs are as follows :

Mine	Power Plant			
	D	E	F	G
A	3.5	4.0	2.5	5.5
B	4.5	3.7	6.0	4.7
C	4.2	3.8	5.2	2.9

a. an unbalanced transportation problem
b. a balanced transportation problem
c. an assignment problem
d. none of the above

[GATE, 02]

3. Which one of the following operation research problems is solved by the Branch and Bound algorithm?

a. transportation problem b. assignment problem
c. integer programming problem
d. queuing problem

4. A mining machinery company has four plants at locations A, B, C, and D and three warehouses at destinations X, Y and Z. The number of machine units available at the plants A, B, C and D are 5, 8, 7 and 14 respectively. The demands at X, Y and Z are 7, 9 and 18 respectively. The unit costs of transportation in rupees as follows :

		Destination		
		X	Y	Z
Location	A	20	70	40
	B	30	30	10
	C	50	40	70
	D	10	60	20

The total cost of transportation in rupees based on the initial basic feasible solution obtained by the north-west corner rule is

a. 760 b. 800
c. 1020 d. 1110

5. In the above question, the total number of basic variables obtained based on the initial basic feasible solution is

a. 3 b. 4 c. 6 d. 7

[GATE, 06]

6. Three jobs A, B, and C are to be assigned to three machines X, Y and Z. The processing costs are given below :

		Machine		
		X	Y	Z
Job	A	19	28	31
	B	11	17	31
	C	12	15	13

The minimum total cost of assigning the jobs to the machines is

a. 60 b. 54
c. 51 d. 49

[GATE, 07]

7. In PERT analysis, the expected time to complete an activity is computed from the three time estimates: a most likely time (m), an optimistic time (a), and pessimistic time (b) using the expression

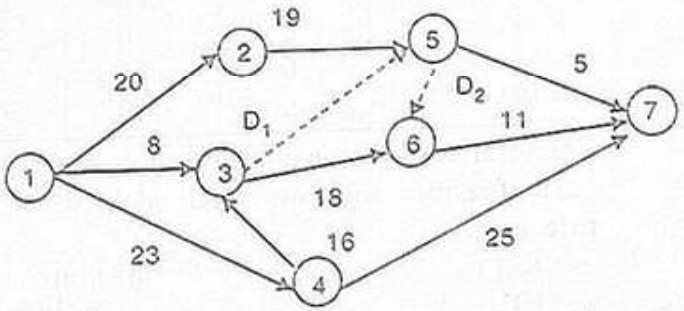
a. $(a + 4m + b)/6$ b. $(4a + m + b)/6$
c. $(a + m + 4b)/6$ d. $(a + m + b)/3$

8. The probabilistic nature of activity times in PERT network is described by which one of the following distributions?

a. normal b. exponential
c. beta d. uniform

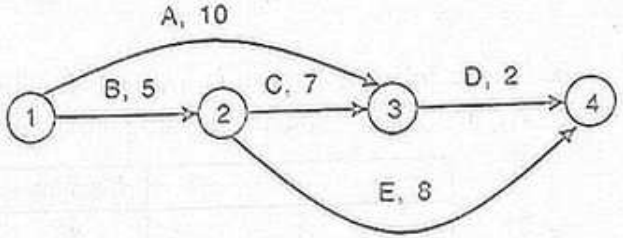
9. The following project network is given with arrows as activities and the numbers by each arrow representing job durations in days. The network has two dummy activities D_1 and D_2. The minimum completion time of the project in days is

a. 13
b. 19
c. 75
d. 68



10. In the project network shown activity durations are given in months. If the activity can be crashed to expedite the project, which one would you crash and by how many months?

a. C, 1
b. C, 2
c. D, 1
d. E, 1



11. The three time estimates of PERT network are assumed to follow

a. exponential b. normal

12. Activity durations in PERT and CPM are decided on the following consideration

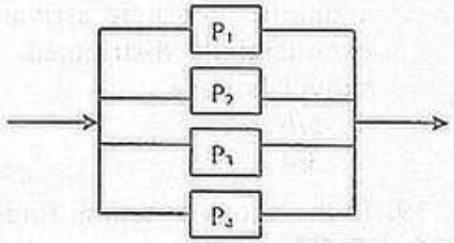
a. PERT – stochastic CPM - deterministic
b. PERT – stochastic CPM – stochastic
c. PERT – deterministicCPM – deterministic
d. PERT – deterministicCPM – stochastic

13. In an opencast mine with shovel-dumper combination have 1 shovel and 6 dumpers. The reliability of the shovel is 0.9 and all the dumpers have equal reliability of 0.8. What is the system reliability?

a. 0.658 b. 0.576
c. 0.720 d. 0.160

14. A pumping station in a mine has 4 pumps, P_1, P_2, P_3 and P_4 which are connected functionally in parallel configuration as shown in the figure. The individual reliability of the pumps P_1, P_2, P_3 and P_4 are 0.80, 0.90, 0.85 and 0.95 respectively. For an effective dewatering operation in the mine denoting system success, at least three pumps should be in operation. The reliability of the system is

a. 0.19
b. 0.93
c. 0.99
d. 0.58



[GATE, 04]

15. The lifetime of a shovel is exponentially distributed with a mean life of 10 years. If the shovel has worked for 10 years in an opencast mine, the probability that it will be working for an additional 10 years is

a. 0.003 b. 0.135
c. 0.367 d. 0.500

[GATE, 06]

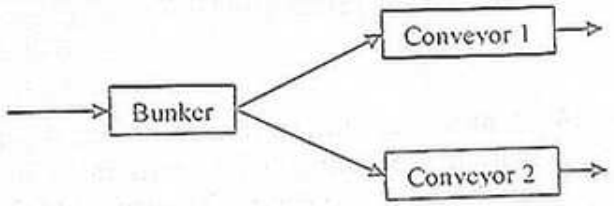
16. The failure data analysis of a longwall mining system consisting of three major subsystems revealed that the failure rate of shearer, face conveyor and stage loader are 0.15/ hour, 0.2/ hour and 0.1/ hour respectively. If all the subsystems must be running for a satisfactory system operation, the reliability of the longwall system for a period of 4 hours is

a. 0.165 b. 0.259
c. 0.405 d. 0.630

[GATE, 06]

17. A bunker-conveyor system as shown in the figure works under the following scheme:
   i. Bunker and two conveyors running
   ii. Bunker and either of the two conveyors running

   If the bunker and the two conveyors have identical reliability of 0.9, the system reliability is
   a. 0.729
   b. 0.810
   c. 0.999
   d. 0.891



18. In an opencast mine loading and transporting has been carried out by shovel dumper combination. There is only one shovel is in operation. Dumpers arrive at the rate of 10 dumpers per hour and it takes shovel on an average 5 minutes to load a dumper. Assume arrival is Poisson and service time is exponentially distributed. Find the proportion of time the shovel is busy.
   a. 5/6 b. 7/8
   c. 3/4 d. 1/2

19. In the above question, find the probability that the shovel is sitting idle.
   a. 1/6 b. 1/8
   c. 1/4 d. 1/2

20. In the above question, find the expected number of dumpers in the system.
   a. 2 b. 3
   c. 4 d. 5

21. In the above question, find the expected number of dumpers waiting to be load in the system.
   a. 100/24 b. 100/12
   c. 2 d. 4

22. In the above question what is expected waiting time of the dumper in the queue?
   a. 5/7 hour b. 5/ 12 hour
   c. 1/2 hour

23. In the above question what is expected idle time of a dumper in the system?
   a. 30 min b. 60 min
   c. 45 min d. 15 min

24. In an opencast mine, dumpers arrive at a shovel location with an average arrival rate of 12 per/ hour. The inter-arrival times of the dumpers follow an exponential distribution. The service times of the shovel also follow an exponential distribution with an average of 3 minute. The average waiting time of a dumper in the queue in minutes is
   a. 2.70 b. 6.00
   c. 4.50 d. 7.50

[GATE, 05]

25. In industrial work study, work sampling is a technique of
   a. method study b. work measurement
   c. time study d. motion study

[GATE, 05]

Answer

1. b	2. b	3. b	4. c	5. c	6. d
7. a	8. c	9. d	10. a	11. c	12. a
13. b	14. b	15. b	16. a	17. d	18. a
19. a	20. d	21. a	22. b	23. a	24. c
25.b					

□

8.0 Mine Economics

8.1 Appraisal Techniques

There are several appraisal techniques for judging the profitability of new investments in assets. They may be classified as Non-discounting and Discounting techniques. These are as follows

- Non-discounting techniques :
  - Payback period
  - Average rate of return (ARR)
- Discounting techniques
  - Net Present Value (NPV)
  - Profitability Index (P.I)
  - Net Terminal Value (NTV)
  - Internal Rate of Return (IRR)

8.2 Determination of Future Worth

- **For Annual Compound Interest Rate 'i' :**

Future Worth = FW = PV $(1 + i)^n$

Where, FW = Future worth after n^{th} year

i = interest rate compounded annually

n = number of years

- **For Half Yearly compound Interest Rate 'i'**

Future Worth = FW = PV $(1 + i/2)^{2n}$

Where, FW = Future worth after nth year

i = interest rate compounded half yearly

n = number of years

- **For Equal Yearly Payments (annuity) :**

$$FW = A * \left[\frac{(1+i)^n - 1}{i} \right]$$

Where, FW = future worth after n^{th} year

i = interest rate compounded annually

n = number of years

A = yearly payments

8.3 Determination of Present Value

Present value of equal yearly payments (annuity) can be

$$\text{Present Value} = PV = A * \left[\frac{(1+i)^n - 1}{i(1+i)} \right]$$

Where, i = interest rate compounded annually

n = no. of years.

A = annual payment

8.4 Payback period

Time required to repaying the loan with equal yearly payments (annuity) called the pay back period. The loan has been rapid when the net present value (NPV) is equal to zero.

$$NPV = -X + A * \left[\frac{(1+i)^n - 1}{i(1+i)^n} \right] = 0$$

Where, X = Loan amount

A = annual payment

n = payback period

i = interest rate compounded annually

8.5 Rate of Return on an investment

The interest rate 'i' which makes the future worth equal [Case 1 : Rs. X is invested for n number of years, Case 2 : Annual profit after tax of Rs. Y will be generated through initial investment of Rs. X] is called the rate of return (ROR) on an investment.

For Case 1 : $FW = X\ (1 + i)^n$

For Case 2 : $FW = Y \left[\frac{(1+i)^n - 1}{i} \right]$

In this case, $X(1+i)^n = Y \left[\frac{(1+i)^n - 1}{i} \right]$

Where, i is rate of return on an investment

8.6 Discounted Cash Flow (DCF)

This is the present value of each of the invididual cash flows. The net present value assuming a minimum acceptable discount rate of i% is

$$NPV = \frac{-X_0}{(1+i)^0} + \frac{X_1}{(1+i)^1} + \frac{X_2}{(1+i)^2} + \frac{X}{(1+i)^3} + \ldots\ldots \frac{X_n}{(1+i)^n}$$

Where, $X_0, X_1, X_2, X_3 \ldots X_n$ are the cash flows in the year

8.7 Discounted Cash Flow Rate of Return (DCFROR) or Internal Rate of Return (IRR)

Discounted cash flow rate of return (DCFROR) or internal rate of return (IRR) may be defined as the discount rate that equates present value of the project's future net cash flows with the projects initial cash outlay. In other words we can say that the net present value at the end of the project will be equal to zero.

Thus in the above equation if i is the DCFROR or IRR then NPV = 0.

8.8 Profitability Index

The profitability index (also known as benefit/cost ratio) is the ratio of the present value of the future net cash flows to the initial outlay of the project. This index provides a relative measure for judging desirability and evaluating the worth of an investment proposal. It can written as

$$PI = \frac{\sum_{i=1}^{n} \frac{CFi}{(1+r)^i}}{I_0}$$

Where, CF_i = net cash flow in time period i

r = rate of discount

n = the project's expected life

I_o = initial cash outlay

The accept-reject criterion in case of P.I. is as under:

Accept the project if it's P.I. > 1

Reject the project if its P.I < 1

8.9 Depreciation

The amount of wear and tear (of a property etc.) for which a reduction may be made in a valuation. Standard straight line depreciation yields a yearly depreciation value of

$$\text{Yearly Depreciation value} = \frac{\text{Investment} - \text{Salvage Value after completion of the project}}{\text{Life of the project}}$$

8.10 Redemption

Recover the capital by expenditure of effort or by a stipulated payment.

8.11 Depletion

When natural resources are being consumed in producing decrease in value of the resource base that has occurred. The term is commonly used in connection with non-renewable resources like mineral, oil etc. The basic difference between the depreciation and depletion is that in the case of depreciation the property involved usually may be replaced with similar property when it has become fully depreciated whereas in the case of depletion of mineral resources such replacement usually is not possible.

There are two ways to compute depletion allowances:

- The cost method
- The percentage method

EXERCISE

1. What will be the effective rate of interest for nominal rate of interest 10 %, when money is compounded quarterly?
   a. 10.381 % b. 09.381 %
   c. 11.381 % d. 12.381 %

2. The cash flow of a project is given as follows :

Year	0	1	2	3	4
Cash flow (Rs.)	-50000	15000	15000	20000	22500

   The internal rate of return (IRR) in percent lies between
   a. 15 and 16 b. 14 and 15
   c. 16 and 17 d. 13 and 14

   [GATE 2002]

3. The present value of an annuity of Rs. 1000 per year for 8 years with the redemption of capital at market interest rate of 5 % and with risk rate of 12 % is
   a. Rs. 5550.3 b. Rs. 4554.9
   c. Rs. 4207.1 d. Rs. 4449.9

4. A firm, having a cost of capital of 12 %, evaluates a small mining project of initial investment of Rs. 100 lakhs. For the benefits shown in the table below, the benefit-cost ratio of the project is

Year	1	2	3	4
Benefits (Rs. In lakhs)	25	40	40	50

   a. 0.985 b. 1.145 c. 1.384 d. 1.550

   [GATE 2005]

5. At a discount rate of 12 %, uniform annual savings producing Rs. 6 lakhs at the end of 10 years is

a. Rs. 105740 b. Rs. 52850
c. Rs. 19320 d. Rs. 34190

[GATE 2001]

6. Four alternative investment opportunities are available in a mining project each offering different revenues for the next two years as shown in the table. If the interest rate is 10 % annually, which alternative provides the best investment opportunity based on the net present value?

	Case I	Case II	Case III	Case IV
Initial Investment (Rs.)	40,000	1,00,000	80,000	1,20,000
Return in Year 2	30,000	75,000	65,000	80,000
Return in Year 3	30,000	75,000	65,000	80,000

a. I b. II c. III d. IV

[GATE 2004]

7. The probability distribution of the outcome of an investment is

Outcome (Rs.)	1000	700	300
Probability	0.2	0.5	0.3

The measure of risk of the investment based on standard deviation (σ) is given as

a. 183.20 b. 204.00
c. 249.80 d. 260.82

[GATE 2004]

8. Hoskold's formula is used for
a. Salting b. Speculative rate of return
c. Depreciation d. Ore reserve calculation

9. Sum of the years digit method is used for
a. Salting b. Speculative rate of return
c. Depreciation d. Ore reserve calculation

10. Using Hoskold's formula, the present worth, in rupees of annuity of Rs. 1 per year for the 10 years after the deferment period of 5 years and allowing the purchaser 10 % interest is
a. 3.81 b. 5.22 c. 5.57 d. 8.41

11. A company producing bearing plates for rock bolts has the selling price Rs. 20/unit and the variable cost has Rs. 12/unit. If the total fixed cost is Rs. 5,60,000, then the break even output units of bearing plates is
a. 17,500 b. 28,000

12. A company producing bearing plates for rock bolts has the selling price Rs. 20/unit and the variable cost has Rs. 12/unit. If the total fixed cost is Rs. 5,60,000, then the profit earned for an output of 1,00,000 units of bearing plates is
a. Rs. 2,40,000 b. Rs. 6,40,000
c. Rs. 8,00,000 d. Rs. 14,40,000

13. A company producing bearing plates for rock bolts has the selling price Rs. 20/unit and the variable cost has Rs. 12/unit. If the total fixed cost is Rs. 5,60,000, then for a target profit of Rs. 4,00,000, the total output units of bearing plates is
a. 30,000 b. 50,000
c. 1,20,000 d. 1,60,000

14. A bank lends a mine owner Rs. 1, 00,000 today to purchase hand-held drills. If a note of agreement is signed to pay the bank Rs. 1,76,234 at the end of 5 years, the rate of interest in percent compounded yearly on the loan becomes
a. 6 b. 8 c. 10 d. 12

15. The net present value of Rs. 400 at the end of 5 years at 10% discount rate is
a. Rs. 273 b. Rs. 226
c. Rs. 248 d. Rs. 465

[GATE 2000]

16. The uniform end of year payment, R, which can be realized for n years from a single present investment, P, at i rate of interest is

(a) $R = P(1+i)^n\left[\frac{i}{(1+i)^n - 1}\right]$ (b) $R = P(1+i)^n\left[\frac{(1+i)^n - 1}{i}\right]$

(c) $R = P\left[\frac{(1+i)^n - 1}{i(1+i)^n}\right]$ (d) $R = \frac{P}{(1+i)^n}\left[\frac{i}{(1+i)^n - 1}\right]$

[GATE 1999]

17. A company takes a loan of Rs. 25,000 at 8 % interest for the extension of a mine. To repay the loan, an additional saving of Rs. 2500 per annum is necessary. The period taken for repayment is
a. 25 years b. 28 years
c. 21 years d. 30 years

[GATE 1999]

17. As per DCF analysis a project report is acceptable if

a. Discounting rate >IRR
b. Discounting rate < IRR
c. Discounting rate = IRR
d. There exists no relation between the two

[GATE 1998]

18. The present value of a 4 year annuity of Rs. 10,000 discounted at 10 % is
a. 35,600 b. 33,500
c. 32,400 d. 31,700

[GATE 1998]

19. For an open pit operation the value of metal is Rs. 210/kg, and recoverable grade is 1.2 %. Production cost per tonne of ore inclusive of mining and processing but excluding stripping is Rs. 2000. If the break even stripping ratio is 3.49 m^3/te, then the stripping cost is (1te = 1000 kg)
a. Rs. 149 /m^3 b. Rs. 140 /m^3
c. Rs. 72 /m^3 d. Rs. 60 /m^3

[GATE 1997]

20. The net present value in Rs. of a 3 year annuity of Rs. 10,000 discounted at 10 % is
a. 9,091 b. 17,355
c. 24,869 d. 26,446

21. A company invested Rs. 4 lakhs in a machine. The net income expected from the operation of the machine is Rs. 80,000 per annum. The payback period for the machine in years is
a. 5 b. 6
c. 7 d. 8

22. An O.C. mine requires 16 blocks of overburden of 3000 te each to mine 4 blocks of coal of 2000 te each. The mining cost of both overburden and coal is Rs. 50 per tonne. The cost of transportation of overburden to dump areas is Rs. 50 per tonne. Coal transportation cost to the selling point is Rs. 75 per tonne. The price of coal is Rs. 800 per tonne. If the royalty and tax to the state is Rs. 50 per tonne, the nominal profit in rupees is
a. 2,00,000 b. 20,00,000
c. 30,00,000 d. 50,00,000

[GATE, 2005]

23. A person borrows a sum of Rs 50,000 @ 6 % interest rate for 10 years. Determine the equal annual sum to be paid at

a. 6795 b. 5672
c. 6749 d. 6592

24. An equipment cost Rs. 8,00,000 today and has a service life of 15 years. The salvage value at the end of 15 years is Rs. 2,00,000 and the interest rate is 10%. What will be the appraisal value at the end of 10 years?
a. 4,99,070 b. 5,66,876
c. 4,56,873 d. 3,45,456

25. What will be the profitability index with discount rate of 12 % for the cash follows given below?

Year	0	1	2	3	4	5
Net Cash Flow in (Rs.)	-40000	15000	14000	13000	12000	11000

a. 1.192 b. 2.186
c. 2.193 d. 0.292

26. The ABC copper company recently bought an ore-bearing parcel of land for Rs. 2,000,000. The recoverable reserves in the mine were estimated to be 5,00,000 tons. If 75,000 tons of ore were mined during the first year and 50,000 tons sold, what was the depletion allowance for one year?
a. Rs. 2,00,000 b. Rs. 2,50,000
c. Rs. 3,00,000 d. Rs. 4,00,000

27. The ABC copper company recently bought an ore-bearing parcel of land for Rs. 2,000,000. The recoverable reserves in the mine were estimated to be 5,00,000 tons. If 75,000 tons of ore were mined during the first year and 50,000 tons sold. Suppose at the end of year one reserves were reevaluated and found to be only 4,00,000 tons. If 50,000 additional tons are sold in the second year, what is the depletion allowance for year two?
a. Rs. 2,00,000 b. Rs. 2,25,000
c. Rs. 2,50,000 d. Rs. 3,00,000

28. A mining industry has the following total cost and total revenue functions :

Total cost = TC = $50 + 100q - 6q^2 + 1/3\ q^3$

Total revenue = TR = 64 q

Where q is the output of mine in tonne/day

The profit will be maximized for what level of production?

a. 4 b. 6

Answer

1. a	2. b	3. c	4. b	5. d	6. c
7. c	8. b	9. c	10. a	11. d	12. a
13. c	14. d	15. c	16. a	17. c	18. d
19. a	20. c	21. d	22. a	23. a	24. d
25. a	26. a	27. b	28. b		

□

9.0 Mineral Dressing

9.1 Classification

Classification is a method of separating mixtures of minerals into two or more products on the basis of the velocity with which the grains fall through a fluid medium.

