

THIRD EDITION



# COAL GEOLOGY

LARRY THOMAS

WILEY Blackwell

## Coal Geology

# Coal Geology

*Third Edition*

*Larry Thomas*

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## Contents

	<b>Preface to Third Edition</b>	<i>xv</i>
	<b>Preface to Second Edition</b>	<i>xvii</i>
	<b>Preface to First Edition</b>	<i>xix</i>
	<b>List of Acronyms</b>	<i>xxi</i>
<b>1</b>	<b>Preview</b>	<i>1</i>
1.1	Scope	<i>1</i>
1.2	Coal Geology	<i>1</i>
1.3	Coal Use	<i>2</i>
1.4	Background	<i>2</i>
<b>2</b>	<b>Origin of Coal</b>	<i>5</i>
2.1	Introduction	<i>5</i>
2.2	Sedimentation of Coal and Coal-Bearing Sequences	<i>5</i>
2.2.1	Depositional Models	<i>5</i>
2.2.2	The Traditional Model	<i>6</i>
2.2.2.1	Prodelta and Delta Front Facies	<i>8</i>
2.2.2.2	Lower Delta Plain Facies	<i>8</i>
2.2.2.3	Upper Delta Plain Facies	<i>11</i>
2.2.2.4	Fluvial Facies	<i>11</i>
2.2.3	Modern Peat Analogues	<i>11</i>
2.2.3.1	Palaeobotanical Composition of Ancient Mires	<i>19</i>
2.2.3.2	Case Studies	<i>24</i>
2.2.4	Sequence Stratigraphy	<i>25</i>
2.2.5	Facies Correlation	<i>27</i>
2.2.6	Facies Maps	<i>29</i>
2.2.6.1	Seam Splitting	<i>31</i>
2.2.6.2	Washouts	<i>34</i>
2.2.6.3	Floor Rolls	<i>34</i>
2.2.6.4	Coal Seam Thickness Variations	<i>35</i>
2.2.6.5	Interburden/Overburden Thickness	<i>37</i>
2.2.6.6	Coal Seam Quality Variations	<i>38</i>
2.3	Structural Effects on Coal	<i>40</i>
2.3.1	Syn depositional Effects	<i>40</i>
2.3.1.1	Microstructural Effects	<i>40</i>
2.3.1.2	Macrostructural Effects	<i>41</i>

2.3.2	Post-Depositional Effects	44
2.3.2.1	Jointing/Cleats in Coal	44
2.3.2.2	Faulting	45
2.3.2.3	Folding	50
2.3.2.4	Igneous Associations	52
2.3.2.5	Mineral Precipitates	53
<b>3</b>	<b>Age and Occurrence of Coal</b>	<b>57</b>
3.1	Introduction	57
3.2	Plate Tectonics	57
3.3	Stratigraphy	61
3.4	Age and Geographical Distribution of Coal	64
3.4.1	United States of America	67
3.4.2	Canada	74
3.4.3	Europe	75
3.4.3.1	Albania	75
3.4.3.2	Austria	75
3.4.3.3	Belgium	75
3.4.3.4	Bosnia	75
3.4.3.5	Bulgaria	75
3.4.3.6	Czech Republic	75
3.4.3.7	Denmark	76
3.4.3.8	France	76
3.4.3.9	Germany	76
3.4.3.10	Georgia	76
3.4.3.11	Greece	77
3.4.3.12	Greenland	77
3.4.3.13	Holland	77
3.4.3.14	Hungary	77
3.4.3.15	Ireland	77
3.4.3.16	Italy	77
3.4.3.17	Kosovo	77
3.4.3.18	Montenegro	78
3.4.3.19	Poland	78
3.4.3.20	Portugal	78
3.4.3.21	Romania	78
3.4.3.22	Serbia	78
3.4.3.23	Spain	79
3.4.3.24	Spitzbergen	79
3.4.3.25	Sweden	79
3.4.3.26	Turkey	79
3.4.3.27	United Kingdom	79
3.4.4	Africa	80
3.4.4.1	Angola	80
3.4.4.2	Botswana	80
3.4.4.3	Cameroon	81
3.4.4.4	Egypt	81
3.4.4.5	Ethiopia	81

- 3.4.4.6 Malagasy Republic 81
- 3.4.4.7 Malawi 81
- 3.4.4.8 Mali 81
- 3.4.4.9 Morocco 81
- 3.4.4.10 Mozambique 82
- 3.4.4.11 Namibia 82
- 3.4.4.12 Niger 82
- 3.4.4.13 Nigeria 82
- 3.4.4.14 South Africa 82
- 3.4.4.15 Swaziland 83
- 3.4.4.16 Tanzania 83
- 3.4.4.17 Zaire 83
- 3.4.4.18 Zambia 83
- 3.4.4.19 Zimbabwe 83
- 3.4.5 The Indian Subcontinent 84
  - 3.4.5.1 Afghanistan 84
  - 3.4.5.2 Bangladesh 84
  - 3.4.5.3 India 84
  - 3.4.5.4 Iran 85
  - 3.4.5.5 Pakistan 85
- 3.4.6 Central and South America 85
  - 3.4.6.1 Argentina 85
  - 3.4.6.2 Bolivia 86
  - 3.4.6.3 Brazil 86
  - 3.4.6.4 Chile 86
  - 3.4.6.5 Colombia 87
  - 3.4.6.6 Costa Rica 87
  - 3.4.6.7 Ecuador 87
  - 3.4.6.8 Mexico 87
  - 3.4.6.9 Peru 87
  - 3.4.6.10 Uruguay 88
  - 3.4.6.11 Venezuela 88
- 3.4.7 Commonwealth of Independent States 88
  - 3.4.7.1 Kazakhstan 88
  - 3.4.7.2 Russian Federation 88
  - 3.4.7.3 Tajikistan 89
  - 3.4.7.4 Ukraine 89
  - 3.4.7.5 Uzbekistan 89
- 3.4.8 Far East 89
  - 3.4.8.1 Brunei 89
  - 3.4.8.2 Democratic Republic of (North) Korea 90
  - 3.4.8.3 Indonesia 90
  - 3.4.8.4 Japan 91
  - 3.4.8.5 Laos 91
  - 3.4.8.6 Malaysia 91
  - 3.4.8.7 Mongolia 91
  - 3.4.8.8 Myanmar (Burma) 92

- 3.4.8.9 People's Republic of China 92
- 3.4.8.10 People's Republic of Vietnam 93
- 3.4.8.11 Philippines 93
- 3.4.8.12 Republic of (South) Korea 94
- 3.4.8.13 Taiwan 94
- 3.4.8.14 Thailand 94
- 3.4.9 Australasia 95
- 3.4.9.1 Australia 95
- 3.4.9.2 New Zealand 96
- 3.4.9.3 Antarctica 96

## **4 Coal as a Substance 97**

- 4.1 Physical Description of Coal 97
  - 4.1.1 Macroscopic Description of Coal 97
    - 4.1.1.1 Humic Coals 97
    - 4.1.1.2 Sapropelic Coals 101
  - 4.1.2 Microscopic Description of Coal 102
  - 4.1.3 Mineral Content of Coals 106
  - 4.1.4 Coal Petrography 113
- 4.2 Coalification (Rank) 116
  - 4.2.1 Coalification 116
  - 4.2.2 Causes of Coalification 118
    - 4.2.2.1 Temperature 120
    - 4.2.2.2 Time 120
    - 4.2.2.3 Pressure 120
    - 4.2.2.4 Radioactivity 121
- 4.3 Coal Quality 121
  - 4.3.1 Chemical Properties of Coal 122
    - 4.3.1.1 Basis of Analytical Data 122
    - 4.3.1.2 Proximate Analysis 123
    - 4.3.1.3 Ultimate Analysis 125
    - 4.3.1.4 Other Analysis 126
  - 4.3.2 Combustion Properties of Coal 127
    - 4.3.2.1 Calorific Value 127
    - 4.3.2.2 Ash Fusion Temperatures 128
    - 4.3.2.3 Caking Tests 128
    - 4.3.2.4 Coking Tests 129
  - 4.3.3 Physical Properties of Coal 131
    - 4.3.3.1 Mechanical Strength 131
    - 4.3.3.2 Density 132
    - 4.3.3.3 Hardness and Grindability 132
    - 4.3.3.4 Abrasion Index 133
    - 4.3.3.5 Particle Size Distribution 133
    - 4.3.3.6 Float–Sink Tests 133
- 4.3.4 Coal Oxidation 135
- 4.4 Classification of Coals 136
  - 4.4.1 North America 136

- 4.4.2 United Kingdom 136
- 4.4.3 Europe 137
- 4.4.4 Australia 146
- 4.4.5 South Africa 146
- 4.4.6 United Nations 146
- 4.4.7 Russia 148
- 4.4.8 People's Republic of China 149

## **5 Coal Sampling and Analysis 151**

- 5.1 Coal Sampling 151
  - 5.1.1 In-Situ Coal Sampling 151
    - 5.1.1.1 Grab Samples 151
    - 5.1.1.2 Channel Samples 151
    - 5.1.1.3 Pillar Samples 154
    - 5.1.1.4 Core Samples 154
    - 5.1.1.5 Cuttings Samples 155
    - 5.1.1.6 Specimen Samples 155
    - 5.1.1.7 Bulk Samples 156
    - 5.1.1.8 Sample Storage 156
  - 5.1.2 Ex-Situ Sampling 157
- 5.2 Coal Analysis 162
  - 5.2.1 Outcrop/Core Samples 162
  - 5.2.2 Bulk Samples 162
  - 5.2.3 Ex-Situ Samples 162

## **6 Coal Exploration and Data Collection 169**

- 6.1 Introduction 169
- 6.2 Field Techniques 169
  - 6.2.1 Outcrop Mapping 172
  - 6.2.2 Global Positioning System 179
  - 6.2.3 Portable Personal Computers 179
  - 6.2.4 Remote Sensing 180
    - 6.2.4.1 Satellite Imagery 180
    - 6.2.4.2 Airborne Imagery 181
- 6.3 Drilling 183
  - 6.3.1 Openhole Drilling 184
  - 6.3.2 Core Drilling 188
  - 6.3.3 Portable Drilling 189
  - 6.3.4 Core and Openhole Logging 190
    - 6.3.4.1 Core Logging 190
    - 6.3.4.2 Openhole Logging 193
- 6.4 Geotechnical Properties 194
  - 6.4.1 Strength 195
  - 6.4.2 Weathering 196
  - 6.4.3 Texture and Structure 196
  - 6.4.4 Colour 196
  - 6.4.5 Grain Size 198

- 6.4.6 Total Core Recovery 198
- 6.4.7 Solid Core Recovery 198
- 6.4.8 Rock Quality Designation 198
- 6.4.9 Fracture Spacing Index 198
- 6.4.10 Fracture Logging 199
- 6.4.11 Rock Mass Rating 201
- 6.5 Computer Applications 201

## **7 Coal Resources and Reserves 207**

- 7.1 Introduction 207
- 7.2 Coal Resources and Reserves Classification 208
  - 7.2.1 Australia 209
    - 7.2.1.1 Coal Resources 209
    - 7.2.1.2 Coal Reserves 211
  - 7.2.2 Canada 211
  - 7.2.3 Europe (Including the UK) 212
  - 7.2.4 South Africa 213
  - 7.2.5 United Nations 213
  - 7.2.6 United States of America 216
  - 7.2.7 Russian Federation 219
  - 7.2.8 People's Republic of China 222
  - 7.2.9 India 222
  - 7.2.10 Other Countries 224
- 7.3 Reporting of Resources/Reserves 225
  - 7.3.1 Coal Resources and Reserves 225
  - 7.3.2 Coal Resources and Reserves Maps 226
  - 7.3.3 Calculation of Coal Resources 227
    - 7.3.3.1 In-Situ Tonnage Calculations 227
    - 7.3.3.2 Geostatistics and Computer Modelling 229
    - 7.3.3.3 Opencast Coal Mining 232
    - 7.3.3.4 Geological Losses 233
    - 7.3.3.5 Reserves Reporting 235
    - 7.3.3.6 Reserve Economics 235
- 7.4 World Coal Reserves and Production 235
  - 7.4.1 World Coal Reserves 235
  - 7.4.2 World Coal Production 237
    - 7.4.2.1 Coal Production Statistics 237
    - 7.4.2.2 Regional Production and Consumption 241
    - 7.4.2.3 Reserves/Production Ratio 242

## **8 Geophysics of Coal 243**

- 8.1 Introduction 243
- 8.2 Physical Properties of Coal-Bearing Sequences 244
  - 8.2.1 Density 244
  - 8.2.2 Seismic Velocity 244
  - 8.2.3 Seismic Reflection Coefficients 245
  - 8.2.4 Magnetic Susceptibility 245

8.2.5	Electrical Conductivity	245
8.2.6	Radiometric Properties	245
8.3	Surface Geophysical Methods	246
8.3.1	Seismic Surveys	246
8.3.1.1	Seismic Reflection Surveys	246
8.3.1.2	Seismic Refraction Surveys	256
8.3.1.3	Passive Seismic Surveys	257
8.3.2	Gravity Surveys	257
8.3.3	Magnetic Surveys	259
8.3.4	Electrical Methods	262
8.3.4.1	Electrical Resistivity Methods	262
8.3.4.2	Ground-Penetrating Radar Methods	262
8.3.4.3	Electromagnetic Surveys	263
8.3.5	Radioactive Methods	264
8.4	Underground Geophysical Methods	264
8.4.1	In-Seam Seismic Surveys	264
8.4.2	Underground Gravity Surveys	269
8.4.3	Ground-Penetrating Radar Techniques	269
8.5	Geophysical Borehole Logging	269
8.5.1	Radiation Logs	271
8.5.1.1	Gamma-Ray Log	271
8.5.1.2	Density Log	273
8.5.1.3	Neutron Log	274
8.5.1.4	Gamma Spectrometry	276
8.5.2	Calliper Log	276
8.5.3	Electric Logs	277
8.5.4	Dipmeter Log	277
8.5.5	Sonic Log	278
8.5.6	Acoustic Scanning Tools	279
8.5.7	Temperature Log	280
8.5.8	Advanced Interpretation	282
<b>9</b>	<b>Hydrogeology of Coal</b>	<b>289</b>
9.1	Introduction	289
9.2	The Nature of Groundwater and Surface Flow	289
9.2.1	Surface Water	289
9.2.2	Groundwater	290
9.3	Hydrogeological Characteristics of Coals and Coal-Bearing Sequences	292
9.4	Collection and Handling of Hydrogeological Data	295
9.4.1	Surface Water	295
9.4.2	Groundwater	295
9.5	Groundwater Inflows in Mines	298
9.5.1	Dewatering of Open-pit Mines	299
9.5.2	Dewatering of Underground Mines	306
9.5.3	Water Quality	307
9.6	Groundwater Rebound	307

<b>10</b>	<b>Geology and Coal Mining</b>	<b>311</b>
10.1	Introduction	311
10.2	Underground Mining	312
10.2.1	Geological Factors	313
10.2.2	Mining Methods	314
10.2.2.1	Longwall Mining	314
10.2.2.2	Room-and-Pillar Mining	316
10.2.2.3	Stress Fields	318
10.2.2.4	Coal Bursts	326
10.2.2.5	Strata and Air Temperatures	327
10.2.2.6	Spontaneous Combustion	328
10.3	Surface Mining	328
10.3.1	Geological Factors	328
10.3.2	Mining Equipment	330
10.3.2.1	Dragline	330
10.3.2.2	Powered Shovels	331
10.3.2.3	Bucketwheel Excavators	334
10.3.3	Surface Mining Methods	335
10.3.3.1	Strip Mining	335
10.3.3.2	Opencast or Open-pit Mining	335
10.3.3.3	Highwall Mining	338
10.4	Coal Production	339
10.4.1	Underground Coal Production	340
10.4.2	Surface Coal Production	340
<b>11</b>	<b>Coal as an Alternative Energy Source</b>	<b>343</b>
11.1	Introduction	343
11.2	Gas in Coal	343
11.2.1	Coal-bed Methane	345
11.2.1.1	Coal-bed Methane Generation	345
11.2.1.2	Coal-bed Methane Retention	346
11.2.1.3	Coal-bed Methane Production	349
11.3	Underground Coal Gasification	365
11.3.1	Underground Coal Gasification: The Case For and Against	365
11.3.2	Underground Coal Gasification Technology	366
11.3.2.1	Coal Gasification Reactions	366
11.3.3	Global Development of Underground Coal Gasification	373
11.4	Coal as a Liquid Fuel	375
11.4.1	Petroleum Potential of Coal	375
11.4.2	Coal Properties as an Oil-Source Rock	375
11.4.3	Coal Liquefaction Technology	378
11.4.4	Future Development of Coal Liquefaction	379
11.4.5	Coal-Sourced Oil and Gas Occurrences	381
<b>12</b>	<b>Coal Use and the Environment</b>	<b>385</b>
12.1	Introduction	385
12.2	Coalmining	386



12.2.1	Effects on Water Supply	387
12.2.1.1	Surface Water	387
12.2.1.2	Underground Water	387
12.2.2	Contamination of Mine Waters	387
12.2.3	Other Water Pollution	391
12.2.4	Run-off, Erosion, and Sedimentation	391
12.2.5	Spoil Dumping	392
12.2.6	Spontaneous Combustion	396
12.2.7	Dust Suppression	397
12.2.8	Subsidence	398
12.3	Coal Use	401
12.3.1	Electricity Generation	404
12.3.1.1	Emissions	405
12.3.1.2	Flue Gas Desulfurisation	409
12.3.1.3	Other Emission Controls	410
12.3.1.4	Fluidised-Bed Combustion	411
12.3.2	Other Major Users	413
12.3.2.1	Iron and Steel Production	413
12.3.2.2	Industrial Use	414
12.3.2.3	Domestic Use	415
12.3.3	Coal Transportation	415
12.4	Health	415
12.5	Carbon Capture and Storage	416
12.6	Environmental Regulations	418
12.6.1	Introduction	418
12.6.2	United Nations Economic Commission for Europe Conventions	419
12.6.3	European Union	420
12.6.4	World Bank	420
12.6.5	Kyoto Protocol	420
12.6.6	Copenhagen Accord	421
12.6.7	Durban Platform for Enhanced Action	421
12.6.8	Paris Agreement	421
12.7	Future Implications	422
<b>13</b>	<b>Coal Marketing</b>	<b>423</b>
13.1	Introduction	423
13.2	Coal Quality	423
13.3	Transportation	425
13.3.1	Land Transportation	425
13.3.1.1	Conveyors	426
13.3.1.2	Road	426
13.3.1.3	Rail	427
13.3.2	Water Transportation	428
13.3.2.1	Barges	428
13.3.2.2	Bulk Carriers	429
13.4	Coal Markets	430
13.5	Coal Contracts	431

- 13.5.1 Spot Purchases 431
- 13.5.2 Term Contracts 431
- 13.5.3 Indexed Contracts 432
- 13.6 Coal Price and Indexing 433

**Appendix A List of International and National Standards Used in Coal and Coke  
Analysis and Evaluation 435**

- A.1 British Standards Institution (BS) 435
- A.2 International Organization for Standardization (ISO) 438
- A.3 ASTM International, Formerly Known as American Society for Testing and Materials (ASTM) 441
- A.4 Standards Association of Australia (AS) 444
- A.5 National Standards of People's Republic of China 446
- A.6 Bureau of Indian Standards 449
- A.7 State Standards of Russia – GOST (GOST = Gosudarstvennyy Standart) 451

**Appendix B Tables of True and Apparent Dip, Slope Angles, Gradients, and  
Percentage Slope 455**

**Appendix C Calorific Values Expressed in Different Units 457**

**Appendix D Coal Statistics 463**

**Appendix E Methane Units Converter 465**

**Glossary 467**

**Bibliography 475**

**Index 497**

## Preface to Third Edition

The first and second editions of *Coal Geology* have provided the coal geologist and those associated with the coal industry with the background to the origins and characteristics of coal together with exploration techniques, including geophysics and hydrogeology. Details of coalmining techniques, resource calculations, alternative uses of coal, and environmental issues were also described.

Although broadly following the layout of the previous edition, additional information has been added to coal origins, geographical distribution of coal, and coal exploration. The chapter on coal resources and reserves has been updated with current resource classifications, together with recent world reserves/production figures.

The chapter on the alternative uses of coal, particularly coal-bed and coalmine methane extraction, have been expanded to reflect the increase in activity in these areas. Developments in environmental requirements and regulations have also been updated.

Again, numerous sources of information have been consulted, the majority of which are

listed in the bibliography section. International standards relating to coal, listed in Appendix A, have been updated and expanded to include the People's Republic of China, India, and Russia, and a list of acronyms has been added to assist the reader.

I would like to thank all those colleagues and friends who have helped and encouraged me with the third edition. In particular, special thanks are due to Steve Frankland of Dargo Associates Ltd, Dr Gareth George for his expertise on sedimentary sequences, Rob Evans for his help with coal geophysics, and to the following for their contributions and support: Professor Vladimir Pavlovic, Dr Dave Pearson of Pearson Coal Petrography and Argus Media Ltd, as well as the staff at John Wiley & Sons Ltd.

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## Preface to Second Edition

The first edition of *Coal Geology* has provided the coal geologist and those associated with the coal industry with the background to the origins and characteristics of coal together with exploration techniques including geophysics and hydrogeology. Details of coal mining techniques, resource calculations, alternative uses of coal and environmental issues were also described.

Although broadly following the layout of the first edition, additional information has been added to coal origins, geographical distribution of coal and coal exploration. The chapter on coal resources and reserves has been brought up to date with current resource classifications together with recent world reserves/production figures.

The chapter on geophysics of coal has been enlarged and the alternative uses of coal, in particular, methane extraction and underground coal gasification have been expanded to reflect the increase in activity in these areas. Developments in environmental requirements have also been updated.

Again, numerous sources of information have been consulted, the majority of which are

listed in the bibliography section. International Standards relating to coal, listed in Appendix 1, have been updated and expanded to include P R China, India and Russia.

I would like to thank all those colleagues and friends who have helped and encouraged me with the second edition. In particular, special thanks are due to Steve Frankland of Dargo Associates Ltd, Rob Evans for his invaluable help with coal geophysics, Paul Ahner in the U.S.A. for providing data on underground coal gasification, and to the following for their contributions and support, Professor Vladimir Pavlovic of Belgrade University, Mike Coultas, Dave Pearson of Pearson Coal Petrography, Oracle Coalfields plc and Robertson Geologging, as well as the staff at John Wiley & Sons Ltd.

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## Preface to First Edition

The Handbook of Practical Coal Geology (Thomas 1992) was intended as a basic guide for coal geologists to use in their everyday duties, whether on site, in the office or instructing others. It was not intended as a definitive work on all or any particular aspect of coal geology, rather as a handbook to use as a precursor to, or in conjunction with more specific and detailed works.

This new volume is designed to give both the coal geologist and others associated with the coal industry background information regarding the chemical and physical properties of coal, its likely origins, its classification and current terminology. In addition I have highlighted the currently known geographical distribution of coal deposits together with recent estimates of world resources and production. I have also outlined the exploration techniques employed in the search for, and development of these coal deposits and the geophysical and hydrogeological characteristics of coal-bearing sequences, together with the calculation and categorisation of resources/reserves.

Chapters are devoted to the mining of coal, to the means of extracting energy from coal other than by conventional mining techniques, and to the environmental concerns associated with the mining and utilisation of coal.

Also covered is the development of computer technology in the geological and mining fields, and the final chapter is a condensed account of the marketing of coal, its uses, transportation and price.

Many sources of information have been consulted, the majority of which are listed in the reference section. A set of appendices contains information of use to the reader.

I would like to thank all those colleagues and friends who have helped and encouraged me with the book from conception to completion. In particular special thanks are due to Steve and Ghislaine Frankland of Dargo Associates Ltd, Alan Oakes, Rob Evans, Dr Keith Ball, Professor Brian Williams, Mike Coultas, Reeves Oilfield Services, IMC Geophysics Ltd and Palladian Publications, as well as the staff at John Wiley & Sons Ltd.

I should also like to thank those authors and organisations whose permission to reproduce their work is gratefully acknowledged.

Finally I would like to thank my family for their support, encouragement and assistance with the manuscript.

*Larry Thomas*

## List of Acronyms

AMD	Acid mine drainage	GHG	Greenhouse gas
AMM	Abandoned-mine methane	IGCC	Integrated gasification combined cycle
ASTM	American Society for Testing and Materials	ISP	Indian Standard Procedure
BAP	Bali Action Plan	JORC	Joint Ore Reserves Committee
BFBC	Bubbling fluidised-bed combustion	LRTAP	Long-range trans-boundary air pollution
BOF	Basic oxygen furnace	NAEN	Russian Code for Public Reporting of Exploration Results, Mineral Resources & Reserves
CBM	Coal-bed methane	NDC	Nationally determined contribution
CCS	Carbon capture and storage	NRO	National Reporting Organisation
CDM	Clean development mechanism	PCDDs	Polychlorinated dibenzo- <i>para</i> -dioxins
CFBC	Circulating fluidised-bed combustion	PCDFs	Polychlorinated dibenzofurans
CFR	Code of Federal Regulations	PERC	Pan-European Reserves and Resources Reporting Committee
CHP	Combined heat and power	PM	Particulate matter
CIM	Canadian Institute for Mining, Metallurgy and Petroleum	PRC	People's Republic of China
CMM	Coalmine methane	R/P	Reserves/production ratio
CMMI	Council of Mining & Metallurgical Institutions	SAMREC	South African Code for Reporting of Exploration Results, Mineral Resources and Reserves
CRIRSCO	Committee for Mineral Reserves International Reporting Standards	SAMVAL	South African Code for Reporting of Mineral Asset Valuation
CSG	Coal seam gas	SCR	Selective catalytic reduction
EAF	Electric arc furnace	SEC	United States Securities and Exchange Commission
EFG	European Federation of Geologists	SME	Society for Mining, Metallurgy and Exploration (USA)
EISA	Environmental Impact and Social Assessment	TDS	Total dissolved solids
EPA	Environmental Protection Agency		
FBC	Fluidised-bed combustion		
FCDD	Framework Convention on Climate Change		
FGD	Flue gas desulfurisation		

TEO	Technical–economic justification	UNFCCC	United Nations Convention on Climate Change
TERI	The Energy Research Institute (India)	USGS	United States Geological Survey
UCG	Underground coal gasification	VALMIN	Australian Code for the Public Reporting of Technical Assessments and Valuations of Mineral Assets
UNECE	United Nations Economic Commission for Europe	VAM	Ventilation air methane
UNEP	United Nations Environment Programme	WCA	World Coal Association
UNFC	United Nations Framework Classification for Fossil Energy and Mineral Resources	WEC	World Energy Council

# 1

## Preview

### 1.1 Scope

The object of this book remains unchanged. It is to provide geologists and those associated with the coal industry, as well as teachers of courses on coal about its geology and uses, with a background of the nature of coal and its varying properties, together with the practice and techniques required in order to compile geological data that will enable a coal sequence under investigation to be ultimately evaluated in terms of mineability and saleability. In addition, the alternative uses of coal as a source of energy together with the environmental implications of coal usage are also addressed.

Each of these subjects is a major topic in itself, and the book only covers a brief review of each, highlighting the relationship between geology and the development and commercial exploitation of coal.

### 1.2 Coal Geology

Coal is a unique rock type in the geological column. It has a wide range of chemical and physical properties, and it has been studied over a long period of time. This volume is intended to be a basic guide to understanding the variation in coals and their modes of origin and of the techniques required to evaluate coal occurrences.

The episodes of coal development in the geological column are given together with the principal coal occurrences worldwide. It is

accepted that this is not totally exhaustive, as coal does occur in small areas not indicated in the figures or tables.

The reporting of coal resources/reserves is an important aspect of coal geology, and international standards and guidelines are in place to ensure the correct reporting procedures to be undertaken. Most national standards are now being reconciled with these, and the principal resources/reserves classifications are given. Current estimates of global resources and reserves of coal, together with coal production figures, are listed. Although these obviously become dated, they do serve to indicate where the major deposits and mining activities are currently concentrated.

In relation to the extraction of coal, the understanding of the geophysical and hydrogeological properties of coals is an integral part of any coalmine development, and these are reviewed together with the principal methods of mining coal. The increasing use of computer technology has had a profound impact on geological and mining studies. Some of the applications of computers to these are discussed.

An important development in recent years has been the attempts to use coal as an alternative energy source by either removing methane ( $\text{CH}_4$ ) gas from the in-situ coal and coal mines, or by liquefying the coal as a direct fuel source, or by gasification of coal in situ underground. These technologies together are particularly significant in areas where conventional coalmining has ceased or where coal deposits are situated either at depths uneconomic to



mine or in areas where mining is considered environmentally undesirable.

### 1.3 Coal Use

The principal uses of traded coals worldwide are for electricity generation and steel manufacture, with other industrial users and domestic consumption making up the remainder.

Lack of environmental controls in the use of coal in the past has led to both land and air pollution, as well as destruction of habitat. Modern environmental guidelines and legislation are both repairing the damage of the past and preventing a reoccurrence of such phenomena. An outline is given of the types of environmental concerns that exist where coal is utilised, together with the current position on the improvements in technology in mining techniques, industrial processes, and electricity generation emissions.

The marketing of coal is outlined, together with the contractual and pricing mechanisms commonly employed in the coal producer/coal user situation.

### 1.4 Background

In most industrial countries, coal has historically been a key source of energy and a major contributor to economic growth. In today's choice of alternative sources of energy, industrialised economies have seen a change in the role for coal.

Originally, coal was used as a source of heat and power in homes and industry. During the 1950s and 1960s cheap oil curtailed the growth of coal use, but the uncertainties of oil supply in the 1970s led to a resumption in coal consumption and a rapid growth in international coal trade. This, in turn, was followed by an increasingly unfavourable image for coal as a contributor to greenhouse gas (GHG) emissions and had been closely identified with global warming. The coal industry has

responded positively to this accusation, and modern industrial plants have much lower emissions levels than in previous years. Current figures show that coal accounts for 45% of all carbon dioxide (CO<sub>2</sub>) emissions.

The world consumption of fossil fuels, and thus emissions of CO<sub>2</sub>, will continue to increase, and fossil fuels still meet around 86% of primary energy requirements. The objective of the United Nations Framework Convention on Climate Change (UNFCCC) signed at the 1992 Earth Summit in Rio de Janeiro is to 'stabilise GHG concentrations in the atmosphere at a level that would prevent dangerous anthropogenic interference with the climate system'. No set levels were identified, but emissions in developed countries were expected to be reduced to 1990 levels. A series of annual meetings by the international body under the UNFCCC, the Conference of the Parties (COP), have taken place, notably COP-3 in Kyoto, Japan, in 1997, at which the Kyoto Protocol was drawn up, setting emissions targets for all the countries attending. However, government ministers at COP-6 in The Hague in November 2000 failed to agree on the way forward to meet the Kyoto Protocol targets. This placed the whole of the Kyoto Protocol's ambitious and optimistic plan for a global agreement on GHG emissions reduction in an uncertain position. This could be an indication of overambitious goals rather than any failure in the negotiations, and it is up to the parties concerned to establish a realistic set of targets for emissions reductions in the future. The Copenhagen Accord in 2009 reinforced the need for emissions reductions, together with providing financial assistance to help developing countries cut carbon emissions. In 2015, at the Paris Agreement, parties to the UNFCCC reached agreement to combat climate change and to accelerate and intensify the actions and investments for a sustainable low-carbon future. This was the first legally binding global climate deal.

It remains a fact that many economies still depend on coal for a significant portion of

their energy needs. Coal currently accounts for 28% of the world's consumption of primary energy, and, importantly, coal provides fuel for the generation of around 39% of the total of the world's electricity. In 2018, internationally traded coal was 1169 Mt, the bulk of which was steam or thermal coal. Globally, 5.6 Gt of coal was consumed in 2016 (BP plc 2017).

Coal reserves are currently estimated to be around 900 Gt, and the world coal reserves to production ratio is nearly six times that for oil and four times that for natural gas.

This, together with the globally democratic distribution and secure nature of coal deposits, will ensure that coal will continue to be a major energy resource for some considerable time to come.

With this scenario in mind, this volume is intended to assist those associated with the coal industry, as well as educationalists and those required to make economic and legislative decisions about coal.

The philosophy and views expressed in this book are those of the author and not the publisher.

## 2

# Origin of Coal

## 2.1 Introduction

Sedimentary sequences containing coal or peat beds are found throughout the world and range in age from middle Palaeozoic to Recent.

Coals are the result of the accumulation of vegetable debris in a specialised environment of deposition. Such accumulations have been affected by synsedimentary and post-sedimentary influences to produce coals of differing rank and differing degrees of structural complexity, the two being closely interlinked. The plant types that make up coals have evolved over geological time, providing a variety of lithotypes in coals of differing ages.

Remarkable similarities exist in coal-bearing sequences, due for the greater part to the particular sedimentary associations required to generate and preserve coals. Sequences of vastly different ages from areas geographically separate have a similar lithological framework and can react in similar fashions structurally.

It is a fact, however, that the origin of coal has been studied for over a century and that no one model has been identified that can predict the occurrence, development, and type of coal. A variety of models exist that attempt to identify the environment of deposition, but no single one can adequately give a satisfactory explanation for the cyclic nature of coal sequences, the lateral continuity of coal beds, and the physical and chemical characteristics of coals. However, the advent of sequence stratigraphy

has recognised the pattern of geological events leading to the different phases of deposition and erosion within coal-bearing sequences.

## 2.2 Sedimentation of Coal and Coal-Bearing Sequences

During the last 90 years, interest has grown rapidly in the study of sedimentological processes, particularly those characteristic of fluvial and deltaic environments. It is these, in particular, that have been closely identified with coal-bearing sequences.

It is important to give consideration both to the recognition of the principal environments of deposition and to the recent changes in emphasis regarding those physical processes required, in order to produce coals of economic value. In addition, the understanding of the shape, morphology, and quality of coal seams is of fundamental significance for the future planning and mining of coals. Although the genesis of coal has been the subject of numerous studies, models that are used to determine the occurrence, distribution, and quality of coal are often still too imprecise to allow such accurate predictions.

### 2.2.1 Depositional Models

The recognition of depositional models to explain the origin of coal-bearing sequences

and their relationship to surrounding sediments has been achieved by a comparison of the environments under which modern peats are formed and ancient sequences containing coals.

Cecil et al. (1993) suggested that the current models often concentrate on the physical description of the sediments associated with coal rather than concentrating on the geological factors that control the genesis of coal beds. They also suggest that models that combine sedimentation and tectonics with eustasy and chemical change have not yet been fully developed. Such integrated models would give an improved explanation of physical and chemical processes of sedimentation. It should be noted that the use of sequence stratigraphy in facies modelling is based on physical processes and does not take into account chemical stratigraphy. This will prove a deficiency when predicting the occurrence and character of coal beds.

The traditional depositional model used by numerous workers was based on the 'cyclothem', a series of lithotypes occurring in repeated 'cycles'. Weller (1930) and Wanless and Weller (1932) remarked on the similarity of stratigraphic sections associated with every coal bed; i.e. marine sediments consisted of black shaly shale with large concretions, limestone with marine fossils and shale with ironstone nodules and bands, whereas continental sequences comprised sandstone lying unconformably on lower beds, sandy shale, limestone without marine fossils, rootlet bed or seatearth, and then coal. Although all of the members of each cyclothem vary in thickness and lithology from place to place, the character of some beds is remarkably similar at localities great distances apart. Their studies showed that the entire Pennsylvanian (upper Carboniferous) system in the Eastern Interior and northern Appalachian Basins and the Lower Pennsylvanian strata in the northern part of the Western Interior Basin consist of similar successions of cyclothem. Individual cyclothem are persistent, and correlation of cyclothem at widely separated localities is

possible. This concept has been modified to a model that relates lateral and vertical sequential changes to depositional settings that have been recognised in modern fluvial, deltaic, and coastal barrier systems. Further studies on the traditional model are based on work carried out in the USA by Horne (1979), Horne et al. (1978, 1979), Ferm (1979), Ferm et al. (1979), Ferm and Staub (1984), and Staub and Cohen (1979). The sequences, or lithofacies, are characterised by the sedimentary features listed in Table 2.1. Other workers include Thornton (1979) and Jones and Hutton (1984) on coal sequences in Australia, Galloway and Hobday (1996), and Guion et al. (1995) and George (2014) in the UK. More recent studies have compared such established depositional models with modern coastal plain sedimentation, e.g. in equatorial South-East Asia, and have concentrated in particular on modern tropical peat deposits (Cecil et al. 1993; Clymo 1987; Gastaldo et al. 1993; McCabe and Parrish 1992). Studies by Hobday (1987), Diessel (1992), Lawrence (1992), Jerzykiewicz (1992), Dreesen et al. (1995), Cohen and Spackman (1972, 1980), Flint et al. (1995), and McCabe (1984, 1987, 1991) have all further developed the model for coal deposits of differing ages, using the traditional model but relating it to modern sedimentary processes. Galloway and Hobday (1996), in their textbook, give a detailed analysis of coal-bearing environments with worldwide examples.

In parallel with this work, detailed studies of peat mires have both raised and answered questions on the development of coal geometry, i.e. thickness and lateral extent, together with the resultant coal chemistry.

The traditional model is still a basis for modern coal studies, but linked to detailed interpretation of sedimentary sequences and a better understanding of peat development and preservation.

### 2.2.2 The Traditional Model

The interrelationship of fluvial and marine processes results in significant variation in

**Table 2.1** Sedimentary features used to identify depositional environments.

Recognition characteristics	Fluvial and upper delta plain <sup>a</sup>	Transitional lower delta plain <sup>a</sup>	Lower delta plain <sup>a</sup>	Back-barrier <sup>a</sup>	Barrier <sup>a</sup>
I Coarsening upwards					
A Shale and siltstone sequences	2-3	2	1	2-1	3-2
(i) >50 ft	4	3-4	2-1	2-1	3-2
(ii) 5-25 ft	2-3	2-1	2-1	2-1	3-2
B Sandstone sequences	3-4	3-2	2-1	2	2-1
(i) >50 ft	4	4	2-1	3	2-1
(ii) 5-25 ft	3	3-2	2-1	2	2
II Channel deposits					
A Fine-grained abandoned fill	3	2-3	1-2	2	3-2
(i) Clay and silt	3	2-3	1-2	2	3-2
(ii) Organic debris	3	2-3	1-2	2-3	3
B Active sandstone fill	1	2	2-3	2-3	2
(i) Fine grained	2	2	2-3	2-3	2
(ii) Medium- and coarse-grained	1	2-3	3	3	2-3
(iii) Pebble lags	1	1	2	2-3	3-2
(iv) Coal spars	1	1	2	2-3	3-2
III Contacts					
A Abrupt (scour)	1	1	2	2	2-1
B Gradational	2-3	2	2-1	2	2
IV Bedding					
A Cross beds	1	1	1	1-2	1-2
(i) Ripples	2	2-1	1	1	1
(ii) Ripple drift	2-1	2	2-3	3-2	3-2
(iii) Trough cross beds	1	1-2	2-1	2	2-1
(iv) Graded beds	3	3	2-1	3-2	3-2
(v) Point bar accretion	1	2	3-4	3-4	3-4
(vi) Irregular bedding	1	2	3-2	3-2	3-2
V Levee deposits					
A Irregularly interbedded sandstones and shales, rooted	1	1-2	3-2	3	4
VI Mineralogy of sandstones					
A Lithic greywacke	1	1	1-2	3	3
B Orthoquartzite	4	4	4-3	1-2	1
VII Fossils					
A Marine	4	3-2	2-1	1-2	1-2
B Brackish	3	2	2	2-3	2-3
C Fresh	2-3	3-2	3-4	4	4
D Burrow	3	2	1	1	1

a) 1, abundant; 2, common; 3, rare; 4, not present.

Source: From Horne et al. (1979).

the geometry, thickness, quality, and continuity of coal seams. The resultant distribution and composition of the facies characterising marine deltas are controlled by a number of variables, such as climate, sediment availability, salinity, and depth of tidal influences, together with sea-level changes relating to eustatic, tectonic, or compactional processes.

#### 2.2.2.1 Prodelta and Delta Front Facies

The coastal or prodelta end of the depositional model is characterised by clean barrier sandstones. These become finer grained in a seaward direction and intercalate with red and green calcareous shales and carbonate rocks, with the latter containing marine faunas. Towards the delta front, the back-barrier facies are characterised by a gradation into dark grey lagoonal shales with brackish water faunas and into marginal swamp areas on which vegetation was established. The barrier sandstones have been constantly reworked and, therefore, are more quartzose than those sandstones in surrounding environments with the same source area. Wave and tidal reworking at the delta front produces barrier sand bars parallel to the shore, whereas tidal reworking generates bars parallel to the ebb and flow currents (George 2014).

Barrier sandbars exhibit a variety of bedding styles: first, extensive sheets of plane-bedded sandstones with rippled and burrowed upper surfaces, interpreted as storm washover sands; second, wedge-shaped bodies that extend landward, which can attain thicknesses of up to 6 m and contain landward-dipping planar and trough cross-beds, interpreted as floodtide delta deposits; and third, channel-fill sandstones, which may scour to depths of over 10 m into the underlying sediments, interpreted as tidal channel deposits.

A depositional reconstruction is shown in Figure 2.1a based on studies by Horne et al. (1979).

The lagoonal back-barrier environment is characterised by upwards-coarsening, organic-rich grey shales and siltstones overlain

by thin and discontinuous coals. This sequence exhibits extensive bioturbation zones, together with bands and concretions of chemically precipitated iron carbonate (sideritic ironstone). The extent of such sequences is considered to be in the order of 20–30 m in thickness and 5–25 km in width. A typical vertical sequence of back-barrier deposition is shown in Figure 2.1b.

#### 2.2.2.2 Lower Delta Plain Facies

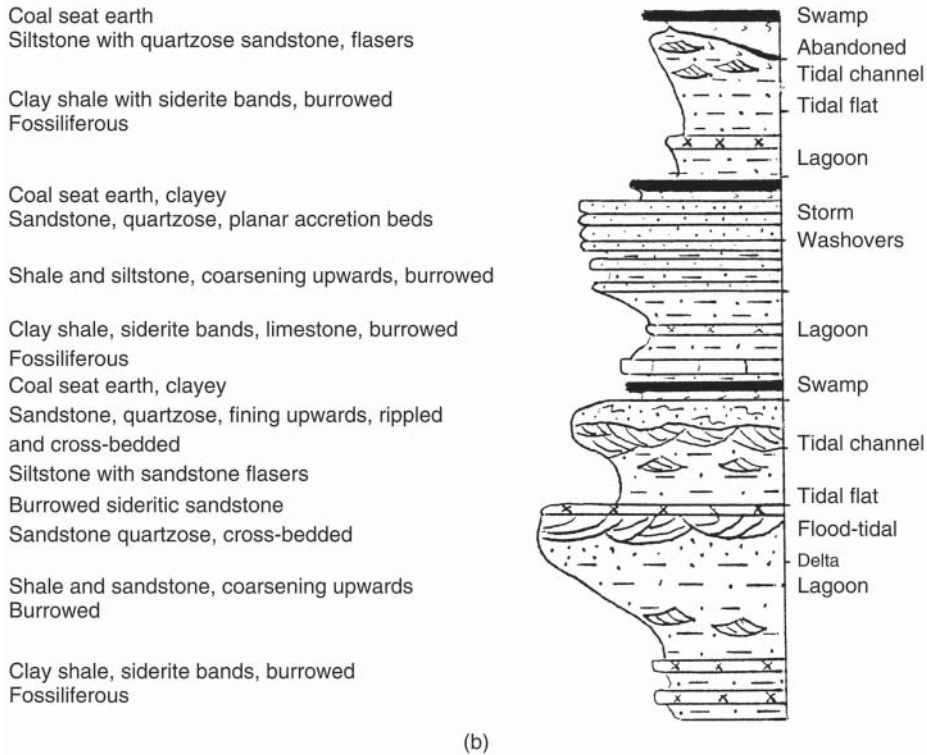
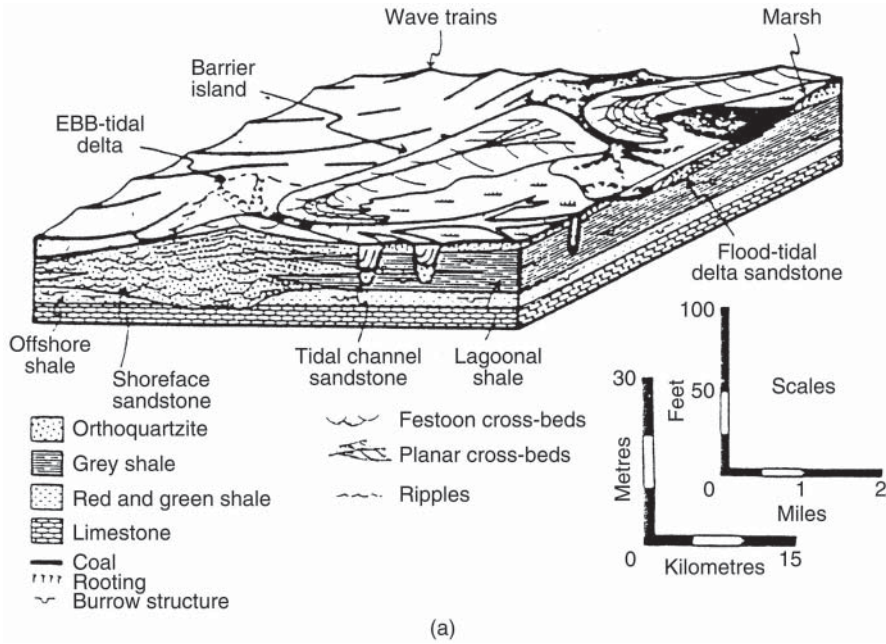
Lower delta plain deposits are dominated by coarsening-upwards sequences of mudstone and siltstone, ranging from 15 to 55 m in thickness and 8–110 km in lateral extent. The lower part of these sequences is characterised by dark grey to black mudstones with irregularly distributed limestones and siderite (Figure 2.2a).

In the upper part, sandstones are common, reflecting the increasing energy of the shallow water as the bay fills with sediment. Where the bays have filled sufficiently to allow plant growth, coals have formed. Where the bays did not fill completely, bioturbated, siderite-cemented sandstones and siltstones have formed.

This upwards-coarsening pattern is interrupted in many areas by crevasse-splays (Figure 2.2b). In the Pennsylvanian of the USA, crevasse-splay deposits can be >10 m in thickness and 30 m to 8 km wide.

Overlying and laterally equivalent to the bay-fill sequences are thick lithic sandstones up to 25 m in thickness and up to 5 km in width. These are interpreted as distributary mouth bar deposits; they are widest at the base and have gradational contacts. They coarsen upwards and towards the middle of the sand body. In some places, fining-upwards sequences are developed on the top of the distributary mouth bar and bay-fill deposits.

These distributary channel-fill deposits have an irregular sharp basal contact, produced by scouring of the underlying sediments. At the base, pebble and coal-fragment lag deposits are common.



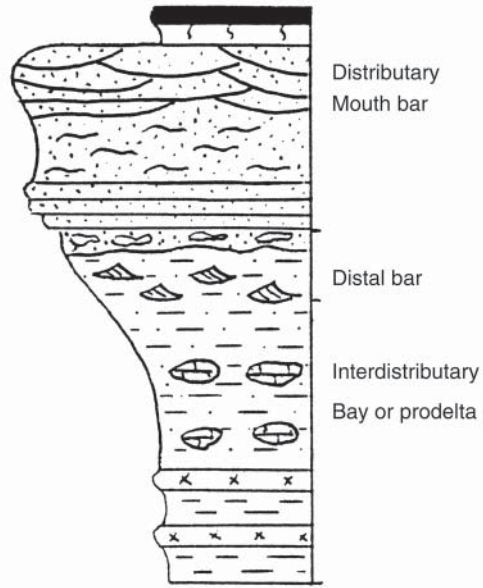
**Figure 2.1** (a) Barrier and back-barrier environments including tidal channels and flood-tidal deltas, based on exposures in Kentucky, USA. *Source:* From Horne et al. (1979). (b) Generalised vertical section through back-barrier deposits in the Carboniferous of eastern Kentucky, USA. *Source:* From Horne et al. (1979).



Coal  
 Seat earth, clayey  
 Sandstone, fine to medium grained  
 Multi-directional planar and  
 Festoon cross beds  
 Sandstone, fine grained, rippled  
 Sandstone, fine grained, graded beds  
 Sandstone, flow rolls  
 Sandstone, fine grained, flaser bedded  
 and siltstone

Silty shale and siltstone with calcareous  
 Concretions, thin-bedded, burrowed  
 Occasional fossil

Clay shale with siderite bands, burrowed  
 Fossiliferous



(a)

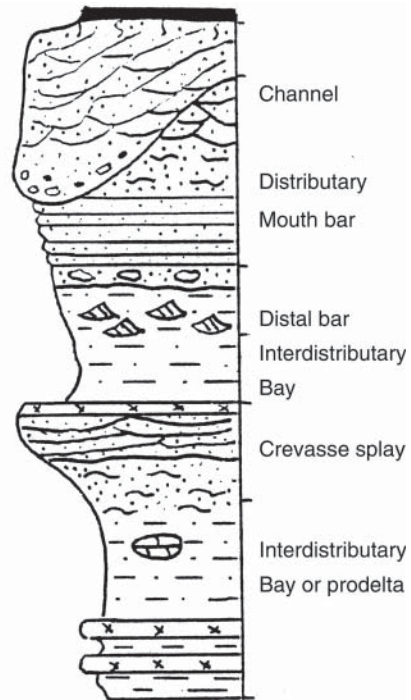
Coal  
 Rooted sandstone  
 Sandstone, fine grained, climbing ripples  
 Sandstone fine to medium grained  
 Sandstone, medium grained, festoon cross-beds  
 Conglomerate lag, siderite and coal pebbles  
 Sandstone, siltstone, graded beds

Sandstone, flow rolls  
 Sandstone, siltstone, flaser bedded  
 Siltstone/silty shale, thin bedded, burrowed

Burrowed sideritic sandstone  
 Sandstone, fine grained  
 Sandstone, fine grained, rippled

Silty shale, siltstone with calcareous  
 Concretions, thin bedded, burrowed

Clay shale with siderite bands, burrowed,  
 Fossiliferous



(b)

**Figure 2.2** Generalised vertical sequences through lower delta plain deposits in eastern Kentucky, USA. (a) Typical coarsening-upward sequence. (b) Same sequence interrupted by crevasse-splay deposits. *Source:* From Horne et al. (1979).



Because of the rapid abandonment of distributaries, fine-grained mudstone fills are common in lower delta plain deposits. They represent silt and organic debris that has settled from suspension in the abandoned distributary. In some areas, thick organic accumulations filled these channels, resulting in the formation of lenticular coals. Apart from those formed in the abandoned channels, coals of the lower delta plain are generally relatively thin and widespread. Such coals are oriented parallel to the distributary patterns.

### 2.2.2.3 Upper Delta Plain Facies

Between the upper and lower delta plains there is a transition zone that exhibits characteristics of both sequences. This zone consists of a widespread platform on which peat mires were formed. This platform was cut by numerous channels and the sequence disrupted by crevasse-splay deposits. The coals formed on the platform are thicker and more widespread than the coals of the lower delta plain; such a sequence is shown in Figure 2.3b.

Upper delta plain deposits are dominated by linear, lenticular sandstone bodies up to 25 m thick and up to 11 km wide. These sandstones have scoured bases and pass laterally in the upper part into grey shales, siltstones, and coals. The sandstones fine upwards with abundant pebble conglomerates, and there are coal clasts in the lower part. The sandstones are characterised by massive bedding and are overlain by siltstones.

These sandstone bodies widen upwards in cross-section and are considered to have been deposited in the channels and on the flanks of streams that migrated across the upper delta plain; see Figure 2.4a. Laterally, muds and silts with vegetation and invertebrate activity are characteristic. These areas are colonised by plants, and extensive mires may develop as the channel system becomes infilled with an increase in available sediment.

Coal seams in the upper delta plain facies may be >10 m in thickness but are of limited lateral extent. Figure 2.4b illustrates a

vertical sequence of upper delta plain facies from eastern Kentucky and southern West Virginia, USA.

### 2.2.2.4 Fluvial Facies

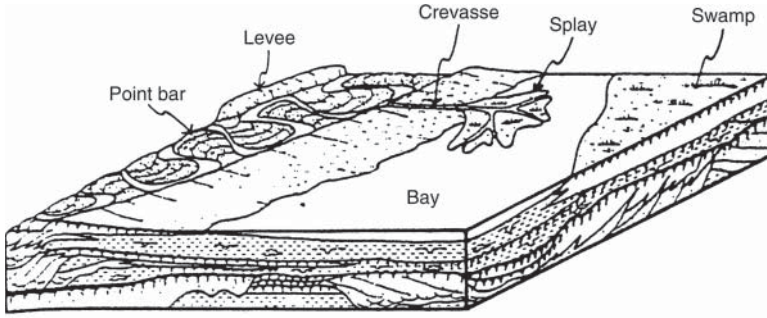
Meandering rivers are developed on floodplains close to base level, where they transport suspended sediment within sinuous channels. Downcutting is minimal, with river energy being expended laterally. The evolution and extensive colonisation of alluvial terrains by plants – whose beginnings lie in the Late Silurian to Devonian and which increased dramatically during the Carboniferous – resulted in rooted plants having the ability to form soils and stabilise river margins and to produce a decrease in surface run-off. As such, coals commonly overlie upwards-fining sequences accumulating on riverbanks and in backswamps adjacent to migrating channels. The thickest coal development occurs between the channels, with splitting and thinning at the channel margins. Detailed studies of the influence of vegetation on the evolution of fluvial systems have been documented by Davies and Gibling (2010) and Gibling et al. (2014).

Anastomosed rivers comprise two or more interconnected channels separated by areas of floodplain deposits on which lacustrine and coal deposits accumulate. Their morphology and development are described in detail by Makaske (2001) in his comprehensive review of the classification, origin, and characteristic sedimentation of anastomosing rivers. To illustrate the complex depositional relationships, a depositional model of an anastomosed river is shown in Figure 2.5 (George 2014).

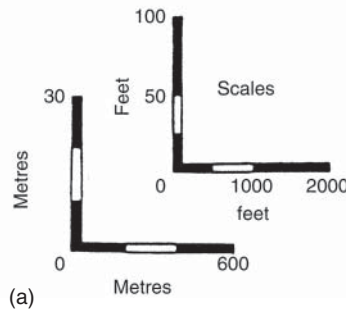
These sequences characterised by meandering and anastomosed rivers are considered to be virtually absent prior to the evolution and extensive colonisation of alluvial terrains (George 2014; Gibling et al. 2014).

### 2.2.3 Modern Peat Analogues

The principal characteristics of a coal are its thickness, lateral continuity, rank, maceral



- Sandstone
- Sandstone and siltstone
- Shale
- Coal
- Rooting
- Burrow structure
- Marine fossil
- Bedding planes
- Trough cross-beds



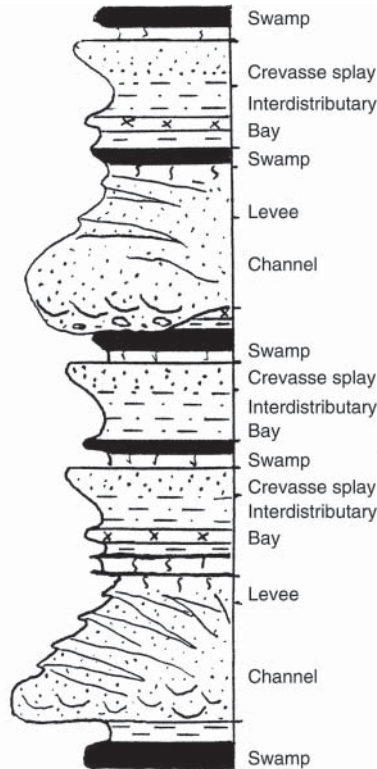
Coal, seat earth, clayey

Sandstone, fine grained, rippled  
 Shale and siltstone, coarsening upwards  
 Siderite bands, burrowed  
 Coal, seat earth, silty

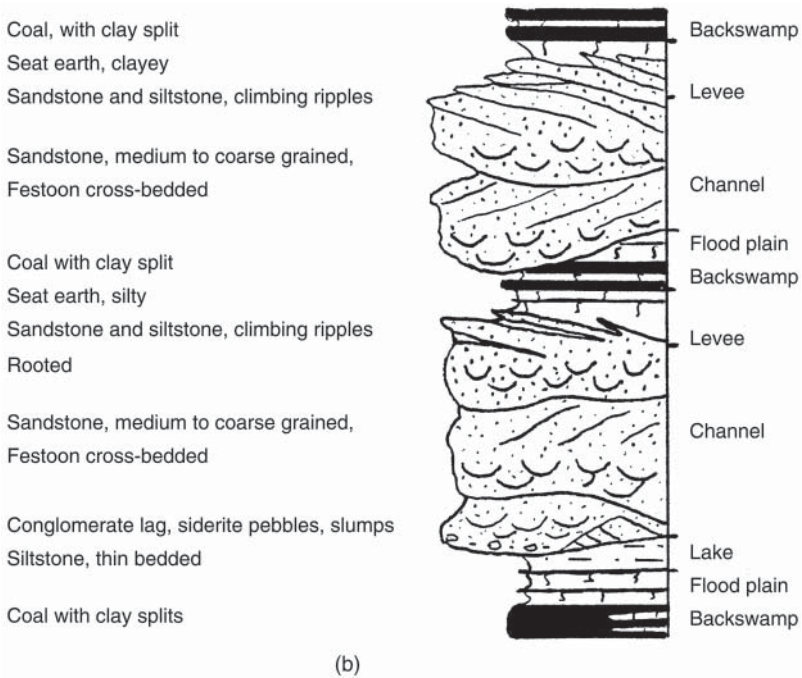
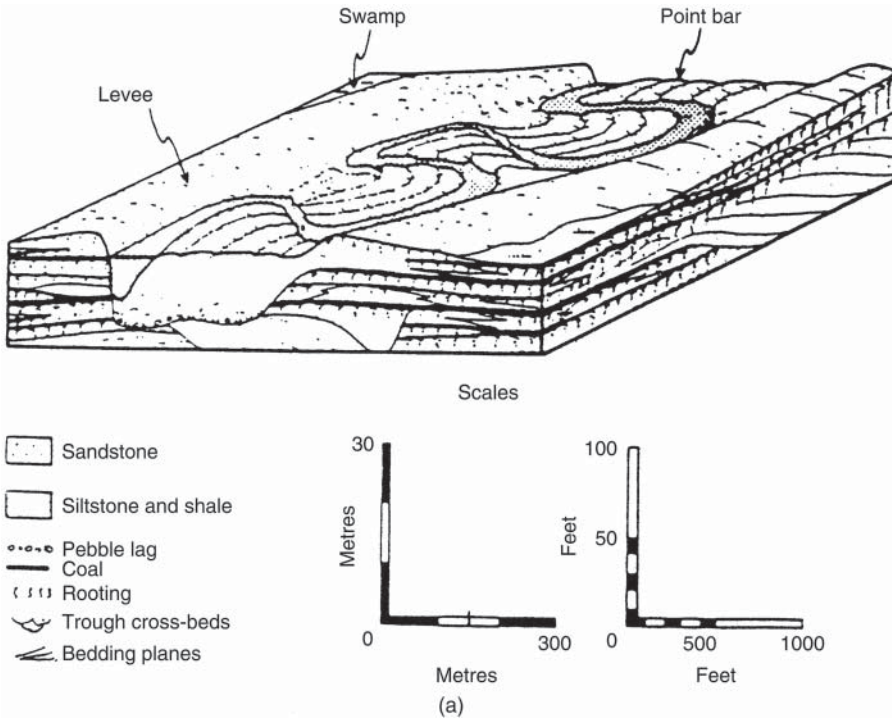
Sandstone and siltstone, climbing ripples,  
 Rooted  
 Sandstone, fine to medium grained, festoon  
 Cross-bedded  
 Conglomerate lag, siderite pebbles  
 Coal, seat earth, clayey  
 Shale and siltstone, coarsening upwards  
 Burrowed

Coal, seat earth  
 Sandstone fine grained rippled  
 Shale and siltstone coarsening upwards  
 Siderite bands, burrowed  
 Coal with seat earth splits  
 Sandstone and siltstone, climbing ripples,  
 Rooted

Sandstone, fine to medium grained, festoon  
 Cross-bedded  
 Clay shale, burrowed  
 Coal



**Figure 2.3** (a) Reconstruction of transitional lower delta plain environments in Kentucky, USA. *Source:* From Horne et al. (1979). (b) Generalised vertical sequence through transitional lower delta plain deposits of eastern Kentucky and southern West Virginia, USA. *Source:* From Horne et al. (1979).



**Figure 2.4** (a) Reconstruction of upper delta plain–fluvial environments in Kentucky, USA. *Source:* From Horne et al. (1979). (b) Generalised vertical sequence through upper delta plain–fluvial deposits of eastern Kentucky and southern West Virginia, USA. *Source:* From Horne et al. (1979).

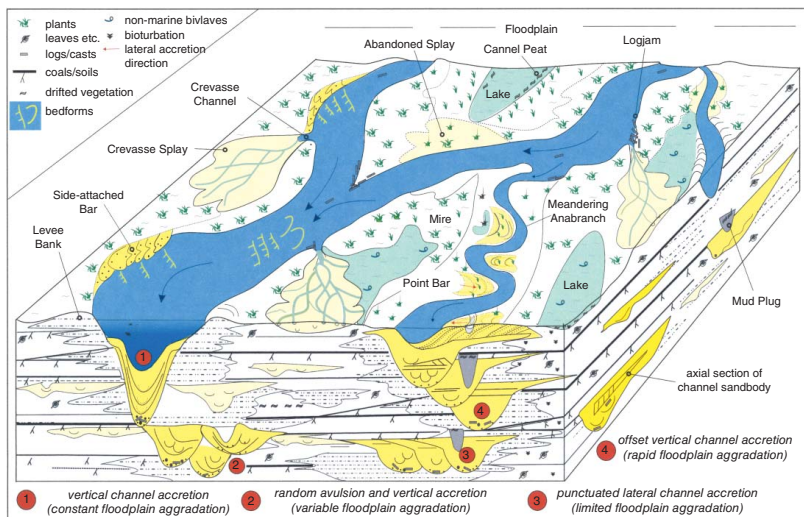


Figure 2.5 Depositional model of a humid anastomosing river based on Westphalian sequences and modern/ancient examples (George 2014).

content, and quality. Apart from rank, which is governed by burial and subsequent tectonic history, the remaining properties are determined by factors controlling the mire where the peat originally formed. These factors include type of mire, type(s) of vegetation, growth rate, degree of humification, base-level changes, and rate of clastic sediment input (McCabe and Parrish 1992).

About 3% of the Earth's surface is covered by peat, totalling  $310 \times 10^6$  ha (World Energy Council 1998). This includes the tropical peats (>1 m thick) of South-East Asia, which cover almost 200 000 km<sup>2</sup>.

During the last 15 years, numerous studies have attempted to understand more fully how peat-producing wetlands or mires are developed and maintained and how, in particular, post-depositional factors influence the formation of coals.

Diessel (1992) divides peat-producing wetlands into ombrogenous peatlands, or mires (owing their origin to rainfall) and topogenous peatlands (owing their origin to a place and its surface/groundwater regime). A great variety of topogenous peats form when waterlogging of vegetation is caused by groundwater, whereas ombrogenous peats are of greater extent but less varied in character.

Based on this distinction, Diessel (1992) gives a classification of peatlands or mires as shown in Table 2.2. This is illustrated in Figure 2.6, which shows the relationship between ombrotrophic and rheotrophic mires in terms of the influence of rainwater and groundwater in their hydrological input. The inorganic content of mires is seen to increase in the topogenous rheotrophic mires.

The classification of the two hydrological categories of mire lists a number of widely used terms. Moore (1987) has defined a number of these:

<i>Mire</i>	is now accepted as a general term for peat-forming ecosystems of all types.
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<i>Bog</i>	is generally confined to ombrotrophic peat-forming ecosystems.
<i>Bog forest</i>	consists of ombrotrophic forested vegetation, usually an upper storey of coniferous trees and a ground layer of <i>sphagnum</i> moss.
<i>Marsh</i>	is an imprecise term used to denote wetlands characterised by floating vegetation of different kinds, including reeds and sedges, but controlled by rheotrophic hydrology.
<i>Fen</i>	is a rheotrophic ecosystem in which the dry-season water table may be below the surface of the peat.
<i>Swamps</i>	are a rheotrophic ecosystem in which the dry-season water table is almost always above the surface of the sediment. It is an aquatic ecosystem dominated by emergent vegetation.
<i>Floating swamps</i>	develop around the fringes of lakes and estuaries and extend out over open water. These platforms can be thick and extensive, particularly in tropical areas.
<i>Swamp forest</i>	is a specific type of swamp in which trees are an important constituent, e.g. mangrove swamps.

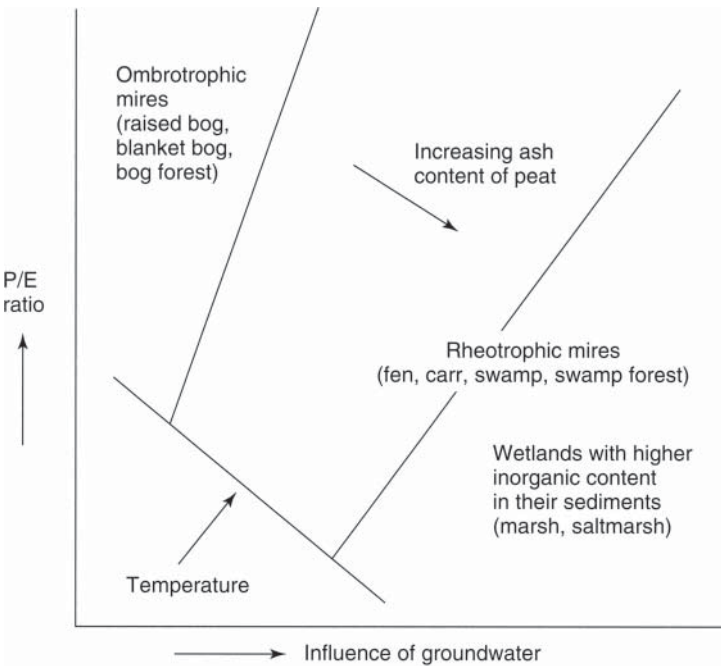
The resultant characteristics of coals are primarily influenced by the following factors during peat formation: type of deposition, the peat-forming plant communities, the nutrient supply, acidity, bacterial activity, temperature, and redox potential.



**Table 2.2** Classification of mires.

Peatlands (Mires)	
Ombrogenous	Topogenous
<b>Ombrotrophic</b> = rain fed	<b>Mineralotrophic</b> = mineral fed
<b>Oligotrophic</b> = poorly fed	<b>Rheotrophic</b> = flow fed
	<b>Eutrophic</b> = well fed
Raised bog	Tree cover increases
<i>Sphagnum</i> bog	Marsh
Bog forest	Fen
	Swamps
	Floating swamps
	Swamp forest
Transitional or mixed mires	
<b>Mesotrophic</b>	

Source: Adapted from Diessel (1992).



**Figure 2.6** Proposed relationship between mires in terms of the relative influence of rainwater and groundwater in their hydrological input (Moore 1987).

In order for a mire to build up and for peat to accumulate, the following equation must balance:

$$\begin{aligned} \text{Inflow} + \text{Precipitation} &= \text{Outflow} \\ &+ \text{Evapotranspiration} + \text{Retention} \end{aligned}$$

The conditions necessary for peat accumulation, therefore, are a balance between plant production and organic decay. Both are a function of climate, plant production, and organic decay. Such decay of plant material within the peat profile is known as

humification. The upper part of the peat profile is subject to fluctuations in the water table and is where humification is most active. The preservation of organic matter requires rapid burial or anoxic conditions (McCabe and Parrish 1992), the latter being present in the waterlogged section of the peat profile. In addition, an organic-rich system will become anoxic faster than an organic-poor one, as the decay process consumes oxygen. This process is influenced by higher temperatures, with decay rates being fastest in hot climates. Rates of humification are also affected by the acidity of the groundwater, as high acidity suppresses microbial activity in the peat.

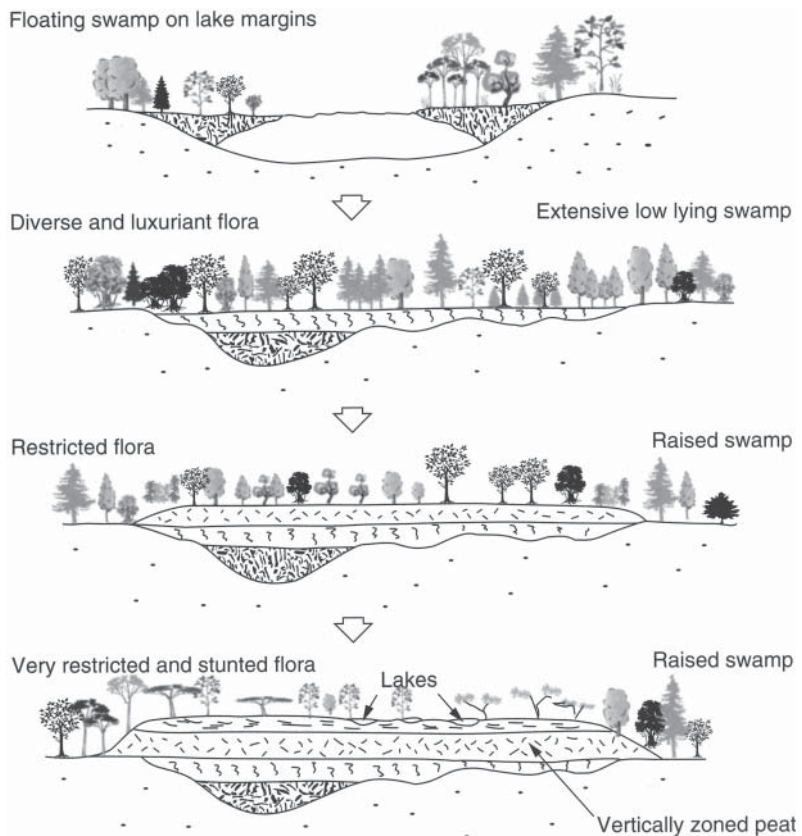
Peat formation can be initiated by:

- Terrestrialisation, which is the replacement due to the setting up of a body of water (pond, lake, lagoon, interdistributary bay) by a mire.
- Paludification, which is the replacement of dry land by a mire, e.g. due to a rising groundwater table.

As peat is relatively impermeable, its growth may progressively impede drainage over wide areas, so that low-lying mires may become very extensive. In those areas where annual precipitation exceeds evaporation, and where there are no long dry periods, a raised mire may develop. Such mires are able to build upwards because they maintain their own water table. The progression of a peat-forming environment from the infilling of a water course or lake to a low-lying mire and finally to a raised mire should produce zonation in the peat accumulated, as shown in Figure 2.7.

Depositional models may show peat formation adjacent to and intercalated with areas of active clastic deposition. Such peats accumulating on interchannel areas on the delta plain

**Figure 2.7** Evolutionary sequence of swamp types showing the development of a raised swamp with distinct peat zonations. *Source:* From McCabe (1984). Reproduced with permission of Blackwell Scientific Publications.



may be disrupted by clastic contamination from crevasse splays or by subsidence of the interchannel area resulting in submergence of the peat, cessation of peat development, and clastic influx. Sediment may also be introduced into low-lying mires by floods, storm surges, or exceptionally high tides. The overall result of clastic contamination is an increase in the ash content of the peat. Also, inundation of mires by aerated waters helps to degrade the peat and enrich it with inorganics.

Basin subsidence combined with ombrogenous peat accumulation such that the rise in the peat surface continues to outstrip the rate of subsidence will lead to the formation of thick and clean (low mineral matter content) coals (McCabe 1984). Low-ash coals, therefore, must have formed in areas removed or cut off from active clastic deposition for long periods of time, e.g. centuries. Partings in coals, such as mudstone, indicate the interruption of peat formation and may represent intervals of thousands of years.

For a thick peat layer to form in a topogenous setting it is essential that the rise in the water table and the rate of peat accumulation are balanced. In the case of a slower rise in water level, peat accumulation could be terminated by oxidation, but in a very wet climate the peat formation might continue under high moor conditions. Actual rates of peat accumulation or accretion vary in different climates and with the type of vegetation. Assuming a compaction ratio of 10 : 1 (Ryer and Langer 1980) to operate in the transition from peat to bituminous coal, and considering that some of the coal seams are tens of metres thick, optimum peat-forming conditions must, therefore, require the maintenance of a high groundwater table over very long periods of time, i.e. 5–10 ka for every metre of clean bituminous coal.

As peat accumulation is regulated by temperature and precipitation, tropical and subtropical regions are well suited for large peat development, where rates of decay are higher. Most modern peats are situated in low terrains not far above sea level. However, even

in conditions of slow plant accumulation, peat can still develop in large quantities. Diessel (1992) quotes evidence that most Gondwana coal deposits have been formed under cool to temperate conditions, whereas the European Palaeogene–Neogene coal formations began in tropical conditions in the Eocene, changing to temperate conditions in the Miocene.

A number of peat types have been summarised by Diessel (1992):

- Fibrous or woody peat, which shows the original plant structures only slightly altered by decay and may include branches, trunks, and roots of trees.
- Pseudo-fibrous peat, comprised of soft plastic material.
- Amorphous peat, in which the original structure of the plant's cell tissue has been destroyed by decomposition, resulting in a fine organic plastic mass.
- Intermediate forms of peat consisting of more resistant elements set in an altered matrix. Mixed peats are alternating layers of fibrous peat and amorphous peat.

However, these types can display overlapping characteristics dependent upon types of vegetation and mire setting.

In contrast to the traditional depositional model, studies of modern environments suggest that significant areas of low-ash peats are not present on delta plains and that most mires on coastal or floodplain areas are not sites of true peat accumulation. The exception appears to be those areas where raised mires have developed. Floating mires may also produce low-ash peats, but these are thought generally to be of limited extent. Examination of modern delta plain peats shows that they have an ash content of over 50% on a dry basis and that peats with less than 25% ash on a dry basis rarely exceed 1 m in thickness. These peats, if preserved in the geological record, would form carbonaceous mudstones with coaly stringers.

Studies of raised mires indicate that ash levels can be less than 5%, and over large areas they may be as low as 1–2%. Rates of organic



accumulation in raised mires outstrip rates of sedimentation from overbank or tidal flooding. However, although some low-ash coals have doubtless originated as products of raised mires, many coals are thought to have formed under palaeoclimates unsuitable for raised mire development. One suggestion is that low-ash coals originated as high-ash peats and were depleted in ash during the coalification process. Acidic waters may hasten the dissolution of many minerals; but not all mires are acidic, and some may even contain calcareous material. Another concept is that peat accumulation was not contemporaneous with local clastic deposition, suggesting that resulting coals are distinct from the sediments above and below the coal. Those areas of the mire that have been penetrated by marine waters may be identified in the resultant coal by high sulfur, hydrogen, and nitrogen contents.

As a corollary to the mechanism of clastic contamination of peats, those raised mires that are able to keep pace with channel aggradation could confine the fluvial sediments to defined narrow courses. If this is so, the presence of thick peats could influence the depositional geometry of adjacent clastic accumulations, (Figure 2.8).

The majority of coals are developed from plants that have formed peat close to where they grew. Such coals are underlain by seatearths or rootlet beds and are known as autochthonous coals. However, coals that

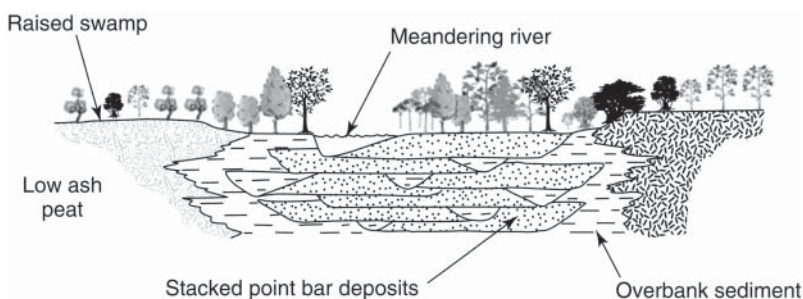
have formed from plant remains which have been transported considerable distances from their original growth site are known as allochthonous coals, e.g. large rafts of peat or trees drifting on lakes or estuaries. Allochthonous coals do not have an underlying rootlet bed; instead, they rest directly on the bed below. In the Cooper Basin, South Australia, thick Gondwana (Permian) coals show evidence of both autochthonous and allochthonous deposition. The allochthonous coals are closely associated with lacustrine sediments and are thick and widespread (B.P.J. Williams, personal communication).