9.2 Crushers

9.2.1 Introduction

Crushing is the first mechanical stage in the process of comminution in which the main objective is the liberation of the valuable minerals from the gangue. Crushing may be in open or closed circuit depending on product size. In open circuit crushing, undersize material from the screen is combined with the crusher product and is then routed to the next operation.

In closed circuit crushing operation, undersize from the screen is the finished product. The crusher product is returned to the screen so that any over over-size material will be recirculated. The open circuit and closed circuit crushing systems are shown in Figure 9.1 and Figure 9.2.

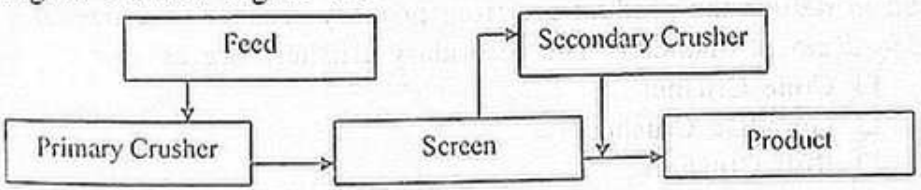


Figure 9.1 : Open-circuit crushing

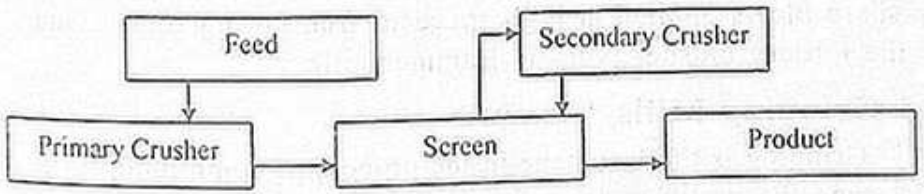


Figure 9.2 : Closed-circuit crushing

9.2.2 Types of Crushers

duty machines, used to reduce the run-of-mine ore as large as 1.5 m across down to a suitable size in the order of 10 -20 cm for transport and for feeding the secondary crushers. They are always operated in open circuit.

There are mainly two types of primary crushers :

- Jaw Crushers : Jaw crushers are further classified by the method of pivoting the swing jaw.
  - Blake Crusher : In the Blake crusher the jaw is pivoted at the top and thus has a fixed receiving area and a variable discharge opening.
  - Dodge Crusher : In the Dodge crusher the jaw is pivoted at the bottom, giving it a variable feed area but fixed delivery area. It is never used for heavy duty crushing as it chokes very easily.
  - Universal Crusher : The Universal crusher is pivoted in an intermediate position, and thus has a variable delivery and receiving area.
- Gyratory Crushers : Gyratory crusher consists essentially of a long spindle, carrying a hard steel conical grinding element, the head, seated in an eccentric sleeve. Gyratory crushers are principally used in surface- crushing plants, although a few currently operate underground. It has higher capacity in comparison to Jaw crushers.

9.2.2.2 Secondary Crushers : Secondary crushers further used to reduce the product ore from primary crusher to a size of 0.5 – 2 cm in diameter. The secondary crushers are as

- Cone Crusher
- Gyradisc Crusher
- Roll Crushers

9.2.2.3 Tertiary Crushers or Impact Crushers : In this class of crusher, comminution is by impact rather than compression, by sharp blows applied at high speed to free falling rock. One of the tertiary crusher types is Hammer mill.

9.3 Grinding Mills

Grinding is the last stage in the process of comminution. In this stage the particles are reduced in size by a combination of impact and abrasion, either dry or in suspension in water.

According to the ways by which motion is imparted to the

9.3.1 Tumbling Mills

Tumbling mills are of three basic types: Rod mills, Ball mills, Autogeneous mills. Tumbling mills are employed in the mineral industry for coarse grinding processes, in which particles between 5 and 250 mm are reduced in size to between 40 and 300 μm.

9.3.1.1 Rod Mill and Ball Mill : A rod mill and ball mill grinding is a tumbling mill that utilises the steel balls and rods respectively as grinding media. The path of the ball or rod in a tumbling mill (Ball mill or Rod Mill) is circular followed by parabolic as shown in Figure 9.3. The rod abandons its circular path for a parabolic path, when the weight of the ball/rod is just balanced by the centrifugal force.

i.e., when $mV^2/R = mg \cos\alpha$

where, m = mass of the ball or rod

V = linear velocity

g = acceleration due to gravity

$$\Rightarrow \frac{(2\pi RN/60)^2}{R} = g\cos\alpha$$

$$\Rightarrow \cos\alpha = \frac{4\pi^2 RN^2}{3600\,g} = \frac{4\pi^2 N^2 (D-d)/2}{3600\,g}$$

where, N = speed of the mill in rev per minute

D = diameter of the mill

d = diameter of the ball or rod

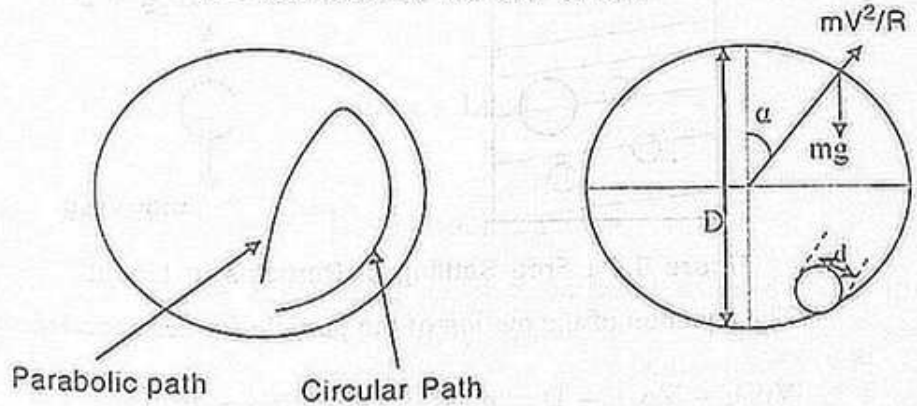


Figure 9.3 : Path of the ball or rod in a tumbling mill

The critical speed of the mill occurs when α = 0, i.e., the medium abandons its circular path at the highest vertical point. At this point, cos α = 1.

Critical speed of the mill= $Nc = \dfrac{3600\,g}{4\pi^2(D-d)/2}\ rev/\min$

9.3.1.2 **Autogenous Mills** : An autogenous grinding is a tumbling mill that utilises the ore itself as grinding media. The semi autogenous mill is an autogenous mill that utilises steel balls in addition to the natural grinding media.

9.3.2 Stirred Mills

These employ stirrers comprising a shaft with pins or disks. Stirred mills find application in fine (15–40 μm) and ultra-fine (<15 μm) grinding. Finer grinding is normally achieved with stirred mills.

9.4 Principles of Classification in Liquid Medium

Classifiers consist essentially of a sorting column in which a fluid is rising at a uniform rate. Particles introduced into the sorting column either sinks or rise according to whether their terminal velocities are greater or lesser than the upward velocity of the fluid.

9.4.1 Free Settling

Free settling refers to the sinking of particles in a volume of fluid which is large with respect to the total volume of particles, hence particles crowding is negligible. The force diagram and the movement of particles is shown in Figure 9.4.

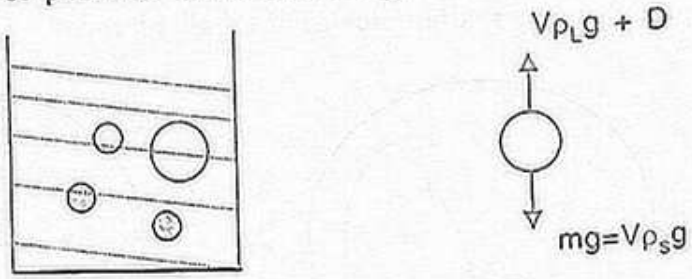


Figure 9.4 : Free Settling of Particles in Liquid

The equation of the motion of the particle for zero acceleration is

$$V\rho_S g - V\rho_L g - D = \text{mass*acceleration} = 0$$

Where,

V = volume of the solid particle

ρ_S = density of the solid

According to Stokes, drag force on the particle is due to viscous resistance and is generally applicable for finer particles.

Drag force = $3\pi d\eta v$

Where,

d = diameter of the particle

η = viscosity of the fluid

v = terminal velocity

Therefore, $v = \dfrac{gd^2(\rho_s - \rho_L)}{18\eta}$

According to Newton, drag force acting on the particle is due to turbulent resistance and is generally applicable to coarser particles.

Drag force = $D = 0.055\pi d^2 v^2\ \rho_L = V\rho_S g - V\rho_L g$

Therefore, $v = \left[\dfrac{3gd(\rho_s - \rho_L)}{\rho_L}\right]^{1/2}$

9.4.2 Hindered Settling

As the proportion of solids in the pulp increases, the effect of particle crowding becomes more apparent and the falling rate of the particles begins to decrease. Hindered settling condition begins when the proportion of solids in the pulp increases and density of the fluid is that of the pulp rather than that of the carrier liquid.

Therefore for hindered settling the settling velocity is given by

$$v = \frac{gd^2(\rho_s - \rho_p)}{18\eta} \quad \text{[Stoke's Law]}$$

$$v = \left[\frac{3gd(\rho_s - \rho_p)}{\rho_L}\right]^{1/2} \quad \text{[Newton's Law]}$$

The above expressions depends on the density of the pulp rather than density of the carrier liquid.

9.5 Jigging

Jigging process is used to separate the minerals from gangue materials with different specific gravity. In the jig, the separation of minerals of different specific gravity is accomplished in a bed which is rendered fluid by a pulsating current of water so as to

relatively coarse material. When the specific gravity difference is large, good concentration is possible with a wider size range.

Idealised jigging process can be described as in Figure 9.5.

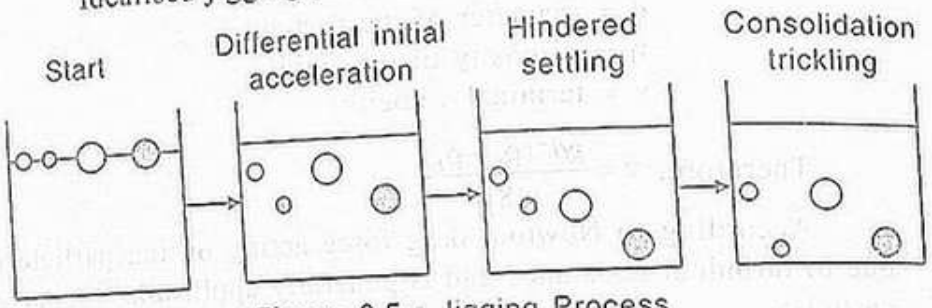


Figure 9.5 : Jigging Process

9.6 Dense Media Separation/ Heavy Media Separation

DMS is applied to the pre-concentration of minerals, i.e. the rejection of gangue prior to grinding for final liberation. It is also used in coal preparation to produce a commercially graded end-product, clean coal being separated from the heavier shale or high-ash coal. In this process heavy liquids of suitable density are used, so that those minerals lighter than the liquid float, while those denser than it sinks.

Feed

↓

Minerals (S.G. < x) ⟵ Fluid Medium (S.G. = x) ⟶ Minerals (S.G.> x)

Float

Sink

9.7 Froth Floatation

Flotation is a physico-chemical separation process that utilises the difference in surface properties of the valuable minerals and the unwanted gangue minerals.

9.8 Washing

Washing of run-of-mine ore can be carried out to facilitate sorting by removing obscuring dirt from the surfaces of the ore particles. Washing is normally performed after primary crushing as the ore is then of a suitable size to be passed over washing

EXERCISE

1. In-situ ore of 1.5 % Cu got diluted with waste rock of value 0.2 % Cu. If the resultant extracted ore has 1.2 % Cu, what is the percentage of dilution?
   a. 30 b. 25
   c. 23 d. 20
2. For heavy duty crushing on hard material the type of jaw crusher normally selected is
   a. blake type double toggle b. dodge type
   c. single toggle type d. none of the above
3. The mill which accepts the lowest size of feed is
   a. rod mill b. ball mill
   c. autogeneous mill d. none of the above
4. The diameter of a ball mill is 3 m and the diameter of the balls in the ball mill is 50 cm. What is the critical speed of the ball mill?
   a. 511 rev/min b. 520 rev/min
   c. 470 rev/min d. 465 rev/min
5. What will be the value of d_{50} for the specified overflow of 60 % passing 74 m when the value of the multiplier is 2.08?
   a. 62.08 ìm b. 35.52 ìm
   c. 124.8 ìm d. 154.0 ìm
6. To separate two finely divided solids from one another the process to be selected is
   a. flotation b. washing
   c. jigging d. none of the above
7. The method of separation of two or more of minerals of different specific gravity is
   a. flotation b. sieving
   c. washing d. HMS
8. As the ball load increases, the energy consumed by ball mill
   a. decreases b. increases
   c. first increases and then decreases
   d. first decreases and then increases
9. The path of a typical ball in a ball mill is
   a. circular
   b. elliptical
   c. parabolic

10. When the particles of same or different specific gravity come to rest at different instants, the phenomenon is termed as
   a. stratification b. hindered settling
   c. consolidation trickling d. differential acceleration

11. Froth flotation is used in ore treatment plant to separate
   a. two minerals in one b. mineral and gangue material
   c. none of the above d. both (a) and (b)

12. Hydrocyclones are used in ore treatment plant to separate
   a. coarse and fine particle b. mineral and gangue
   c. concrete and tailings d. none of the above

13. Magnetic separation is
   a. a dry method b. an wet method
   c. combination of two d. none of the above

Answer

1. a	2. a	3. d	4. a	5. d	6. a
7. d	8. d	9. d	10. d	11. a	12. a
13. c					

□

10.0 Geostatistics

10.1 Arithmetic Mean

If the variable x takes the n values $X_1, X_2 \ldots X_n$ the arithmetic

$$\bar{x} = \frac{x_1 + x_2 + \ldots x_n}{n} = \frac{1}{n}\sum_{i=1}^{n} x_i$$

If the distribution, the variable $x_1, x_2 \ldots x_n$ occur with respective frequencies $f_1, f_2, \ldots f_n$ the A.M. is given by

$$\bar{x} = \frac{f_1x_1 + f_2x_2 + \ldots f_nx_n}{f_1 + f_2 + \ldots f_n} = \sum_{i=1}^{n} f_ix_i / \sum_{i}^{n} f_i$$

The above formula gives weighted arithmetic mean, since the value of the variable x_i is weighted by frequency f_i.

10.2 Median

Median is the value of the middle variable when variables are arranged in ascending or descending order. If the number of the variables is odd, say 2n+1, the value of the (n + 1)th variables gives the median value, while if they are even, say 2n, then the mean of the nth and (n+1)th values gives the median. In the case of grouped frequency distribution, the median by simple interpolation is given by

$$M_d = L + \frac{\frac{N}{2} - F}{f} * i$$

Where, M_d = median
L = lower limit of the median class
f = frequency of the median class
F = total of all the frequencies before the median class
i = width of class interval of the median class

10.3 Mode

Mode is the value of the variable which has the maximum frequency. It is the maximum ordinate of the ideal curve which gives the closest possible fit to the actual distribution. For a grouped distribution, the mode is given by the formula

$$\text{Mode} = L + \frac{f - f_{-1}}{2f - f_1 - f_{-1}} * i$$

Where, L = lower limit of the modal class

i = width of the modal class

f_1 and f_{-1} = frequencies of the class following and preceding the modal class respectively.

f = frequency of the modal class

Empirical relationship among mean, median and mode :

For moderately symmetrical distribution, the following relationship approximately holds.

Mode = Mean − 3 (Mean − Median)

For a symmetrical distribution, the mean, median and mode coincide.

10.4 Standard Deviation

If we take our measure of dispersion as the sum of the deviations of the values of variate from the mean, the sum will be zero even though the actual dispersion from the central value may be quite large. In order to remove this discrepancy, either we take the mean of the absolute deviations from the average value or square the deviations to do away the negative sign of the deviations. Standard Deviation (σ) is the positive square root of the mean of the squares of the deviations from the mean.

$$\text{i.e. } \sigma = \left\{ \frac{1}{N} \sum_i (x_i - \bar{x})^2 \right\}$$

Where, N is the total frequency.

In case of frequency distribution

$$\sigma = \left\{ \frac{1}{N} \sum_i f_i (x_i - \bar{x})^2 \right\}$$

10.5 Mean Deviation or the Average Deviation

Another measure of dispersion in which the absolute values of the deviations are considered is used and is known as mean deviation. The mean deviation about the mean

$$\text{Mean Deviation} = \frac{1}{N} \sum f_i / |x_i - \bar{x}|$$

10.6 Coefficient of Variation

A measure of dispersion known as coefficient of variation is given by

Coeff

10.7 Probability Distribution Function

For a discrete random variable, there are spikes of probability associated with the values that the random variable assumes. In case of discrete random variables, the probability distribution is called a probability mass function and in case of continuous random variable it is called a probability density function. The probability density function curve is shown in Figure 10.1.

The cumulative distribution function. F(x), represents the probability that X is less than or equal to x_k, for a discrete random variable is given by

$$F(x_k) = P(x \le x_k) = \sum_{k=1}^{n} P(x_k)$$

The cumulative probability density function curve is shown in Figure 10.2.

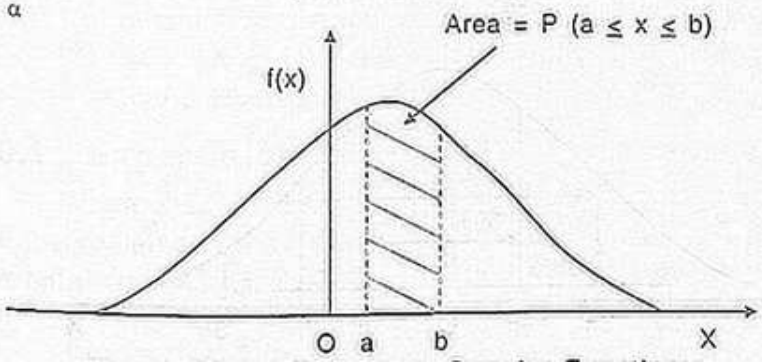
Probability distributions of continuous random variables are smooth curves. The probability density function (pdf) of a continuous random variable is denoted by f(x). The cumulative distribution function (cdf) of a continuous random variable is denoted by F(x). It is a non-decreasing function with a maximum value of 1. The cdf represents the probability that X is less than or equal to x,

$$\text{i.e., } F(x) = P(X \le x) = \int_{-\infty}^{x} f(x)dx$$

Which is represented by the area under the probability curve $y = f(x)$ between the values x = a and $x = b$ under the following conditions

f (x) ≥ 0 for all values of x

$$\int_{\alpha}^{\beta} f(x)dx = 1, \text{ if } (\alpha, \beta) \text{ is in the range of } x$$



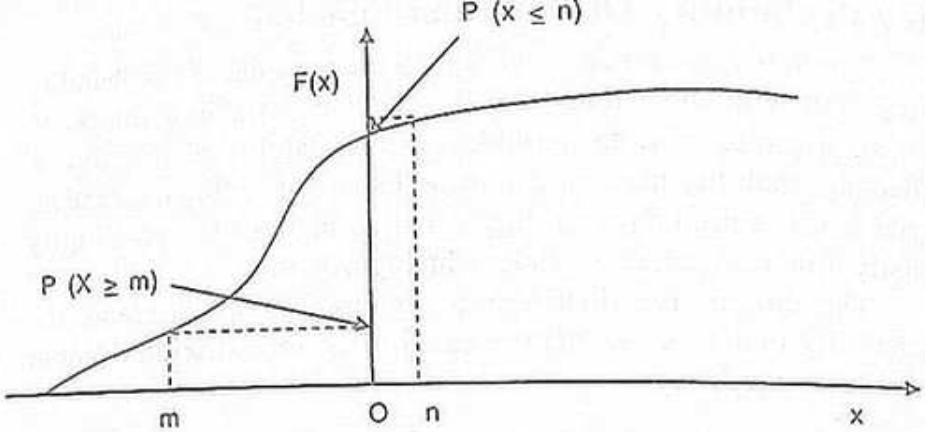


Figure 10.2 : Cumulative Probablity Distribution Function

Probability of the random variable taking a value exactly equal to a given value is zero, because

$$P(x=m)=P(m\le x\le m)=\int_m^m f(x)dx=0$$

The probability density is given by $f(x)=f(x)=\frac{d}{dx}F(x)$

10.8 Normal Distribution

Amongst all the probability distribution, the normal probability distribution is by far the most important and frequently used continuous probability distribution. The distribution is shown in Figure 10.3.

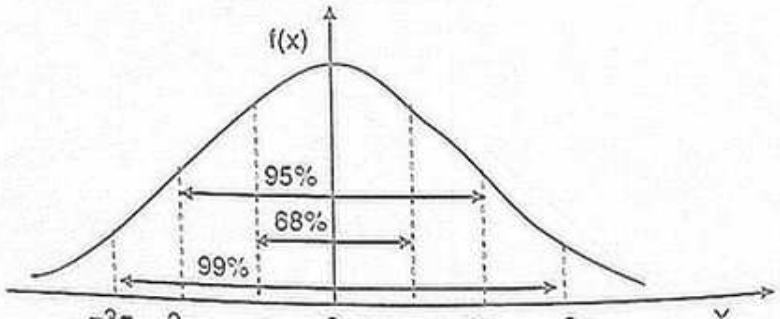
The probability function of this distribution is generally written as under

$$f(X_i=x)=\left(\frac{1}{\sqrt{2\pi\sigma}}\right)e^{-\left(\frac{1}{2}\right)[(x-\mu)^2/\sigma^2]}$$

Where,

μ = mean of the distribution

σ^2 = variance of the distribution



If the mean is μ and the standard deviation is σ, then about 68 % of the values lies within $\mu \pm \sigma$ and 95% values lies within $\mu \pm 2\sigma$ in a normal distribution.

10.9 Poisson distribution

The random variable of interest in Poisson distribution is the number of occurrences of a given event during a given interval.

The probability function of this distribution is generally written as under

$$f(X_i=x)=\frac{\lambda^x e^{-\lambda}}{!x}$$

x = 0, 1, 2.....

Where,

λ = average number of occurrences per specified interval or the mean of the distribution = n*p.

x = number of occurrences of a given event.

n = number of trials.

p = probability of success in a single trial.

10.10 Exponential distribution

Exponential distribution is the probability distribution of time, (say t), between events and such it is a continuous probability distribution concerned with the continuous random variable that takes on any value between zero and positive infinity.

The probability function of the exponential distribution is given by

$$f(x)=\mu e^{-\mu x}$$

Where, $x \ge 0$

μ = the average length of the interval between two occurrences

The expected value or mean of the exponential distribution is $1/\mu = \lambda$, and its variance is $1/\mu^2$

The cumulative distribution of the exponential distribution is

$F(x) = P(X \le x) = 1 - e^{-\mu x}$; $\quad x \ge 0$

$= 0$; elsewhere

10.11 Expectation of a Variate

When x is a discrete variable which may take n mutually exclusive values x_i = (i = 1, 2, 3.....n) and no others, with respective probabilities p_i (i = 1, 2, 3....n), the expectation of x is

$$E(x)=p_1.x_1+p_2.x_2+p_3.x_3+\ldots p_n.x_n=\sum_i^n p_i.x_i$$

Where x is a continuous variable, the expectation of x is given by

$$E(x) = \int_{-\infty}^{\infty} x.\phi(x)\,dx$$

Where, $\phi(x)$ is the probability density defining the function

The mean of a random variable X, denoted as μ, is equal to the expected value, i.e.

$$\mu = E(X)$$

The variance, $\sigma^2 = E(X-\mu)^2 = \int_{-\infty}^{\infty} (x-\mu)^2 . f(x)dx$

[For continuous random variable]

The variance of a constant time randon variable is the constant squared times the variance of random variable and is generally denoted simply as

$$Var(cX) = c^2\ Var(X)$$

The mathematical expectation of a sum of a number of random variables is equal to the sum of their expectations.

$$E(x_1 + x_2 + \ldots\ldots\ldots x_n) = E(x_1) + E(x_2) + \ldots\ldots\ldots E(x_n)$$

The mathematical expectation of the product of a number of independent random variables is equal to the product of their expectations.

$$E(x_1 * x_2 * \ldots\ldots\ldots x_n) = E(x_1) * E(x_2) * \ldots\ldots\ldots E(x_n)$$

10.12 Skewness

Skewness is the lack of symmetry and is termed as positive if the longer tail of the frequency curve is towards the higher value of the variate or in other words the mean is greater than the mode or median and negative in the reverse case. For a symmetrical distribution, the mean, mode and median coincide.

$$Coefficient\ of\ Skewness = \frac{Mean - Mode}{S.D.}$$

10.13 Correlation Coefficient

The relationship between two variables such that a change in one variable result in a positive or negative change in the other and also greater change in one variable results in a corresponding greater change in the other is known as correlation.

10.14 Rank Correlation

judges who have graded or ranked the same individuals regarding the same characteristics, for which there is no known objective standard of measurement.

$$\text{Rank correlation} = R = 1 - \frac{6\sum d_i^2}{n^3 - n}$$

Where, n = number of character

d_i = differences in agreement of the two judges.

For example,

Rank correlation between the two judges :

x	1	6	5	10	3	2	4	9	7	8
y	3	5	8	4	7	10	2	1	6	9
d^2	4	1	9	36	16	25	4	64	1	1

In the above example, n = 10, Σd^2 = 200

Therefore, R = 1 – (6*200)/ (1000 – 10) = –0.2

10.15 Probability

If there are S exhaustive, mutually exclusive and equally likely outcomes of a random experiment and 'r' of them are favourable to the happenings of an event A, the probability of the happening of A is

$$P(A) = \frac{r}{S}$$

It is sometimes expressed as odds in favour of A are r to S-r or the odds against A are S-r to r.

10.15.1 Additional Theory of Probability

If events A and B are mutually exclusive (i.e. both the events are not happened simultaneously), the probabilities of A and B or both happening is given by

$$P(A \cup B) = P(A) + P(B)$$

In general,

$$P(A_1 + A_2 + A_3 \ldots\ldots + A_n) = P(A_1) + P(A_2) + \ldots P(A_n)$$

If the events are not mutually exclusive, then

$$P(A \cup B) = P(A) + P(B) - P(AB)$$

In case of three events :

$$P(A \cup B \cup C) = P(A) + P(B) + P(C) - P(AB) - P(BC) - P(AC) + P(ABC)$$

P (B) = chances of occurring the event B

P (C) = chances of occurring the event C

P (AB) = chances of occurring both the events A and B simultaneously

P (BC) = chances of occurring both the events C and B simultaneously

P (AC) = chances of occurring both the events A and C simultaneously

P (ABC) = chances of occurring all the three events A, B and C simultaneously

10.15.2 Theorem of Total Probability for Compound Events :

If A is any event in a sample space S, then

$$P(A)=\sum_{i=1}^{n}P(A_i)P(A/A_i)=\sum_{i=1}^{n}P(A\cap A_i),$$

Provided P (Ai) $\neq$ 0, i = 1, 2, 3.....n

10.16 Bayes's Theorem

If the events A_1, A_2....An are pair-wise independent and A_1UA_2.....UA_n = S, then for any event A

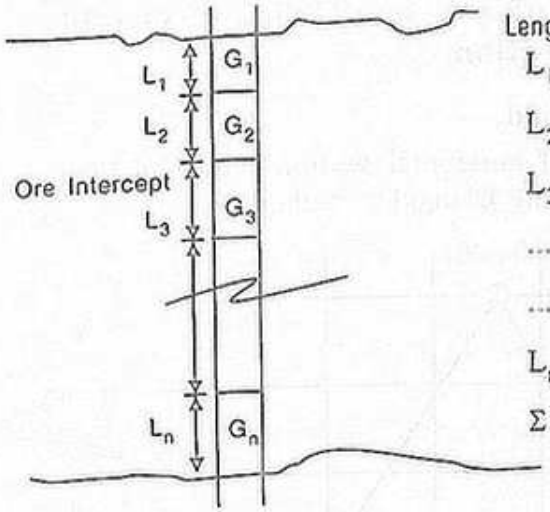
$$P(A_k/A)=\frac{P(A_k)P(A/A_k)}{P(A)}=\frac{P(A_k)P(A/A_k)}{\sum_{i=1}^{n}P(A_i)P(A/A_i)}$$

Baye's theorem serves to connect the conditional probability of A_k, given $P(A_k)$, with the conditional probability of A, given Ai and the probability of A_i themselves. If the evens Ai are called causes then the above relation can be regarded as a formula for the probability that the event A, which has occurred, is the result of the cause A_k.

The probabilities P (A_i), i = 1, 2, 3...n are called 'a priori probabilities' because they exist before we get any information from the experiments itself. The probabilities P (A/A_i), i = 1, 2 ...n are called 'posteriori' probabilities, because they are determined after the results of the experiment are known.

10.17 Determination of Average Grade of Ore in a Vertical Section

The average grade of the core of vertical section is shown



Length	Grade	Length*Grade
L_1	G_1	L_1*G_1
L_2	G_2	L_2*G_2
L_3	G_3	L_3*G_3
.....	.....	.....
.....	.....	.....
L_n	G_n	L_n*G_n
ΣL_i	ΣG_i	ΣL_i*G_i

Figure 10.4 : Length and Grade of Section

The average grade is given by

$$g=\frac{\sum L_iG_i}{\sum L_i}$$

10.18 Tonnage Factor

In mining, payment is normally made on the basis of the weight of the valuable material contained. The conversion from volume 'V' to weight 'W' and vice versa is done with the help of a tonnage factor (TF).

TF = Volume (V)/ Weight (W)

If the specific gravity of the mined material is 'S', the density of the mined material is given by

$$\rho_m = S * \rho_w$$

Where, ρ_w = density of water

ρ_m = density of mined out material

$$\text{Average grade of the ore actually mined} = \frac{X_1G_1+X_2G_2-X_3G_3}{X_1+X_2+X_3}$$

Where,

X_1 = Tonnage of ore in place

X_2 = tonnage of diluted ore

X_3 = tonnage of ore loss

G_1 = grade of ore in place

G_2 = grade of diluted ore

10.19 Determination of Average Grade of Ore in a Horizontal Section

10.19.1 Triangular Method

The average grade of horizontal section shown in Figure 10.5 can be determined using triangular method

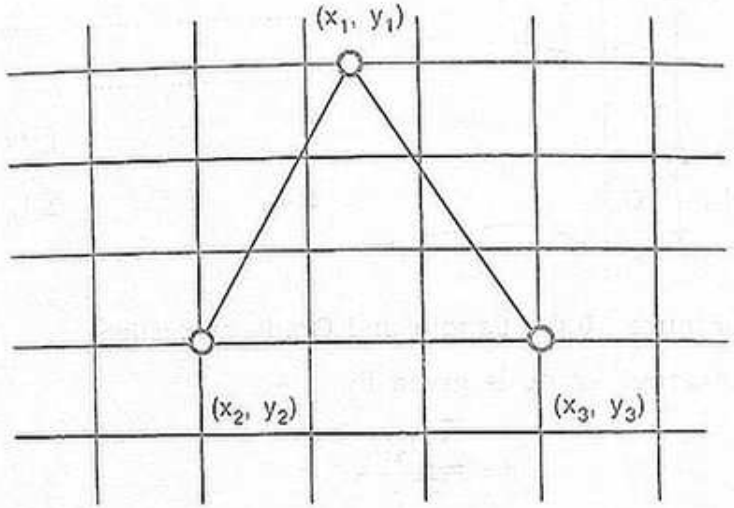


Figure 10.5 : Sample of Bore hole

The average grade of the triangular is given by

$$g = \frac{g_1 + g_2 + g_3}{3} \quad \text{[For the same borehole thickness]}$$

If the thickness of the boreholes are different

$$g = \frac{g_1 t_1 + g_2 t_3 + g_3 t_3}{t_1 + t_2 + t_3}$$

Average thickness is given by

$$\bar{t} = \frac{t_1 + t_2 + t_3}{3}$$

10.19.2 Polygons

In the polygon method, each drill hole lies within the centre of a polygon. The polygon is constructed such that its boundary is always equidistant from the nearest neighbouring hole. Within the polygon, the grade is assumed constant and equal to that of the hole it includes.

10.19.3 Constant Distance Weighting Technique

The grade of any point can be calculated on the basis the distance between the desired point and the neighbourhood holes. It is given by

Average grade of the desired block (P)

$$= g = \frac{\sum_{i=1}^{n} \frac{g_i}{d_i}}{\sum_{i=1}^{n} \frac{1}{d_i}}$$

Where,

g_i is the given grade at a distnace d_i away from the desired point, P

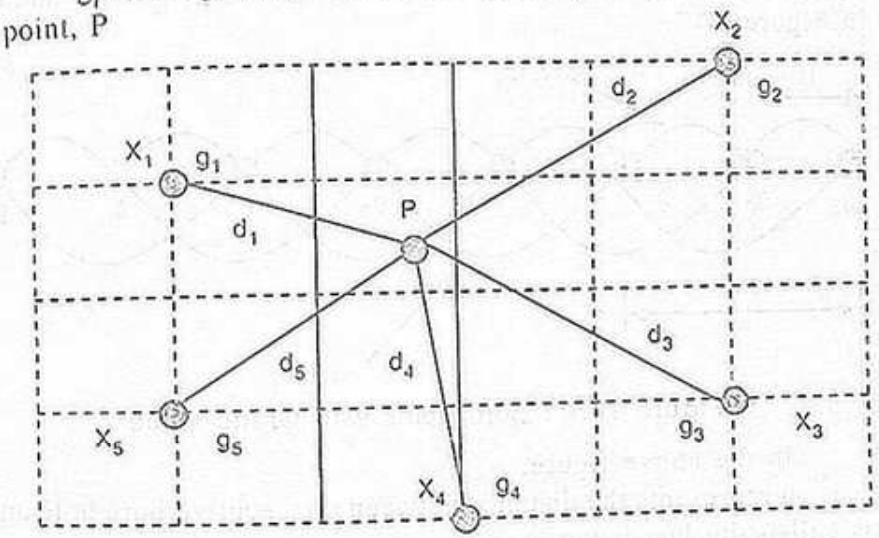


Figure 10.6 : Bore holes with grade values

10.19.4 Inverse Distance Weighting Technique

In the above formula, it was assumed that the grade values are changes linearly from the hole to the desired point. That is the grade of the desired block is more close to the nearer points than those far away. To emphasize this dependence, the weighting with distance can be increased. This is done by changing the power of d_i. If the dependence varies inversely with the square of the distance rather than linearly, the method is called inverse distance weighting method. Thus the grade of the desired block can be calculated by

$$\text{Average grade of the desired block (P)} = g = \frac{\sum_{i=1}^{n} \frac{g_i}{d_i^2}}{\sum_{i=1}^{n} \frac{1}{d_i^2}}$$

10.20 Applications of Statistics on the Orebody

Average grade is given by $= \bar{g} = \frac{1}{6}\sum_{i=1}^{n} g_1$

Variance in the grade values $= \sigma^2 = \frac{1}{6}\sum_{i=1}^{6}(g_i - \bar{g})^2$

Therefore, Standard Deviation $= \sqrt{Variation} = \sigma$

Determination of range of influence, lag distance and variogram : The boreholes with corresponding grades are shown in Figure 10.7

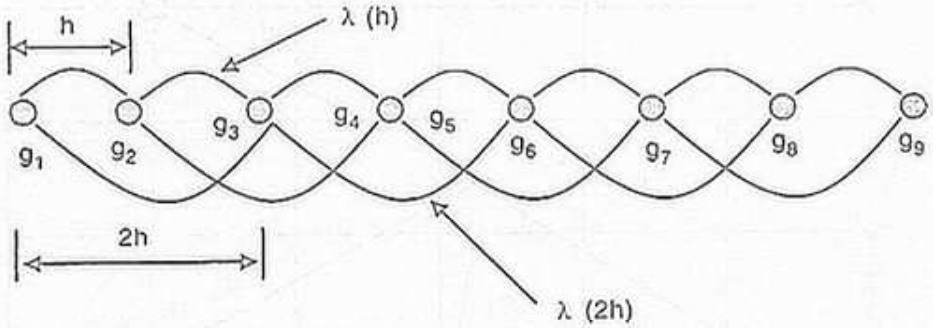


Figure 10.7 : Bore holes with Grade value

In the above figure,

h represents the distance between consecutive bore hole and is called the lag distance

g_i represents the grade values of the respectable holes

λ(h) represents the geostatistical variance or the semi-variance

Geostatistical variance can be determined by

Geostatistical Variance for the lag distance 'h'= λ (h) = (Statistical Variance)/ 2

$$= \frac{1}{2}s^2(h) = \frac{1}{2}\left[\frac{1}{n(h)}\sum_{n}(g_i - g_j)^2\right]$$

Geostatistical Variance for the lag distance '2h'= λ (2h) = (Statistical Variance)/ 2

$$= \frac{1}{2}s^2(2h) = \frac{1}{2}\left[\frac{1}{n(2h)}\sum_{n-1}(g_i - g_j)^2\right]$$

10.21 Trend of the Curve of Geostatistical Variance vs. Lag Distance

Wijsian, and the Spherical Model. But the Spherical model (Figure 10.8) is one in which the variogram reaches a finite value as h increases indefinitely. This finite value, referred to as the sill of the spherical variogram, is the overall variance of the deposit.