### 2.2.3.1 Palaeobotanical Composition of Ancient Mires

The petrographic composition of a coal seam is genetically linked to the composition of its ancestral peat deposit. This is determined by the kinds of peat-forming plants and the biochemical conditions under which they were converted to peat.

Cellulose, pectin, and lignin form the bulk of material contained in plant cells and, therefore, are significant contributors to the composition of a coal seam.

The plant communities that make up the composition of peat have changed and evolved over geological time. Embryophytes (land plants) first appeared in the Late Silurian and Early Devonian, and are significant in becoming life forms on land rather than in water, although most began in swampy environments



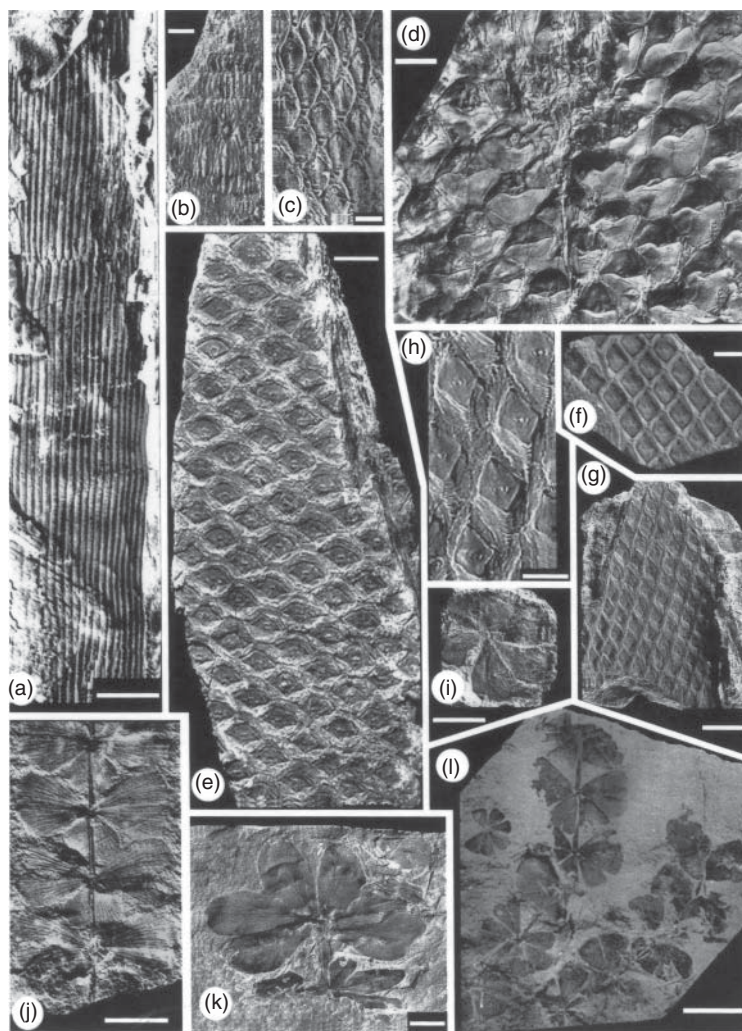
**Figure 2.8** Theoretical model of fluvial architecture in areas of raised swamps. The elevated swamp restricts overbank flooding and prevents avulsion, leading to the development of stacked channel sandstones. *Source:* From McCabe (1984). Reproduced with permission of Blackwell Scientific Publications.

and over time exerted a profound influence on fluvial sedimentation. Davies and Gibling (2010) described such influences on erosion, sediment transport, and deposition, links between biological and physical systems that in turn would have influenced the natural selection of plant types. Charcoal is known from the Pridolian (Late Silurian) of the Welsh Borderland and from the Emsian (Early Devonian) of Quebec; and the carbonaceous shales of Emsian age found in the Eifel (Germany) contain thin layers of vitrinite derived from land plants (Diessel 1992). Coal accumulation reached a peak in the Carboniferous Period in the Northern Hemisphere. This was a period of slow and repeated subsidence in tectonic basinal settings. The predominant plant group was the pteridophytes, consisting of lycopsida (lycopods), sphenopsida (horsetails), and pteropsida (true ferns). These are all wetland plants with shallow root systems susceptible to changes in the groundwater levels. A drop in groundwater level resulted in such vegetation dying back, and this accounts for the numerous thin stringers and bands of coal found throughout the Carboniferous coal measures of Europe. Collinson and Scott (1987) described those plant features that influence peat formation as being anchoring systems, reproductive biology, leaf and shoot biology, and the detailed structure of woody axes. The lycopsids were not a diverse group; they had a poorly developed root system, of which *Stigmaria* is an example. Other associated forms had root systems, details of which are poorly known. The lycopsids reproduced using the heterosporous technique, i.e. the ability to produce both megaspores and microspores (e.g. *Lepidocarpon* and *Sigillaria*), whereas some ferns were homosporous, producing only one kind of spore. All of these groups are thought to have had difficulty surviving in drier environments. Raymond et al. (2010) examined cordaites from the Carboniferous of the USA. These are an extinct group of gymnosperm trees and shrubs characterised by large strap leaves and woody stems and

were seed bearing. Their nearest living relatives are the modern conifers. Plants in cordaite-dominated peats probably grew in coastal mires in climate zones with seasons of low rainfall. In the Middle and Late Pennsylvanian in Eastern Kentucky, coal-forming floras consist of a diverse assemblage of large and small lycopod trees and tree ferns, characteristic of rheotrophic–ombitrophic mires, together with domed mires (precipitation dependent) with perched water tables. The range in diversity may be a response to changes in nutrient availability, or variations in soil character, together with changes in water table levels (Johnson et al. 2017). Some authors interpret such cordaite-rich peats as indicative of mangrove habitats.

Zhao and Wu (1979) examined Carboniferous macrofloras from South China and established a *Lepidodendron gaolishense–Eolepidodendron* assemblage for the early Carboniferous, and *Neuropteris gigantea–Mariopteris acuta* f. *obtusata* assemblage for the middle Carboniferous. The late Carboniferous is represented by transgressive marine strata with no plant content. Wang (2010) studied the late Palaeozoic (Carboniferous–Permian) macrofossil assemblages in the Weibei Coalfield, Central Shaanxi Province, People’s Republic of China (PRC). Four floral assemblages were established, each reflecting the impact of climate changes, the so-called ‘icehouse–greenhouse’ climatic changes (Gastaldo et al. 1996). Within these assemblages, some plant types are present throughout the period; for example, species of *Lepidodendron*, *Stigmaria*, *Sphenophyllum*, *Calamites*, and *Cordaites* (Figure 2.9), as well as forms of *Pecopteris* and *Neuropteris*. Correlation of these Cathaysian floras with Euroamerican floras is still problematic. This is in part influenced by the very nature of sedimentation in peat-forming areas, creating the persistent problem of difficulties in correlation between chronostratigraphic and lithostratigraphic units.

**Figure 2.9** Late Palaeozoic plant assemblages, Weibei Coalfield, PRC. (A) *Calamites cistii* Brongniart, Upper Shihhotse Formation; (B) *Calamites cf. schutzeiformis* Longmans, Upper Shihhotse Formation; (C) *Lepidodendron tienii* (Lee), Taiyuan Formation; (D) *Lepidodendron oculus-felis* Abb. Lower Shihhotse Formation; (E) *Cathaysiodendron acutangulum* (Halle), Upper Shihhotse Formation; (F) *Cathaysiodendron nanpiaoense* Lee, Taiyuan Formation; (G, H) *Lepidodendron posthumii* Jongmans et Gothan, Shanxi Formation; (I) *Sphenophyllum thonii* Mahr, Shanxi Formation; (J) *Sphenophyllum speciosum* (Royle), Upper Shihhotse Formation; (K) *Sphenophyllum cf. sinense* Zhang et Shen, Upper Shihhotse Formation; (L) *Sphenophyllum emarginatum* Brongniart, Lower Shihhotse Formation. Source: (Wang 2010). Reproduced with permission of Elsevier Publications.

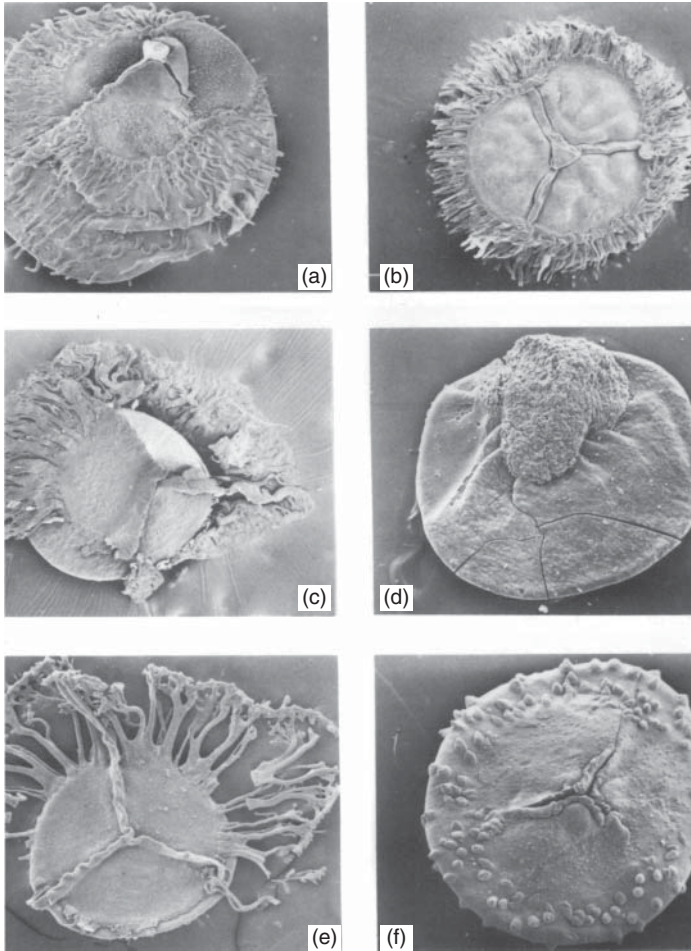


Bartram (1987) studied the distribution of megaspores (Figure 2.10) and the relationship to coal petrology using the Low Barnsley Seam (Westphalian B) from Yorkshire, UK. Six megaspore phases were recognised within the Barnsley Seam, suggesting a floral progression with changing environment (Figure 2.11). However, no positive correlation was found between individual species and lithology. Smith (1968) produced a profile through a Carboniferous coal seam showing miospore phases and petrographic types (Figure 4.4). The leaf and shoot biology will affect peat-forming environments. Frequent leaf fall will allow

continuous decomposition, whereas seasonal leaf fall may prevent further decomposition in the lower layers of leaf litter. Leaf fall was not characteristic of Carboniferous plants, unlike the later floral forms of the Cretaceous and Palaeogene–Neogene Periods. The early plants had a different internal anatomy from the later flora, and such differences may have influenced decomposition rates.

Post-Carboniferous vegetation adjusted to a larger tolerance of groundwater levels, which enabled thicker accumulations of coals to occur; e.g. the Permian coals of the Sydney Basin, Australia (Diessel 1992). The Permian



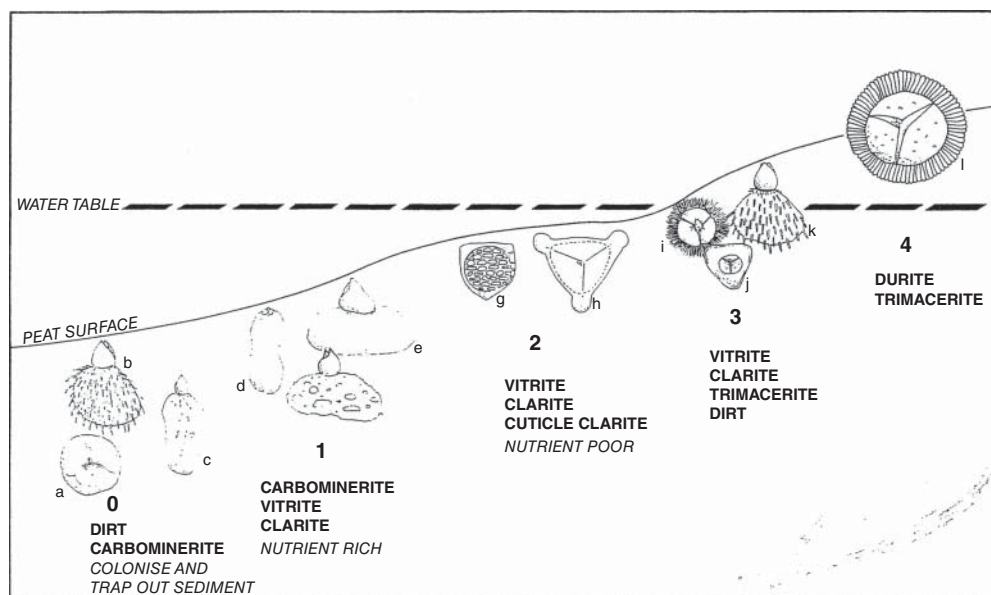


**Figure 2.10** Selected megaspores from the Low Barnsley Seam. (a) *Lagenicula subpilosa* (Ibrahim) Potonie & Kremp  $\times 50$ ; (b) *Setosporites hirsutus* (Loose) Ibrahim  $\times 50$ ; (c) *Zonalesporites brasserti* (Stach & Zerndt) Potonie & Kremp  $\times 25$ ; (d) *Cystosporites varius* (Wicher) Dijkstra  $\times 50$ ; (e) *Zonalesporites rotates* (Bartlett) Spinner  $\times 50$ ; (f) *Tuberculatisporites mamillarius* (Bartlett) Potonie & Kremp  $\times 25$ . Source: (Bartram 1987). Reproduced with permission of the Geological Society.

Period was dominated by the spermatophyta (seed plants); these had existed in the Carboniferous, but the Permian Period marked the end of the dominance of pteridophytes. Pteridosperms (seed ferns) reached their maximum development in the late Carboniferous and Permian Periods (Gondwana) in the Southern Hemisphere. This led to the formation of large, extensive coal deposits characterised by the plant *Glossopteris*, after which the flora is named. Iannuzzi (2010) has reviewed Early Permian (Gondwana) floras in the Parana Basin, Brazil. Post-glacial global warming during the Permian led to the appearance of spore-producing lycophytes (e.g. *Brasilodendron*, pecopterids, and sphenopterid ferns) and pollen-producing

glossopterids (Figure 2.12) together with an increase in sphenophytes (leaf-bearing) plants. Cordaites and lowland-habitat conifers continued from the Carboniferous into the Permian without significant change. These floras represent swamp (lycophytes, sphenophytes), floodplain (sphenophytes, glossopterids), and elevated terrains or more upland plant communities (conifers).

Tian (1979) examined coal balls from the Late Permian of China; these contained well-preserved plants of the Cathaysian flora. The coal balls were considered to result from floating blocks of vegetation, saturated and accumulated in paralic peat bogs. *Psaronius* is preserved in great abundance and is



**Figure 2.11** Schematic diagram of idealised uninterrupted sequence of megaspore assemblages through a coal seam. *Source:* From Bartram (1987). Reproduced with permission of Geological Society.

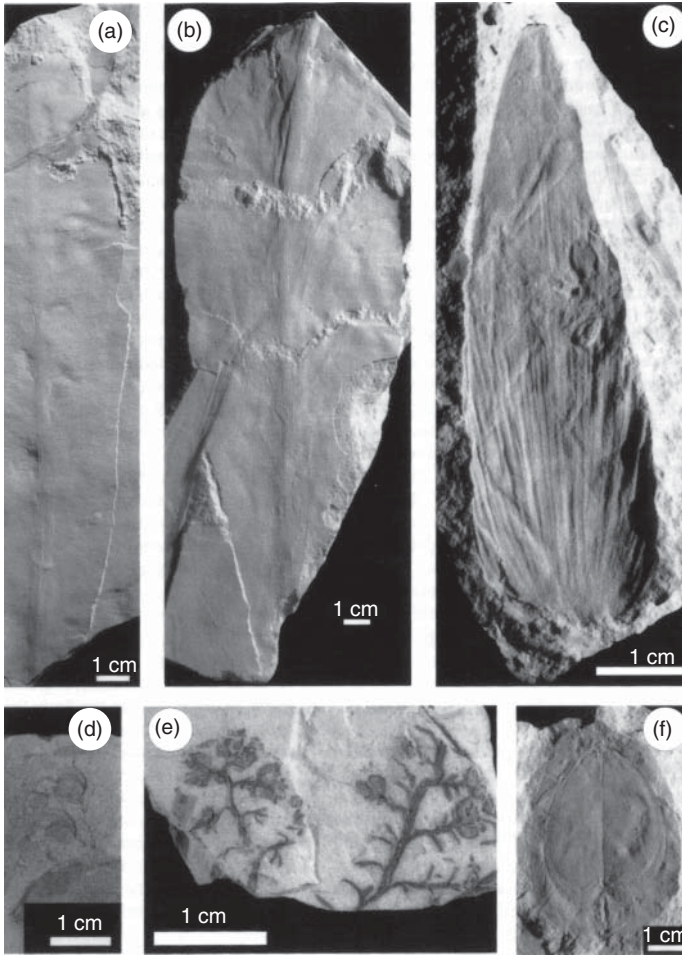
indicative of a hot, humid climate, such as tropical rain forest.

Gymnosperms, i.e. plants with naked seeds, became the dominant plant type from the Permian Period to the Cretaceous Period. Also during this time, the angiosperms, which first appeared in the Triassic, developed rapidly and are characterised by plants with covered seeds – namely the grasses, palms, shrubs, and trees, all of which make up the present-day vegetation. The large diversity of angiosperm pollen, fruits, and seeds meant a better resistance to changes in environment.

During the Palaeogene–Neogene Period, gymnosperms and angiosperms dominate. Duigan (1965) carried out a review of Palaeogene–Neogene brown coal flora from the Yallourn area of Victoria, Australia. In all, five gymnosperms and 11 angiosperms were identified. Patton (1958) concluded that the forests which formed this brown coal were predominantly coniferous, and deciduous trees, although occasionally present, were subordinate. Collinson and Scott (1987) also

stressed the importance of the taxodiaceous conifers in coal formation from Cretaceous to Recent. In central Europe, Kasinski (1989) described the formation of lacustrine lignite deposits in Poland. In his designated lignite microfacies, he listed the frequency of occurrences of taxodiaceous conifers and *Sequoia* wood tissue, water plant pollen grains and fungal spores, together with the related maceral content in each microfacies (see Table 4.11). More recently, Çelik et al. (2017), in a study of late Oligocene coals in the Thrace Basin (north-west Turkey), identified 5 gymnosperms and 20 angiosperms and interpreted the environment as one of deposition in freshwater mires in back mangrove areas and in a forest zone along rivers and lakes in moistured open areas where the palaeomire surface was covered with arboreal vegetation.

These studies all indicate the changes in the palaeobotanical make up of peats, and subsequently coal, and account for different maceral make-up in coals of differing geological age and geographical location.



**Figure 2.12** Early Permian macrofossil assemblages, Parana Basin, Brazil. (A) *Glossopteris occidentalis*; (B) *Gangamopteris obovata* var. *major*; (C) *Kawizophyllum* sp.; (D) *Arberia minasica*; (E) *Coricladus quiterensis*; (F) *Samaropsis gigas*. Source: (Iannuzzi 2010). Reproduced with permission of Elsevier Publications.

### 2.2.3.2 Case Studies

A number of studies of peat accumulation in tropical South-East Asia (Neuzil et al. 1993; Cecil et al. 1993; Ruppert et al. 1993; Gastaldo et al. 1993; Gastaldo 2010) show possible similarities between the extensive coastal plain peat deposits in Indonesia and Malaysia and the North American/European Carboniferous coal-bearing sequences in terms of sediment accumulation, mineralogy, geochemistry, and maceral content.

Cecil et al. (1993) produced an Indonesian analogue for Carboniferous coal-bearing sequences, in which it is suggested that the domed (convex upper surface) ombrogenous peat deposits of the ever-wet tropical area may represent the modern equivalent of Lower

to mid-Middle Pennsylvanian (Carboniferous) coal deposits of the eastern USA. In their study on the island of Sumatra, they conclude that peat formation has been controlled primarily by the allogenic processes of sea-level change and the modern ever-wet climate. Autogenic processes, such as delta switching, channel cutting, and barrier-bar migration, are considered to be of secondary importance as a control on the formation of peat. This is in contrast to the traditional model described earlier. The tropical climate is seen to favour formation of laterally extensive, thick, low-sulfur, low-ash peat deposits rather than active clastic sedimentation. Erosion and sediment transportation are restricted in the tropical rainforest environment. Gastaldo

(2010) compared the Rajang and Mahakam deltas in Borneo and observed that only the Rajang delta, on the western side of the island of Borneo, contains thick, extensive blanket peat. High degradation and ombrogenous mire accumulation rates have resulted in domed peat bodies up to 15 m in thickness, the surfaces of which reach heights above nearby channels. This contrasts with the organic accumulation in the Mahakam delta on the east side of Borneo. Gastaldo et al. (1993) and Gastaldo (2010) observed no autochthonous peat formation, but rather accumulations of allochthonous (derived and transported) peat on the lower delta plain tidal flats, the peats having derived from swamps and tropical forest higher on the delta plain. This is in part due to the deposition of expandable clays, a high proportion in the Rajang delta and a low proportion in the Mahakam delta, resulting in a regionally extensive aquiclude. Such an aquiclude promotes ombrogenous mire development and accumulation.

Neuzil et al. (1993) studied the inorganic geochemistry of a domed peat in Indonesia. The inorganic constituents in peat are the primary source for mineral matter in coals, and the study showed that large amounts of low-ash peat can develop close to marine conditions and above a marine substrate without high sulfur or pyrite contents. In domed ombromorphic peats, the geochemical controls on mineral matter are dominantly autogenic, independent of surrounding depositional environments. Neuzil et al. (1993) also considered that quality predictions for coal derived from domed peat deposits cannot be based on facies relations with enclosing sedimentary rocks. Rather, prediction of coal quality should be based on autogenic geochemical processes and controls of peat formation, recognised by the composition and distribution of mineral matter in coal.

Grady et al. (1993) carried out petrographic analysis on Indonesian peat samples and found that the optical characteristics of peat constituents are comparable to the maceral

content of brown coals. The distribution of maceral types in the modern peat was also found to be analogous to maceral profiles from Carboniferous coals in the USA and could be used to interpret the changing conditions in the original peat mire. Styant and Bustin (1983) studied the sedimentology of the Frazer River delta peat and used it as a modern analogue for some ancient deltaic coals.

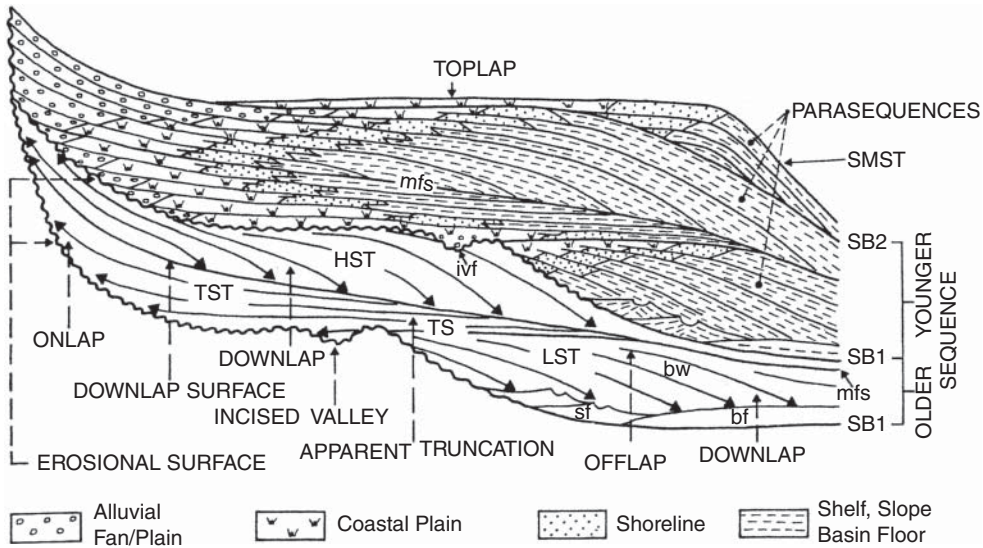
Studies of modern peat-forming environments, both in tropical and non-tropical areas, will continue to improve the understanding of coal-forming environments and, more importantly, the mechanisms for the retained accumulation of peat, its lateral development, and chemical constituents. This will have economic significance when applied to higher rank coals.

#### 2.2.4 Sequence Stratigraphy

The concept of sequence stratigraphy has been described by Vail (1987), Van Wagoner (1987), and Wilson (1991), and in relation to coal deposition by Diessel (1992). The principal four elements to be considered are eustatic sea-level change, basin subsidence, sediment supply, and climate. Sequence stratigraphic units consist of genetically related lithological sequences, referred to as parasequences. These parasequences and their depositional settings are then combined into systems tracts, identifiable in outcrops, borehole logs, and seismic profiles. Such systems tracts can be recognised by their three-dimensional (3D) geometries, including vertical characteristics and identifiable stratal surfaces. Parasequences are selected representing depositional types within differing systems tracts. Vail (1987) identified four kinds of system tract, and Diessel (1992) summarised this concept (Figure 2.13).

Systems tracts are a response to changes in sea level and availability of sediment supply. Changes in sea level are caused by rates of basin subsidence and variations in eustatic sea level. Eustatic changes have a strong influence on changes in shoreline morphology, and





**Figure 2.13** Cartoon illustrating the concept of sequence stratigraphy and its terminology. SB1, type 1 boundaries; TST, transgressive systems tract; LST, lowstand systems tract; HST, highstand systems tract; SMST, shelf margin systems tract; Ts, transgressive surface; ivf, incised valley fill; mfs, maximum flooding surface; sf, slope fan; pw, prograding composite wedge; bf, basin floor fan. *Source:* From Diessel (1992). *Coal-bearing Depositional Systems*, with permission from Springer-Science & Business Media.

when combined with climate they determine lithofacies types over geological time.

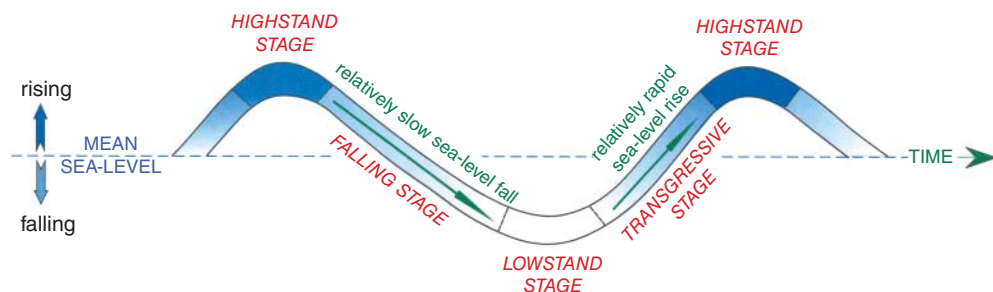
The result of a eustatic drop in sea level exceeding the rate of basin subsidence is erosion on the exposed shelf platform and associated land surfaces together with deposition on the sea-floor as deep-water sedimentary sequences in the form of a basin floor fan and slope fan with a greater development of prograding composite wedge sediments. These are nominated as part of a *lowstand systems tract*. The erosion of the shelf and adjoining areas plus the increase of surface water gradients produces deltas dominated by rivers with a high rate of progradation that comprise immature or reworked clastic sediments. This is further defined when deposition during relative sea-level fall produces a distinctive *falling stage systems tract* (FSST). This lies basinward of the highstand systems tract (HST) and is overlain by the lowstand systems tract (LST) (Plint and Nummedal 2000).

Figure 2.14 illustrates the relationship of rising and falling mean sea level and the relative

positions of highstand, falling stage, lowstand, and transgressive systems tracts (TSTs) and their resultant sediment profiles (George 2014).

In the opposite sense, a relative rise in sea level due to a eustatic rise combined with basin subsidence produces a TST. This marine transgression covers the delta plain and incised valley fill sediments of the lowstand systems tract, producing onlap sedimentation. This forms a maximum flooding surface that represents the upward termination of the TSTs. The periods of eustatic change may be short and numerous, producing a sequence of delta development and curtailment. The reversal of the eustatic rise in sea level in balance with basin subsidence produces periods when sedimentation occurs on the exposed shelf and adjoining upland, referred to as the HST. It is during this phase that coal formation will be maximised and sedimentation reflecting a regressive sequence will be established. These depositional settings of transgression and regression control coal formation, particularly the types of coal and their composition, and





**Figure 2.14** Simplified asymmetrical glacio-eustatic sea-level curves showing relative positions of highstand, falling stage, lowstand, and transgressive tracts (George 2014).

together with fluctuations in groundwater levels and the prevailing climatic conditions determine the extent and thickness of organic material that will be produced. Areas subject to episodic tectonic activity will produce coals of different properties to those formed in geologically quiescent environments.

Sequence stratigraphy has been applied to coal measure sequences by a number of authors in all the major coal deposits. Diessel (1992) makes the valid point that, in order to place coals in a sequence stratigraphic model, it is important to study the nature and genesis of those sediments enclosing a coal, the composition of the coal, and the lateral development of the coal, whether synchronous or diachronous.

Figure 2.15 (Diessel 1992) illustrates a number of systems tracts in a coal measure sequence. The pattern of coal seam development relates to small-scale changes in sea level. Such changes in water levels produce the features of variable coal thickness and coal seam splitting. The sequential position of coal seams in relation to the TST and HST illustrates the changes in the types of surrounding sediments to the coal seam. Coals are designated ‘T’ or ‘R’, depending on whether they have formed during a marine transgressive or regressive phase.

During the TST, the presence of seawater on or within the peat accumulation can produce high sulfur levels that may be preserved in the resultant coal seam. Coal maceral types may also vary dependent on whether the peat

accumulated in the transgressive or regressive phases.

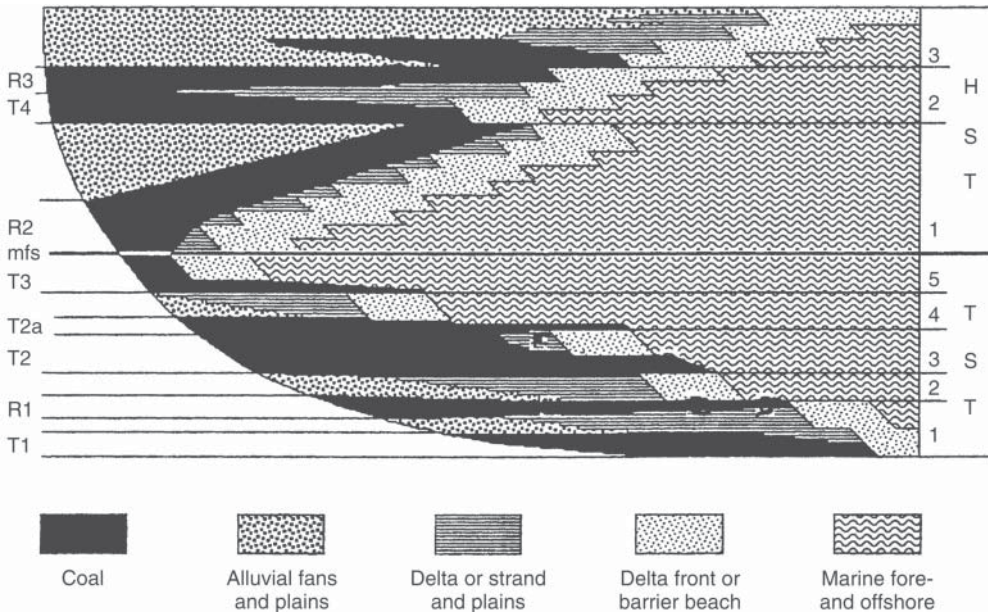
Flint et al. (1995) applied sequence stratigraphy to Westphalian coal measure sequences of the UK; their conclusions were that the development of regionally extensive mires required a slow rising water table, characteristic of a TST. Coals of economic significance occur principally in the TST of high-frequency sequences. Thick coals may be time equivalents of flooding surfaces.

### 2.2.5 Facies Correlation

The recognition of the variety of facies types described in the facies model is essential in order that their lateral and vertical relationships can be determined and correlated to produce the geometry of lithotypes within a study area.

In order to achieve this, examination of surface exposures, both natural and man-made, and borehole data are required to establish the particular lithological sequence present at each data point. It is the correlation between data points that is critical to the understanding of the patterns of coal development and preservation in any given area of interest.

It is an unfortunate fact that, for a great number of coal-bearing sequences, good recognisable and widespread marker horizons are rare. In part, this stems from the very localised patterns of deposition within many coal-forming environments. However, some distinctive deposits may be present;



**Figure 2.15** Diagrammatic model of a number of sequence stratigraphic settings of coal seams. TST, transgressive systems tract; HST, highstand systems tract;  $R_1$ – $R_4$ , marine regression;  $T_1$ – $T_4$ , marine transgression; mfs, maximum flooding surface. *Source:* From Diessel (1992).

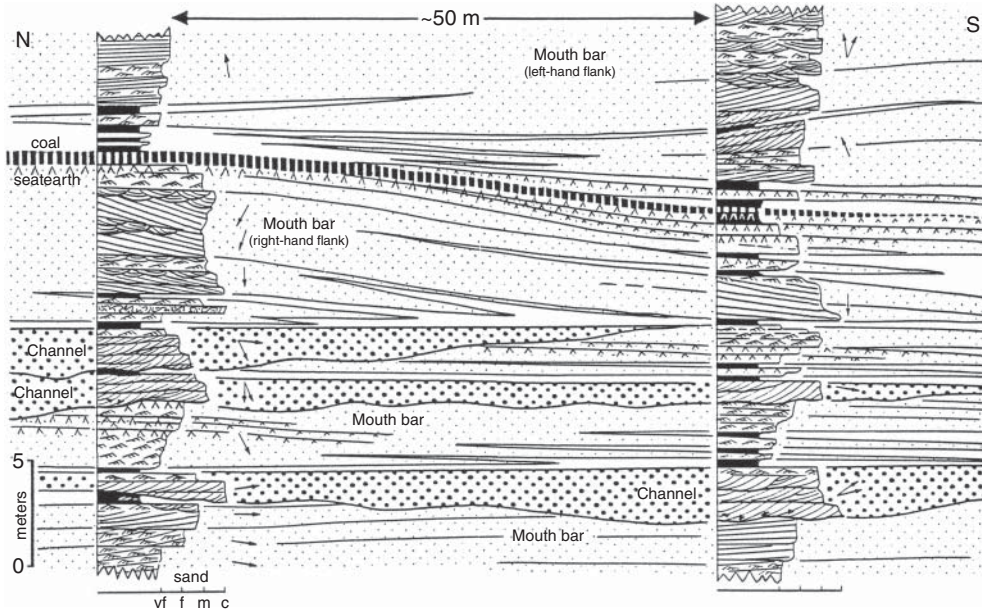
for example, marine mudstones, usually overlying a coal seam, may contain marine or brackish-marine fauna and may also have a particular geochemical/geophysical profile. In areas where contemporary volcanic activity took place during coal deposition, deposits of fine-grained volcanic ash intercalated with coal-bearing sediments produce widespread ‘tonstein’ horizons, which also have a distinctive geochemical/geophysical signature.

Other, less reliable, lithotypes can be used, certainly on a local scale (for example, within a mine lease area), such as sandstone complexes, freshwater limestones with their associated fauna, and the coal seams themselves.

Lithotype correlation from boreholes and surface exposures is dependent on the use of identifiable lithological horizons. Figure 2.16 shows correlation of irregular sand bodies within a coal-bearing sequence (Nemec 1992); such complexity makes individual coal seam correlation difficult, and in some cases impossible. The use of a widespread coal horizon is commonly used; however, owing to the

differential rates of sedimentation both above and below the chosen coal horizon, depiction of the sequence can result in distortions of the succeeding coal bed, as shown in Figure 2.17, where borehole sections have been used to determine the facies character of coal-bearing strata in the Arkoma Basin, USA (Rieke and Kirr 1984). Similar lithotype correlations are shown in Figure 2.18 from the USA and Figure 2.19 from India, the latter illustrated by the much-used ‘fence’ diagram presentation.

In modern coal sequence correlation, increasing use is being made of downhole geophysical logs of boreholes. The individual profile of each borehole can be compared with its neighbouring boreholes. An example of Canadian coal-bearing sequences showing the correlation of lithotypes with their geophysical profiles is shown in Figure 2.20. Details of the variety of geophysical logs used in coal sequence correlation are described in Chapter 8.



**Figure 2.16** Cross-section showing portions of superimposed mouth-bar lobes with associated seatearths and coal (Drønbreen locality, central Spitsbergen). The lobes prograde to the east (away from the viewer). Source: From Nemeč (1992).

Once the distribution pattern of the various lithotypes present in an area has been established, it may be possible to predict the likely sequence in adjacent areas. This is particularly important for neighbouring areas with proven coal reserves, which may be concealed beneath younger deposits, or which may lack quantitative geological data. If it is likely that coal is developed at economic thickness and depth, then a facies study of the known area may guide predictions for drill sites in adjacent areas. In the early stages of exploration, this can be an important tool to deploy.

In the example shown in Figure 2.21, Area 1 has a known distribution pattern of coal and non-coal deposits, determined from the correlation of the boreholes present. Area 2 is as yet unexplored, but from the data available in Area 1, together with an appraisal of the topography in Area 2, it is likely that Coal A will be present at similar depth and thickness, at least in that part closest to the nearest known data points in Area 1, i.e. at points 2a–2d. If Area 2 is considered for development,

exploratory drill sites would be located at sites 2a–2u before any close-spaced drilling would be sanctioned. Similarly, the areas of split coal and channel sandstone in Area 1 would need to be identified in Area 2 to determine how much coal loss is likely to occur here.

### 2.2.6 Facies Maps

In close association with the correlation of facies, the most significant sedimentological features for the coal geologist are those of seam splitting, washouts, and floor rolls, as well as the more obvious variations in seam thickness, seam quality, and interburden and overburden nature and thickness, together with the identification of igneous intrusions in the coal-bearing sequence. From borehole and surface data, all of these features can be quantified and portrayed in plan or map form.

Facies maps are usually compiled for the area of immediate interest, i.e. the mine lease area, but plans covering larger areas can be produced that give a useful regional picture



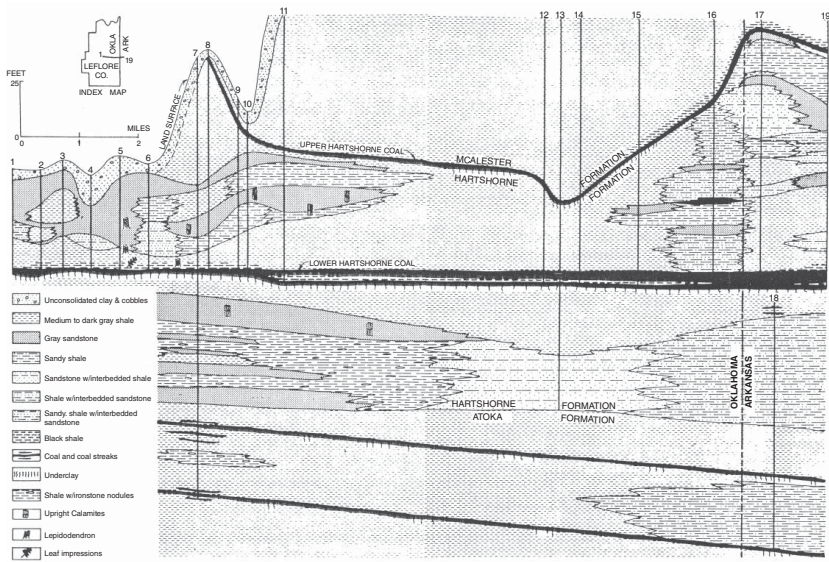
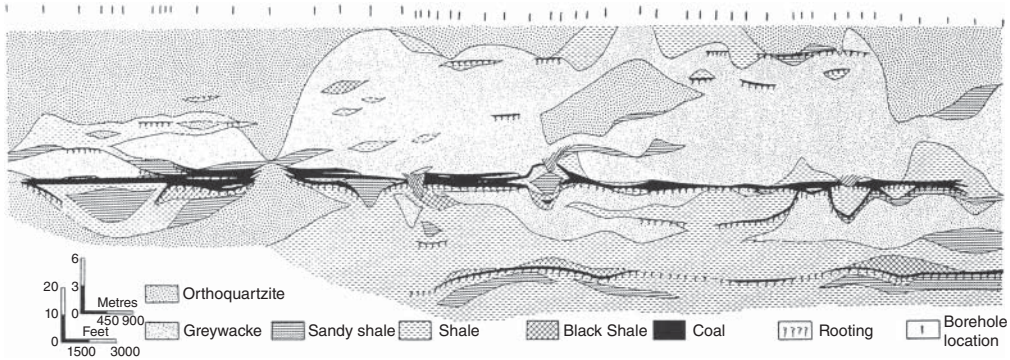


Figure 2.17 Cross-section showing stratigraphic relations of coals and sandstones in the Hartshorne and Atoka Formations, Le Flore County, Oklahoma, and Sebastian County, Arkansas, USA. Source: From Rieke and Kirr (1984).



**Figure 2.18** Cross-section showing correlation of lithofacies and associated coals above and below the Beckley Seam, West Virginia, USA, based on borehole data. *Source:* From Ferm et al. (1979).

of coal development. Such a large-scale study is shown in Figure 2.22, which illustrates a palaeogeographic reconstruction of the depositional setting of the Beckley Seam of southern West Virginia. The reconstruction is based on 1000 cored boreholes in an area of 1000 km<sup>2</sup>. In this example, coal thickness variations are closely related to the pre-existing topography, produced by depositional environments that existed prior to coal formation. The shape of the coal body also has been modified by contemporaneous and post-depositional environments, such as channels. Consideration of these features during mine planning can maximise the recovery of the thicker areas of coal whilst avoiding the areas of ‘want’, i.e. those areas depleted in coal.

Figure 2.23 shows a lithofacies map of part of the Patchawarra formation (Permian) in Australia, on which clastics-to-coal ratio contours are plotted, indicating areas of coal and no coal (or non-coal deposition). Figure 2.24 then shows the palaeogeographic reconstruction of the same interval, and the area of high clastics-to-coal ratio represents an area dominated by fluvial channel deposits (Thornton 1979).

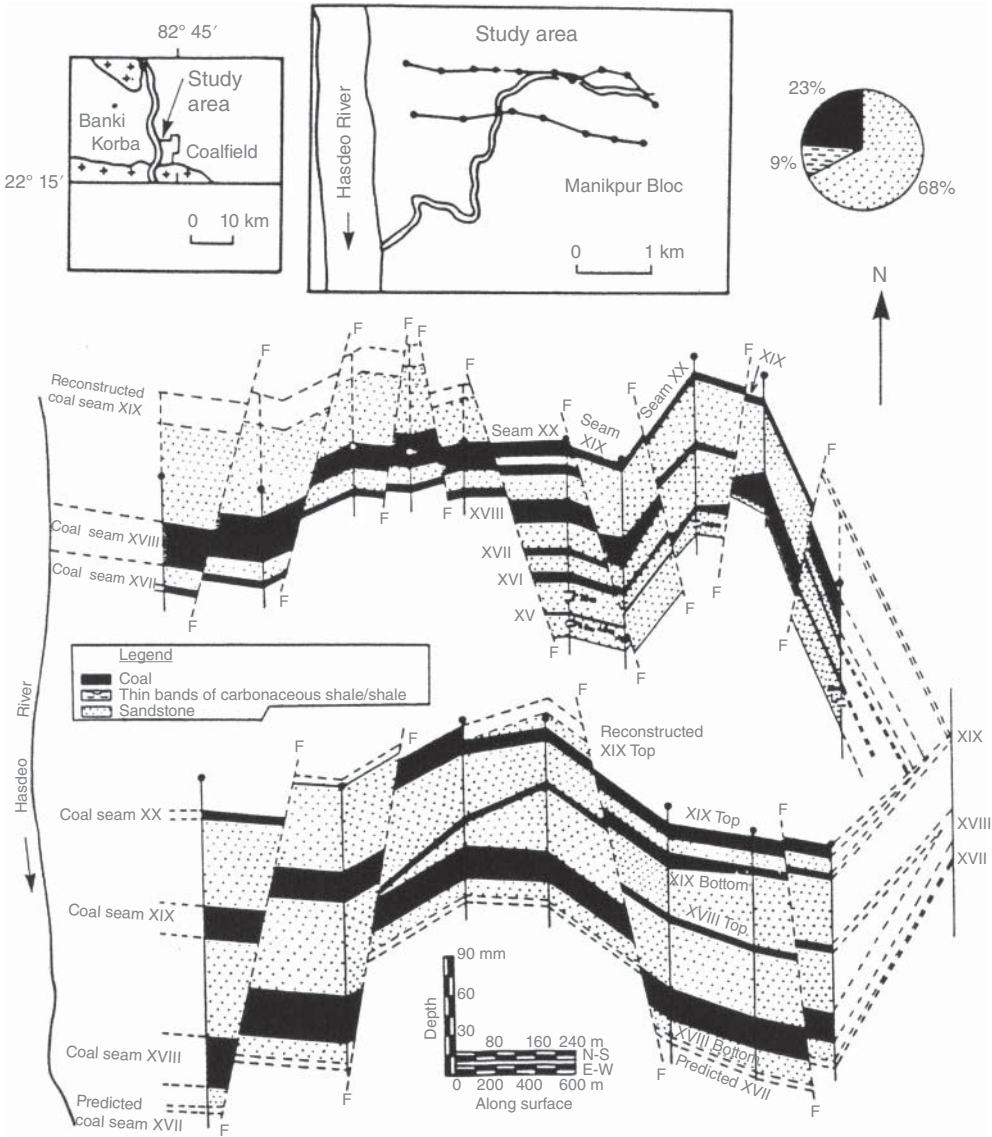
### 2.2.6.1 Seam Splitting

This common phenomenon occurs when a coal seam, traced laterally, is seen to ‘split’ into a minimum of two individual coals or ‘leaves’ separated by a significant thickness of non-coal

strata. Such non-coal materials within a seam are referred to as ‘partings’ or ‘bands’ and may composed of a variety of lithotypes. Such partings and bands are the result of clastic deposition replacing organic accumulation. They may represent crevasse-splay overbank deposits, or, if the partings are well developed laterally, represent either widespread flooding of the mire from adjacent river courses or periodic marine flooding into those mires close to the coast.

Seam splitting can be simple or form a complex series of layered organic and clastic materials. Simple splits occur when organic accumulation is interrupted and replaced for a short period by clastic deposition. Once the influx of detrital material ceases, vegetation is re-established, and organic accumulation thus continues. This may occur once or many times during the deposition of a coal seam. When traced laterally, splits may coalesce or further divide. This has the detrimental effect of reducing good sections of coal that can be mined, particularly if the partings are quartz rich, thus creating mining difficulties, particularly in underground workings. Figure 2.25 illustrates differential splitting of a coal seam across a mine working; such variations are significant to the economics of coalmining, more particularly in underground operations.

Other types of seam splitting are known as ‘S’ or ‘Z’ splits (Figure 2.26). This feature

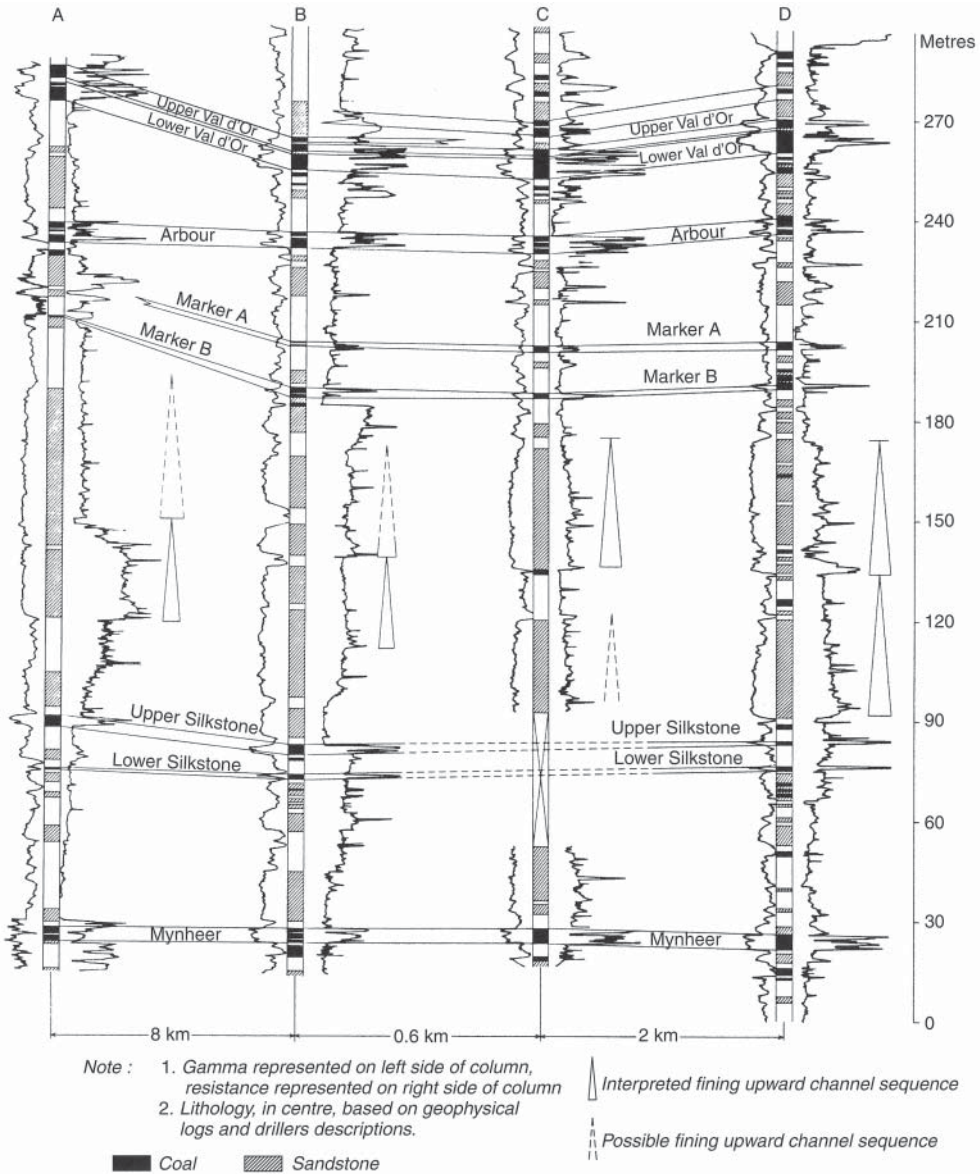


**Figure 2.19** Fence correlation diagram showing the geometry of a coal and sandstone sequence in the Lower Permian Barakar Formation, Korba Coalfield, India. *Source:* From Casshyap and Tewari (1984). Reproduced with permission of the author and Blackwell Scientific Publications.

is characterised by two seams usually separated by 20–30 m of sediment. The upper seam splits and the bottom leaf apparently descends through the clastic interval to unite with the lower seam. Such features are well documented in the UK (Fielding 1984) and Australia; they are considered to be produced

by accelerated subsidence induced by differential compaction of peat, clay, and sand-rich lithotypes that have been deposited in two adjacent ‘basin’ areas on the delta plain and require continuous peat formation and accumulation on the abandoned ‘basin’ surfaces and in inter-basin areas. Coal seam splits are also formed by





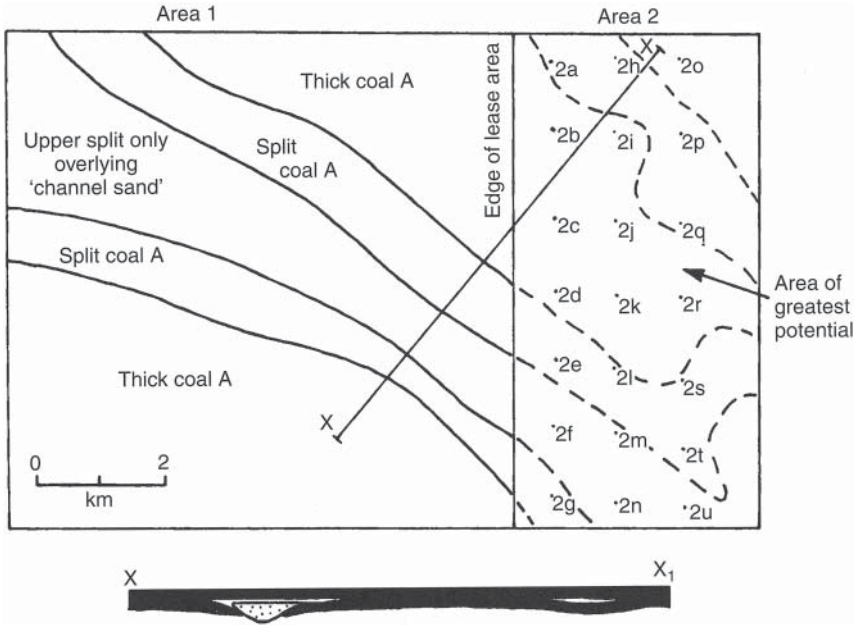
**Figure 2.20** Correlation based on geophysical logs, Coalspur Beds, Upper Cretaceous–Palaeogene, Alberta, Canada. *Source:* From Jerzykiewicz and McLean (1980), Geological Survey of Canada, Department of Energy, Mines, and Resources paper 79–12. Reproduced with permission of the Minister of Supply and Services, Canada.

the influence of growth faulting, as described in Section 2.3.1.2 and illustrated in Figure 2.33.

Splits are of considerable significance: coals that have been identified as being of workable thickness may in one or more areas split into two or more thinner seams that are

uneconomic to exploit. Such splitting effectively limits those areas of economically recoverable coal reserves.

High-angle splitting can produce instability, particularly in opencast workings, where mudstones or fractured sandstones overlying such



**Figure 2.21** Lithofacies map illustrating how such mapping can be extended to an adjoining area. To locate an additional area of thick coal and an area of coal split.

an inclined split may readily allow passage of groundwater and/or produce slope failure.

#### 2.2.6.2 Washouts

Washouts occur where a coal seam has been eroded away by wave or river current action and the resultant channel is filled with sediment. The coal may be wholly or partly removed by this process. Washouts are usually elongate in plan and infilled with clastic material, either as mudstone, siltstone, or sandstone depending on whether the erosive phase was followed by a reduction in current energy, so reducing the grain size of the sediment transported to infill the channel (Figure 2.27).

Initially, the edges of the washout tend to be sharp but then may have become diffused by differential compaction of the coal and non-coal materials.

Washouts are a major problem in mining operations, particularly in underground workings. Washouts can seriously reduce the area of workable coal; therefore, the delineation of such features is an essential prerequisite to mine planning. Detailed interpretation of the sedimentary sequences exposed in outcrops,

boreholes, and in underground workings allows a facies model to be constructed; this, in turn, may help to predict the orientation of washouts. Contemporaneous fault and fold influences during sedimentation can result in clastic wedges pinching out against these positive elements, with coal seams tending to merge over the structural highs. Another feature is the 'stacking' or localisation of channelling along the flanks of such flexures, producing elongate sandstone bodies that can influence mine planning operations.

#### 2.2.6.3 Floor Rolls

These are the opposite phenomenon to washouts. They are characterised by ridges of rock material protruding upwards into the coal seam. Like washouts, they reduce the mineable thickness of the coal seam. If they have to be mined with the seam, as is commonly the case, the dilution of the coal quality will result in an increase in the ash content. Floor rolls are often the result of differential compaction of peat around clastic deposits in the lower part of the seam as the upper part of the seam accumulates.



**Figure 2.22** Mapped lithotypes compiled from 1000 boreholes over an area of 1000 km<sup>2</sup>, illustrating the regional depositional setting of the Beckley Seam, West Virginia, USA. *Source:* From Horne et al. (1979).

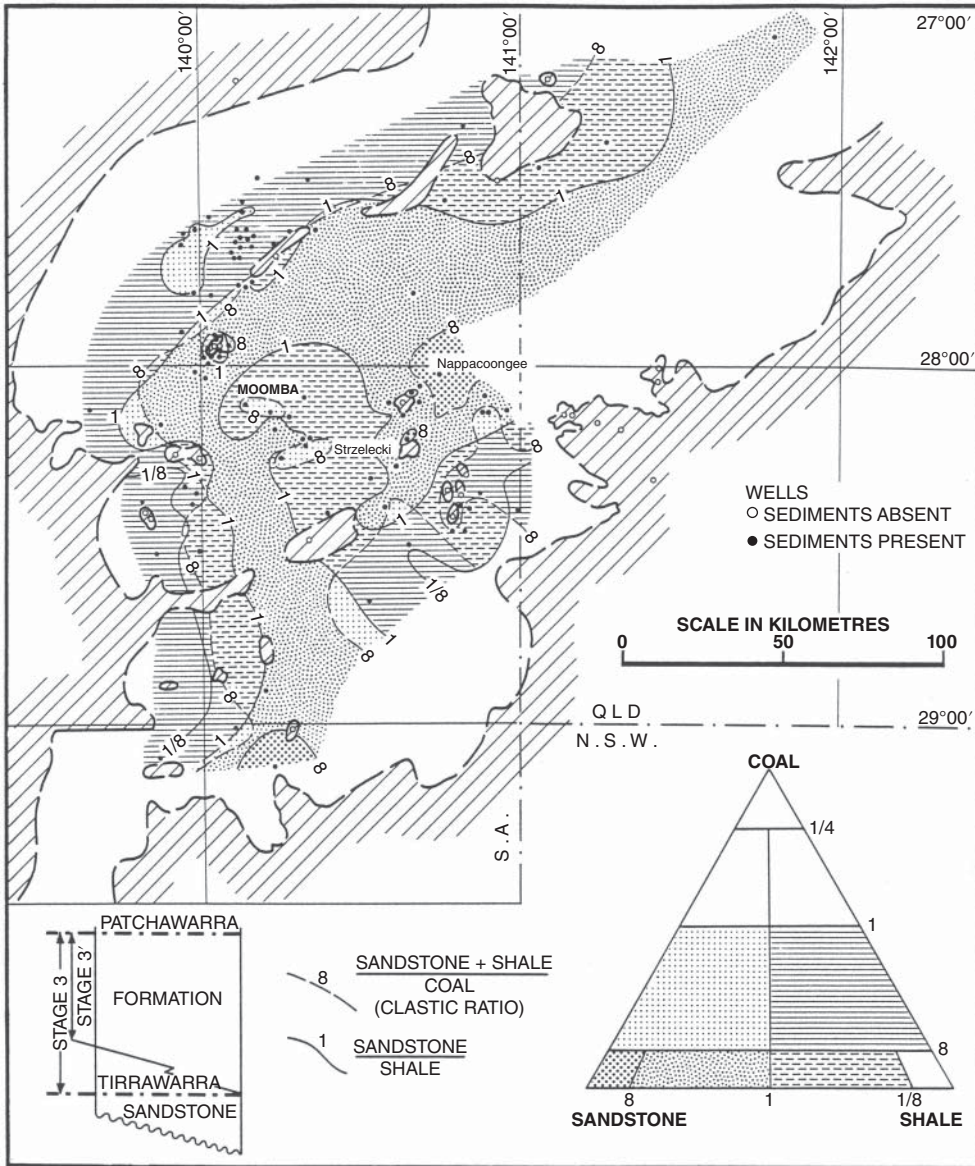


#### 2.2.6.4 Coal Seam Thickness Variations

The production of isopach maps of coal seams, sandstone thickness, or percentages of lithotypes present in any area of interest is an important guide to the eventual exploitation of coals. Figure 2.28 shows two contour maps of the north-eastern Fuxin Basin, PRC (Wu et al. 1992), in which sand percentage decreases to the west, which coincides with thicker coal development. Similar maps showing splitting of coal seams, their change in thickness and distribution, and the thickness and trend of

the non-coal interburden are essential to the mine planning process (see also Figure 2.23).

The importance of these is self-evident, since, depending on the economics of the mine site, coal less than a predetermined thickness will not be mined. This means that an area containing significant reserves may not be exploitable due to the thinness of a good mining section, particularly if the coal becomes inferior above and below the good coal section, i.e. makes a poor floor and roof to the seam. Conversely, there can be problems

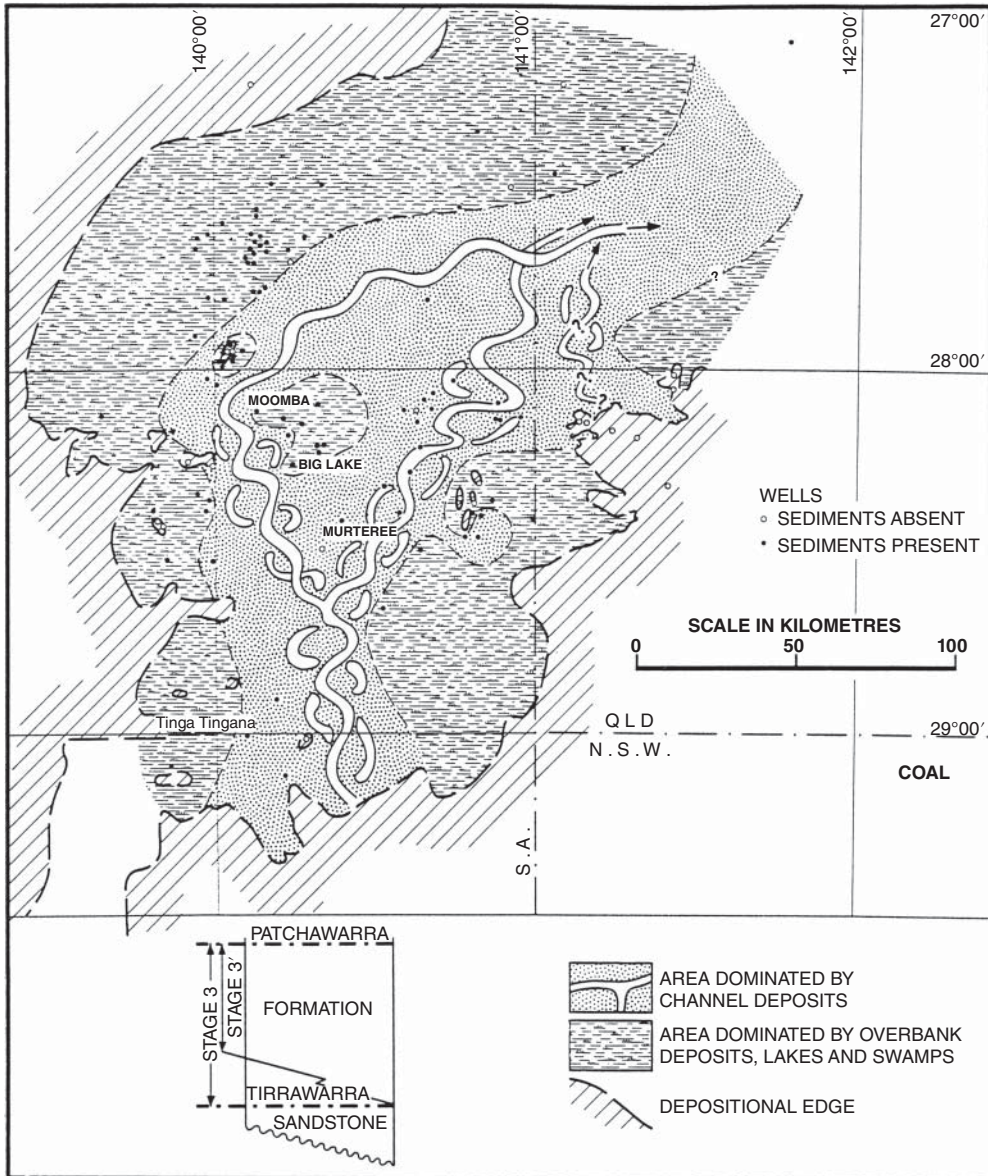


**Figure 2.23** Lithofacies map of part of the Patchawarra Formation (Permian) South Australia, showing sandstone/shale and sandstone + shale/coal ratios. *Source:* From Thornton (1979).

with an excessively thick coal in underground conditions producing poor roof or floor conditions. In opencast mines, thick coals are desirable, and the geotechnical nature of the overlying and underlying strata is important, particularly with regard to water movement and collection, as well as ground and slope instability.

Figure 2.29 shows an example of the variations in thickness of a coal in which the areas of thick/thin coal are clearly defined. The areas of coal thinning may indicate the attenuation of the seam or that the seam is splitting, producing a thinner upper leaf. Such occurrences are influential on the siting of mining panels in underground workings, and





**Figure 2.24** Palaeogeographic reconstruction of the same interval as shown in Figure 2.23, indicating the area of high clastics-to-coal ratio represents an area dominated by fluvial channel deposits. *Source:* From Thornton (1979).

they also affect the coal/overburden ratios in opencast operations.

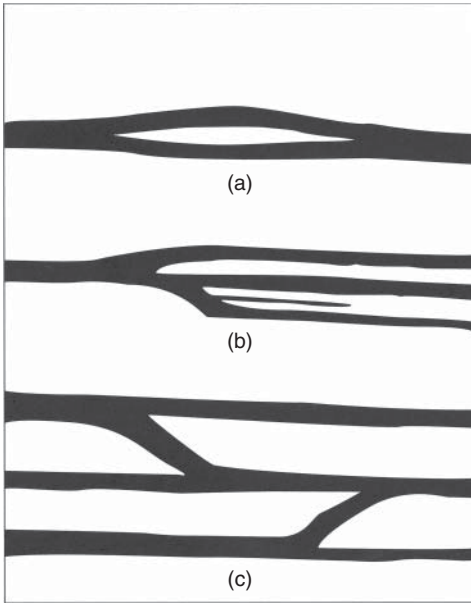
#### 2.2.6.5 Interburden/Overburden Thickness

The amount and nature of the lithotypes present between coal seams and between

the uppermost coal seam and the present land surface all have particular relevance to opencast mining operations. If the ratio of the thickness of such sediments to the thickness of workable coal is excessive then the deposit will be deemed uneconomic. Such



**Figure 2.25** Development of a coal seam splitting in the Beckley Seam across a mine working. *Source:* From Ferm et al. (1979).



**Figure 2.26** Common types of coal seam split: (a) simple splitting; (b) multiple splitting; (c) 'Z' - 'S'-shaped splitting.

ratios are variable and may be dependent on other costs, such as labour and transport. Most desirable coal/interburden/overburden ratios are in the order of 2:1–5:1, although they may be higher in certain circumstances, i.e. 10:1–15:1. In addition, if the lithotypes include hard indurated sandstone that will require blasting, then this is an added cost that has to be allowed for in the economic appraisal of the coal deposit.

#### 2.2.6.6 Coal Seam Quality Variations

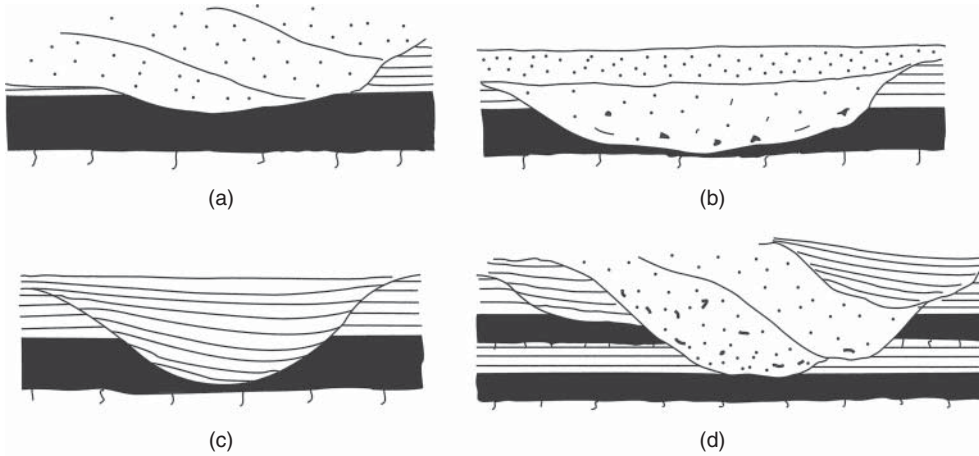
Variations in the environments of deposition strongly influence the resultant quality of coals. As described in Section 2.2.3, peat mires

can intermittently receive influxes of detritus by marine invasion, overbank flooding, or from airborne sources such as contemporaneous volcanism. Such occurrences will cause all or part of the coal seam to contain a higher ash level, which may be local or widespread. If the peat mire has been invaded by marine waters for a long period of time, precipitation of minerals into the uppermost part of the peat is likely; in particular, the sulfur content in those parts of a coal seam so affected may be greatly increased.

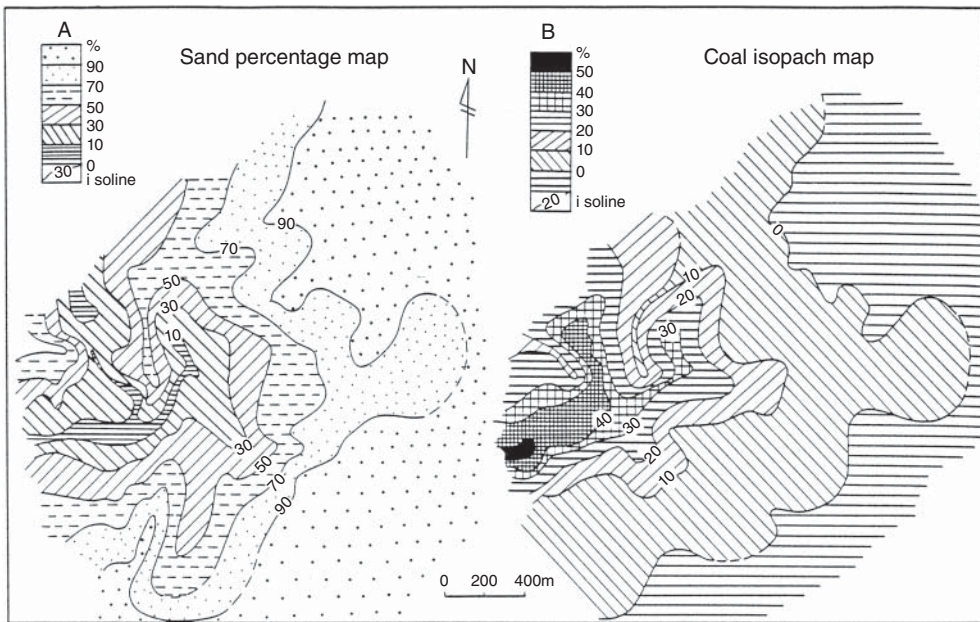
The plotting of coal seam quality parameters will not only give an indication of their distribution but also indicate the palaeoenvironmental influences that existed during the depositional and post-depositional phases of coal formation. Conversely, the interpretation of the palaeoenvironment will help to predict coal quality in selected areas. That is, those areas considered distant from marine influence should have lower sulfur contents, and coals deposited away from the main distributary channels and only subjected to low-energy currents can be expected to have lower ash contents.

The above relationships have been summarised in the literature as follows: rapid subsidence during sedimentation generally results in abrupt variations in coal seams, but is accompanied by low sulfur and trace-element content, whereas slower subsidence favours greater lateral continuity but a higher content of chemically precipitated material.

The examination of the coal quality analyses, particularly from cored boreholes, allows the coal geologist to plot coal quality variations



**Figure 2.27** Channelling in coal seams. (a) Sand-filled channel producing a sandstone roof to the coal seam. (b) Sand- and coal-detritus-filled channel with coal seam eroded. (c) Mudstone-filled channel with coal seam eroded. (d) Multiple channel sequence with sandstone and mudstone fills; the channel has removed the upper leaf of the coal seam.

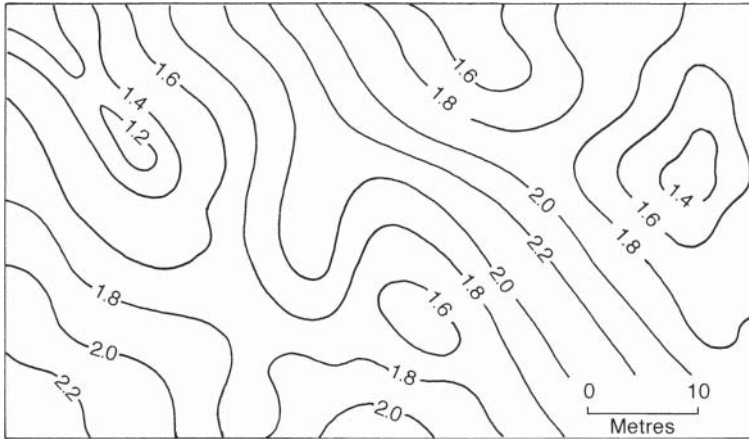


**Figure 2.28** Isopach maps of the north-eastern Fuxin Basin, PRC, showing (a) percentage of sandstone and (b) coal isopachs for the middle part of the Haizhou Formation (Lower Cretaceous) showing the relationship of increased sandstone thickness with decreasing coal thickness. *Source:* From Wu et al. (1992).

across the study area. The parameters that are particularly relevant are volatile matter, ash, and sulfur content. Deficiencies in the volatile matter content (notably in proximity to igneous intrusions) and too high amounts of

ash and sulfur can lead to the coal under consideration being discarded as uneconomic due to increased costs of preparation or by simply just not having those properties required for the market that the coal is targeted for.





**Figure 2.29** Coal seam thickness isopach map; hypothetical example (thickness values are in metres).

## 2.3 Structural Effects on Coal

Any significant lateral or vertical structural change in a coal seam has a direct bearing on its thickness, quality, and mineability.

Such changes can be on a small or large scale, affect the internal character of the coal, or simply displace the coal spatially, replacing it with non-coal sediment, or, in certain circumstances, with igneous intrusives. Disruption to coal seam thickness and continuity can lead to the interruption or cessation of mining, which will have economic repercussions, particularly in underground mines where mining flexibility is reduced. Therefore, an understanding of the structural character of a coal deposit is essential in order to perform stratigraphic correlation, to calculate coal resource/reserves, and to determine the distribution of coal quality prior to mine planning.

### 2.3.1 Syndepositional Effects

The majority of coal-bearing sediments are deposited in or on the margins of tectonic basins. Such a structural environment has a profound influence on the accumulating sediments, both in terms of the nature and the amount of supply of detrital material required to form such sequences and on the distribution and character of the environments of sedimentation.

In addition, diagenetic effects within the accumulating sediments produce structural deformation; this may be due to downward pressure from the overlying strata and may be combined with water loss from the sediments whilst still in a non-indurated or plastic state.

#### 2.3.1.1 Microstructural Effects

The combination of thick sediment accumulation and rapid basin subsidence can produce instability, particularly along the basin margins.

The effects on coal-bearing sediments are frequently seen in the form of slumping and loading structures and in liquefaction effects, the latter being characterised by the disruption of bedding laminae and the injection of sediment into the layer above and below. Under such loading effects, coal may be squeezed into overlying strata and the original seam structure may be completely disrupted. In addition, coals may be injected by surrounding sediment in the form of sedimentary dykes. Interbedded sequences of mudstone, sandstone, and coal that have undergone loading deformation exhibit a variety of structures, such as accentuated loading on the bases of erosive sandstones, flame structures, distorted and dislocated ripples, and folded and contorted bedding (Figure 2.30).