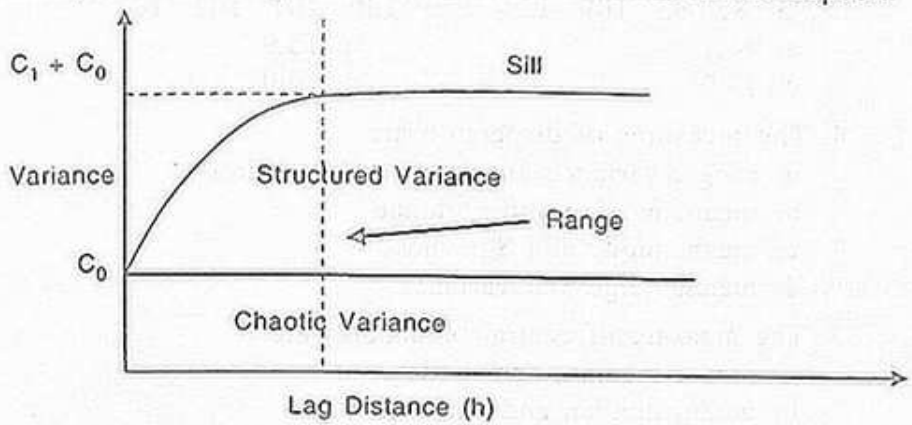


Figure 10.8 : Spherical Model

The spherical scheme is defined by the forumla

$$\lambda(h) = \begin{cases} c_1 + c_0 \ldots\ldots\ldots\ldots & [h > a] \\ c_1\left(\frac{3h}{2a} - \frac{h^3}{2a^3}\right) + c_0 \ldots [h \le a] \end{cases}$$

Where, $c_1 + c_0 = \lambda(\infty)$ and is called the sill value.

c_0 = nugget effect

a = range or maximum zone of influence

10.22 Quantification of Deposits through Variogram

Following types of quantification information provided by variograms :

- A measure of continuity of mineralization
- A measure of the area of influence of a sample
- A measure of mineral trend of the deposit

EXERCISE

1. What is the standard deviation of the following series?
   83 87 93 109 124 126 126 101 102 108
   a. 14.96 b. 15.96
   c. 13.96 d. 19.96
2. What is the mean of the following series?

a. 105.9 b. 150.9
c. 103.9 d. 190.9

3. What is the coefficient of variation of the following series?

83 87 93 109 124 126 126 101 102 108

a. 14.1 b. 15.9
c. 13.9 d. 19.9

4. The measures of dispersion are
   a. range, variance and standard deviation
   b. mean, median and variance
   c. mean, mode and Skewness
   d. mean, range and variance

5. The measure of central tendencies are
   a. mean, median, and mode
   b. mean, median and variance
   c. range, variance and standard deviation
   d. mean, variance and standard deviation

6. The correlation coefficient between two variables y (dependent) and x is 0.5. Given that the coefficient of determination is the ratio of explained variation and total variation of the regression model between the two variables y and x, the percentage of the total variation remains unexplained by the regression equation is

a. 25 b. 50
c. 60 d. 75

[GATE, 06]

7. A random variable, X is described by a probability density function

$$f(x)=\begin{cases}3x^2, 0\le x\le 1\\ 0,\ elsewhere\end{cases}$$

What will be the cumulative distribution function?

a. x b. x^2
c. x^3 d. 2/3 x^2

8. A random variable, X is described by a probability density function

$$f(x)=\begin{cases}3x^2, 0\le x\le 1\\ 0,\ elsewhere\end{cases}$$

What will be the expectation E(X)?

a. 2/3 b. 3/4
c. 1/4 d. 4/5

9. A random variable, X is described by a probability density function

$$f(x)=\begin{cases}3x^2, 0\le x\le 1\\ 0,\ elsewhere\end{cases}$$

What will be the variance of X?

a. 0.0375 b. 0.0435
c. 0.0325 d. 0.0425

10. If the variance of random variable X is 0.76, what is the variance of random variable 2X?

a. 3.04 b. 2.34
c. 4.02 d. 3.40

11. A random variable, X is described by a probability density function

$$f(x)=\begin{cases}3x^2, 0\le x\le 1\\ 0,\ elsewhere\end{cases}$$

What will be value of P [X ≥ 0.6]?

a. 0.784 b. 0.765
c. 0.576 d. 0.450

12. The grade of copper in a deposit is represented by the following probability density function

$$f(x)=\begin{cases}x/2 \ldots\ldots for\ .[0\le x\le 2]\\ 0 \ldots\ldots\ldots\ otherwise\end{cases}$$

The average grade of copper in the deposit is

a. 1.00 b. 1.33
c. 1.55 d. 2.00

13. Suppose the time in minutes that a person has to wait at a certain station for a train, is found to be a random phenomenon, a probability function specified by the distribution function.

F (x) = 0	for $x \le 0$
= x/2	$0 \le x \le 1$
= 1/2	$1 \le x \le 2$
= x/4	$2 \le x \le 4$
= 1	$x \ge 4$

a. 1/4 b. 2/3
c. 4/5 d. 1/2

14. In the above question, what is the probability that the person will have to wait between 1 and 3 minutes?
a. 1/2 b. 1/3
c. 1/4 d. 1/5

15. A set of measurements is found to be normally distributed with mean μ and standard deviation σ. The interval defined by (μ ± a) includes 68 % of the measurements. The value of a is
a. 0.5σ b. σ
c. 2σ d. 3σ

[GATE, 06]

16. A set of measurements is found to be normally distributed with mean μ and standard deviation σ. The interval defined by (μ ± a) includes 95 % of the measurements. The value of a is
a. 0.5σ b. σ
c. 2σ d. 3σ

17. The life in hours of a radio tube has the density $f(x) = 100/x^2$, when $x \geq 100$ and zero when $x < 100$. What is the probability that none of three such tubes in a given radio set will have to be replaced during the first 150 hours of operation?
a. 1/3 b. 1/2
c. 2/3 d. 1/4

18. The life in hours of a radio tube has the density $f(x) = 100/x^2$, when $x \geq 100$ and zero when $x < 100$. What is the probability that all three of the original tubes in a given radio set will have to be replaced during the first 150 hours of operation?
a. 1/3 b. 1/2
c. 2/3 d. 1/4

19. A bombing plane carrying three bombs flies directly above a railroad track. If a bomb falls within 10 m of the track, the track will be sufficiently damaged to disrupt traffic. With a certain sight, the [illegible] bomb is

$$f(x) = \begin{cases} \dfrac{(100-x)}{10{,}000}, & 0 < x < 100 \\ 0, & elsewhere \end{cases}$$

x represents the vertical deviation from the aiming point which is the track in this case. If all the bombs are used, what is the probability that the track will not be damaged?
a. $1 - (19/20)^3$ b. $1 - (16/20)^3$
c. $1 - (8/20)^3$ d. $1 - (4/20)^3$

20. If $f(x)$ and $g(x)$ are real functions defined in the interval $a < x < b$, such that $\int_a^b f^2(x)dx$ and $\int_a^b g^2(x)dx$ are finite then

a. $\left(\int_a^b f(x).g(x)dx\right)^2 \leq \int_a^b f(x)dx \int_a^b g(x)dx$

b. $\left(\int_a^b f(x).g(x)dx\right)^2 \geq \int_a^b f(x)dx \int_a^b g(x)dx$

c. $\left(\int_a^b f(x).g(x)dx\right)^2 = \int_a^b f(x)dx \int_a^b g(x)dx$

d. None of the above

21. If X and Y are random variables taking real values then
a. $[E(XY)]^2 \leq E(X)^2 E(Y)^2$
b. $[E(XY)]^2 \geq E(X)^2 E(Y)^2$
c. $[E(XY)]^2 = E(X)^2 E(Y)^2$
d. None of the above

22. Let X be uniformly distributed over (0, 1), $E(X^3)$ is
a. 1/3 b. 1/8
c. 1/4 d. 1/5

[GATE, 02]

23. The random variable X has the following probability mass function

P (4) = 1/4, P (8) = 1/4, P (12) = 1/4, P (16) = 1/4

The expected value of X is

a. 1 b. 3
c. 10 d. 12

24. For normal distribution which one of the followings statements is true?
a. mean, median and mode will coincide
b. mean and median coincide but they are different from mode
c. mean, median and mode are different from each other
d. median and mode coincide but they are different from mean

[GATE, 02]

25. A and B throw one dice for a prize of Rs. 11 which is to be won by a player who first throws 6. If A has the first throw, what is the expectation of A?
a. Rs. 6 b. Rs. 5
c. Rs. 8 d. Rs. 11

26. A and B throw one dice for a prize of Rs. 11 which is to be won by a player who first throws 6. If A has the first throw, what is the expectation of B?
a. Rs. 6 b. Rs. 5
c. Rs. 8 d. Rs. 11

27. From a bag containing 5 one rupee coins and 3 coins of 20 paisa each, a person is allowed to draw 2 coins indiscriminately. Find the value of his expectation
a. Rs. 1.40 b. Rs. 2.40
c. Rs. 3.40 d. Rs. 4.40

28. An urn contains n tickets numbered from 1 to n and m tickets are drawn one by one without replacement from the urn. What is the mathematical expectation of the sum of the numbers drawn?
a. m(n + 1) / 2 b. n(m + 1)2
c. mn / 2 d. m / n

29. A company manufactured product has 2 defects per unit of product inspected. What will be the probabilities of finding a product without any defect using Poisson distribution?
a. 0.135 b. 0.143
c. 0.125

30. A company manufactured product has 2 defects per unit of product inspected. What will be the probabilities of finding a product with 3 defects using Poisson distribution?
a. 0.135 b. 0.180
c. 0.153 d. 0.125

31. If the probability is 0.4 that an engineering job applicant has a post graduate degree, 0.6 that he has some work experience as an engineer, and 0.3 that he has both, the probability that he would have a post graduate degree or some work experience is
a. 0.3 b. 0.1
c. 0.7 d. 0.8

[GATE, 02]

32. One fifth percent of the bolts produced by a bolts manufacturing company turn out to be defective. The bolts are supplied in packets of 10. What will be the approximate number of packets containing no defectives in a consignments of 1, 00,000, if the distribution is assumed Poisson?
a. 0.98 b. 0.96
c. 0.95 d. 0.94

33. In a mining industry, the average number of fatal accidents per month is one-half. The number of accidents per month is adequately described by a Poisson distribution. What is the probability that four months will pass without a fatal accidents?
a. 0.00034 b. 0.00054
c. 0.00024 d. 0.00340

34. A bauxite deposit has been intersected by 5 drill holes. The values of alumina (% by weight) and silica (% by weight) in these drill holes are as follows :

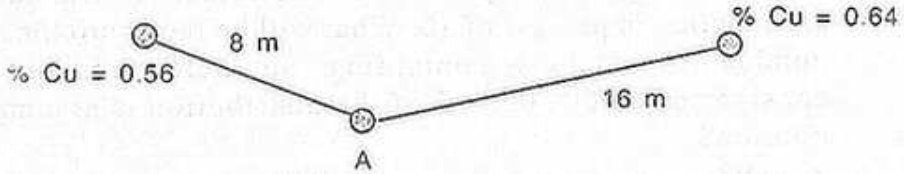
Drill hole number	Alumina (%)	Silica (%)
1	46	1
2	42	5
3	45	2
4	43	4
5	44	3

a. positive linear b. exponential
c. negative linear d. random

35. In the above question the unbiased estimate of variances of alumina and silica in $(\%)^2$ respectively are
a. 2.5 2.5 b. 2.0 2.5
c. 2.5 2.0 d. 2.0 2.0

36. The grade values of 30 iron ore samples in a mine are normally distributed. The sample mean (Fe % by weight) and the standard deviation of these sample observations are 60 % and 4 % respectively. The 95 % confidence limits for the population- mean are (given $z_{0.975} = 1.96$)
a. 60 ± 1.43 b. 60 ± 1.79
c. 60 ± 2.01 d. 60 ± 3.50

37. The grade of copper (%) at point A using the inverse distance weighting method



a. 0.47 b. 0.58
c. 0.61 d. 1.20

38. Assume the grade at A to be the average grade of copper, mill recovery to be 85 % and the smelting & refining loses to be 1 kg of copper per tonne of ore. The amount of saleable copper in kg/te of ore is
a. 2.93 b. 3.93
c. 4.93 d. 5.93

39. Five sample observations have been taken from a lead-zinc deposit. The observed values of zinc % and lead % are given in the following table

Zinc %	Lead %
6.0	4.2
5.2	3.0
6.0	6.8
2.4	2.0
4.0	3.2

The sample covariance between two variables is

Answer

1. a	2. c	3. a	4. a	5. a	6. d
7. c	8. b	9. a	10. a	11. a	12. b
13. a	14. c	15. b	16. c	17. c	18. a
19. a	20. a	21. a	22. b	23. c	24. a
25. a	26. b	27. a	28. a	29. a	30. b
31. c	32. a	33. a	34. c	35. a	36. a
37. b	38. c	39. b			

□

11.0 Selected Questions from GATE 2008 and GATE 2009

Mine System Engineering

1. A balanced transportation problem is characterised by
   a. total supply exceeds total demand
   b. total demand exceeds total supply
   c. total demand is equal to total supply
   d. total supply is either equal to or more than total demand

   [GATE 2009]

2. In the context of project management techniques, the true statement is
   a. CPM is stochastic and PERT is deterministic
   b. CPM is deterministic and PERT is stochastic
   c. Both CPM and PERT are deterministic
   d. Both CPM and PERT are stochastic

   [GATE 2009]

3. There are 50 lemon trees in a reclaimed mine area. Each tree produces 800 lemons per year. For each additional tree planted in this area, considering all trees, the output number of fruits per tree drops by 10 lemons in a year. The number of trees that to be added to the existing reclaimed area in order to maximise the total number of lemons in the year is
   a. 10 b. 15
   c. 16 d. 26

   [GATE 2009]

4. Consider the following linear programming problem:
   Maximize $z = 3x + 2y$
   s.t. $3x + 2y \geq 15$
   $2x + 3y \leq 6$
   $x \geq 0, y \geq 0$

   The above linear programming problem has
   a. Unique optimal solution
   b. Multiple optimal solution
   c. Unbounded solution
   d. Infeasible solution

5. A mine workshop has 4 lathe machines and 4 tasks for completion. Each of the machines can perform each of the 4 tasks. Each task can be assigned to one and only one machine. Estimated cost in rupees to complete each task is given in the matrix below.

Machine / Task	M1	M2	M3	M4
T1	61	92	52	72
T2	42	49	69	85
T3	47	59	80	71
T4	65	70	68	72

   The total optimum cost in Rupees for assigning the tasks to the machine is
   a. 210 b. 215
   c. 220 d. 286

   [GATE 2009]

6. Workmen arrive at a mine workshop to receive tools for maintenance. The inter-arrival time of workmen at the service counter is exponentially distributed with an average time of 10 min. The service time at the counter is also distributed exponentially with a mean time of 6 min. The probability that there is a queue (more than one workman) at the service counter is
   a. 0.24 b. 0.36
   c. 0.40 d. 0.60

   [GATE 2009]

7. In the above problem, the average time spent by a workman waiting for his turn to be served in min is
   a. 9 b. 12
   c. 15 d. 18

8. In PERT network, the activity duration is assumed to follow
   a. Beta distribution b. Normal distribution
   c. Binomial distribution d. Weibull distribution

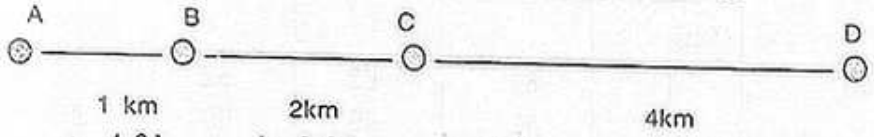
   [GATE 2008]

9. For an LP problem, identify the incorrect statement
   a. Optimal point lies in one of the corner points
   b. Objective function is linear
   c. All the constraints are linear

d. Optimal point lies in any one of the interior points of the feasible region.

[GATE 2008]

10. Four mines A, B, C, and D are located along a road as shown with production in Mt per year 1, 2, 1 and 3 respectively. In order to handle total coal produced, the ideal distance of a coal washery along the road from the mine A in Km is



a. 4.01 b. 3.91 c. 3.81 d. 3.71

[GATE 2008]

11. The solution of the following system of linear equations is

$x + 4y + 3z = 0$

$3x + 5y + 2z = 0$

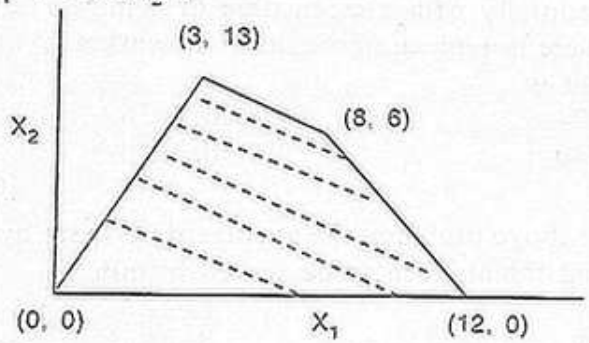
$8x + 10y + 12z = 0$

a. (0, 0, 0) b. (1, −1, 1)

c. (2, −1, −2) d. (−3, 0, 1)

[GATE 2008]

12. The feasible region of an LP problem is shown as given below. The maximum value of the objective function $Z = 1600X_1 + 1200X_2$ is



a. 20400 b. 20000

c. 19200 d. 16800

[GATE 2008]

13. A system of two identical mine pumps were to connected in series has reliability 0.49. If the pumps were to be connected in parallel, the system reliability would be

a. 0.21 b. 0.60

c. 0.91 d. 0.95

[GATE 2008]

14. Unit cost of matrix of a transportation problem is given below in certain monetary units

Destination / Source	1	2	3	
1	2	2	8	15
2	1	5	7	40
3	6	4	3	20
	10	25	40	Supply / Demand

The total cost of transportation based on the initial basic feasible solution obtained by the North-West corner rule is

a. 250 b. 290

c. 330 d. 360

[GATE 2008]

15. The optimal solution for the transportation problem has allocation as shown below :

Destination / Source	1	2	3	
1		15		15
2	10	10	20	40
3			20	20
	10	25	40	Supply / Demand

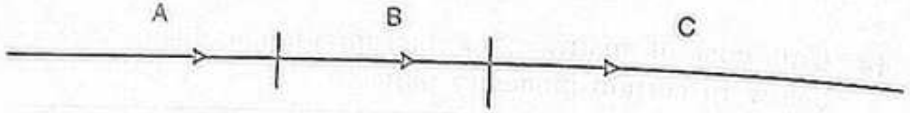
When compared to initial basic feasible solution from the above, the optimal allocation results in savings of

a. 10 b. 20

c. 30 d. 40

[GATE 2008]

16. A mining project comprising of A, B, and C activities is scheduled for 90 days at a cost of Rs. 1200 million. The manager of the project decides to reduce the time for completion of the project to 85 days. The decision was taken after 45 days.



Activity	A	B	C
Duration (days)	40	15	35
Crashing cost/day (million rupees)	15	25	20

The minimum cost of the project in million rupees after crashing 5 days is

a. 100
b. 1300
c. 1475
d. 1825

[GATE 2008]

Mineral Economics

17. A mining equipment has a life of 5 years with no salvage value. Assuming that the depreciation of the equipment is calculated by the straight line method, the average annual value of the equipment percentage of its original value is

a. 20
b. 40
c. 50
d. 60

[GATE 2009]

18. The cash flow table of a manganese mine for a particular year is shown below :

Item	Amount (Rs. in lakhs)
Revenue	900
Cost (other than depreciation)	300
Depreciation	100
Profit before tax	500

If the corporate tax is 50 % of the profit before tax, operating cash inflow in lakhs of rupees is

a. 400
b. 350
c. 250
d. 200

[GATE 2009]

19. In a mine site, the cost of shaft sinking in lakhs of rupees is given as 2.64D + 34.8, where D is the shaft depth in m. In the same site, the corresponding cost of driving an incline is 0.96 L, where L is the length of the incline in m. Assuming L by D ratio is 3.0, the depth in m beyond which the shaft sinking becomes more economical is

a. 43
b. 48
c. 145
d. 155

[GATE 2009]

20. The following information is provided for an ore deposit :

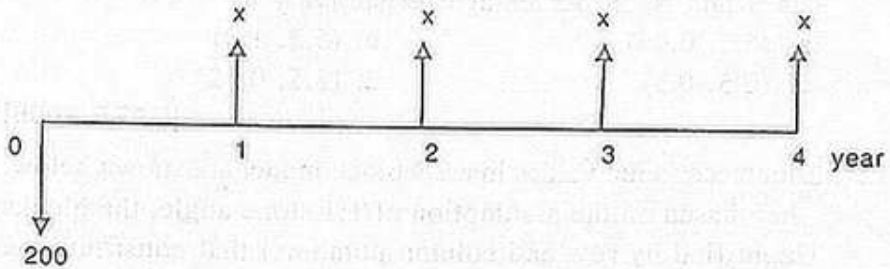
Number of waste blocks	= 10
Number of ore blocks	= 5
Volume of each waste block, m^3	= 600
Total cost of waste handling per m^3	= Rs. 100
Tonnage of each ore block	= 400
Total cost of ore handling per ton	= Rs. 150
Sale price of ore per ton	= Rs. 500

The net cash flow of mining the deposit in lakhs of rupees, is

a. 3.4
b. 2.5
c. 1.0
d. 0.8

[GATE 2008]

21. A cash flow diagram is shown below. Based on NPV, at 10 % rate of interest, the minimum annuity x at which the investment becomes viable is



a. 63
b. 54
c. 42
d. 35

[GATE 2008]

Geostatistics

22. A measure of dispersion of a sample of data set is

a. Mean
b. Median
c. Mode
d. Standard deviation

[GATE 2009]

23. In a normal distribution curve, the area between one standard deviation from mean on either side in percent is

a. 50
b. 68
c. 86
d. 95

[GATE 2009]

24. If X is a continuous random variable and f(x) defines its

probability density function, then the expected value of X is

a. $\int_{-\infty}^{+\infty} f(x)dx$ b. $\int_{-\infty}^{+\infty} x.f(x)dx$

c. $\sum_{i=-\infty}^{+\infty} x_i$ d. $\sum_{i=-\infty}^{+\infty} x_i$

[GATE 2008]

25. Daily production measured for a period of 50 days in a coal mine exhibits normal distribution with mean 1200 tpd and standard deviation 100 tpd. The 95 % confidence interval of daily production (standard normal variable z at 0.025 level of significance is 1.96) in tpd is

a. 1200 ± 120.5 b. 1200 ± 96.0

c. 1200 ± 39.6 d. 1200 ± 27.7

[GATE 2008]

26. In an iron ore deposit alumina is distributed with μ = 3% and σ = 0.5 %; whereas silica is distributed with μ = 2.5 % and σ = 0.8 %. The combined alumina and silica (as impurities) has μ and σ, in percentage respectively as

a. (5.5, 0.94) b. (5.5, 1.3)

c. (0.5, 0.3) d. (5.5, 0.62)

[GATE 2008]

27. Block economic values in a 2D block model are shown below. Then based on the assumption of 1:1 slope angle, the blocks (identified by row and column numbers) that constitute the ultimate pit are

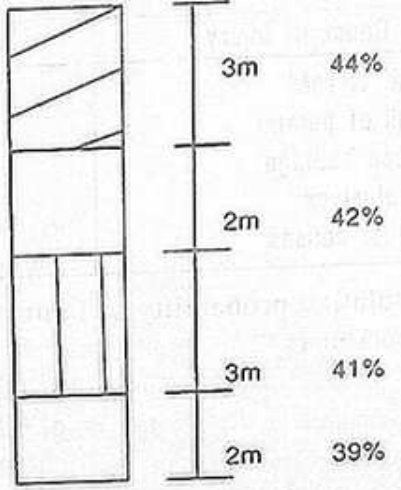
	1	2	3	4
1	−1	−1	1	−1
2	−1	1	3	−1
3	−1	−1	−1	−1

a. (1,1), (1,2), (1,3), (2,2) b. (1,2), (1,3), (1,4), (2,3)

c. (1,3), (2,4) d. (1,3), (1,4), (2,4)

[GATE 2008]

28. An assay value of alumina in a borehole from a bauxite deposit is as shown below. If the cut-off grade is 40 %, the composite value of ore in the borehole in percent is



a. 31.6 b. 33.9

c. 41.7 d. 42.2

[GATE 2008]

29. In a normal Gaussian distribution curve, the area between one standard deviation from mean on either side in percent is

a. 50 b. 68

c. 86 d. 95

[GATE 2009]

30. A measure of dispersion of a sample data set is

a. Mean b. Median

c. Mode d. Standard Deviation

[GATE 2009]

31. An iron ore deposit has a mean grade of 63 % Fe. During the course of mining, 30 % fines by weight are generated at a grade of 72 % Fe which are rejected. The effective mean grade of the deposit in Fe percentage is

a. 59.1 b. 53.1

c. 50.4 d. 41.4

[GATE 2009]

32. A mine having a reserve of 320 Mt produces 4 Mt of ore at the end of 1st year. If the mine increases production by 10 % every year, the percentage of the reserve that still remains at the end of 21st year is

a. 50 b. 35

c. 25 d. 20

[GATE 2009]

33. Cause wise data of injuries in an underground coal mine for a five-year period is given below :

Cause of injury	Number of injuries
Fall of roof	27
Fall of person	22
Rope haulage	17
Explosives	5
Other causes	4

The cumulative probability of injury due to fall of roof and fall of person is

a. 0.65 b. 0.50
c. 0.36 d. 0.29

[GATE 2009]

34. The following data represent the number of workers suffering from pneumoconiosis in 10 coal mines.

Mine	I	II	III	IV	V	VI	VII	VIII	IX	X
Number	10	16	14	15	14	12	17	13	15	12

The number of mines falling above the 50th percentile in terms of the number of workers suffering from pneumoconiosis is

a. 2 b. 3
c. 4 d. 5

[GATE 2009]

Mine Environment

35. The concentration of OH^- ion in a mine water sample is 10^{-11} mol/L. The pH of the sample is

a. 2 b. 3 c. 4 d. 11

[GATE 2009]

36. In an area within a surface mine, under static condition the following gases are found: NO_2, CO_2, O_3, and SO_2. Assuming no diffusion, reaction and bonding of the gases, the concentration of the gases from bottom upwards will be in the order of

a. NO_2, CO_2, O_3, and SO_2 b. SO_2, NO_2, CO_2, and O_3
c. SO_2, O_3, NO_2, and CO_2 d. NO_2, CO_2, SO_2 and O_3

[GATE 2009]

37. Electrostatic precipitator works on the principle of

a. Capacitance change b. Ionisation of the particles
c. Electro heating of gases
d. Centrifuging the gaseous molecules

[GATE 2008]

38. Precipiation of metallic ions in mine water drainage is carried out by

a. $CaSO_4$ and $MgSO_4$ b. $CaCO_3$ and $MgCO_3$
c. $Ca(OH)_2$ and NaOH d. $CaCO_3$ and $MgSO_4$

[GATE 2008]

39. A Roadheader district produces 20 mg/m^3 of airborne dust with the following size distribution :

Size up to	Cumulative wt %
1 μm	1
5 μm	5
10 μm	10
20 μm	20
50 μm	50
> 50 μm	100

The concentration of respirable fraction of dust in mg/m^3 is

a. 0.2 b. 2.0
c. 10.0 d. 1.0

[GATE 2008]

40. For a person working in an atmosphere containing 21 % O_2, the exhaled air contains 4.5% CO_2 and 16% O_2. The respiratory quotient of breathing is

a. 0.21 b. 0.9
c. 0.28 d. 1.11

[GATE 2008]

41. Mine water flowing at 1.5 m^3/s with 2 mg/l dissolved oxygen, joins river water flowing at 7 m^3/s containing 6 mg/l dissolved oxygen. The dissolved oxygen concentration of the mixture in mg/l is

a. 5.3 b. 4.8
c. 4.2 d. 3.9

[GATE 2008]

42. Mine water flowing at 1.5 m^3/s with 2 mg/l dissolved oxygen, joins river water flowing at 7 m^3/s containing 6 mg/l dissolved oxygen. The saturated value of the dissolved oxygen in the mixture is given to be 9.3 mg/l. On the basis, the initial oxygen deficit of the mixture in mg/l is

a. 2.4 b. 4.0
c. 6.8 d. 14.6

[GATE 2008]

b. Equal to atmospheric pressure
c. More than barometric pressure
d. Less than barometric pressure

Mine Ventilation

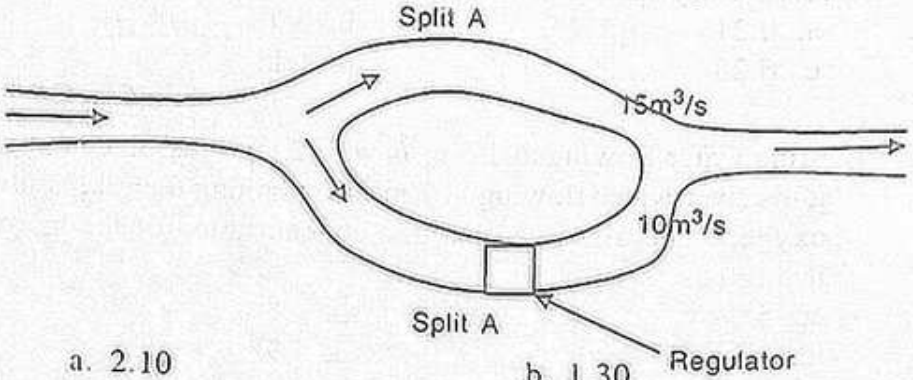
44. A gas mask does not include
a. Check valve
b. Warning device
c. Face piece assembly
d. Coolant canister

[GATE 2009]

45. Air flows at 2 m^3/s through a forcing fan duct of 0.3 m^2 having uniform cross-section. The duct resistance is 40 Ns^2m^{-8} and air density is 1.2 kg/m^3. The total pressure generated by the fan in Pa is
a. 186.7
b. 160.0
c. 133.3
d. 26.7

[GATE 2009]

46. Two splits A and B are ventilated from an intake airway. Resistances of the splits are 0.5 Ns^2m^{-8} and 0.8 Ns^2m^{-8} respectively. A regulator is placed in split B to maintain a flow of 15 m^3/s and 10 m^3/s in splits A and B respectively as shown in the figure. The size of the regulator in m^2 is



a. 2.10
b. 1.30
c. 1.20
d. 1.13

[GATE 2009]

47. A fan running at a speed of 280 rpm circulates 105 m^3/s of air in a mine. If the power input to the motor for driving the fan is recorded to be 75 kW, with the combined efficiency of fan and motor at 70%, the fan pressure in Pa is
a. 50
b. 350
c. 500
d. 650

[GATE 2009]

air in a mine. If the power input to the motor for driving the fan is recorded to be 75 kW, with the combined efficiency of fan and motor at 70%. If the fan pressure is to be increased by 200 Pa by changing the fan speed, the fan speed in rpm will become
a. 768
b. 549
c. 392
d. 332

[GATE 2009]

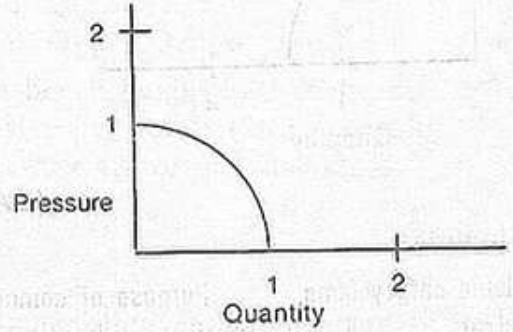
49. Moody diagram represents resistance coefficient in terms of
a. Reynolds number and asperity ratio
b. Viscosity and aspect ratio
c. Surface tension and viscosity
d. Reynolds number and surface tension

[GATE 2009]

50. Effective temperature is estimated from
a. Wet-bulb temperature, relative humidity, and air velocity
b. Dry-bulb temperature, relative humidity, and air velocity
c. Dry-bulb temperature, Wet-bulb temperature, and air velocity
d. Dry-bulb temperature, Wet-bulb temperature, and relative humidity

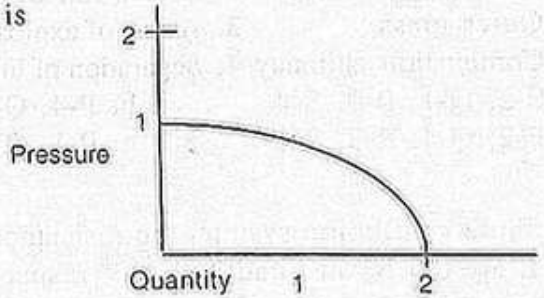
[GATE 2008]

51. Pressure-quantity characteristics of a mine fan is given below :

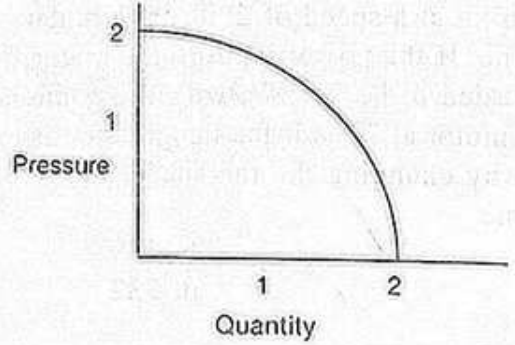


The combined characteristic of two such identical fans installed in parallel is

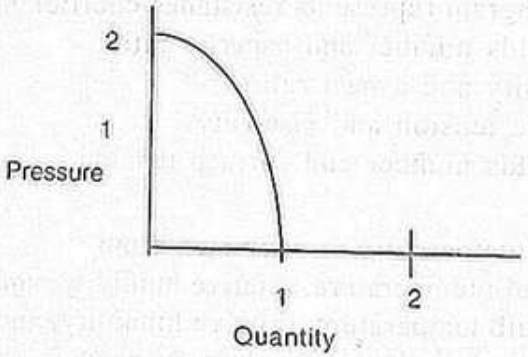
a.



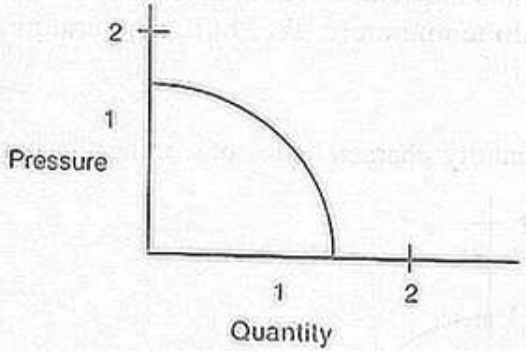
b.



c.



d.