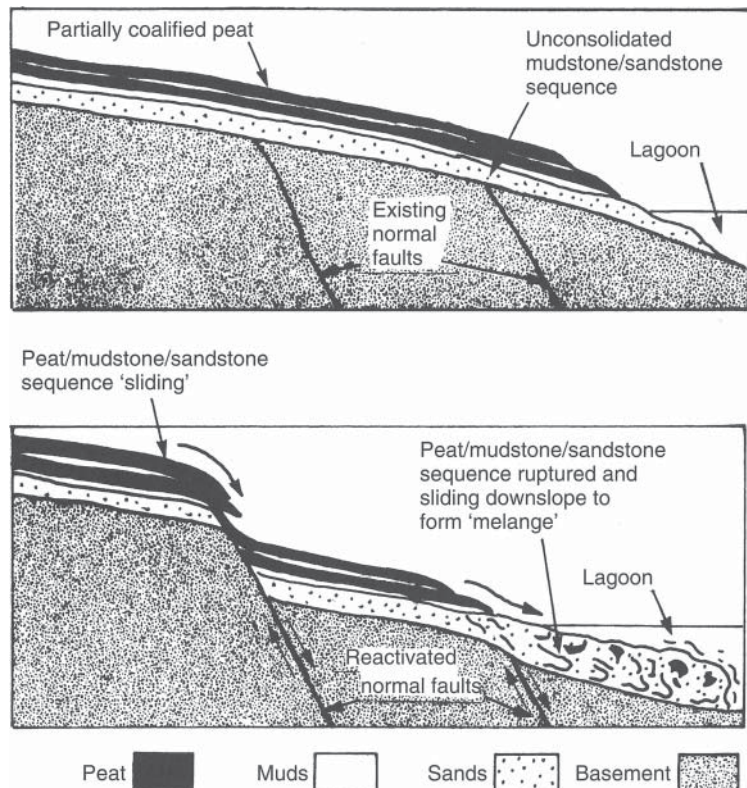
Instability within environments of deposition, whether induced by fault activity or



**Figure 2.30** Deformed bedding in Palaeogene–Neogene coal-bearing sediments, East Kalimantan, Indonesia. Hammer length is 37 cm. Source: Photograph by LPT.

simply by overloading of accumulated sediment, can produce movement of sediments in the form of gravity flows. Figure 2.31 illustrates such a phenomenon. If a coal is transported in this fashion, the result can be an admixture of coal material and other sediment with no obvious bedding characteristics. Figure 2.32 shows a coal that has become intermixed with

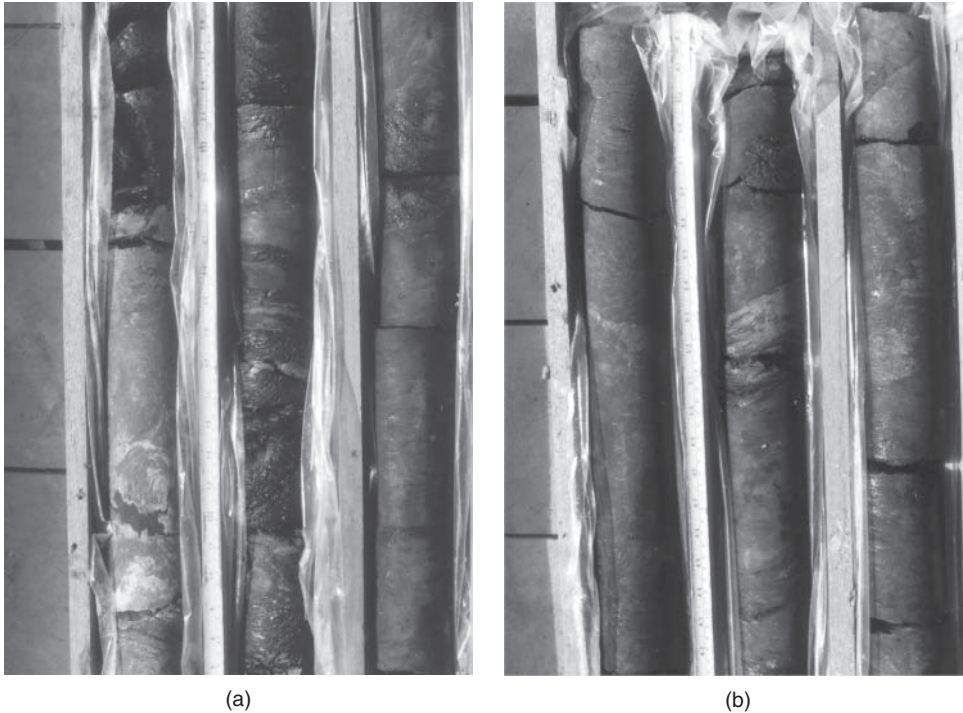
**Figure 2.31** Normal-fault reactivation causing instability in a partially coalified peat sequence; downslope slumping produces a 'melange' of coal and intermixed sediment.



the surrounding sediment; it now is in an unworkable state as the ash content is too high and the geometry of the coal seam is irregular.

### 2.3.1.2 Macrostructural Effects

Within sedimentary basins, existing faults in the underlying basement may continue to be active and influence the location, thickness, and character of the sedimentary sequence. Many coal-bearing basinal sediments display evidence of growth faulting. In West Virginia and Pennsylvania, USA, broad-scale tectonic features have caused local thickening of the sequence in response to an increased rate of subsidence, as distinct from more stable platform areas (i.e. less rapidly subsiding), where sedimentation prograded rapidly over the shelf. In South Wales, UK, growth faults have again influenced sedimentation; here, in addition to active basement elements, faults are developed that owe their origin to gravity sliding within the sedimentary pile (Elliott and



**Figure 2.32** Cores exhibiting a 'melange' or mixing of lithotypes due to gravity sliding in Palaeogene–Neogene coal-bearing sediments, East Kalimantan, Indonesia. (a) Left core: mixing of sandstone and siltstone with subordinate coal. Centre core: coal mixed with mudstone and siltstone. Right core: coal and mudstone mixing. (b) All cores: sandstone, siltstone, and coal mixing. *Source:* Photographs by LPT.

Lapido 1981). Overpressured, non-compacted argillaceous sediments initiate faults on gentle gradients. Such faults tend to have a curved cross-sectional profile, steep at the top and flattening progressively into bedding plane faults, often along the roof of a coal. In many cases, such faults are partially eroded before the succeeding sediments are laid down.

Seam splitting can also, in certain circumstances, be attributed to growth faulting. Reactivation of faults with changes in the sense of movement can result in the downwarping of sections of peat beds; this is then followed by non-peat deposition on the downwarped section, and then peat deposition is resumed at the original level of the first peat. Figure 2.33 shows the possible mechanism for the formation of such a coal seam split.

Periodic changes in base level in deltaic areas through fault activation will result in changes

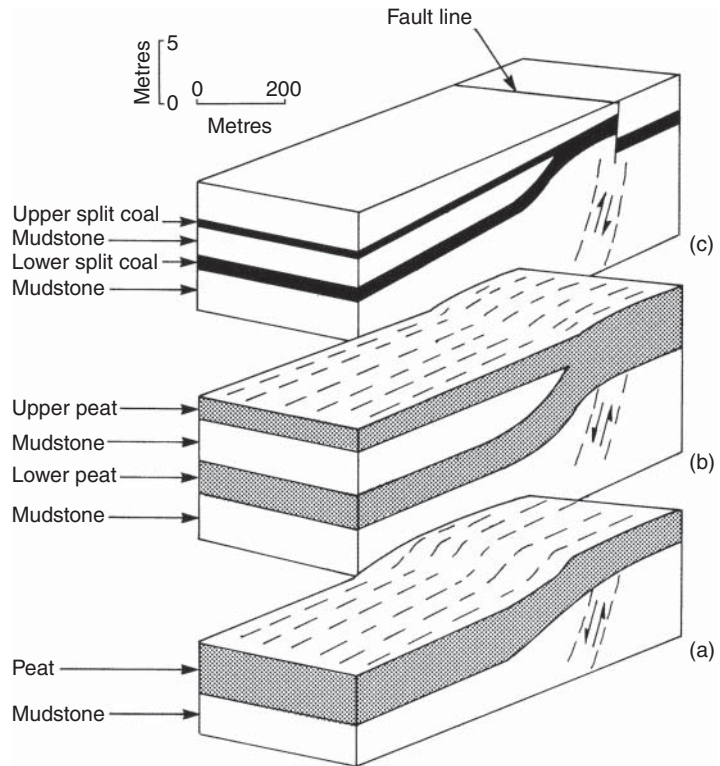
in the development and character of coals. With emergence, coals may become more extensive and, where the influx of detritus is curtailed, have a lower ash content. If submergence occurs, coals may be restricted areally or receive increased amounts of detritus that may increase the ash content, or even cease to develop at all. Furthermore, submerged coals may be contaminated with marine waters, which could result in a higher sulfur content in the uppermost parts of the seam.

Growth folds also influence the deposition patterns in coal basins; local upwarping can accelerate the rates of erosion and deposition in some parts of a basin, but they can also have the effect of cutting off sediment supply by uplift or by producing a barrier to the influx of detritus.

In very thick sedimentary sequences, the continued growth of such folds can result in the production of oversteepened fold axes.



**Figure 2.33** Seam splitting caused by differential movement of faults during peat deposition. (a) Fault downthrow results in downwarping of the peat. (b) Downwarp filled in with mudstone; peat development resumed at original level. (c) Fault throw sense reversed; uplifting split coal and downthrowing unsplit section of the coal seam. *Source:* From Broadhurst and Simpson (1983).

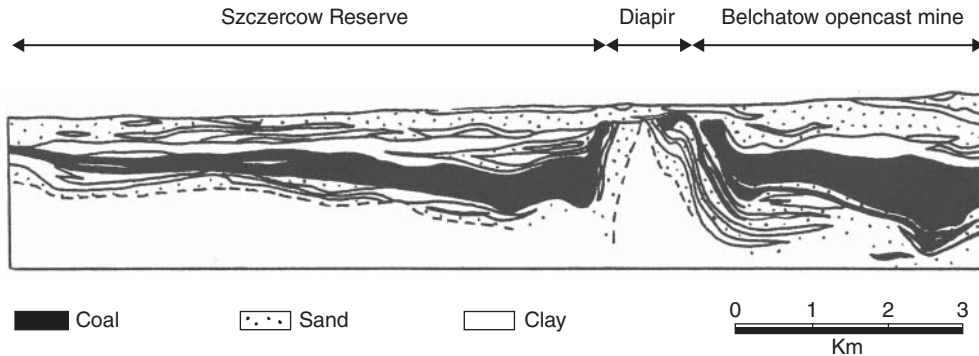


Where this occurs, overpressured mudstone at depth may be forced upwards and actually breach the anticlinal axial areas; this can be seen by the breaking up of the surface strata and the intrusion of material from below. Such diapiric intrusion breccia can be found in East Kalimantan, Indonesia, and this is often accompanied by the development of mud volcanoes along the axial region of the anticlines. Development of diapiric structures can disrupt as well as distort coal beds; in the Belchatow opencast coalmine in Poland, a large diapir has intruded into the coal-bearing sequence, dividing the coal reserve into two distinct areas (Figure 2.34).

In the Kutei Basin in East Kalimantan, Indonesia, the established structural pattern continually evolved throughout the Palaeogene–Neogene Period. In this area, the anticlines are tight with steep or overturned dips accompanied by steep reverse and normal faults in the complex axial regions. The

synclines are broad and wide with very low dips; the transition between the two structures can be abrupt, now represented by steep reverse faults. These growth folds are thought to have been further accentuated by gravity sliding associated with very thick accumulation of sediment (up to 9000 m) in the Kutei Basin and rifting in the Makassar Strait to the east. The structural grain and the palaeostrike were roughly parallel in this region; the resultant sequence is characterised in its upper part by upper delta plain and alluvial plain sedimentation with numerous coals. This structural pattern is shown in the map in Figure 2.48b (Land and Jones 1987).

Penecontemporaneous volcanism can also have a profound effect on the character of coals. Large amounts of airborne ash and dust together with waterborne volcanic detritus may result in the deposition of characteristic dark lithic sandstones, possible increases in



**Figure 2.34** Section across Belchatow opencast mine, Poland, showing effect of a diapiric intrusion into the thick coal seam. The diapir in this instance is a salt dome and effectively divides the coal reserves into two distinct areas.

the ash content in the peat mires, and the formation of tonstein horizons.

### 2.3.2 Post-Depositional Effects

All coal-bearing sequences have undergone some structural change since diagenesis. This can range from gentle warping and jointing up to complex thrust and folded coalfields usually containing high rank coals.

These post-depositional structural elements can be simply summarised as faults, joints (cleats), folds, and igneous associations. Mineral precipitation may also produce some changes in the original form and bedding of coal-bearing sequences.

#### 2.3.2.1 Jointing/Cleats in Coal

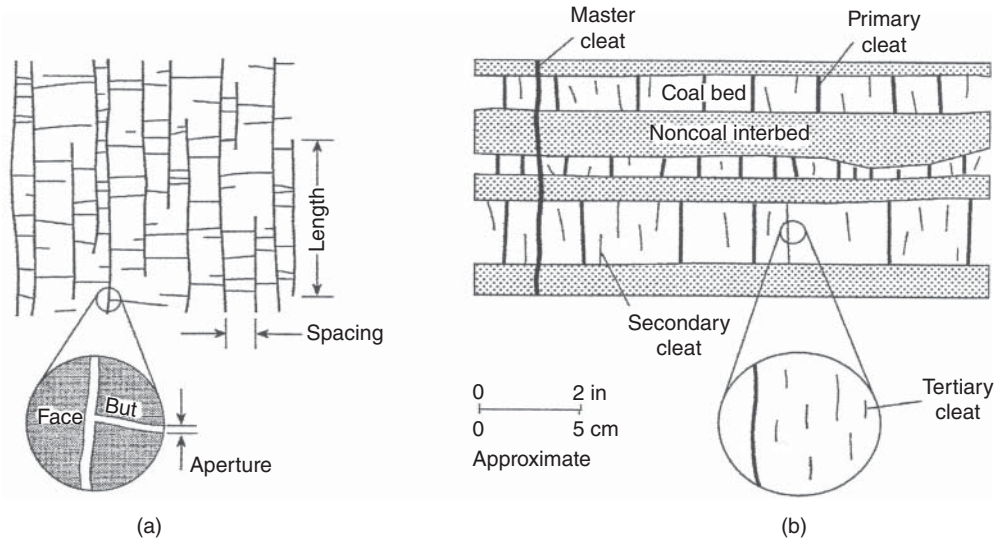
Coal, and particularly all ranks of black coal, is noted for the development of its jointing, more commonly referred to as cleats. This regular pattern of cracking in the coal may have originated during coalification, the burial, compaction, and continued diagenesis of the organic constituents results in the progressive reduction of porosity and permeability. At this stage, microfracturing of the coal is thought to be generated. The surfaces and spaces thus created may be coated and filled with mineral precipitates, chiefly carbonates and sulfides.

Cleats are fractures that occur in two sets that are, in most cases, mutually perpendicular, and also perpendicular to bedding.

Abutting relations between cleats generally show one set pre-dates the other (Figure 2.35). Through-going cleats are formed first and are referred to as face cleats; and cleats that end at intersections with through-going cleats formed later are called butt cleats. These fracture sets and partings along bedding planes impart a blocky character to coal. Figure 2.36a shows a well-developed orthogonal cleat pattern in a Carboniferous bituminous coal from the Midlands, UK, and Figure 2.36b shows a strong cleat pattern in an anthracite of Triassic age from the Republic of Vietnam. Figure 2.36c shows cleat development together with conchoidal fracture in a Palaeogene–Neogene brown coal from the Republic of Serbia. Figure 2.36d shows a well-developed cleat and joint pattern in a Gondwana bituminous coal from Central India. It is noticeable that cleats can be seen in all thicknesses of coal, even in the thinnest films of coal included within other lithotypes.

Cleats are subvertical in flat-lying beds and are usually orientated at right angles to the bedding, even when strata are folded. In a number of cases, cleats are confined to individual coal beds, or to layers composed of a particular maceral type. These are usually uniform in strike and arranged in subparallel sets that have regional trends (Laubach et al. 1998).

Research has shown that cleat spacing varies with coal rank, decreasing from lignite, via



**Figure 2.35** Schematic illustrating cleat geometrics. (a) Cleat-trace patterns in plan view. (b) Cleat hierarchies in cross-section view. The following conventions are used for cleat: length is the dimension parallel to the cleat surface and parallel to the bedding; height is parallel to the cleat surface and perpendicular to the bedding; aperture is the dimension perpendicular to the fracture surface; spacing between two cleats is the distance between them at right angles to the cleat surface. *Source:* From Laubach et al. (1998). Reproduced with permission from Elsevier Publications.

medium-volatile bituminous coal, increasing to anthracite coals. Law (1993) found cleat spacing ranged from 22 cm in lignites ( $R_0$  [vitrinite reflectance] values of 0.25–0.38%) to 0.2 cm in anthracites ( $R_0$  values >2.6%). Such spacing in higher rank coals may reflect competing processes of fracture formation and annealing. Tectonism may obliterate previously formed cleats in anthracites; if this is so, then regular variations in cleat spacing with rank in the lignite–high-volatile bituminous coal range might not occur. Another parameter that has been studied with regard to cleat spacing is coal type and ash content. Dawson and Esterle (2010) identified four major classes of cleats in coals from Queensland, Australia. These are master cleats, single vitrain (bright coal) cleats, multiple vitrain package cleats, and durain (dull coal) cleats. Bright coal lithotypes (vitrain) have smaller cleat spacings than dull coal lithotypes (durain) do (Stach 1982); and coals with low ash content have smaller cleat spacings than coals with high ash content

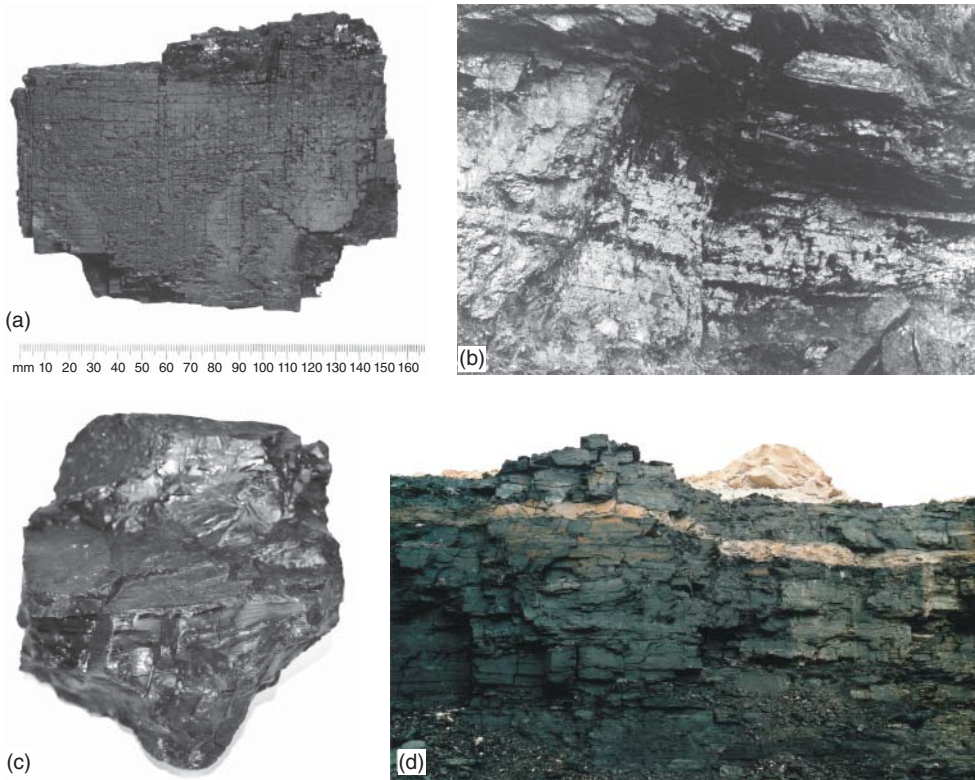
do. Laubach et al. (1998) have produced a comprehensive review of the origins of coal cleat.

### 2.3.2.2 Faulting

The development of strong joint and fault patterns in coal-bearing sequences is the commonest post-depositional structural expression. Briefly, the principal fault types are normal, reverse, and strike-slip faults.

*Normal faults* are produced by dominantly vertical stress resulting in the reduction of horizontal compression, leaving gravity as the active compression; this results in the horizontal extension of the rock sequence.

This form of faulting is common, movements can be on the order of a few metres to hundreds of metres. Figure 2.37 shows a normal fault with a throw of about 2 m. Such faulting is not too problematical in opencast workings, but larger throws can result in the cessation of opencast mining either locally or totally. Figure 2.38 shows a local highwall termination due to faulting in an opencast mine in Bosnia-Herzegovina, and Figure 2.39



**Figure 2.36** (a) Orthogonal cleat pattern in Meltonfleet Coal, Upper Carboniferous, Yorkshire, UK. *Source:* Reproduced with permission: IPR/25-6C British Geological Survey. © NERC. All rights reserved. (b) Well-developed joint and cleat pattern in Triassic anthracite, Republic of Vietnam. *Source:* Photograph by LPT. (c) Cleat development and conchoidal fracture in Palaeogene–Neogene brown coal, Republic of Serbia. *Source:* Photograph by courtesy of Dargo Associates Ltd. (d) Well-developed joint pattern in Permian (Gondwana) coal, Central India. *Source:* Photograph by courtesy of Dargo Associates Ltd.

illustrates a normal fault downthrowing overburden (light colour) against a coal seam in an opencast mine in India. In underground workings, even small-scale faulting can result in cessation of the mining of fully automated faces, resulting in loss of available reserves.

The dip of normal faults ranges widely. In coalfields, most are thought to be in the region of 60–70°. Some normal faults die out along their length by a decrease of throw towards either one or both ends of the fault. Again, a fault may pass into a monoclinical flexure, particularly in overlying softer strata. Such faulting also produces drag along the fault plane, with the country rock being pulled along in the direction of movement. Where

large faults have moved on more than one occasion, and this applies to all kinds of faulting, a zone of crushed coal and rock may extend along the fault plane and have a width of several metres. Such a crush zone can be seen in the highwall of an opencast working in the UK shown in Figure 2.40.

Large-scale normal faults are produced by tensional forces pulling apart or spreading the crustal layer. Where these faults run parallel with downfaulted areas in between, they are known as graben structures. Many coalfields are preserved in such structures; the brown coalfields of northern Germany and eastern Europe, and the Gondwana coalfields of India and Bangladesh are examples.





**Figure 2.37** Normal fault with downthrow of 2 m to the right. Palaeogene–Neogene coal-bearing sediments, Sumatra, Indonesia. *Source:* Photograph by LPT.



**Figure 2.39** Normal fault downthrowing overburden (light colour) against a coal seam in an opencast mine in Central India. *Source:* Photograph by courtesy of Dargo Associates Ltd.



**Figure 2.38** Highwall termination due to faulting in Palaeogene–Neogene brown coal opencast mine in Bosnia-Herzegovina. *Source:* Photograph by courtesy of Dargo Associates Ltd.

Low-angle faults with normal fault displacements are known as lag faults. They originate from retardation of the hanging wall during regional movement, as shown in Figure 2.41. Lag faults are common in the coalfields of South Wales, UK.

*Reverse faults* are produced by horizontal stress with little vertical compression. This results in the shortening of the rock section in the direction of maximum compression.

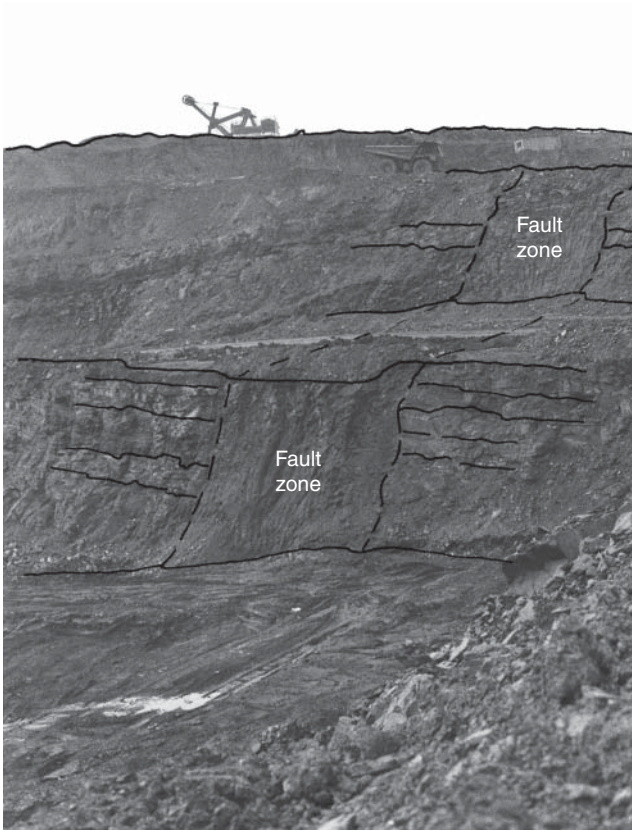
Very high angle reverse faults are usually large structures, associated with regional uplift and accompanying igneous activity. In coal geology, those reverse faults with low angles ( $<45^\circ$ ) are more significant. A typical reverse fault structure is seen in Figure 2.42, where the fault has dislocated a coal seam by several

metres. When the angle is very low and the lateral displacement is very pronounced, such faults are termed thrust faults. The shape of such low-angle reverse faults is controlled by the nature of the faulted rocks, especially when a thrust plane may prefer to follow the bedding plane rather than to cut across it.

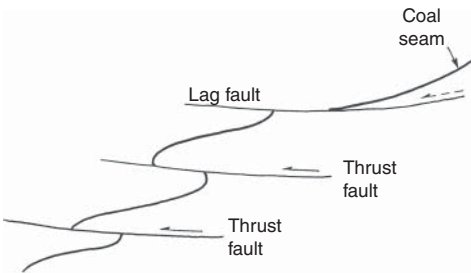
In typical sequences of coal, seatearth, and mudstone with subordinate sandstone, such low-angle faults often follow the roof and/or the floor of coal seams, as these allow ease of movement, with the seatearths often acting as a lubricant. One detrimental effect is the contamination of the coal seam with surrounding country rock, so reducing its quality and, in some cases, its mineability.

In highly tectonised coal deposits, a great number of coal seam contacts have undergone some movement and shearing. In some cases the whole seam will have been compressed and moved. This may be displayed in coals as arcuate shear planes throughout. Figure 2.43 portrays this feature.

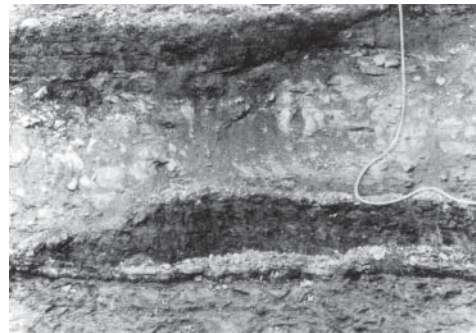
The development of thrust zones in coal sequences is illustrated in Figure 2.44. Here, lateral compression has produced thrusting along preferred lithological horizons, and continued compression has resulted in the upper part of the sequence being more tectonically disturbed than the lower part. This deformation is now termed progressive easy-slip



**Figure 2.40** Large fault zone exposed in highwall in an opencast mine, South Wales, UK. *Source:* Photograph by LPT.



**Figure 2.41** Lag fault produced by retardation of the upper part of the sequence during the forward movement of the lower sequence by thrust faulting. *Source:* From Sherborn Hills (1975) *Elements of Structural Geology*, 2nd ed., with kind permission of Kluwer Academic Publishers.



**Figure 2.42** Coal seam dislocated by reverse fault, with throw of 1.5 m, in UK opencast mine. *Source:* Photograph by M C Coultas.

thrusting. Such events are particularly common in coalfields that have suffered crustal shortening, as is the case in South Wales, UK (Gayer et al. 1991), and in the Appalachians, USA. Thrusting is also accentuated where

coal and mudstone sequences are sandwiched between thick sequences of coarse clastics, the upper and lower portions of the sequence reacting to compressive forces quite differently to the incompetent coals and mudstones.



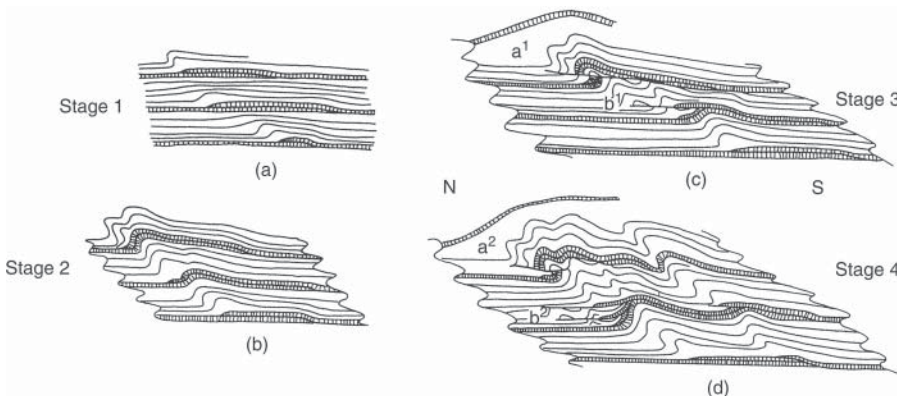
**Figure 2.43** Highly sheared anthracite coal seam (seam thickness 1.2 m) in opencast mine, South Wales, UK. *Source:* Photograph by LPT.

*Strike-slip faults* have maximum and minimum stress in the two horizontal planes normal to one another. This has the effect of producing a horizontal movement either in a clockwise (dextral) or anticlockwise (sinistral) sense. Strike-slip faulting is usually found on a regional scale. Although important, it has a lesser influence on the analysis of small coal deposits and mine lease areas.

Evidence of faulting on the rock surface can be seen in the form of slickensides, which are

striations on the fault plane parallel to the sense of movement. Some fault planes have a polished appearance, particularly where high rank coal has been compressed along the fault plane. Conical shear surfaces are characteristically developed in coal. These are known as cone-in-cone structures and are the result of compression between the top and bottom of the coal.

Coal responds in a highly brittle manner to increasing deformation by undergoing failure and subsequent displacement along ever-increasing numbers of fracture surfaces (Frodsham and Gayer 1999). In tectonically deformed coalfields, and particularly in mine workings, it is important that a rapid assessment of the physical state of the coal can be made. Visual assessment of the appearance of coal can be made in hand-specimen samples, core samples, or by outcrop observations (see Section 6.4.10). The average structural index (ASI) can be used to assess the relative strength of deformed coal samples on the basis of appearance and the frequency of fractures in the specimen. Table 2.3 shows the coal types and the ASI rating for coals of differing levels



**Figure 2.44** Model for four stages in progressive easy-slip thrusting. (a) Thrusts develop simultaneously as flats along the floors of overpressured coal seams, cutting up to the roofs of the seams along short ramps; propagation folds grow at the thrust tips. (b) Thrusts continue to propagate with amplification of the tip folds, until a lower propagation fold locks up a higher thrust, producing downward-facing cut-offs. (c) Continued out-of-sequence movement on higher thrusts results in break-back thrusting in the hanging wall and footwall areas; thrusts locally cut down stratigraphy in the transport direction. (d) Progressive out of sequence hanging wall break-back produces distinctive geometry, with the structure in a lower thrust being apparently unrelated to that in a higher thrust slice. Progressive footwall break-back produces folded thrusts. *Source:* From Gayer et al. (1991).

**Table 2.3** The average structural index (ASI) for coals.

Basic coal type	ASI value	Type and frequency of fracturing	Structural state of the coal
Normal	1	Entirely non-tectonic	Undisturbed bright hard coal
Normal	2	Mainly non-tectonic; 1–2 striated fractures	Strong, bright, and hard but easily split along fractures
Normal	3	Mainly non-tectonic; 1–2 polished fractures	Strong, bright, and hard but easily split along fractures
Normal	4	Mainly non-tectonic, several tectonics of either kind	Bright, but coal becoming noticeably weakened by tectonic fractures
Abnormal	5	Mainly tectonic fractures of either kind but also several non-tectonic	Coal exhibiting a change in overall structure, has largely dull or shiny lustre and lacks strength
Abnormal	6	Striated fracture planes dominant with only a few non-tectonic	Disturbed and very dull
Abnormal	7	Polished fracture planes dominant with only a few non-tectonic	Disturbed and excessively shiny
Abnormal	8	Wholly tectonic fractures, no non-tectonic left	Disturbed, either dull or excessively shiny
Outburst	9	Pervasive microfractures	Highly friable, soft sooty texture, no in-situ strength

Source: From Frodsham and Gayer (1999).

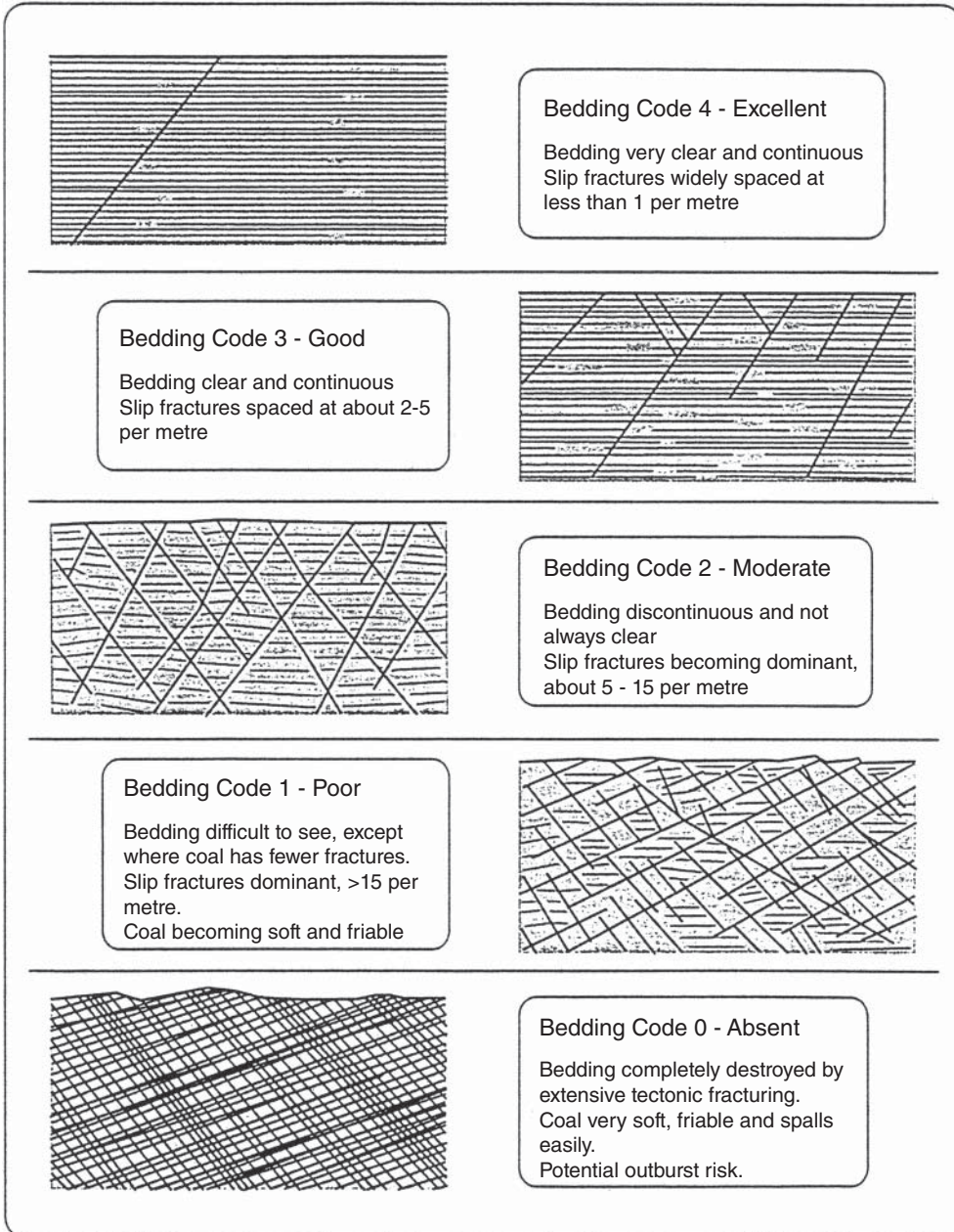
of structural intensity. On a larger scale, the coal Bedding Code is based on the principle that the bedding planes in coal seams become progressively obscured with increasing frequency of tectonic fracturing. Frodsham and Gayer (1999) describe five categories of bedding plane obscurity and large-scale fracture intensity within coal seams from South Wales, UK (Figure 2.45). These categories range from Bedding Code 4 (Excellent, where coal exhibits very clear bedding with fractures moderately spaced at less than one per metre) to Bedding Code 0 (Absent, where the bedding is completely destroyed by tectonic fracturing). The use of coal bedding codes has proved successful in predicting the location of coal outbursts as more deformed parts of the mine are approached (Frodsham and Gayer 1999).

### 2.3.2.3 Folding

Coals in coal-bearing sequences may be folded into any number of fold styles. An example is shown in Figure 2.46. In coalfield evaluation,

the axial planes of the folds need to be located and the dips on the limbs of the folds calculated. In poorly exposed country the problem of both true and apparent dips being seen has to be carefully examined. Also, in dissected terrain, dips taken at exposures on valley sides may not give a true reflection of the structural attitude of the beds at this locality; many valley sides are unstable areas and mass movement of strata is common, resulting in the recording of oversteepened dips. This is characteristic of areas of thick vegetation cover where a view of the valley side is obscured and any evidences of movement may be concealed. If the field data suggest steeper dipping strata this will give less favourable stripping ratio calculations for an opencast prospect and may contribute to the cancellation of further investigations. Similarly, in underground operations, if the dip of the coal seams steepens, it can make the working of the coal difficult, and in the case of longwall mining can prevent further extraction. Therefore, it is important to be sure





**Figure 2.45** The coal Bedding Code, showing five categories of bedding plane obscurity and large-scale fracture intensity within coal seams in South Wales, UK. *Source:* From Frodsham and Gayer (1999).

that all readings taken reflect the true nature of the structure in the area of investigation.

Compression of coal seams during folding can produce tight anticlinal folds with thrusting along the nose of the fold; these have been

termed queue anticlines. Coal seams can be pinched out along the fold limbs and appear to have flowed into the axial areas of the anticlines. Where this occurs from two directions approximately normal to one another,



**Figure 2.46** Intensely folded Carboniferous coal-bearing sediments, Little Haven, Pembrokehire, UK. *Source:* Photograph by courtesy of Dargo Associates Ltd.

coals can be concentrated in ‘pepperpot’-type structures. Such features are usually found only in highly tectonised coalfields. Examples of such intense deformation are illustrated in Figure 2.47a, and a coal seam squeezed in this way is shown in Figure 2.47b, where the coal has been compressed in and around the overlying sandstone. In many instances, such structural complications will render a coal seam unmineable except by the most primitive

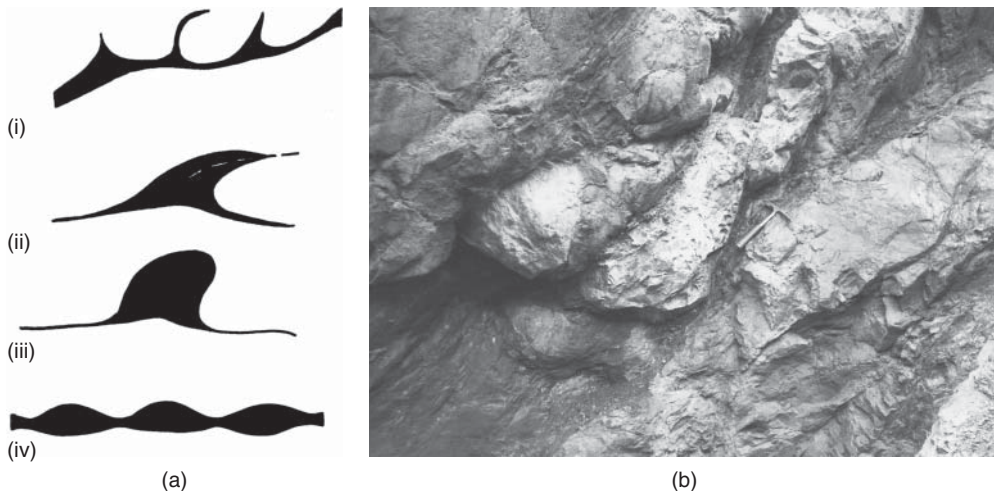
of methods, but coal concentrations in the axial regions of folds have been mined in the same fashion as mineral ‘saddle reefs’.

Detailed mapping of folded coal deposits is an essential part of the exploration process. Examples of folded coal deposits are illustrated in Figure 2.48a, where a pattern of zigzag folds characterises the Wurm Coal Basin, Germany (Stutzer and Noe 1940), and Figure 2.48b shows a series of asymmetrical folds with associated thrusting from the Kutei Basin, Indonesia (Land and Jones 1987). Such examples serve to show the necessity of acquiring a good understanding of the structural elements and style of the coal deposit in order to identify those areas where coal is preserved in such quantities, attitude, and depth as to allow mining to develop.

#### 2.3.2.4 Igneous Associations

In many coalfields, associated igneous activity has resulted in dykes and sills being intruded into the coal-bearing sequence.

The intrusion of hot molten rock into the coals produces a cindering of the coal and



**Figure 2.47** (a) Tectonic deformation of coal seams due to compression: (i) squeezing into the overlying formation; (ii) queue anticline; (iii) ‘pepper-pot’ structure; (iv) ‘rosary’ structure. (b) Carboniferous anthracite squeezed in and around overlying sandstone, Samcheog Coalfield, Republic of Korea. *Source:* Photograph by LPT.

a marked loss in volatile matter content, which has been driven off by heat. This can have the effect of locally raising the rank of lower rank coals and can, therefore, in certain circumstances make the coal attractive for exploitation. Such 'amelioration' of coal seams is a common feature in areas of igneous activity. Good examples are found in Indonesia and the Philippines, where Palaeogene–Neogene sub-bituminous coals have been ameliorated up to low-volatile bituminous and some even to anthracite rank.

The majority of dykes and sills are doleritic in composition, as in the case of South African and Indian coalfields, but occasionally other types are found: in the Republic of Korea, acidic dykes and sills are intruded into the coals. Figure 2.49 shows acidic igneous material intruding a coal seam in an underground working.

In areas where igneous intrusions are prevalent in mine workings, plans showing the distribution and size of igneous bodies are required in order to determine areas of volatile loss where the coal has been baked; and because of the hardness of the igneous material, tunnelling has to be planned with

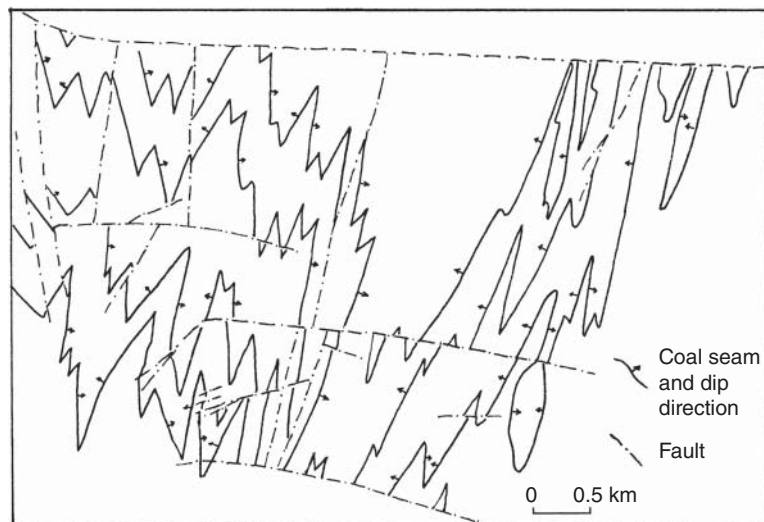
the position of intrusions in mind. Igneous sills have a tendency to jump from one coal seam to another, so that close-spaced drilling is often required to identify precisely the nature and position of such intrusions. Igneous intrusions are found in coal sequences worldwide, but they are a particularly common feature of South African coal workings. Where such igneous bodies exist, the coal geologist must identify the areas occupied by igneous material within the mine area, and also those seams affected by igneous activity.

In addition, the possibility of methane gas driven off during intrusion may have collected in intervening or overlying porous sandstones. Mine operatives need to investigate this possibility when entering an intruded area of coal.

### 2.3.2.5 Mineral Precipitates

A common feature of coal-bearing sequences is the formation of ironstone, either as bands or as nodules. They usually consist of siderite ( $\text{FeCO}_3$ ) and can be extremely hard. Where ironstone nucleation and development takes place either in, or in close proximity to, a coal seam, this can deform the coal, cause mining

**Figure 2.48** Outcrop patterns in folded coalfields. (a) Zigzag folding of coal seams and associated faulting, Wurm Coal Basin, Germany. *Source:* From Stutzer and Noe (1940) By permission of the University of Chicago Press. (b) Asymmetrical folding, broad synclines, and sharp anticlines associated with thrusting, East Kalimantan, Indonesia. *Source:* From Land and Jones (1987). Reproduced with permission of the author.



(a)



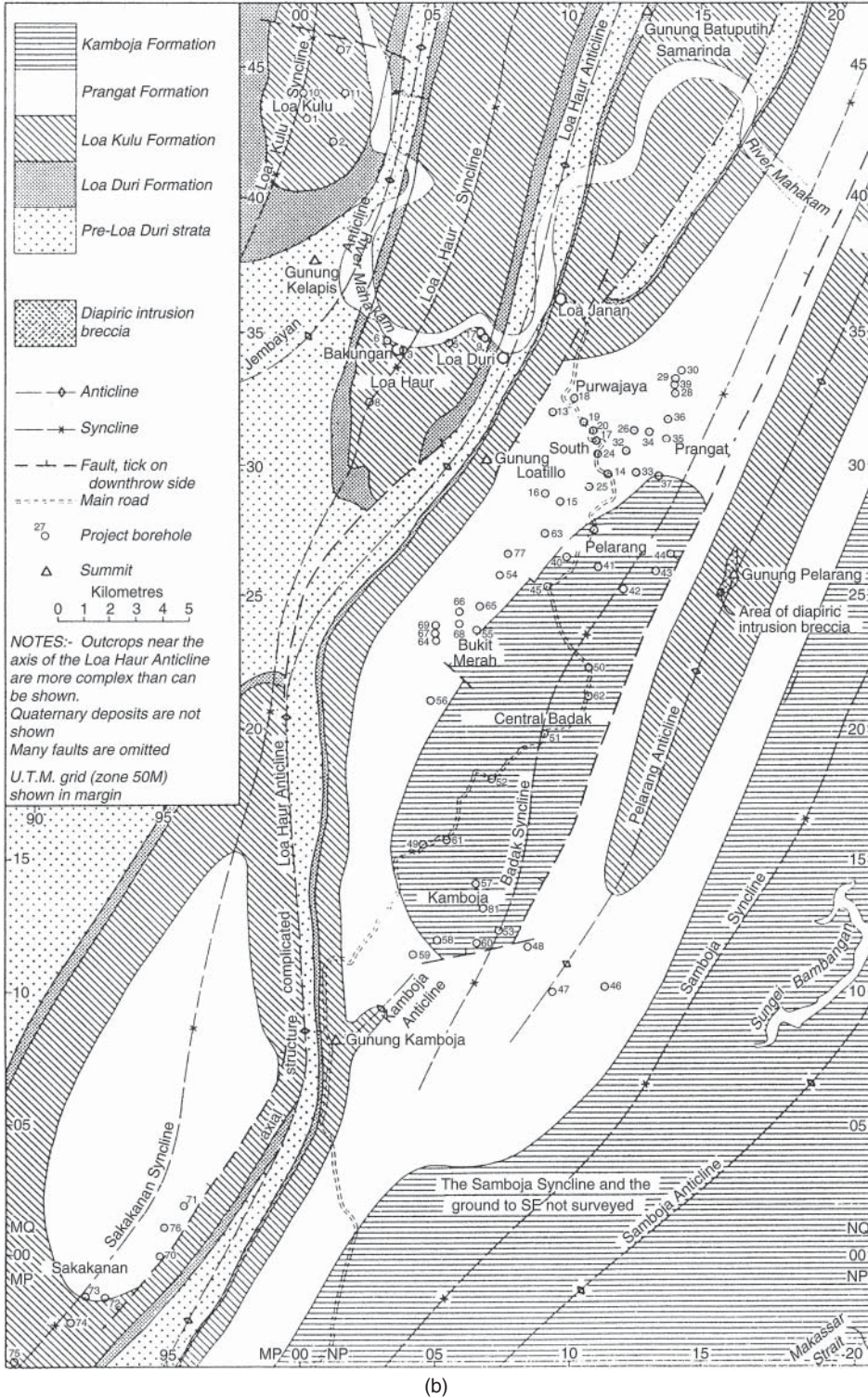


Figure 2.48 (Continued)



**Figure 2.49** Jurassic anthracite (dark colour) intruded by granitic dykes and sills (light colour), Chungnam Coalfield, Republic of Korea. *Source:* Photograph by LPT.

difficulties, and, because of the difficulty in separating coal and ironstone whilst mining, will have an effect on the quality of the run-of-mine product.

Iron sulfide in the form of iron pyrite may be precipitated as disseminated particles, as thin bands, or, as is more common, as coatings on cleat and bedding surfaces (the coal specimen in Figure 6.7a displays pyrite in this form). Inorganic sulfur held in this form in coal can be removed by crushing and passing the coal through a heavy liquid medium. Organic sulfur held elsewhere in the coal cannot be readily removed and remains an inherent constituent of the coal.

Other mineral precipitates usually are in the form of carbonates, coating cleat surfaces or occasionally as mineral veins. Where quartz veining occurs, this has the detrimental effects of being hard, liable to produce sparks in an underground environment where gas is a hazard, and, when crushed, is an industrial respiratory health hazard.

## 3

# Age and Occurrence of Coal

## 3.1 Introduction

Although land plants first developed in the Palaeozoic Era, and coal deposits of Late Silurian and Early Devonian age are the earliest known, it was not until the Upper Palaeozoic, particularly the Carboniferous and Permian Periods, that sufficient plant cover was established and preserved to produce significant coal accumulations.

Coal deposits have been formed throughout the geological column, i.e. from the Carboniferous Period to the Quaternary Period. Within this time range, there were three major episodes of coal accumulation.

The first took place during the late Carboniferous–Early Permian Periods. Coals formed at this time now form the bulk of the black coal reserves of the world, and they are represented on all of the continents. The coals are usually of high rank and may have undergone significant structural change. Carboniferous–Permian coal deposits stretch across the Northern Hemisphere from Canada and the USA, through Europe and the Commonwealth of Independent States (CIS) to the Far East. In the Southern Hemisphere, the Carboniferous–Permian coals of Gondwanaland are preserved in South America, Africa, the Indian subcontinent, South East Asia, Australasia, and Antarctica.

The second episode occurred during the Jurassic–Cretaceous Period, and coals of this age are present in Canada, the USA, the People’s Republic of China (PRC), and the CIS.

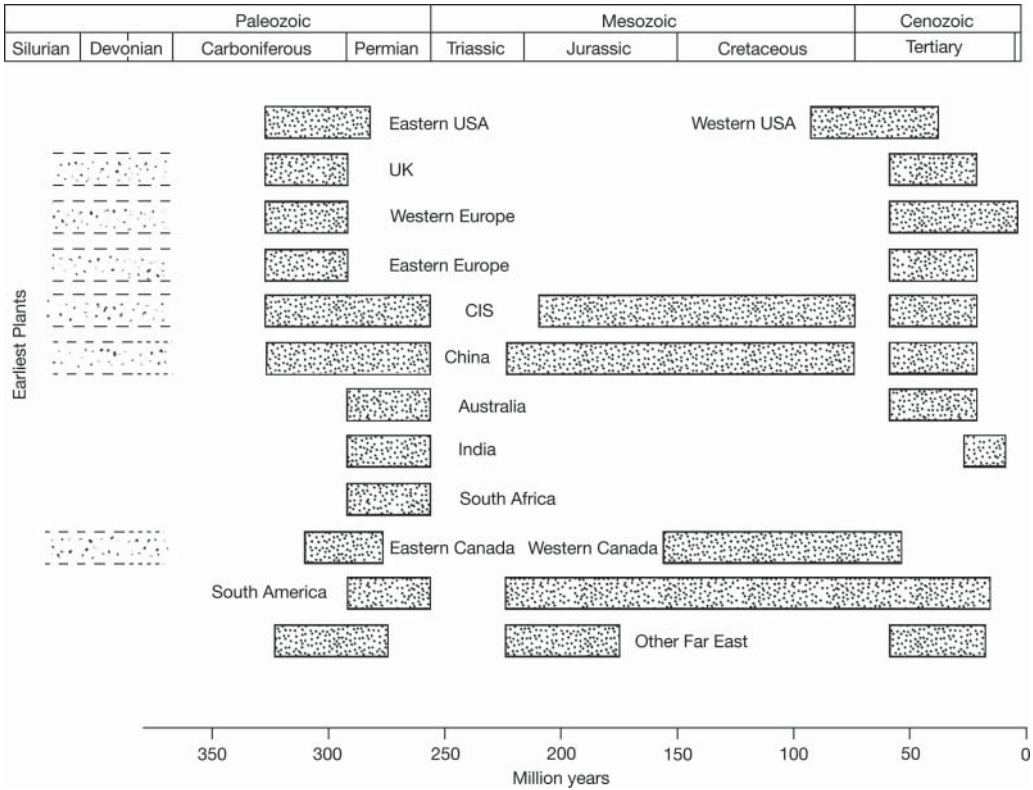
The third major episode occurred during the Palaeogene–Neogene Period. Coals formed during this period range from lignite to anthracite. Palaeogene–Neogene coals form the bulk of the world’s brown coal reserves, but also make up a significant percentage of black coals currently mined. They are characterised by thick seams and have often undergone minimal structural change. Palaeogene–Neogene coals are also found worldwide, and they are the focus of current exploration and production as the traditional Carboniferous coalfields become depleted or geologically too difficult to mine.

Figure 3.1 shows the generalised distribution of world coal deposits in terms of geological age and area (modified from Walker 2000).

## 3.2 Plate Tectonics

Evidence of ocean floor spreading and the identification of modern plate margins has enabled the mechanism of plate tectonics to be understood, i.e. the Earth’s crust and upper part of the mantle consist of a number of mobile plates that have responded to



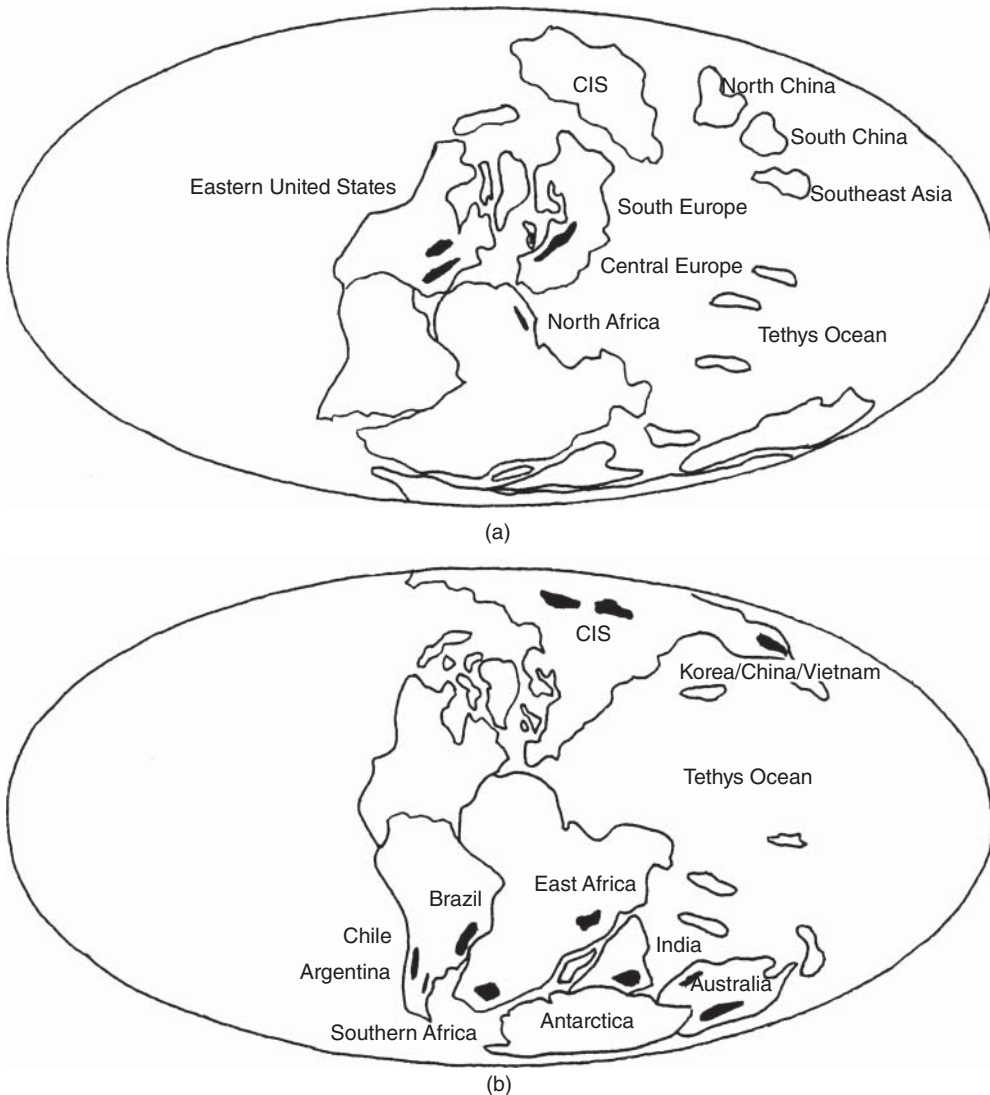


**Figure 3.1** Geological age distribution of the world's black coal and lignite deposits. *Source:* Modified from Walker (2000).

convection currents in the mantle. This has resulted in the amalgamation and fragmentation of plates throughout geological time. du Toit (1937) proposed that the supercontinent *Pangaea* consisted of *Laurasia* in the Northern Hemisphere and *Gondwanaland* in the Southern Hemisphere. These two land areas split apart in the Early Triassic, followed by further rifting that has produced the various smaller continents that exist today.

During the Carboniferous Period, in the northern part of *Pangaea* (*Laurasia*), the coal basins of western and central Europe, eastern USA, and the CIS were equatorial in nature, and tropical peat mires containing a flora of *Lepidodendron*, *Sigillaria*, and *Chordaites* were characteristic of coal deposition (Figure 3.2a).

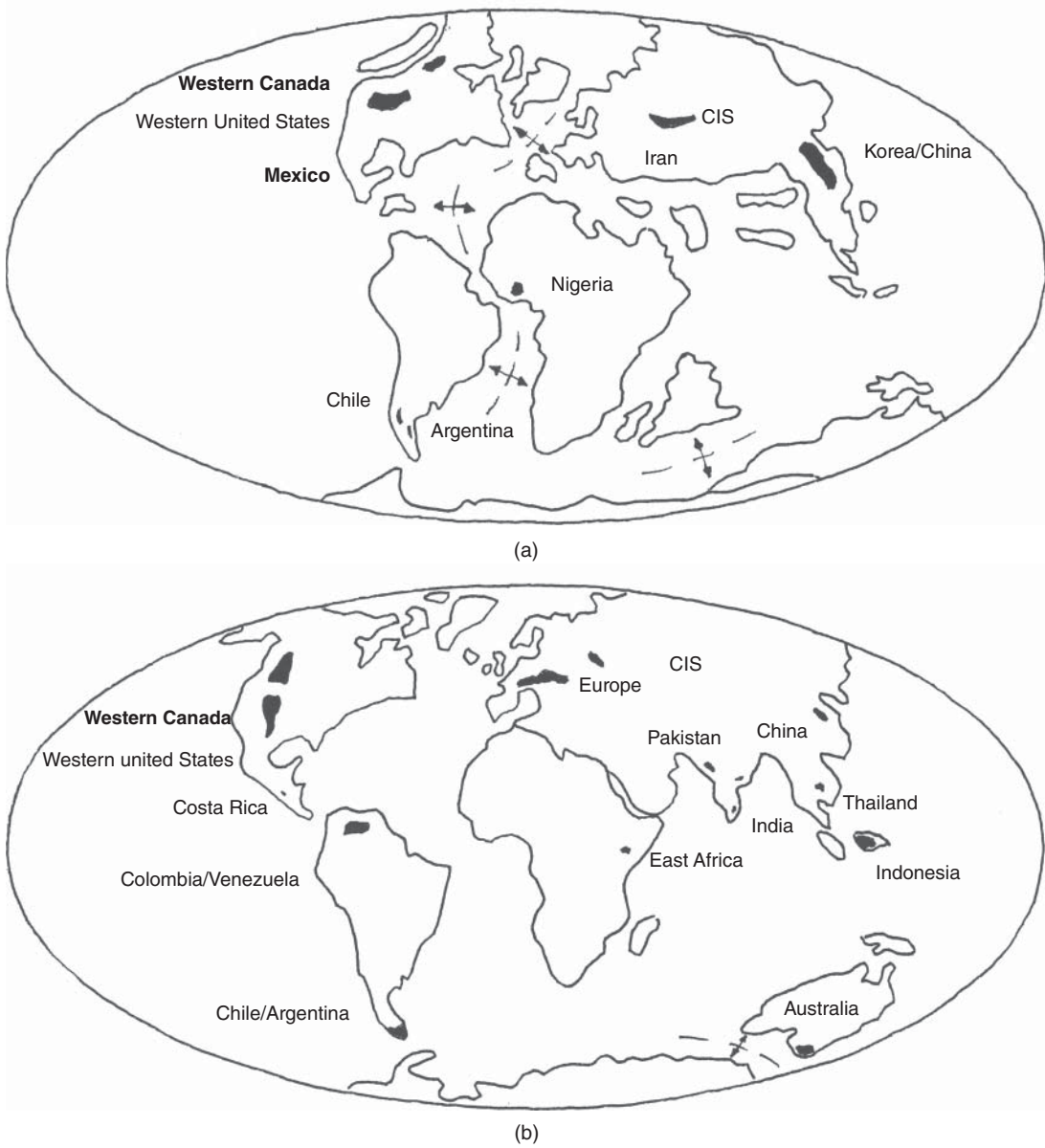
The climate changed during the Permian Period and coal deposition ceased in the northern area. In the southern part of *Pangaea* (*Gondwanaland*), covering what is now South America, southern Africa, India, Australia, and Antarctica, peat mires formed under cooler, more temperate conditions, characterised by the *Glossopteris* flora (Figure 3.2b). After the break-up of *Pangaea*, coal deposition continued through the Triassic, Jurassic, and Cretaceous Periods (Figure 3.3a) and the Palaeogene–Neogene Period (Figure 3.3b), where another change in the floral types took place, heralding the onset of angiosperm floras. These changes in vegetation type are reflected in the type and proportion of maceral types present in the coals (see Section 2.2.3.1).



**Figure 3.2** Palaeogeographical reconstructions of (a) late Carboniferous and (b) Permian–Triassic showing principal areas of coal deposition.

The Laurasian coals are rich in the vitrinite group of macerals, whereas the Gondwana coals have a much higher percentage of the inertinite group of macerals with varying amounts of vitrinite. Gondwana coals have a higher content of mineral matter but lower sulfur contents than the Laurasian coals (see Section 4.1.2).

The Palaeogene–Neogene coals are, for the most part, lignites and are found worldwide, although in some areas they have undergone severe temperature and pressure changes. This has produced higher rank coal, ranging from subbituminous to high-volatile bituminous in areas such as Indonesia, Colombia, and Venezuela.



**Figure 3.3** Palaeogeographic reconstruction of (a) Jurassic–Cretaceous and (b) Palaeogene–Neogene showing principal areas of coal deposition.

### 3.3 Stratigraphy

The age of all the major coal deposits is well documented, and the stratigraphy of each deposit has been studied in detail. This is particularly true for those deposits that have an economic potential.

The origin of coal is characterised by deposition in foredeep and cratonic basins. The essentially non-marine nature of these coal-bearing sequences has meant that detailed chronostratigraphy has often been difficult to apply due to the lack of biostratigraphic evidence, notwithstanding studies of floras and non-marine faunas.

In the Carboniferous of western Europe, a number of marine transgressions have enabled the coal-bearing sequences to be divided into a number of stratigraphic sections, and where individual coal seams have either an overlying marine mudstone or non-marine bivalve band then correlations are possible over large distances. In the Carboniferous of the USA, and also of the PRC, discrete limestone beds within coal-bearing sequences have enabled good broad stratigraphic control over large areas. The later Permian, Mesozoic, and Cenozoic coal deposits all have similar constraints on detailed correlation. In the UK, the long history in studying the Carboniferous (Westphalian) has enabled the chronostratigraphy and biostratigraphy to be classified as shown in Table 3.1 (Lake 1999).

The establishment of a stratigraphic framework for a coal-bearing sequence can be approached in two ways: an examination of the sedimentary sequence in which the coals occur, and a detailed study of the coals themselves. It is usual to apply a combination of chronostratigraphy (where possible) and lithostratigraphy for individual coal deposits. This may be supported by geophysics and detailed sedimentological studies. In addition,

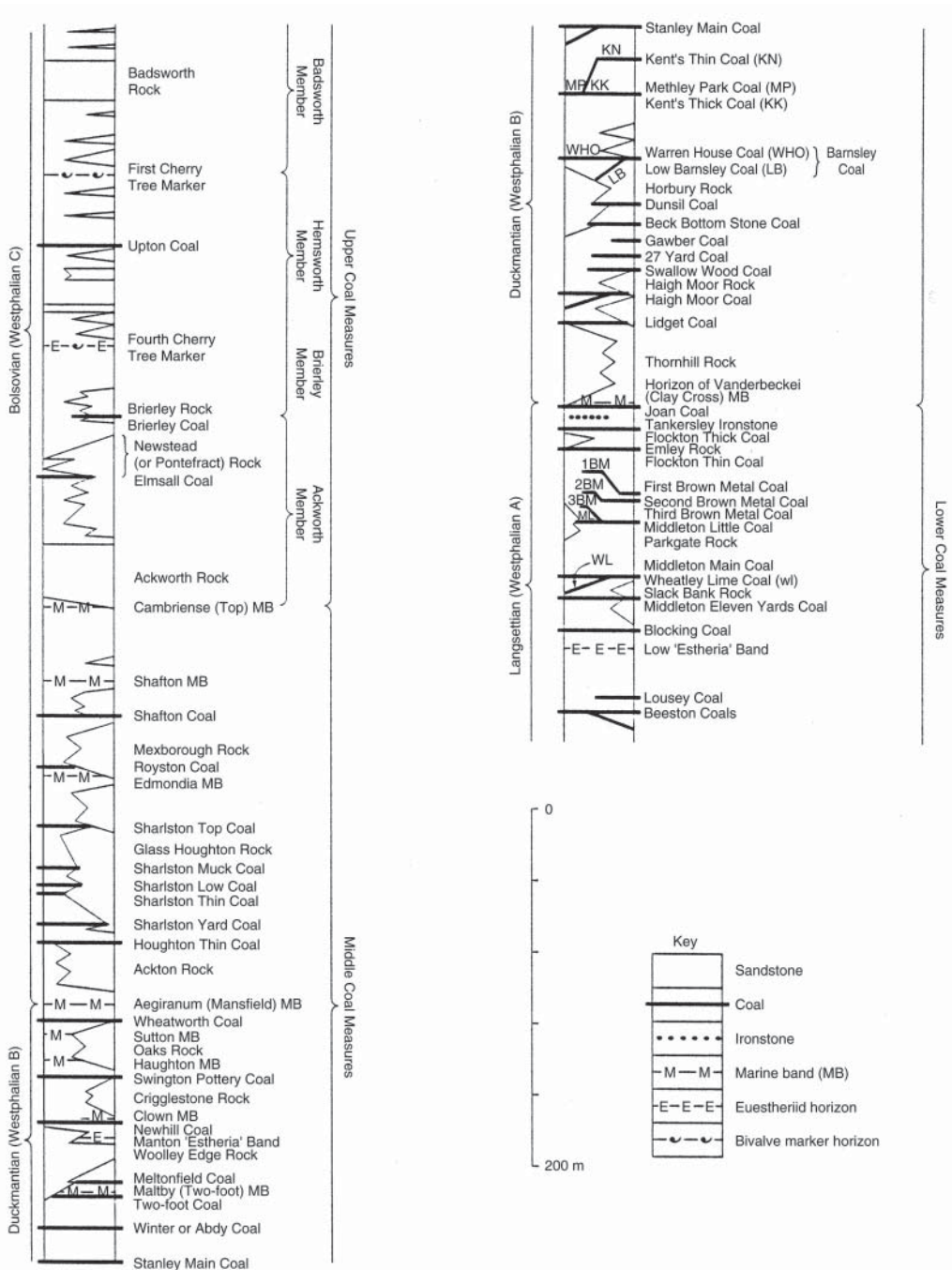
this can be augmented by petrographic analysis of the coals and palynological studies, which will aid the identification of individual coals or series of coals. The combination of all these studies is the basis on which to build the geological model and to develop a 3D picture of the coal deposit.

In the UK, detailed studies over the last 150 years of the various coalfield areas has resulted in a wealth of geological data being collected and interpreted. For example, in the Yorkshire coalfield, the identification of coals and marine and non-marine fossiliferous horizons has enabled a detailed stratigraphy to be built up. Figure 3.4 shows the stratigraphic column of 1120 m of the Westphalian succession for the Wakefield district of Yorkshire (Lake 1999). The compilation of detailed stratigraphic logs from sections, boreholes, and mines then allows correlations to be made. Figure 3.5 depicts a series of logs within the Leicestershire coalfield and shows correlations based on faunal horizons and coals (Worssam and Old 1988).

In the USA, a large amount of geological investigation took place in the Pennsylvanian coal-bearing sequence in the Illinois Basin. Correlation studies have been based on spores (Kosanke 1950) and on sedimentological analysis of the shape and distribution patterns of sand bodies and their effect on the principal economic coal seams in the basin (Potter 1962, 1963; Potter and Glass 1958; Potter and Simon 1961). Figure 3.6 illustrates the use of lithological and electric logs to show the stratigraphic relationships due to the development of the Anvil Rock Sandstone and its effect upon the Herrin No.6 seam (Potter and Simon 1961). Similar studies have been carried out in all the major coal deposits worldwide with a view to identifying the effects on coalmining economics by geological processes.

**Table 3.1** Detailed chronostratigraphy and biostratigraphy of the Carboniferous (Westphalian A–C) coal-bearing sequence of the UK (Lake 1999). IPR/23-10C British Geological Survey. © NERC. All rights reserved.

CHRONOSTRATIGRAPHY				BIOSTRATIGRAPHY			
SERIES	STAGE	CHRONOZONE	MARINE BAND	NONMARINE BIVALVES	MIOSPORES	MEGAFLORA	
WESTPHALIAN	BOLSOVIAN	Phillipsi		<i>Anthraconauta phillipsi</i> Zone	<i>Torispora securis</i> (X)	<i>Paripteris linguaeifolia</i>	
		Upper Similis-Pulchra	Cambriense				<i>Vestispora magna</i> (IX)
			Shafton				
			Edmondia		<i>Anthraconaia adamsi-hindi</i>		
			Carway Fawr				
			Aegiranum				
	DUCKMANTIAN	Lower Similis-Pulchra	Sutton		<i>Anthracosia atra</i>	<i>Dictyotriletes bireticulatus</i> (VIII)	
			Haughton				
			Clown				
			Maltby				
		Modiolaris		Lowton Estheria		<i>Anthracosia caledonica</i>	<i>Schulzospora rara</i> (VII)
						<i>Anthracosia phrygiana</i>	
						<i>Anthracosia ovum</i>	
						<i>Anthracosia regularis</i>	
						<i>Carbonicola crista-galli</i>	
	LANGSETTIAN	Communis		Vanderbeckei	<i>Carbonicola pseudorobusta</i>	<i>Radiizonates aligerens</i> (VI)	
					<i>Carbonicola bipennis</i>		
				Low Estheria	<i>Carbonicola torus</i>		
				Kilburn			
		Lenisulcata			Burton Joyce	<i>Carbonicola Proxima</i>	<i>Triquitrites sinani</i> <i>Cirratiradites saturni</i> (SS)
						<i>Carbonicola extenuata</i>	
					Langley		
					Amaliae		
				Meadow Farm			
				Parkhouse			
				Listeri			
				Honley			
			Springwood				
			<i>Carbonicola fallax-protea</i>				
					<i>Lyginopteris hoeninghausii</i>		
			Holbrook				



**Figure 3.4** Detailed stratigraphy of the Westphalian succession for the Wakefield area of Yorkshire, UK (Lake 1999). IPR/23-10C. British Geological Survey, © NERC. All rights reserved.



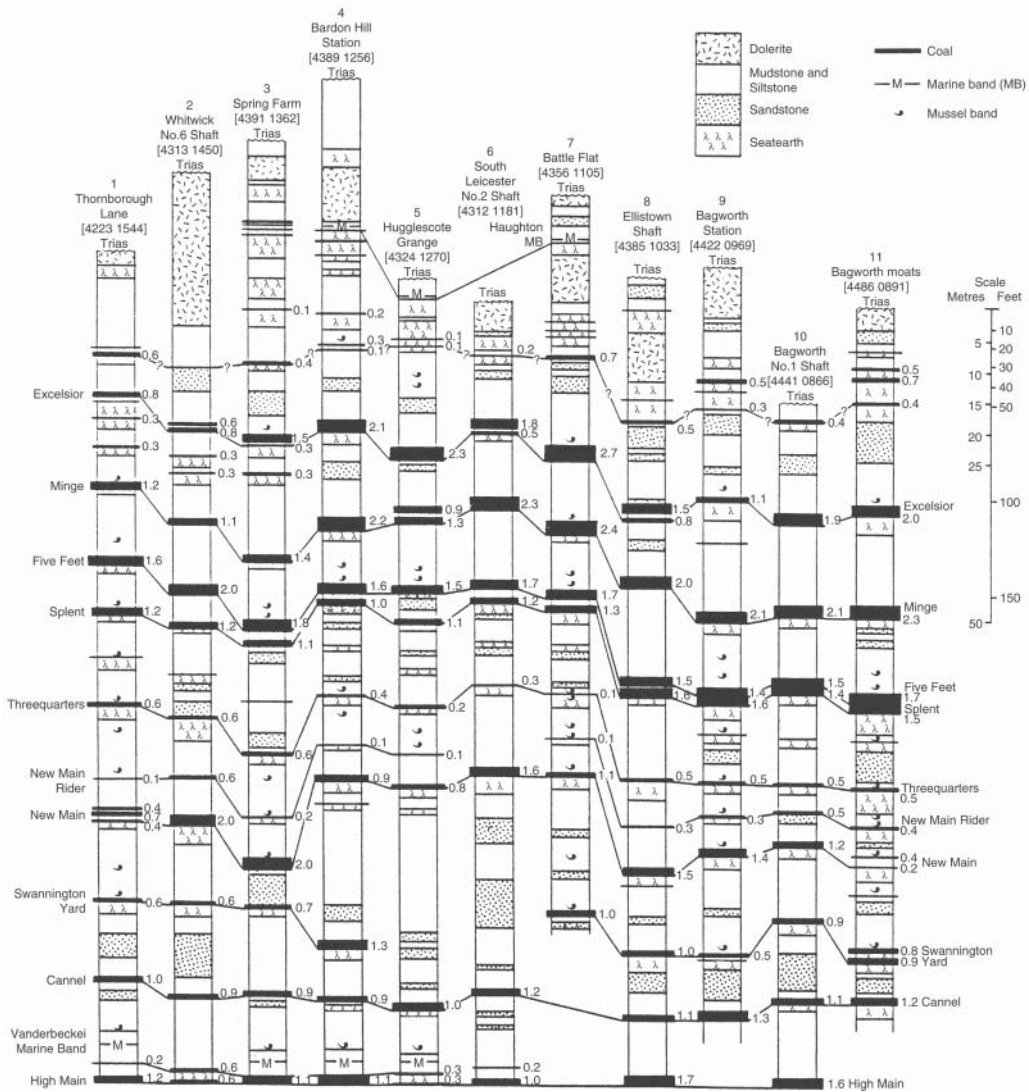


Figure 3.5 Stratigraphic correlation based on faunal horizons and coals (Worsam and Old 1988). IPR/23-10C. British Geological Survey. © NERC. All rights reserved.

### 3.4 Age and Geographical Distribution of Coal

A brief summary is given of the geographical distribution of the known coal deposits of the world. This is designed as a guide to the location of the principal coalfields throughout the world. The detailed stratigraphic ages of the deposits are not given; usually, only the

period in which they were formed is given. The distribution of coal deposits throughout the world is shown in Figures 3.7–3.15 and is dealt with based on nine geographical regions. Details are principally based on Walker’s (2000) review of the major coal-producing countries of the world and Saus and Schiffer’s (1999) review of lignite in Europe.

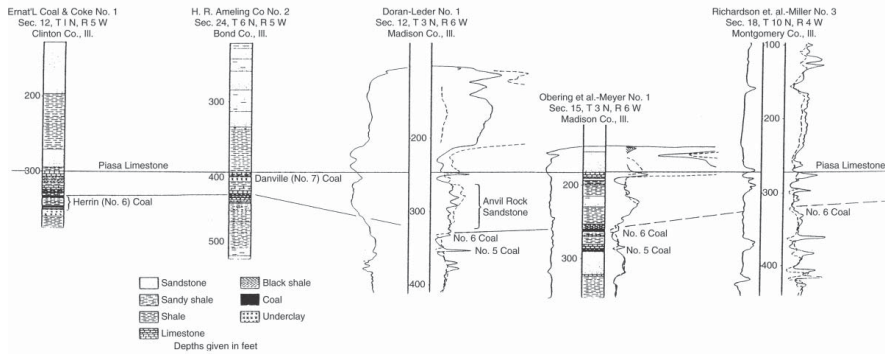
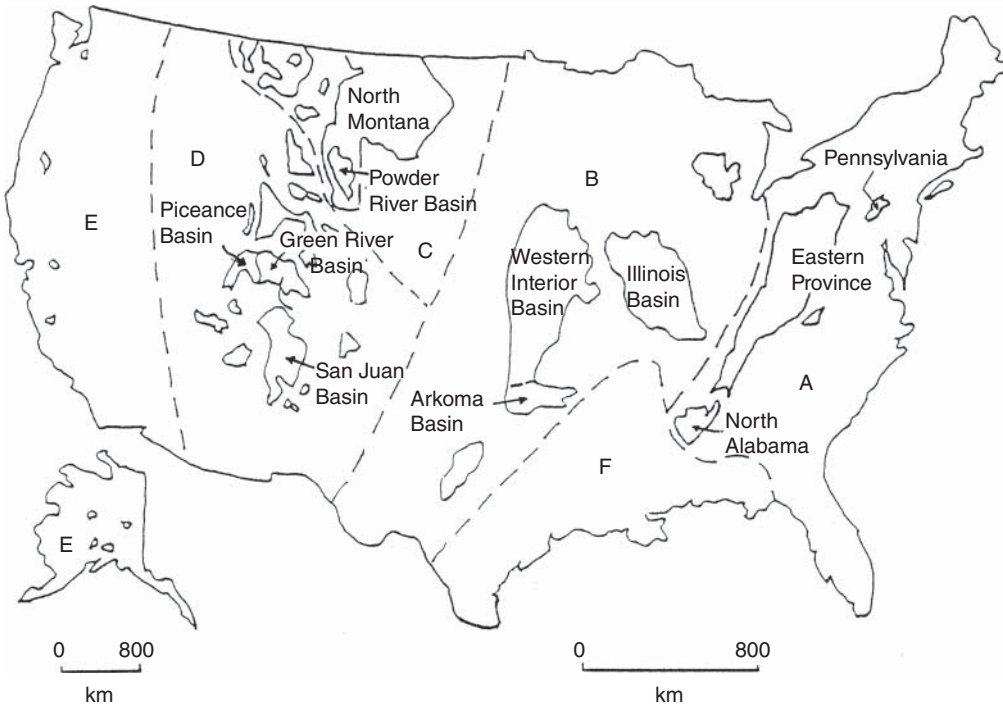
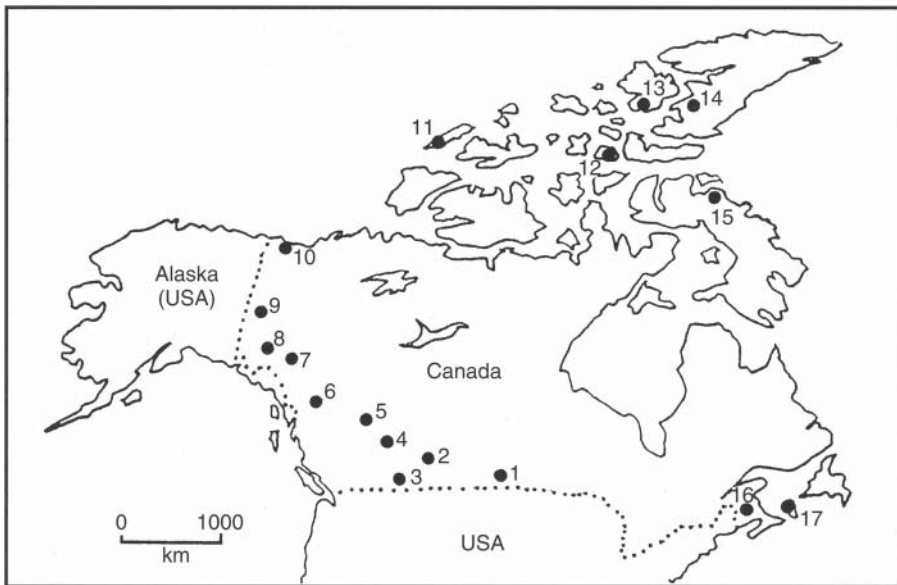


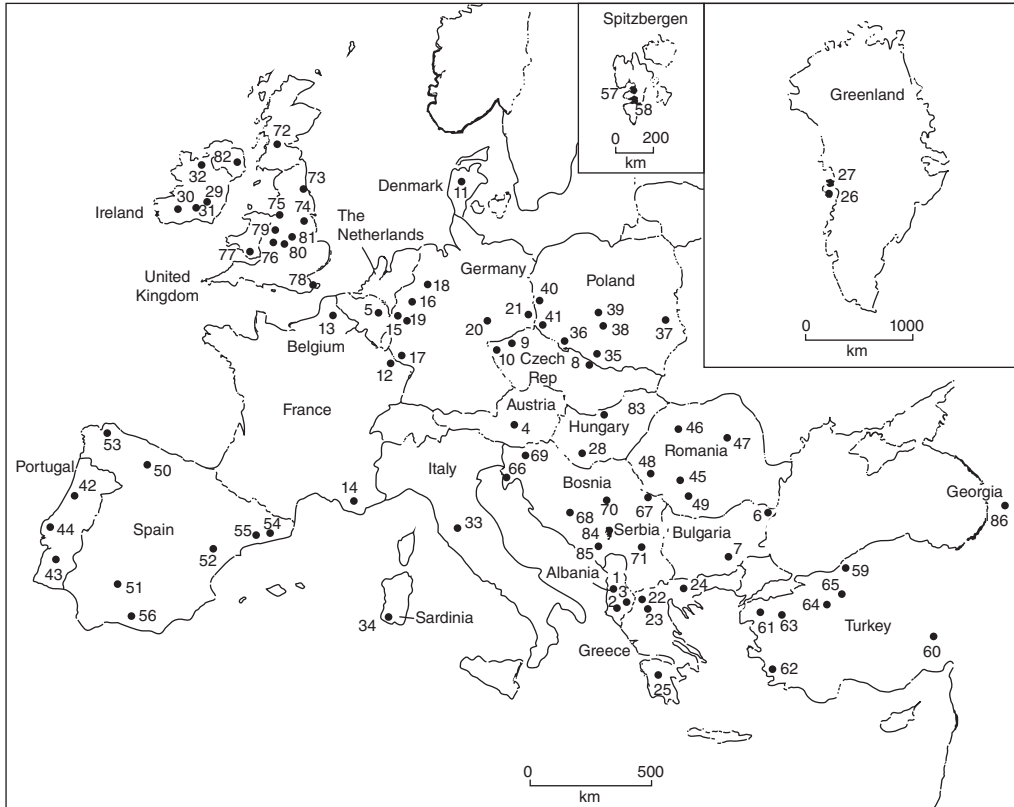
Figure 3.6 Use of lithological and electric logs to show the stratigraphic relationships between the development of the Anvil Rock sandstone, and its effect on the Herrin No. 6 coal (Potter and Simon 1961).



**Figure 3.7** Coal deposits of the USA. A, Eastern Province; B, Interior Province; C, Northern Great Plains Province; D, Rocky Mountains Province; E, Pacific Coast Province; F, Gulf Coastal Plain Province.



**Figure 3.8** Coal deposits of Canada: (1) south Saskatchewan; (2) central and eastern Alberta; (3) south-east British Columbia; (4) west central Alberta; (5) north-east Coal Block; (6) north-west British Columbia; (7) Watson Lake; (8) Whitehorse; (9) Dawson; (10) Mackenzie Bay; (11) Prince Patrick Island; (12) Cornwallis Island; (13) Axel Heiberg Island; (14) Ellesmere Island; (15) Baffin Island; (16) Minto Coalfield; (17) Sydney Coalfield.



**Figure 3.9** Coal deposits of Europe: (1) Tirane; (2) Tepelene; (3) Korce; (4) West Styria; (5) Kempen; (6) Dobrudza; (7) Maritsa; (8) Ostrava-Karvina; (9) North Bohemia; (10) Sokolov; (11) Herring; (12) Lorraine; (13) Nord et Pas de Calais; (14) Provence; (15) Aachen; (16) Ruhr; (17) Saar; (18) Lower Saxony; (19) Rhenish; (20) Halle Leipzig Borna; (21) Lower Lausitz; (22) Florina Amyndaeon; (23) Ptolemais; (24) Serrae; (25) Megalopolis; (26) Disko; (27) Nugssuaq; (28) Mecsek; (29) Leinster; (30) Kanturk; (31) Slieveardagh; (32) Connaught; (33) Valdarno; (34) Sulsis; (35) Upper Silesia; (36) Lower Silesia; (37) Lublin; (38) Belchatow; (39) Konin; (40) Lausitz; (41) Turoscow; (42) Sao Pedro da Cova; (43) Cabo Mondego; (44) Rio Maior; (45) Jiu; (46) Almas; (47) Comanesti; (48) Banat; (49) Oltenia; (50) Leon; (51) Puertollano; (52) Teruel; (53) Garcia Rodriguez; (54) Calaf; (55) Mequinenza; (56) Arenas del Rey; (57) Longyearbyen; (58) Svea; (59) Zonguldak; (60) Elbistan; (61) Canakkale; (62) Mugla; (63) Bursa; (64) Ankara; (65) Cankiri; (66) Istra; (67) Dobra; (68) Sarajevo-Zenica; (69) Trans-Sava; (70) Stanari; (71) Kosovo; (72) Scotland; (73) North-east England; (74) Yorkshire–Nottinghamshire; (75) Lancashire–North Wales; (76) East and West Midlands; (77) South Wales; (78) Kent; (79) North and South Staffordshire; (80) Warwickshire; (81) Leicestershire; (82) Northern Ireland; (83) Visonta-Bukkabrany; (84) Kolubara; (85) Gacko; (86) Tkibuli.

### 3.4.1 United States of America

The US coal resource base is the largest in the world; they are widely distributed, being found in 38 states. The coal deposits of the USA have been divided into six separate areas or provinces, based on the findings of the US Geological Survey (Figure 3.7) and Nelson (1987).

The *Eastern Province* is the oldest and most extensively developed coal province in the USA. The coal is of Carboniferous age (Pennsylvanian), and the province contains two-fifths of the nation's bituminous coal plus almost all the anthracite. Coal rank increases from west to east, so that high-volatile bituminous coal gives way to low-volatile bituminous

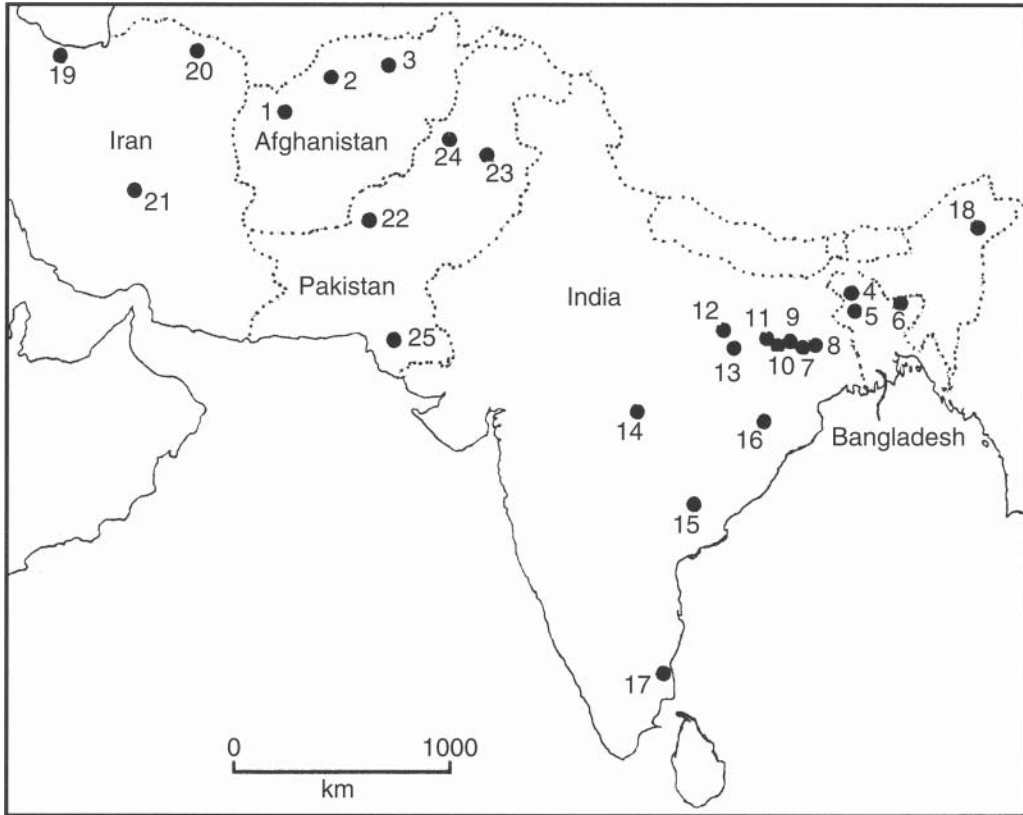


**Figure 3.10** Coal deposits of Africa. (1) Lungue-Bungo; (2) Luanda; (3) Moropule; (4) Mmamabula; (5) Bamenda; (6) Al Maghara; (7) Imaloto–Vohibory; (8) Sakoa; (9) Sakamena; (10) Antanifotsy; (11) Livingstonia; (12) North Rukuru; (13) Ngana; (14) Lengwe; (15) Mwabvi; (16) Jerada; (17) Ezzhiliga; (18) Tindouf-Draa; (19) Meknes-Fez; (20) Moatize; (21) Mmambansavu; (22) Chiomo; (23) Itule; (24) Enugu-Ezimo; (25) Orukpa–Okaba–Ogboyoga; (26) Asaba; (27) Karoo Basin; (28) Waterberg; (29) Springbok Flats; (30) Limpopo; (31) Lebombo; (32) Mhlume; (33) Mpaka; (34) Maloma; (35) Ketewaka–Mchuchma; (36) Ngaka; (37) Songwe–Kiwira; (38) Galula; (39) Njuga; (40) Ufipa; (41) Luene; (42) Lukuga; (43) Luangwa; (44) Luano; (45) Maamba; (46) Kahare; (47) Wankie; (48) Lubimbi; (49) Sessami–Kaonga; (50) Tuli; (51) Bubyee; (52) Chelga; (53) Wuchalle; (54) Dobre-Brehan; (55) Bourem.

and anthracite. The sulfur content of the coals is higher in the west and decreases to the east, with the older coals containing the highest sulfur levels.

The Appalachian Basin extends for over 1500 km from Pennsylvania in the north to Alabama in the south-west, with a width of 400 km in the north tapering to 25 km





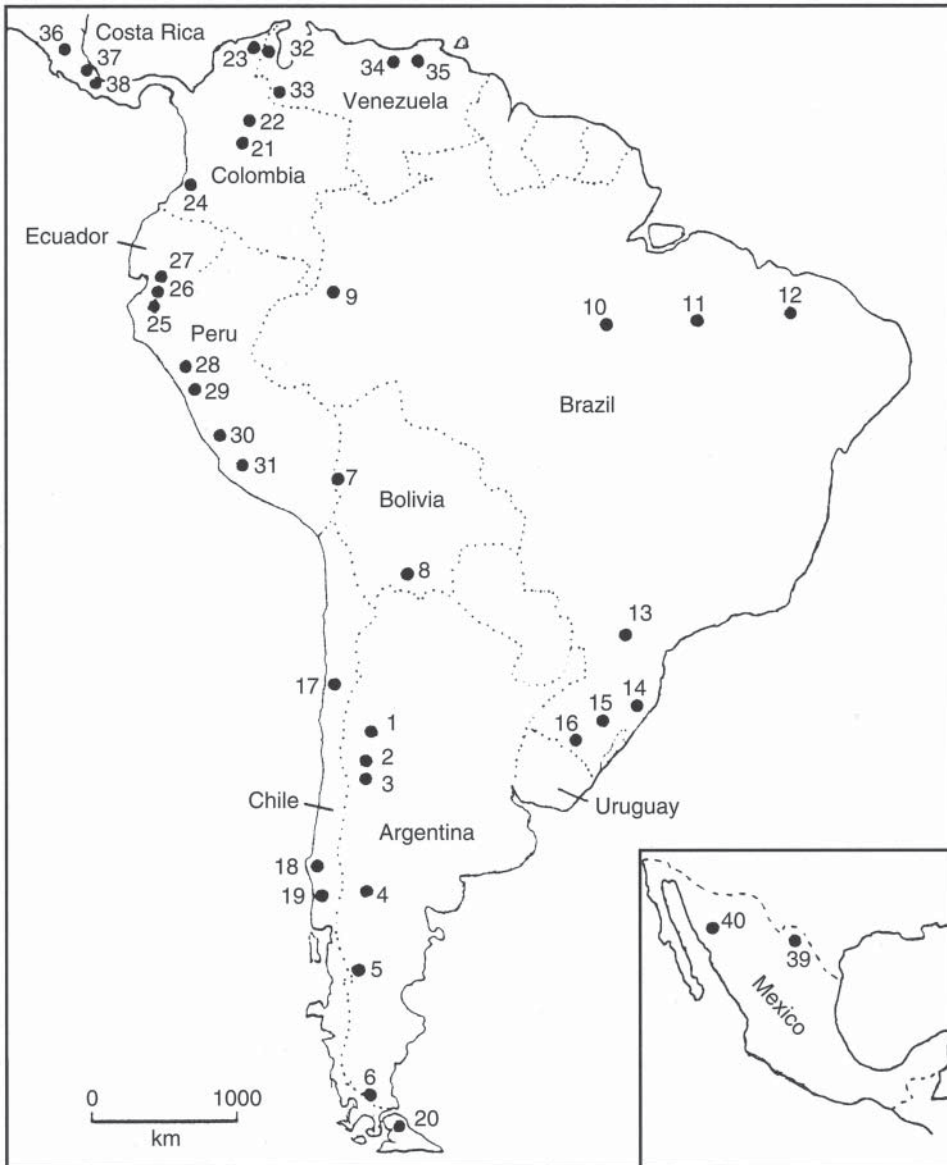
**Figure 3.11** Coal deposits of the Indian subcontinent. (1) Heart; (2) Sari-i-Pul; (3) Dara-i-Suf; (4) Barapukuria–Khalaspir; (5) Jamalganj; (6) Sylhet; (7) Raniganj; (8) Jharia; (9) Bokaro; (10) Ramgarh; (11) Karanpura; (12) Singrauli; (13) Bisrampur; (14) Pench–Kanhan–Tawa; (15) Godavari; (16) Talchir; (17) Neyveli; (18) Makum; (19) Elburz; (20) Khorasan; (21) Kerman; (22) Quetta–Kalat; (23) Salt Range; (24) Makerwal; (25) Tharparkar.

in the south. The Pennsylvanian sequence is around 1000 m thick and the basin is subdivided into regions: a northern region, comprising south-west Pennsylvania, eastern Ohio, northern Virginia, and north-eastern Kentucky; a central area, which reaches across southern West Virginia, eastern Kentucky, western Virginia, and northern Tennessee; and a southern region, covering southern Tennessee and northern Alabama. The northern region contains over 60 coal seams of varying economic significance, the central area contains around 50 coals, and the south-central area has up to 26 economic coal seams. Seams are usually between 0.5 and 3.6 m in thickness, with varying degrees of structural intensity.

Deep mines characterise the older workings; more recently, opencast mining in the form of contour mining and mountain top removal has been established.

Most reserves are in high- and medium-volatile bituminous coal, used chiefly as steam coal. The USA's largest reserves of coking coal are low-volatile, low-sulfur coals situated in central Pennsylvania and West Virginia.

The *Interior Province* comprises a number of separate basins containing Carboniferous bituminous coal. The eastern area is the Illinois Basin, which extends across central and southern Illinois, south-west Indiana, and western Kentucky. Pennsylvanian sediments reach 1200 m in thickness, with around 25 coals



**Figure 3.12** Coal deposits of South America. (1) La Rioja; (2) San Juan; (3) Mendoza; (4) Neuquen; (5) Pico Quemado; (6) Rio Turbio; (7) Copacabana Peninsula; (8) Tarija Basin; (9) Amazon; (10) Rio Fresco; (11) Tocantins–Araguaia; (12) Western Piaui; (13) Parana; (14) Santa Catarina; (15) Rio Grande Do Sul; (16) Candiota; (17) Copiapo; (18) Arauco; (19) Valdivia; (20) Magallanes; (21) Cundinamarca; (22) Santander; (23) El Cerrejon; (24) Valle del Cauca; (25) Malacatus; (26) Loja; (27) Canar–Azuay; (28) Alto Chicama; (29) Santa; (30) Oyon; (31) Jatunhuasi; (32) Zulia; (33) Lobatera; (34) Caracas–Barcelona; (35) Naricual; (36) Venado; (37) Zent; (38) Uatsi; (39) Coahuila; (40) Sonora.

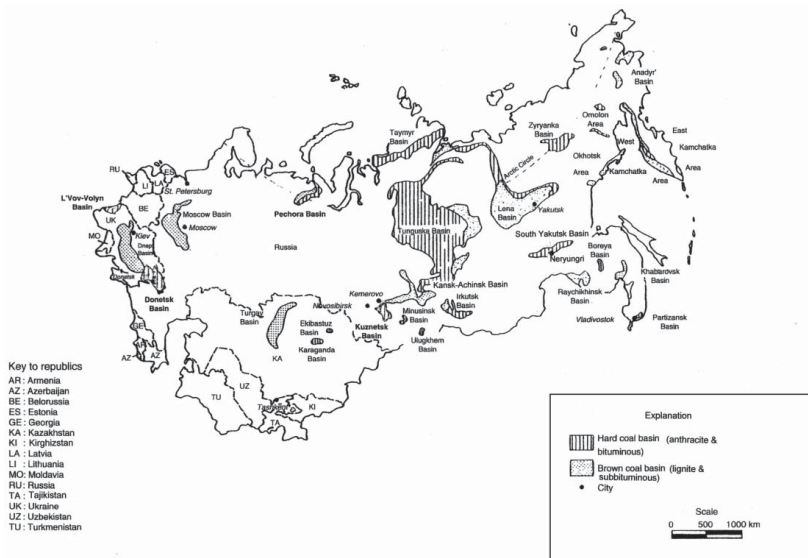
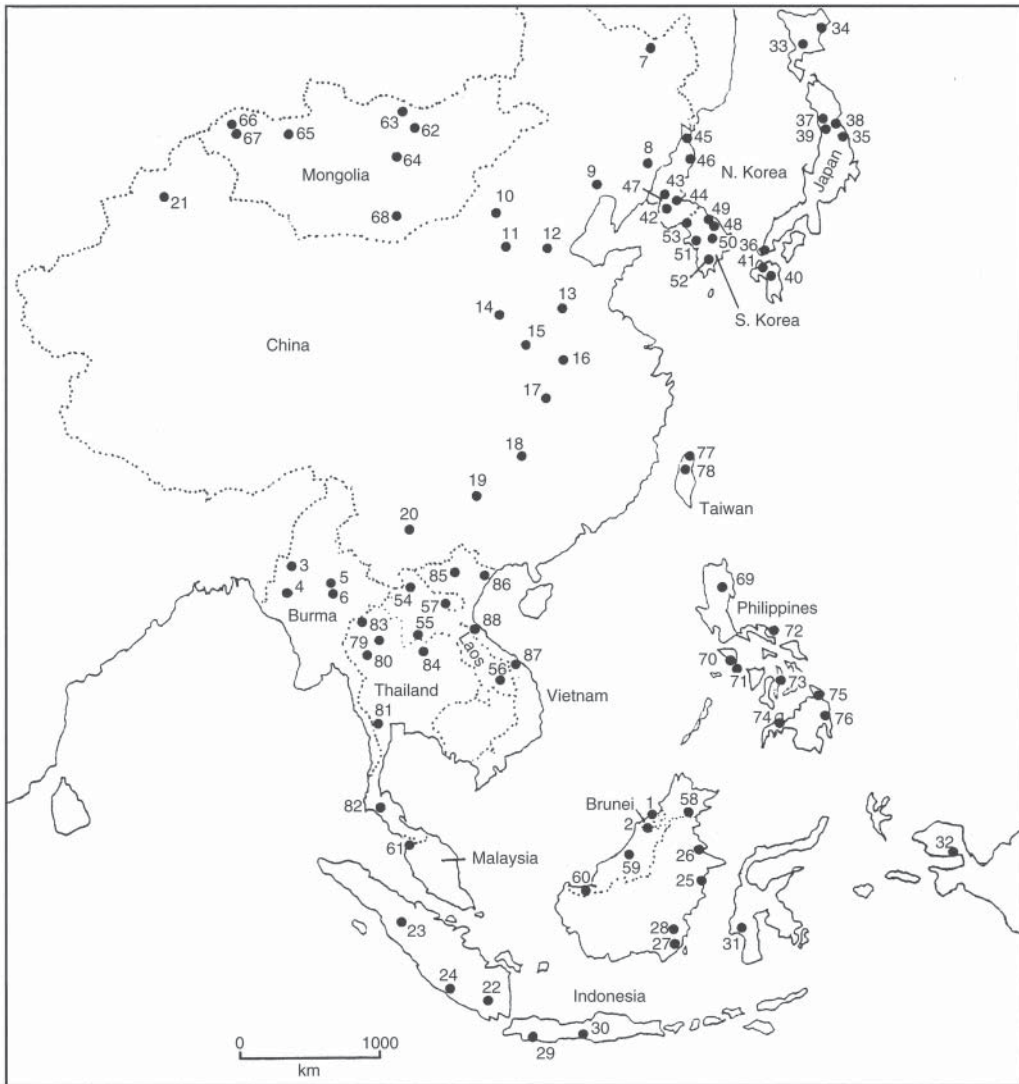
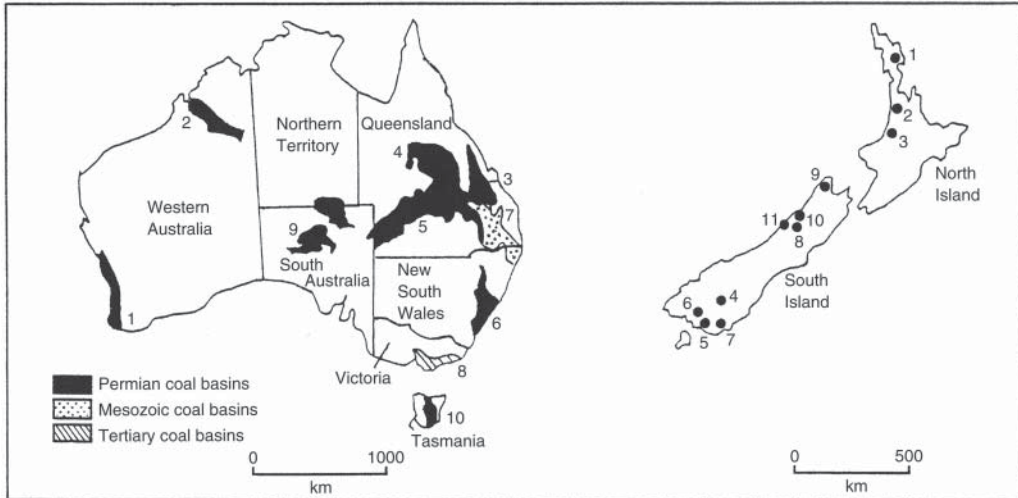


Figure 3.13 Coal deposits of the Commonwealth of Independent States: Donbass Basin; Kuzbass Basin; Karaganda Basin; Pechora Basin; South Yakutsk Basin; Ekibastuz Basin; Kansk–Achinsk Basin; Moscow Basin; Dnepr Basin; Neryungri; Lena Basin; Tunguska Basin; Turgay; West Kamchatka Area; Tajikistan; Georgia.



**Figure 3.14** Coal deposits of the Far East. (1) Bandar Seri Begawan; (2) Belait Basin; (3) Kalewa; (4) Pakkoku; (5) Panlaung; (6) Henzada; (7) Heilung-kiang; (8) Kirin; (9) Liaoning; (10) Shen Fu–Dong Shen; (11) Shanxi; (12) Hopeh; (13) Shantung; (14) Shaanxi; (15) Honan; (16) Anhwei; (17) Hupeh; (18) Hunan; (19) Kweichow; (20) Yunnan–Guizhou; (21) Xinjiang–Uyghur; (22) Bukit Asam; (23) Ombilin; (24) Bengkulu; (25) Sangatta; (26) Berau; (27) Senakin–Tanah Grogot; (28) Tanjung; (29) South Java; (30) Central Java; (31) South Sulawesi; (32) Bintuni; (33) Ishikari; (34) Kushiro; (35) Joban; (36) Omine; (37) Mogami; (38) Miyagi; (39) Nishitagawa; (40) Miike; (41) Chikuhō; (42) Pyongyang; (43) North Pyongyang; (44) Kowon–Muchon; (45) Kyongsang; (46) Kilchu–Myongchon; (47) Anju; (48) Samcheog; (49) Jeongseon–Kangnung; (50) Mungyeong; (51) Chungnam; (52) Boeun; (53) Kimpo–Yeongcheon; (54) Phongsaly; (55) Ventiane; (56) Saravan; (57) Muongphan; (58) Silimpopon; (59) Bintulu; (60) Silantek; (61) Bukit Arang; (62) Baganur; (63) Sharin Gol; (64) Nalayh; (65) Mogoyon Gol; (66) Achit Nuur; (67) Khartabagat; (68) Tavan Tolgoi; (69) Cagayan; (70) Mindoro; (71) Semirara; (72) Catanduanes; (73) Cebu; (74) Zamboanga; (75) Gigaquit; (76) Bislig; (77) Chilung; (78) Hsinchu; (79) Mae Moh–Li; (80) Mae Tun; (81) Nong Ya Plong; (82) Krabi; (83) Vaeng Haeng; (84) Na Duang; (85) Nan Meo–Phan Me–Bo Ha; (86) Quang Yen; (87) Nong Son; (88) Huong Khe.



**Figure 3.15** Coal deposits of Australasia. Australia: (1) Collie Basin; (2) Fitzroy Basin; (3) Bowen Basin; (4) Galilee Basin; (5) Cooper Basin; (6) Sydney Basin; (7) Brisbane Basin; (8) Gippsland Basin; (9) Ackaringa Basin; (10) Tasmania. New Zealand: (1) Northland Coalfield; (2) Waikato Coalfields; (3) Taranaki Coalfields; (4) Otago; (5) Southland; (6) Ohai; (7) Kaitangata; (8) Reefton; (9) Collingwood; (10) Buller; (11) Pike River–Greymouth.

of economic interest. In Illinois, coal rank increases from high-volatile bituminous C coal in the central and northern areas, to high-volatile bituminous B and A in west Kentucky. Coals generally have a high sulfur content of 3–7%, although low-sulfur coals are present.

Coal seams range from 0.5 to 2.5 m in thickness, and two seams, the Springfield and Herrin Coals, are >1.5 m and cover thousands of square kilometres. These seams have accounted for more than 90% of production from this region. The western Interior Province includes deposits in Iowa, Missouri, Kansas, Oklahoma, Arkansas, and Texas and is characterised by thinner (<1.5 m) but laterally extensive seams, mined exclusively by opencast stripping.

The southernmost part, the Arkoma Basin, has coals of higher rank, including semi-anthracites with low sulfur contents. Some of the low-sulfur coal has coking properties.

The *Northern Great Plains Province* contains coals of Cretaceous to Palaeogene–Neogene age. The chief coal-producing area is the

Powder River Basin, which lies in northern Wyoming and south-eastern Montana. Coals of Cretaceous age are present, but the major coal development occurred in the early Palaeogene–Neogene. Between 8 and 12 important coals averaging 6–30 m in thickness are present in 600–900 m of sediments. These are overlain by a further 300–600 m containing up to eight major coal seams. Some bituminous coals are present, but the main reserves are of subbituminous coal and lignite. These coals have assumed a greater significance in recent years because of their low sulfur content. This makes them attractive to users constrained by environmental legislation.

Cretaceous coals also occur in northern Montana, and these are bituminous in rank.

The *Rocky Mountain Province* contains coal preserved in a series of intermontane basins. In north-west Colorado and south-west Wyoming occur the two principal structural basins: the Green River Basin in the west and the Great Divide Basin in the east. The coal-bearing sediments range from late Cretaceous to early Palaeogene–Neogene in age and are more than 900 m in thickness, containing



multiple bedded coal seams. In west central Colorado and central Utah, the Piceance Basin contains over 3000 m of sediments of Late Cretaceous to early Palaeogene–Neogene age. In south-western Colorado and north-eastern New Mexico, Late Cretaceous coals are present in the San Juan Basin.

Most Cretaceous coal is bituminous and subbituminous, and the Palaeogene–Neogene coals are subbituminous and lignite. The seams are thick, in the range 3.0–10.0 m. Some are structurally affected, and some are ameliorated by igneous intrusives. In general, the coals have low sulfur contents (<1%), with some small areas of coal suitable for coking purposes. The usual mining practice is by opencast and contour stripping mines, with some drift mining.

The *Pacific Coast Province* covers those coal deposits found west of the Rocky Mountains and those in Alaska. The coal is Palaeogene–Neogene in age and is found in small, widely scattered basins from California in the south to Washington in the north. The coal has been tectonised and metamorphosed.

In Alaska, numerous deposits of subbituminous and high-volatile bituminous coal have been identified. The Usibelli open-pit mine contains low-sulfur subbituminous coal. Indications are that the coal resources could be up to 10% of the economically recoverable coal in the world.

Finally, the *Gulf Coastal Plain Province* contains lignites of Palaeogene–Neogene age. The lignite is produced from large opencast pits, where seams range from 1.0 to 7.5 m in thickness. Production has greatly increased in the last 20 years and is primarily for electricity generation.

### 3.4.2 Canada

Canada has very large coal resources and contains all ranks from lignite to anthracite. The largest coal-bearing region is located in western Canada, stretching from south Saskatchewan across Alberta into British

Columbia (Figure 3.8). The coals that underlie the plains are relatively undisturbed Palaeogene–Neogene and Mesozoic (Late Jurassic–Early Cretaceous) lignites and subbituminous coals, whereas those occurring in the mountains are Mesozoic high-volatile to low-volatile bituminous coals. The Late Jurassic–Early Cretaceous sediments are up to 2700 m in thickness in the mountainous region of south-west Alberta and south-east British Columbia; numerous coal seams are present, of which 14 have been mined with thicknesses of 2 m in Alberta to 14 m in British Columbia. Lower to Upper Cretaceous coals are present in west-central Alberta and north-eastern British Columbia, ranging from 2 to 13 m in thickness. In the central Alberta Plains region, coals of Late Cretaceous to Palaeogene–Neogene age occur; these are subbituminous coals that decrease in rank eastwards to Saskatchewan, where they are preserved as lignites. The majority of the western Canadian coals are low in sulfur content.

In eastern Canada, Carboniferous coals were mined in the Minto Coalfield, New Brunswick, and in the Sydney Coalfield, Nova Scotia (Figure 3.8). All are bituminous coals, some with high sulfur contents and coking properties. The Minto Coalfield supplied coal for generating electricity from a single 0.5 m seam. The Sydney Coalfield is now restricted to mining offshore, and again it supplies power stations and some coking coal for export.

In northern Canada, coals are found in the Yukon Territory and Northwest Territories. They are of the same age as the coals described for western Canada, comprising Mesozoic high-, medium-, and low-volatile bituminous coals, often highly tectonised, and Palaeogene–Neogene subbituminous coals and lignites. The Mesozoic coals are principally from the Yukon Territory, and Palaeogene–Neogene coals are found in both Yukon Territory and the Northwest Territories, including the Canadian Arctic islands. Older coals (Devonian) are known to occur in the Arctic islands (Ricketts and Embry 1986).

### 3.4.3 Europe

Coal deposits of Palaeozoic (Carboniferous), Mesozoic, and Cenozoic (Palaeogene–Neogene) age are developed in a series of basins that stretch from the UK in the west to Georgia in the east. The full range of black and brown coals is present, and all of the most accessible deposits have been extensively worked over the last 150 years.

Those European countries with recorded coal deposits are listed alphabetically and shown in Figure 3.9.

#### 3.4.3.1 Albania

Albania has small isolated occurrences of subbituminous coal and lignite scattered throughout the country. They produce small amounts of coal for local industrial use, chiefly from Tirane, Memaliaj, and Korçe-Pogradec. The use of coal in thermal power plants is planned.

#### 3.4.3.2 Austria

In Austria there are a number of small coal deposits; all are structurally complex. They include anthracite and bituminous coals, principally in the west of the country. Palaeogene–Neogene lignite and subbituminous coal are present in the West Styria area. Mining of lignite is on a small scale only, and production of bituminous coal has ceased.

#### 3.4.3.3 Belgium

Coal was mined in the provinces of Hainaut and Liège. The coal ranges from anthracite to low-volatile bituminous, with low sulfur content, some of which is used as coking coal. Seams may be up to 2.0 m thick and were worked by underground methods. Smaller coal deposits are known to the south, but mining has long ceased because of thin seams and difficult mining conditions.

#### 3.4.3.4 Bosnia

Palaeogene–Neogene lignite intermontane basins are present in several areas of Bosnia. In the Sarajevo-Zenica area, subbituminous coal

and lignite are present in large quantities. The coals have high moisture and ash contents. In the east of Bosnia, at Gacko, lignite seams are up to 18 m, and in the Ugljevic area the seams are up to 28 m in thickness. In west Bosnia, seams of lignite up to 25 m in thickness are present in the Kongora area but are as yet unmined, and a 7 m lignite seam is currently mined at Stanari. These lignites have variable ash contents due to the presence of numerous thin non-lignite partings, and sulfur contents are around 1%. Current lignite production is used for power generation.

#### 3.4.3.5 Bulgaria

The most extensive coal deposits in Bulgaria are brown coals of Palaeogene–Neogene age. Most of the mining areas are situated in the south-west and central parts of the country. The principal producers are the opencast mines at Chukurova, Pernik, Beli Breg, and Maritsa East, with both underground and opencast production from Bobov Dol mine complex and Maritsa West reaching depths of 200 m. The seams are thick, some over 20 m, and have undergone little structural disturbance. They are often low grade with high sulfur content. Lignite is currently used to provide up to 37% of Bulgarian power generation.

High-rank bituminous coals are known from the Dobrunda area in the north-east of the country; here, the coals are deep and so far have not been mined on a significant scale.

#### 3.4.3.6 Czech Republic

The Czech Republic has numerous deposits of black and brown coals spread widely across the country. The chief black coalfield is that of Ostrava-Karvina on the north-eastern border, containing Upper Carboniferous strata that represent a continuation of the Upper Silesian Coalfield in Poland. The lower part of the sequence varies between 1500 and 3000 m in thickness and contains 170 coals with an average thickness of 0.7 m. The upper sequence is 1200 m thick with 90 coal seams of between 5 and 15 m in thickness. The area is structurally

complex. The coal produced is low-volatile bituminous, some of which is strongly caking, and anthracite. These coals usually have low ash and sulfur contents. Other smaller bituminous coal deposits are found north and west of Prague. All black coal is mined in underground operations.

Palaeogene–Neogene lignites are located in the north-west of the Czech Republic, in North Bohemia (Most), western Bohemia (Sokolov), and in southern Moravia (Hodonin) and have been mined in all these areas. Seams are up to 30.0 m in thickness, with low sulfur contents and little structural disturbance. Underground mining of brown coal has now ceased. Other lignite deposits are present in the south of the country but are not currently being exploited.

#### 3.4.3.7 Denmark

Lower and Middle Jurassic coals are recorded on the island of Bornholm, and Palaeogene–Neogene lignites are found at Herning, but they are no longer worked as reserves are small.

#### 3.4.3.8 France

In northern France, two large Carboniferous basins, those of Lorraine and Nord et Pas de Calais, contain high-volatile bituminous coal, some suitable as coking coal. Seams are up to 2.0 m thick and have been highly tectonised; all have been mined by underground methods. To the south lies the Carboniferous Cevennes Basin, and other smaller scattered coal deposits are known from western and south-western France; all are structurally complex, and mining has ceased in these areas.

The coalmining industry in France has now virtually ceased, due to increasing production costs and competition from other fuel sources.

Lignite has been mined in the Provence district, where the seams are relatively undisturbed, to supply the local electricity industry.

#### 3.4.3.9 Germany

The black coals of Germany are located in the Carboniferous basins of Aachen, Ruhr,

and Saar, with small deposits present in the south-east of the country.

The Aachen Basin produces high- and low-volatile bituminous coal with low ash and sulfur, some of which is coking coal, from seams up to 1.5 m in thickness.

The Ruhr basin contains 3000 m of upper Carboniferous sediments containing around 300 coal seams. The area has been subjected to block faulting and folding, and the Ruhr basin produces high-volatile bituminous coal with low ash and sulfur and is a good coking coal. Seam thicknesses range from 0.5 to 3.0 m. A northern extension of this area is the Lower Saxony Coalfield, which produces low-volatile anthracite.

The Saar Basin is less structurally disturbed and produces bituminous coal from seams 0.5–2.0 m in thickness. This coal is not suitable for coking purposes.

The brown coals of Germany are of Palaeogene–Neogene age, and significant deposits are found in three basins: the Rhenish, the Halle Leipzig Borna (the central German mining area), and Lower Lausitz (the Lusatian mining area).

The Rhenish Basin is situated close to the Ruhr; it contains thick seams (up to 90.0 m) of lignite with low ash and sulfur contents. The coalfield has suffered little structural disturbance and is mined on a large scale by opencast methods for use in the electricity generation industry.

The Halle Leipzig Borna and Lower Lausitz coalfields are situated in the east of the country. Again, they contain thick seams of lignite that has high volatiles, low ash, and sulfur contents of 0.3–2.0%. Both areas are mined on a large scale by opencast methods.

Germany is Europe's largest lignite producer; however, pressure from the use of renewables and environmental considerations is likely to cause a reduction in lignite production in the years to come.

#### 3.4.3.10 Georgia

Large reserves of Jurassic black coal are present in the Tkibuli–Shaori and Tkvarcheli

coalfields in central and north-west Georgia. This is currently mined underground at the Mindeli Mine. In the south, the Akhaltsikhe coalfield has large resources of brown coal that are, as yet, undeveloped.

#### 3.4.3.11 Greece

Lignite deposits occur widely across Greece and is the second largest lignite producer in Europe. Greece has no black coal reserves and, consequently, imports such coal. The principal deposits are late Palaeogene–Neogene to Quaternary in age. The principal deposits are situated in the north of Greece at Florina, Amyndaeon, Ptolemais, and Ellassona. To the south occurs the Megalopolis deposit, and to the north-east is the Drama lignite field. The number and thickness of the lignite seams varies both between and within individual deposits. The Ptolemais basin contains one major thick lignite seam that is 60 m in thickness; and in Amyndaeon and Megalopolis, lignite seams are up to 30.0 m thick. The lignite is generally of poor quality due to the presence of numerous non-lignite interbeds. Although the ash and sulfur contents can vary greatly, the current mining areas produce lignite with typically 15–20% ash and sulfur <1%. All is surface mined; chiefly for electricity generation.

#### 3.4.3.12 Greenland

Although geographically separate from Europe, Greenland is included here for convenience. Coal occurrences have been reported from several areas in Greenland. Middle Jurassic coals are recorded on the north-east coast of Greenland. The most significant are the sub-bituminous coals with low sulfur contents that are found on the coast at Disko and Nugssuaq.

#### 3.4.3.13 Holland

Coal-bearing Carboniferous sequences are known at depth beneath younger sediments; however, seams are thought to be thin. No coalmining is presently carried out in Holland.

#### 3.4.3.14 Hungary

Black coal has been mined in the south-west of Hungary, in the area around Mecsek. Here, coals of Mesozoic age are present in a structurally complex area, strongly folded and faulted, with associated igneous intrusives. Seams are steeply dipping but can be thick locally (>5.0 m). The coal is weakly caking bituminous with high ash and sulfur contents. The area is a well-established coalfield, mining in difficult conditions; but mining has declined in recent years. The opencast mine at Nagymanyok is currently being investigated to produce bituminous coal.

Brown coal deposits of Mesozoic and Palaeogene–Neogene age are found in a north-east–south-west belt of country in the north of Hungary. Lignite has been mined at Oroszlany, Tatabanya Dorog, and the mines at Visonta–Bukkabrany are operating on a small scale. The coals are mined by underground and opencast methods, with ash and sulfur contents of 20% and 1% respectively. All is utilised as fuel for local power plants.

#### 3.4.3.15 Ireland

The coal deposits of Ireland are all of Carboniferous age. Four main coal-bearing areas have been mined, at Leinster, Kanturk, and Slieveardagh in the south and at Connaught in the north-west. The coal is anthracite with variable sulfur contents in the south, and it is medium-volatile bituminous coal in the north-west. Coal has been mined on a small scale from thin seams.

#### 3.4.3.16 Italy

Carboniferous coals are present in the structurally complex areas of the Alps and Sardinia.

Palaeogene–Neogene lignites and subbituminous coals are found in the Apennines and Sardinia. The latter coalfield at Sulcis has subbituminous coal as a result of volcanic amelioration. Mining is on a very small scale.

#### 3.4.3.17 Kosovo

Kosovo has large lignite resources situated in three areas: the Kosovo, Dukagjin, and

Drenica Basins. The coals are of Pliocene age, and seam average thickness is 40 m; mining has concentrated in the Kosovo Basin, with the Sibovc mining area being targeted for future development.

#### 3.4.3.18 Montenegro

The Republic of Montenegro has the Pljevlja opencast mine providing lignite from one thick seam to the local power plant.

#### 3.4.3.19 Poland

Poland has large reserves of coal and a long-established coalmining industry. Black coal in Poland is centred around three coalfields: Upper Silesia, Lower Silesia, and Lublin. These areas represent large basins containing Carboniferous sediments that have undergone varying degrees of structural disturbance.

The Upper Silesian Basin contains a thick sequence of upper Carboniferous sediments, up to 8500 m. The lower part of the sequence contains 250 coal seams and the upper part has 60 coal seams, with the thicker seams reaching 6–7 m. The basin is highly tectonised, so that mining operations are complicated by large-scale faulting and folding. Coal rank is also affected by igneous intrusions of Permian, Triassic, and Miocene age. The coal is primarily high-volatile bituminous, with low ash and sulfur contents. The coals have caking and coking properties in the far south of Poland. These are a major export commodity.

The Lower Silesian Coalfield is a much smaller area containing thinner seams that are highly tectonised. The remaining reserves are deep, but have been an important supplier of coking coal. Mining operations have ceased in this area.

The Lublin Coalfield, only discovered relatively recently, is a very large area with potentially enormous reserves. The seams appear to be less structurally disturbed than in the Silesian coalfields. Lublin has bituminous coals with low ash and sulfur contents together with strong coking properties. The sequence of productive coal seams lies at

depths of up to 700 m, requiring the development of underground coalmining. The Bogdanka underground mine is currently the only working mine in the Lublin Coalfield.

Palaeogene–Neogene lignite basins are present in central and south-western Poland, many of which are fault bounded. Production is centred on four areas; namely, Adamov, Belchatow, Konin, and Turow. Lignite thickness ranges from 5 m to over 70 m; ash and sulfur contents are low in the thicker lignites, which are mined to supply local power plants. Poland is the third largest European producer of lignite.

#### 3.4.3.20 Portugal

Anthracite of Carboniferous age is present in the north-west of the country, but it is not extensively worked. At Cabo Mondego, bituminous coal, high in ash and sulfur, is produced on a small scale.

A number of basins contain lignite; in particular, that at Rio Maior is a small producer.

#### 3.4.3.21 Romania

The Palaeogene–Neogene lignite reserves of Romania lie in a series of deposits aligned east–west in the south of the country. The principal deposit is at Oltenia, where up to 75% of production is obtained, mining seams up to 30 m in thickness by both opencast and underground methods. Other deposits at Berbesti and Ploiesti are also exploited. The lignite is high in ash and sulfur content and is used exclusively for electricity generation, providing 25% of Romania's power.

Higher rank Palaeozoic coal is present at Banat in south-west Romania, and in the Jiu Valley, where four remaining underground mines supply thermal coal to local power plants.

#### 3.4.3.22 Serbia

Serbia, in common with many other European countries, has black coal of Palaeozoic (Carboniferous) and Mesozoic ages, preserved in the older structurally complex regions of the



country, and Palaeogene–Neogene brown coals that are not affected structurally. The latter are more extensive than the older deposits.

The black coal deposits occur in the south-east of the country; most are subbituminous to low-volatile bituminous coal with relatively high sulfur contents. The majority of the underground workings have now closed.

Serbia has large proved deposits of lignite, and current production is surface mined at Kolubara, Kostolac, and Kovin, all providing fuel for electricity generation. The Kostolac Basin has 45 m of lignite in three seams, and the Kolubara basin has three lignite seams totalling 18–45 m. These lignites have low ash and sulfur contents.

#### 3.4.3.23 Spain

The principal black coal basin is that in the Leon region in the north of the country. The Nalon Valley and Leon–Palencia areas contain low-volatile bituminous coals with low sulfur contents, some of which are usable as coking coal. Seam thicknesses are up to 1.5 m, and the area is structurally complex. In the Astur-Leonese Basin in the region of Castilla y Leon, anthracite with high calorific values and low volatiles are mined on a small scale.

In the south of Spain, south-west of Puertollano, low-volatile bituminous coals are found, and Spanish brown coals are located at Teruel, where shallow lignites of Cretaceous age are mined for power generation.

The major lignite production comes from the Puentes de Garcia Rodriguez and Meirama areas in the north-west of the country. Up to 17 lignite seams are exploited in opencast mines to depths of 170 m; the lignite is high in sulfur and is used for local electricity needs. Other lignite deposits are present at Calaf and Mequinenza in the north-east, and Arenas del Rey in the extreme south of Spain. These are also used for local electricity requirements.

#### 3.4.3.24 Spitzbergen

Carboniferous and Palaeogene–Neogene coals are present on the western side of the island. At

Longyearbyen and Svea underground mines, high-volatile bituminous coal with low sulfur is mined in seams up to 5.0 m in thickness.

#### 3.4.3.25 Sweden

Thin Triassic (Rhaetian) coals have been recorded and studied in the Höganäs Basin, southern Sweden, but are not of commercial significance.

#### 3.4.3.26 Turkey

Turkey has considerable reserves of Carboniferous black coal and Palaeogene–Neogene brown coal. The principal black coal deposit is the Zonguldak Coalfield on the northern coast, where numerous seams ranging from 0.7 to 10.0 m are present. The coal is bituminous with low ash and sulfur contents and is suitable for use as a coking coal. The coalfield is structurally complex, and mining is heavily subsidised.

Palaeogene–Neogene lignite basins are present across west and central Turkey. Older Eocene lignites are found in the north, with seams up to 6 m in thickness. Younger Oligocene and Miocene lignites occur in the north-west and west of the country respectively. The Oligocene is characterised by numerous thin seams, whereas the Miocene lignites form the larger deposits with seams up to 25 m in thickness in the Alpagut-Dodurga basin and in the Turgut coalfield in south-west Turkey. In central and eastern Turkey, Pliocene deposits contain large lignite resources, with one or two seams averaging 40 m in thickness. The largest opencast operation at Afsin-Elbistan mines a seam 5–58 m thick. The older lignites have a higher heating value together with high sulfur contents, and the youngest lignites have high ash and moisture values. Current and future use of lignite is planned for electricity generation

#### 3.4.3.27 United Kingdom

In the UK, a series of coal-bearing basins is distributed throughout the country. The coals are of Carboniferous age and are principally

bituminous with some anthracite, notably in South Wales.

The principal areas are those of Scotland, North-east England, the Yorkshire–Nottinghamshire region, the Lancashire–North Wales region, the East and West Midlands, South Wales, and Kent. Virtually all of the underground mining in the UK has now disappeared and been replaced by opencast operations.

The Yorkshire–Nottinghamshire region has been the most important coal-producing area in the UK, supplying coal for electricity generation and coking coal to industry. The region is less structurally affected than other areas and has a long coalmining history. The north Nottinghamshire area together with the South Yorkshire area produce high-volatile bituminous coal, some of which is strongly caking. This coal is used chiefly for the electricity generating industry, and declining amounts for the steel-making industry.

The Lancashire–North Wales region in the past has produced high-volatile bituminous coal with low sulfur content. Owing to extensive working and difficult mining conditions, mining in the region has now ceased.

In North-east England, bituminous coals suitable for coking purposes are produced in opencast operations.

The East and West Midlands contain four coalmining areas, North and South Staffordshire, Warwickshire, and Leicestershire. The region has produced high-volatile bituminous coal for electricity generation. Warwickshire currently produces from the Warwickshire Thick Seam, which can be up to 8.0 m in thickness, and is used for power generation.

In Scotland, high-volatile bituminous coals with low sulfur are opencast mined. This area has been extensively mined in the past.

South Wales, once the principal coalfield in the UK, still produces bituminous coal and anthracite with low sulfur contents; seams are usually between 1.0 and 3.5 m. Underground mining conditions are difficult and expensive; currently, only one small underground mine

is in operation. The bulk of production comes from two opencast mines in the coal outcrop areas.

Kent, in the south-east of the UK, has a small coalfield that contains high-quality bituminous and anthracite coal, but it is no longer a producer.

Palaeogene–Neogene lignites are found in south-west England and in Northern Ireland. In Northern Ireland, lignite deposits have been identified for future use in local power stations.

### 3.4.4 Africa

Black coal occurs in Africa first in those deposits of Carboniferous age found on the northern coast, in Morocco in the west and Egypt in the east, and second, and more importantly, in the widespread Karoo deposits of late Carboniferous–Permian age that are found throughout central and southern Africa (Haughton 1969). The Karoo sequences were deposited on the Gondwana supercontinent, which split apart in the Mesozoic Era; hence the similarities of African Gondwana coals with those of India and South America.

Brown coals of Palaeogene–Neogene age are present, but in Africa it is the black coals that are of prime interest. The principal coal occurrences are shown in Figure 3.10.

#### 3.4.4.1 Angola

Lignites of Palaeogene–Neogene age have been identified in Angola. These are in the east, around the headwaters of the Lungue-Bungo River, where seams of lignite up to 2.5 m have been recorded, and in the west around Luanda, where lignites are present in the Palaeogene–Neogene coastal sediments. None of these deposits have been worked.

#### 3.4.4.2 Botswana

Botswana has large reserves of black coal of Karoo age. These coal deposits extend from north to south along the eastern edge of the country. The more important coalfields are those of Morupule and Mmamabula. At

Morupule, seams up to 9.5, 4.5, and 2.0 m in thickness are present. At Mmamabula, seam thicknesses average 2.8, 5.4, and 2.0 m. In both these coalfields, the coals are relatively undisturbed and contain bituminous coal with a high ash and sulfur content; these coals have no coking properties.

Other smaller coalfields are present in close proximity to Morupule and Mmamabula.

Botswana has the potential to be a significant coal producer, but at the present time it is geographically disadvantaged to be a coal exporter.

#### 3.4.4.3 Cameroon

In the Bamenda district, lignites are found interbedded with lava flows. They are of Cretaceous–Palaeogene–Neogene age, and locally can be up to 6.0 m in thickness, but they are undeveloped.

#### 3.4.4.4 Egypt

Carboniferous coal is present in the Sinai Peninsula. Coals are of bituminous and sub-bituminous rank. Coal has been produced from workings at Al Maghara, at which future development is to be considered.

#### 3.4.4.5 Ethiopia

Palaeogene–Neogene brown coals are known from many localities on the Ethiopian Plateau, beds are up to 15.0 m, and range from lignite to subbituminous coal. They have high ash and low sulfur content. Principal localities are Chelga, Wuchalle, and Dobre-Brehan.

#### 3.4.4.6 Malagasy Republic

Black coal is present in the Karoo sediments on the western side of the island, where they overlie the Precambrian basement. At the southern end of the Karoo outcrop, five coal-bearing areas have been identified.

The northernmost area is the Imaloto Coalfield, which contains seams averaging 1.0 m in thickness. The coal is medium-volatile bituminous with high ash and some high sulfur contents. The Vohibory and Ianapera coalfields have seams up to 2.3 m and 0.6 m respectively;

both areas are structurally complex. The Sakoa Coalfield is the best-known area, with seams of 3.0 and 7.0 m in thickness. The coals are high-volatile bituminous with high ash and low sulfur contents; they are non-coking. The Sakamena Coalfield is similar to Sakoa except that the seams are thinner.

Lignite deposits of Palaeogene–Neogene age are present in the region of Antanifotsy and are thought to cover a large area.

#### 3.4.4.7 Malawi

In Malawi, a series of separate basins contain coal-bearing Karoo sediments; these are located in the extreme north and south of the country.

The main coalfields are those of Livingstonia, Ngana, and North Rukuru, with small deposits at Lengwe and Mwabvi in the south.

The Livingstonia Coalfield contains seams 1.0–2.0 m in thickness, which are mined to supply fuel to local industry.

At Ngana, one seam is up to 15.0 m in thickness, but seams usually average around 1.0 m, and they show rapid vertical and lateral variations in thickness. The southern coalfields have thin seams and are not well developed.

Malawian coals are subbituminous to high-volatile bituminous with high ash and low sulfur contents.

#### 3.4.4.8 Mali

Upper Cretaceous and Palaeogene–Neogene brown coals are recorded from the Mali–Niger Basin in the south-eastern part of the country, around Bourem. Seams are thought to reach 2.0 m in thickness, and they have moisture values of 24% and ash values of 21%.

#### 3.4.4.9 Morocco

Carboniferous black coals are found in the north-east of Morocco, at Jerada, and have been identified at depth beneath younger sediments at Ezzhiliga and Tindouf-Draa.

At Jerada, four seams of up to 0.7 m in thickness are mined. They are structurally unaffected, and the coal is low-volatile anthracite with low ash and high sulfur contents.

Lignites have been identified at Meknes-Fez in northern Morocco.

#### 3.4.4.10 Mozambique

Karoo sediments are preserved in a series of basins in the Precambrian basement. The coal-bearing Karoo outcrop is a long strip running eastward from the southern tip of Lake Malawi. Four coalfields have been identified; of these, the Moatize deposit is the most important.

At Moatize, seams range from 0.4 to 4.0 m in thickness; structurally, the coalfield is heavily faulted. Both coking and thermal coals are produced, and the coal is low-volatile bituminous, with high ash and low sulfur contents, chiefly for export. The coal deposits at Mmam-bansavu, Chiomo, and Itule are as yet little known.

#### 3.4.4.11 Namibia

The eastern half of Namibia is covered by post-Karoo sediments of the Kalahari Group. It is possible that Karoo sediments underlie a portion of this area and may contain coals of similar aspect to those found in Botswana.

#### 3.4.4.12 Niger

The Mali-Niger Basin contains coals of late Cretaceous to Palaeogene-Neogene age. Little information is known but these coals may be worked locally.

#### 3.4.4.13 Nigeria

Coal-bearing sediments of Cretaceous and Palaeogene-Neogene age overlie Precambrian basement in the south-eastern part of Nigeria. These sediments dip to the west, where they are overlain by floodplain deposits of the River Niger.

The Nigerian Coalfield is divided into several mining areas: the Enugu, Ezimo, Orukpa, Okaba, and Ogboyoga coalfields. Seams range from less than 1.0 m to 3.0 m, and all the coalfields are affected by faulting and gentle folding. Coals from these coalfields are high-volatile subbituminous with high ash and

low sulfur contents. Similar coal is reported from the Lafia area situated to the north of the Enugu Coalfields; here, the coal is similar but has a higher sulfur content. After a period of decline, there is current interest in Nigerian coal.

Palaeogene-Neogene lignites are found in the Asaba region, close to the River Niger; seams are 3.0-7.0 m in thickness. They are high-volatile, high-ash lignite with low sulfur content.

#### 3.4.4.14 South Africa

The coal deposits of South Africa are found in a series of basins situated in the north and east of the country. The Karoo System contains coal-bearing sediments of Carboniferous-Permian (Gondwana) age.

The main Karoo basin extends for 200 km from Free State Province in the west to south and east Mpumalanga Province (formerly Transvaal), and for 400 km from Mpumalanga in the north to Kwazulu-Natal in the south. The Karoo sequence was deposited directly onto basement, and the coal seams are shallow and almost horizontal. They have been affected by numerous igneous intrusions that have produced a great variation in rank, often very localised. The western area consists of the Vereeniging-Sasolberg and South Rand coalfields, which contain coals 10-25 m thick. The northern area comprises the Witbank, Eastern Mpumalanga (formerly Eastern Transvaal) and Highveld coalfields, where up to five seams are present; two of these, up to 10 m thick, are worked. The southern Kwazulu-Natal area includes the Vryheid and Utrecht coalfields, again with five seams, of which two are worked, together with the Kliprivier Coalfield, which has two coal seams with a thickness of up to 15 m. The basin as a whole produces high-volatile bituminous coal, with high ash and variable sulfur contents; some of the coal is weakly caking, for electricity generation, and to produce liquid fuel. The majority of the mines are underground operations, and the region is a major exporter of steam coals. In

eastern Mpumalanga and Natal, anthracite is also produced.

In Cape Province, the Molteno–Indwe Coalfield has coals that are of lower rank than those to the north-east.

Other coalfield basins in the north-east of the country are less developed. Of these, the Waterberg Coalfield, on the Botswana border, and the Springbok Flats area appear to have future potential. The Limpopo and Lebombo coalfields have bituminous coals with high ash contents. South Africa's coals are generally low in sulfur but high in ash. Beneficiation is essential for export-quality coal. Lower quality coal is used for the local power generation market.

#### **3.4.4.15 Swaziland**

Coal-bearing Karoo sediments are located on the eastern side of the country. The seams are thicker in the north and are flat lying; some have been ameliorated by dolerite intrusions. The Mhlume Coalfield, in central Swaziland, produces anthracite on a small scale. Coal is also known from the Maloma area in the south of the country.

#### **3.4.4.16 Tanzania**

There are eight coalfields in Tanzania, the coal-bearing sediments are preserved in depressions in the Precambrian basement; all the coalfields are located in the south-west of the country. The Ruhuhu Coalfields have been known for a century but have never been fully developed. These include the Ketewaka-Mchuchuma, Ngaka, Songwe-Kiwira, Galula, Njuga, and Ufipa coalfields. Mines at Kiwira and Mchuchuma are being developed. In these coalfields, coals occur in two zones, with the lower one containing the better coals; seams can be as thick as 7.0 m, but this is exceptional. Coals are high- to low-volatile bituminous with high ash and low sulfur contents.

#### **3.4.4.17 Zaire**

Small, separate basins of coal-bearing Karoo sediments occur in the south-east of the

country at Luena and Lukuga. Seams are up to 2.0 m in thickness and are disrupted by faulting. The coals are bituminous with high ash contents and are used locally for electricity generation.

#### **3.4.4.18 Zambia**

Karoo sediments are preserved in depressions in the Precambrian basement. A series of such basins is present in the east and south-east of the country – namely the Luangwa, Luano, and Maamba areas – and also in the west-central district around Kahare.

The Luangwa coals are up to 1.6 m thick, and are high-volatile bituminous high-ash coal. The Luano area has fairly thin seams that are high-volatile bituminous, with high ash content; some coal has coking properties. The Maamba area in the south-east has seams 2.0–3.0 m in thickness, and is high volatile bituminous with high ash content. The Maamba area currently has two operating mines which produce most of Zambia's coal.

At Kahare, coals are preserved beneath younger sediments, coal quality is similar to other Zambian coals, this area has not yet been fully investigated.

#### **3.4.4.19 Zimbabwe**

The Karoo sequence in Zimbabwe is preserved in the Zambezi basin in the north-west and the Limpopo basin in the south-east. The north-west includes the coalfield districts of Wankie and Lubimbi, with Sessami–Kaonga to the east of these. In these coalfields, the coal is the Wankie Main seam, a medium- to high-volatile bituminous coal, comprising a lower coking coal up to 4.0 m in thickness and an upper steam coal up to 8.0 m, all generally with low sulfur contents. Hwange Colliery Company is the biggest producer, with a number of new mining companies becoming interested.

In the southern coalfields of Buby and Tuli, the coals have variable qualities. Some low-sulfur coking coal has been identified in the Tuli Coalfield.



### 3.4.5 The Indian Subcontinent

The area delineated the Indian subcontinent extends from Iran in the west to Bangladesh in the east. Black coals are of Palaeozoic (Carboniferous–Permian), Mesozoic, and Cenozoic age, Brown coals are of Cenozoic age.

Palaeozoic Gondwana coals are found in India, Pakistan, and Bangladesh; Mesozoic coals are present in Afghanistan, India, Pakistan, and Iran; and Cenozoic coals are found in all the countries listed in this region. For distribution, see Figure 3.11.

#### 3.4.5.1 Afghanistan

Mesozoic (Jurassic) black coals are present in the northern mountainous regions of Takhar and Badakhshan. The coal is relatively undisturbed, with seams up to 1.5 m in thickness. The coal is bituminous with low ash and sulfur contents, with coking properties. Coal is mined at Herat in the west and at several other sites in the north. All are small operations and produce for the local market only.

#### 3.4.5.2 Bangladesh

Gondwana coals are found at depth, concealed beneath Palaeogene–Neogene sediments in north-western and eastern Bangladesh. The Gondwana sediments represent the infilling of depressions in the underlying crystalline basement. These basins have been faulted at the margins, resulting in gently dipping coal seams being preserved in graben structures.

In the north-west, the concealed coal basins of Barapukuria, Khalaspir, and Jamalganj contain numerous seams ranging in thickness from less than 1.0 m to 20.0 m and 30.0 m. The coals are medium- to high-volatile bituminous, with high ash and low sulfur contents. The Barapukuria Basin produces coal from Dinajpur, Phulbari, and Khalashpur.

In the east of Bangladesh, lower rank coal is located at Sylhet; the coal is subbituminous and lignite.

#### 3.4.5.3 India

In India, coal resources are of Palaeozoic (Gondwana) and Cenozoic (Palaeogene–Neogene) age.

About 98% of India's coal reserves are of Gondwana coal, which also accounts for 95% of production, chiefly for electricity generation and the metallurgical industries. The Gondwana coals are present in over 14 separate basins centred in the north-eastern and central eastern parts of peninsular India.

Palaeogene–Neogene brown coals are present in the north-eastern and north-western parts of the country, together with an important lignite deposit in the south at Neyveli.

The principal coalfields containing Gondwana coals are those of Raniganj in West Bengal State, Ramgarh, Jharia, Karanpura, and Bokaro in Bihar State, Singrauli, Bistrampur, Pench–Kanhan, and Tawa Valley in Madhya Pradesh, Kamptee, Bandar, and Wardha Valley in Maharashtra State, Ib river and Talcher coalfields in Orissa, and Godavari in Andhra Pradesh. In addition, numerous other fields in the same region are producing coal. Seams in these coalfields range in thickness from 1.0 to 30.0 m, with an exceptionally thick seam of 134.0 m discovered in the Singrauli Coalfield. The coalfields have been faulted but otherwise are not highly tectonised. Coals range from high- to low-volatile bituminous with high ash and variable sulfur contents. In the Jharia and Raniganj coalfields, good-quality coking coals are produced.

The Palaeogene–Neogene coals are highly disturbed tectonically and are located in the mountainous regions of north-east India. In the Makum Coalfield in Assam, seams are lens-shaped, in places reaching thicknesses of 33.0 m. Coals are subbituminous to high-volatile bituminous with high sulfur contents.

In southern India, in the state of Tamil Nadu, Palaeogene–Neogene lignites are found in Neyveli, the thickest seam being up to 20.0 m in thickness. Here, the lignite is low volatile, with low ash and sulfur content. Lignite is

now also being mined in the north-western states of Gujarat and Rajasthan. All these areas will increase in importance as the demand for electricity generation increases.

Although India's coal reserves cover all ranks of coal from lignite to bituminous, they tend to have high ash content and a low calorific value. This prevents India from being a major exporter of coal.

#### 3.4.5.4 Iran

The black coal deposits of Iran are Mesozoic (Jurassic) in age, with some lignites of Palaeogene–Neogene age. The Jurassic coals are bituminous with high ash and sulfur contents, and they have coking properties. All are strongly tectonised, with seam thicknesses ranging from 1.0 to 4.0 m. The coal supplies local needs and the metallurgical industry. Principal coalfields are located at Golestan and Mazandaram provinces in the north and at Kerman in central Iran. Lignites are found in north-west Iran but are not worked.

#### 3.4.5.5 Pakistan

All the principal coalfields in Pakistan are of Palaeogene–Neogene age, although Palaeozoic and Mesozoic coals are present. The coalfields of economic importance are situated in three distinct coal regions: Sindh, Quetta–Kalat, and Salt Range–Makerwal. Most of these coalfields have been structurally disturbed.

The central part of Sindh province contains the coalfields of Lakhra, Sonda–Thatta, and Meting–Shimpir. Seams are up to 2.0 m in thickness, and the coal is subbituminous, noncoking with high sulfur content.

In eastern Sindh province, the Thar Coalfield covers an area of 9000 km<sup>2</sup>. Miocene lignites are low in sulfur and can be in excess of 30 m in thickness. This coalfield is targeted to provide fuel for electricity generation and industrial use.

The Quetta–Kalat province contains the coalfields of Sor Range–Daghari, Khost Sharig–Harnai, and Duki–Chamalang. Again, the coal is subbituminous with high ash and

sulfur contents. The Salt Range–Makerwal province comprises the coalfields of eastern, central, and western Salt Range, together with the Makerwal Coalfield to the west of these. Coals are subbituminous, with high ash and sulfur contents. Overall production is small, the coal being used chiefly for electricity generation.

### 3.4.6 Central and South America

Coal deposits are distributed throughout Central and South America and make up a significant proportion of world reserves of black coal. The majority of coals are of Cenozoic (Palaeogene–Neogene) age. Coals of Palaeozoic (Gondwana) age are present in eastern South America, in Brazil and Uruguay, and Mesozoic coals are found in discrete deposits throughout the region (see Figure 3.12).

#### 3.4.6.1 Argentina

The coal deposits of Argentina are preserved in a series of basins in the Andean Cordillera and pre-Cordillera and in Austral Patagonia. Coals are of Carboniferous–Triassic, Jurassic, and Palaeogene–Neogene ages. Of these, the Palaeogene–Neogene coals are the principal deposits of economic interest.

Coals of Carboniferous and Permo-Triassic age are found in the La Rioja–San Juan region. They consist of thin discontinuous seams less than 1.0 m thick and highly tectonised. Coals are low- to medium-volatile bituminous with some anthracites at Mendoza. Jurassic coals are found south of San Juan in the Neuquen region, preserved in a series of small basins. Seams are thin, normally less than 1.0 m thick, and are medium-volatile bituminous at Neuquen.

Palaeogene–Neogene coal-bearing sediments are preserved in a large basin that extends from Pico Quemado in the north to Tierra del Fuego in the south. At Pico Quemado, coal seams 1.0–2.0 m in thickness are high-volatile bituminous with coking properties, their high rank possibly being due to

a locally high geothermal gradient related to magmatic phenomena. In the southern part of the basin around Rio Turbio, two coal zones contain seams up to 2.0 m in thickness. They are subbituminous to bituminous with no coking properties. All the Palaeogene–Neogene coals have low sulfur contents and are suitable for the electricity generating industry and, in the case of the Pico Quemado coal, can be used in the metallurgical industry.

#### 3.4.6.2 Bolivia

Two types of coal are known from Bolivia: anthracite of Permian age and lignites of Palaeogene–Neogene age.

Anthracite is located on the Copacabana Peninsula and on the Isla del Sol, Lake Titicaca. Seams are in the form of coal lenses or very thin beds of anthracite with low sulfur content. The Palaeogene–Neogene lignites are found in the Tarija basin, where seams are thin, under 1.0 m, and have a high sulfur content (6–8%).

#### 3.4.6.3 Brazil

Brazil has five coal-bearing regions that may have potential: the Upper Amazon, the Rio Fresco, Tocantins–Araguaia, Western Piauí, and Southern Brazil. Only the Southern Brazil region is currently considered prospective.

The Amazon region contains lignites of Palaeogene–Neogene age; seams are thin (less than 1.5 m) with high ash and sulfur contents. The Rio Fresco region contains thin seams of anthracite with very high ash contents (40%); seams up to 1.7 m have been reported.

The Tocantins–Araguaia region has very thin coals of Carboniferous age and is not considered of economic importance.

The Western Piauí region also contains Carboniferous coals, which are thin and not significant.

The principal Brazilian coal deposits are situated in the Southern Brazil region. They are of Carboniferous–Permian (Gondwana) age and are exposed in a lenticular belt that runs from the states of Parana in the north

through Santa Catarina to Rio Grande do Sul in the south. The Rio Grande do Sul Coalfield contains numerous seams up to 3.0 m in thickness; the Santa Catarina region contains 10 coal seams, of which the thickest is 2.2 m. In Parana, seams are usually less than 1.0 m thick. The coals are high-volatile bituminous with high ash contents. At Candiota in Rio Grande do Sul a 5 m seam is mined; the coal has a high ash content (50%) and a sulfur content of 1%. Santa Catarina and Parana have coals with high sulfur values (3–10%), and in the Parana area the coals become low-volatile bituminous–semi-anthracite due to the intrusion of dolerite dykes into the coals. Some Santa Catarina coals have some coking properties, but the bulk of Southern Brazilian coal is mined as a thermal coal product.

Some lignites are present in the Sao Paulo region but their economic potential is unknown.

#### 3.4.6.4 Chile

There are four areas of coal-bearing sediments in Chile; these are, from north to south, the Copiapo region, the Arauco region, the Valdivia region, and the Magallanes region.

The Copiapo coals are of Mesozoic (Rhaetic) age; they are strongly folded, with seams occurring as thin lenses of anthracite with high ash and variable sulfur contents.

The Arauco region lies on the Chilean coast just south of Concepción, and the coal field extends offshore to a distance of 7 km. Dips are steep and faulting common. Seams are of Palaeogene–Neogene age and average 1.0–1.5 m in thickness; the coals are high-volatile bituminous with low ash and variable sulfur contents, and they have poor coking properties. Coals of the Valdivia region are concealed beneath younger sediments. Seams reach 3.0 m in thickness and are subbituminous with low sulfur contents. These coals are used for local purposes.

The Magallanes region forms part of a large sedimentary basin in which over 3800 m

of Late Cretaceous– Palaeogene–Neogene sediments are preserved. Coal seams up to 7.0 m are present, and upwards of 12 coal seams have been identified. All the coals are high-volatile subbituminous, non-coking with low ash and sulfur contents. This large coal deposit is geographically remote but is a large resource and may be of future importance.

#### 3.4.6.5 Colombia

Coal deposits of Mesozoic (Cretaceous) and Palaeogene–Neogene age are found in numerous localities in the northern half of Colombia. All have been highly tectonised, and coals range from lignite to anthracite. Cretaceous coals are found just north of Bogota, in the Cundinamarca–Santander region. In this area the coals are bituminous with low ash and sulfur contents, strongly coking, and are suitable for coke production. Palaeogene–Neogene coal deposits are located north of Santander on the Venezuelan border, in the extreme north of Colombia at El Cerrejon, around Cordoba on the north coast, and at Valle del Cauca in the west of Colombia. These coalfields produce non-coking, high-volatile bituminous coal with generally low ash and sulfur contents. It is ideally suited for use in the electricity generating industry.

The deposit at El Cerrejon is one of the most important in South America. Coals dip gently eastwards, and more than 40 seams are greater than 1.0 m in thickness, locally reaching 26.0 m. Because of the high quality of these coals, El Cerrejon is now a significant exporter of thermal coal.

#### 3.4.6.6 Costa Rica

Costa Rica contains deposits of Palaeogene–Neogene coals and lignites. Individual seams are up to 1.0 m in thickness. Locally, the subbituminous coals and lignites have been ameliorated by igneous intrusions. Sulfur contents range from 1.0% to 4.0%. The three principal coal deposits are at Uatsi and Zent on the south-east coast, and at Venado in the north of Costa Rica.

Areally extensive peat deposits (up to 2.0 m thick) are present in the Talamanca Cordillera and may represent a large resource for future development.

#### 3.4.6.7 Ecuador

Small lignite deposits are present in the Palaeogene–Neogene sequences of the Amazon basin, the Pacific coast, and in intermontane basins in the Andes. Only the latter are considered to be of significance. The Malacatus Basin contains seams of up to 4.0 m in thickness, disrupted by faulting. To the north the Loja Basin contains seams of up to 2.0 m in thickness and the Canar–Azuay Basin has seams of up to 5.0 m. All the coals are high-volatile subbituminous coals with high ash and sulfur contents.