[GATE 2008]

52. Match the followings

Component of flame safety lamp	Purpose of component
P- Asbestos ring	1. Dissipation of heat of flue gas
Q- Wire gauges	2. Formation of air-tight joints
R- Outer glass	3. Arrest of explosion inside the lamp
S- Combustion chimney	4. Separation of inlet air from flue gas

a. P-2, Q-1, R-3, S-4 b. P-4, Q-1, R-2, S-3
c. P-2, Q-4, R-3, S-1 a. P-1, Q-2, R-4, S-3

[GATE 2008]

53. In a mine ventilation system, the resistances of two splits A and B are 0.5 Ns^2m^{-8} and 2 Ns^2m^{-8} respectively. Combined

resistance of two shafts and trunk airways is 0.7 Ns^2m^{-8}. A quantity of 20 m^3/s of air passes through split A. The total air quantity passing the mine in m^3/s is

a. 30 b. 27
c. 25 d. 17

[GATE 2008]

54. In the above question, the total air power of the ventilation system in kW are

a. 82.9 b. 48.9
c. 24.9 d. 27.9

[GATE 2008]

Opencast Mining

55. Match the following in the context of Indian mining practice.

Equipment	Power source
P- Rocker shovel	1. Battery
Q- Locomotive	2. Compressed air
R- Shearer	3. Electricity (Maximum voltage 6.6 kV AC)
S- Dragline (24 m^3 bucket capacity)	4. Electricity (Maximum voltage 1.1 kV AC)

a. P-1, Q-2, R-3, S-4 b. P-2, Q-1, R-4, S-3
c. P-2, Q-1, R-3, S-4 d. P-1, Q-3, R-2, S-4

56. Given bench height : 12 m, burden : 4 m, Spacing : 5 m, explosive per hole : 120 kg, density of rock : 2600 kg/m^3, the powder factor in tonne/kg is

a. 2.0 b. 4.6
c. 5.2 d. 7.3

[GATE 2008]

57. Match the following in the context of Indian mining practice:

Equipment	Slice Thickness (m)	Action
P- Dragline	1. 6-12	a. Crowding
Q- Shovel	2. 30-40	b. Hoisting
R- Surface Miner	3. 0.2-0.4	c. Cutting

a. P-1-b, Q-2-a, R-3-c b. P-2-b, Q-1-a, R-3-c
c. P-2-a, Q-1-b, R-3-c d. P-2-b, Q-3-c, R-1-a

[GATE 2008]

58. If the value of ore is Rs. 600 per tonne, production cost Rs.

m^3, the break-even stripping ratio in m^3/tonne is

a. 4:1 b. 3:1
c. 1:3 d. 1:4

[GATE 2008]

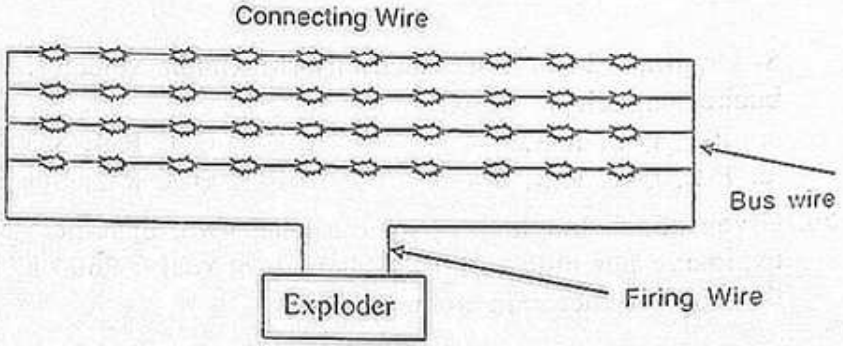
59. Total number of injuries in an opencast coal mine employing 800 persons is 16 in a year. As per DGMS norms, the injury rate per 1000 persons employed is

a. 13 b. 15
c. 20 d. 25

[GATE 2003]

60. A surface mine blast pattern shown below has the following details :

Accessory	Resistance (in Ohms)	Number or Length
Detonator	2 per detonator	40 nos.
Connecting wire	0.5/m	100 m
Bus wire	0.5/m	100 m
Firing wire	0.01/m	200 m



If the exploder supplies 440 V, the current in the blasting circuit in ampere is

a. 5.36 b. 7.31
c. 4.83 d. 2.53

[GATE 2008]

61. In a surface mine blast, the peak particle velocity (V in mm/s) is estimated from the equation $V = 120(\sqrt{SD})^{-1.0}$, where SD is square root scaled distance. If at a distance of 100 m from the blast site the permissible peak particle velocity is 25 mm/s, the maximum charge per delay in kg is

a. 404 b. 414
c. 434 d. 464

62. A surface mine blast design has 9 holes in a row, each of 8 m length and 200mm diameter. The spacing and burden are 6 m and 5 m respectively. The length of subgrade drilling is 1 m and the density of in-situ rock is 2.43 t/m^3. Assuming no back break, the output per blast in t is

a. 4593 b. 5905
c. 6124 d. 6299

[GATE 2009]

63. A surface mine blast design has 9 holes in a row, each of 8 m length and 200mm diameter. The spacing and burden are 6 m and 5 m respectively. The length of subgrade drilling is 1 m and the density of in-situ rock is 2.43 t/m^3. Considering an explosive density of 0.9 t/m^3 and stemming length of 2 m, the powder factor from the blast in t/kg is

a. 4.12 b. 4.00
c. 3.86 d. 3.01

[GATE 2009]

64. A non-electric detonating relay does not contain

a. Delay element b. Fuse head
c. Metal sleeve d. Neoprene connecting tube

[GATE 2009]

Mining Machinery

65. Koepe system of winding does not include

a. Tapper guide b. Limit switches
c. Safety hook d. Brake

[GATE 2009]

66. Two belt conveyors load a ground bunker, each at a rate of 400 tph, which is initially filled with 10000 t of coal. Coal is discharged from the bottom of the ground bunker onto a belt conveyor at a rate of 1200 tph. The time elapsed in hours before the bottom conveyor starts to operate below its rated capacity is

a. 6.5 b. 8.5
c. 12.5 d. 25.0

[GATE 2009]

67. A 1100 V, 3ϕ power supply system of a mine draws a load of 185 kW. The ammeter reading shows 115 A. The power factor of the system is

a. 0.84 b. 0.73
c. 0.64 d. 0.48

[GATE 2009]

68. A turbine pump of efficiency 70% discharges water at the rate of 2100 L/min at a total head of 100 m. If the pump is run by a motor of efficiency 90 %, the input power required for the motor in kW is

a. 22.49 b. 34.31
c. 44.11 d. 54.50

[GATE 2009]

69. A turbine pump of efficiency 70% discharges water at the rate of 2100 L/min at a total head of 100 m. If the velocity of water in suction and delivery pipes of the pump are 1.8 m/s and 2.5 m/s respectively, the diameter of suction and delivery pipes in cm are

a. 15.73 and 13.35 b. 7.86 and 6.67
c. 5.78 and 6.02 d. 4.97 and 4.22

[GATE 2009]

70. The tool used for borehole deviation is

a. String shot b. Kelly
c. Whipstock d. Rachet

[GATE 2008]

71. During over-winding, a cage is safely suspended in the headgear due to

a. Bull chain b. Rope capel
c. D-Link d. Detaching hook

[GATE 2008]

72. In the order of the chronological development, the longwall support systems are arranged as

P Powered support Q Link bar and friction support
R Frame support S Hydraulic support

a. P > Q > R > S b. R > S > Q > P
c. S > R > P > Q d. Q > S > R > P

[GATE 2008]

73. A drum winder of radius 2.5 m draws a power of 308 kW. When the maximum rope speed is 7 m/s, the RMS torque in kNm is

a. 55 b. 76
c. 110 d. 144

[GATE 2008]

74. A belt conveyor conveys material of average cross-sectional area of 0.09 m^2, of bulk density 1.5 tonnes/m^3, at a speed 2 m/s. The carrying capacity of the belt conveyor in

a. 972 b. 864
c. 732 d. 643

[GATE 2008]

75. A conveyor of rated power 100 kW hauls coal up-dip at 30 kg/s along an inclination of 15° and distance 300 m. Heat added by the conveyor to the air in kW is

a. 56.4 b. 65.9
c. 77.2 d. 82.3

[GATE 2008]

76. A cage of floor area 5 m^2 suspended in a shaft has a drag coefficient 2.5. If the velocity of air in the shaft is 6 m/s, the drag force (N) experienced by the cage is

a. 120 b. 170
c. 200 d. 270

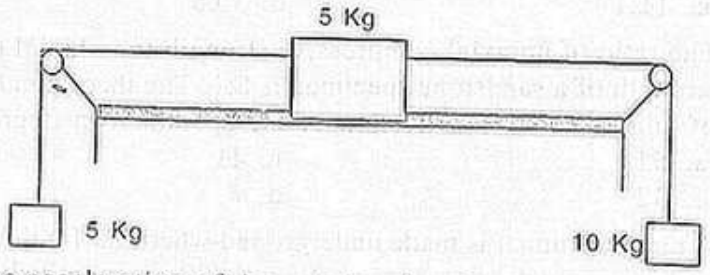
[GATE 2008]

77. In a coal handling plant wagons of 8 m length are loaded, at rake travel speed of 0.48 km/hr. The chute loading rate is 6000 tonne/hr. As the rake moves continuously, the chute stops for a total of 24s in between two wagons. The quantity of coal in tonne loaded in each wagon is

a. 52 b. 60
c. 76 d. 94

[GATE 2008]

78. Two blocks of mass 5 kg and 10 kg are connected with cords and frictionless pulleys as shown. Friction coefficient between the 5 kg block and table is 0.2



The acceleration of the system when the blocks are released from rest ('g' acceleration due to gravity), tension in the cord connected to 10 kg block and 5 kg block are

a. g/5, 8g, 6g b. g/6, 6g, 8g
c. g, 4g, 6g d. g/2, 2g, 4g

[GATE 2008]

79. A loco of mass 10000 kg

the tracks as 0.25. The loco offers a running resistance equal to 10 % of its weight. The draw-bar-pull generated by the loco on a level ground in kN is

a. 11.3 b. 14.7
c. 15.8 d. 17.2

[GATE 2008]

80. In the above questions, the draw-bar-pull generated by the loco when the upward gradient of the track is 5° in kN is

a. 6.16 b. 7.9
c. 9.5 d. 11.5

[GATE 2008]

Rock Mechanics

81. The rock mass classification system that considers "active stress" factor is

a. Q-system b. RMR
c. RQD d. GSI

[GATE 2009]

82. In a triaxial compression test if σ_1 is axial stress and σ_2 and σ_3 are confining stresses, then

a. $\sigma_3 > \sigma_2 = \sigma_1$ b. $\sigma_1 > \sigma_2 = \sigma_3$
c. $\sigma_1 = \sigma_2 > \sigma_3$ d. $\sigma_1 < \sigma_2 = \sigma_3$

83. The grain density and bulk density of a dry coarse gained sandstone rock sample are 3.0 gm/cc and 2.7 gm/cc respectively. The void ratio of the sample in percentage is

a. 8.4 b. 10.0
c. 11.1 d. 30.5

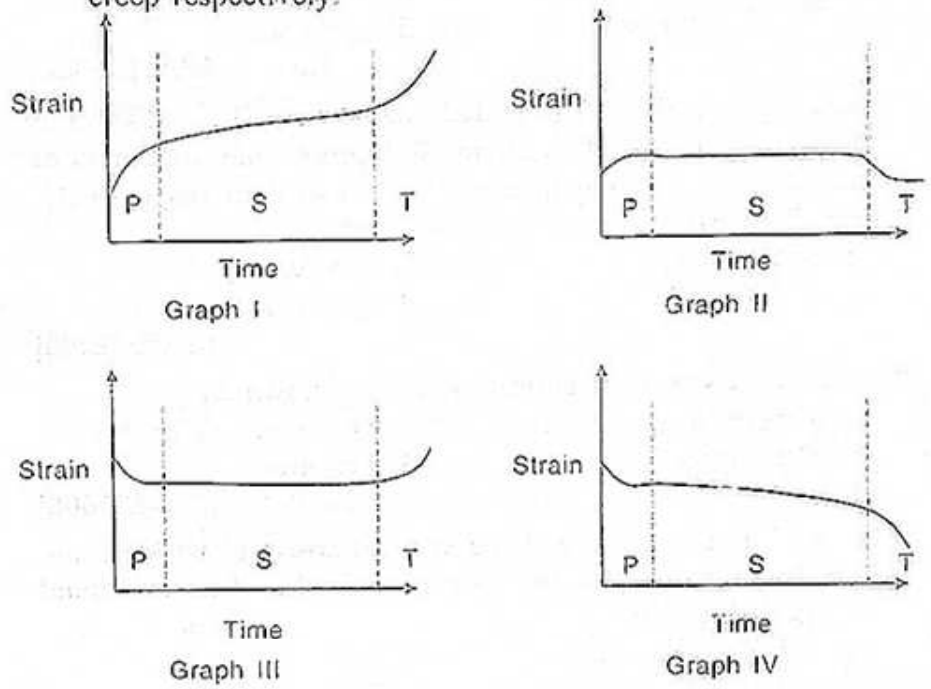
84. The ratio of uniaxial compressive strength to uniaxial tensile strength of a sandstone specimen is 8:1. The theoretical value of angle of internal friction of the specimen in degree is

a. 51 b. 41
c. 32 d. 7

85. A circular tunnel is made underground where far field vertical and horizontal stresses are P_0 and KP_0 respectively. The tangential stress ($\sigma_{\theta\theta}$) at the boundary of the tunnel for θ = 45° from the horizontal plane is $3P_0$. The value of K is

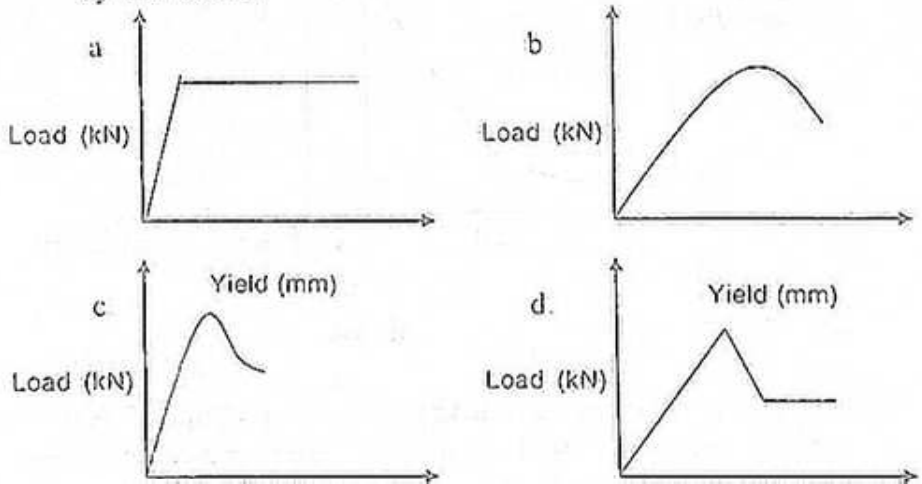
a. 0 b. 1
c. 2 d. 3

rock material under constant temperature? P, S and T in the figures refer to primary creep, secondary creep and tertiary creep respectively.



a. Graph I
b. Graph II
c. Graph III
d. Graph IV

87. The load yield characteristic of a hydraulic prop is represented by the curve



88. The velocity of the wave type that determines the 'rippability' of rockmass is

a. P wave
b. S wave
c. Raleigh wave
d. Love wave

[GATE 2008]

89. Under identical water head and roadway conditions for water dam construction, if P, Q, and R represent the thickness of flat dam, cylindrical dam and spherical dam respectively, the thickness would follow the order

a. R > P > Q
b. P > R > Q
c. P > Q > R
d. Q > P > R

[GATE 2008]

90. The grain size distribution of soil is known as

a. Permeability
b. Structure
c. Porosity
d. Texture

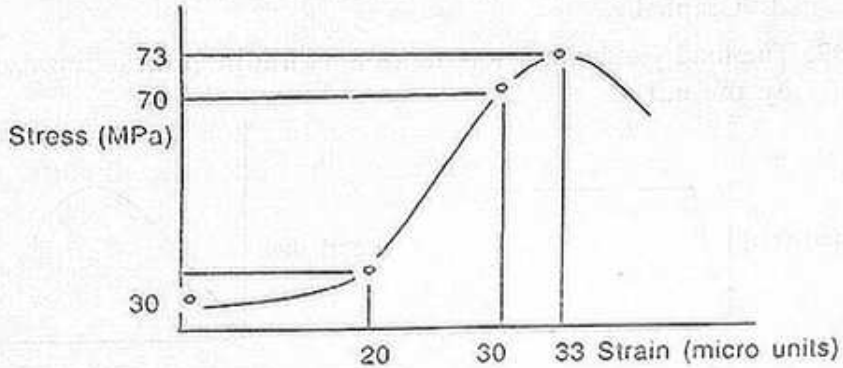
[GATE 2008]

91. In a bi-axial stress field the vertical stress is 10 MPa and the Poisson ratio for the rock mass is 0.2. The horizontal stress in MPa is,

a. 1.5
b. 2.5
c. 2.0
d. 5.0

[GATE 2008]

92. From the stress-strain diagram shown below, the tangent and the secant moduli of elasticity in GPa are



a. 4.0, 2.2
b. 3.3, 2.3
c. 3.3, 1.5
d. 4.0, 1.5

[GATE 2008]

93. A bord and pillar operation is planned at a depth of 300 m in a strata of average unit weight 24.5 kN/m^3 and compressive considering a factor of safety of 1, the maximum possible extraction ratio in percentage is

a. 28
b. 34
c. 45
d. 53

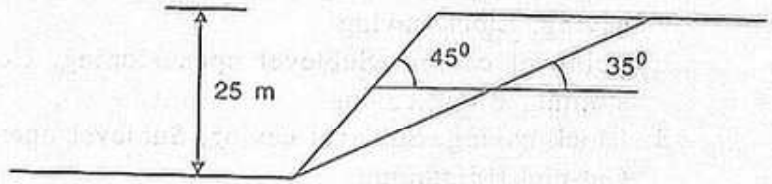
[GATE 2008]

94. A confined aquifer of 75 m thickness has 2 monitoring wells spaced 2500 m apart along the direction of water flow. The hydraulic conductivity of the aquifer is 40 m per day. The water head difference between the wells is 1.5 m. Applying the Darcy's law, the rate of flow per meter of distance perpendicular to the direction of flow in m^3/day is

a. 2.1
b. 1.8
c. 1.45
d. 1.21

[GATE 2008]

95. An open pit mine bench has a potential failure plane as indicated below. The unit weight, cohesion, and angle of internal friction of the rock mass are 24.5 kN/m^3, 0.02 MPa and 30° respectively.



The force for failure, on the potential failure plane is

a. 187 N
b. 1.87 kN
c. 18.7 kN
d. 1.87 MN

[GATE 2008]

96. In the above problem, the factor of safety of slope is

a. 0.7
b. 0.9
c. 1.1
d. 1.3

[GATE 2008]

97. Match the following :

Failure criteria	Relation
P. Drucker - Prager	1. $\sigma_1 = \sigma_3 + \sqrt{m\sigma_3 + s^2}$
Q. Hoek - Brown	2. $\tau = c + \sigma_n \tan\varphi$
R. Mohr - Column	3. $\sqrt{2/3}\left[\left(\frac{\sigma_1-\sigma_2}{2}\right)^2 + \left(\frac{\sigma_2-\sigma_3}{2}\right) + \left(\frac{\sigma_3-\sigma_1}{2}\right)^2\right]^{1/2}$

a. P-1, Q-3, R-2 b. P-3, Q-1, R-2
c. P-1, Q-2, R-3 d. P-3, Q-2, R-1

[GATE 2008]

Mining Methods

98. In longwall caving, the thickness of immediate roof is calculated from
   a. Bulking factor and width of longwall face
   b. Seam thickness and width of longwall face
   c. Seam thickness and Bulking factor
   d. Bulking factor and length of the panel

[GATE 2008]

99. Depending on the decreasing ability of surrounding rock to store strain energy, the underground metal mining methods can be ordered as
   a. Cut-and-fill stoping, Sublevel caving, Sublevel open stoping, Block caving
   b. Sublevel open stoping, Cut-and-fill stoping, Sublevel caving, Block caving
   c. Sublevel caving, Sublevel open stoping, Cut-and-fill stoping, Block caving
   d. Block caving, Sublevel caving, Sublevel open stoping, Cut-and-fill stoping

[GATE 2008]

100. If swell factor of ore in a shrinkage stope is 1.4, the output from the stope in percent of broken ore is
   a. 0 b. 29
   c. 40 d. 100

[GATE 2008]

101. Match the following :

Blast Problem	Cause
P Misfire	1. Poor stemming
Q Vibration	2. Low current
R Blown-out shot	3. Excess charge
S Cut-off shot	4. Improper delays

a. P- 3, Q- 2, R – 4, S-1 b. P- 4, Q- 1, R – 2, S-3
c. P- 2, Q- 3, R – 1, S-4 d. P- 1, Q- 2, R – 4, S-3

[GATE 2008]

102. Match the following :

Stoping method	Ore handling systems	Support System
P Breast Stoping	1. LHD	a. In-situ pillars
Q Cut and fill stoping	2. Scraper	b. Unsupported
R Sublevel stoping	3. Gravity flow	c. Mill tailings

a. P- 2-a, Q- 1-c, R –3-b b. P- 1-a, Q- 3-c, R – 2-b
c. P- 2-b, Q- 1-a, R –3-c d. P- 1-c, Q- 3-a, R – 2-b

[GATE 2008]

103. Match the following

Access	Haulage	Mineralisation
P Shaft	1. Track	a. Moderate depth
Q Decline	2. Trackless	b. Deep seated
R Adit	3. Hoisting	c. Hillock

a. P- 1-a, Q- 3-b, R –2-c b. P- 3-b, Q- 2-a, R – a-c
c. P- 2-a, Q- 1-b, R –3-c d. P- 2-b, Q- 3-c, R – 1-a

[GATE 2008]