#### 3.4.6.8 Mexico

Coals of Mesozoic (Cretaceous) age are found throughout Mexico; all are highly tectonised and are structurally complex. The principal coalfield is at Coahuila, close to the border with Texas, USA, where shallow, gently dipping seams reach 2.0 m in thickness. The coal is low volatile and bituminous with high ash and low sulfur contents and with no coking properties. Output is used for local industry and power generation. Another location of note is in the north-west of Mexico at Sonora where anthracites averaging 1.0 m thickness are found.

Numerous other small deposits of bituminous coal are present in Mexico. Seam development is irregular and often ameliorated by volcanic activity.

#### 3.4.6.9 Peru

Mesozoic coals are located within the Andean Cordillera, which extends throughout Peru from north to south.

The northern coalfields are highly tectonised and affected by associated igneous activity, resulting in the formation of anthracite as well as bituminous coal. The principal areas are those of Alto Chicama and Santa. Subbituminous and bituminous coals are found in the

southern coalfields of Oyon and Jatunhuasi. All production is for local needs.

#### 3.4.6.10 Uruguay

The north-east of Uruguay contains Carboniferous–Permian (Gondwana) sediments that represent the southern extension of the Southern Brazilian coalfields. Coals are found in this area, but no development has yet occurred.

#### 3.4.6.11 Venezuela

All the known coal-bearing sequences in Venezuela are Palaeogene–Neogene in age and occur in a series of basins across the country north of the Orinoco River. The principal areas of interest are Zulia, Lobatera, and the Caracas–Barcelona Basin. Other coal occurrences are known in the Lara region and within the eastern Orinoco Basin.

The Zulia deposit is the most important so far identified in Venezuela and is situated in the extreme north-west of the country. Between 25 and 30 seams with thicknesses of between 0.5 and 15.0 m are present. The coal is high-volatile, non-coking bituminous with variable ash and low sulfur contents, suitable as a steam coal for export.

The Lobatera Coalfield is in the west of Venezuela, close to the Columbian border. Here, 35 seams over 0.3 m thick are present. The coal is high volatile with low ash and sulfur contents.

The Caracas–Barcelona Coalfield contains the deposits of Naricual and Fila Maestra. Naricual contains 15 seams, ranging from 1.0 to 10.0 m in thickness, of high-volatile bituminous coal with a low sulfur content; some of these seams have coking properties. The deposit at Fila Maestra is currently being investigated.

In the Lara region, thin lenticular seams of low-volatile bituminous coals occur with low sulfur content. In the eastern Orinoco Basin, seams of lignite occur of up to 1.2 m thick, with high sulfur contents. These have not been considered significant.

### 3.4.7 Commonwealth of Independent States

The CIS has vast reserves of all ranks of coal stretching across the whole of the region (Figure 3.13). Thick coal-bearing sequences range from Palaeozoic (Carboniferous–Permian), Mesozoic (Triassic, Jurassic, and Cretaceous), to Cenozoic in age. These are preserved in a series of large sedimentary basins, which generally become younger from west to east. Most of the older basins are structurally disturbed, resulting in steeply dipping seams and extensive faulting. The potential for production is enormous; however, geographical position, severe climatic conditions, and poor infrastructure may curtail the development of many of these deposits.

#### 3.4.7.1 Kazakhstan

In Kazakhstan, the greater part of measured reserves consists of bituminous coal found in the Karaganda, Ekibastuz, and Teniz-Korzhanokol basins: the Kushokinsk, Borly, Shubarkol, and Karazhyr deposits. Lignite is found mainly in the Turgay, Nizhne-Illyskiy, and Maikuben basins. The Karaganda Basin contains a thick sequence of carboniferous sediments; numerous coal seams are present, varying from <1 to 3.5 m in thickness. The seams range from high-volatile bituminous to anthracite, with high ash and medium sulfur contents. The lower seams have good coking properties. The majority of mines are underground operations.

The Ekibastuz Basin contains the same Carboniferous sequence, and the basin is fault bounded. In this area, a number of coal seams have coalesced to form a single seam 130–200 m thick. The coals are mined in open-cast operations and supply coal to the power generation sector.

#### 3.4.7.2 Russian Federation

Russia has very large coal reserves, which are widely dispersed and occur in a number of major basins. These range from the Moscow Basin in the far west to the eastern end of the



Donetsk Basin in the south, the Pechora Basin in the far north-east of European Russia, and the Irkutsk, Kuznetsk, Kamsk–Achinsk, Lena, South Yakutia, and Tunguska basins extending across Siberia and the Far East. Current black coal production has centred on the Kuznetsk and Pechora basins and the Russian part of the Donetsk Basin, but coalfields in Eastern Siberia and the Russian Far East are still largely unexploited.

The Kuznetsk Basin is structurally complex, and the Carboniferous to Jurassic sequence is 7000–8000 m thick. Around 90% of the coal seams (c. 300) are found in the Permian, of which 130 are workable with average thicknesses of 1–35 m. Coals range from subbituminous to semi-anthracite. The ash content is variable, and sulfur content is generally low. Coal is produced from both surface and underground mines. The Pechora Basin also contains Permian coal-bearing sediments that are intensely folded. Up to 5000 m of sediments contain 20–30 workable coal seams of 3–20 m in thickness. Coal rank increases from west to east and with depth. The high-volatile bituminous coals have variable ash contents (10–40%), and sulfur is usually less than 1.5%, with occasional high-sulfur coals up to 4%. Many coals are semi-coking.

In the Donetsk Basin, coal rank increases towards the central and eastern parts, and ranges from subbituminous up to anthracite. The greater part of this basin is located in the Ukraine.

The Moscow Basin contains thin seams with difficult mining conditions. This area has traditionally been a large coal producer, but in recent years has declined and so that little mining now takes place.

In Eastern Siberia, the Kamsk–Achinsk and South Yakutsk basins have coal deposits of Jurassic age. The coals are subbituminous with thicknesses of 40–70 m. Above these occur a number of Lower Cretaceous coals. These basins have simple structure. They supply local power stations and chemical plants, the principal mining area being Neryungri.

### 3.4.7.3 Tajikistan

Tajikistan has important coal deposits at Shurob and Fon-Yaghnob and is beginning to exploit the bituminous coals of East Zidi by open-pit methods and the anthracite coal at Nazarailok by underground mining.

### 3.4.7.4 Ukraine

As for Russia, the Donetsk Basin contains a thick succession of Carboniferous sediments in which numerous coal seams are present, ranging from 0.5 to 2.5 m in thickness. The rank ranges from subbituminous to anthracite; all have variable ash contents and high sulfur contents (average 2–3%). Some seams have good coking properties. The large Dnieper Basin produces lignite for the local power stations.

### 3.4.7.5 Uzbekistan

Two coal fields are currently exploited: the Angren brown coal field in the Tashkent region using open-pit mining, and the Shargun anthracite deposit in the Surkhandarya region, together with some bituminous coal produced from the Baisun field in south Surkhandarya. 85% of lignite production supplies the power generating sector; bituminous coal output remains on a small scale.

## 3.4.8 Far East

The Far East region contains 13 countries with known coal deposits (Figure 3.14). By far the largest of these is the PRC, which has vast resources of all ranks of coal.

The coals of the Far East range in age from Palaeozoic to Cenozoic, and all ranks of coal are present.

### 3.4.8.1 Brunei

Coals in Brunei are Palaeogene–Neogene in age and occur in the north-east of the country close to the capital Bandar Seri Bagawan, and also in the headwaters of the Belait River in the south-west of Brunei. Coal seams are 0.5–5.0 m in thickness and are high-volatile bituminous

with low ash and variable sulfur content. Those seams close to Bandar Seri Bagawan have been extensively worked in the past, whereas those in the Belait River basin are undeveloped but are geographically remote.

#### 3.4.8.2 Democratic Republic of (North) Korea

Coals of Palaeozoic, Mesozoic, and Cenozoic age are present throughout the Korean peninsula.

The principal Palaeozoic coalfields are Pyongyang and North Pyongyang in the north-west and Kowon-Muchon in the east. All have been highly tectonised. Consequently, seam thicknesses are variable due to intense folding; however, thicknesses of 5.0 and 15.0 m are reached. The coals are low-volatile anthracites with low ash and sulfur contents. All the coal is mined by underground methods and is used for local industry and domestic heating.

Mesozoic coals form small deposits of anthracite; these are also strongly folded, but to a lesser extent than the Palaeozoic coals.

The Cenozoic coalfields contain subbituminous coal and lignite and are found chiefly in the north-east of the country. The Kyongsang and Kilchu-Myongchon coalfields contain lignites, and the Tumangang in the extreme north-east contains subbituminous coal that is used for electricity generation. The Anju Coalfield is located north of Pyongyang and is a large deposit of subbituminous coal that has been developed as an opencast operation.

#### 3.4.8.3 Indonesia

Indonesian coal deposits are Palaeogene-Neogene in age and are situated on the islands of Sumatra, Borneo, Java, Sulawesi, and Irian Jaya. There is a range in rank from lignite to low-volatile bituminous, the higher rank coals being affected by local igneous intrusions or more importantly by regional heating due to magmatic activity at relatively shallow depths.

On the island of Sumatra, a number of coalfield areas are currently exploited.

At Bukit Asam, at the south-eastern end of the island, seams up to 12.0 m in thickness are present. Coals are generally subbituminous with low ash and sulfur contents, but some bituminous coal is present in close proximity to igneous intrusions. The coals are mined by opencast methods and used primarily for electricity generation.

Ombilin, located in central Sumatra, has a few thick seams of high-volatile bituminous and subbituminous coal with low ash and sulfur contents. Mining has been by both opencast and underground methods, and the coal is used for electricity generation and cement manufacture.

In the Bengkulu region on the south-west coast of Sumatra, small occurrences of mostly subbituminous coals with low sulfur content are mined by opencast methods.

On the island of Borneo, the Indonesian territory of Kalimantan has coal deposits situated along the east coast. In East Kalimantan, subbituminous and bituminous coals are found, notably in the Sangatta and Berau areas. These coals are up to 10.0 m in thickness with extremely low ash and low sulfur contents. Some of these coals are now exported as prime-quality steam coals. In South Kalimantan, in the Senakin, Tanah Grogot, and Tanjung areas, subbituminous and bituminous coals with similar characteristics are mined both for export and for local power generation needs. In the north-eastern part of Kalimantan, north of Berau, bituminous coals are present at Tarakan, but high sulfur contents have halted the development of these deposits.

In Java, subbituminous coals have been worked on a very small scale in central and western parts. These coals are thin and irregularly developed.

In South Sulawesi, similar subbituminous coals are present that have been mined for local needs.

In West Irian, the western half of the island of New Guinea, subbituminous coals and lignites are present in the Bintuni region at the western

end of the island, but they have not yet been developed.

Large deposits of Recent peat are present in West Kalimantan and have been investigated for commercial development.

#### 3.4.8.4 Japan

Japanese coal deposits are widespread and range from Permian to Palaeogene–Neogene in age. The productive coals are Palaeogene–Neogene, whereas the Permian and Mesozoic coals are of minor importance, except for the Omine Coalfield in western Honshu.

The principal Palaeogene–Neogene coalfields are located on the three Japanese islands of Hokkaido, Honshu, and Kyushu.

On Hokkaido Island, the structurally complex area of the Ishikari Coalfield provides strongly caking bituminous coal with high ash and low sulfur content. The coals are produced for local use. The Kushiro Coalfield is less disturbed and produces non-coking bituminous and subbituminous coal.

On the island of Honshu, the Joban Coalfield has seams up to 3.0 m in thickness and is thought to extend eastwards offshore. The Omine Coalfield, on the south-west coast, is important as a source of anthracite for Japanese industry. The Mogami, Nishitagawa, and Miyagi coalfields are situated in the northern half of the island and are the chief lignite producers in Japan.

On Kyushu Island, the Miike Coalfield is structurally undisturbed and is mined offshore. The coal is bituminous with good coking properties. The Chikuho Coalfield has similar coals to Miike and is a source of coking coal for the metallurgical industry.

Numerous smaller coalfields containing bituminous coals and lignites are worked on a small scale.

#### 3.4.8.5 Laos

Palaeozoic, Mesozoic, and Cenozoic coals are present in Laos. The Palaeozoic deposits are chiefly anthracite with a high ash content. There are three principal occurrences:

Phongsaly in the north, the Ventiane coal basin in west-central Laos, and the Saravan coal basin in the south of the country. In the Ventiane basin, five seams ranging from 2.6 to 6.0 m are present; in the other areas, the seams are considerably thinner.

Some Mesozoic (Triassic–Jurassic) coals are found in the Phongsaly region; all are steeply dipping, and seams range in thickness from 0.1 to 10.0 m. The coals are high-volatile bituminous with low ash and low sulfur contents.

Cenozoic brown coals are present in several Palaeogene–Neogene basins located in the east of the country, chiefly at Muongphan, with other occurrences at Khang Phanieng, Hua Xieng, and Bam O. These Palaeogene–Neogene basins are highly faulted and contain subbituminous coals and lignite. At Muongphan, lignite seams are 1.0–6.0 m in thickness and have high volatile and ash contents.

All Laotian coal produced is used for local needs.

#### 3.4.8.6 Malaysia

Malaysian coals are found on the west coast of the West Malaysian peninsula, and on the East Malaysian side of the island of Borneo in the states of Sabah and Sarawak.

All the coals are of Palaeogene–Neogene age. Those in Sabah are subbituminous with some coking properties, but often with high sulfur contents; these have been mined at Silimpon in east Sabah. In Sarawak, higher quality bituminous and subbituminous coals with low sulfur contents have been identified at Bintulu, Balingian, and Silantek, and mined on a local scale.

In West Malaysia, at Bukit Arang on the Malaysian–Thailand border, extensive lignite deposits have been identified; another occurrence of lignite is reported north of Kuala Lumpur at Batu Arang.

#### 3.4.8.7 Mongolia

Coal deposits in Mongolia are concentrated in the north of the country. Highly tectonised Palaeozoic coals in the form of anthracite and

low-volatile bituminous are found in small isolated deposits.

Mesozoic (Cretaceous) coals are less deformed and consist of low-volatile bituminous coal with low sulfur contents, found principally in the Baganuur Coalfield, where seam thicknesses can be up to 25.0 m. In the same region occur the coalfields of Sharin Gol and Nalayh, and in the west of the country are the coalfields of Aчит Nuur and Khartarbagat. At Ovoot Tolgoi, Late Permian coals are mined that produce high-grade coking coal. In southern Mongolia, at Tavan Tolgoi, is a large deposit of high-quality bituminous coal. The Permian coal-bearing sequence includes 16 coal seams ranging in thickness from 2 to 72 m. Development of road and rail links to the PRC will enable coal production and exports to increase.

#### 3.4.8.8 Myanmar (Burma)

The coal deposits of Burma consist of scattered occurrences of Palaeogene–Neogene lignite, with some Mesozoic black coals, which have been highly tectonised.

Lignites are found in the western and southern parts of the country, notably at Kalewa and Pakokku. The black coals are situated inland in the east-central region of Burma, in the Panlaung and Henzada districts. The coals are reported to be of poor quality and are only worked on a very small scale.

#### 3.4.8.9 People's Republic of China

The PRC is the world's largest coal producer. Three-quarters of all proven recoverable coal reserves occur in the northern half of the PRC; of these, two-thirds are present in the provinces of Inner Mongolia, Shanxi, and Shaanxi. These areas provide the bulk of the coal for export and power generation. The majority of mines are underground operations and range from 70 years old to new (Thomas and Frankland 1999).

China has extensive black coal deposits of Carboniferous, Permian, Triassic, Jurassic, and Early Cretaceous age, plus lignite reserves

of Palaeogene–Neogene age. Carboniferous and Permian coals are found throughout eastern PRC, and Triassic coals are located in south-east PRC. Coals of Jurassic and Early Cretaceous age are located in Inner Mongolia and north-eastern PRC.

The Shenmu–Dengfeng Coalfield is located on the Inner Mongolia–Shaanxi border and contains structurally undisturbed coals up to 10 m in thickness. The coal is high-volatile bituminous with low sulfur content (0.4%) and supplies power stations both domestically and for export. Although mining is principally by underground methods, the Haerwusu opencast mine is one of the largest in the PRC.

In north-west China, the Xinjiang–Uyghur region contains large reserves of coal; up to 13 coal seams with a total thickness of 175 m are reported. These have yet to be exploited.

The Datong Coalfield in northern Shanxi Province is also relatively undisturbed structurally. It produces medium-volatile bituminous coal with <1% sulfur, again for both domestic use and for export. The Lu'an coalmining area in south Shanxi Province produces both steam and coking coal for home and export markets.

Shaanxi Province contains five major coalfields. Of these, the southern Huang Ling and Tongchuan coalmining districts have a number of large underground mines in operation. All are producing high- and medium-volatile bituminous coal with sulfur contents of around 1%.

In Henan Province, the Hebi Coalfield, situated in the north of the province, is the largest producer. Together with the Gaocheng coalmines in the south, it produce a coal range of high-volatile bituminous to anthracite coal.

Anhui Province in the east has large coal deposits, all exploited by underground mines. The coalfield areas are structurally complex, seam thicknesses are up to 6 m, and the coals are bituminous with low sulfur contents. Some coals have coking properties.

To the north of Anhui, Shandong Province is an important producer of export-quality coals.

Coals are bituminous, low in ash and sulfur, and are amongst the best coking coals in the PRC. Seams range in thickness from 1 to 10 m.

In the north-east of the PRC, the Liaoning region has numerous coal seams up to 100 m in thickness, with little structural disturbance. Coals are high-volatile bituminous with low ash and sulfur and with good coking properties. The Heilongjiang and Jilin coalfields in the far north-east also have thick seams of similar quality.

In Guizhou Province, in southern PRC, the Pangjiang Coalfield produces medium- and low-volatile bituminous coal for both power generation and steel production. The coalfield is structurally complex, and anthracite occurs in the more intensely tectonised areas.

To the south of Guizhou, in Yunnan Province, low-volatile bituminous coking coals with low sulfur contents are produced.

There are numerous other coal deposits in the PRC, mostly close in location to those listed. The PRC's potential for coal production is enormous but depends heavily on underground operations and has poor infrastructure in some areas.

#### 3.4.8.10 People's Republic of Vietnam

In Vietnam, black coals are of Mesozoic (Triassic) age and are located, first, in a broad belt running east to west, situated north and north-east of Hanoi. This belt consists of four sedimentary basins, each containing coal-bearing strata; these are the Nan Meo, Phan Me, Bo Ha, and Quang Yen basins. Second, they are found in central Vietnam at Nong San and Huong Khe.

The Nan Meo, Phan Me, and Bo Ha basins contain low-volatile bituminous coals, some with coking properties. These areas were worked on a small scale in the past.

The most important coal basin is the Quang Yen Basin, the eastern part of which borders the north-east coast. Coals are preserved in a series of folds orientated parallel to the coast and are bounded by large east-west-running faults.

In the east part of the Quang Yen Basin, the Hong Gai Coalfield is the chief coal producer in Vietnam; up to six seams with thicknesses 2.0–8.0 m are worked. Coals are low-volatile anthracites with low ash and sulfur contents.

In central Vietnam, the Nong Son area has a thick seam up to 20.0 m in thickness and is low-volatile bituminous to semi-anthracite with a variable sulfur content. The Huong Khe area is believed to contain several seams of anthracite.

#### 3.4.8.11 Philippines

Throughout the Philippines archipelago are situated a series of Palaeogene–Neogene basins containing coal-bearing sediments. The coals are predominantly of subbituminous rank, although variations in rank do occur related to local structure and contemporaneous and recent igneous activity.

The northern island of Luzon contains the Cagayan Basin; this area is only partially explored, but it is known to contain seams up to 2.0 m in thickness and is structurally undisturbed. The coals are high-volatile subbituminous with low ash and sulfur. The deposit covers a large area and is amenable to open-cast mining operations; such coals would be suitable for local electricity generation.

The island of Mindoro has coal deposits in the south; the seams are up to 2.8 m in thickness and are subbituminous with variable sulfur contents.

Semirara Island lies to the south of Mindoro and contains coals up to 6.0 and 12.0 m in thickness. The coals are subbituminous with low ash and sulfur.

Catanduanes Island contains lenticular seams up to 5.0 m in thickness; these are steeply dipping and are ameliorated by igneous intrusions. This has resulted in the formation of bituminous coals with high sulfur and moderate coking properties.

Cebu Island contains several coal deposits. Seams are up to 4.0 m in thickness, dip steeply, and are high-volatile subbituminous coals with low ash and variable sulfur contents.



Mindanao Island has coal deposits at Malangas and Zamboanga in the west and at Gigaquit and Bislig in the east.

The Malangas-Zamboanga area has ameliorated coals, anthracite, and bituminous coking coal. At Gigaquit, low-rank coals with high ash contents are characteristic; and at Bislig, some bituminous coal with locally high sulfur content is mined.

Numerous other small coal deposits are worked locally throughout the archipelago.

Small-scale underground mining characterises the bulk of the coal exploration in the Philippines; however, those deposits at Cagayan, Semirara, and Zamboanga could be developed further.

#### 3.4.8.12 Republic of (South) Korea

Coals in South Korea are of similar age and character to those in the north of the peninsula. All of the mining operations were underground, but due to difficult mining conditions and unsuitable coal quality, little mining is currently in operation.

The principal Palaeozoic coalfields are Samcheog, Jeongseon, Kangnung, Danyang, and Mungyeong. These coalfields are highly tectonised and intensely folded, with seam thicknesses varying considerably due to the squeezing of the coals; 1.0–2.0 m is usual. All the coal is anthracite with a low sulfur content and is exclusively used for local industry and domestic heating.

Mesozoic (Jurassic) coal deposits are present at Mungyeong and Chungnam. The latter is structurally complex; again, all the coal is anthracite.

Small anthracite deposits are found at Boeun and Honam in the south and at Kimpo and Yeongcheon on the northern border of the country; small workings produce anthracite for local use.

Palaeogene–Neogene deposits containing thin seams of lignite are found in small areas bordering the south-east coast of South Korea.

#### 3.4.8.13 Taiwan

Coals in Taiwan are Palaeogene–Neogene in age, and the coalfields are grouped into a northern and a central province; of these, the northern province only has economic significance. The Taiwan coalfields have been highly tectonised, and some have been ameliorated by igneous intrusions. The coals are high-volatile bituminous and subbituminous with low ash and sulfur contents. At Chilung, in the north of Taiwan, ameliorated coals (semi-anthracites) have been mined in small areas. Further south, low-volatile bituminous coals, low in sulfur and with good coking properties, have been mined at Hsinchu, Nanchuang, Shuangchi, and Mushan. Four seams exceed 1.0 m in thickness. Because of the high level of tectonic disturbance, there are only underground operations working at increasingly deeper levels. This will eventually result in the cessation of mining in these areas.

#### 3.4.8.14 Thailand

In Thailand, virtually all the known coal deposits are of Palaeogene–Neogene age, together with some Mesozoic coals found in the north-east of Thailand at Na Duang. The Palaeogene–Neogene sediments are preserved in a series of basins; of these, the Mae Moh Basin in north-west Thailand is the most extensive. Other basins in close proximity are Mae Tip, Li, Mae Tun, and Vaeng Haeng. Other Palaeogene–Neogene coals are found east of Bangkok at Nong Ya Plong and at Krabi in the extreme south-west.

Seams in Thailand generally range from 2.0 to 12.0 m; however, at Mae Moh and Krabi, seams up to 30.0 m are worked. Most coals are relatively undisturbed structurally. These Palaeogene–Neogene coals range from lignite to high-volatile subbituminous, with generally low sulfur contents, as found at Mae Moh, Mae Tip, and Li; and some with higher rank, high-volatile bituminous are found at Mae Tun and Nong Ya Plong.

The Mesozoic coal at Na Duang is semi-anthracite with low sulfur content.

The bulk of Thailand's coals are mined and supplied to the electricity generating and cement manufacturing industries.

### 3.4.9 Australasia

Australasia is one of the major coal producers in the world. The bulk of the coal resources are located in the eastern part of Australia, with smaller coal deposits in Western Australia and New Zealand (Figure 3.15).

#### 3.4.9.1 Australia

Australia contains coals of Palaeozoic, Mesozoic, and Cenozoic age. The whole of the black coal resources are of Palaeozoic age and are located in Western Australia, Queensland, and New South Wales. Mesozoic coal is present in Queensland, and an important deposit of Cenozoic coal is found in Victoria. The black coals of Queensland and New South Wales are both steam and coking coals, and the bulk of production is for export. In the other coal-producing areas of Australia, coal is primarily used for domestic power generation.

The Palaeozoic coals of Australia are Permian (Gondwana) in age and have been generated in a series of basins. The principal ones are the Bowen, Galilee, and Cooper basins in Queensland, the Sydney Basin in New South Wales, and the Collie and Fitzroy basins in Western Australia. Other smaller areas are known from South Australia and Tasmania.

In Queensland, the Bowen Basin has been explored extensively. The eastern side of the basin has subsided more rapidly than the west and has received more sediments, and the lack of structural disturbance has resulted in the preservation of shallow flat-lying coals. The oldest coals are low-ash and low-sulfur seams that reach thicknesses of up to 30.0 m. The uppermost Permian contains four workable seams. Seam splitting is common, and igneous intrusions have locally affected the coals in the west. The topmost coal-bearing sediments are the most widespread, with 12 coal seams having thicknesses up to 4 m. The coal is

high-volatile bituminous, with variable ash and low sulfur content; the coals have good coking properties. The coal is worked by large opencast operations, and large reserves have been identified. In the other coal-bearing areas of Queensland (e.g. the Galilee and Cooper basins), large reserves of coal have yet to be developed.

In New South Wales, the Sydney Basin is the most important coal-producing area in Australia. Again, as in the Bowen Basin, there is little structural disturbance. Two Permian formations contain coals that are exposed over large areas. The lower part contains up to six seams and is heavily faulted and intruded. The upper formations have 14–40 coal seams, many of which exhibit splitting; in the west, these are severely affected by igneous intrusions that, although producing an increase in rank, have destroyed large reserves of coal. Seam thicknesses reach 10.0 m, and the coal produced is high-volatile bituminous with variable ash and low sulfur contents; some of the coals have good coking properties. Coals are mined by underground and opencast methods, with the principal mining districts being the Western District, the Burragorang Valley, the Hunter Valley, and the Southern District. Much of the coal is exported as steam and coking coal, as well as supplying local needs.

In Western Australia, the Collie basin contains seams ranging from 1.5 to 11.2 m; these are structurally undisturbed and are mined in the Cardiff and Muja areas. The coals are subbituminous with low ash and sulfur contents. Coals have been located in northern Western Australia in the Fitzroy Basin, and brown coal deposits have been identified close to the south coast. In South Australia, the Ackaringa Basin is currently being explored, and in Tasmania some development of the coal deposits may occur in the future.

The Mesozoic coals of southeast Queensland, in the Brisbane area, are subbituminous coals and have not been extensively developed.

The Palaeogene–Neogene coals of the Gippsland Basin in Victoria are thick developments

of lignite; the principal seams reach enormous thicknesses of 300.0 m. The seams are shallow and flat-lying and are high-volatile lignite with low ash and low sulfur contents. This basin is worked in a number of separate coalfields, the most important of which is the Latrobe Valley. This lignite is used exclusively for the Victorian electricity industry.

#### 3.4.9.2 New Zealand

With the exception of a few thin uneconomic coals of Jurassic age, all significant New Zealand coals are Cretaceous–Palaeogene–Neogene in age. The coalfields are located in the western part of North Island, and in the north-western and south-eastern districts of South Island.

In North Island, the Waikato Coal region contains New Zealand's major subbituminous coal resource. Coal seams are discontinuous but are thick locally, up to 30.0 m. They are subbituminous with low ash and low sulfur contents. To the north, the Northland area has a few seams up to 2.0 m thick and these are of poorer quality.

To the south of the Waikato Coal region lies the Taranaki Coal area, where seams are usually less than 3.0 m in thickness and are subbituminous with higher sulfur contents.

In South Island are located the Cretaceous–Palaeogene–Neogene coalfields of Otago, Southland, Ohai, and Kaitangata in the far

south, and on the north-west coast the Westland Coal region includes the Greymouth, Pike River, Charleston, Buller, Garvey Creek, and Collingwood coalfields.

The southern coalfields have seams up to 6.0–10.0 m, although they are discontinuous and lensoid. The coals vary from lignite to high-volatile subbituminous with variable sulfur contents, some of which may be as high as 6%.

In the Westland Coal region, the coals range from subbituminous at Charleston to high-volatile bituminous at Pike River and Collingwood. Hard coking coal is mined at Greymouth, and low-ash semi-soft coking coal at Reefton in the Garvey Creek coalfield. In the Buller and Greymouth coalfields the coals range from high- to low-volatile bituminous.

The Southland lignite deposit makes up 80% of New Zealand's coal resources.

Production from the New Zealand coalfields is small at the present time.

#### 3.4.9.3 Antarctica

Cretaceous coals of mixed quality have been recorded from James Ross Island, on the south-east flank of the Weddell Sea. Other occurrences have been in the area of the Transantarctic Mountains.

Present legislation will prohibit any development of these possible resources for many years to come.

## 4

### Coal as a Substance

#### 4.1 Physical Description of Coal

Coal has been defined by numerous authors. Essentially, it is a sediment, organoclastic in nature, composed of lithified plant remains, which has the important distinction of being a combustible material. The composition and character of each coal will be determined, first, by the nature of the make-up of the original organic and inorganic accumulation and, second, by the degree of diagenesis it has undergone.

The inherent constituents of any coal can be divided into ‘macerals’ (i.e. the organic equivalent of minerals) and ‘mineral matter’ (i.e. the inorganic fraction, made up of a variety of primary and secondary minerals). Note that the latter is sometimes erroneously referred to as ‘ash’ when in fact ‘ash’ is the mineral residue remaining after combustion of the coal. The composition and ratio of the two fractions reflects the make-up of the original material and indicates the coal **type**.

The degree of diagenesis or coalification that a coal has undergone by burial and tectonic effect determines the coal **rank**.

The term **brown coal** is used for low-rank coals, such as lignite and subbituminous coal, and **black** or **hard** coal is used for coals of higher rank; that is, the bituminous, semi-anthracite, and anthracite coals.

The majority of coals are composed of discrete layers of organic material. Such layers may possess different physical and chemical properties. It is the relative proportions and petrological characteristics of these layers that determine the character of the coal as a whole, and its usefulness as a mined product.

Coals are divisible into two main groups: the **humic** coals and the **sapropelic** coals.

**Humic** coals are composed of a diversified mixture of macroscopic plant debris, with the coals typically having a banded appearance. **Sapropelic** coals are composed of a restricted variety of microscopic plant debris; such coals have a homogeneous appearance.

##### 4.1.1 Macroscopic Description of Coal

###### 4.1.1.1 Humic Coals

The use of a simple but distinctive system of description is fundamental to field examination of coals. Several systems to describe the physical character of coal have been proposed and are briefly outlined in the following.

The term lithotype is applied to the different macroscopically identifiable layers in coal seams.

Stopes (1919) proposed four lithological types (lithotypes) for describing humic coals.

- (i) **Vitrain** is black, glassy, vitreous material with a bright lustre, occurring as thin bands and is brittle. Vitrain breaks into

fine angular fragments and is commonly concentrated in the fine fraction of mined coal. Vitrain is found in most humic coals and usually consists of the microlithotype vitrite with some vitrinite-rich clarite.

- (ii) **Clarain** is bright with a silky lustre between vitrain and durain, and occurs in fine laminations. Clarain comprises alternating thin layers, often <1 mm. It can include the microlithotypes vitrite, clarite, durite, fusite, and trimacerite.
- (iii) **Durain** is grey to black with a dull lustre, fracturing into rough-surfaced fragments. Only lenses thicker than 3–10 mm are referred to as durain. Durain is less common than vitrain and clarain in humic coals, but it can occur as extensive layers within a coal seam. Durain is composed of the microlithotypes durite and trimacerite.
- (iv) **Fusain** is black, soft, friable, and easily disintegrates into a black fibrous powder. Fusain occurs in coals as lenses, usually several millimetres thick, often concentrating in discrete layers in the coal. In most coals, fusain is a minor lithotype composed of the microlithotype fusite.

However, difficulties have arisen in using these terms to describe coals in borehole cores and in exposures. The four lithotypes

often occur as thin layers or lenses, often only millimetres in thickness. Strict usage of Stopes' terms would lead to extremely detailed lithological descriptions, whereas in practice only a limited amount of lithologically distinct units are required. For practical purposes, various sources have proposed alternative terminology that, although essentially retaining the basic classification of Stopes, has a more descriptive lithological bias. The principal types of humic and sapropelic coals, according to McCabe (1984), are summarised in Table 4.1.

In the USA, Schopf (1960) introduced the term **attrital** coal to include all coal not precisely defined as vitrain or fusain and which can be subdivided into five levels of lustre ranging from bright to dull. The Australian system is broadly similar in approach, but more descriptive in terminology (Table 4.2). The Australian coal industry defines vitrain and fusain as bright and dull coal respectively, and the five categories of attrital coal are graded according to major and minor constituents of each end member. This is very much a physical description and is eminently more suitable for field recording of coals.

In addition, coals with high mineral content contained in discrete bands or as nodules or veins can be best described as **impure** coals. Commonly, such mineral matter is in the form of pyrite, calcite, siderite, ankerite, or as clay

**Table 4.1** Lithotypes of humic and sapropelic coals.

Lithotype	Description	Composition
Vitrain	Black, very bright lustre; thin layers break cubically; thick layers have conchoidal fracture	Vitrinite macerals with <20% exinite macerals
Clarain	Finely stratified layers of vitrain, durain, and, in some instances, fusain, medium lustre	Variable
Durain	Black or grey, dull, rough fracture surfaces	Mainly inertinite and exinite macerals
Fusain	Black, silky lustre, friable, and soft	Mainly fusinite
Cannel coal	Black, dull, lustre 'greasy', breaks with conchoidal fracture	Fine maceral particles usually dominated by sporinite
Boghead coal	Black or brown, dull, homogeneous, breaks with conchoidal fracture, lustre may be 'greasy'	Dominated by alginite

Source: From McCabe (1984).



**Table 4.2** Macroscopic description of coals in sections and boreholes.

Stopes (1919)	Schopf (1960); ASTM D2796 (1994)	Australian standard K183-1970
<b>Banded (humic) coals</b>		
Vitrain	Vitrain	Coal, bright
Clarain	Bright	Coal, bright, dull bands
	Moderately bright	Coal, dull, and bright
Durain	Attrital mid-lustre coal	Coal, mainly dull with numerous bright bands
	Moderately dull	
	Dull	Coal, dull minor bright bands
Fusain	Fusain	Coal, dull
<b>Non-banded (sapropelic) coals</b>		
Cannel coal	Cannel coal	Coal, dull conchoidal (canneloid)
Boghead coal	Boghead coal	
Impure coals	Bone coal	Coal, stony (or shaly)
	Mineralised coal	Coal, heat altered
		Coal, weathered

Source: From Ward (1984).

coatings and infillings. In the USA, coal that contains clay disseminated throughout the coal rather than in layers is termed **bone** coal and is of dull appearance.

In coals of lower rank, that is **brown** coals, the aforementioned lithological descriptions are difficult to apply. Brown coals range from **lignite**, which may be anything from soft, dull brown to black in colour, to **subbituminous** coal, which is black, hard, and banded. Brown coals are usually described in terms of colour and texture; for example, they crack and disintegrate when dried out.

Hagemann (1978, 1980) adopted a macroscopic description of lignites and applied it to Saskatchewan lignites and lignite-subbituminous coals. The important criteria in Hagemann's descriptions are the relative proportions of groundmass and woody (xylic) remains plus the relative abundance of mineral impurities, and the texture or banded characteristics. The groundmass comprises the more finely comminuted particles of varied origin too small to be identified macroscopically. In

addition, intensity and hue of colour, degree of gellification, and presence or absence of inclusions are all incorporated into the system shown in Table 4.3.

Following the work of Hagemann, the International Committee for Coal and Organic Petrology (ICCP) in 1993 adopted the classification for soft brown coals shown in Table 4.4 (Taylor et al. 1998). The classification recognises lithotype groups, lithotypes, and lithotype varieties.

The structure and constituents of the lithotype of soft brown coal can be recognised with the naked eye, and lithotypes can be distinguished by their degree of gellification and colour. The ICCP classification recognises four coal types, as described by Taylor et al. (1998).

1. **Matrix coal** consists of a fine detrital groundmass, yellow to dark brown in colour. Plant fragments may be embedded in the groundmass, and matrix coal may be homogeneous in appearance or show some stratification. The homogeneous matrix coals may have originated from peats found

**Table 4.3** Macroscopic description of lignites.

Field observations		Laboratory observations			Additional features
Structure	Texture	Colour	Gellification	Inclusions	
1. Pure coal (non-xylitic)	Unbanded coal	Pale yellow	Gellified groundmass	Resin bodies	Cracking to non-cracking
2. Pure coal (xylitic); fibrous/brittle; tree stumps, trunks, etc.	Moderately banded coal	Medium light yellow	Gellified tissues	Cuticles	Fracture even
3. Impure coal (non-xylitic) clayey/sandy calcareous coal, iron sulfides, etc.	Banded coal	Pale brown	Microgranular humic gel particles	Charcoal	Size break-up coarse to fine
4. Impure coal (xylitic)	Highly banded coal	Medium brown/dark brown/black	—	—	

Source: From Bustin et al. (1983), based on Hagemann (1980), by permission of Geological Association of Canada.

**Table 4.4** Lithotype classification for soft brown coals.

Lithotype group (constituent elements)	Lithotype (structure)	Lithotype variety (colour; gellification)
Matrix coal	Stratified coal	Brown (weakly gellified) coal
		Black (gellified) coal
	Unstratified coal	Yellow (ungellified) coal
		Brown (weakly gellified) coal
Xylite-rich coal		Black (gellified) coal
Charcoal-rich coal		
Mineral-rich coal		

Source: From Taylor et al. 1998, after International Committee for Coal and Organic Petrology (1993).

in low-lying mires or from decomposition of swamp forest peats, whereas banded matrix coals are considered to be the product of an open-swamp environment. Matrix coals are common in Palaeogene–Neogene soft brown coals.

2. **Xylite-rich coal** includes coals in which xylite (woody tissue) comprises more than 10% of the coal. The groundmass is detrital

and may or may not be stratified. Xylite occurs as fibrous tissue and may be mineralised. Inclusions of charcoal or gellified nodules may be present.

Xylite-rich coal occurs in all brown coals and is the dominant lithotype. Its characteristics are thought to be the decomposition of trees and shrubs in the peat-forming mire.

3. **Charcoal-rich coal** contains >10% charcoal. The coal can be weakly or strongly stratified, occurring as lenses and occasional more persistent layers. The coal is brownish black and has a coke-like appearance. It is a minor constituent of soft brown coals. Charcoal-rich coals are considered to be the product of burned forest swamps. Where such coal is stratified, it is indicative of water- or wind-transported residues in an open-swamp environment.
4. **Mineral-rich coal** includes all kinds of mineralisation of the different brown coal lithotype groups and should be visible to the naked eye. The inorganic materials present typically include quartz, clay, carbonates and sulfides, and other minerals.

In Australia, the State Electricity Commission of Victoria has used a classification of brown coal based on colour and texture. Table 4.5 shows the classification, including the additional characteristics of gellification level, weathering character, and physical

**Table 4.5** Typical characteristics of air-dried brown coal lithotypes from Latrobe Valley, Australia.

Lithotype	Abbreviation	Colour	Texture <sup>a</sup>	Gellification	Weathering pattern	Physical properties <sup>b</sup>
Dark	Dk	Dark brown to medium brown	High (20–30%) wood content. Often small (<25 cm) fragments	Gellification, particularly of woody material, common	Cracks wide and deep. Regular pattern	Strong, hard, heavy (high SG)
Medium–dark	M–d	Dark brown to medium brown	High to medium (10–20%) wood content. Often large (>25 cm) pieces	Some gellification but not extensive	Cracks wide. Some regularity of pattern	Strength variable, hardness and SG above average
Medium–light	M–l	Medium brown to light brown	High to low (0–10%) wood content. Often well preserved	Gellification uncommon. Confined mainly to wood	Cracks shallow. Irregular pattern	Intermediate physical properties
Light	Lt	Light brown	Medium to low wood content	Gellification rare	Cracks generally fine. Random orientation	Generally soft and relatively light (low SG)
Pale	Pa	Pale brown to yellow brown	Wood present but uncommon	Gellification very rare	Few extensive cracks	Soft, crumbles readily, very low SG

a) Wood content includes all plant fragments clearly distinguishable from the groundmass.

b) SG, specific gravity.

Source: From Taylor et al. (1998) modified after George (1975).

properties. The classification should be assessed on air-dried coal; colour is based on shades of brown, and texture refers to the amount of xylitic material present. The classification does not take into account mineral matter content because of the low ash levels in Victorian brown coals (George and Mackay 1991).

#### 4.1.1.2 Sapropelic Coals

Sapropelic coals are formed from the biological and physical degradation products of coal peat-forming environments, with the addition of other materials such as plant spores and algae. The resultant sediment is an accumulation of colloidal organic mud in which concentrations of spore remains and/or algae are present. Sapropelic coals are characteristically fine grained, homogeneous, dark in colour, and display a marked conchoidal fracture. They may occur in association with humic coals or as individual coal layers.

**Cannel coal** is black and dull; it is homogeneous and breaks with a conchoidal fracture. It is composed largely of miospores and organic mud laid down under water, such as in a shallow lake.

**Boghead coal** is algal coal, and the criteria for the assignment of a coal to a boghead is that the whole mass of that coal originated from algal material without consideration of the state of preservation of the algal colonies, i.e. whether they are well preserved or completely decomposed. Boghead coals may grade laterally or vertically into true oil shales.

Between these two major types of sapropelic coals, transitional or intermediate forms such as cannel–boghead or boghead–cannel are recognised. Essentially, all sapropelic coals look similar in hand specimen and can only be readily distinguished microscopically.

Coal descriptions using the above terms result in a considerable amount of data that can be used in conjunction with the laboratory

analysis of the coal. Such lithological logs can also provide information on coal quality that will influence the mining and preparation of the coal.

#### 4.1.2 Microscopic Description of Coal

The organic units or **macerals** that comprise the coal mass can be identified in all ranks of coal. Essentially, macerals are divided into three groups:

1. **Huminite–vitrinite** – woody materials.
2. **Exinite** – spores, resins, and cuticles.
3. **Inertinite** – oxidised plant material.

The original classification of maceral groups is referred to as the Stopes–Heerlen system and is given in Table 4.6 (Stopes 1935). Other detailed descriptions are well summarised by Speight (1994); see Table 4.7.

**Table 4.6** Stopes–Heerlen classification of maceral groups, macerals, and submacerals of hard coals.

Maceral group	Maceral	Submaceral
Vitrinite		Telocollinite
	Telinite	Gelocollinite
	Collinite	Desmocollinite
		Corpocollinite
Exinite (liptinite)	Sporinite	
	Cutinite	
	Suberinite	
	Resinite	
	Alginite	
	Liptodetrinite	
	Fluorinite	
	Bituminite	
	Exudatinite	
Inertinite	Fusinite	
	Semifusinite	
	Macrinite	
	Micrinite	
	Sclerotinite	
	Inertodetrinite	

Source: From Ward (1984), after Stopes (1935).

However, coals may be made up largely of a single maceral or, more usually, associations of macerals. These associations when studied microscopically are called microlithotypes.

In order to distinguish between the different microlithotypes, the ICCP has agreed that a lithotype can only be recorded if it forms a band >50 µm and that lithotypes are not composed purely of macerals from one or two maceral groups; they must contain 5% of accessory macerals. All microlithotypes may contain amounts of mineral matter; if this reaches 20% then the microlithotype is referred to as a carbominerite (Taylor et al. 1998).

The composition of the microlithotypes is listed in Table 4.8 (McCabe 1984) and described in the following. Their interrelationship is shown in Figure 4.1 (Bustin et al. 1983). Taylor et al. (1998) used the ICCP recommendations and described the microlithotypes of humic coals; the principal types are shown in Figure 4.2.

**Vitrinite** comprises 95% of the vitrinite macerals, telinite, and collinite in bands at least 50 µm thick (Figure 4.2).

Vitrinite occurs in coal seams as elongated lenses several millimetres thick. Vitrinite originated in anaerobic conditions due to high groundwater table levels in the peat mire.

Vitrinite makes up 40–50% of the Carboniferous coals in the Northern Hemisphere. In Gondwana coals; however, it rarely exceeds 20–30%. As a group, Late Cretaceous and Palaeogene–Neogene coals are generally rich in vitrinite and comparatively rich in liptinite, usually having >20% inertinite.

**Liptinite** layers form thin lenses or bands a few millimetres thick and have been deposited in water. Concentrations of up to 95% liptinite group macerals are rare.

**Inertinite** microlithotypes contain >95% inertinite macerals, which include inertodetrinite, semifusite, and fusite (Figure 4.2). In most coals, fusite comprises no more than 5–10% as thin bands and lenses. Fusite-rich coals are thought to be the result of the onset of aerobic conditions in peat formation.

**Table 4.7** Maceral terminology and origin (Speight 1994).

Maceral group	Maceral	Origin
Vitrinite	Telinite	Humified plant remains typically derived from woody, leaf, or root tissue with well to poorly preserved cell structures
	Collinite	Humified material showing no trace of cellular structure, probably colloidal in origin
	Vitrodetrinite	Humified attrital or less commonly detrital plant tissue with particles typically being cell fragments
Liptinite (exinite)	Sporinite	Outer casing of spores and pollens
	Cutinite	Outer waxy coating from leaves, roots, and some related tissues
	Resinite	Resin filling in cells and ducts in wood; resinous exudations from damaged wood
	Fluorinite	Essential oils in part; some fluorinate may be produced during physico-chemical coalification and represent non-migrated petroleum
	Suberinite	Cork cell and related tissues
	Bituminite	Uncertain, but probable algal origin
	Alginite	Tests of some groups of green algae; material referred to alginite shows moderate to strong fluorescence
	Exudatinitite	Veins of bitumen-related material expelled from organic matter during coalification
	Liptodetrinite	Detrital forms of liptinite that cannot be differentiated
Inertinite	Fusinite	Wood and leaf tissue oxidation
	Semifusinite	Wood or leaf tissue weakly altered by decay or by biochemical alteration
	Inertodetrinite	Similar to fusinite or semifusinite, but occurring as small fragments
	Macrinite	Humic tissue probably first gellified and then oxidised by processes similar to those producing semifusinite
	Sclerotinite	Moderately reflecting tissue of fungal origin; largely restricted to Palaeogene–Neogene coals
	Micrinite	Largely of secondary origin formed by disproportionation of lipid or lipid-like compounds

Inertodetrinite, which consists of 95% inertodetrinite, is common in Gondwana coals. These coals are composed of numerous inertite layers in which inertodetrinite and semifusinite make up over 95%. Inertodetrinite is present in Northern Hemisphere Carboniferous coals and other coals as a minor constituent. The high levels of inertinite in Gondwana coals have been attributed to the peat being oxidised to a high degree during formation (Plumstead 1962). Taylor et al. (1998) considered the characteristic petrographic composition of many Gondwana coals to be attributable to a climate of wet, cool summers and freezing winters.

The oxygen content of inertinite was retained in its structure at an early stage as a result of drying or freeze-drying. Similar material in warmer climates would have proceeded towards vitrinitization.

**Clarite** comprises microlithotypes that contain >95% of vitrinite and liptinite (Figure 4.2), each being >5% of the total. Vitrite and clarite are commonly associated, particularly in Carboniferous coals in the Northern Hemisphere and in Palaeogene–Neogene hard coals. Liptinite-rich clarites may owe their formation to algae, lipid-rich plants, and animal plankton and, as such, may grade into sapropelic coals.



**Table 4.8** Composition of microlithotypes.

Microlithotype	Composition
Vitrite	Vitrinite >95%
Liptite	Exinite >95%
Inertite	Inertinite >95%
Fusite	Inertite with no macrinite or micrinite
Clarite	Vitrinite and exinite >95%
Durite	Exinite and inertinite >95%
Vitrinertite	Vitrinite and inertinite >95%
Trimacerite	Vitrinite, exinite, inertinite, each >5%

Source: From McCabe (1984).

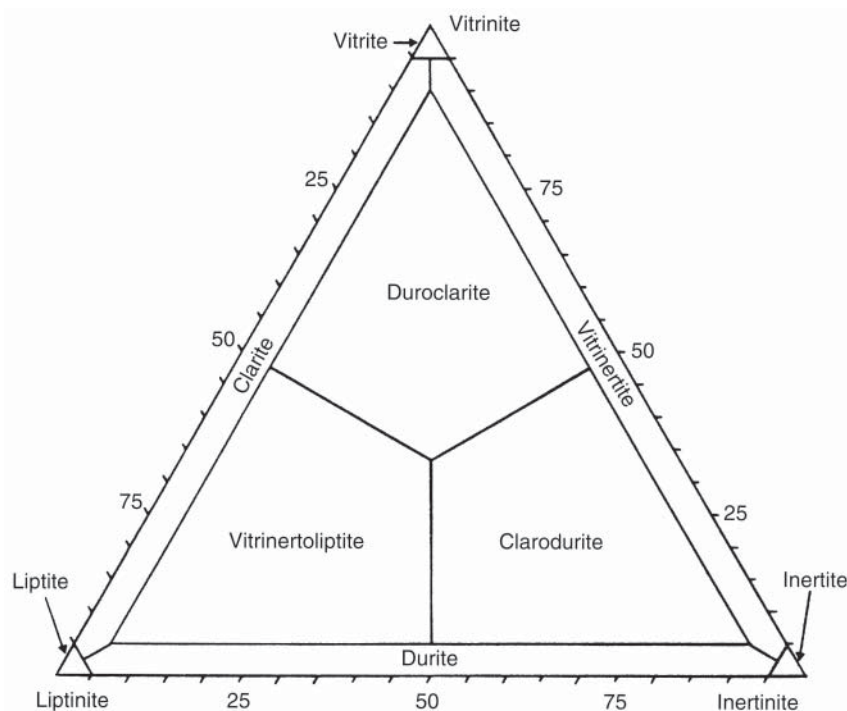
**Vitrinertite** contains 95% vitrinite and inertite. It is rare in Carboniferous coals and common in inertinite-rich Gondwana coals.

**Durite** is composed of 95% liptinite and inertinite (Figure 4.2). There is a wide variation in the proportion of durite in different coals.

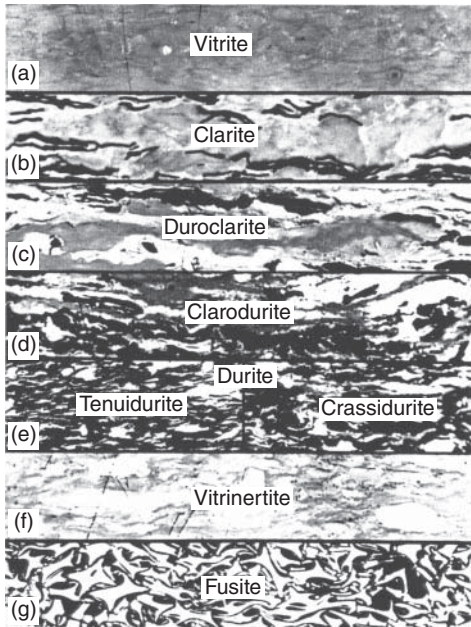
Taylor et al. (1998) suggested that durite occurs near to the margins of coal basins, as in the case of the Upper Silesian Carboniferous Basin. Some Gondwana coals are particularly durite rich, as found in South Africa. It is thought that the groundwater tables of the Gondwana peat mires were subject to greater fluctuation than those in the Carboniferous in the Northern Hemisphere.

**Trimacerite** is the only microlithotype group in which all three maceral groups are present. The trimacerite group is further divided into three microlithotypes: duroclarite, in which vitrinite is more abundant than liptinite; clarodurite, in which the proportion of inertinite is greater than vitrinite and liptinite; and vitrinertoliptite, in which liptinite predominates. In most coals, apart from vitrite, trimacerite occurs most frequently.

In low rank coals, i.e. lignites and subbituminous coals, the **vitrinite** maceral group is referred to as **huminite** and is regarded as



**Figure 4.1** Diagrammatic representation of microlithotype classification. Source: From Bustin et al. (1983), By permission of Geological Association of Canada.



**Figure 4.2** Microlithotypes: (a) vitrite from a high-volatile Ruhr coal, polished surface, oil immersion; (b) clarite from a Saar coal, polished surface, oil immersion; (c) duroclarite from a high-volatile Ruhr coal, polished surface, oil immersion; (d) clarodurite from a high-volatile Ruhr coal, polished surface, oil immersion; (e) durite from a high-volatile Ruhr coal, showing both tenuidurite and crassidurite, polished surface, oil immersion; (f) vitrinertite from a high-volatile Ruhr coal, polished surface, oil immersion; (g) fusite from a high-volatile Ruhr coal, polished surface, oil immersion. Original magnification for all images was  $\times 300$ . *Source:* From Taylor et al. 1998.

equivalent to, and the precursor of, the vitrinite macerals found in higher rank coals. Bustin et al. (1983) classified huminite macerals as summarised in Table 4.9 and gives details of their origin and their equivalents in the hard coals. The increase of coal rank leads to the homogenisation of the macerals of the huminite/vitrinite group, the term collinite being used to describe homogeneous structureless vitrite.

The relationship between maceral type and the original plant material has been well documented. The plant materials that make up coal have different chemical compositions, which in turn determine the types of group macerals.

There are variations in terminology when comparing maceral usage; for example, George and Mackay (1991) gave a maceral classification of brown coals based on the Australian Standard 2856.2 (AS 2856.2-1998 1998; updated 2013; Table 4.10). The relationship of types of plant debris to microlithotypes in lignite microfacies was proposed by Kasinski (1989) and is shown in Table 4.11. Such chemical differences are clear in lower rank coals, but it becomes increasingly difficult to distinguish petrographically between the various macerals with increasing coalification. This can be illustrated by an analysis of miospore floras and the petrographic types. Certain relationships have been established based on the investigation of thin layers of coal representing a moment in time during which environmental change was minimal. To illustrate this, in a thick coal seam in a stable area, the ascending miospore sequence and the resultant microlithotypes are shown in Figure 4.3. If the coal seam has splits, then the sequence may revert to the early phase of seam development. Above the split the normal sequence of phases may become re-established unless the sequence is again interrupted by splitting.

Smyth (1984) related the microlithotype composition of Permian coals in eastern Australia to depositional environments, as shown in Figure 4.4. Lower delta plain environments have produced coals relatively vitrinite rich, whereas upper delta plain and meandering fluvial coals are vitrinite poor. High subsidence rates prevailed during the accumulation of both, but water tables were high in the Early Permian and low in the Late Permian. A microlithotype analysis can give an indication of the texture of a coal. If two coals have equal overall contents of vitrinite and one has a higher vitrite content than the other, then this may be due to different thicknesses in the bands of vitrinite, which in turn may influence the preparation of the coal. Similarly, the size distribution of masses of inertinite may be important in the coking behaviour of the coal.

**Table 4.9** Huminite macerals.

Maceral	Origin	Petrological features	Equivalentents in hard coals
Textinite	Woody tissue	Primary cell wall structure still distinguishable; cell lumina mostly open	Telinite/telocollinite
Ulminite	Woody tissue	Higher degree of humification; texto-ulminite = cell wall structure still visible; eu-ulminite = no visible cell wall structure, cell lumina mostly closed	Telinite/telocollinite
Attrinite	Finely comminuted	Particle size <10 µm, product of degradation of huminite macerals	Desmocollinite
Densinite	Same as attrinite	Tighter packed than attrinite	
Gelinite	Derived from colloidal humic solutions that migrate into existing cavities and precipitate as gels	Secondary cell filling	Gelocollinite
Corpohumite	Condensation products of tannins characteristic of bark tissues	In cross-section, globular to tabular shape	Corpocollinite

Source: From Bustin et al. (1983), by permission of Geological Association of Canada.

**Table 4.10** Australian maceral classification of brown coals.

Maceral	Maceral subgroup
Telovitrinite	Textinite
	Texto-ulminite
	Eu-ulminite
	Telocollinite
Detrovitrinite	Attrinite
	Densinite
	Desmocollinite
Gelovitrinite	Corpogelinite
	Porigelinite
Inertinite	Fusinite
	Macrinite/micrinite
	Sclerotinite
Liptinite	Sporinite
	Cutinite
	Suberinite
	Resinite
	Liptodetrinite

Source: Based on Australian Standard 2856.2 (AS 2856.2-1998 1998) and George and Mackay (1991).

### 4.1.3 Mineral Content of Coals

The mineral content of coal is the non-combustible inorganic fraction. This is made up of minerals that are either detrital or authigenic in origin, and which are introduced into coal in the first or second phases of coalification. The principal mineral associations are outlined in Table 4.12 (Taylor et al. 1998).

**Detrital** minerals are those transported into a swamp or bog by air or water. A large variety of minerals can be found in coal, commonly, these are dominated by quartz, carbonate, iron, and clay minerals with a diverse suite of accessory minerals that may be peculiar to the local source rock.

Waterborne mineral matter is transported into coal swamps along channels that cut through the accumulating organic debris. When such channels are in flood, detritus is laid down on top of the organic material; such events are usually preserved as mineral-rich partings in coals. Mineral-rich materials present in the floor of the peat swamp may be

**Table 4.11** Characteristics of Palaeogene–Neogene lignite facies.

Microfacies	Pollen-spore spectrum	Phytodetritus content	Microlithotypes content
Pollen grains	Higher frequency of the water plant pollen grains and algal cells	Highest pollen grain frequency (>80%); low fungi-spores and wood tissue frequency (5%); corrosional structures on surfaces of pollen grains	High sapropelite content (70–80%); low content of telinite, detritinite, and collinite (10%); high sporinite content; significant addition of clay minerals; strong bituminisation, traces of decay
Pollen-resin grains		Relatively high frequency of pollen grains (60%), resin grains (20%), fungi spores (10%), relatively low wood-tissue frequency (15%)	Distinct addition of sapropelite (10–20%); higher inertinite content, low content telinite, detritinite, collinite (20% total); small distinct addition of clay minerals
Coniferae wood-tissue	Highest frequency of Taxodiaceae; Cupressaceae–Nyssaceae association (20%); relatively high frequency of Polypodiaceae spores	High wood-tissue frequency (>50%), relatively low frequency of pollen grains (<40%)	High content of telinite (40%) and detritinite (20%)
Fungi-spores and wood tissue, <i>Sequoia</i> wood-tissue	Highest frequency of Myricaceae–Cyrillaceae– <i>Alnus</i> association (30–40%); addition of Polypodiaceae spores; highest frequency of <i>Sequoia</i> – <i>Pinus</i> association	Relatively high frequency of resin grains (20%) and fungi spores (10%); relatively low pollen grains frequency (40%); high content of wood tissue with texture for <i>Sequoia</i>	Highest collinite content (40%); relatively high telinite content (30%) and detritinite (20%); high content of gelinite and significant resinite addition; high content of telinine and detritinite with the characteristic vein texture

Source: Based on Kasinski (1989).

incorporated into the organic layer by differential compaction within the swamp and by bioturbation action.

Windborne mineral matter is important, as this can be a significant contributor to the mineral contents of coals because of the slow accumulation rates in peat swamps. Coal swamp areas located in close proximity to active volcanic regions may receive high amounts of mineral matter. Associated lithologies with coals, such as flint clays and tonsteins, are indicative of such volcanic mineral deposition, and, if the volcanic event was short-lived but widespread, are extremely useful as stratigraphic marker horizons in coal sequences.

**Authigenic** minerals are those introduced into a peat during or after deposition, or into a

coal during coalification. Precipitated minerals may be disseminated through the peat or present as aggregates, whereas mineral-rich fluids present during the later stages of coalification tend to precipitate minerals on joints and any open voids within the coal. Common products of mineralisation are the calcium–iron minerals such as calcite, ankerite and siderite, and pyrite, with silica in the form of quartz. The element sulfur is present in almost all coals; it is usually present in the organic fraction of the coal, but inorganic or mineral sulfur is in the form of pyrite. Pyrite may be present as a primary detrital mineral or as secondary pyrite as a result of sulfur reduction of marine waters; thus, there is now considered to be a strong correlation between high sulfur coals and marine depositional environments.

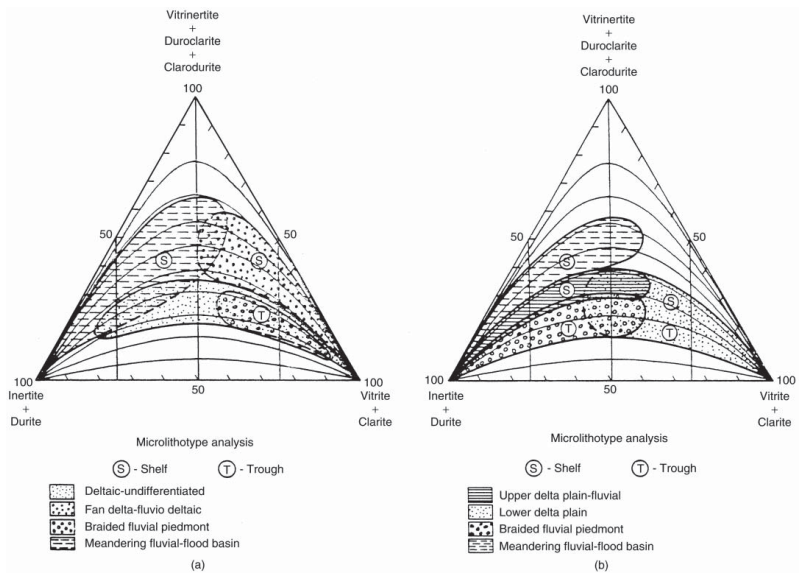
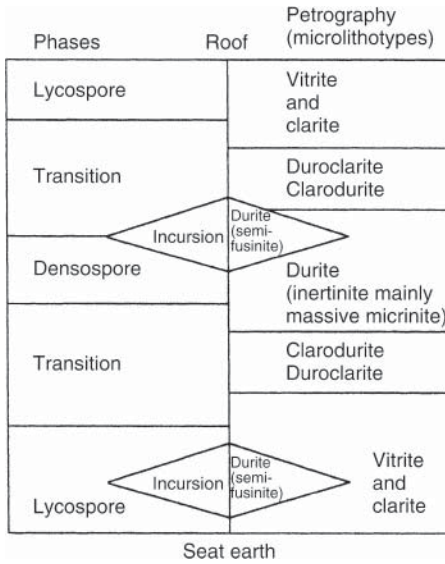


Figure 4.3 Microolithotype analysis related to depositional environment, for Early and Late Permian Coals, Australia (Smyth 1984).





**Figure 4.4** Diagrammatic profile of a coal seam showing the sequence of miospore phases and petrographic types. *Source:* From Smith (1968) with permission.

Clay minerals, on average, make up 60–80% of the total mineral matter associated with coal. Their genesis is complex; they can have a detrital origin or be a secondary product from aqueous solutions. Chemical conditions at the site of deposition also influence the type of clay minerals associated with coal. In particular, freshwater swamps with their low pH tend to favour in situ alteration of smectites, illite, and mixed-layer clays to kaolinite. Generally, illite is dominant in coals with marine roofs, whereas kaolinite is dominant in non-marine-influenced coals. Secondary clays are produced from alteration of primary clays; for example, chlorite is expected to occur in coals subjected to greater pressure and temperature.

Clay minerals occur in coal in two ways: either in tonsteins or as finely dispersed inclusions in maceral lithotypes.

Tonsteins have been formed by detrital and authigenic processes, and in particular are associated with volcanic activity. They usually contain kaolinite, smectite, and mixed-layer clays with accessory minerals.

Clay minerals can contaminate all microlithotypes. Those with less than 20% (by volume)

**Table 4.12** Minerals identified in coal (not exhaustive).

Mineral	Occurrence <sup>a</sup>
<b>Clay minerals</b>	
Illite-sericite	common–abundant
Montmorillonite	rare–common
Kaolinite	common–abundant
Halloysite	rare
<b>Iron disulfides</b>	
Pyrite	rare–common
Marcasite	rare–common
<b>Carbonates</b>	
Siderite	common–very common
Ankerite	common–very common
Calcite	common–very common
Dolomite	rare–common
Aragonite	rare
Witherite	rare
Strontianite	rare
<b>Oxides</b>	
Haematite	rare
Quartz	rare–common
Magnetite	very rare
Rutile	very rare
<b>Hydroxides</b>	
Limonite	rare–common
Goethite	rare
Diaspore	rare
<b>Sulfides (other than iron)</b>	
Sphalerite	rare
Galena	rare
Millerite	very rare
Chalcopyrite	very rare
Pyrrhotite	very rare
<b>Phosphates</b>	
Apatite	rare
Phosphorite	rare
Goyazite	rare
Gorceixite	rare
<b>Sulfates</b>	
Baryte	rare
Gypsum	very rare

(continued)

**Table 4.12** (Continued)

Mineral	Occurrence <sup>a</sup>
<b>Silicates (other than clays)</b>	
Zircon	rare
Biotite	very rare
Staurolite	very rare
Tourmaline	very rare
Garnet	very rare
Epidote	very rare
Sanidine	rare
Orthoclase	very rare
Augite	very rare
Amphibole	very rare
Kyanite	very rare
Chlorite	rare
<b>Salts</b>	
Gypsum	rare
Bischofite	very rare–common
Sylvin (sylvite)	very rare–common
Halite	very rare–common
Kieserite	very rare–common
Mirabilite	very rare–rare
Melanterite	very rare
Keramohalite	very rare

- a) Minerals classed as abundant to common occur in many coals in significant proportions (5–30% of mineral matter in coal). Minerals classed as rare or very rare commonly occur in small amounts (<5% of the total mineral matter), but also include some minerals that occur in somewhat larger amounts in only a few coals.

Source: From Taylor et al. (1998)

clay minerals are described as being ‘contaminated by clay’; for clay mineral contents of 20–60% (by volume) the term ‘carbargilite’ is used; if higher proportions of clay minerals are present the lithology is no longer a coal but an argillaceous shale.

Clay minerals have the property of swelling in the presence of water. Swelling is accompanied by reduction in strength, and disintegration is an end result. This is most significant in mines where coals have clay-rich roofs and floors, which can result in instability, as well as

difficulties encountered in drainage and dewatering in both underground and open-pit mine operations.

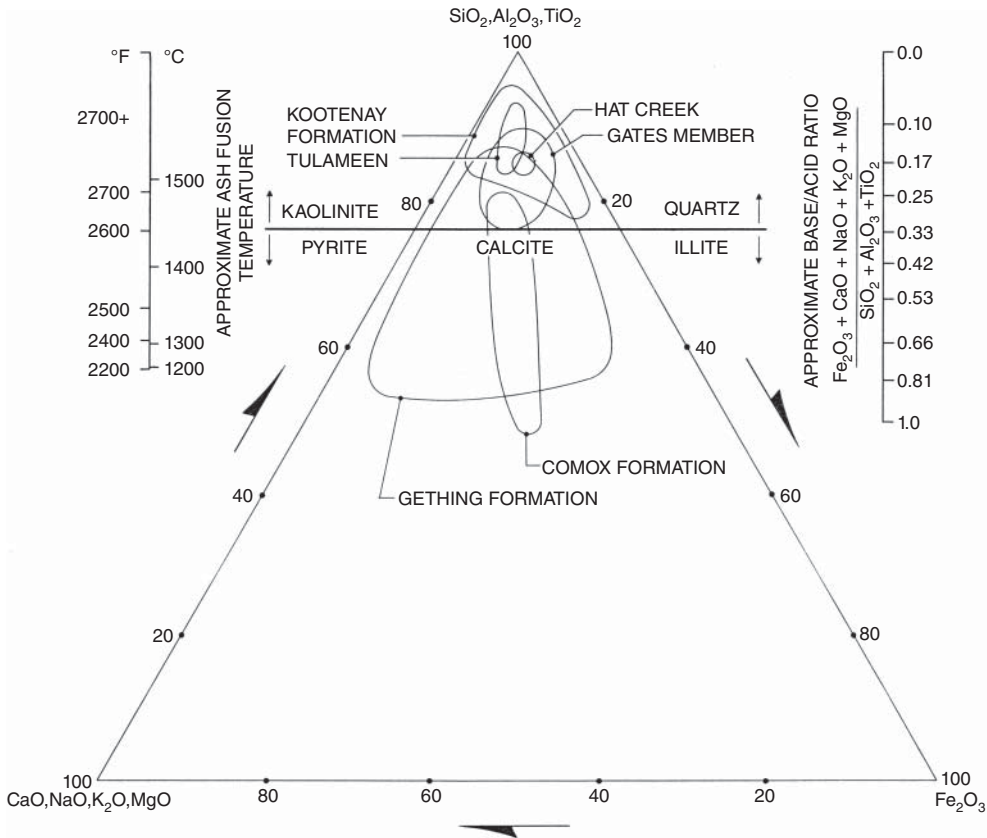
All of the aforementioned forms of mineral content in coals can be identified macroscopically by the field geologist in outcrop and borehole cores. There are other minerals that may be present in coal that affect its future potential use; these cannot be seen in hand specimens, being detectable only by chemical analysis.

The mineral matter content of coals and the surrounding country rock will influence the properties of the coal roof and floor, particularly their resistance and response to water. It will also influence the composition of mine dust with a diameter of below 5 µm, particularly in underground operations. Significant amounts of quartz in dust affect the incidence of silicosis.

The mineral matter in the coal will also affect the washability of the coal and, consequently, the yield and ash content of the clean coal.

Mineral impurities affect the suitability of a coal as a boiler fuel, with a low ash fusion point causing deposition of ash and corrosion in the heating chamber and convection passes of the boiler. Figure 4.5 shows that coal ash with low base/acid ratios (<0.25) has an excess of refractory acidic oxides (kaolinite–quartz mineral matter assemblages) that produce high ash fusion temperatures. Coals that have illite–calcite–pyrite mineral matter assemblages have proportionately more of the basic oxides (alkalis and ferric iron), so that ash fusion temperatures are correspondingly reduced. Ash fusion temperatures are used to predict boiler deposit build-up and slagging performance, when used as thermal coals (Pearson 1985). The presence in coal of phosphorous minerals, usually in the form of phosphorite or apatite, causes slagging in certain boilers, and steel produced from such phosphorus-rich coals tends to be brittle.

Halide minerals, such as chlorides, and sulfates and nitrates are present in coal usually as infiltration products deposited from brines migrating through the sedimentary sequence.



**Figure 4.5** Ternary diagram showing ash chemistry of some of Western Canada's coals, together with approximate ash fusion temperatures, and base/acid ratios. Approximate boundaries of pyrite–calcite–illite (marine influenced) coal ashes and kaolin–quartz (freshwater-dominated) coal ashes are also shown. *Source:* Pearson 1985 with permission.

They become significant in mining operations when mine waters enriched with, for example, nitrates create serious corrosion effects on pipework and other metal installations in the mine workings. In addition, chlorine causes severe corrosion in coal-fired boilers.

The more important trace element minerals found in coals are summarised in Table 4.13. They may originate from the original plant material or be components of other minerals in the coal. Several of them, notably boron, titanium, vanadium, and zinc, can have detrimental effects in the metallurgical industry.

The mineral matter in Carboniferous and Gondwana coals is broadly similar. There are,

however, differences in the total content and distribution of types of mineral matter. Gondwana coals commonly have higher contents of mineral matter, particularly as well-defined layers of mineral-rich material. Such bands are composed of kaolinite or other clay minerals and quartz. In addition, Gondwana coals tend to have fine clay or other mineral matter dispersed throughout the organic fraction.

The inorganic components of Paleogene–Neogene coals are strongly affected by the level of rank that the coal will have achieved. Groundwater leaching can lead to precipitation of minerals such as gypsum, barite, and other sulfates (Taylor et al. 1998).

**Table 4.13** Contents of trace elements in coals, soils, and shales (as ppm).

Element	Most coals	Soils	Shales
Antimony (Sb)	0.05–10	0.2–10	1.5
Arsenic (As)	0.5–80	1–50	13
Barium (Ba)	20–1000	100–3 000	550
Beryllium (Be)	0.1–15	<5–40	3
Boron (B)	5–400	2–100	130
Cadmium (Cd)	0.1–3	0.02–10	0.22
Caesium (Cs)	0.3–5	0.3–20	5
Chlorine (Cl)	50–2000	8–1 800	160
Chromium (Cr)	0.5–60	5–1 000	90
Cobalt (Co)	0.5–30	1–40	19
Copper (Cu)	0.5–50	2–100	39
Fluorine (F)	20–500	20–700	800
Gallium (Ga)	1–20	5–70	23
Germanium (Ge)	0.5–50	0.1–50	2
Gold (Au)	up to 0.01	0.001–0.02	0.002
Hafnium (Hf)	0.4–5	0.5–34	2.8
Lanthanum (La)	1–40	2–180	4.9
Lead (Pb)	2–80	2–100	23
Lithium (Li)	1–80	5–200	76
Manganese (Mn)	5–300	200–3 000	850
Mercury (Hg)	0.02–1	0.01–0.5	0.18
Molybdenum (Mo)	0.1–10	0.2–5	2.6
Nickel (Ni)	0.5–50	5–500	68
Niobium (Nb)	1–20	6–300	18
Phosphorus (P)	10–3000	35–5 300	700
Rubidium (Rb)	2–50	20–1 000	160
Scandium (Sc)	1–10	<10–25	13
Selenium (Se)	0.2–10	0.1–2	0.5
Silver (Ag)	0.02–2	0.01–8	0.07
Strontium (Sr)	15–500	50–1 000	300
Tantalum (Ta)	0.1–1	0.4–6	2
Thallium (Tl)	<0.2–1	0.1–0.8	1.2
Thorium (Th)	0.5–10	1–35	12
Tin (Sn)	1–10	1–20	6
Titanium (Ti)	10–2000	1 000–10 000	4600
Tungsten (W)	0.5–5	0.5–80	1.9
Uranium (U)	0.5–10	0.7–9	3.7
Vanadium (V)	2–100	20–500	130
Yttrium (Y)	2–50	10–250	41
Zinc (Zn)	5–300	10–300	120
Zirconium (Zr)	5–200	60–2 000	160

Source: From Taylor et al. (1998)

#### 4.1.4 Coal Petrography

The microscopic study of coal has enabled a better understanding of its organic and mineral components and its industrial utilisation. Classical petrographic studies in relation to geology were carried out by Thiessen (1920), Stach (1982), and Teichmüller and Teichmüller (1982); Teichmüller (1987, 1989) on Carboniferous coals in Europe. Teichmüller (1987, 1989) compared coal petrography and genesis of coal and the process of coalification, and other studies dealt with the relationship of petrography of the Carboniferous coals and their depositional environment (Diessel 1992).

Carboniferous coals have a typically bright lustrous appearance and consist of predominantly vitrite and clarite. Permian coals were principally formed in the Gondwana supercontinent with the exception of early Permian coals in northern China. Many of these coals have a dull appearance and contain a higher percentage of inertinite. There is also the tendency for a greater percentage of mineral matter to be present in these coals.

Late Mesozoic and Palaeogene–Neogene coals are variable and can be more complex in their make-up. They are usually rich in huminite and vitrinite with variable amounts of liptinite and they are low in inertinite. The majority of these coals are low rank and do not exhibit the fine banding that characterises the older coals (Taylor et al. 1998).

The principal uses of black coals on a world-wide basis are to generate electricity and to produce iron and steel. The latter still depends chiefly on coal, whereas in the electricity generation industry coal has competition from other energy sources; even so, coal still retains a 39% share of this market.

The relationship between coal properties and coal usage has been outlined by Taylor and Shibaoka (1976), Pearson (1980, 1985), Callcott and Callcott (1990), and Taylor et al. (1998).

Coals that are to be used for conventional coke production must have three essential properties:

- (i) they must be within a specific range in rank for the coking process to occur, i.e. bituminous coal;
- (ii) they must possess a high proportion of fusible macerals (>40% vitrinite) to form a strong, well-fused coke;
- (iii) they must have low levels of certain elements, notably sulfur and phosphorus, and be generally low in mineral matter.

Steam or thermal coals used for electricity generation are required to have a low mineral matter level with a high calorific value (CV). Ash fusion temperatures are preferred to be high and sulfur, nitrogen and trace elements to be low. Local power stations can operate on a wide range of coals including brown coals, whereas export steam coals are dominated by high-volatile bituminous coals with mineral matter contents <15%.

The various macerals and maceral groups react differently to physical stress. Vitrinite is brittle and fractures easily, whereas liptinite–inertinite associations are more durable. Therefore, when coal is crushed, a higher percentage of vitrinite will be found in the fine fraction, with inertinite concentrated in the coarser fraction.

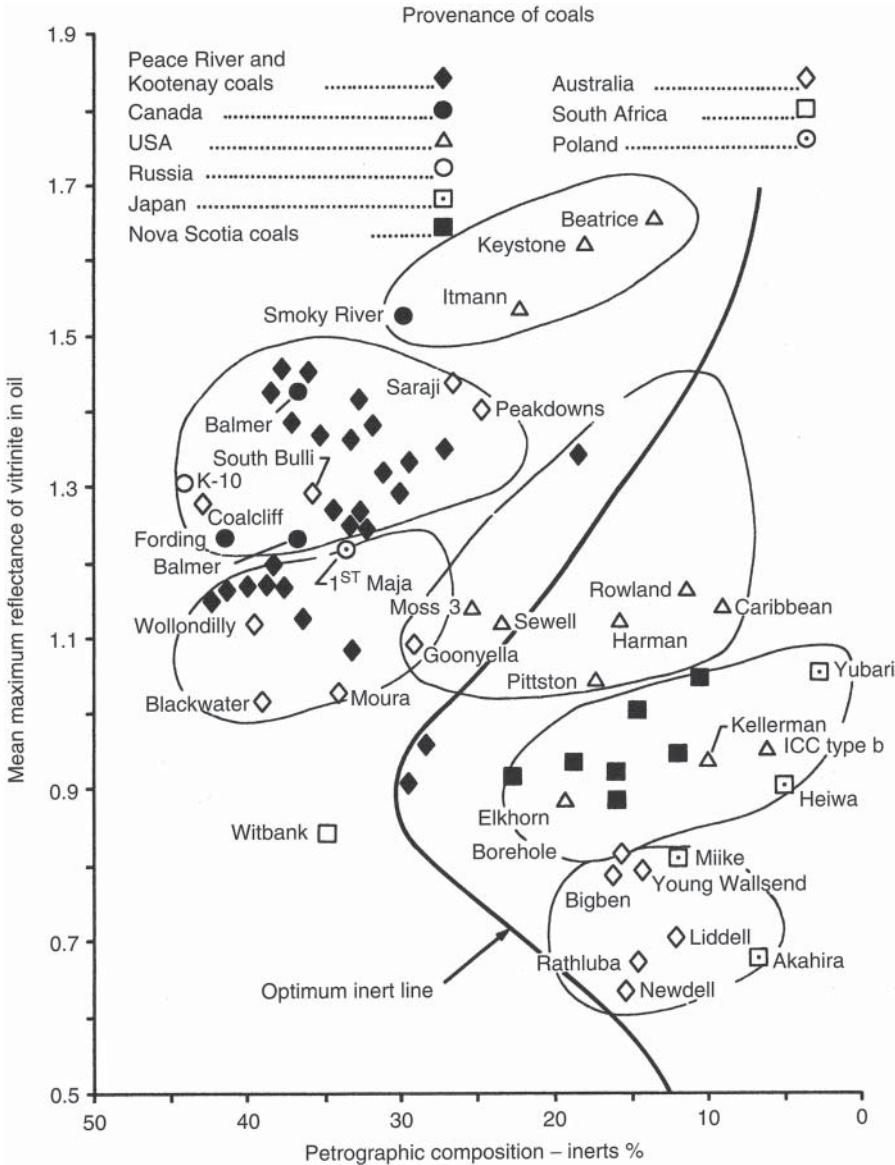
In coke production, vitrinite is the maceral group that contributes most to its formation. However, a stronger coke is obtained if the vitrinite is reinforced by inertinite. The liptinite group is characterised by high hydrogen/carbon ratios and therefore produce large amounts of gas on heating, all of which contributes to the fluidity and swelling properties of the coal. However, abundant liptinites are relatively resistant to thermal breakdown and remain after vitrinite has become plastic. In the inertinite maceral group, fusinite and semi-fusinite do not fuse during carbonisation due to their insufficient hydrogen content. These macerals are characterised by higher oxygen/carbon ratios. The inertinite maceral group is thought to have little influence during coke-making, although numerous studies on the coking properties of coal suggest that some



inertinite is completely fused during the coke-making process. Figure 4.6 shows the petrographic composition expressed as inerts, or the percentage of inertinite macerals plus inert semifusinite, plus ash composition calculated by the Parr formula (see Section 4.3.1.1) and rank (expressed as  $R_{o,max}$ ) of world-traded coking coals. The 'optimum inert' line represents

the optimum amount of inert components that would produce the strongest coke for each rank. Coal compositions to the left of the line are inertinite rich and those to the right are reactive rich (Pearson 1980).

The application of vitrinite reflectance methods (see Section 4.2.1) to reactive macerals has shown that there is a direct relationship



**Figure 4.6** Petrographic compositions of coking coals traded internationally. *Source:* Pearson (1980) with permission.

between the types of reactive macerals and the amount of inerts or non-reactives in making coke (Zimmerman 1979). By testing coals of different reflectance values in relation to the quantity of inerts in the coal, the relative strength (or stability) of the coal can be determined based on petrographic analyses. It has been shown that reactivities in themselves will not make a good coke, but require inert material in proper proportion. The amount of inerts required will vary with the types of reactivities present. This ratio of inerts to reactivities is used to determine the coke strength, and a balance index is calculated from this ratio and called the composition balance index (CBI). The amount of inerts present and the strength properties of each reactive type can be shown as a series of curves, where each curve shows the greatest strength when the highest amount of inerts is present in the coal. Low-reflectance types have low relative

strengths. The strength index (SI) is the comparative coke strength of the reactive macerals present in the coal, and the SI together with the CBI is required to predict the strength of the coke produced from any coal, and will indicate the ability of the coke to perform in a blast furnace.

Figure 4.7 shows the relationship between SI and CBI and its effect on coke strength and coke resistance for bituminous coals (Zimmerman 1979).

Coals used for combustion are less specific in terms of coal rank and type. It is the CV of the coal that is of prime interest, i.e. the percentage of combustible matter against non-combustible matter (mineral matter and water). Liptinite with high hydrogen/carbon ratio has the highest CV, followed by vitrinite and inertinite; however, vitrinite and inertinite increase in CV with increasing rank, whereas liptinite declines.

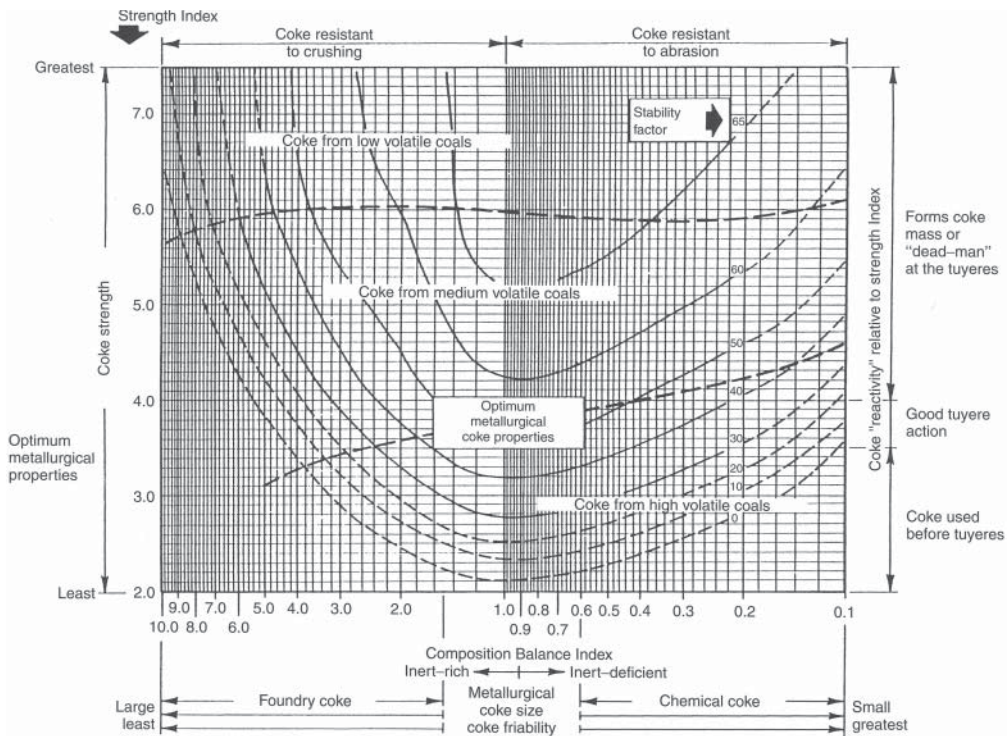


Figure 4.7 Characteristics of coke attainable from bituminous coals. Source: From Zimmerman (1979).

## 4.2 Coalification (Rank)

### 4.2.1 Coalification

Coalification is the alteration of vegetation to form peat, succeeded by the transformation of peat through lignite, subbituminous, bituminous, semi-anthracite, to anthracite and meta-anthracite coal.

The degree of transformation or coalification is termed the coal rank, and the early identification of the rank of the coal deposit being investigated will determine the future potential and interest in the deposit.

To understand coal rank, a brief examination of the coalification process is given, particularly those conditions under which coals of different rank are produced. A detailed account of coalification and its physical and chemical processes is given by Taylor et al. 1998. They described the major stages of coalification

from peat to meta-anthracite. These are summarised in Table 4.14, which outlines not only the denoted rank of coal but also the dominant processes and physico-chemical changes undergone in each stage in order to produce an increase in rank.

The coalification process is essentially an initial biochemical phase followed by a geochemical or metamorphic phase. The biochemical phase includes those processes that occur in the peat swamp following deposition and burial, i.e. during diagenesis. This process is considered to be in operation until the hard brown-coal stage is reached. The most intense biochemical changes occur at very shallow depths in the peat swamps. This is chiefly in the form of bacteriological activity, which degrades the peat and may be assisted in this by the rate of burial, pH, and levels of groundwater in the swamp. Bacteriological activity ceases with increased burial, and it is

**Table 4.14** Major stages of the development from peat to meta-anthracite.

Coalification stage	Approximate ASTM rank range	Predominant processes	Predominant physico-chemical changes
1. Peatification	Peat	Maceration, humification, gellification, fermentation, concentration of resistant substances	Formation of humic substances, increase in aromaticity
2. Dehydration	Lignite to subbituminous	Dehydration, compaction, loss of O-bearing groups, expulsion of —COOH, CO <sub>2</sub> , and H <sub>2</sub> O	Decreased moisture contents and O/C ratio, increased heating value, cleat growth
3. Bituminization	Upper subbituminous A to high-volatile A bituminous	Generation and entrapment of hydrocarbons, depolymerization of matrix, increased hydrogen bonding	Increased vitrinite R <sub>o</sub> , increased fluorescence, increased extract yields, decrease in density and sorbate accessibility, increased strength
4. Debituminization	Uppermost high-volatile bituminous A to low-volatile bituminous	Cracking, expulsion of low molecular weight hydrocarbons, especially CH <sub>4</sub>	Decreased fluorescence, decreased molecular weight of extract, decreased H/C ratio, decreased strength, cleat growth
5. Graphitization	Semi-anthracite to anthracite to meta-anthracite	Coalescence and ordering of pre-graphitic aromatic lamellae, loss of hydrogen, loss of nitrogen	Decrease in H/C ratio, stronger X-ray diffraction peaks, increased sorbate accessibility, anisotropy, strength, ring condensation and cleat healing

Source: From Taylor et al. (1998) According to Levine (1993).

considered absent at depths greater than 10 m. Carbon-rich components and volatile content of the peat are little affected during the biochemical stage of coalification; however, with increased compaction of the peat, moisture content decreases and CV increases.

From the brown coal stage, the alteration of the organic material is severe and can be regarded as metamorphism. Coals react to changes in temperature and pressure much more quickly than do mineral suites in rocks. Coals, therefore, can indicate a degree of metamorphism in sequences that show no mineralogical change.

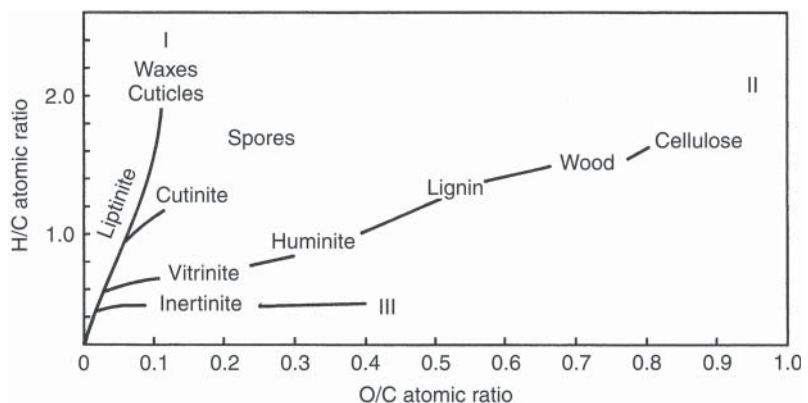
During the geochemical or metamorphic stage, the progressive changes that occur within coals are an increase in the carbon content and a decrease in the hydrogen and oxygen content, resulting in a loss of volatiles. This, together with continued water loss and compaction, results in the reduction of the coal volume. Products of such coalification are methane ( $\text{CH}_4$ ), carbon dioxide ( $\text{CO}_2$ ), and water; water is quickly lost, and the  $\text{CH}_4$  to  $\text{CO}_2$  ratio increases with rank.

These changes in the physical and chemical properties of the coal are, in reality, the changes to the inherent coal constituents. During coalification the three maceral groups become enriched in carbon, with each maceral group (i.e. exinite, inertinite, and huminite [vitrinite]) following a distinct coalification path. Figure 4.8, after van Krevelen (1961),

illustrates the distinct coalification paths. The petrographic properties of vitrinite change uniformly with increasing rank.

The reflectance progressively increases in reflected light, whereas organic materials become opaque and plant structure becomes difficult to recognise in transmitted light. The optical properties of vitrinite have enabled it to be used as an indicator of rank. Teichmüller and Teichmüller (1982) describe the method used in detail as applied to the medium-volatile bituminous to meta-anthracite and semi-graphite range of coals, i.e. coals with less than 30% volatile matter. Also, reflectance is considered the best rank parameter for anthracites, and reflectance is nearly comparable to moisture content as a rank indicator in high-volatile bituminous coals. It was originally suggested that this is not so for lower rank coals; however, later studies showed the utility of reflectance in low rank lignitic coals, provided that care is taken in the selection of the component measured. Ward (1984) suggested rank classes in terms of vitrinite reflectance (Tables 4.15 and 4.16) show the changing pattern of coal composition with increasing coalification (Diessel 1992). This increase in vitrinite reflectance with increase in coal rank is shown in Figure 4.9a for New Zealand coals; these have high proportions of vitrinite, and most fall within a restricted band on a volatile matter/CV plot. The mean reflectance values given in Figure 4.9b are reported to be on the

**Figure 4.8** Diagram showing the coalification tracks of liptinite, inertite, and huminite-vitrinite. Source: From Bustin et al. (1983), based on van Krevelen (1961), by permission of Geological Association of Canada.



**Table 4.15** Rank classes in terms of vitrinite reflectance.

Rank	Maximum reflectance $R_{o,max}$ (%)
Subbituminous	<0.47
High-volatile bituminous C	0.47–0.57
High-volatile bituminous B	0.57–0.71
High-volatile bituminous A	0.71–1.10
Medium-volatile bituminous	1.10–1.50
Low-volatile bituminous	1.50–2.05
Semi-anthracite	2.05–3.00 (approx.)
Anthracite	>3.00

Source: From Ward (1984).

high side; nevertheless, the reflectance–rank relationship is a meaningful one (Suggate and Lowery 1982).

It should be noted that, in the case of high-volatile South African Gondwana coals, reflectance is a better indicator than moisture due to the presence of higher amounts of inertinite, which has a lower moisture content.

Figure 4.9c shows the relationship between  $R_{o,max}$  and volatile matter (dry, mineral matter free, dmmf) for non-marine Canadian Cretaceous coal, Australian non-marine Gondwana

coal, and marine-influenced Pennsylvanian coal from the USA. In general, the non-marine Cretaceous coals possess lower volatile yields than the non-marine Gondwana and marine influenced Pennsylvanian coals. This variation is a consequence of compositional differences, with the Cretaceous coals having a higher inertinite content (Pearson 1985).

Fluorescence microscopy of the liptinite macerals and the coloration of the liptinite (thermal alteration index) are useful for coals of low rank, but these methods are not as refined as vitrinite reflectance.

During coalification, sapropelic coals undergo alteration similar to that of the liptinite component of humic coals. At the peat stage, sapropelic coals are enriched in hydrogen relative to humic coals, but at advanced stages of coalification (90% carbon) the chemical composition of boghead, cannel, and humic coals is similar. During coalification, significant amounts of bitumen may be generated from sapropelic coals.

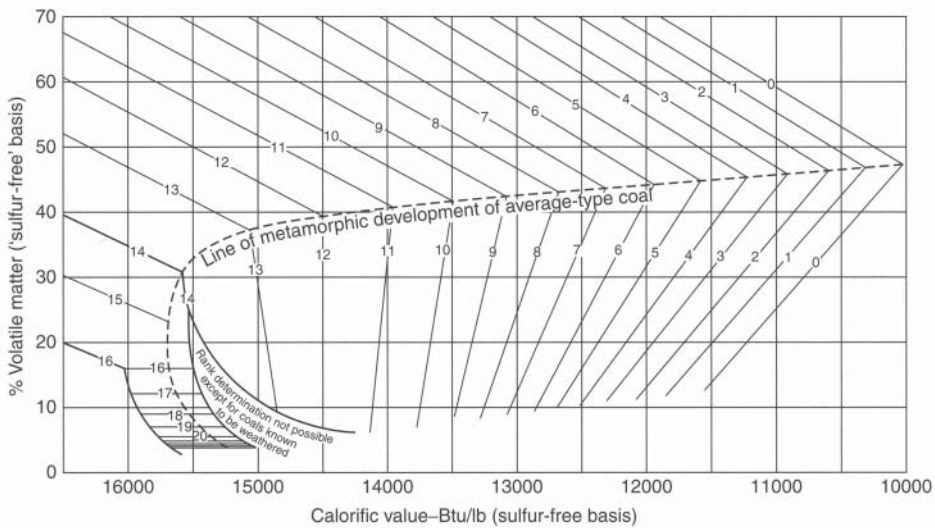
#### 4.2.2 Causes of Coalification

The coalification process is governed primarily by rises in temperature and the time during which this occurs.

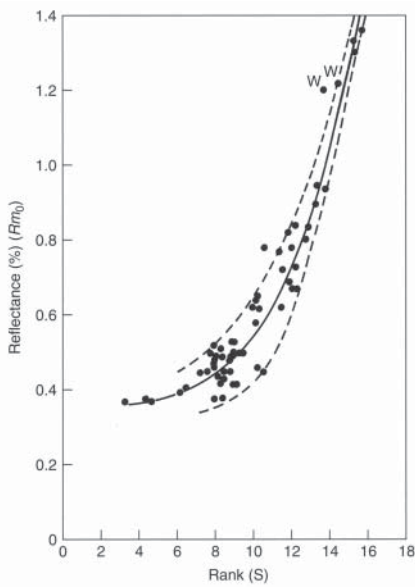
**Table 4.16** Some rank parameters showing the changing pattern of coal composition with increasing coalification (Diessel 1992).

Rank stage	Carbon (% daf)	Volatile matter (% daf)	Gross CV ( $\text{MJ kg}^{-1}$ )	In-situ moisture (%)	Vitrinite reflectance random $R_o$ (%)	Vitrinite reflectance $R_{max}$ (%)
Wood	50	> 65	11.7			
Peat	60	> 60	14.7	75	0.20	0.20
Brown coal	71	52	23.0	30	0.40	0.42
Subbituminous coal	80	40	33.5	5	0.60	0.63
High-volatile bituminous coal	86	31	35.6	3	0.97	1.03
Medium-volatile bituminous coal	90	22	36.0	< 1	1.49	1.58
Low-volatile bituminous coal	91	14	36.4	1	1.85	1.97
Semi-anthracite	92	8	36.0	1	2.65	2.83
Anthracite	95	2	35.2	2	6.55	7.00

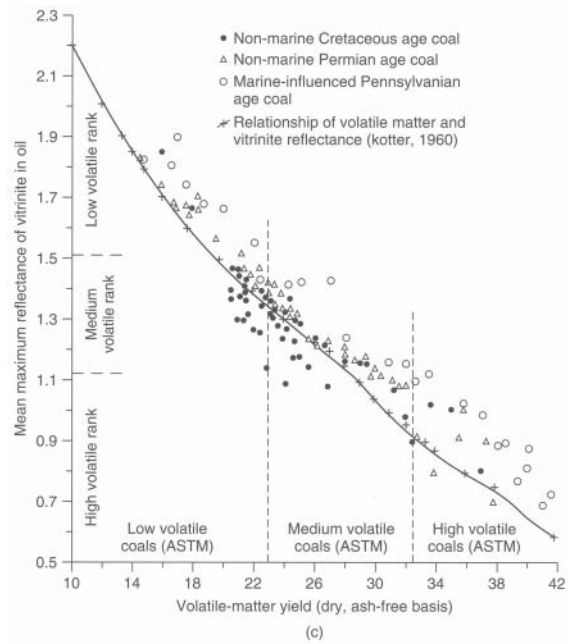




(a)



(b)



(c)

**Figure 4.9** (a) Rank scale of coal using axes of volatile matter and CV (Suggate 1959), with permission of the Royal Society of New Zealand. (b) Reflectance/rank relationship (w: weathered coal). *Source:* From Suggate and Lowery (1982), with permission of the Royal Society of New Zealand. (c) Relationship between  $R_{o,max}$  and volatile matter yield (dmmf) for non-marine Canadian Cretaceous coals, Permian Australian coals, and marine-influenced Pennsylvanian coals from the USA. *Source:* From Pearson (1985) with permission.

#### 4.2.2.1 Temperature

Temperature changes can be achieved in two ways:

- (i) The direct contact of the coal with igneous material, either as minor intrusions or as deep-seated major intrusions. The coals exhibit loss of volatiles, oxygen, CH<sub>4</sub>, and water, and the surrounding sediments will show evidence of contact metamorphism, e.g. the local development of high-rank coal in the Gondwana coals of South Africa and India and in the Palaeogene–Neogene coals of Sumatra, Indonesia.
- (ii) The rise in temperature associated with the depth of burial. Increasing depth of burial results in a decrease in the oxygen content of the coals and the increase in the ratio of fixed carbon to volatile matter. Professor Carl Hilt (1873) observed this phenomenon, and Hilt's law states that 'in a vertical sequence, at any one locality in a coalfield, the rank of the coal seams rises with increasing depth'.

The rate of rank increase known as the rank gradient, is dependent on the geothermal gradient and the heat conductivity of the rocks. Where the geothermal gradient is high (70–80 °C km<sup>-1</sup> depth), bituminous rank can be attained at depths of 1500 m (Upper Rhine Graben, West Germany); but in the same area, the same rank is reached at depths of 2600 m when the geothermal gradient is lower (40 °C km<sup>-1</sup>) (Stach 1982). Similar basal studies have shown variations in geothermal gradient in different parts of the basin (Teichmüller 1987; Teichmüller and Teichmüller 1982). Studies of the Remus Basin in the Canadian Arctic show differing geothermal gradients of 55 °C km<sup>-1</sup> in the eastern part and 20 °C km<sup>-1</sup> in the western part. The Remus Basin contains 90 seams of coal with ranks ranging from lignite to high-volatile bituminous with a maximum palaeothickness of 4500 m. In South Wales, it is suggested that the coalification that has produced anthracitisation is due to the proximity of a magmatic heat

source. The anthracite field has a present-day geothermal gradient of 25 °C km<sup>-1</sup>.

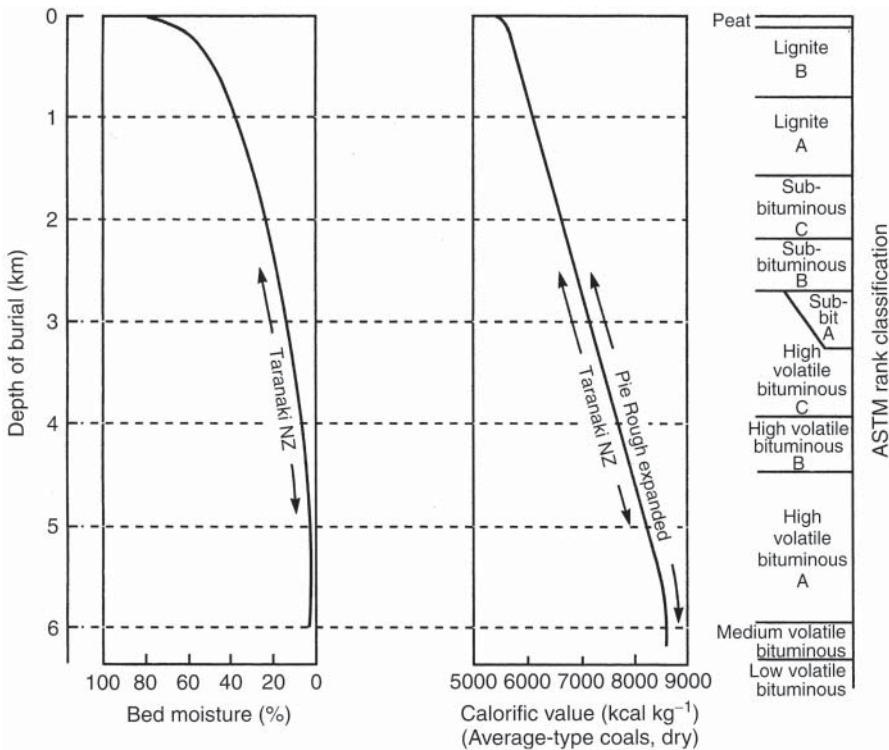
Figure 4.10 illustrates the manner in which American Society for Testing and Materials (ASTM) rank boundaries vary in depth from the surface according to the geothermal gradient, as reflected by variations in the moisture and CV relationships (Suggate 1982).

#### 4.2.2.2 Time

Usually, coalification temperatures are lower than was once inferred from experimental coalification studies. Stach (1982) quoted temperatures of the order of 100–150 °C as sufficient for bituminous coal formation according to geological observations. To attain higher rank, higher temperatures are required with more rapid rates of heating (contact metamorphism) rather than with slower heating rates (subsidence and depth of burial). Therefore, it is apparent that the degree of coalification is less where sediments have subsided rapidly and the 'cooking time' was short, and time only has a real effect when the temperature is sufficiently high to allow chemical reaction to occur. Where very low temperatures occur over a very long period, little coalification takes place; for example, the lower Carboniferous lignites in the Moscow Basin. The influence of time, therefore, is all the greater the higher the temperature.

#### 4.2.2.3 Pressure

The influence of pressure is at its greatest during compaction and is most evident from the peat to subbituminous coal stages, in the decrease of porosity and the reduction of moisture content with depth. Stach (1982) states that the pressure promotes 'physico-structural coalification', whereas rise of temperature accelerates 'chemical coalification'. With gradual subsidence of coal, both influences run parallel, but occasionally physico-structural coalification may precede chemical coalification; for example, where relatively low-moisture coals have been produced by early folding. Chemical coalification will



**Figure 4.10** Composite sequence providing an example of the relationship between depth, CV, and ASTM rank. The average geothermal gradient for this sequence is estimated at 26–27 °C km<sup>-1</sup>. Source: From Suggate (1982).

advance when additional heat is supplied; for example, from intrusive bodies. With increasing chemical coalification, pressure has less influence. Laboratory experiments suggest that the confining pressure may inhibit chemical coalification and retard the process; for example, the removal of gas is more difficult, and the alteration of macerals is postponed by pressure.

Local rises in rank can occur along shear planes, which is probably due to frictional heat.

#### 4.2.2.4 Radioactivity

Increase of rank by radioactivity is rarely observed and is likely to be only in the form of microscopic contact haloes of higher reflectivity around uranium/thorium concentrations in the coal.

## 4.3 Coal Quality

Coal quality, in essence, means those chemical and physical properties of a coal that influence its potential use.

It is essential to have an understanding of the chemical and physical properties of coal, especially those properties that will determine whether the coal can be used commercially. Coals need to possess particular qualities for selected usage. Should they meet such requirements, then they can be mined and sold as a pure product or, if the quality could be improved, they can be blended with other selected coals to achieve a saleable product.

The quality of a coal is determined by the make-up of the original maceral and mineral matter content of the coal, and its degree of coalification (rank). In order for this to be

understood in analytical terms, set procedures for determining chemical and physical properties of coals have been set up (see Karr 1978). A number of countries and organisations have defined standards of procedure that should be consulted (Appendix A).

A knowledge of the most commonly determined properties of a coal is important, particularly those that are deleterious to the coal. Such coal analyses are essential in the evaluation of a coal deposit, i.e. to be aware of which seams or parts of seams will be unacceptable when mining commences or, conversely, those seams or parts of seams that will yield a premium product for the predetermined market. It is possible that, after analysing a coal, hitherto undetected properties may enhance the product or even suggest a different end usage for the coal, e.g. the discovery that a coal has good coking properties when it was originally considered for a steam coal product.

An outline is now given of the fundamental chemical and physical properties of coal and what they mean in terms of the coal's usability.

#### 4.3.1 Chemical Properties of Coal

In simple terms, coal can be regarded as being made up of moisture, pure coal, and mineral matter.

The moisture consists of surface moisture and chemically bound moisture, the pure coal is the amount of organic matter present, and the mineral matter is the amount of inorganic material present, which when the coal is burnt produces ash. Clearly, decomposition during heating of some inorganic minerals means that ash and mineral matter composition cannot be equal.

Coal analyses are often reported as proximate or ultimate analysis. Proximate analysis is a broad analysis that determines the amounts of moisture, volatile matter, fixed carbon, and ash. This is the most fundamental of all coal analyses and is of great importance in the practical use of coal. The tests are highly dependent on the procedure used, and different results

are obtained using different times and temperatures. It is important, therefore, to know the procedure used and the reported basis (see Section 4.3.1.1).

Ultimate analysis is the determination of the chemical elements in the coal, i.e. carbon, hydrogen, oxygen, nitrogen, and sulfur. In addition, the calculation of the amounts of those elements that have a direct bearing on the usability of the coal is necessary. These may include forms of sulfur, chlorine, and phosphorus, an analysis of those elements making up the mineral matter content of the coal, and selected trace elements.

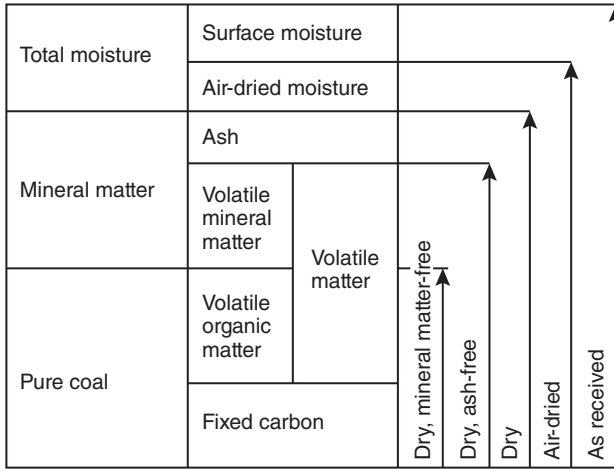
##### 4.3.1.1 Basis of Analytical Data

Before proceeding to the analysis of the coal, it is important to understand how the moisture, ash, volatile matter, and fixed carbon relate to one another, and to the basis that analytical data are presented.

It is important in evaluating previous coal analyses that the basis on which they are presented is known. It is unfortunately a common problem that analyses are given which do not indicate on what basis they are presented. Indeed, they are often listed together on different bases that are not stated.

Coal analyses may be reported as follows (see Table 4.17):

- (i) 'As received' basis (ar), also 'as sampled'. The data are expressed as percentages of the coal including the total moisture content, i.e. including both the surface and the air-dried moisture content of the coal.
- (ii) 'Air-dried' basis (adb). The data are expressed as percentages of the air-dried coal; this includes the air-dried moisture but not the surface moisture of the coal.
- (iii) 'Dry' basis (dry). The data are expressed as percentages of the coal after all the moisture has been removed.
- (iv) 'Dry ash-free' basis (daf). The coal is considered to consist of volatile matter and fixed carbon on the basis of recalculation with moisture and ash removed. It should

**Table 4.17** Components of coal reporting to different bases.

Source: From Ward (1984) with permission of Blackwell Scientific Publications.

be noted that this does not allow for the volatile matter derived from minerals present in the air-dried coal. This basis is used as the easiest way to compare organic fractions of coals.

- (v) Dmmf. Here, it is necessary that the total amount of mineral matter rather than ash is determined, so that the volatile matter content in the mineral matter can be removed.

Table 4.18 gives the required formulas for the calculation of results to the above bases (reference BS ISO 1170:2008, Appendix B).

In addition, the following countries have developed equations to calculate the mineral matter content of their coals.

North America

original Parr formula

$$MM = 1.08A + 0.55S$$

modified Parr formula

$$MM = 1.13A + 0.47Sp\text{pyr} + Cl$$

UK

British Coal Utilisation Research Association formula

$$MM = 1.10A + 0.53S + 0.74CO_2 - 0.36$$

King–Maries–Crossley formula (revised by British Coal)

$$MM = 1.13A + 0.5Sp\text{pyr} + 0.8CO_2 - 2.8S\text{ash} + 2.8SS\text{sulf} + 0.3Cl$$

Australia

Standards Association of Australia formula

$$MM = 1.1A$$

where MM (%) is mineral matter, A (%) is ash, S (%) is total sulfur, Spyr (%) is pyritic sulfur, SSulf (%) is sulfate sulfur, Sash (%) is Sulfur in ash, Cl (%) is chlorine, CO<sub>2</sub> (%) is carbon dioxide.

All values are expressed on an air-dried basis.

#### 4.3.1.2 Proximate Analysis

**4.3.1.2.1 Moisture** The terminology used in describing the moisture content of coals can be confusing and needs to be clarified. The most confusing term is inherent moisture, which has many different definitions and should be avoided if possible. If used in any tests it is necessary to ascertain the exact definition that the reference is using.

There is no exact method of determining moisture content. The coal industry has therefore developed a set of empirically determined definitions, which are as follows:



**Table 4.18** Formulae for calculation of results to different bases.

Given result	As sampled (as received) (as despatched) (as fired)	Air dry	Wanted result dry	Dry, ash free	Dry, mineral matter free
As sampled					
(as received)	—	100 – Mad	100	100	100
(as despatched)		100 – Mar	100 – Mar	100 – (Mar + Aar)	100 – (Mar + MMar)
(as fired)					
As analysed	100 – Mar	—	100	100	100
(air dry)	100 – Mad		100 – Mad	100 – (Mad + Aad)	100 – (Mad + MMad)
Dry	100 – Mar	100 – Mad	—	100	100
	100	100		100 – Ad	100 – MMd
Dry, ash free	100 – (Mar + Aar)	100 – (Mad + Aad)	100 – Ad	—	100 – Ad
	100	100	100		100 – MMd
Dry, mineral matter free	100 – (Mar + MMar)	100 – (Mad + MMad)	100 – MMd	100 – MMd	—
	100	100	100	100 – Ad	

M = moisture %; A = ash %; MM = mineral matter %; ar = as-received basis; ad = air-dried basis; d = dry basis.

Source: From BS 1016-100 (1994).

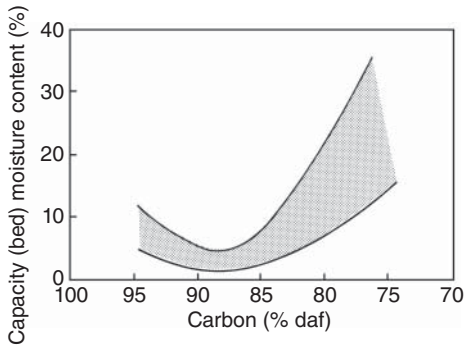
1. *Surface moisture.* This is adventitious moisture, not naturally occurring with the coal and which can be removed by low-temperature air drying (~40 °C). This drying step is usually the first in any analysis, and the moisture remaining after this step is known as air-dried moisture.
2. *As-received or as-delivered moisture.* This is the total moisture of the coal sample when received or delivered to the laboratory. Usually, a laboratory will air dry a coal sample, thereby obtaining the ‘loss on air drying’. An aggressive drying step is then carried out, which determines the air-dried moisture. These results are added together to give the total/as-received/as-delivered moisture.
3. *Total moisture.* This is all the moisture that can be removed by aggressive drying (~150 °C in vacuum or nitrogen atmosphere).
4. *Air-dried moisture.* This is the moisture remaining after air drying and which can be removed by aggressive drying.

In addition to those generally used terms, the following terms are used for scientific purposes: moisture holding capacity (MHC) and capacity moisture or equilibrium moisture. It is not within the scope of this book to detail the analytical procedure required; but, suffice it to say, that it is lengthy and expensive.

These terms relate to the in-bed or in-situ moisture of a coal. Technically, it is the MHC that increases with decreasing rank, as shown in Figure 4.11 (Berkowitz 1979).

High moisture is undesirable in coals as it is chemically inert and absorbs heat during combustion, and it creates difficulties in handling and transport. It lowers the CV in steam coals and lowers the amount of carbon available in coking coals.

**4.3.1.2.2 Ash** The ash of a coal is the inorganic residue that remains after combustion. It should be remembered that the determined ash content is not equivalent to the mineral matter content of the coal. It does, however, represent the bulk of the mineral matter in the



**Figure 4.11** Generalised variation of capacity (or air-dried) moisture contents with rank. *Source:* From Berkowitz (1979).

coal after losing the volatile components, such as  $\text{CO}_2$ , sulfur dioxide ( $\text{SO}_2$ ), and water, which have been driven off from mineral compounds such as carbonates, sulfides, and clays.

In a steam coal, a high ash content will effectively reduce its CV. Recommended maximum ash contents for steam coals for use as pulverised fuel (pf) are around 20% (adb), but much lower values are desirable for some stoker-fired boilers. In coking coals, a maximum of 10–20% (adb) is recommended, as higher ash contents reduce the efficiency in the blast furnace.

**4.3.1.2.3 Volatile Matter** Volatile matter represents the component of the coal, except for moisture, that is liberated at high temperature in the absence of air. This material is derived chiefly from the organic fraction of the coal, but minor amounts may also be from the mineral matter present. Correction for the volatile matter derived from the latter may be made in technical works, but this is not usually necessary in commercial practice.

In pf firing for electricity generation, most boilers are designed for a minimum volatile matter of 20–25% (daf). In stoker firing for electricity generation, the volatile matter limits recommended are 25–40% (daf). There is virtually no limit for the volatile matter for coals used in the production of cement. In coke production, high volatile matter content will

give a lower coke yield, so that the best quality coking coals have a volatile matter range of 20–35% (adb), but values of 16–36% (adb) can be used.

**4.3.1.2.4 Fixed Carbon** The fixed carbon content of coal is that carbon found in the residue remaining after the volatile matter has been liberated. Fixed carbon is not determined directly, but is the difference, in an air-dried coal, between the total percentages of the other components (i.e. moisture, ash, and volatile matter) and 100%

#### 4.3.1.3 Ultimate Analysis

Ultimate analysis of coal consists of the determination of carbon and hydrogen as gaseous products of its complete combustion, the determination of sulfur, nitrogen, and ash in the material as a whole, and the estimation of oxygen by difference.

**4.3.1.3.1 Carbon and Hydrogen** These are liberated as  $\text{CO}_2$  and water when the coal is burned and are most easily determined together. However,  $\text{CO}_2$  may be liberated from any carbonate minerals present, and water may be derived from clay minerals or from any inherent moisture in the air-dried coal, or both. Allowances have to be made for these inorganic sources of carbon and hydrogen.

**4.3.1.3.2 Nitrogen** The nitrogen content of coal is significant, particularly in relation to atmospheric pollution. Upon combustion of the coal, nitrogen helps to form oxides that may be released as flue gases and thereby pollute the atmosphere; as a result, coals that are low in nitrogen are preferred in industry.

Coals should not, as a rule, have nitrogen contents of more than 1.5–2.0% (daf) because of these  $\text{NO}_x$  emissions.

**4.3.1.3.3 Sulfur** As in the case of nitrogen, the sulfur content of coals presents problems with utilisation and resultant pollution. Sulfur causes corrosion and fouling of boiler tubes

and atmospheric pollution when released in flue gases.

Sulfur can be present in coal in three forms:

- (i) organic sulfur, present in the organic compounds of the coal;
- (ii) pyritic sulfur, present as sulfide minerals in the coal, principally iron pyrite;
- (iii) sulfate minerals, usually hydrous iron or calcium sulfates, produced by oxidation of the sulfide fraction of the coal.

In the ultimate analysis of the coal, only the total sulfur content is determined. However, in many instances, the relative amount of sulfur in each form is required. This is carried out as a separate analysis.

The total sulfur content in steam coals used for electricity generation should not exceed 0.8–1.0% (adb); the maximum value will depend upon local emission regulations. In the cement industry, a total sulfur content of up to 2.0% (adb) is acceptable, but a maximum of 0.8% (adb) is required in coking coals because higher values affect the quality of steel.

**4.3.1.3.4 Oxygen** Oxygen is a component of many of the organic and inorganic compounds in coal, as well as the moisture content. When the coal is oxidised, oxygen may be present in oxides, hydroxides, and sulfate minerals, as well as oxidised organic material.

It should be remembered that oxygen is an important indicator of rank in coal.

Oxygen is traditionally determined by subtracting the amount of the other elements (carbon, hydrogen, nitrogen, and sulfur) from 100%.

#### 4.3.1.4 Other Analysis

**4.3.1.4.1 Forms of Sulfur** The proportions of organic, inorganic, and sulfate forms of sulfur are important when considering the commercial usefulness of a coal. Coal preparation can reduce the inorganic (pyritic) and sulfate fractions, but it will not reduce the organic sulfur content. Therefore, if a coal has a high sulfur content, it is essential to know if this can be

reduced by coal preparation methods; if not, then it may mean that the coal is unusable, or at best used in a blend with a low-sulfur product. Also, pyritic sulfur can be linked to liability to spontaneous combustion.

**4.3.1.4.2 Carbon Dioxide** CO<sub>2</sub> in coal occurs in the carbonate mineral matter fraction. The carbonates liberate CO<sub>2</sub> on combustion and contribute to the total carbon content of the coal. This reaction, however, reduces the amount of energy available from the coal.

**4.3.1.4.3 Chlorine** The chlorine content of coal is low, usually occurring as the inorganic salts of sodium, potassium, and calcium. The presence of relatively high amounts of chlorine in coal is detrimental to its use. In boilers, chlorine causes corrosion and fouling, and when present in flue gas it contributes to atmospheric pollution.

Steam coals should have a maximum chlorine content of 0.2–0.3% (adb), and for coals used in the production of cement, a maximum of 0.1% (adb) is recommended.

**4.3.1.4.4 Phosphorus** Phosphorus may be present in coal, usually concentrated in the mineral apatite. It is undesirable for large amounts of phosphorus to be present in coking coals to be used in the metallurgical industry, as it contributes to producing brittle steel. It is also undesirable in stoker-firing coal as it causes fouling in the boiler.

Coking coals should have a maximum phosphorus content of 0.1% (air-dried).

**4.3.1.4.5 Ash Analysis** The ash in coal represents the residue of the combusted mineral matter, and it can be broken down and expressed as the series of metal oxides aluminium oxide, titanium dioxide, calcium oxide, magnesium oxide, potassium oxide, sodium oxide, and iron(III) oxide together with phosphorus pentoxide, silica, and sulfite that make up the lithosphere. These data are important in determining how a coal will

behave, such as steam coal in boilers, where slagging and fouling can result, because the presence of large amounts of the oxides of iron, calcium, sodium, and/or potassium can result in ashes with low ash-fusion temperatures. In coking coals, sodium and potassium oxide contents should be a maximum of 3% in ash, as high alkalis cause high coke reactivity.

**4.3.1.4.6 Trace Element Analysis** Coals contain diverse amounts of trace elements in their overall composition. Those predominantly associated with the organic fraction are boron, beryllium, and germanium; those predominantly associated with the inorganic fraction include arsenic, cadmium, mercury, manganese, molybdenum, lead, zinc, and zirconium. Other trace elements have varying associations with the organic and inorganic fractions: those usually associated with the organic fraction are gallium, phosphorus, antimony, titanium, and vanadium, and those with the inorganic fraction are cobalt, chromium, nickel, and selenium.

Boron can be a useful index in indicating the palaeosalinity of the coal's depositional conditions.

Certain trace elements, such as lead, arsenic, cadmium, chromium, and mercury, if present in high amounts, could preclude the coal from being used in environmentally sensitive situations. Others have detrimental effects on the metallurgical industry; these include boron, titanium, vanadium, and zinc.

As a result of the high tonnages of coal used in industry, significant amounts of trace elements may be concentrated in residues after combustion. Therefore, trace element determinations are carried out before the coal is accepted for industrial usage.

### 4.3.2 Combustion Properties of Coal

The determination of the effects of combustion on coal will influence the selection of coals for particular industrial uses.

Tests are carried out to determine a coal's performance in a furnace, i.e. its CV and its ash fusion temperatures. In addition the caking and coking properties of coals need to be determined if the coal is intended for use in the metallurgical industry.

These parameters are particularly significant as they form the basis for the classification of coals (see Section 4.4).

#### 4.3.2.1 Calorific Value

The CV of a coal is the amount of heat per unit mass of coal when combusted, often being referred to as specific energy (SE), particularly in Australia.

The CV of a coal is expressed in two ways:

- (i) The gross CV or higher heating value. This is the amount of heat liberated during testing in a laboratory, when a coal is combusted under standardised conditions at constant volume, so that all the water in the products remains in the liquid form.
- (ii) The net CV or lower heating value. During actual combustion in furnaces, the gross CV is never achieved because some products, especially water, are lost with their associated latent heat of vaporisation. The maximum achievable CV under these conditions is the net CV at constant pressure. This can be calculated and expressed in absolute joules, calories per gram, or British thermal units (Btu) per pound. The simplified equations for these are as follows:

$$\text{MJ kg}^{-1}, \quad \text{Net CV} = \text{Gross CV} - 0.212H - 0.024M$$

$$\text{kcal kg}^{-1}, \quad \text{Net CV} = \text{Gross CV} - 50.7H - 5.83M$$

$$\text{Btu lb}^{-1}, \quad \text{Net CV} = \text{Gross CV} - 91.2H - 10.5M$$

where  $H$  is the percentage of hydrogen and  $M$  is the percentage of moisture.

As an approximate value, in bituminous coals, gross as-received CV can be converted to net as-received CV by subtracting the following values:  $1.09 \text{ MJ kg}^{-1}$ ;  $260 \text{ kcal kg}^{-1}$ ;  $470 \text{ Btu lb}^{-1}$ .

It should be noted that, in practice, the USA uses  $\text{Btu lb}^{-1}$ , and the UK has used  $\text{Btu lb}^{-1}$ , although then the British coal industry uses gigajoules per tonne (this is not used elsewhere). South Africa and Australia use megajoules per kilogram, and the rest of the world usually use kilocalories per kilogram. A conversion chart is given in Appendix C.

#### 4.3.2.2 Ash Fusion Temperatures

How the coal's ash residue reacts at high temperatures can be critical in selecting coals for combustion, i.e. how it will behave in a furnace or boiler.

A laboratory-prepared and moulded ash sample (in the shape of a cone, cube, or cylinder) is heated in a mildly reducing or oxidising atmosphere, usually to about  $1000\text{--}1600^\circ\text{C}$ .

Four critical temperature points are recognised:

- (i) Initial deformation temperature (IT, but sometimes seen as IDT), the temperature at which the first rounding of the apex or corners of the sample occurs.
- (ii) Softening (sphere) temperature (ST), the temperature at which the moulded sample has fused down to a lump, the width of which equals its height.
- (iii) Hemisphere temperature, the temperature at which the mould sample has fused down to a lump the height of which is half of its width.
- (iv) Fluid temperature (FT), the temperature at which the mould has collapsed as a flattened layer.

Temperatures recorded under a reducing atmosphere are lower or equal to those recorded under an oxidising atmosphere. The IT and FT are the most difficult to reproduce.

The behaviour of ash at high temperatures is a direct response to its chemical composition. Oxides of iron, calcium, and potassium

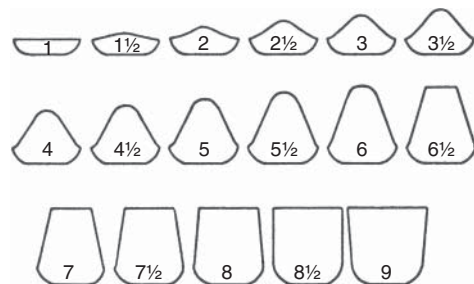
act as fluxes and reduce the temperature at which fusion occurs; high aluminium is the most refractory. In stoker boilers, a minimum IT of  $1200^\circ\text{C}$  is recommended, as lower values lead to excessive clinker formation. In pf combustion, in dry-bottom boilers a minimum IT of  $1200^\circ\text{C}$  is recommended and in wet-bottom boilers a maximum of  $1300^\circ\text{C}$  is recommended.

#### 4.3.2.3 Caking Tests

**4.3.2.3.1 Free Swelling Index** The free swelling index (FSI) in BSI nomenclature – the crucible swelling number (CSN) in International Organisation for Standardisation (ISO) nomenclature – is a measure of the increase in the volume of coal when heated, with the exclusion of air. The test is useful in evaluating coals for coking and combustion.

The coal sample is heated for a specific time. When all the volatiles have been liberated, a small coke 'button' remains. The cross-section of the button is then compared with a series of standard profiles, as in BS 1016-107.1 (1991) (see Figure 4.12).

Coals with a low swelling index (0–2) are not suitable for coke manufacture. Coals with high swelling numbers ( $\geq 8$ ) cannot be used by themselves to produce coke, as the resultant coke is usually weak and will not support the loads imposed within the blast furnace. However, they are often blended to produce strong coke.



**Figure 4.12** Characteristic profiles of coke buttons for different values of the crucible swelling number (FSI). *Source:* From BS 1016-107.1 1991. Reproduced with permission of BSI under licence number 2002SK/0003.



**4.3.2.3.2 Roga Index Test** The Roga index (RI) test again indicates the caking properties of coals. A sample of coal is combined with a standard measure of anthracite and then heated. The resultant button is then tested for mechanical strength, rather than the change in dimensions, by being rotated in a drum for a specific time.

There is a correlation between RI values and FSI values. For example, RI values 0–5 are equivalent to  $FSI = 0\text{--}1/2$ , values  $>5\text{--}20$  are equivalent to  $FSI = 1\text{--}2$ , values  $>20\text{--}45$  are equivalent to  $FSI = 2\text{--}4$ , and values  $>45$  are equivalent to  $FSI = >4$ .

#### 4.3.2.4 Coking Tests

**4.3.2.4.1 Gray–King Coke Type** Finely crushed coal is heated slowly in a sealed tube, and the appearance and texture of the coke residue is compared with standards and assigned a letter, the Gray–King coke type.

Values range from A, for no coking properties at all, to G, where the coal has retained its volume and forms a well-fused product. If it swells beyond its volume, it is said to have superior coking properties and is further tested and designated coke type G1–G8. Table 4.19 outlines the characteristics of the Gray–King coke types.

Gray–King coke types approximate to FSIs as follows: Gray–King coke type A–B is equivalent to  $FSI = 0\text{--}1/2$ , C–G2 is equivalent to  $FSI = 1\text{--}4$ , F–G4 is equivalent to  $FSI = 4\text{--}6$ , G3–G6 is equivalent to  $FSI = 6\text{--}8$ , and  $\geq G7$  is equivalent to  $FSI = 8\text{--}9$ .

**4.3.2.4.2 Fischer Assay** This test is most widely used for testing low-rank coals to low-temperature carbonisation. The percentages of coke, tar, and water driven off by the dry coal are determined; gas is calculated by subtraction.

**4.3.2.4.3 Gieseler Plastometer** To form coke, coal passes from a solid form through a fluid or plastic state to become a fused, porous solid. The temperature range of the fluid phase and

the viscosity of the fluid are important features when blending coals for coke manufacture. These parameters are measured by the Gieseler plastometer, in which a coal sample is pressed around a spindle under torque; as the coal reaches its fluid state, the spindle begins to revolve. The rate at which it turns is measured in ‘dial divisions per minute’ (d.d.p.m.), which are then plotted against temperature. Until recently, Gieseler plastometer motors were only capable of measuring to 30 000 d.d.p.m., but newer instruments can now measure up to 180 000 d.d.p.m. (Pearson 2011).

Coals with high and low fluidity may be blended to obtain improved coking properties.

**4.3.2.4.4 Audibert–Arnu Dilatometer** Coals shrink during carbonisation; such volume changes that accompany the heating of a coking coal are measured with a dilatometer. Several have been developed for this purpose, the most widely used are the Audibert–Arnu and the Ruhr dilatometers.

Dimensional changes in a coal can be measured as a function of time. While the temperature of the coal is being raised at a constant rate, curves record the length of a coal sample to define the extent of contraction and dilatation, and the temperatures at which these changes begin or end.

These properties are significant in determining the volume of coal that can be fed into a coke oven, and also in blending different coals for coke production.

The resultant coke is itself subjected to rigorous testing to confirm its strength and quality for use in commercial operations.

**4.3.2.4.5 Plastic Layer Test** Coal is heated in the absence of air and a steel needle is inserted into the coal. The amount the needle penetrates is measured and is a determination of the coking property of the coal. It is designated  $y$  and measured in millimetres. Plastic layer thickness is used along with volatile matter and vitrinite reflectance for classifying a broad range of ranks and types of coking coals. This

Table 4.19 Characteristics for classification of Gray-King coke type.

A, B, and C			D, E, and F			G	G1 to Gx		
Retains initial cross-section			Shrunken			Retains initial volume	Swollen		
Examine for strength			Examine for strength			Examine for strength	Examine for degree of swelling		
Non-coherent	Badly coherent	Coherent	Moderately hard and shrunken	Hard and very shrunken	Hard, strong, and shrunken	Hard and strong	Slightly swollen	Moderately swollen	Highly swollen
Usually in powder form but may contain some pieces which, however, cannot be handled without breaking.	In several pieces and some loose powder. Pieces can be picked up but break into powder on handling	Usually in one piece but easily broken; may be in two pieces with practically no loose powder; very friable and dull.	May be fissured but can be scratched with fingernail and stains the fingers on rubbing the curved surface vigorously; usually dull and black and appearing fritted rather than fused.	Usually very fissured, moderate metallic ring when tapped on a hard wooden surface, does not stain the fingers on rubbing; grey or black with slight lustre.	May be fissured; moderate metallic ring when tapped on a hard wooden surface; does not stain the fingers on rubbing. Cross-section well fused and greyish.	Well fused with a good metallic ring when tapped on a hard wooden surface.			G3 and higher. Guided by swelling number, blend with minimum number of parts of electrode carbon to give a standard G-type coke.
A	B	C	D	E	F	G	G1	G2	Gx

Source: From BS 1016-107.2 (1991). Reproduced with permission of BSI under licence Number 2002SK/0003.

test is widely used in the People's Republic of China (PRC), Russia, and other eastern European countries. The test was known as the Sapozhnikov plastometric test and first developed by Sapozhnikov and Bazilevich (1938).

Recent work by Pearson et al. (2017a,b) has identified blends at the plastic layer produced from the earliest melting vitrinites which engulf and thermally isolate unmelted coal grains that possess a higher melting temperature. This engulfing fluid hardens to a semi-coke envelope within which the isolated material subsequently melts and develops an internal high-pressure suite. Such examples are known as 'encapsulites'. Encapsulites have an effect on the coke-forming capacity of blended coals which in recent years has only been predicted on single coals using petrographic analysis and then applied to coal blends.

**4.3.2.4.6 The G Value Test** The PRC uses a variation of the Roga test, known as the *G* value test. A sample of coal is ground to a fine size and mixed with a standard anthracite. A coke is produced and the resulting button is rotated in a cylinder for a defined number of rotations. The cylinder contents are screened and the amount of coal remaining on the sieve is the *G* value of the coal. Values over 65 are considered good, with values less than 30 being considered non-coking (GB/T 5447-2014 2014).

**4.3.2.4.7 Vitrinite Fluorescence** Recent studies have shown that Gieseler fluidity correlates with vitrinite fluorescence and is independent of coal rank and coal type (Pearson 2011). Different populations of vitrinite possess different levels of Gieseler fluidity and fluorescence intensity. This method can identify single-sourced coals and blended coals by the difference in the vitrinite populations (Figure 4.13).

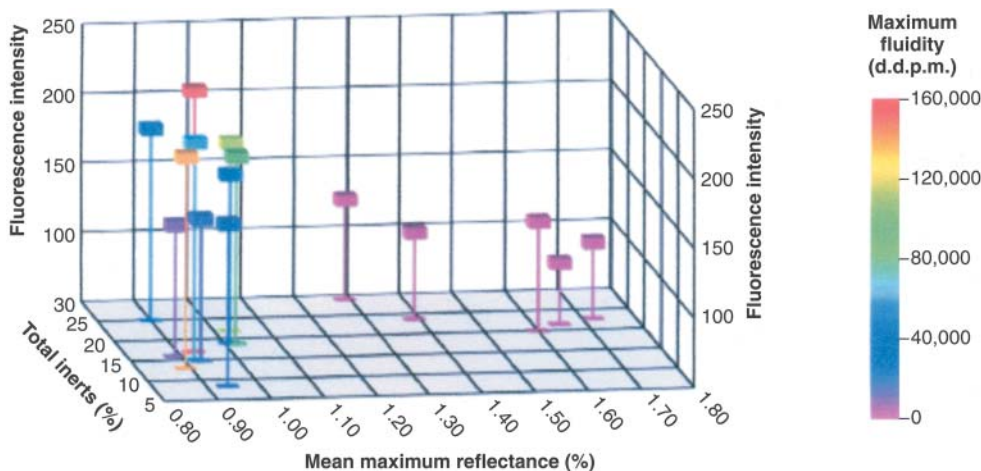
### 4.3.3 Physical Properties of Coal

In addition to the chemical and combustion properties of a coal, its evaluation for commercial usage requires the determination of several physical properties. These are the coal's strength, density, hardness, grindability, abrasiveness, size distribution, and float-sink tests.

#### 4.3.3.1 Mechanical Strength

The mechanical strength of coal refers to its capacity to resist external force and is related to factors such as degree of coalification, lithotype, mineral content, and weathering. The mechanical strengths of high- and low-rank coals are greater than those of medium-rank coals and coking coals. The mechanical strengths of coals with high mineral contents are high and are reduced by weathering.

Mechanical strength can be determined by the drop test. This method uses coal lumps



**Figure 4.13** Relationship between coal rank, coal type, and fluorescence. *Source:* From Pearson (2011) with permission.

**Table 4.20** Grading standards for mechanical strength of coal (Xie 2015).

Grade	Mechanical strength of coal	Proportion of particles >25 mm (%)
Grade I	High-strength coal	>65
Grade II	Medium-strength coal	50–65
Grade III	Low-strength coal	30–50
Grade IV	Ultralow-strength coal	<30

of size 60–100 mm that are allowed to fall freely from a point 2 m above a steel plate. The coal is sieved using a 25 mm sieve and the process is repeated three times. The mass of coal sample larger than 25 mm is determined and the percentage with respect to the mass of the original coal sample is taken as the shatter indices of the coal (Xie 2015). Table 4.20 gives the grading standards for mechanical strength of coal.

#### 4.3.3.2 Density

The density of a coal will depend on its rank and mineral matter content. It is an essential factor in converting coal units of volume into units of mass for coal reserves calculation.

Density is determined by the loss of weight incurred when immersed in water. The testing of field samples and core samples in this way gives ‘apparent density’, because air remains

trapped within the coal. True density is determined by crushing the coal and using a standard density bottle or pycnometer. The ease with which apparent densities can be determined in the field is an important facility available to the geologist when describing coal types whose mineral matter contents may fluctuate up to levels where the coal could become uneconomic on quality grounds.

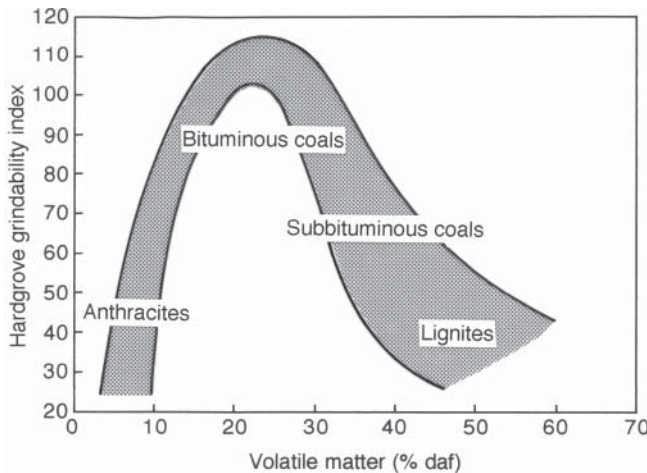
It should be noted that density is not synonymous with specific gravity or relative density; the former is defined as the weight per unit volume given as grams per cubic centimetre, whereas specific gravity or relative density is its density with reference to water at 4 °C.

#### 4.3.3.3 Hardness and Grindability

In modern commercial operations, coals are required to be crushed to a fine powder (pulverised) before being fed into a boiler. The relative ease with which a coal can be pulverised depends on the strength of the coal and is measured by the Hardgrove grindability index (HGI). This is an index of how easily a coal can be pulverised in comparison with coals chosen as standards.

Coals with a high HGI are relatively soft and easy to grind. Those coals with low HGI values (less than 50) are hard and difficult to grind into a pulverised product.

HGI varies with coal rank, as shown in Figure 4.14 (Berkowitz 1979).



**Figure 4.14** Generalised variation of the Hardgrove grindability index with rank. Source: From Berkowitz (1979).

#### 4.3.3.4 Abrasion Index

Coarse mineral matter in coal, particularly quartz, can cause serious abrasion of machinery used to pulverise coal. Coal samples are tested in a mill equipped with four metal blades. The loss in mass of these blades determines the 'abrasion index', which is expressed as milligrams of metal per kilogram of coal used.

#### 4.3.3.5 Particle Size Distribution

Size distribution in a coal depends on the mining and handling it undergoes, together with its hardness, strength, and its inherent degree of fracturing.

The size of coal particles affects coal preparation plant design, which in turn is related to the sized product to be sold. Tests are based on sieve analysis as for other geological materials, and the results expressed in various size-distribution parameters, such as mean particle size and cumulative size percentages.

#### 4.3.3.6 Float-Sink Tests

The particles in coal are of different relative densities. The densities represent the

varying amounts of mineral matter present. Consequently, the coal preparation process is designed to remove these (so that the ash level of the coal is reduced) in order to improve the product to be used or sold.

Coal particles are separated into density fractions by immersion in a series of liquids of known relative density, usually ranging from 1.30 to 2.00. Commencing with the lowest relative density, the sinking fraction is transferred to the next liquid in the series, and so on. An example of a float-sink analysis is shown in Table 4.21 (S.C. Frankland, personal communication). Using the results given in Table 4.21, these may be plotted graphically as a series of 'washability curves'. These are used to calculate the amount of coal that can be obtained at a particular quality, the density required to effect such a separation, and the quality of the discard left behind.

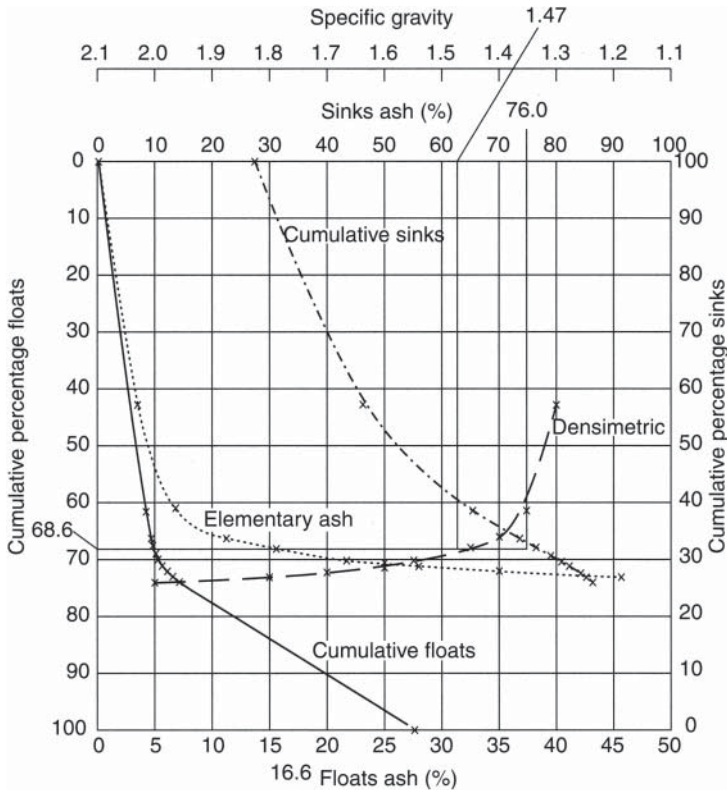
The curves shown in Figure 4.15 are the classical washability curves: the cumulative floats curve, which plots column I values against column G; the densimetric curve, which plots column G against column C; the cumulative sinks

**Table 4.21** Washability data.

Relative density			Fractional			Cumulative floats			Cumulative sinks		
A	B	C	D	E	F	G	H	I	J	K	L
Sink RD	Float RD	Midpoint RD $\frac{A+B}{2}$	Weight %	Ash %	Ash points $\frac{D \times E}{100}$	Weight % $\Sigma D \downarrow$	Ash points $\Sigma F \downarrow$	Ash % $\frac{H}{G} \times 100$	Weight % $\Sigma D \uparrow$	Ash points $\Sigma F \uparrow$	Ash % $\frac{K}{J} \times 100$
—	1.30		43.31	3.10	1.34	43.31	1.34	3.10	100.00	27.66	27.66
1.30	1.35	1.325	18.47	6.61	1.22	61.78	2.56	4.15	56.69	26.31	46.42
1.35	1.40	1.375	4.91	11.11	0.55	66.69	3.11	4.66	38.22	25.09	65.66
1.40	1.45	1.425	1.41	15.26	0.22	68.10	3.32	4.88	33.31	24.55	73.70
1.45	1.50	1.475	1.45	19.04	0.28	69.55	3.60	5.18	31.90	24.33	76.28
1.50	1.55	1.525	1.04	21.69	0.23	70.59	3.83	5.42	30.45	24.06	79.00
1.55	1.60	1.575	0.77	28.08	0.22	71.36	4.04	5.66	29.41	23.83	81.03
1.60	1.70	1.650	1.07	34.70	0.37	72.43	4.41	6.09	28.64	23.62	82.46
1.70	1.80	1.750	0.68	45.85	0.31	73.11	4.73	6.46	27.57	23.24	84.31
1.80	2.00	1.900	1.02	55.38	0.56	74.13	5.29	7.14	26.89	22.93	85.28
2.00	—		25.87	86.46	22.37	100.00	27.66	27.66	25.87	22.37	86.46

Source: Reproduced by courtesy of Dargo Associates Ltd.





**Figure 4.15** Washability curves based on data given in Table 4.18. *Source:* Reproduced by courtesy of S.C. Frankland, Dargo Associates Ltd.

curve, which plots column L against column J; and the elementary ash curve, which plots column G against column E (see Table 4.21).

Quantitatively, an examination of the cumulative floats curve will give yield values for a given quality, and the densimetric curve will indicate the density at which to wash (i.e. washing density) in order to obtain that yield and quality. This can also be calculated in reverse.

The curves can also be used on a more qualitative basis. For example, if the density value that is required is on the steep part of the densimetric curve then it will be more difficult to maintain a consistent quality.

The significance of this is that the amounts of coal and mineral matter or discard can be determined for a specific relative density, so enabling a product of specified ash content to be produced using liquids of known relative density.

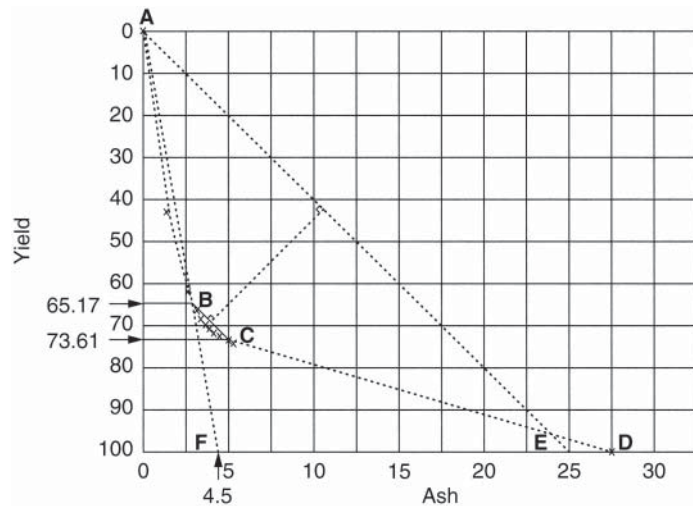
For example, in Figure 4.15, a coal with an ash content of 5% will give a yield of 68.6%, and

a density of 1.47 will be needed to achieve this. The ash of the sinks (reject) will be 76% and the percentage of those ash particles in the floats will be 16.6%. The latter figure is useful to the coal preparation engineer for coal blending calculations.

Sometimes the coal is cleaned to produce two products, a prime product and a lower quality product (the so called 'middlings'), plus a discard. Classical washability curves cannot be used to calculate yield or quality of middlings. In order to determine these values, an M curve is used (M = Mayer, middlings, or mean-value curve), as shown in Figure 4.16. The M curve is produced by plotting column G against column I (see Table 4.21). The angle of a line drawn from point A to intersect the abscissa represents the ash value, and the value of the ordinate represents the yield.

For example, in Figure 4.16, to calculate the yield of a prime product of 4.5% ash, a line is drawn from A to intersect the ash axis at

**Figure 4.16** Middlings or M curve. *Source:* Reproduced by courtesy of S.C. Frankland, Dargo Associates Ltd.



4.5% (F); where this line crosses the M curve at B, a yield of 65.17% can be read off from the yield axis. To calculate the yield of a 25% ash middling, a line is drawn from A to intersect the ash axis at 25% (E). A line is drawn from B parallel to the 25% ash line A-E to intersect the M curve at C. This gives a total yield of 4.5% ash prime product and 25% ash middlings of 73.61%. Therefore, the yield of 25% ash middlings is  $73.61 - 65.17\% = 8.44\%$ .

The densimetric curve may also be drawn on the M curve and used in an identical way to the classical washability curves. Intersects of the densimetric curve with all lines drawn from the yield axis give densities of separation.

#### 4.3.4 Coal Oxidation

Exposure of coals to weathering in the atmosphere, or by oxygenated groundwaters, results in the oxidation of the organic and inorganic constituents of the coal. Oxidation reduces the coal quality by altering the chemical and physical properties of coal. In particular, the CV is lowered and caking is eliminated. There is also a loss of floatability during washing of the coal.

The weathering of coal results in its physical breakdown to fine particles, which enhances hydration and hydrolysis. If the coal is structurally fractured, the extent of oxidation will be greater.

The degree of oxidation is determined by the maceral and mineral matter content. Vitrinite is considered by some to be the most readily oxidised maceral; however, Gondwana coals high in inertinite have a high propensity for spontaneous combustion, which would indicate a rapid oxidation of the inertinite. In addition, pyrite and other sulfides readily oxidise to sulfates.

All ranks of coal are affected by oxidation, and the degree to which this may occur is influenced by the coal rank, pyrite content, climate, hydrology, and by the surface area within the coal accessible to oxidation.

It is extremely important to establish how much of a coal deposit has been oxidised. The oxidised coal may well be excluded from the tonnage produced.

One direct side effect of oxidation is that of spontaneous combustion. This occurs when the rate of heat generation by oxidation exceeds the rate of heat dissipation. All coals have the propensity to heat spontaneously, but lower rank coals have a greater tendency to self-heat. When the temperature of the coal is raised, the rate of oxidation is also increased; it is suggested that the oxidation rate doubles for every  $10^{\circ}\text{C}$  rise in temperature at least up to  $100^{\circ}\text{C}$ . It has also been demonstrated that low-rank coal produces heat when wetted, and

that reactivity is increased 10-fold if dispersed pyrite is present.

Where coals possessing some or all of these properties are stockpiled or loaded into vessels, tests and monitoring are rigorously carried out. Procedures carried out to lessen heating effects include compaction of the coal, which reduces the oxidation rate, and protection of the coal from heat sources, such as solar radiation.

Spontaneous combustion is also a hazard to underground mining. Oxidation of in-situ coals and coal dust particles produces a potential danger. Factors contributing to the possibility of combustion are if the coal is thicker than its mined section, steep dips, faulting, and coal outbursts. Where workings are deep, the natural strata temperature is higher, and therefore so will be the base temperature of the in-situ coal. Care in mine design and careful monitoring are needed in these circumstances to minimise heating effects. Potential fires or explosions are costly in terms of labour, materials, and time, with a corresponding loss in production. This is particularly true if an area of mine has to be abandoned and sealed off through spontaneous combustion, so losing the potential reserves of coal in that area.

## 4.4 Classification of Coals

Coals have usually been classified according to their chemical properties in relation to their industrial usage.

Several classifications are in common usage, which classify both humic and brown coals, and refer to particular parameters; these include the percentage of fixed carbon and volatile matter (on a dmmf basis), the CV (on a moist, mineral-matter-free basis, maf – equivalent to daf), the caking properties of coal (FSI and RI), and the coking properties of coal (dilatometer and Gray–King tests).

Coals have been classified either for ‘scientific’ purposes or for coal use. The scientific classifications use carbon/oxygen or carbon/hydrogen correlations; of these, the best known

is that of Seyler (1931, 1938) (Figure 4.17). This classification, however, is applicable only to British Carboniferous coals and takes little account of lower rank coals. It uses the terms ‘perhydrous’ for hydrogen-rich material and ‘subhydrous’ for hydrogen-poor samples; these prefixes plus terms for each rank are given in Table 4.22.

The principal commercial classifications of coal in current use are those discussed in the following sections.

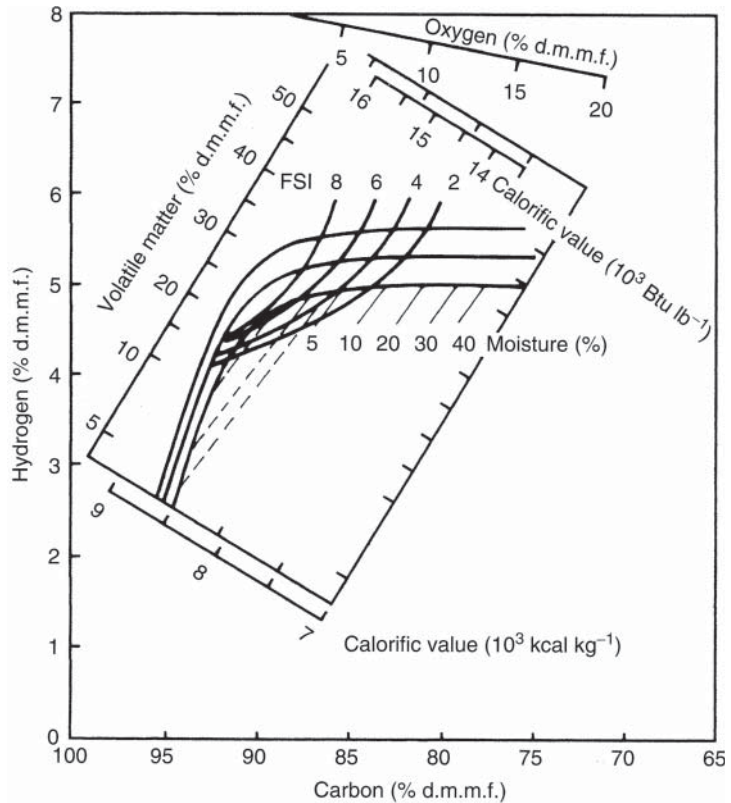
### 4.4.1 North America

The ASTM classification D 388-99 (now updated to D 388-17A) is used in the USA and Canada as well as on a worldwide basis (Appendix A). This is based on two coal properties: the fixed carbon values and the CVs (on a dmmf basis). The higher rank coals are classified according to fixed carbon on a dry basis (db), the lower rank coals are classified according to gross CV on a moist basis. A correlation of the ranks property and volatile matter with the mean maximum reflectance group of macerals (D 2798-09, now updated as D2798-11) is used as supplemental information. Further classification is given for those coals with agglomerating or coking properties; see Table 4.23. The classification is applicable to coals composed mainly of vitrinite, so that coals rich in inertinite or liptinite (exinite) cannot be properly classified because the properties that determine rank differ greatly between these maceral groups. In North America, such coals are mostly non-banded varieties containing only a small proportion of vitrinite and consist mainly of attrital materials.