104. Match the following

Mining Method	Operation
P Bord and Pillar	1. Longhole radial drilling
Q Sublevel caving	2. Splitting and Slicing
R Longwall retreating	3. Loosening under strata pressure
S Integrated Caving	4. Mechanical cutting

a. P- 1, Q- 4, R – 3, S-2 b. P- 2, Q- 3, R – 1, S-4
c. P- 4, Q- 2, R – 3, S-1 d. P- 2, Q- 1, R – 4, S-3

[GATE 2008]

105. The wt% of solids in a sand-water stowing pipe is 60. If the solids density is 3000 kg/m^3, the pulp density of the slurry in kg/m^3 is
   a. 1380 b. 1420
   c. 1560 d. 1670

[GATE 2008]

106. An SDL working at different faces gives the following performance

Operating face	Production per blast (tonne)	Muck clearance time (hrs)
Development face	16	1.25
Splitting face	17	1.20
Slicing face	18	1.30
Heightening face	19	1.40

In 5 hrs operation maximum output is obtainable from the
a. Heightening face b. Slicing face
c. Development face d. Splitting face
[GATE 2008]

107. In longwall mining subsidence phenomenon, the "angle of break" is the angle between
a. The vertical line at the panel edge and line connecting the panel edge and zero subsidence on the surface
b. The vertical line at the panel edge and line connecting the panel edge and point of critical deformation on the surface
c. The vertical line at the panel edge and line connecting the panel edge and point of the maximum tensile strain on the surface
d. The horizontal line and the line connecting the panel edge and zero subsidence on the surface
[GATE 2009]

108. Resuing stoping method is adopted when ore body is
a. Flat and thick b. Very steep and thick
c. Flat and thin d. Very steep and thin
[GATE 2009]

109. Pocket and wing technique of pillar extraction is relevant to
a. Room and pillar method b. Bord and pillar method
c. Wongawilli method d. Shortwall method
[GATE 2009]

Mine Surveying

110. An area of 100 m^2 is measured on a plan having a R.F. of 1/800. If the R.F. were to be 1/2000, the area in m^2 would be
a. 16 b. 40
c. 250 d. 625
[GATE 2009]

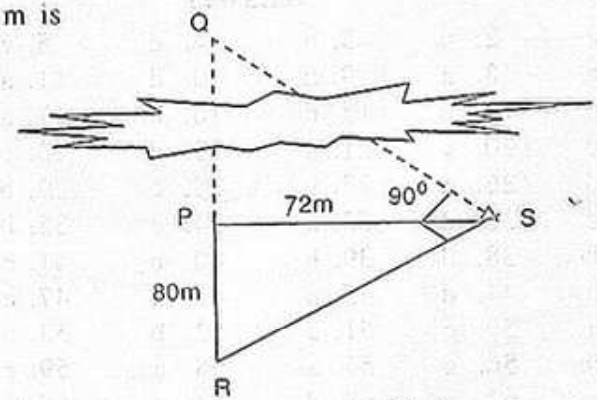
111. A tacheometer is set up at a station B. The R.L. of the station B is 150 m above the MSL. By holding a staff vertically at a station A, the following readings are taken:

Vertical Angle	Staff Readings (m)		
26° 36'	Lower	Middle	Upper
	0.80	3.08	5.36

are 100 and 1.9 respectively. The horizontal distance between the stations A and B in m is
a. 364.6 b. 366.3
c. 409.4 d. 457.6
[GATE 2009]

112. In the above problem, if the height of the instrument is 1.2 m, the RL of the station A above the MSL in m is
a. 337.6 b. 334.5
c. 331.5 d. 330.3
[GATE 2009]

113. In the figure shown below, the distances RP and PS are measured to be 80 m, and 72 m respectively. The distance PQ in m is



a. 60.4 b. 66.4
c. 64.8 d. 68.4
[GATE 2008]

114. A 30 m tape has an error of ± 0.005. If a length of 1500 m is measured with this tape, the expected total error made in the measurement in m is
a. ± 0.025 b. ± 0.030
c. ± 0.035 d. ± 0.040
[GATE 2008]

115. Match the following :

Instrument	Principal features	Application
P Tilting level	1 Micrometer	a Levelling
Q Microoptic theodolite	2 Magnetic needle	b Traversing
R Telescopic alidade	3 U-tube	c Azimuth (Bearing)
S Compass	4 Plane table surveying	d Contouring

a. P-1-b, Q-2-c, R-4-a, S-3-d
b. P-4-b, Q-3-a, R-1-c, S-2-d
c. P-2-c, Q-3-b, R-4-a, S-1-d
d. P-3-a, Q-1-b, R-4-d, S-2-c

[GATE 2008]

116. The local mean time at longitude 75°30' is 8hr. 45 min. The corresponding standard time with reference to 82°30' meridian is

a. 8 hr. 13 min
b. 9 hr. 13 min
c. 9 hr. 17 min
d. 10 hr. 17 min

[GATE 2008]

Answer

1. c	2. b	3. b	4. d	5. c	6. d
7. a	8. a	9. d	10. d	11. a	12. a
13. c	14. c	15. d	16. b	17. a	18. c
19. c	20. c	21. a	22. d	23. b	24. b
25. d	26. a	27. b	28. c	29. b	30. d
31. d	32. d	33. a	34. c	35. b	36. c
37. b	38. d	39. b	40. b	41. a	42. b
43. b	44. d	45. a	46. a	47. c	48. d
49. a	50. c	51. a	52. b	53. a	54. c
55. b	56. c	57. a	58. a	59. c	60. a
61. c	62. a	63. d	64. c	65. c	66. d
67. a	68. d	69. a	70. c	71. d	72. b
73. c	74. a	75. c	76. d	77. b	78. a
79. b	80. a	81. a	82. b	83. b	84. a
85. c	86. a	87. a	88. a	89. a	90. b
91. b	92. a	93. d	94. b	95. d	96. d
97. c	98. d	99. b	100. c	101. a	102. b
103. d	104. d	105. d	106. a	107. d	108. a
109. b	110. b	111. d	112. c	113. c	114. d
115. b					

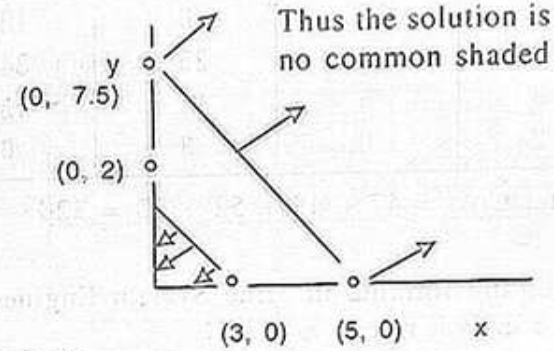
Hints :

3. Total Number of Lemons Production = z = (50 + x)*(800 − 10x)

For maximize dz/dx = 0 = (50 + x)*(-10) + (800-10x)*1

⇒ x = 15

4. Soln.



Thus the solution is infeasible as there is no common shaded region.

5. Soln.

Task \ Machine	M1	M2	M3	M4
T1	61	92	52	72
T2	42	49	69	85
T3	47	59	80	71
T4	65	70	68	72

Refer to Assignment problem solution method :

Step I :

9	40	0	20	Min Row = 52
0	7	27	43	Min Row = 42
0	12	13	24	Min Row = 47
0	5	3	7	Min Row = 65

Step II :

9	40	0	20
0	7	27	43
0	12	13	24
0	5	3	7
Min Column = 0	Min Column = 5	Min Column = 0	Min Column = 7

Step III :

9	35	0	13
0	2	27	36
0	7	13	17
0	0	3	0

Step IV :

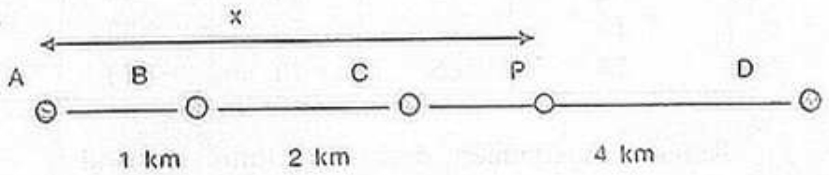
11	35	0	13
0	0	25	34
0	5	11	15
2	0	3	0

Total Cost = 47 + 49 + 52 + 72 = 220

6. Refer to scheduling formula in Mine System Engineering
Mean effective arrival rate = λ = 1/10
Mean service rate = μ = 1/6
Expected waiting time per customer in the system = Ws = λ / μ = 6/10 = 0.6

7. The average time spent by a workman waiting for his turn = Wq = $\lambda / \mu (\mu - \lambda)$ = 9

10.



Assuming the ideal distance of washery from mine A is x to minimise the transportation cost.

⇨ 7x= 26

⇨ x = 26/7 = 3.71 km

12. Soln. Z (0, 0) = 0

Z (12, 0) = 1600*12 + 1200*0 = 19200

Z (8, 6) = 1600*8 + 1200*6 = 12800 + 7200 = 20000

Z (3, 13) = 1600*3 + 1200*13 = 4800 + 15600 = 20400

Thus, Maximum Z = 20400

13. Refer to the chapter of Mine system engineering

Reliability of a system connected in series = R*R = 0.49

⇨ R = 0.7

Reliability of a system connected in parallel = 1 – (1-R)*(1–R) = 1– 0.3*0.3 = 0.91

14. Soln. For details refer to Mine System Engineering chapter

Source \ Destination	1	2	3	
1	(10) 2	(5) 2	(0) 8	15
2	(0) 1	(20) 5	(20) 7	40
3	(0) 6	(0) 4	(20) 3	20
	10	25	40	Supply / Demand

Total cost of transportation = 10*2 + 5*2 + 20*5 + 20*7 + 20*3 = 330

15. Soln.

Source \ Destination	1	2	3	
1		15		15
2	10	10	20	40
3			20	20
	10	25	40	Supply / Demand

Optimal cost of transportation = 15*2 + 10*1 + 10*5 + 20*7 + 20*3 = 290

Therefore, Cost saving = 330 – 290 = 40

16. Soln. The minimum cost of the project =
Total cost + Crashing cost (Minimum for activity C)
= 1200 + 5*20 = 1300

17. Soln. Salvage value after 5 years = 0
Therefore, Annual depreciation = 1/5*100 = 20

18. Soln.

Item	Amount (Rs. in lakhs)
Revenue	900
Cost (other than depreciation)	300
Depreciation	100
Profit before tax	500
Corporate Tax (50% profit before tax)	250
Operating Cash Inflow	[illegible]

19. Soln. For shaft excavation become more economical

$2.64D + 34.8 \leq 0.96\ L$

=> $2.64D + 34.8 \leq 0.96 * 3D$ [for L/D = 3]

=> $0.24\ D \geq 34.8$

=> $D \geq 145$

20. Soln. Net Cash Flow

= Sale Price of ore - Total cost of waste handling - Total cost of ore handling

= 400*5*500–10*600*100 - 400*5*150 = Rs.100000 = 1 lacs

21. Soln. For details, refer to chapter of Mineral Economics

Minimum annuity x at which the investment becomes viable can be determined by

NPV ≥ 0

$$\Rightarrow -200 + x * \left[\frac{(1+i)^n - 1}{i(1+i)^n}\right] \geq 0$$

$\Rightarrow -200 + x*0.4641/0.1*1.4641 \geq 0$

$\Rightarrow x \geq 63$

22. Dispersion is a measure of Range, Variance and Standard Deviation.

23. See the normal distribution curve in Geostatistics chapter.

25. Soln.

The 95 % confidence interval of daily production = Mean ± $1.96\sigma/\sqrt{n}$

$= 1200 \pm 1.96*100/\sqrt{50} = 1200 \pm 27.71$

26. Soln.

The combined mean (μ) = $\mu_1 + \mu_2 = 3 + 2.5 = 5.5$

The combined Standard Deviation (σ) = $\sqrt{Combined\ Variation}$

$= \sqrt{Var(Alu\ min\ a) + Var(Silica) + Co.Var\ (Alu\ min\ a, Silica)}$

$= \sqrt{\sigma_1^2 + \sigma_2^2 + 0} = \sqrt{0.5^2 + 0.8^2} = 0.94$

28. Soln.

Composite value of ore grade

$$\frac{L_1 * g_1 + L_2 * g_2 + L_2 * g_2 + L_2 * g_4}{3+2+3+2}$$

31. Soln.

The effective mean grade of thedeposit in Fe percentage can be determined by

(W*63 – 0.3W*72) = 0.7W*G

=> G = 59.1

32. Let, x be the percentage reserve that still remains at the end of 21st year

Therefore,

$$x = \frac{[340 - (4 + 1.1*4 + ...4*1.1^{20})]}{340} * 100 = 24.7 \approx 25Mt$$

33. The cumulative probability of injury due to fall of roof andfall of person =

$$\frac{\text{Number of injury due to Fallofroof+ Fall of person}}{\text{Total number ofinjury}}$$

= 49/75 = 0.65

34 **Determination of percentile :**

Percentile score of rth sample =

$$\frac{\text{No. of sample scored less than or equal to a rth sample}}{\text{Total no.of Sample}} * 100$$

$\Rightarrow 50 = \frac{n}{10} * 100$

$\Rightarrow n = 5$

Thus, the number of mines falling below or equal to 50th percentile = 5

And hence the number of mines falling above the 50th percentile = 10 – (5 + 1) = 4

35. Since, pH + pOH = 14

=> pH + 11 = 14 [pOH = – log[OH⁻]] = – log 10^{-11} = 11]

=> pH = 3

39. Respirable size ≤ 10 μm

Concentration of respirable fraction of dust in mg/m³ = 10*20/100 = 2mg/m³

40. Respiratory quotient of breathing
= 4.5/5 = 0.9

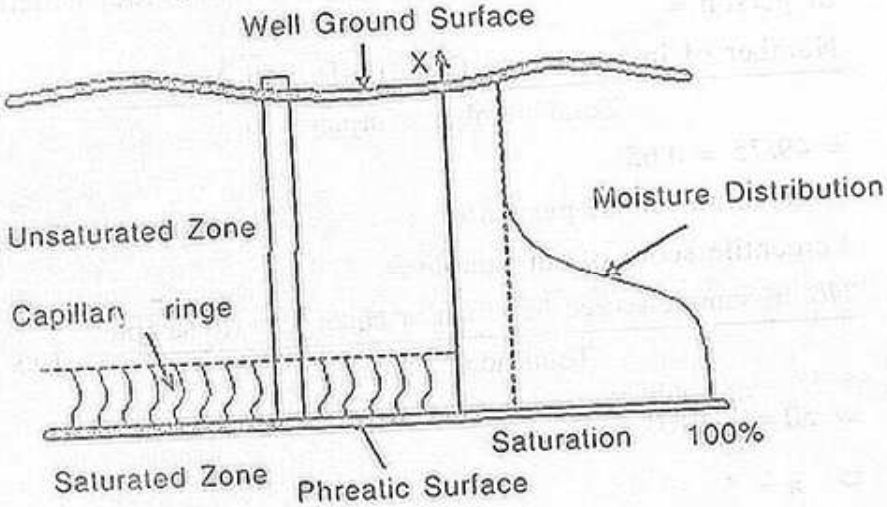
41. Do concentration of the mixture

$$= \frac{C_1Q_1 + C_2Q_2}{Q_1 + Q_2} = \frac{1.5*2 + 7*6}{1.5+7} = 5.3$$

42. Initial Oxygen deficit = Saturated value – DO concentration in the mixture

= 9.3- 5.3 = 4 mg/l

43. The phreatic surface (water table), defined as the surface at every point of which the pressure in the water is atmospheric, separates between the saturated and the unsaturated zones. The unsaturated zone is bounded from above by ground surface, and from below by the water table (= phreatic surface).



45. Total pressure generated by the fan = Velocity pressure + Flow pressure

$= (½ \rho v^2) + RQ^2$

$= 1/2 * 1.2*(2/0.3)^2 + 40*2^2$

$= 26.67 + 160 = 186.67$

46. We have,

$P = R_1Q_1^2 = (R_2 + R)\ Q_2^2$

$0.5*15^2 = (0.8 + R)*10^2$

$R = 0.325\ Ns^2m^{-8}$

Size of the regulator = $= 1.2/\sqrt{R} = 1.2\sqrt{0.325} = 2.10\ m^2$

=> 75000 0.7

=> Pr = 500 Pa

48. We have, $\frac{P_1}{P_2} = \frac{n_1^2 * Q_1^2}{n_2^2 * Q_2^2}$

$\Rightarrow \frac{500}{500+200} = \frac{280^2 * 105^2}{n^2 * 105^2}$

$\Rightarrow n = 332$ rpm

53. We have, $P = R_1Q_1^2 = R_2\ Q_2^2$

$\Rightarrow 0.5*20^2 = 2*Q_2^2$

$\Rightarrow Q_2^2 = 100$

$\Rightarrow Q_2 = 10$

Therefore, Total air flow through mine (Q) = $Q_1 + Q_2 = 20 + 10 = 30\ m^3/s$

54. Let, equivalent resistance of the two splits is R

Therefore, $= 1/\sqrt{R} = 1/\sqrt{R_1} + 1/\sqrt{R_2}$ [Splits are in parallel]

$\Rightarrow 1/\sqrt{R} = 1/\sqrt{0.5} + 1/\sqrt{2}$

$\Rightarrow R = 0.22$

Total resistance of the mine (R_e) = Equivalent resistance of the splits + Combined resistance of trunk roadways and shaft [For series connection]

= 0.22 + 0.7 = 0.92

We have, Total air power of the system = $R_eQ^3 = 0.92*30^3$ = 24.9 kW

56. We have,

Powder factor = Ore production (te) per hole/ Explosive required (kg) per hole

= 12*4*5*2.6/120 = 5.2 te/kg

58. We have,

Break-even stripping ratio =

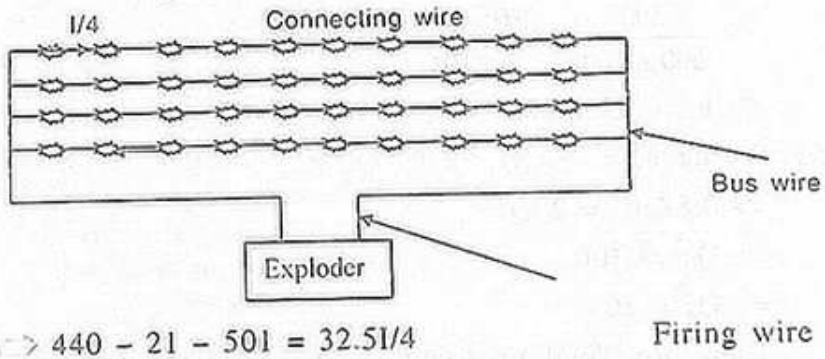
$$\frac{\text{Recoverable price per te of ore} - [\text{O.C. prodction cost per te of ore} + \text{Profit Margin}]}{\text{Cost of extraction per cubic meter of overburden material}}$$

= [600 – (400 + 0)] / 50 = 4:1

59. Injury rate = 1000*16/800 = 20 per 1000 persons employed

60. We have, $V - (R_1+R_2)*I = R_3*I/4$

Where, I = Current flow through blasting circuit
R_1 = Resistance of firing wire
R_2 = Resistance of bus wire
R_3 = Resistance of connecting wire and detonator



⇒ 440 – 21 – 501 = 32.5I/4

⇒ I = 7.31 Amp

61. We have, $V = 120(\sqrt{SD})^{-1.0}$

Where, SD = Square root scaled distance

Again, Scaled Distance = S.D = $D/\sqrt{Q}$

Where, D = distance of the measuring transducer
Q = Maximum charge weight per delay

Therefore, $= 25 = 120*(100/\sqrt{Q})^{-1}$

⇒ Q = 434 kg

62. Output per blast =

No. of Holes *Spacing * Burden *(Length of holes – Subgrade length)* Density

= 9*6*5*(8 – 1)*2.43 = 4592.7 ≈ 4593 te

63. We have, Powder Factor =

Output (te) per round of blasting/ Explosive (kg) required per round of blasting

Explosive (kg) required per round of blasting =

$\pi/4*d^2$*No. of Hole*(Length of blast hole – Stemming length)*Explosive density

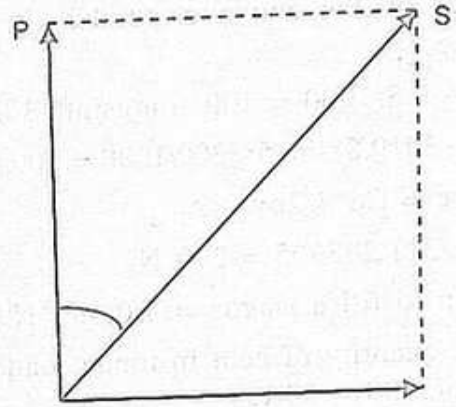
= 3.14*0.2*0.2*9*6*900/4 = 1526.04

66. We have,

10000 + 400T ≥ 1200T

⇒ T = 10000/800 = 12.5 hrs.