### 4.4.2 United Kingdom

The British classification system was devised by British Coal before privatisation and is shown in Table 4.24. It uses a three-figure numeral code to classify bituminous and anthracite coals. The first two digits are based on the amount of volatile matter in the coal

**Figure 4.17** Seyler's coal chart. This version shows relationships between elemental composition, volatile matter contents, moisture contents, and caking properties. *Source:* From Berkowitz (1979), based on Seyler (1899, 1931).



(on dmmf basis) and the third digit is based on the Gray–King assay value. Coals with less than 19.6% volatile matter (dmmf) are classified by this property alone. It should be noted that coals with ash contents greater than 10% must be cleaned prior to analysis. Coals that have been thermally altered by igneous intrusions have the suffix H added to the coal code, and coals that have been oxidised by weathering may be distinguished by adding the suffix W to the coal code.

#### 4.4.3 Europe

In Europe, in 1988, the codification system for medium- and high-rank coals was published by the United Nations Economic Commission for Europe (UNECE) and approved by the ISO. The codification system uses a series of numbers to illustrate the chemical and physical characteristics that determine the usage of the coal. It does not include low-rank coals,

which are defined as coals with a gross CV <24.0 MJ kg<sup>-1</sup> (5700 kcal kg<sup>-1</sup>) and a mean random reflectance  $R_r$  <0.6% (Figure 4.18).

The codification system is applicable to both run-of-mine and washed coals. The coals are characterised by using a 14-digit code number based on eight property-related parameters providing information concerning rank, type, and grade of a coal (Table 4.25). In addition, a list of 'supplementary parameters' is used where appropriate, e.g. chlorine content, ash fusion temperature.

The code is based on:

1. Mean  $R_r$  as two digits. Codes 2–50 cover  $R_r$  values from 0.20 to >5.00.
2. Description of a reflectogram; the third digit covering codes 0–5, dependent on whether the coal is a single seam or a blend.
3. Maceral group composition; the fourth digit provides the lower limit of a 10% range of the inertinite content, and the fifth digit

Table 4.22 Parameters used in Seyler's coal classification.

Genus	Hydrogen (%)	Class (% carbon)						
		Anthracite (>93.3)	Carbonaceous (93.3–91.2)	Bituminous			Lignitous	
				Meta- (91.2–89.0)	Ortho- (89.0–87.0)	Para- (87.0–84.0)	Meta- (84–85)	Ortho- (80–75)
Per-bituminous	>5.8	—	—	Per-bituminous (per-meta-bituminous)	Per-bituminous (per-ortho-bituminous)	Per-bituminous (per-para-bituminous)	Per-lignitous	
Bituminous	5–5.8	—	Pseudo-bituminous species	Meta-bituminous	Ortho-bituminous	Para-bituminous	Lignitous (meta, ortho)	
Semi-bituminous	4.5–5.0	—	Semi-bituminous species (ortho-semi-bituminous)	Subbituminous (sub-meta-bituminous)	Subbituminous (sub-ortho-bituminous)	Subbituminous	Sub-lignitous (meta, ortho)	
Carbonaceous	4.0–4.5	Semi-anthracitic species Dry steam coal	Carbonaceous species (ortho-carbonaceous)	Pseudo-carbonaceous (sub-meta-bituminous)	Pseudo-carbonaceous (sub-ortho-bituminous)	Pseudo-carbonaceous (sub-para-bituminous)		
Anthracitic	<4	Ortho-anthracite True anthracite	Pseudo-anthracite (sub-carbonaceous)	Pseudo-anthracite (sub-meta-bituminous)	Pseudo-anthracite (sub-ortho-bituminous)	Pseudo-anthracite (sub-para-bituminous)		

Source: From Ward (1984) with permission of Blackwell Scientific Publications.



Table 4.23 ASTM classification of coals by rank<sup>a</sup>.

Class/group	Fixed carbon limits (dry, mineral-matter-free basis), %		Volatile matter limits (dry, mineral-matter-free basis), %		Gross CV limits (moist, <sup>b</sup> mineral-matter-free basis)				Agglomerating character
					Btu lb <sup>-1</sup>		MJ kg <sup>-1</sup> c		
	Equal or greater than	Less than	Greater than	Equal or less than	Equal or greater than	less than	Equal or greater than	Less than	
<b>Anthracitic</b>									
Meta-anthracite	98	—	—	2	—	—	—	—	} Non-agglomerating
Anthracite	92	98	2	8	—	—	—	—	
Semi-anthracite <sup>d</sup>	86	92	8	14	—	—	—	—	
<b>Bituminous</b>									
Low-volatile bituminous coal	78	86	14	22	—	—	—	—	} Commonly agglomerating <sup>f</sup> Agglomerating
Medium-volatile bituminous coal	69	78	22	31	—	—	—	—	
High-volatile A bituminous coal	—	69	31	—	14 000 <sup>e</sup>	—	32.6	—	
High-volatile B bituminous coal	—	—	—	—	13 000 <sup>e</sup>	14 000	30.2	32.6	
High-volatile C bituminous coal	—	—	—	—	11 500	13 000	26.7	30.2	
					10 500	11 500	24.4	26.7	
<b>Subbituminous</b>									
Subbituminous A coal	—	—	—	—	10 500	11 500	24.4	26.7	} Non-agglomerating
Subbituminous B coal	—	—	—	—	9 500	10 500	22.1	24.4	
Subbituminous C coal	—	—	—	—	8 300	9 500	19.3	22.1	
<b>Lignitic</b>									
Lignite A	—	—	—	—	6 300 <sup>g</sup>	8 300	14.7	19.3	}
Lignite B	—	—	—	—	—	6 300	—	14.7	

a) This classification does not apply to certain coals.

b) Moist refers to coal containing its natural inherent moisture but not including visible water on the surface of the coal.

c) To convert British thermal units per pound to megajoules per kilogram, multiply by 0.002326

d) If agglomerating, classify in low-volatile group of the bituminous class.

e) Coals having 69% or more fixed carbon on the dry, mineral-matter-free basis shall be classified according to fixed carbon, regardless of gross CV.

f) It is recognized that there may be non-agglomerating varieties in these groups of the bituminous class, and that there are notable exceptions in the high-volatile C bituminous group.

g) Editorially corrected.

Source: Reprinted with permission from ASTM D388-1999.

**Table 4.24** Coal classification system used by British Coal (revision of 1964).

Main class(es)	Coal rank code		Volatile matter (%, dmmf)	Gray–King coke type <sup>a</sup>	General description
	Class	Subclass			
100			Under 9.1	A	} Anthracites
	101 <sup>b</sup>		Under 6.1	} A	
	102 <sup>b</sup>		6.1–9.0		
200			9.1–19.5	A–G8	} Low-volatile steam coals
	201		9.1–13.5	A–C	
		201a	9.1–11.5	A–B	} Dry steam coals
		201b	11.6–13.5	B–C	
	202		13.6–15.0	B–G	} Coking steam coals
	203		15.1–17.0	E–G4	
	204		17.1–19.5	G1–G8	
300			19.6–32.0	A–G9 and over	} Medium-volatile coals
	301		19.6–32.0	G4 and over	
		301a	19.6–27.5	} G4 and over	} Prime coking coals
		301b	27.6–32.0		
	302		19.6–32.0	G–G3	} Medium volatile, medium-caking, or weakly caking coals
	303		19.6–32.0	A–F	
400–900			Over 32.0	A–G9 and over	} High-volatile coals
400			Over 32.0	G9 and over	
	401		32.1–36.0	} G9 and over	} High volatile, very strongly caking coals
	402		Over 36.0		
500			Over 32.0	G5–G8	} High volatile, strongly caking coals
	501		32.1–36.0	} G5–G8	
	502		Over 36.0		
600			Over 32.0	G1–G4	} High volatile, medium-caking coals
	601		32.1–36.0	} G1–G4	
	602		Over 36.0		
700			Over 32.0	E–G	} High volatile, weakly caking coals
	701		32.1–36.0	} E–G	
	702		Over 36.0		
800			Over 32.0	C–D	} High volatile, very weakly caking coals
	801		32.1–36.0	} C–D	
	802		Over 36.0		

*(continued)*

Table 4.24 (Continued)

Coal rank code			Volatile matter (%, dmmf)	Gray-King coke type <sup>a</sup>	General description
Main class(es)	Class	Subclass			
900			Over 32.0	A-B	} High volatile, non-caking coals
		901	32.1-36.0	} A-B	
		902	Over 36.0		

Notes:

- (1) Coals that have been affected by igneous intrusions ('heat-altered' coals) occur mainly in classes 100, 200, and 300, and when recognised should be distinguished by adding the suffix H to the coal rank code, e.g. 102H, 201bH.
  - (2) Coals that have been oxidised by weathering may occur in any class and when recognised should be distinguished by adding the suffix W to the coal rank code, e.g. 801W.
  - a) Coals with volatile matter of less than 19.6% are classified by using the parameter of volatile matter alone; the Gray-King coke types quoted for these coals indicate the general ranges found in practice and are not criteria for classification.
  - b) To divide anthracites into two classes, it is sometimes convenient to use a hydrogen content of 3.35% (dmmf) instead of a volatile matter of 6.0% as the limiting criterion. In the original Coal Survey rank coding system the anthracites were divided into four classes then designated 101, 102, 103, and 104. Although the present division into two classes satisfies most requirements, it may sometimes be necessary to recognise more than two classes.
- Source: Reproduced by permission of British Coal Corporation.

indicates the upper limit of a 5% range of the liptinite content.

4. The sixth digit indicates the CSN in terms of the lower limit of two half-step numbers.
5. The seventh and eighth digits indicate the lower limit of a 2% range of volatile matter down to 10% (daf) and a 1% range when volatile matter <10%.
6. The ninth and tenth digits indicate the lower limit of 1% range of the ash (db).
7. The 11th and 12th digits indicate the lower limit of a 0.10% range of the total sulfur content (db) multiplied by 10.
8. The 13th and 14th digits indicate the lower limit of 1 MJ kg<sup>-1</sup> range of the gross CV (daf).

All of the testing is in accordance with ISO standards as listed in Appendix A.

An example of the codification system when applied to an Australian coal is as follows:

		Digit no.
$R_r$	1.25	12
Reflectogram	$s = 0.14$ , no gap	1
Maceral composition		1
Inertinite	51	5
Liptinite	3	1
FSI	6½	6
Volatile matter (% daf)	24.3	24
Ash content (% db)	5.53	05
Total sulfur content (% db)	0.42	04
Gross CV (MJ kg <sup>-1</sup> , daf)	35.9	35
Code no.	12 1 51 6 24 05 04 35	

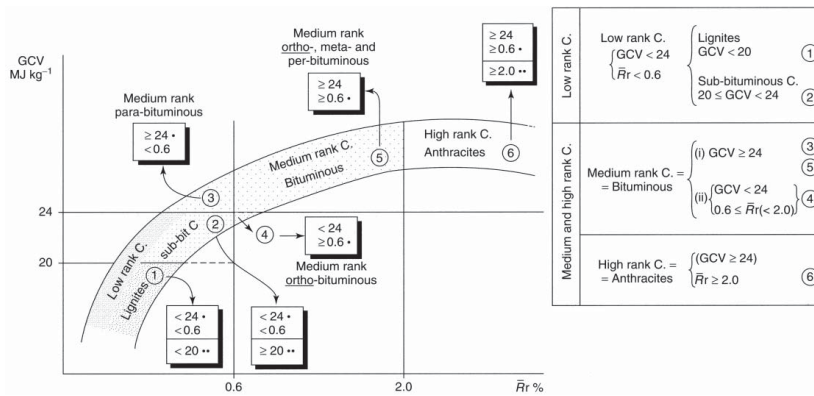


Figure 4.18 UNECE main coal classification categories, defined by gross CV ( $\text{MJ kg}^{-1}$  maf) and vitrinite reflectance percentage in oil ( $\bar{R}r$ , %). Source: Reproduced with permission of UNECE.

Table 4.25 Codification system used by UNECE (1988) for medium- and high-rank coals.

Digit	Random reflectance of vitrinite (%)		Characteristics of reflectogram	Maceral group composition index (vol.%, mmf)		Swelling crucible number	Volatile matter mass (% daf)		Ash mass (% db)	Total sulfur mass (% db)		Gross CV (MJ kg <sup>-1</sup> , daf)					
	1;2	3		4	5		6	7;8		9;10	11;12	13;14					
Code no.	02	0.20-0.29	0	0	0- $<10$	0	Exempted	0	0- $\frac{1}{2}$	48	$\geq 48$	00	0- $<1$	00	0.0- $<0.1$	21	$<22$
	03	0.30-0.39	1	1	10- $<20$	1	0- $<5$	1	1- $1\frac{1}{2}$	46	46- $<48$	01	0- $<2$	01	0.1- $<0.2$	22	22- $<23$
	04	0.40-0.49	2	2	20- $<30$	2	5- $<10$	2	2- $2\frac{1}{2}$	44	44- $<46$	02	2- $<3$	02	0.2- $<0.3$	23	23- $<24$
	05	0.50-0.59	3	3	30- $<40$	3	10- $<15$	3	3- $3\frac{1}{2}$	42	42- $<44$	03	3- $<4$	03	0.3- $<0.4$	24	24- $<25$
	06	0.60-0.69	4	4	40- $<50$	4	15- $<20$	4	4- $4\frac{1}{2}$	40	40- $<42$	04	4- $<5$	04	0.4- $<0.5$	25	25- $<26$
	07	0.70-0.79	5	5	50- $<60$	5	20- $<25$	5	5- $5\frac{1}{2}$	38	38- $<40$	05	5- $<6$	05	0.5- $<0.6$	26	26- $<27$
	08	0.80-0.89		6	60- $<70$	6	25- $<30$	6	6- $6\frac{1}{2}$	36	36- $<38$	06	6- $<7$	06	0.6- $<0.7$	27	27- $<28$
	09	0.90-0.99		7	70- $<80$	7	30- $<35$	7	7- $7\frac{1}{2}$	34	34- $<36$	07	7- $<8$	07	0.7- $<0.8$	28	28- $<29$
	10	1.00-1.09		8	80- $<90$	8	35- $<40$	8	8- $8\frac{1}{2}$	32	32- $<34$	08	8- $<9$	08	0.8- $<0.9$	29	29- $<30$
	11	1.10-1.19		9	$\geq 90$	9	$\geq 40$	9	9- $9\frac{1}{2}$	30	30- $<32$	09	9- $<10$	09	0.9- $<1.0$	30	30- $<31$
	12	1.20-1.29							28	28- $<30$	10	10- $<11$	10	1.0- $<1.1$	31	31- $<32$	
	13	1.30-1.39							26	26- $<28$	11	11- $<12$	11	1.1- $<1.2$	32	32- $<33$	
	14	1.40-1.49							24	24- $<26$	12	12- $<13$	12	1.2- $<1.3$	33	33- $<34$	
	15	1.50-1.59							22	22- $<24$	13	13- $<14$	13	1.3- $<1.4$	34	34- $<35$	
	16	1.60-1.69							20	20- $<22$	14	14- $<15$	14	1.4- $<1.5$	35	35- $<36$	
	17	1.70-1.79							18	18- $<20$	15	15- $<16$	15	1.5- $<1.6$	36	36- $<37$	
	18	1.80-1.89							16	16- $<18$	16	16- $<17$	16	1.6- $<1.7$	37	37- $<38$	
	19	1.90-1.99							14	14- $<16$	17	17- $<18$	17	1.7- $<1.8$	38	38- $<39$	
	20	2.00-2.09							12	12- $<14$	18	18- $<19$	18	1.8- $<1.9$	39	$\geq 39$	

(continued)



Table 4.25 (Continued)

Digit	Random reflectance of vitrinite (%)	Characteristics of reflectogram	Maceral group composition index (vol%, mmf) 4 = inertinite; 5 = liptinite		Swelling crucible number	Volatile matter mass (% daf)		Ash mass (% db)		Total sulfur mass (% db)	Gross CV (MJ kg <sup>-1</sup> daf)
	1;2	3	4	5	6	7;8	9;10	11;12	13;14		
21	2.10-2.19					10	10-<12	19	19-<20	19	1.9-<2.0
22	2.20-2.29					09	9-<10	20	20-<21	20	2.0-<2.1
23	2.30-2.39					08	8-<9			21	2.1-<2.2
24	2.40-2.49					07	7-<8			22	2.2-<2.3
25	2.50-2.59					06	6-<7			23	2.3-<2.4
26	2.60-2.69					05	5-<6			24	2.4-<2.5
27	2.70-2.79					04	4-<5			25	2.5-<2.6
28	2.80-2.89					03	3-<4			26	2.6-<2.7
29	2.90-2.99					02	2-<3			27	2.7-<2.8
30	3.00-3.09					01	1-<2			28	2.8-<2.9
50	>5.00										

Source: Reproduced with permission of UNECE.

**Table 4.26** Australian classification of hard coal.

	Value	Volatile matter (%, dmmf)	Gross calorific value (MJ kg <sup>-1</sup> , daf)
1st digit (coal class)	1	<10.0	
	2	10.1–14.0	
	3	14.1–20.0	
	4A	20.1–24.0	
	4B	24.1–28.0	
	5	28.1–33.0	
	6	33–41 <sup>a</sup>	>33.82
	7	33–44 <sup>a</sup>	32.02–33.82
	8	35–50 <sup>a</sup>	28.43–32.02
9	42–50 <sup>a</sup>	27.08–28.42	
	Value	Crucible swelling no.	
2nd digit (coal group)	0	0–1/2	
	1	1–2	
	2	2 1/2–4	
	3	4 1/2–6	
	4	6 1/2–9	
	Value	Gray-King coke type	
3rd digit (coal sub-group)	0		
	1	A	
	2	B–D	
	3	E–G	
	4	G1–G4	
	5	G5–G8 G9–	
	Value	Ash (dry basis %)	
4th digit (ash number)	(0)	<4.0	
	(1)	4.1–8.0	
	(2)	8.1–12.0	
	(3)	12.1–16.0	
	(4)	16.1–20.0	
	(5)	20.1–24.0	
	(6)	24.1–28.0	
	(7)	28.1–32.0	
	(8)	>32.0	

a) Values for information only.

Source: From Ward (1984), based on Australian Standard 2096-1987.

#### 4.4.4 Australia

The Australian standard coal classification for hard coals (AS 2096-1987) again assigns a multi-digit number to determine coal type. The first digit represents volatile matter for coals with less than 33% volatile matter (dmmf) and gross CV (daf) for other coals. The second digit is the FSI of the coal, the third digit is the Gray-King assay value, and the fourth digit (given in parentheses) is based on the ash content (db) of the coal; see Table 4.26.

#### 4.4.5 South Africa

In South Africa, coals are divided for commercial purposes into three broad classes on the basis of volatile matter (daf). These are South African anthracite, semi-anthracite, and steam coal. Coals of each class are graded on the basis of CV (ad), ash (ad), and ash fusibility.

#### 4.4.6 United Nations

Because of the wide spectrum of criteria used to define the boundary between 'brown' coal and high-rank coals, plus the variation in brown colour, the UNECE decided to abandon the term 'brown' coal, and devised a new classification and codification system for low-rank coals (UNECE 2002); see Tables 4.27 and 4.28.

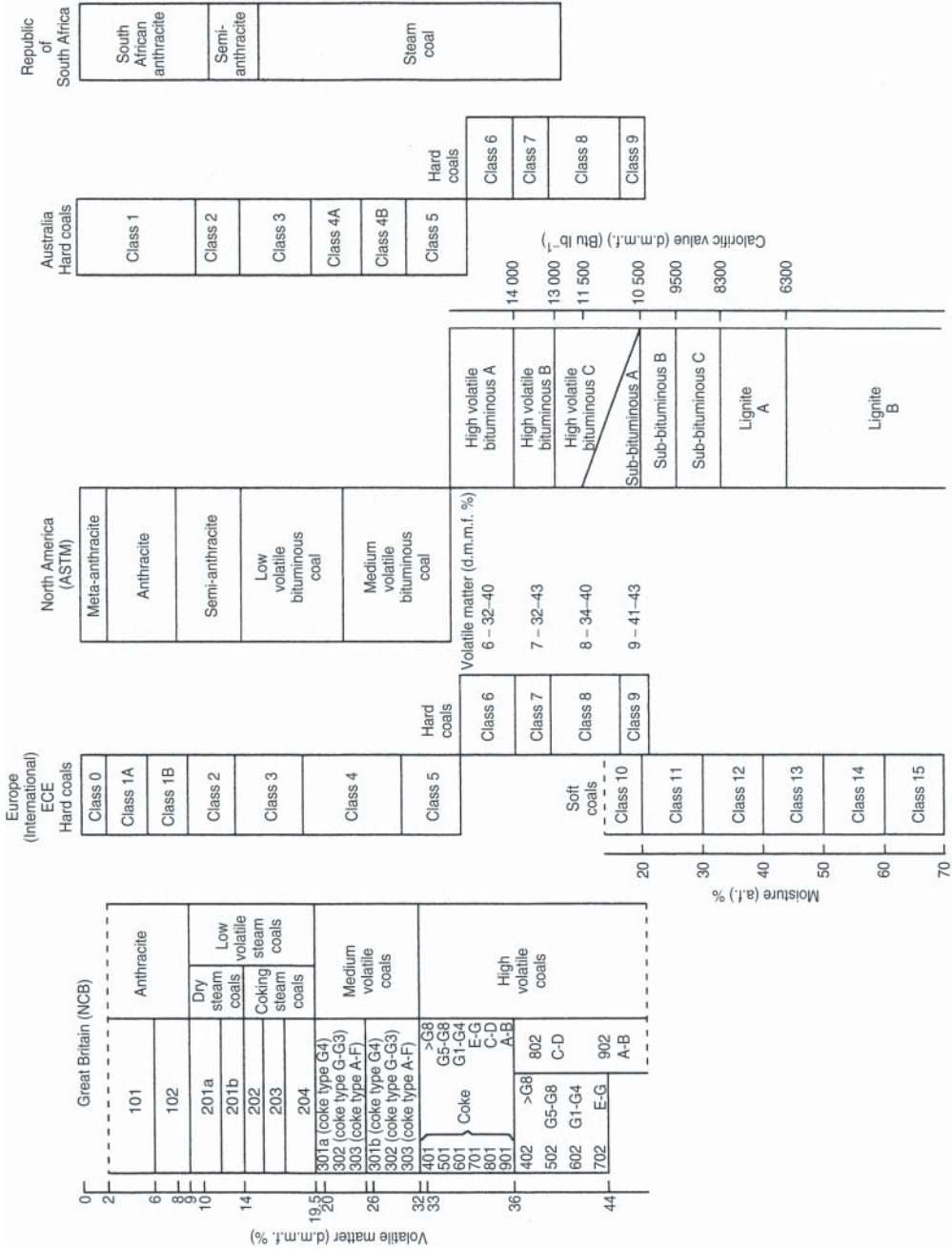
**Table 4.27** Classification of low-rank coals UNECE (2002).

Coal type	Gross CV (MJ kg <sup>-1</sup> , maf)	Moisture (%, ar)	R <sub>10</sub> (%)
Low rank C (ortholignites)	<15.0	<75	—
Low rank B (meta-lignites)	15–20	—	—
Low rank A (subbituminous)	20–24	—	<0.6

**Table 4.28** Codification system used by UNECE (2002) for low-rank coals.

Digit 1, 2 code no.	Gross CV (MJ kg <sup>-1</sup> , daf)	Digit 3, 4 code no.	Total moisture (mass%, ar)	Digit 5, 6 code no.	Ash content (%, db)	Digit 7, 8 code no.	Total sulfur content (%, db)
15	15.00–45.98 incl.			00	0.0–0.9 incl.	00	0.00–0.09 incl.
16	16.00–16.98 incl.			01	1.0–1.9 incl.	01	0.10–0.19 incl.
17	17.00–17.98 incl.			02	2.0–2.9 incl.	02	0.20–0.29 incl.
		20	20.0–20.9 incl.				
		21	21.0–21.9 incl.				
		22	22.0–22.9 incl.			11	1.10–1.19 incl.
						12	1.20–1.29 incl.
						13	1.30–1.39 incl.
		38	38.0–38.9 incl.	28	28.0–28.9 incl.		
		39	39.0–39.9 incl.	29	29.0–29.9 incl.		
		40	40.0–49.9 incl.	30	30.0–30.9 incl.		
						20	2.00–2.09 incl.
		50	50.0–50.9 incl.				
		56	56.0–56.9 incl.	49	49.0–49.9 incl.		
						32	3.20–3.29 incl.
		65	65.0–65.9 incl.				

Source: Reproduced with permission of UNECE.



**Figure 4.19** Interrelationships of coal classification systems used in various countries. *Source:* Unpublished data, reproduced by permission of BP Coal Ltd.

As for medium- and high-rank coals, this system for low-rank coals is based on an eight-digit number.

1. Digits 1 and 2, the gross CV in  $\text{MJ kg}^{-1}$  (maf).
2. Digits 3 and 4, the total moisture percentage recalculated to ar basis.
3. Digits 5 and 6, the ash content percentage recalculated to db.
4. Digits 7 and 8, the total sulfur content percentage recalculated to db.

The coded parameters are always coded in this order, and the make-up of the code numbers is shown in Table 4.27.

Before a final selection of a low-rank coal for a particular purpose, a set of relevant

analyses are carried out. These supplementary parameters consist of the numerous chemical and physical tests outlined previously and which are also used for the medium- and high-rank coals.

The interrelationships of the various coal classifications used in these countries are shown in Figure 4.19. It should be noted, however, that the UNECE high- and low-rank coal classifications shown in the figure have been superseded as described above.

#### 4.4.7 Russia

The Russian system of coal classification as given in the Gosstandart (GOST) 25543-88 (Yeriomina 1988) is based on the degree of

**Table 4.29** Classification of Russian coals Yeriomina (1988).

Letter code	Name of mark	Volatile matter (% daf)	Vitrinite reflectance $R_o$ (%)	Plastic layer $y$ (mm)	Moisture (% ar)	Gross CV ( $\text{MJ kg}^{-1}$ , maf)
B	Brown coal	10–>48	<0.6	—	1B	>40
					2B	30–40
					3B	<30
D	Long flame coal	>30	0.40–0.79	<6		
DG	Long flame gas coal	>30	0.50–0.79	6–9		
G	Gas coal	>30	0.50–0.99	6–12		
GZho	Gas fat semi-lean coal	<38	<0.99	10–16		
GZh	Gas fat coal	>38	0.50–0.99	16–25		
Zh	Fat coal	28–36	0.80–1.19	14–26		
KZh	Coke fat coal	24–30	0.90–1.29	>18		
K	Coke coal	24–28	1.00–1.29	13–17		
			1.30–1.69	13		
KO	Coke semi-lean coal	24–28	0.80–1.39	10–12		
KSN	Coke weakly caking low metamorphic coal	30	0.80–1.09	6–9		
KS	Coke weakly caking coal	<30	1.10–1.69	6–9		
OS	Semi-lean caking coal	20	1.30–1.79	6–12		
TS	Lean caking coal	<20	1.40–1.99	<6 (RI > 13)		
SS	Weakly caking coal	>20	0.70–1.79	<6		
T	Lean coal	8–18	1.30–2.59	—		
A	Anthracite	<8	>2.2	—		

metamorphism, which is defined according to the mean value of vitrinite reflectance index  $R_o$ . Another principal parameter used is volatile matter on the daf basis. The Russian system uses the term mark of coal rather than rank, although these are not considered entirely synonymous. In addition, CV is used for the classification of coals by types, i.e. brown coal, hard coal, and anthracite. Other parameters are used to characterise caking properties, namely the thickness of the plastic layer  $y$ , FSI, and RI. Coals are divided into 17 marks, and each mark is encoded with the Latin equivalent letter of the Russian letter that represents the full name of the coal mark (Table 4.29).

These coal marks have been approximately compared with the ASTM D388-99 standard coal classification (Table 4.23) and is shown in Table 4.30. Only the Russian mark SS within

the GOST standard 25543-88 cannot be equated with the ASTM D388-99 standard, due to the fact that, regardless of the different values of volatile matter, due to the high inertinite content (>60%), they cannot be caked or agglomerated. Also, exact equivalence is approximate because of possible differences in sampling, analytical procedures, and use of terminology.

#### 4.4.8 People's Republic of China

The PRC coal classification, whilst not dissimilar system to the Russian, uses GB standards to classify coals. The Chinese system is based on volatile matter; all coal is divided into brown coal, bituminous coal, and anthracite. Bituminous coals are further divided according to their degree of coalification and properties relating to industrial usage. Brown coals and anthracites are subdivided into two and three

**Table 4.30** Comparison of GOST and ASTM Standard Classification of Coals. US open File Report 01–104 (Brownfield et al 2011).

FSU, GOST 25543-88			USA, ASTM 380-98a	
Brown Coals	Brown (B)	1B	ligB	Lignite
		2B	ligA	
		3B	subC	Subbituminous
Hard Coals	Long-Flame (D)	subB	sub A hvCb hvBb	
	Long-Flame-Gas (DG), Gas (G), Gas-Fat (GZh), Gas-Fat-Mearge (GZhO), and part Fat (Zh)	hvAb		
	Fat (Zh), Coking-Fat (KZh), Coking (K), Coking-Mearge (KO), Coking-Caking (KSN and KS)	mvb		
	Mearge-Caking (OS) and Lean-Caking (TS)	lvb		
	Lean (T)	sa		
	Semi-anthracite (PA)	an	Anthracitic	
	Anthracites (A)			A1
A2				
A3		ma		

Source: US Geological Survey Open File Report 01-104 (Brownfield et al., 2001).



**Table 4.31** Classification of Chinese coals (GB/T5751-86; Lu and Laman 2012).

Symbol	Coal type (abbr)	Volatile matter (% daf)	Caking index GRL	Plastic layer $y$ (mm)	Dilatometer $b$	Moisture (%)	Gross CV (MJ kg <sup>-1</sup> , maf)
WY	Anthracite	(1) 0–3.5 (2) 3.5–6.5 (3) 6.5–10					
PM	Meagre coal	>10–20	<5				
PS	Lean-meagre coal	>10–20	5–20				
SM	Lean coal	>10–20	>20–65				
JM	Coking coal	>20–28 >10–20	>50–65 >65	<25	(<150)		
FM	Fat coal	>10–37	>85	>25			
1/3JM	1/3 Coking coal	>28–37	>65	<25	(<220)		
QF	Gas and fat coal	>37	>85	>25	220		
QM	Gas coal	>28–37 >37	>50–65 >35–65	<25	<220		
1/2ZN	1/2 Middle Sticky coal	>20–37	>30–50				
RN	Weak-sticky coal	>20–37	>5–30				
BN	Un-sticky coal	>20–37	<5				
CY	Long flame coal	>37	<5–35				
HM	Brown coal	(1) >37 (2) <37				<30 >30–50	24

subclasses respectively. Bituminous coals are classified according to their volatile matter content, their caking properties (GRL) including the thickness of the plastic layer ( $y$ ) and dilatometer reading ( $b$ ). The coal type nomenclature uses the old established terminology

of gas coal, fat coal, lean coal, and long flame coal, showing similarities to the Russian coal classification. The Chinese coal classification is shown in Table 4.31 based on GB/T5751-1986 (superseded by GB/T5751-2009; Lu and Laman 2012).

## 5

### Coal Sampling and Analysis

#### 5.1 Coal Sampling

The sampling of coal can be a difficult task, in that coal is a heterogeneous material.

Samples are the representative fractions of a body of material, acquired for testing and analysis, in order to assess the nature and composition of the parent body, collected by approved methods and protected from contamination and chemical change.

Such samples should be differentiated from those materials collected in ways that may not be truly representative of the coal from which they have been collected. These materials may still be useful but should be regarded as specimens rather than samples (Pryor 1965).

Coal samples may be required as part of a greenfields exploration programme to determine whether the coal is suitable for further investigation, or as part of a mine development programme, or as routine samples in opencast and underground mines to ensure that the quality of the coal to be mined will provide the specified run of mine product.

In-situ coal samples are taken from surface exposures, exposed coal seams in opencast and underground workings, and from drill cores and cuttings.

Ex-situ samples are taken from run-of-mine coal streams, coal transport containers, and coal stockpiles.

Such coal samples may have to be taken under widely differing conditions, particularly those of climate and topography. It is essential

that the sample taken is truly representative as it will provide the basic quality data on which decisions to carry out further investigation, development, or to make changes to the mine output will be made.

It is important to avoid weathered coal sections, coals contaminated by extraneous clay or other such materials, coals containing a bias of mineralisation, and coals in close contact with major faults and igneous intrusions.

##### 5.1.1 In-Situ Coal Sampling

Several types of in-situ samples can be taken, dependent upon the analysis required.

###### 5.1.1.1 Grab Samples

Generally, this is a most unsatisfactory method of obtaining coal for analysis, as there are no controls on whether the coal is representative, and this can easily lead to a bias in selection, e.g. the bright coal sections attract attention. However, grab samples can be used to determine vitrinite reflectance measurements, as an indicator of coal rank.

###### 5.1.1.2 Channel Samples

Channel samples are representative of the coal from which they are taken. If the coal to be sampled is a surface exposure, the outcrop must be cleaned and cut back to expose as fresh a section as possible. Ideally, the full seam section should be exposed, but in the case

of thick coals (especially in stream sections), sections of the roof and coal or the coal and floor only may be seen. To obtain a full seam section under these circumstances, two or more channels will need to be cut and the overlap carefully recorded. The resultant samples will consist of broken coal and will not preserve the lithological sequence.

In opencast workings, the complete seam section should be exposed, as this is less likely to be weathered than natural surface exposures.

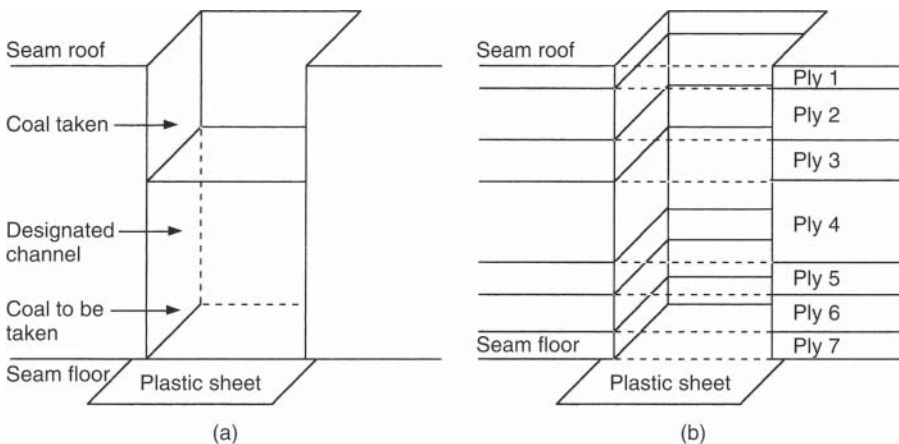
In underground workings, the seam will be unweathered, but the whole seam section may not always be seen, due to the workings only exposing the selected mining section of the seam.

To carry out a channel sample, the coal is normally sampled perpendicular to the bedding. A channel of uniform cross-section is cut manually into the coal seam and all the coal within the cut section is collected on a plastic sheet placed at the base of the channel (Figure 5.1a). Most channels are around 1.0 m across, and samples should not be less than 15 kg per metre of coal thickness. Such channel samples will provide a composite quality analysis for the seam, i.e. an analysis of all the coal and mineral matter present in the seam as a whole.

Though this is suitable for general seam quality assessment, more detailed analysis of the seam from top to bottom may be required. To achieve this, a channel ply sample is taken; this entails a similar procedure as for the whole seam channel sample except that the seam is divided into plies or subsections, as shown in Figure 5.1b.

Coal seams are rarely homogeneous throughout their thickness; most are divisible into distinct lithological sections. Plies are lithological subdivisions of the seam, each of which has a uniform character. When the lithology changes, such as at a clay parting in the seam, a separate ply is designated.

Where the roof and floor of a seam are exposed, ply samples of at least 0.25 m of roof material immediately above the seam and 0.25 m of floor underlying the seam should be included in the samples. This will allow the effects of dilution on coal quality to be assessed. In general, the thickness of coal plies should be a minimum of 0.1 m and a maximum of 1.0 m. In the case of banded coals containing alternating thin (<0.1 m) layers of bright coal–dull coal–clay, the seams may be sampled as a series of composite plies, with the details of the individual layers shown on the record sheet. An interbedded non-coal ply greater than 0.25 m in thickness may be



**Figure 5.1** Channel sampling procedure: (a) whole seam channel sampling; (b) coal seam ply channel sampling.

regarded as a seam split and recorded as such. Ply samples should be at least 2.0 kg where possible; it may be that the sample will be split into two fractions and one stored for later use.

Once the outcrop or face is cleaned, a shallow box cut is made for the total thickness of exposed coal seam. Once this is completed, the seam is divided into plies, each of which is measured and recorded on a record sheet similar to that shown in Figure 5.2.

The channel sample record sheet should show the following information:

- (i) Record card number.
- (ii) Map or aerial photograph number on which locality is located.
- (iii) Location of sample point, grid reference, or reference number.
- (iv) Description of the locality, stream section, working face, etc., including dip, strike, coal seam roof, and floor contacts.
- (v) Extent of weathering, fracturing, mineralisation, etc.
- (vi) Lithological description of each ply interval.

- (vii) Thickness of each ply interval.
- (viii) Designated sample number of each ply interval.

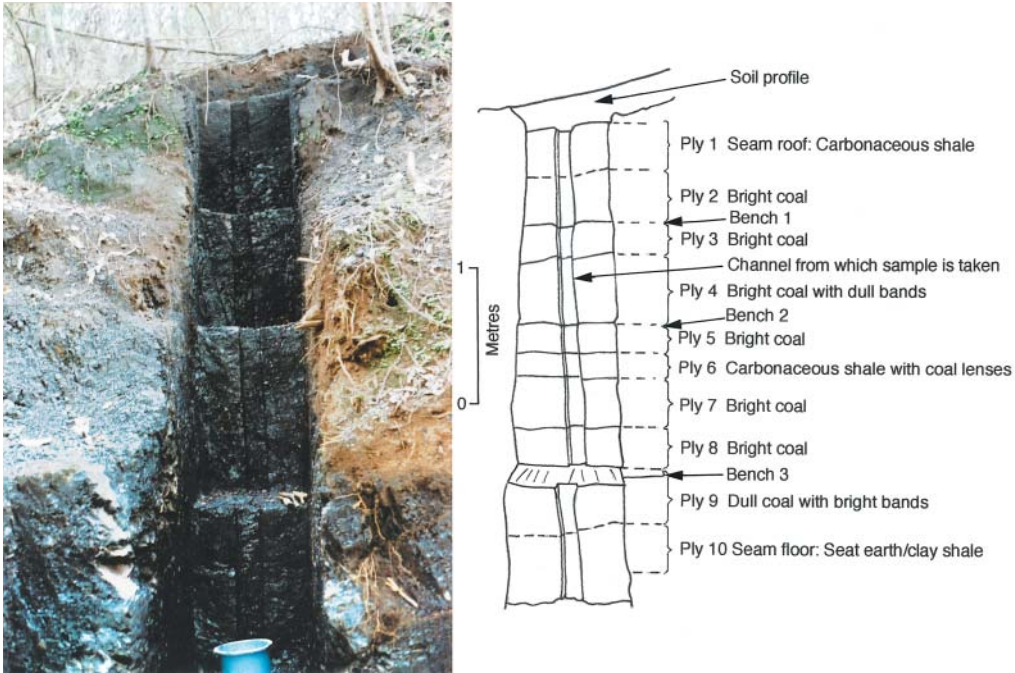
Space can also be allocated on the record card for analytical details, i.e. proximate analysis, to be added later to complete the record.

The fresh surface is then sampled as a channel cut from top to bottom (Figure 5.1b), cutting and collecting all material from each ply section in turn. Each ply sample should be sealed in a strong plastic bag immediately after collection to prevent moisture loss and oxidation. All sample bags must be clearly labelled with a designated number, a copy of which should be placed in a small plastic bag inside the sample bag and another attached to the outside of the sample bag. This number must be recorded on the channel sample record sheet.

Because this task is invariably a dirty one, labels get wet, blackened, and unreadable very easily. It is essential, therefore, that care must be taken to ensure that the sample numbers do not get lost or obliterated during transit to

COAL OUTCROP DATA CARD															
Outcrop Data Card Number		1.1489.R.S		Map	Photo Set	Run/Photo	Locality Name ANYWHERE								
Coord Easting				Northing				Geologist	A. N. DIER		Date	02 01 92			
SEAM NAME/NO ANYCOAL				Location Data JUNCTION OF GREEN RIVER AND RED RIVER EXPOSURE IN SOUTHSIDE BANK.											
PLY	Strike	Dip Direction	Reliability	Apparent Thickness	Outcrop Sample No.	True Coal Thickness	Coal Type	Pyrite	Roof Strata	Contact Type	Floor Strata	Contact Type	Outcrop Type	Coal Weathering	No. of Samples
A	242	S/152	G	0.25	1148/01				CLAY	SHARP			STRICT	PW	1
B	242	S/152	"	0.50	1148/02	0.50	BRIGHT	C					"	PW	1
C	242	S/152	"	1.00	1148/03	1.00	DULL/B	C					"	UW	1
D	242	S/152	"	1.39	1148/04	1.39	BRIGHT	R					"	UW	1
E	242	S/152	"	0.86	1148/05	0.86	DULL/B	R					"	UW	1
F	242	S/152	"	0.31	1148/06	0.31	BRIGHT	R					"	UW	1
G	242	S/152	"	0.52	1148/07	0.52	DULL	R					"	UW	1
H	242	S/152	"	0.25	1148/08						CLAY	GRAD	"	UW	1
Outcrop Sketch Section															

Figure 5.2 Coal outcrop data card. Source: Reproduced with permission of BP Coal Ltd.



**Figure 5.3** Surface coal ply channel sample taken in shallow dipping seam. Central narrow channel taken for ply sample analysis, including coal seam roof and floor. *Source:* Photograph by LPT.

the laboratory, as unidentifiable samples are useless and an expensive waste of time.

The advantage of channel ply sampling is that not only can the analysis of the individual plies be obtained, but also that by combining a fraction of each ply sample a whole seam composite analysis can be made. An example of a channel ply sample from a surface exposure is illustrated in Figure 5.3, which shows a channel cut to expose fresh coal and then a thinner channel (c. 0.25 m wide) cut from the fresh coal from which ply samples are collected for analysis.

#### 5.1.1.3 Pillar Samples

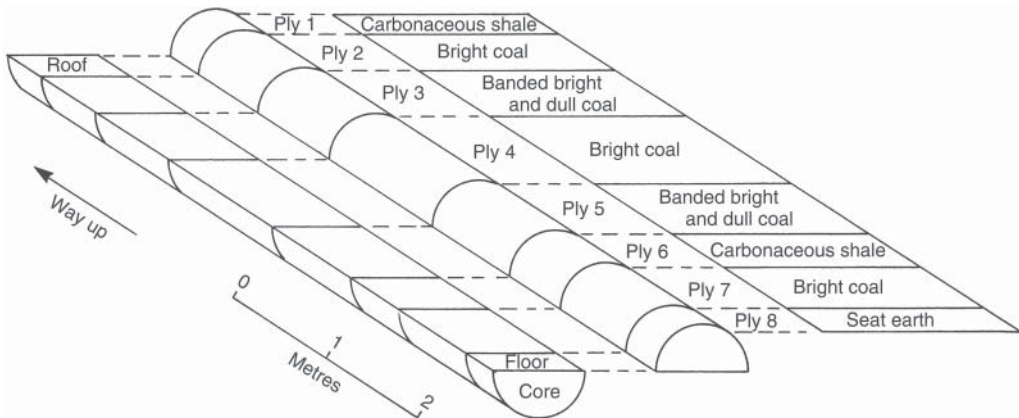
In underground coal mining, samples of large blocks of undisturbed coal are taken to provide technical information on the strength and quality of the coal. These pillar samples are taken when a specific problem may have arisen or is anticipated. Such samples are taken in

much the same way as whole seam channel samples, except that extra care is required not to disturb the cut-out section of coal during removal. Samples are then boxed and taken to the laboratory. Pillar sampling is a long and arduous business and is only undertaken in special circumstance, such as when mining becomes difficult or new roadways or faces are planned.

#### 5.1.1.4 Core Samples

Core sampling is an integral part of coal exploration and mine development. It has the advantage of producing non-weathered coal and including the coal seam floor and roof; and unlike channel samples, core samples preserve the lithological sequence within the coal seam.

First, the borehole core has to be cleaned if drilling fluids have been used and then lithologically logged. Following this, the lithological



**Figure 5.4** Ply sampling of borehole core; run of samples to include all the coal seam and roof and floor. Core may be split in this fashion and one half kept for future examination and analysis.

log should be compared with the geophysical log of the borehole to select ply intervals and to check for core losses and any other length discrepancies.

Once the core has been reconciled to the geophysical logs and the ply intervals have been selected, sampling can commence. Core ply samples are taken in the same way as for surface channel ply samples (as shown in Figure 5.4); again, a ply sample of the coal seam roof and floor (up to 0.25 m) is taken to determine dilution effects. Then the individual plies are sampled, making sure no core is discarded. As in the case of surface samples, bright coal tends to fragment and make up the finer particles that may easily be left in the core tray.

The samples are bagged and labelled as for surface ply samples, and the sample numbers recorded on the core logging sheet in the manner shown in Figure 6.23.

Large-diameter cores may be split lengthways with a bolster chisel and then one half ply sampled, with the other half being retained for future analysis.

#### 5.1.1.5 Cuttings Samples

This method of sampling is considerably less accurate than that of core sampling. As

with core samples, cuttings are unweathered and are a useful indicator as to the general nature of the seam. Air flush and mud flush non-core drilling is a quicker operation than core drilling and will produce cuttings for each horizon encountered in drilling. In the case of mud flush cuttings, they will need to be washed to remove any drilling fluid before sampling.

Cuttings are usually produced for every metre drilled; those cuttings returns that are all coal may be collected, bagged, and numbered in the same way as channel samples. The depth to the top and bottom of the seam sampled should be determined from the geophysical log. The drawback with using cuttings samples is that only a general analysis of the seam can be made, and even this is unlikely to be truly representative; also, contamination from strata above the coal may be included. A close study of the geology will determine whether this is so.

#### 5.1.1.6 Specimen Samples

Orientated specimens of coal may be collected so that their precise orientation can be recreated in the laboratory. The dip and strike of the coal is marked on the specimen before removal. This method is commonly used for the studies



of the optical fabric of the coal, or for the structural features in the coal.

#### 5.1.1.7 Bulk Samples

Bulk samples are taken from outcrops, small pits, or minishafts (i.e. 2 m diameter shaft excavations). A bulk sample is normally 5–25 t and is taken as a whole seam channel sample on a large scale. Such a bulk sample is taken in order to carry out test work on a larger scale; this is designed to indicate the coal's likely performance under actual conditions of usage.

Steam coals are taken for small combustion tests in a pf rig, to simulate conditions in a pf boiler. Pulverised coal firing is the combustion of powdered coal suspended as a cloud of small particles in the combustion air. Substantially more heat is released per unit volume in pf boilers than in stoker-type boilers.

Coking coals are taken to carry out moving-wall oven tests, i.e. to determine how much the coal swells when the coal is combusted, thus putting pressure on the oven walls, which are constructed of uncemented brickwork. High-pressure coals are undesirable and are normally blended with low-pressure coals to reduce the problem. In the USA, low-volatile coking coals (volatile matter: 20–25%; swelling index: 9) are high-pressure coals, whereas, in general, high-volatile coals do not have such high pressures. It is significant that Gondwana coking coals are low-pressure coals, an important factor in Australia being able to export coking coals.

Bulk samples are collected from a site already channel sampled, loaded into drums, numbered, and shipped to the selected test centre.

#### 5.1.1.8 Sample Storage

In the majority of cases, the channel and core samples will be required immediately for laboratory analysis. However, there are circumstances where duplicate coal samples for future reference are taken. Usually, the channel plies are divided into two or the cores are split and one half retained.

If the duplicate samples are to be put into storage, this presents a problem: the exposure of the coal to air will allow oxidation to take place during storage and will result in anomalous quality results when analysed at a later date.

The usual procedure to prevent oxidation of samples is to store them under nitrogen or in water.

To store in nitrogen, place a tube connected to a pressurised cylinder containing nitrogen into a plastic sample bag, then add the coal sample and flush the sample with nitrogen, regulating the flow by means of a flow meter. The nitrogen has to fill the spaces between the coal fragments, so flushing with nitrogen is required for several minutes. One difficulty with this method is that nitrogen is less dense than air and so some is inevitably lost in the process. Once the bag has been thoroughly flushed, it should be heat sealed; no other form of sealing is anywhere near as effective. The coal samples can be as received or air dried and can be in the form of lump or crushed coal. It should be noted that for all in-situ and non-in-situ samples, the top size to which any sample is crushed to is important in determining the weight of the sample required. The size of the sample is calculated as follows:

$$\text{Sample size (kg)} = 5.24 \times \bar{x}$$

where  $\bar{x}$  is the mean particle size, i.e. (top size + bottom size)/2, and 5.24 is an empirically determined number quoted in BS1017-1 (1989) and Australian Standard 4264 (2009) (Appendix 5A).

A cheaper method of storage is by immersing the channel or core sample in the form of lump coal in water. This method has the advantage over storing in nitrogen in that it preserves fluidity of the coal, but it does present handling problems when the sample is required. The sample will have to be air dried before analysis can begin.

Samples can be kept by these methods for one to two years before analysis.

### 5.1.2 Ex-Situ Sampling

The object of collecting coal samples after mining is to determine the quality of coal actually being produced. This coal may differ significantly from the in-situ seam analysis, in that not all of the seam may be included in the mining section or that more than one seam may be worked and fed to the mine mouth and mixed with coal from other seams. In addition, there may be dilution from the seam roof and/or floor that becomes part of the mined coal product.

The mined coal is broken up and therefore contains fragments that vary a great deal in size and shape. Representative samples are collected by taking a definite number of portions, known as increments, distributed throughout the total quantity of coal being sampled. Such increments represent a sample or portion of coal obtained by using a specified sampling procedure, either manually or using some sampling apparatus.

The various practices used in collecting ex-situ samples and the mathematical analysis of the representativeness of samples, i.e. quality control, is reviewed in Laurila and Corriveau (1995) and Membrey (2013).

Increments are taken by three methods:

- (i) Systematic sampling, where increments are spaced evenly in time or in position over the unit.
- (ii) Random sampling, where increments are spaced at random but a prerequisite number are taken.
- (iii) Stratified random sampling, where the unit is divided by time or quantity into a number of equal strata and one or more increments are taken at random from each.

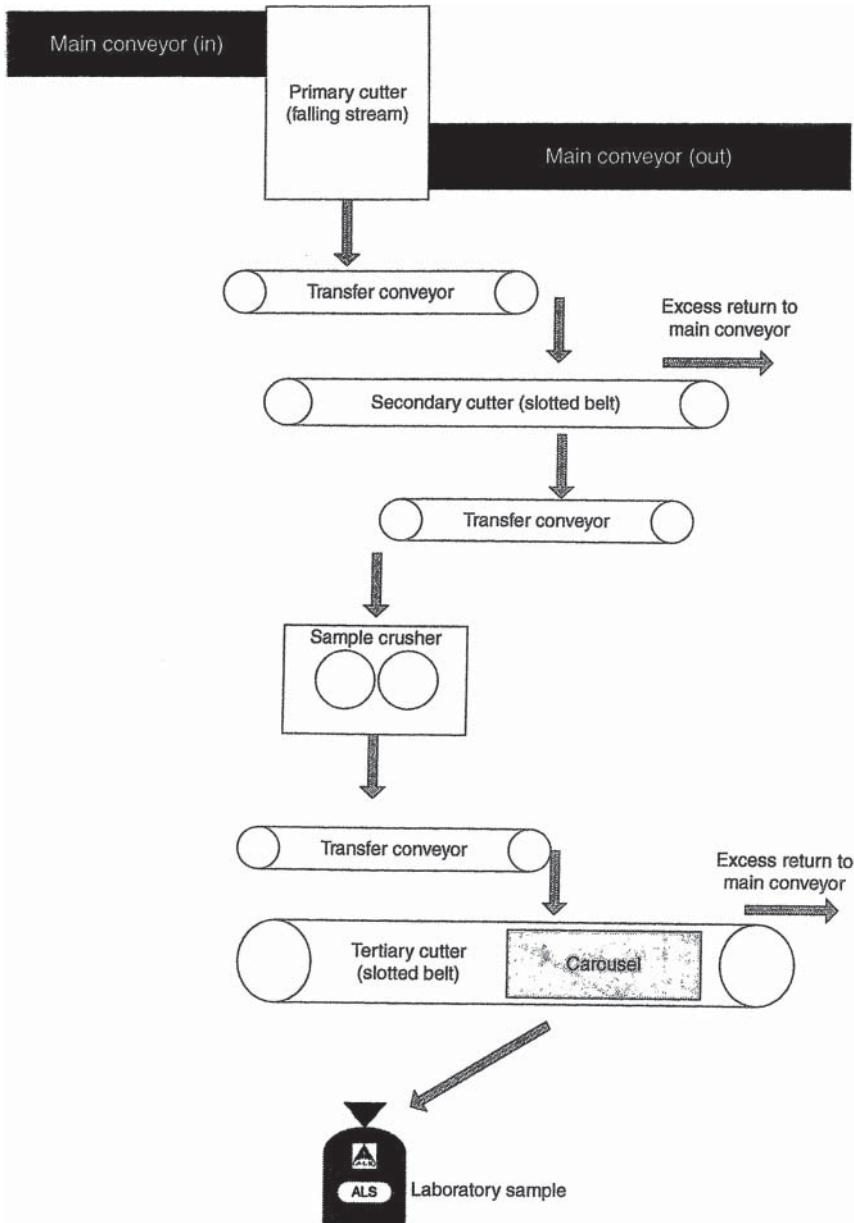
It is good practice that, whatever the method used, duplicate sampling should be employed to verify that the required precision has been attained.

Ex-situ coal sampling is carried out on moving streams of coal, from rail wagons,

trucks, barges, grabs, or conveyors unloading ships, from the holds of ships and from coal stockpiles.

- (i) Hand sampling from streams is carried out using ladles or scoops. The width of the sampler should be 2.5 times the size of the largest lump likely to be encountered. However, this type of sampling is not suitable for coal larger than 80 mm. For larger samples, mechanical sampling equipment is used. Moving streams of coal (conveyors) can be sampled using the following methods:

- (a) Falling-stream samplers or cutters. These make either a linear traverse across the coal conveyor in a straight-line path perpendicular to the direction of flow, or opposite to the direction of flow or in the same direction of flow, or they make a rotational traverse by moving in an arc such that the entire stream is within the radius of the arc. Many modern coal loading facilities use a tertiary sampling system. A falling-stream cutter takes a primary increment at the transfer point on the conveyor, this increment is transferred to a secondary cutter where up to six cuts are taken for each primary increment. The secondary increments are then transferred to a sample crusher and then to a tertiary cutter, which takes 12 divisions and transfers them to a container for shipment to the laboratory, as shown in Figure 5.5 (Membrey 2013). The decision to implement a single-stage, two-stage, or three-stage sampling system is shown in Figure 5.6 (Membrey 2013).
- (b) Cross-belt samplers or cutters move across the belt pushing a section of coal to the side while the belt runs. Single-stage and two-stage sampling systems are used; the former is suitable for very low material flow rates

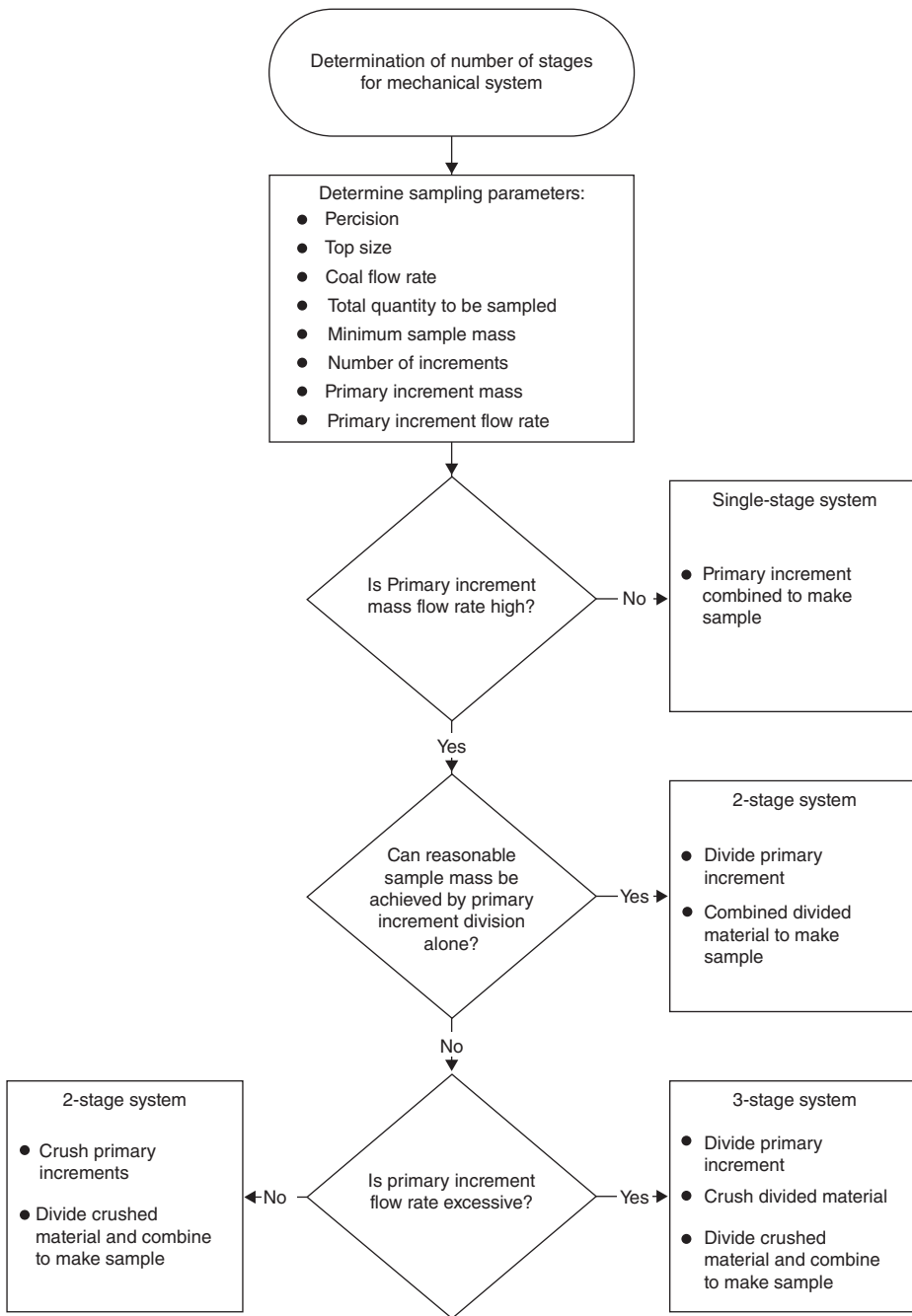


**Figure 5.5** Three-stage mechanical sampling system (Membrey 2013).

and small particle top sizes (Membrey 2013).

- (c) Where a sample is required from a conveyor belt and there is no stream sampling equipment available or where the flow rate is too great

for manual sampling, the stop-belt method is used whereby the conveyor is stopped and all coal occurring within a selected interval, usually a couple of metres, is collected (Figure 5.7). Stop-belt sampling can



**Figure 5.6** Decision flow chart for determining sampling system requirement (Membrey 2013).



**Figure 5.7** The collection of a stop belt sample from a main conveyor (Mazzone 1998).

be disruptive to operations, especially where a large number of increments are required to produce a sample for analysis.

Laurila and Corriveau (1995) stated that the correct increment selection occurs when all the elements of the transversal cross-section are intercepted by the sampling cutter during the same length of time. This should avoid any increase in error. These sampling systems are checked for bias by using a reference sampling method as recommended by BS ISO 13909 (2016) Parts 1–8 or ASTM D2234 (2017) (Appendix 5A).

- (ii) Wagons and trucks are sampled by taking samples from their tops by means of probes, or by sampling from bottom or side-door wagons during discharge, or sampling from the exposed face of

coal as the wagons or trucks are tipping into bunkers or ships, or wagons being emptied via tippers.

- (iii) Ships are sampled either from conveyors loading and unloading coal, at a point where bias can be avoided, or from the hold of the ship. Samples from the hold are taken every 4 m of the depth of the coal within the hold. It is important to estimate the proportion of fine and lump coal in the consignment. It should be noted that free moisture, if present, will tend to settle towards the bottom of the hold. This increase of moisture with depth makes it difficult to collect samples for moisture content determination.
- (iv) Sampling from barges is the same as for ships, except that if the depth of coal is less than 4 m it should be sampled in one stage during unloading, once the bottom of the barge is partially uncovered.
- (v) Sampling from stockpiles: where the preferred procedure of sampling from a conveyor belt during stocking and unstocking cannot be used, then the stockpile is sampled based on collecting increments spaced as evenly as possible over the surface and layers of the stockpile. Sampling is by means of probes or by digging holes. If the stockpile is known to consist of different coals piled in separate areas of the total pile, a separate gross sample must be taken from each such area. The stockpile should be divided into a number of portions, each 1000 t or less, from which a separate sample with a specified number of increments is taken. This normally takes a long time to accomplish, but it can be speeded up if automated auger units are employed. It is important that all levels in the stockpile are sampled.

Table 5.1 (Osborne 1988) indicates the minimum number of increments required for gross samples of a single-load consignment up to 1000 t from all of the above.

Table 5.1 Minimum number of increments required for gross samples of a single coal consignment up to 1000t.

Sampling situation	Common sample for total moisture and general analysis			Total moisture sample		General analysis sample		Size analysis sample
	Sized coals, dry cleaned or washed	Washed smalls (50 mm)	Blended part treated, untreated, run-of-mine, and 'unknown' coals	Sized coals, dry cleaned or washed, unwashed dry coals	Washed smalls (50 mm) blended, part treated, untreated, run-of-mine, and 'unknown' coals	Sized coals, dry cleaned or washed, or unwashed dry coals	Blended part treated, untreated, run-of-mine, and 'unknown' coals	All coals
Moving streams	20	35	35	20	35	20	35	40
Wagons and trucks, barges, grabs, or conveyors unloading ships	25	35	50	20	35	25	50	40
Holds of ships, stockpiles	35	35	65	20	35	35	65	40

Source: From Osborne (1988). Reproduced with kind permission of Kluwer Academic Publishers.



## 5.2 Coal Analysis

The marketability of coals depends on their quality. This will determine whether they are to be sold as steam or coking coals, prime or lower grade coals. Customer requirements vary considerably, from those who will accept a broad spectrum of coal qualities to those who require coal for a specialised purpose and have set restricted specifications for the coal.

The coal producer, i.e. the mining company, will have assessed the potential market before developing any coal deposit, i.e. whether to mine coal for export or local use. The mining company will also need to know the quality limitations of the coal that can be produced from the deposit.

The quality of the coal has to be determined at an early stage of exploration and monitored during all later phases of development.

All coals should be sampled using the procedures outlined in 5.1.1 and 5.1.2, and sent to the laboratory, where they are weighed, crushed and split for analysis.

### 5.2.1 Outcrop/Core Samples

The procedures for weighing, crushing, and splitting outcrop/core samples are shown in Figure 5.8 for steam (thermal) coals and in Figure 5.9 for coking (metallurgical) coals. However, there is no universal set procedure, and differences do occur.

In Australia, the samples are crushed to 11.2 mm. Large-diameter core samples are preferred when sampling Gondwana coals in order to be more confident of yield values obtained during analysis. There should be a correlation in properties between outcrop/small-diameter core, large-diameter core, and bulk samples.

The analysis undertaken for each float-sink fraction is proximate analysis, plus total sulfur, and calorific value (CV). This is intended to produce a simulated product by combining several float-sink fractions. This product is then analysed for ultimate analysis, ash

analysis, ash fusion temperatures, Hardgrove grindability, swelling index, and Geiseler plastometer test, with the latter only if the coal has coking properties.

In South Africa, the coal is generally crushed to only 25 mm; the coal is then analysed for proximate analysis, total sulfur, and CV. Float-sink tests are done, but no simulated product is made.

In Australia and South Africa, the fine fraction (0.5 or 0.1 mm) is screened out before analysis, dependent upon expectations for coal preparation.

In the UK, a similar procedure is used, except that for outcrop/core samples there is no float-sink analysis carried out, only proximate analysis, total sulfur, and CV.

In the USA, there are no defined crushed parameters; proximate analysis, total sulfur, and CV are determined. Float-sink analysis is done with the results reported to zero; this often means that the fines have to be screened out. Because in the USA there is no hard and fast procedure for outcrop/core analysis, the individual procedures have to be verified in order to correctly assess the reliability of the analytical results.

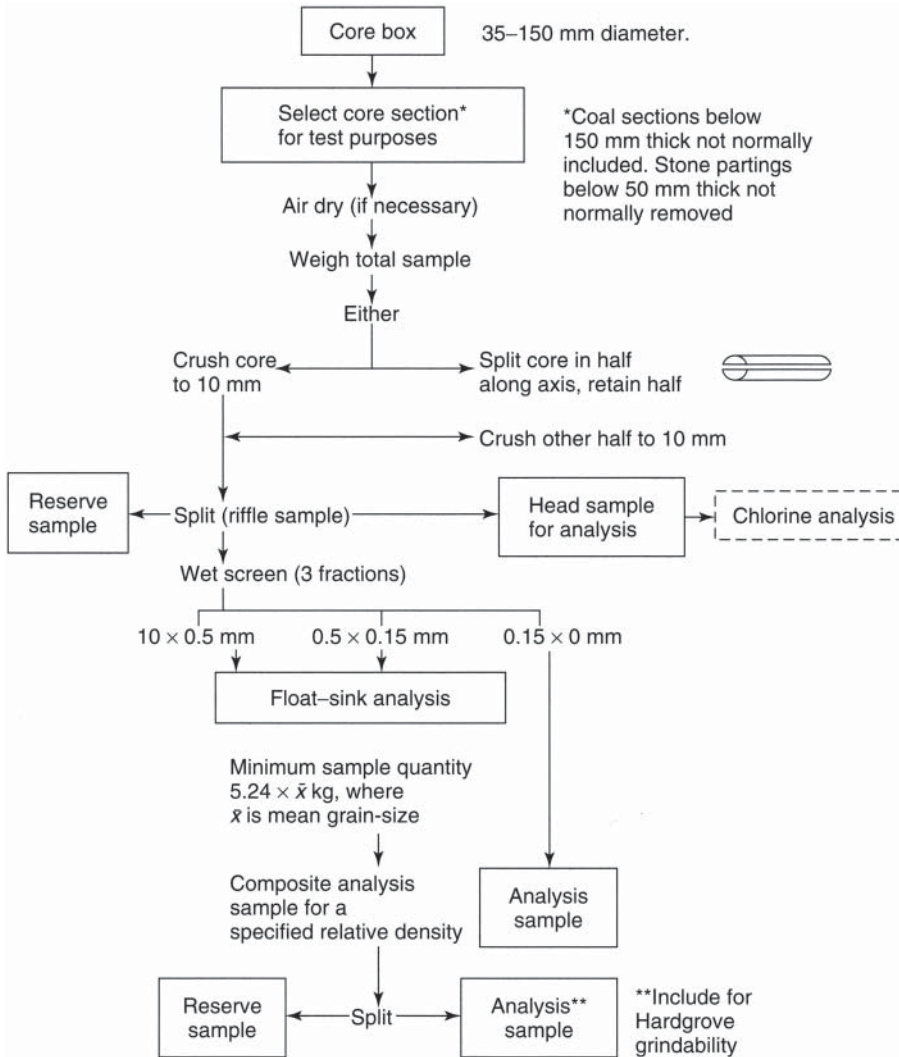
### 5.2.2 Bulk Samples

The procedure for sample preparation of bulk samples of both steam and coking coals is shown in Figure 5.10 for steam (thermal) coals and in Figure 5.11 for coking (metallurgical) coals. The coals are analysed as for outcrop/core samples with the additional tests for combustion and coking properties.

### 5.2.3 Ex-Situ Samples

The analysis of ex-situ coal samples is undertaken to ascertain the quality of the coal leaving the mine, leaving the coal preparation plant, if one is installed, and in stockpiles prior to shipment to ensure that the agreed specification of the coal is maintained.

Stream samples will be crushed and analysed for proximate analysis, total sulfur, and CV,



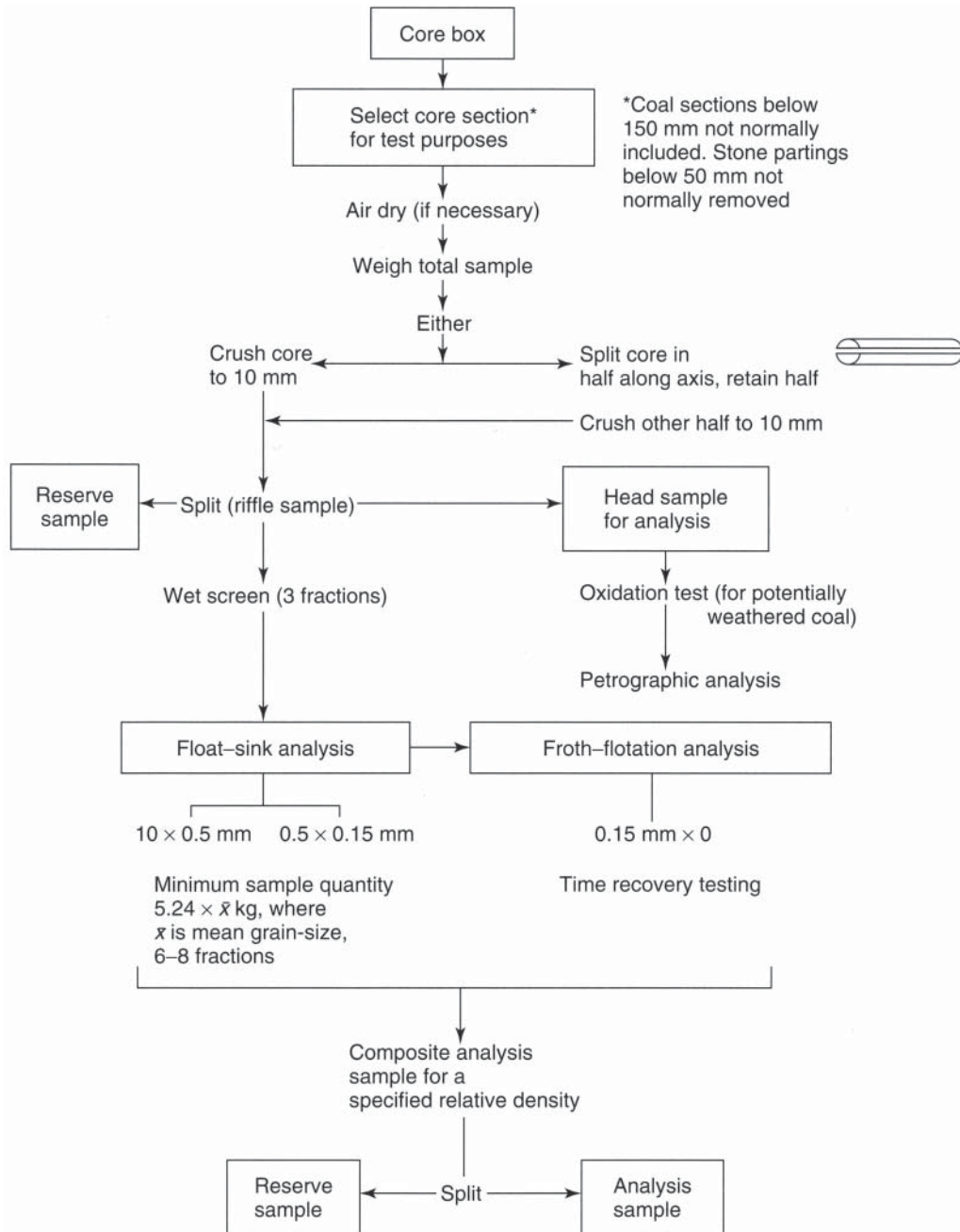
**Figure 5.8** Sample preparation diagram for drill core samples from a steam (thermal) coal deposit. *Source:* From Osborne (1988).

plus other properties if requested; normally, no float-sink analysis is done. In addition, other stream samples, such as stop-belt samples, will be used for size analysis and float-sink analysis. Samples from small stockpiles are rarely taken; and although augured samples may be taken in large stockpiles, it is difficult to obtain a truly representative sample.

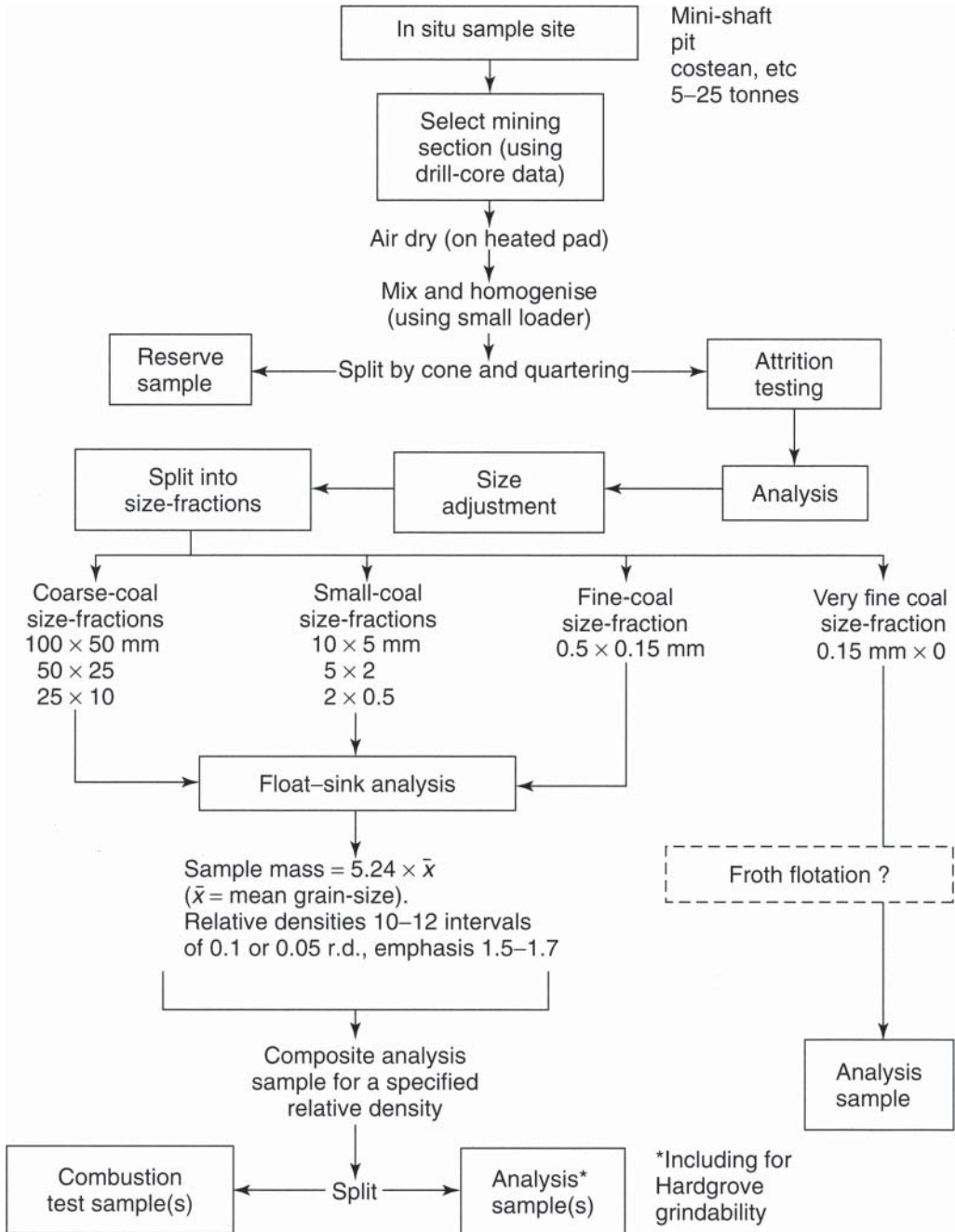
A number of types of on-line analyser are currently used at coal-fired power stations, coal mines, and coal handling facilities. These

are usually ash and moisture monitors or elemental analysers. Modern on-line analysers can provide high-precision ash, moisture, sulfur, and energy monitoring, and they are increasingly used on coal conveyor systems and for coal shipments. These are designed to withstand tropical weather conditions, such as high humidity and heavy rainfall.