67. It is common to define the Power Factor - *PF* - as the cosine of the phase angle between voltage and current - or the "*cosφ*".



318

The power factor defined by IEEE and IEC is the ratio between the applied active (true) power - and the apparent power, and can in general be expressed as:

PF = P / S = P/VA

Where,

PF = power factor
P = active (true or real) power (Watts)
S = apparent power (VA, volts amps)

For 3-Phase power supply
Power factor $= P/\sqrt{3}\,VI$
Thus, $= PF = 185000/\sqrt{3}*1100*115 = 0.84$

68. We have, Input Power*Pump Efficiency*Motor Efficiency = Head*Quantity

⇒ $P*0.7*0.9 = (100*1000*9.8)*(2100*10^{-3}/60)$

⇒ P = 54.5 kW

69. We know, $Q = A_1v_1 = A_2v_2$

=> $0.035 = 3.14*d_1^2*1.8/4 = 3.14*d_2^2*2.5/4$

=> d_1 = 0.1573 m = 15.73 cm

And, d_2 = 0.1335 m = 13.35 cm

73. We have, P = F.V

=> 308000 = F*7

=> F = 44000 N

Therefore, τ = F*r = 44000*2.5 = 110 kNm

74. Carrying capacity of the belt (T) = abv = 0.09*1.5*2 = 0.27 te/s = 972 te/hr

75. Heat added by the conveyor to the air = Rated Power – Work done

= 100 – F.S/1000 = 100 – mgsinθ*300/1000

= 100 – 30*9.8*sin15°*300/1000 = 100 – 22.82 = 77.2 kW

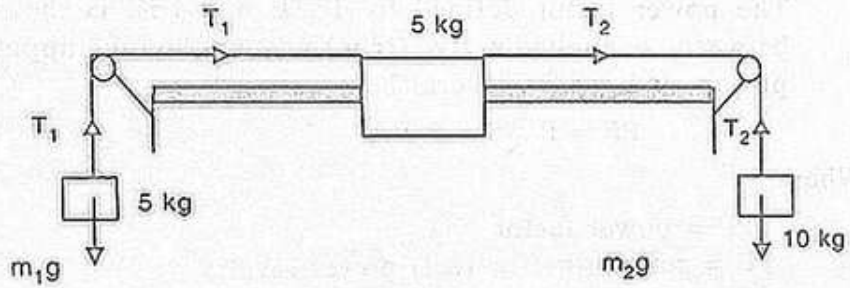
76. Drag Force = Cd*1/2ρv^2 *A

= 2.5*1/2*1.2*36*5 = 270 N

77. Time taken to fill a wagon = 8/0.48 *18/5 – 24 = 36 s

Therefore, quantity of coal in tonne loaded in each wagon = 6000*36/3600 = 60 te

78. Free body force diagram of the situation is as



Thus, we can write

$T_1 - m_1g = m_1a$ => $T_1 - 5g = 5a$

$m_2g - T_2 = m_2a$ => $10g - T_2 = 10a$

$T_2 - T_1 - \mu m_3g = m_3a$ => $T_2 - T_1 - 0.2*5g = 5a$

Solving the above equations, we get

$T_1 = 6g$

$T_2 = 8g$

$a = g/5$

79. Draw-bar-pull= Tractive effort–Force required to move the loco

= 0.25*10000*9.8 - 10000*9.8*10/100 = 14.7 kN

80. Draw-bar-pull = Tractive effort – Running resistance

= 0.25*10000*9.8 – 10000*9.8*10/100 – 10000*9.8*sin5° =

83. Void ratio (Vr)=Volume of Void (Vv)/Volume of Sample (Vs)

=> Vr = (Vs–Vg)/ Vs = 1–Vg/Vs = 1 – $(m/\rho_g)/(m/\rho_s)$

[Mass of air is very less in comparison to mass of grain and hence neglected]

=> Vr = 1– ρ_s/ρ_g = 1– 2.7/3.0 = 0.1 = 10%

84. We have,

$$\sigma_r = \frac{2c.\cos\varphi}{1-\sin\varphi}$$

and, $$\sigma_t = \frac{2c.\cos\varphi}{1+\sin\varphi}$$

Given that, $\frac{\sigma_r}{\sigma_t} = 8$

$$\Rightarrow Sin\ \phi = \frac{7}{9}$$

$$\Rightarrow \phi = 51^0$$

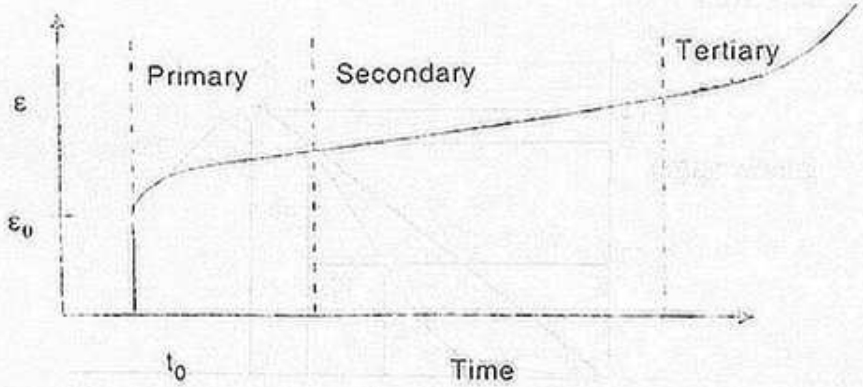
85. WE have, $\sigma_{00} = P(1+k)+(1-K)2\cos 2\theta$

=> $3P_0 = P_0$ [(1 + K) + (1- K)*2*cos90°]

=> 3 = 1+ K

=> K = 2

86. Strain as a function of time due to constant stress over an extended period for a viscoelastic material.



88. Soln P - Wave

[Note : P-waves : P waves (primary waves) are longitudinal or compressional waves. In solids, these waves generally travel almost twice as fast as S waves and can travel through any type of material.

S-waves : S waves (secondary waves) are transverse or shear waves, which means that the ground is displaced perpendicularly to the direction of propagation. Their speed is about 60% of that of P waves in a given material.

Surface waves : Surface waves are analogous to water waves and travel along the Earth's surface. They travel slower than body waves. Because of their low frequency, long duration, and large amplitude, they can be the most destructive type of seismic wave. There are two types of surface waves: Rayleigh waves and Love waves.

Rayleigh waves : Rayleigh waves, also called ground roll, are surface waves that travel as ripples with motions that are similar to those of waves on the surface of water. They are slower than body waves, roughly 90% of the velocity of S waves for typical homogeneous elastic media.

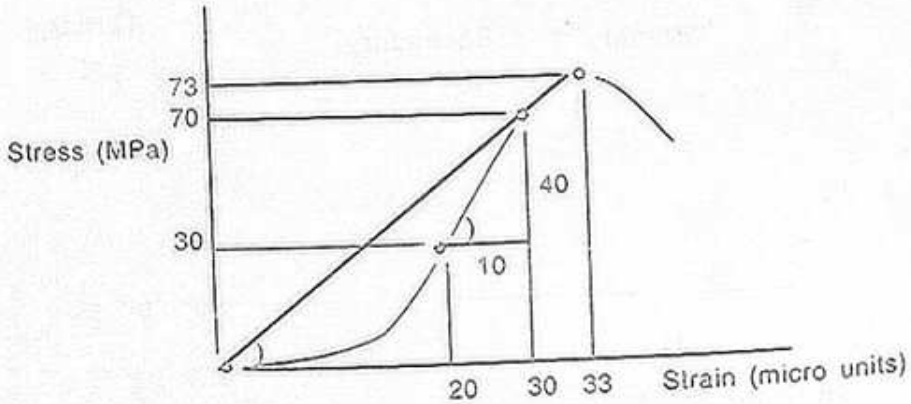
Love Waves : Love waves are surface waves that cause horizontal shearing of the ground. They usually travel slightly faster than Rayleigh waves, about 90% of the S wave velocity. They are the slowest and have the largest amplitude]

91. We have,

$$\frac{\sigma_h}{\sigma_v} = \frac{\nu}{1-\nu}$$

$\Rightarrow \sigma_h = (0.2/1\text{-}0.2)*10 = 2.5$ MPa

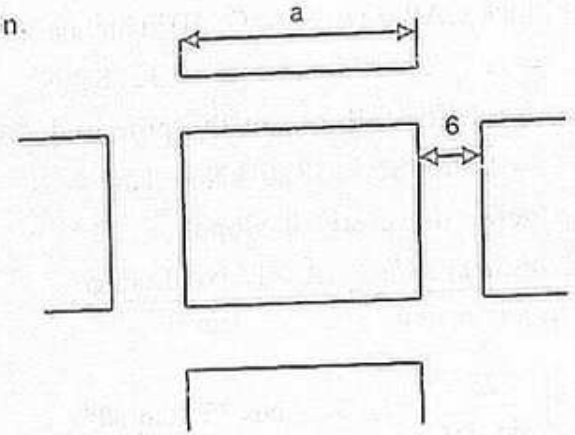
92. Soln.



Tangent Modulus = 40/10 = 4

Secant Modulus = 73/33 2.2

93. Soln.



Compressive strength of pillar = $15.5*10^6* a^2$

Load acting on pillar = $300*(a + 6)^2*24.5*10^3$

F.O.S. = Compressive Strength of pillar/ Load acting on pillar = 1

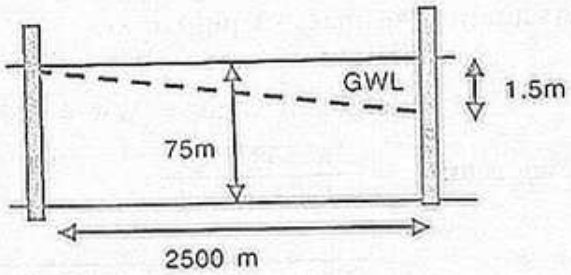
$\Rightarrow 15.5*10^6* a^2 = 300*(a + 6)^2*24.5*10^3$

$\Rightarrow a^2/(a + 6)^2 = 0.4742$

We have, Extraction ration (R) = $[1- a*a/ (a + 6)^2] *100 = (1 - 0.4742)*100$

$= 52.58 \approx 53$

94. Soln.

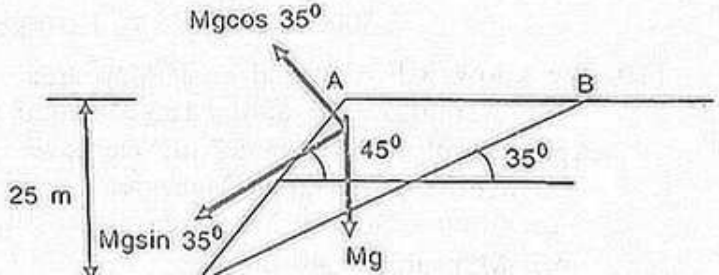


From Darcy's Law, we have

$Q = k*i*A$

$= 40*(1.5/2500)*(75*1) = 1.8$

95. Soln.



Weight of block OAB = (Area of OAB)*Thickness * unit weight

$= ½ * 25*10.7*1*24.5 = 3278$ kN

Therefore, force for failure, on the potential failure plane

$= W\sin 35° = 1870$ kN $= 1.87$ MN

96. We have, factor of safety of slope

$$= \frac{\text{Force resist to slide}}{\text{Force for failure}} = \frac{cA + W.\cos\beta.\tan\varphi}{w\sin\beta}$$

$$= \frac{0.02*\left(\frac{25}{\sin 35^2}\right)*1 + 3.28.\cos 35^0.\tan 30^0}{3.28.\sin 35^0}$$

$= (0.87 + 1.553)/1.88 = 1.3$

100. Let, In-situ Volume = x

Therefore, Output volume = 1.4x

Percent of broken ore = $(0.4x/1.4x)*100 = 28.57 \approx 29$

105. Given that,

Weight% of solids in a sand-water stowing pipe = 60

Therefore, weight % of water in a sand-water stowing pipe = 40

Assuming the mass of pulp = x

Weight of sand = Ws = 0.6x

Weight of water = Ww = 0.4x

$$\text{Pulp density} = \frac{\text{Mass of Pulp}}{\text{Volume of Pulp}}$$

$$= \frac{x}{\text{Volume of Water} + \text{Volume of Sand}}$$

$$= \frac{x}{\frac{0.4x}{1000} + \frac{0.6x}{3000}}$$

$= 3000/1.8 = 1666.7 \approx 1670$ kg/m^3

110. We know, R.F = Actual area/ Map area

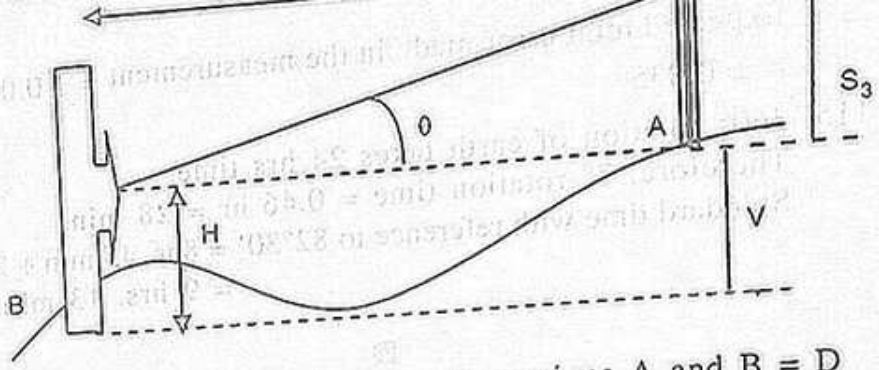
=> Actual area = 800 × 100 = 80000 m^2

Again, applying the same rule we have,

R.F = Actual area/ Map area

=> 2000 = 80000/ Map area

=> Map area = 40 m^2



Horizontal distance between the stations A and B = D

$= f / i * S \cos^2\theta + (f + K)\cos\theta$

$= 100*(5.36 - 0.8)*\cos^2(26.6°) + 1.9*\cos 26.6°$

$= 364.57 + 1.7 = 366.3$ m

112. Vertical height of point A = V =

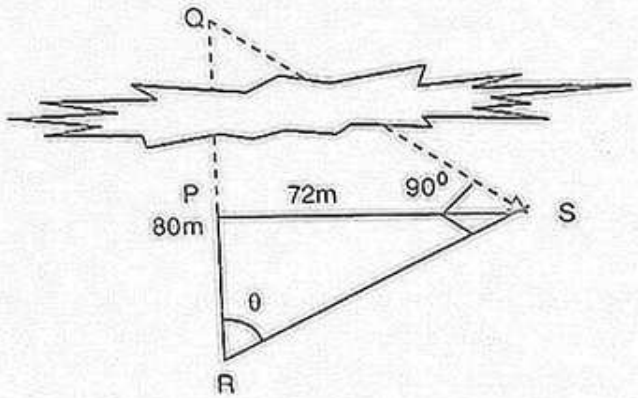
$H + f / i * S\cos\theta * \sin\theta + (f + K)\sin\theta - S_3$

$= 1.2 + 100*(5.36 - 0.8)*\cos 26.6°*\sin 26.6° + 1.9*\sin 26.6° - 3.08$

$= 1.2 - 3.08 + 458*0.4 + 0.85 = 182.17$

Therefore, R.L. of station A = R.L. of B + V = 150 + 182.17 = 332.17 m

113



From right angle triangle PRS

RS = 107.62 m

From similar triangle PRS and QRS, we get

PR/RS = RS/QR

=> 80/107.62 = 107.62/ (PQ+ 80)

=> PQ + 80 = 144.78

=> PQ = 64.78 ≈ 64.8

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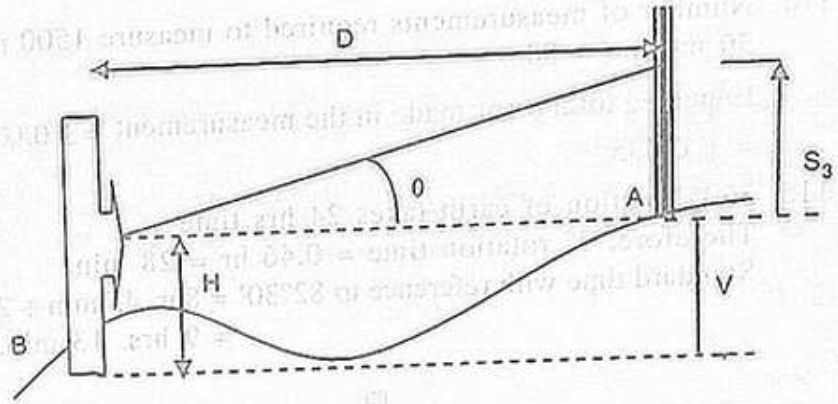

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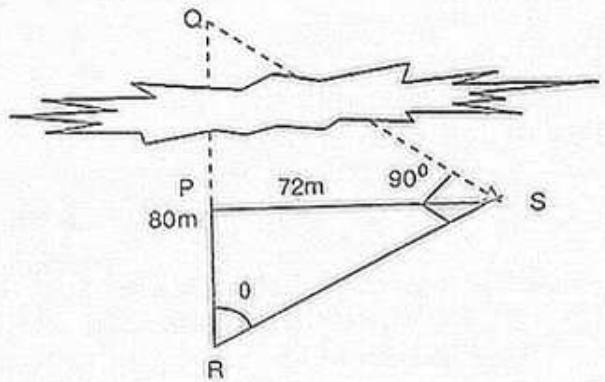

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From similar triangle PRS and QRS, we get

PR/RS = RS/QR

=> 80/107.62 = 107.62/ (PQ+ 80)

=> PQ + 80 = 144.78

=> PQ = 64.78 ≈ 64.8

114. Number of measurements required to measure 1500 m with 30 m tape = 50

Expected total error made in the measurement $= \pm 0.005\sqrt{50}$

$= \pm 0.035$

115. 360° rotation of earth takes 24 hrs time

Therefore, 7° rotation time = 0.46 hr = 28 min.

Standard time with reference to 82°30' = 8hr. 45 min + 28 min

= 9 hrs. 13 min.

□