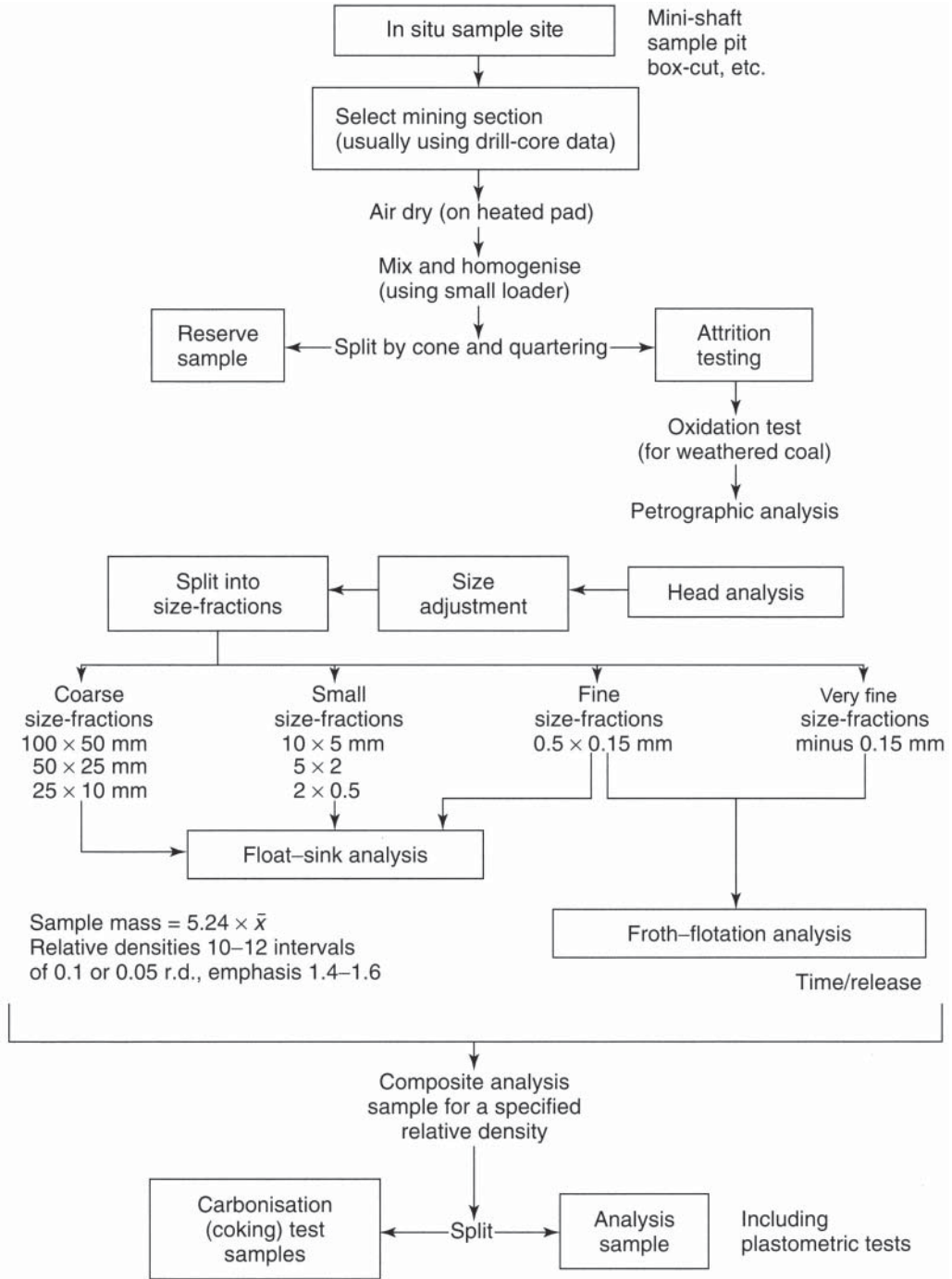
On-line analysers monitor dual-energy gamma-ray transmission for ash and microwave moisture measurement, and there are



**Figure 5.9** Sample preparation diagram for drill core samples from a coking (metallurgical) coal deposit. Source: From Osborne (1988).



**Figure 5.10** Sample preparation diagram for bulk sample(s) from a steam (thermal) coal deposit. *Source:* From Osborne (1988).



**Figure 5.11** Sample preparation diagram for bulk sample(s) from a coking (metallurgical) coal deposit. Source: From Osborne (1988).

more expensive methods for measuring sulfur using gamma neutron activation analysis. Between these, cost-wise, X-ray fluorescence can be used, but this only penetrates a thin layer of material. These types of measurement can be influenced by the composition

of the non-coal material; for example, if the shale fraction included in the coal stream is of marine origin it will have a higher radioactivity, and if pyrite is present it can influence the background scatter by having high fluorescence.



## 6

# Coal Exploration and Data Collection

## 6.1 Introduction

In recent years, the emphasis has been on development of known coal deposits, whether developing better confirmation of the deposit or extending existing mining operations. Greenfield exploration has declined due to a number of factors, including stricter limits on coal quality to meet environmental standards, international financial constraints, and security for personnel.

However, the principles of coal exploration and data collection for known and new coal deposits remain the same.

The principal objective in the exploration for coal is to determine the location, extent, and quality of the resources available in a particular area, and to identify those geological factors that will facilitate or constrain mine development.

Such a role encompasses the evaluation of existing data, geological mapping and sampling, the use of geophysics, and drilling. Once adequate resources of coal of suitable quality have been identified, the geological input will be concentrated on supporting the engineers in the design and development of the mine; this will include additional drilling and sampling, succeeded by geotechnical studies.

The emphasis of geological input will gradually change from exploration to development without a break in continuity. Figure 6.1 illustrates the various stages in this process, from exploration, mapping, and sampling through

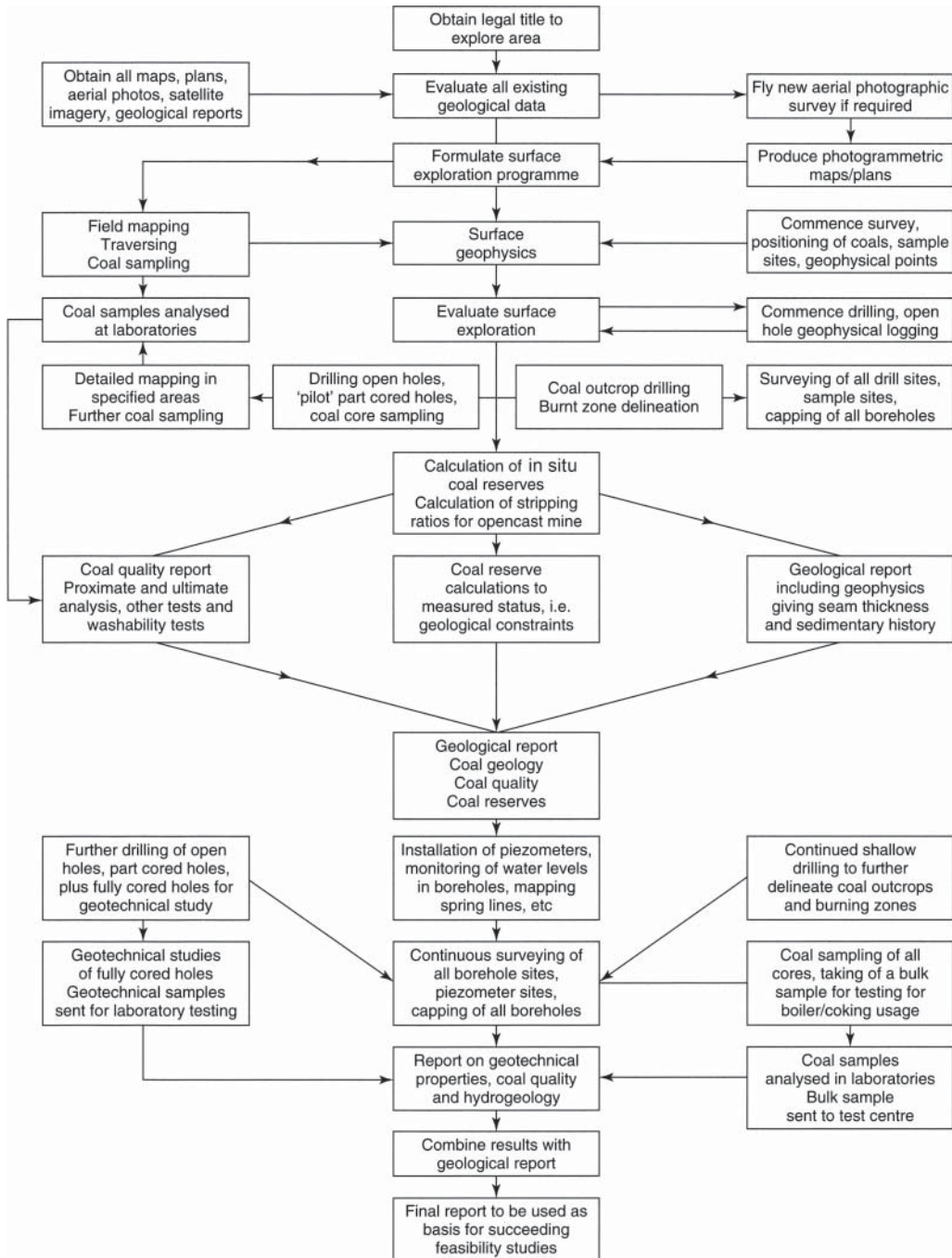
to reserve calculations, coal quality results, and geotechnical investigations.

In the last 20 years, the use of microelectronics-based technology has resulted in significant changes in how coal exploration is carried out, particularly in how data are collected, analysed, and presented. This technology has the ability to handle large amounts of information and maintain consistency, which has resulted in higher standards of data acquisition than was possible using only traditional exploration techniques. Nevertheless, the basic geological practice of observation and data recording is still relevant and forms the geological database on which computerised studies are developed. Methods required for geological input are outlined below, and Waltho et al. (2014) have outlined best practice for coal exploration and resource evaluation.

## 6.2 Field Techniques

The field examination of coal-bearing sequences is an essential component of any exploration programme, particularly the identification and assessment of a new potential coal-bearing area. Field examination of surface exposures of coal is the precursor to formulating a drilling programme to identify coal in the subsurface.

The first step in carrying out a geological study of a selected area is to collate all available



**Figure 6.1** Flow diagram to show principal activities requiring a geological input during the exploration phase of a coalmine development project.

information on that area. This may include published geological maps, topographic and cadastral maps, scientific papers and reports, land records, aerial photographs, and satellite imagery. If such information exists for the selected area, then the geological setting can be ascertained as well as topography, water supply and land access, and the availability of base maps. If no surveyed maps exist, photogrammetric maps constructed from available aerial photographs will be required to carry out ground surveys.

The bulk of this information is usually available from the geological survey or land survey departments of the country in question. Additional data can be obtained from universities, libraries, and local government departments, plus writers such as Knutson (1983) outlined planning and implementation of coal exploration programmes in reconnaissance geology for coal exploration.

The study should make special note of any previous history of coal exploration or mining, and details of which companies were involved. In particular, attention should be given to any coal quality data, and the basis on which it is quoted, reserves calculations, and production figures.

The scale of maps is important. In order to carry out a reconnaissance survey, base maps of 1:20 000 and 1:50 000 will be adequate. However, for further detailed mapping and sampling, and for planning drilling sites at set intervals, a scale of 1:10 000 is necessary. In the case of mine operations, large-scale plans are required; those of 1:2000 and 1:5000 are most commonly used. If the area has not previously been surveyed, aerial photographs will be used for map compilation (see Section 6.2.2). Modern advances in remote sensing, satellite imaging, and global positioning systems (GPSs), together with enormous improvements in communications, have changed the 'pioneering' aspect of field coal exploration. However, fieldwork will be necessary at all stages of coalmine development, with the concentration of field activities in the

reconnaissance and exploration stages of the project.

The style of field surveys varies greatly. In tropical rain forest, traverses are made along water courses either on foot or by using some type of boat. Alternatively, local agriculture or industry may have constructed roads or tracks with exposures of bedrock. In hill or mountain country, outcrops may be frequent but require a helicopter for access and observation, whereas outcrops may be almost non-existent in low-relief arid or scrub country but allow unrestricted access by four-wheel drive vehicles and trail bikes.

Safety during fieldwork is of prime importance; suitable clothing should be worn dependent on the climatic region, e.g. tropical jungle boots for wet jungle conditions, together with strong cotton shirts and trousers with as many pockets as possible. When working in temporary pits or sampling coal, protective headwear and footwear are essential.

The principles of working in hostile terrain are a combination of common sense and experience (or learning from an experienced colleague). The adoption of routines to check one's own health and hygiene are essential at the end of a day's fieldwork. This greatly facilitates the next day's work and monitors any potential problems that might arise, e.g. the daily foot inspection is important as the feet take the brunt of fieldwork, particularly in mountainous and jungle areas. Carelessness about one's health can result in discomfort and an inability to carry out fieldwork, and in extreme cases it gives rise to the necessity to leave a remote field area, which may be difficult and expensive. Although such situations are more likely to occur in geographically difficult areas, such routines are still good practises in logistically accessible countries. In addition to health, at the end of each day's fieldwork all equipment should be checked for damage or losses; all notebooks and field record sheets and maps should be clearly labelled and kept dry and as secure as possible.

Temporary and permanent campsites must be kept clean; food and equipment should not be left lying about. Kitchen, washing, and toilet areas should be clearly separated and adhered to; toilet areas should never be in an upstream position to the campsite.

Campfires must be carefully controlled, especially in areas in which the vegetation is likely to ignite easily. Carelessness in this respect can result in destruction of habitat and/or property, and even loss of life to plants and animals in the area.

Each day's work should be planned in sequence, with time allowed for travelling, sampling, and data recording. It is important to remember that the field mapping is the only method apart from drilling by which basic information on the geology of an area is obtained. All later studies are only as good as the original information collected.

### 6.2.1 Outcrop Mapping

The basic elements of geological mapping, rock identification, and structural measurements have been described by Barnes (1981) and Berkman and Ryall (1987). Mapping is ideally

suited to areas where coal-bearing sequences are exposed due to erosion, folding, and faulting. However, it is common to have to evaluate areas with a scarcity of outcrops to provide at least some basic geological data.

In order to carry out fieldwork the following items of equipment will be needed. Topographical, geological maps and site plans (if any), map case, aerial photographs and pocket stereoscope, 'chinagraph' pencils, notebook/field record sheets, marker pens, geological hammer, chisel, trowel/fold-up spade, polythene bags, sample bags and labels, clinometer/compass, hand lens, small and large tape measures, penknife, and camera. Some of these items are shown in Figure 6.2.

The use of personal and laptop computers has speeded up the field input of data, and readily available software programs provide the ability to construct maps in the field for any desired purpose, e.g. a coal sampling programme.

The first aim of field exploration will be to determine the location, structural attitude, and extent of coals and associated strata, together with structural features such as faults and fold axes and igneous intrusions, all of which, if



**Figure 6.2** Field equipment used by coal geologists.















present, influence future mining conditions. If, for the area of interest, there is already existing geological information available, then further field work may only involve verification of coal seam locations, taking fresh samples, and filling in any gaps in the previous data.

In the absence of published base maps, the traditional method has been to produce plans by field traverses, usually in the form of tape and compass traverses. A long plastic tape measure (30m graduated in centimetres) is used along the traverse in conjunction with a compass bearing at the beginning of each measurement. All such traverses must be connected in closed loops, and the closing errors between the surveys must be corrected before any geological information is plotted. The latter is usually done at base camp. The beginning and end of each traverse must be clearly marked, together with all distinct

physical features, such as hills, river bends, waterfalls, road crossings, and buildings. Geological features, such as thick sandstones, coal seam outcrops, sample locations, faults, and fold axes, will also be put onto the plan. Standard symbols used to portray geological elements on plans, together with mining symbols, are shown in Figure 6.3, and the graphic portrayal of the principal lithotypes found in coal-bearing sequences are shown in Figure 6.4. Figure 6.5 shows the results of a typical traverse survey in dissected terrain using these methods. It is important that these identified features can be revisited at a later date to survey the area and plot elevations accurately using a theodolite.

Field traverses should record all geological features seen; when lithological associations have been recognised they should be linked up wherever possible between traverses.

**Figure 6.3** Symbols for geological maps.

		Geological boundary, definite or indefinite
		Unconformity, definite or indefinite 'u' is towards younger rocks
		Dip (in degrees)
		Horizontal beds
		Vertical beds
		Overtured beds
		Contorted beds
		Anticline
		Overtured anticline
		Syncline
		Overtured syncline
		Monocline

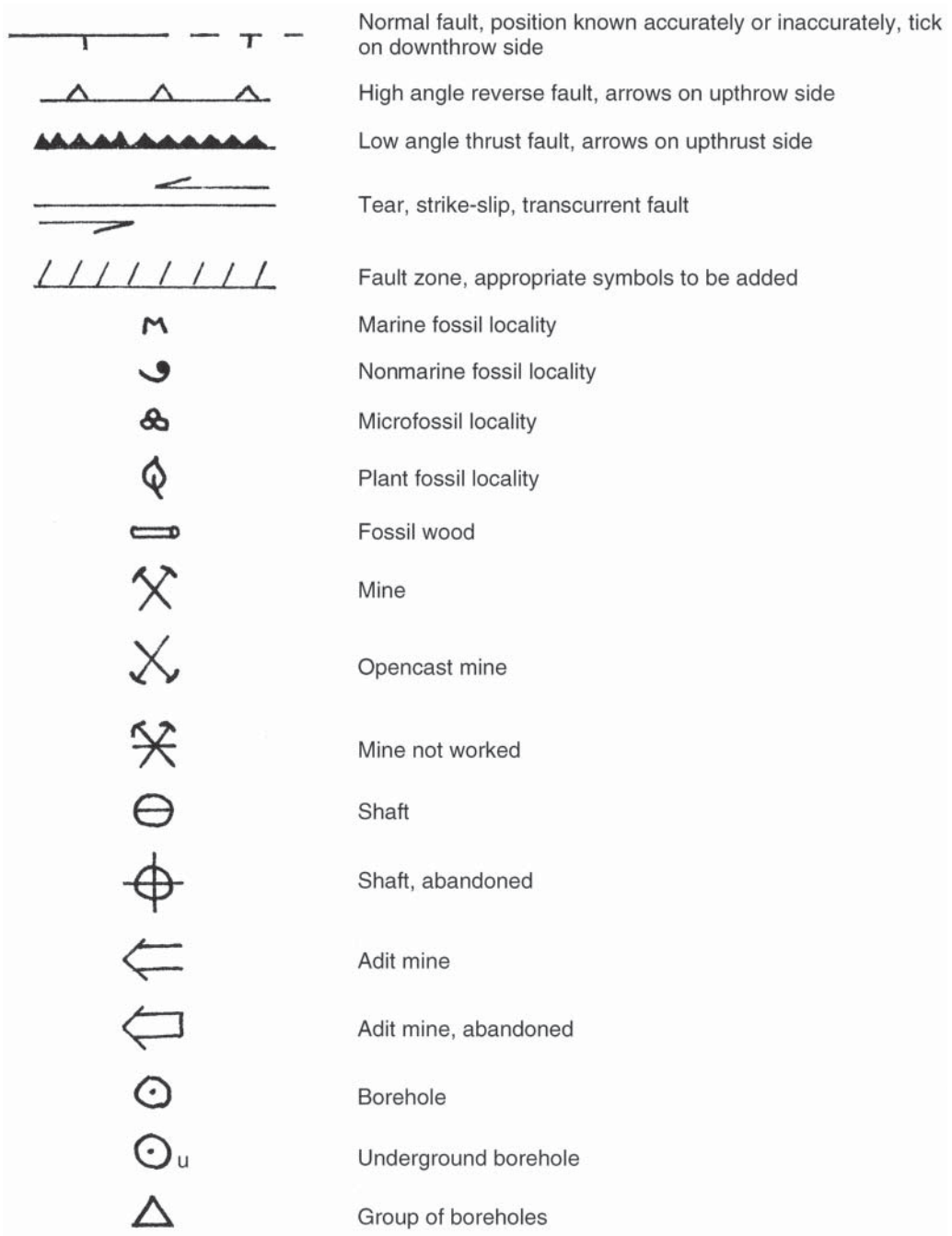
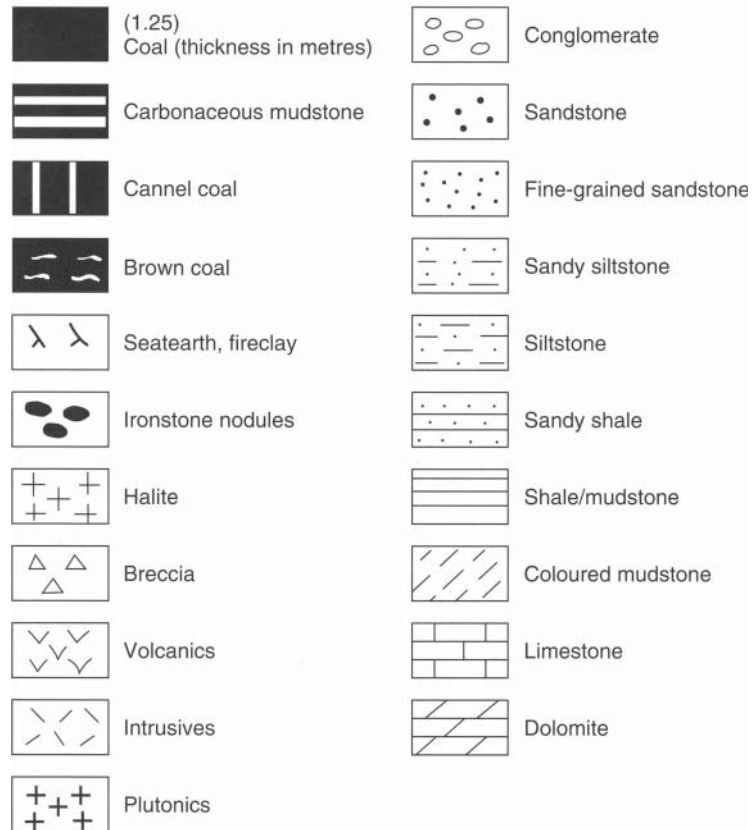


Figure 6.3 (Continued)



**Figure 6.4** Graphic portrayal of principal lithotypes in coal-bearing sequences.



Extrapolation across country from one traverse to another should take into account the effects of dip and topography. Where lines obviously do not tie up, then this may be the effect of faulting and should be checked on the ground and on the aerial photographs.

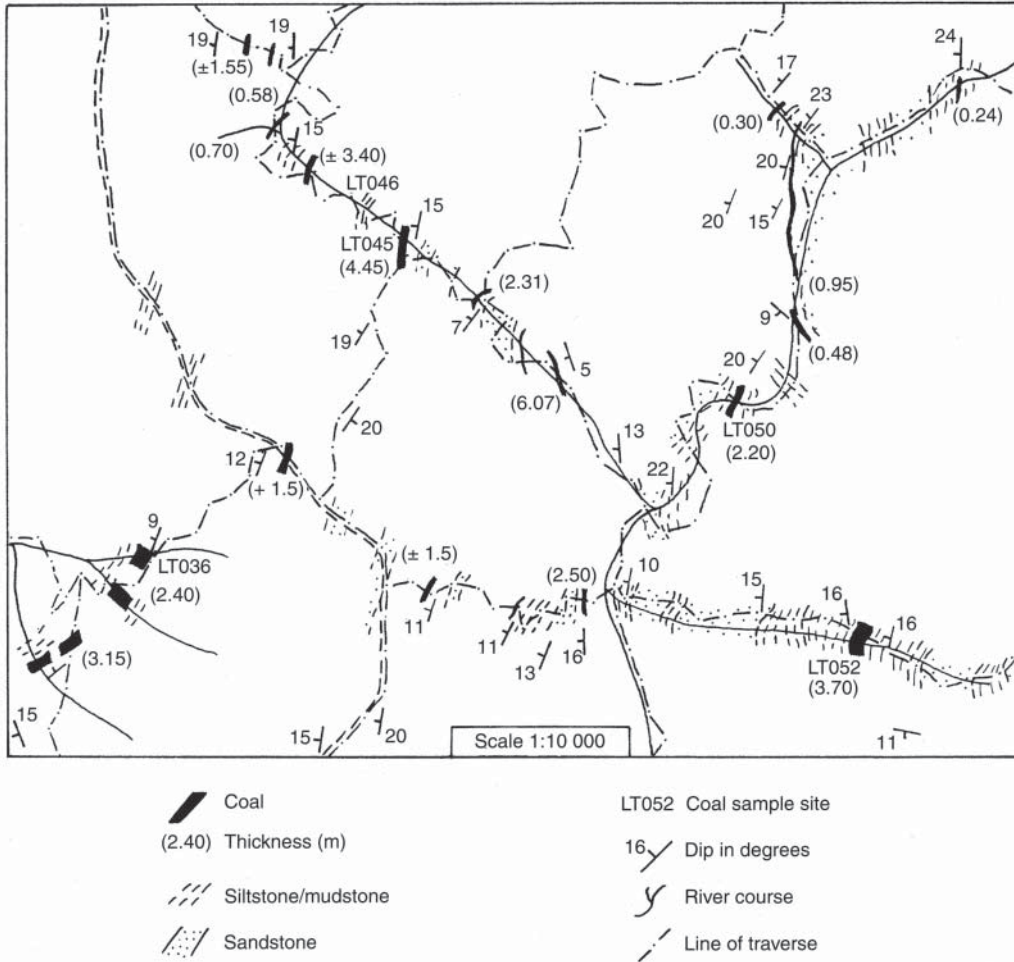
Significant coal-bearing outcrops and the surrounding strata should be mapped in detail; stratigraphic sections should be measured where exposure permits. Individual units, such as coal beds and marker beds, should be traced laterally to ascertain their lateral correlation. Where possible, the environments of deposition should be interpreted during section measuring and coal seam correlation. Care must be taken in measuring sections of strata that may be faulted; low-angle or bedding-plane thrust faults can pass undetected through a coal section, which can result in exaggerated thickness or missing intervals.

Gentle folding can change along strike to isoclinal and recumbent folding.

Recognition and tracing of marker horizons is important; these may be beds of distinctive lithology, such as volcanic deposits or limestone, or beds containing fauna, such as foraminifera, bivalves, and other organisms, or beds containing floral assemblages in the form of plant remains, spores, and pollens. Such distinctive horizons can serve to correlate coal seams, identify structural dislocations, and establish facies patterns.

National geological surveys adopt this approach when mapping coalfield areas; Figure 6.6 shows a typical field map produced by the British Geological Survey for a coalfield area where a large amount of data have been compiled to produce the final geological map.

It is essential to be able to recognise all the lithological types associated with coal

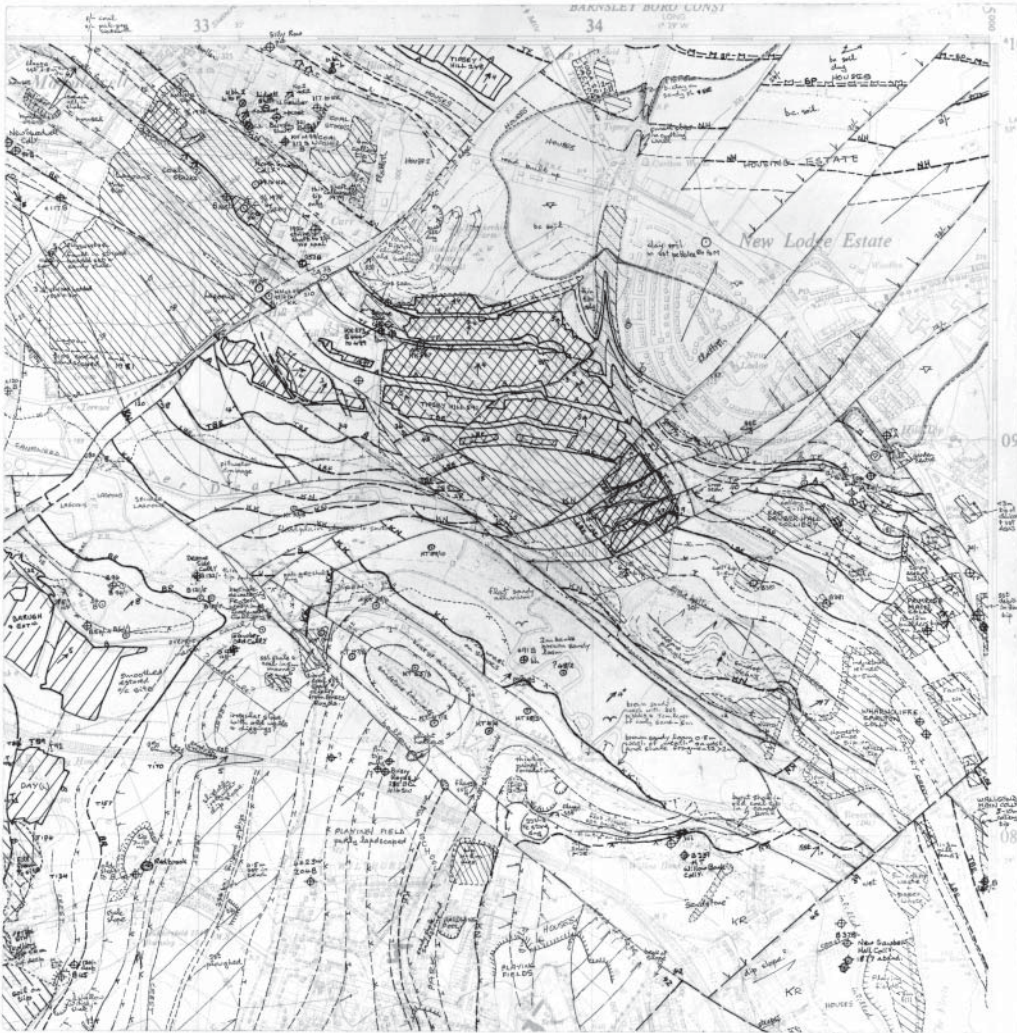


**Figure 6.5** Typical traverse survey showing coal outcrops, sample sites, and structure.

and coal-bearing sequences; the ability to do this undoubtedly improves with experience. In the USA, handbooks have been produced illustrating in colour all the lithologies found in coal-bearing sequences in the eastern USA (Ferm and Smith 1980; Ferm and Weisenfluh 1991). These are invaluable to the inexperienced geologist, and, for the most part, can be used on a global basis, allowing for the occasional local lithological term. The descriptions of the varieties of sandstone, siltstone, shales, mudstones, and coals and their intimate inter-relationships are a fundamental part of the data recording stage of fieldwork.

The lithological description of coal itself is essential to the understanding of the physical subdivisions (plies) of the coal. Most coals have components of bright and dull coal ranging from all bright to all dull with combinations of both in between. Brightness in coal is indicative of the ash content: the brighter the coal the lower the ash content.

Examples of bright coals are shown in Figure 6.7. A banded bituminous coal showing pyrite mineralisation on the cleat face from the Carboniferous of the UK (Figure 6.7a), a bright non-banded high-volatile coal of Palaeogene–Neogene age from Indonesia



**Figure 6.6** Field map of a UK coalfield area, showing geology and past and present coalmining activity; scale 1:10 000.

(Figure 6.7b), a bright high-rank anthracite from the Jurassic, Republic of Korea (Figure 6.7c), and a banded, high-ash Gondwana coal from India (Figure 6.7d).

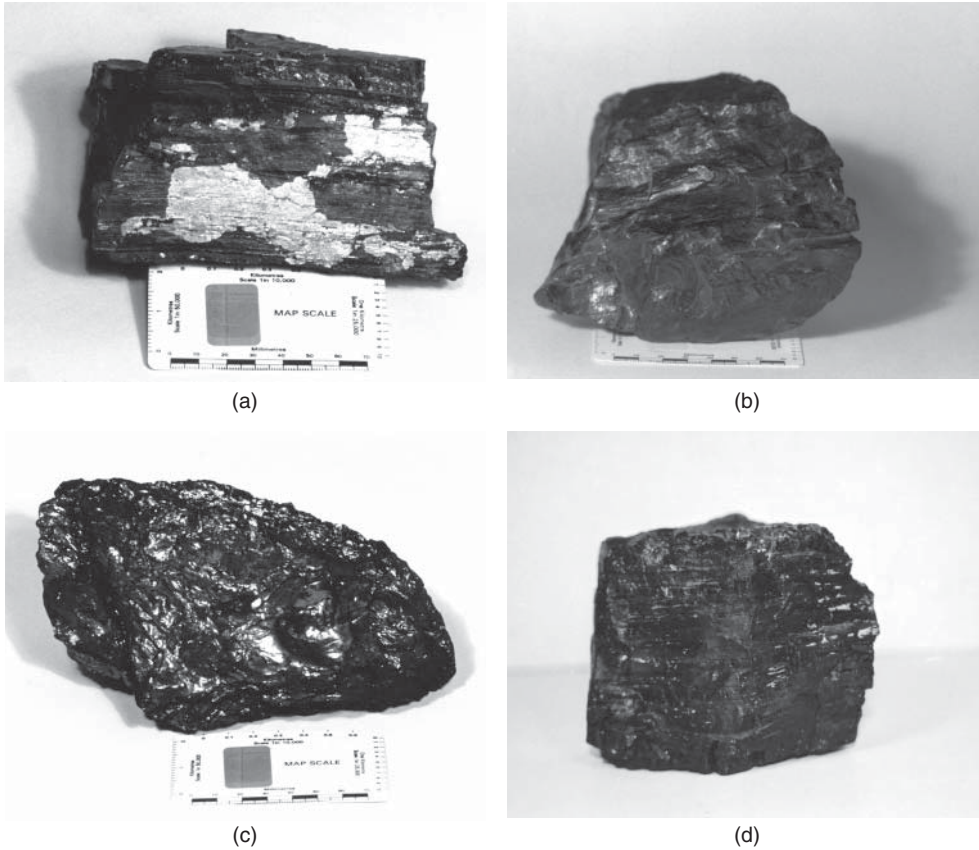
In Australia, the Australian Standard AS2916-1986 (revised 2007) gives the graphic representation of coal as shown in Figure 6.8. A shading, ruling, and letter system is used, beginning with bright coal (>90% bright coal) ranging to dull coal (<1% bright coal); also illustrated are symbols for cannel coal and weathered and heat-altered coals.

The system used in South Africa closely follows the Australian standard, and this is also the case for countries influenced by these areas, e.g. Indonesia.

In the UK, British Coal has used a system of graphic representation for coals in their underground operations. These range from bright coal to banded coal to dull coal; Figure 6.9 shows the symbols used, together with those for cannel and dirty coal.

In the USA, there is no standardised system for the graphic representation of coals;





**Figure 6.7** Varieties in black (hard) coal. (a) Banded bituminous coal from Northumberland, UK, with pyrite mineralisation on the cleat face; (b) high-volatile, low-ash bituminous coal, non-banded, from East Kalimantan, Indonesia; (c) bright, high-rank anthracite, highly tectonised, from Samcheong Coalfield, Republic of Korea; (d) banded, high-ash bituminous coal from Talcher Coalfield, India. *Source:* Photographs by M.C. Coultas & LPT.

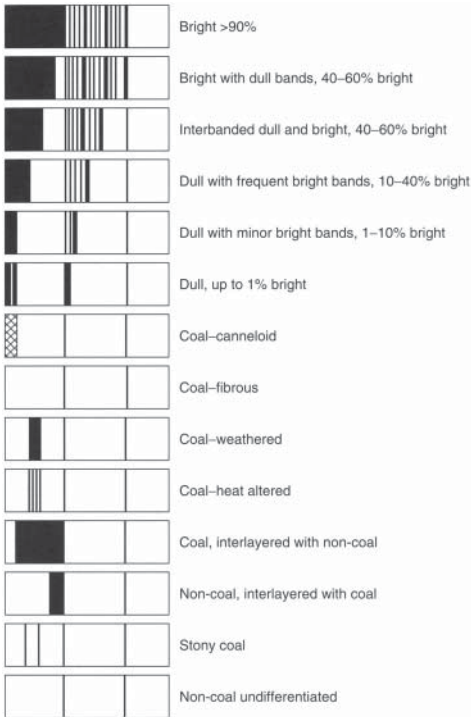
however, an example produced for the American Society for Testing and Materials (ASTM) by Goscinski et al (1978) is shown in Figure 6.10. Coals are described as ranging from bright to intermediate bright, intermediate dull, and dull. Coals with high mineral content (bone coal) are also shown.

In addition to the coal itself, a careful description of the roof and floor of the coal seam is necessary to provide a useful framework for later geotechnical and mining studies.

A description of a coal seam section should include the following:

- 1) Composition of coal seam roof/floor: siltstone, sandstone, carbonaceous shale, etc.

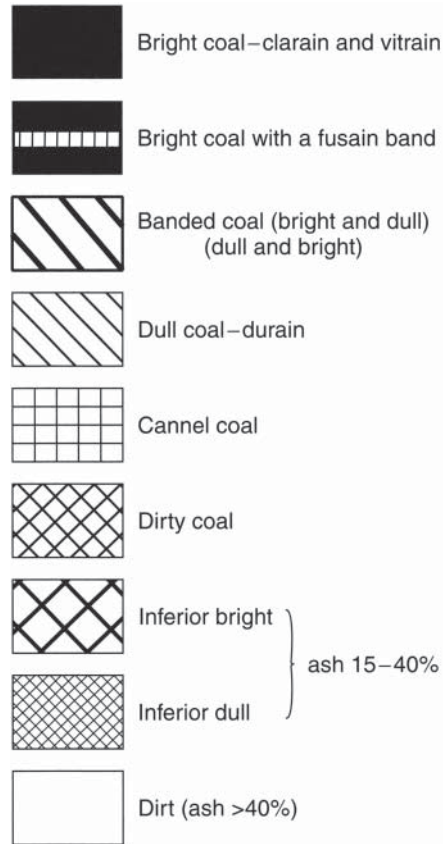
- 2) Structure of coal seam and immediate roof/floor: faults, strike/dip, stratigraphic displacement.
- 3) Coal cleat: face cleat, end cleat, strike.
- 4) Slickensides: frequency, continuity.
- 5) Joints: strike/dip, frequency, continuity.
- 6) Chemical structures: nodules, ironstone, pyrite, concretionary structures.
- 7) Soft sediment structures: slumping, folding, liquefaction structures.
- 8) Degree of weathering: from fresh to completely weathered.
- 9) Roof/floor and/or seam structure: flat, rolling, discontinuities of bedding, splitting.
- 10) Mineralisation.



**Figure 6.8** Graphic representation of coal seams, based on Australian Standard 2916-1986.

### 6.2.2 Global Positioning System

This is a space-based global navigation system that provides reliable location and time information worldwide where and when there is an unobstructed line of sight to four or more GPS satellites. The availability of this system to locate field positions, borehole sites, and other surface features, together with elevation calculations, has revolutionised field exploration, especially in those more remote areas where greenfield exploration takes place. They allow the rapid location and installation of control points for the development of digitally based topographic and geological maps. The readout precision for both elevations and coordinates is generally to 1 m or its equivalent ( $0.000\ 01^\circ$ ) in latitude and longitude (Milsom and Eriksen 2011). A typical hand-held GPS receiver is shown in Figure 6.11a, and these are produced by a number of manufacturers.

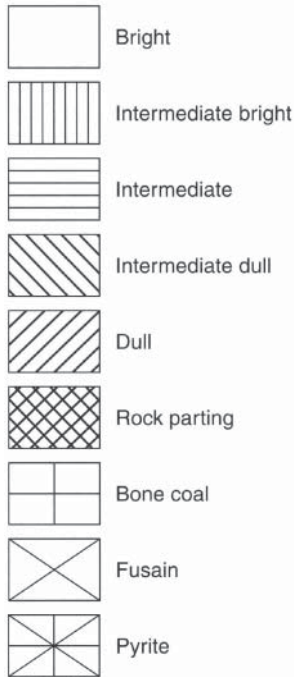


**Figure 6.9** Graphic representation of coal seams as used in UK.

### 6.2.3 Portable Personal Computers

Hand-held personal computers (PCs) or tablets are now widely used, as they are small, light, and portable. These have fast processors with GPS integration and are capable of withstanding all weather conditions. Field data are collected on these, and upon return from the field they are loaded onto a PC workstation for final map preparation. Various models cover a range of software capabilities.

As with GPS receivers, the use of portable PCs is replacing the more traditional methods described in Section 6.2.1. However, the understanding of basic field observation and recording is still a principal requirement for the field geologist. The benefits of using portable PCs are seen by exploration companies to



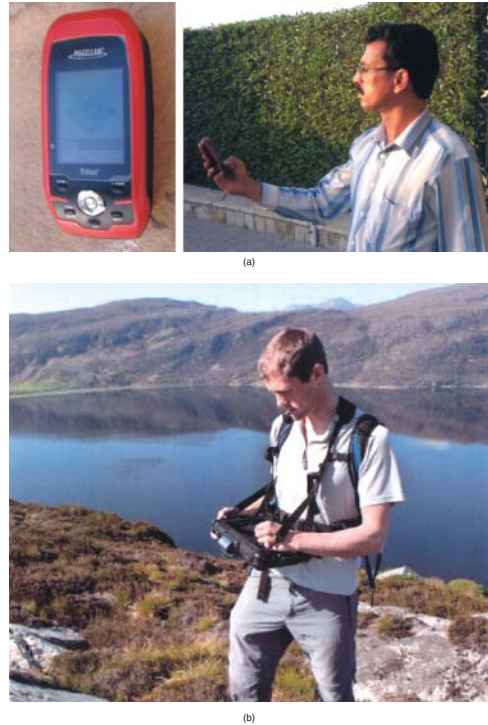
**Figure 6.10** Example of a graphic representation of coal seams as used in the USA. *Source:* From Goscinski and Robinson (1978) ASTM Technical Publication 661.

be the standardisation of data recording by different geologists, the combination with GPS to achieve location and orientation accuracy, and better control on, and faster production of, topographical and geological maps and associated plans.

The British Geological Survey has developed bespoke software for capturing digital geological field data; this enables staff to capture observations and interpretations in the field and make it readily available for use in modelling and visualisation systems. Figure 6.11b shows a tablet PC in use in the field.

### 6.2.4 Remote Sensing

Remote sensing is the acquisition of information about an object or phenomenon without making physical contact with the object and thus in contrast to on-site observation.



**Figure 6.11** (a) Hand-held GPS receiver, Magellan Triton 400. *Source:* Photograph by LPT. Reproduced with permission of Dargo Associates Ltd. (b) Rugged tablet PC for capturing digital geological field data. *Source:* British Geological Survey. © NERC. All rights reserved.

#### 6.2.4.1 Satellite Imagery

The most widely used sources of satellite remote-sensing imagery have been from the Landsat series of the Earth Observation Satellite Co., and from the French SPOT series of spacecraft. Both have worldwide coverage from which data can be obtained in computer-compatible tape and photographic form for ground scenes, each 34 225 km<sup>2</sup> in area. Landsat data are a direct record of the sensor data and can be processed into images that can be enhanced to highlight selected features and enlarged to scales of 1 : 50 000 for map construction or 1 : 24 000 for geological field maps.

Landsat imagery illustrates regional fault patterns and the different geological 'imprint' of a variety of lithological successions within delineated areas. Landsat imagery is provided



in four separate bands of the visible spectrum from blue to infrared. In tropical terrain with thick vegetation cover, the red and infrared spectra are most useful for interpreting geological features.

Landsat interpretation is usually used in conjunction with geological and geophysical maps, if available. As well as highlighting geological structures, variations in rocks, soils, and vegetation are provided in each spectral band. Comparison of data from a number of bands at their different wavelengths allows types of lithology and ground surface cover to be recognised.

SPOT collects data for ground scenes, each of 3600 km<sup>2</sup> in area, and are also available as computer tapes and derived photographic images.

Other sources of satellite imagery include data from a number of international satellites together with photography from manned spacecraft. Both shuttle multispectral infrared radiometer and shuttle imaging radar have geological applications.

Satellite-borne radar has the great advantage of penetrating the persistent cloud cover and forest cover that characterise large areas, particularly those in the tropics.

Lidar uses pulsed laser light and measures the reflected pulses with a sensor. Differences can be used to make digital 3D representations of the target. It is used to measure heights of objects on the ground more accurately than with radar technology.

#### 6.2.4.2 Airborne Imagery

Aerial photographs have been available for the last 90 years, although not for all areas of the world. Photogeological mapping is an established technique in reconnaissance, as it provides an economical and effective map on which geological data are located to ground features, particularly the topography and structural style of an area. Such a framework can then be used for more detailed coalfield exploration in the form of ground surveys.

The use of overlapping aerial photographs to provide stereoscopic (3D) interpretation of areas will give a more detailed geological

picture. The scale of the photographic coverage, usually 1:50 000, but can be 1:25 000, provides a basis for making photointerpretation maps of prospective coal deposits by defining regional geological features such as fault lines and zones, persistent lithological horizons, major fold patterns, and amounts and changes of dip and strike. The symbols used to portray such geological features on plans are shown in Figure 6.12.

When no, or inadequate, aerial photographic coverage is available, it is necessary to prepare a new set of photographs to work from. This enables a ground crew to mark out 'targets' on the ground that can later be accurately surveyed and used as permanent reference points. The scale of the photographs should be of the same order of magnitude as the scale required for the ground plans. For the compilation of maps and plans from aerial photographs, see Barnes (1981) and Berkman and Ryall (1987).

Dense vegetation cover can mask the ground surface, but it can still reflect closely the underlying geology. When combined with fieldwork, subtle lithological differences become apparent. Such photographic interpretation is essential when used in reconnaissance and detailed exploration fieldwork, in identifying traverses, and in pinpointing physical features on those traverses, e.g. river sections, road intersections, and coal outcrops. Such interpretation can be applied to locating potential drilling sites.

The drawbacks with aerial photographs are that, in areas of high rainfall, cloud-free conditions are infrequent and so prohibit good photographic coverage of the area. The topography may be severely dissected, so that the photographs are too strongly distorted for photogrammetric mapping. Colour infrared photography may also be used.

Side-looking airborne radar (SLAR) is also used in exploration, but it has poorer resolution than photography, particularly where vegetation and cloud cover limit the use of aerial photographs. However, linear patterns do show up better than actual geological

	<i>Bedding scarps with dip slopes</i>	<i>Colour (if used)</i>
	<10 degrees	
	10–25 degrees	
	25–45 degrees	
	>45 <90 degrees	
	Vertical	
	<i>Bedding traces dip slopes absent or very short</i>	Purple
	<10 degrees	
	10–25 degrees	
	25–45 degrees	
	>45–<90 degrees	
	Vertical	
	Horizontal bedding	
	Overtured beds	
	Generalised dips, undefined, gentle, medium, steep	
	Joints, certain	Blue
	Joints, uncertain	Blue
	Anticlinal axis, certain, uncertain	Red
	Overtured anticline, certain, uncertain	Red
	Synclinal axis, certain, uncertain	Green
	Overtured syncline, certain, uncertain	Green

**Figure 6.12** Photogeological symbols for use on aerial photographs and photogrammetric plans (from various sources).










	Fault, certain, uncertain	Red
	Normal fault	Red
	Thrust fault	Red
	Tear fault	Red
	Lineament of unknown origin	Purple
	Unconformity	Red
	Lithological boundary, certain, uncertain	Purple
	Lithological boundary for superficial deposits, certain, uncertain	Brown
	Dykes, certain, uncertain	Red

Figure 6.12 (Continued)

features. SLAR can be used to compensate for the lack of aerial photographic cover, as it can highlight structural features not seen at ground level, accurately locate structural elements in the field, and it can provide a regional structural framework that can be used for planning field traverses and exploration drilling grids. SLAR has been used extensively in the USA and Australia (Hartman 1992).

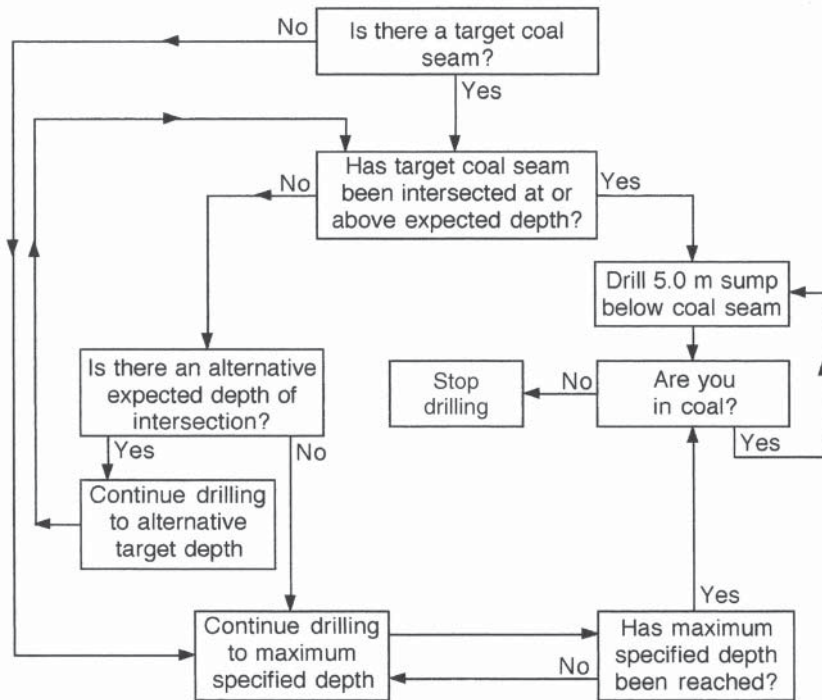
### 6.3 Drilling

Once the field survey has been completed, the position and attitude of all coal seams will be plotted on the base plan. If the dip, structure, and initial quality results indicate that the coal is of economic interest, then the next stage in exploration will be the locating of drill sites to provide data in those areas between known coal outcrops and in areas where no outcrops

have been located but in which coal is thought to occur.

It is important that the geologist maintains a good and close relationship with the drilling supervisor, drillers, and mechanics during the drilling programme. In order to maintain proper records of the drilling operation, the geologist should liaise with the drilling supervisor to ensure that the driller records the following information for each shift drilled: site details, borehole number, details of open-hole drilling, details of hole diameter, details of casing sizes and depths (if used), details of each core run, length of core recovered, details of water encountered, strata description, details of timing of each operation, details of flush losses and bit changes, details of core barrel and core bit type, and details of drilling crew.

If the geologist is supervising junior or less experienced geological staff who may



**Figure 6.13** Drilling procedure to correctly complete a borehole containing one or more targeted coal seams. *Source:* Reproduced by courtesy of M.C. Coultas.

be assigned to a particular drilling rig, the procedure that should be followed is outlined in Figure 6.13. This will ensure that the borehole will be completed in the correct fashion by recording the presence or absence of all coal seams targeted, and that the borehole is drilled to at least 5.0 m below the lowest coal seam to allow geophysical logging tools to record the total coal section.

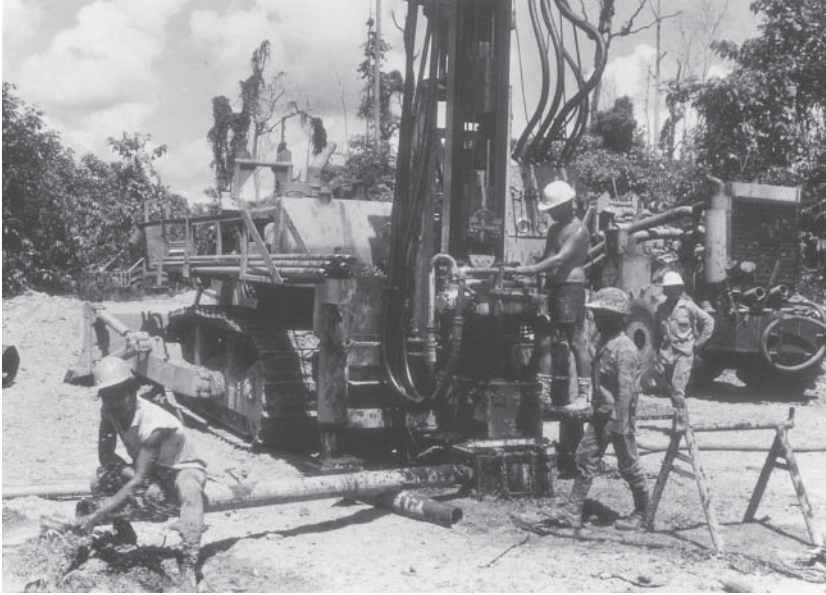
### 6.3.1 Openhole Drilling

Exploration drilling will be carried out to determine coal seam depth, thickness, and quality at any given point within an area, and details of the strata associated with the coal.

For the majority of boreholes drilled in the main area of interest, rotary drilling rigs are used. These rigs have good penetration rates, relatively low cost, and are mobile. They provide the most economical means of shallow (down to 400 m) open-hole and core drilling

of coal deposits. Exploration boreholes are usually vertical, but in areas of high dips inclined boreholes may be drilled, particularly in underground workings.

Rotary rigs are usually truck mounted but can be adapted to fit on a bulldozer (Figure 6.14) or on skids. The rigs can use high-pressure air circulation (air flush), water, or drilling fluid, such as bentonite mud (fluid flush), to cut the rock with tungsten carbide bits. The introduction of high-density polymer foam to facilitate cuttings removal and stabilise hole sidewalls is now replacing bentonite mud. The use of foam has the added advantage of allowing boreholes to be later used as water observation or abstraction wells. In principle, a string of metal rods is rotated axially, and a bit at the base of the string is forced downward under controlled pressure cutting into the sediments, therefore advancing the depth of the hole. Rock cuttings are circulated away from the bit and lifted to



**Figure 6.14** Dando dual air–mud-flush rig mounted on a bulldozer for use in difficult tropical terrain.  
*Source:* Photograph by M.C. Coultas.

the surface by means of the pumped fluid or compressed air. Several types of drill bits are available; the blade bit gives a high penetration rate, needs little maintenance, and the ‘blades’ can be re-sharpened on site. They also have the added advantage of providing larger rock cuttings, so facilitating identification of the lithologies by the geologist logging the borehole. Blade bits are often changed for roller bits when drilling harder strata, such as limestone.

Air-flush drilling rigs have higher penetration rates for non-core rotary drilling, where compressed air is used in place of water to cool the drill bit and flush the cuttings out of the hole. Air-flush also brings up the cuttings much faster, enabling the position of lithological changes to be located more accurately. Air-flush drilling is impeded by high rates of ground water influx; small compressors then find difficulty in maintaining enough pressure to lift the rock cuttings to the surface. However, above the water table, air-flush drilling is particularly useful, although there is the drawback of producing large amounts of dust,

which may be environmentally unacceptable in certain areas. Again, air-flush drilling may be a better option when drilling in zones of burnt coal or broken strata, where there is a likelihood of loss of drilling fluids.

From the point of view of the geologists, water and mud circulation methods are messy and do not allow rapid changes of lithology to be noted easily. The cuttings are slower reaching the surface, which allows a certain amount of mixing, resulting in the less accurate positioning of lithological boundaries. The use of polymer foam has enabled this problem to be overcome and is now used more widely in the industry.

There are numerous makes and types of rotary rigs; examples of those widely used are the Dando (Figure 6.14), Edeco (Figure 6.15a), and Mintec 12.8 (Figure 6.16a) dual air- and fluid-flush rigs and the Mayhew 1000 (Figure 6.15b), a fluid-flush rig. There are now a range of multipurpose rigs for water well and mineral (coal) exploration and geotechnical applications; for example, the Dando Multitec 9000 (Figure 6.16b).





(a)



(b)

**Figure 6.15** (a) Edeco rig operating in the UK; (b) Mayhew 1000 truck mounted rig, operating in tropical conditions, Indonesia. *Source:* Photographs by M.C. Coultas.



**Figure 6.16** (a) Mintec 12.8 exploration drilling rig; (b) Dando Multitec 9000 drilling rig for multipurpose drilling.



(a)



(b)

### 6.3.2 Core Drilling

Cored boreholes are drilled to obtain fresh coal samples and a detailed record of the complete lithological sequence associated with the coals.

Core drilling can be accomplished by using rotary rigs such as the Dando Mintec range of drills, or by diamond drill rigs such as the Boart Longyear and Edeco Stratadrill ranges. These rigs use a tungsten carbide or diamond bit attached to a series of metal rods, the lowest of which is designated a core barrel, and rotated under downward pressure; however, diamond drilling requires the use of drilling fluids. A circle of rock is ground away and a cylindrical core remains in the hollow centre of the core barrel. The recovery of the rock core is facilitated by a non-rotary second metal tube within the core barrel; the core passes into this tube and is protected from damage. This is called a double tube core barrel. Even better core recoveries can be obtained by placing a smooth metal tube, split longitudinally, inside

the non-rotating inner segment of the double tube core barrel; this type of equipment is called a triple tube core barrel.

When the core is to be removed, the split inner tube is withdrawn with the core still inside, it can then be laid out horizontally and the upper half of the inner tube removed to expose the core for examination. It is then transferred to a segmented core box for logging at a later stage.

Removal of the core barrel can be time consuming. Some equipment allows the central part of the core barrel to be drawn up the centre of the hollow drill rods on a steel cable; this technique is known as ‘wire-line’ drilling. The rods themselves are only removed when the drill bit needs to be changed.

The different diameters of core barrels for rotary and diamond drill rigs are given in Table 6.1. As the diameter of the core decreases, there is a tendency for the core to break up and core losses to increase. It is general practice that the core recovery through a coal or coaly horizons should be not less than

**Table 6.1** Core sizes for wireline, conventional, and air-flush drilling.

<b>Wireline core barrels</b>						
Q series (Boart Longyear)	AQ	BQ	NQ	HQ	PQ	
Hole diameter (mm)	48.0	60.0	75.8	96.0	122.6	
Core diameter (mm)	27.0	36.5	47.6	63.5	85.0	
Diamant Boart	ADBG.AQ	BDBG.BQ	NDBG.NQ	HDBG.HQ	PDBG.PQ	SDB.G
Hole diameter (mm)	47.6	59.6	75.3	95.6	122.2	145.3
Core diameter (mm)	27.0	36.4	47.6	63.5	85.0	108.2
<b>Conventional core barrels</b>						
Double tube swivel type	HWF	PWF	SWF	WWF	ZWF	
Hole diameter (mm)	98.8	120.0	145.4	174.5	199.6	
Core diameter (mm)	76.2	92.1	112.8	139.8	165.2	
<b>Air-flush core barrels</b>						
	412F	HWAF				
Hole diameter (mm)	105.2	99.4				
Core diameter (mm)	75.0	70.9				

95%. Boreholes drilled in soft sediments or unconsolidated deposits often have unstable sides, particularly in the top part of the hole. In order for the drill rods to rotate and drill correctly, casing is inserted into the hole to support the collapsing section of the hole. The casing may be of metal or polyvinyl chloride, and it is normally pulled out of the hole once logging has finished.

The core should be placed in a tube of polyethylene sheeting in the core box; this ensures that there is no loss of moisture and that no oxidation of the coal occurs prior to analysis. Ideally, the core should be photographed prior to sealing in the polyethylene using a measuring tape for scale. The core should be labelled with the borehole number and depth indicators.

Core boxes are usually made up of three or four 1 m compartments with lids, usually of wood, as shown in Figure 6.17, but metal ones are used in areas subject to fungal and termite damage.

It is essential that the core is placed in the core box in the correct stratigraphical order; occasionally, drillers put cores in boxes the wrong way up. A comparison with the geophysical log usually shows the error.

The polyethylene sheeting is opened up to log the core, and any core losses are calculated. The roof and floor of the seam are measured; the coal seam lithotypes are recorded in detail together with any partings or splits. In addition, the degree of weathering, mineralisation, and any structural features in the coal and associated strata are recorded.

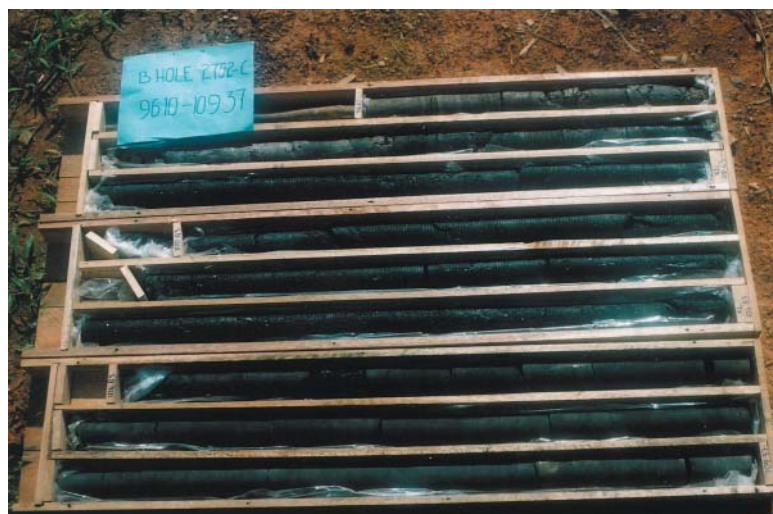
Fully cored boreholes are rare in the exploration phase of a coal deposit; usually, boreholes are only cored for the coals and the coal roof and floor, the depths of these being predetermined by previous open-hole drilling. These part-cored boreholes are sited as 'pilot' holes next to completed open-holes in order to accurately predict the depths and thicknesses of the coal seams. Figure 6.18 shows a typical exploration borehole grid with open-holes and part-cored pilot holes.

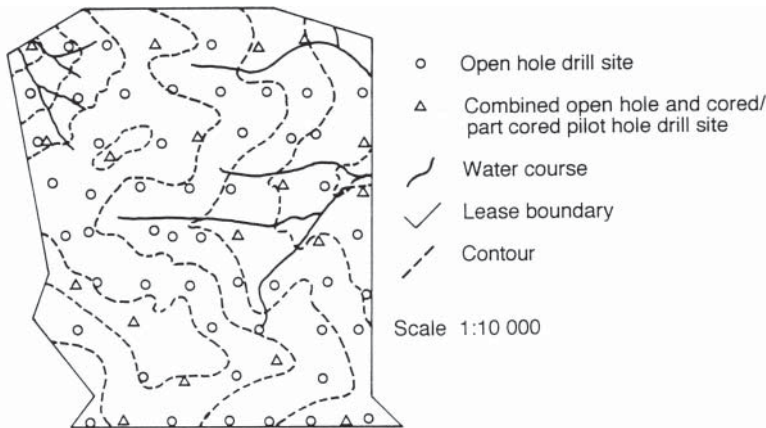
Fully cored boreholes are usually drilled during the geotechnical investigation stage of the project to fully examine the strengths and structural character of the coals and the associated strata.

### 6.3.3 Portable Drilling

Portable drilling, as the name suggests, involves the use of drilling equipment that

**Figure 6.17** Borehole core laid out in wooden boxes with depth markers, awaiting examination by the geologist. *Source:* Photograph by LPT.





**Figure 6.18** Exploration drilling grid showing distribution and position of open holes and part-cored pilot holes.

can be dismantled into person-portable components; this is particularly useful in mountainous and jungle terrain, where access with conventional drilling rigs is difficult. They normally operate by using a small motor (lawnmower or power saw type motor) that drives an axially rotating set of drill rods with a small blade or roller bit at the bottom; the holes are circulated with water or drilling fluid. If the rig motor is not very powerful, then in order to obtain greater penetration the downward pressure may be increased by adding increasing numbers of personnel to sit/stand on the top of the rig, as shown in Figure 6.19.

These small portable rigs are capable of drilling to depths of 60 m and are used to prove coal outcrops, to complete gaps in stream or road sections, and to delineate limits of underground burning zones in coal seams. There are several commercially produced rigs, e.g. the Minuteman and the Voyager V2200 (Figure 6.20), as well as small drills made up by companies for specific use on their own projects.

### 6.3.4 Core and Openhole Logging

#### 6.3.4.1 Core Logging

Because core drilling is an expensive part of the exploration and development programme, cores that are obtained should be logged in as much detail as possible, particularly the coal seams and their roofs and floors.



**Figure 6.19** Portable drilling rig using manpower to exert downward pressure on the drill bit. In use in East Kalimantan, Indonesia. Source: Photograph by LPT.

The total core recovery (TCR) is defined as the proportion of core recovered to the total length of the drilled run. The core run is the length reported by the driller as the actual depth penetrated; this includes both solid core





**Figure 6.20** Voyager V2200 portable drilling rig.

and non-solid core (Valentine and Norbury 2011).

Core logging is usually carried out in a core shed where benching is provided on which are placed those core boxes currently being logged; Figure 6.21 shows typical conditions under which core logging takes place. The core shed also provides storage space for the cores already logged, as well as for new core awaiting examination. Core sheds are important for protecting the cores (and the geologist) from the elements; it is worth remembering that the quality of the core log can vary with the conditions under which the geologist has had to work.

There are three outcomes with core recovery: the core recovery is less than 100% as core has been lost; the core recovery is 100%; or the core recovery is greater than 100%, due to core being lost from one core run and recovered in the subsequent run or from core swelling after recovery (Valentine and Norbury 2011).

The core box will be marked with the depths that the core run commenced and ended; great

care should be taken in relating these markers to the depths and thicknesses of the lithologies actually cored. Core losses may occur, and these should be clearly marked; thickness and depth figures will have been reconciled to the geophysical log, which will have been run after coring has been completed (see Section 8.5).

Where core loss is identified by the TCR measurement, the amount of loss and the depth at which it occurs should be identified as assessed zones of core loss (Valentine and Norbury 2011). The identification of a depth range to zones of core loss allows corrections to be made to the actual depths of the recovered core. The assessed core losses appear as percentages less than 100% of the TCR but provide additional information as to where in the core run the core loss is, with possible reasons as to why such a loss has occurred.

Where possible, a photograph of the cored material with way up and depths clearly marked should be taken for the record, as shown in Figure 6.17, and any special feature



**Figure 6.21** Coal geologist logging borehole core in a core shed on-site. *Source:* Photograph by LPT.

of the coal or its contact with the beds above and below should be photographed. Figure 6.22 shows core exhibiting an erosive sandstone roof to a coal seam. The use of digital photography allows photographs of the core to be reviewed on the computer and then printed as required.

The state and condition of the core should be described; complete solid core is a solid core attached to the roof and floor, and a fragmented solid core is a broken seam but all the fragments join up, so that there is no doubt of the core recovery. Part core indicates that only part of a solid length of core has been recovered but that all lithotypes are present. Fragments indicate that no cores fit together and that it is not possible to accurately state what length of the core the fragments represent.

Once the measurements of seam roof and floor boundaries, major partings, core loss, and any other significant features have been



**Figure 6.22** Borehole core photographed to show special features. In this instance, the erosive sandstone contact above the coal and the siderite nodules contained in the upper part of the coal seam are seen. *Source:* Photograph by M.C. Coultas.

reconciled, and that the individual ply sections have been identified, the core can then be split for detailed lithological logging. Splitting is achieved by using a wide chisel or bolster and hammer to split the core lengthwise, making sure that the split is a fresh surface and does not follow a joint.

Low-rank coals, such as lignite, can be cut lengthwise using a saw; one half can then be examined for colour and lithological variations, whilst the other is left until the core has sufficiently dried out before the texture of the lignite can be described.

Core logging data can be recorded either as a written descriptive log, on printed coding sheets designed for computer usage, or directly into a PC. Figure 6.23 shows a core logging sheet giving depths, lithological description, rock strength, weathering, bedding character, and sample numbers. It is desirable that logging should be carried out to a standard logging procedure to ensure that no element of procedure is omitted and that there is, as far as possible, consistency of approach between different loggers.

As a general guide, the following features should be recorded.

- (i) Lithology: dominant lithotype, colour and shade, grain size and sorting, distinctive



Project Name	Number	Depth, Diameter	Easting	Northing	Elevation	Total Depth	Depth to Base of Weathering	Depth to Water Table	Core Size (mm)	Incl. Azim.	Notes				
664	8711				27.45	115.72			63.5						
Commenced 12/11/87 Completed 15/11/87 Drill Contr. 15187342 Geoph. Contr. 15187342 Flats Casing Depth Other Records HOLE No. 664 871 PAGE 40F 3 ENCODED BY JF															
Depth to Base of Unit	Recovered Thickness	Rock Type	Colour	Lith. Adh. Size	Grain Size	Min. Rock St.	Mass %	Bedding	Bed. Sp.	Base	Dir.	Qual. Adh. A. Adh. Qual.	Form-ation	Seam	Sample Number
60.90	0.61	LG	DB			S5		XL IL	DF	10			LNK		664 871 1132
62.01	1.03	LG	DB			S5		XL IL		10			LNK		664 871 1133
65.40	3.39	CL	DB			S5		XL TM					LNK		
65.55	0.15	CL	DB			S5		XL TM					LNK		
65.70	0.15	CL	MB	SF		S5		XL		IN			LNK		664 871 1135
66.75	1.05	LG	DB			S5		XL		10			LNK		664 871 1136
67.70	0.95	LG	DB			S5		XL					LNK		664 871 1137
68.72	1.02	LG	DB			S5		XL		5			LNK		664 871 1138
69.80	1.00	LG	DB			S5		XL		5			LNK		664 871 1139
70.90	0.90	LG	DB			S5		XL		5			LNK		664 871 1140
72.19	2.29	LG	DB			S5		XL		5			LNK		664 871 1141
74.52	1.30	LG	DB			S5		XL		5			LNK		664 871 1142
76.00	1.48	LG	DB			S5		XL		5			LNK		664 871 1143
82.00	6.00	LG	DB			S5		XL		F3			LNK		664 871 1144
88.00	6.00	LG	DB			S5		XL		5			LNK		664 871 1145
94.00	6.00	LG	DB			S5		XL		5			LNK		664 871 1146
95.18	1.18	LG	DB			S5		XL		F3			LNK		664 871 1147

**Figure 6.23** Example of a core logging sheet used by the coal geologist in the core shed. Key to core logging sheet: CL, clay; CT, clayey lignite; IS, ironstone; LG, lignite; D, dark; BK, black; BF, buff; DB, dark brown; EB, grey-brown; LG, light grey; MB, medium brown; MG, medium grey; OB, orange-brown; RB, red-brown; FC, finely comminuted; SR, sideritic; SF, smooth; ST, silty; WD, woody; FM, fine-medium; S, slightly weathered; R1, very weak rock; R2, weak rock; R3, moderately weak rock; R4, moderately strong rock; S5, very stiff soil; IB, interbedded; TN, thin interbeds; BU, towards base of unit; MU, towards middle of unit; TM, towards middle and top of unit; XN, very thin interbeds (20–60 mm); IL, irregularly laminated; MS, massive bedding; XL, thinly laminated (<6 mm); DF, diffuse base; IN, inclined base; F3, medium spaced (200–600 mm); F4, closely spaced (60–200 mm); L, low angled; V, vertical; W, low and medium angled; C, common; S, sparse; CY, clayey; LM, laminated; LN, lignitic; WL, woody lignite. *Source:* Reproduced with permission of Antrim Coal Company Ltd.

- mineralogy and cementation, associated lithotypes and their relative proportions.
- (ii) Sedimentary characters: thickness and type of bedding and lamination, types and dimensions of cross-bedding, bioturbation, disturbed bedding, contacts with units above and below, fossil content.
- (iii) Mechanical characters: degree of weathering, degree and types of fracturing, orientation and frequency, strength characteristics, mineralisation.

**6.3.4.2 Openhole Logging**

The logging of open boreholes involves the identification of rock chippings collected for

every metre drilled. The chippings are washed, laid out on polyethylene sheets on benches, and then examined by the geologist, as shown in Figure 6.24. The basic lithology is recorded and the depth at which the predominant lithology in the chippings changes. The accuracy of the depths of the top and bottom of important lithologies, such as coals and thick sandstones, can be reconciled when compared with the geophysical log of the borehole. The lithologies are recorded on data sheets in exactly the same fashion as cored boreholes, except that there will be less geological detail for the individual lithologies encountered.



**Figure 6.24** Coal geologist logging borehole chip samples in an on-site core shed. *Source:* Photograph by M.C. Coultas.

Open borehole logs are important as they give the best indication of where to site cored boreholes, and to predict the depths at which coals can be expected to occur; this is particularly important for the siting of part cored boreholes in which only coal cores are required for analysis. The coal geologist can expect to spend the greater part of their time logging open boreholes in the exploration stages of a project.

## 6.4 Geotechnical Properties

The geotechnical logging of surface exposures in trial pits, dug sections, and more particularly in fully cored boreholes is an integral part of the overall geological studies of a coal deposit prior to the engineering studies to determine mine design and the specification of the coal product to satisfy market requirements.

The geotechnical logging of surface excavations is similar to that of cored boreholes: those at the surface will tend to give greater information in a lateral sense, whereas borehole cores give better control in a vertical sense.

The following logging and recording procedures will be carried out by the coal geologist. The results will be incorporated into full geotechnical studies of potential opencast and underground mine developments, in addition to the addition of hydrogeological and geophysical data if available.

The fully cored boreholes drilled during the exploration and reserve proving stages of mine development provide a large amount of information useful to the geotechnical engineer. Therefore, it is part important to record as accurately and as detailed as possible all the relevant geological information. As well as the basic lithological data, the detailed recording of discontinuities (their type, attitude, spacing, and density) will provide valuable data to ensure safe mine design and working methods. The normal indices when logging rock core are solid core recovery (SCR), rock quality designation (RQD), and fracture spacing index (FI).

The drilling history of each borehole may indicate where drilling difficulties, loss of circulation, and core losses have occurred.

These may take on a new significance as the data are evaluated. The potential for inducing fractures into the core is greatest during the extrusion and subsequent handling and transport of the core. The use of core barrel liners is recommended, as this not only enhances recovery but minimises damage to the core during extrusion and transport.

The general description of the rock materials can be as follows: strength (e.g. moderately weak), weathering (e.g. fresh), texture and structure (e.g. thinly cross-laminated), colour (e.g. light grey), grain size (e.g. fine to medium), name (e.g. sandstone), and other properties (e.g. slightly silty with mudstone laminae).

Definitions of the various levels of strength, weathering, texture, structure, and grain size vary. Those given in the following are the definitions given in BS ISO 5930 (1999). Description of the engineering properties of rocks is well documented in Hoek and Brown (1980) and Fookes (1997), and logging techniques are described by Deere (1964), the Geological Society Engineering Group Working Party (1970), and West (1991).

### 6.4.1 Strength

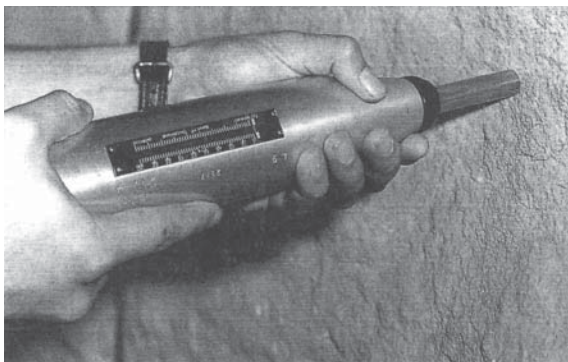
There are several kinds of rock strength, which can only be accurately determined by testing in the laboratory. These are designed to measure both the stress needed to rupture a rock and the strain developed during the application of

stress. It may be argued that, in the absence of laboratory test results, the estimates of material strength are subjective. However, they should always be made, using the guidelines provided, as it is possible to use the later laboratory results from selected samples to 'calibrate' the logger's assessment of the material strength.

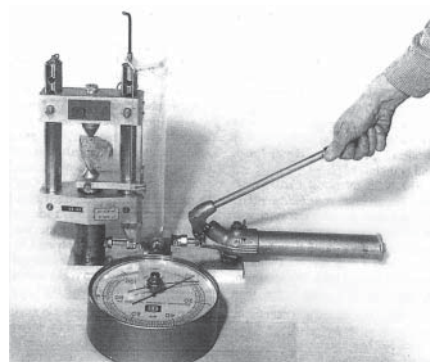
There are two commonly used field methods of assessing the compressive strength of rocks: the Schmidt hammer and the point load test.

The Schmidt hammer (Figure 6.25a) is a portable impact rebound instrument used on rock in situ. The degree of rebound  $R$  is expressed as a percentage of the initial extension of the spring and is read off a scale from 10 to 100. The compressive strength can be read from a correlation curve (Figure 6.26) for type N Schmidt hammer. Uneven surfaces should be avoided, and at least 10 readings should be taken at different points to characterise location. However, absolute accuracy is generally low when testing rock. If the Schmidt hammer has a light alloy housing, it cannot be taken down a coalmine unless a brass housing is made for it.

The point load strength test is a rapid and simple destructive method to measure strength of rock. The equipment (Figure 6.25b) can be used in the field and in the laboratory. It is used for measuring the crushing strength of a hand sample of rock; the sample, c. 5 cm in size, is placed between two conical test platens

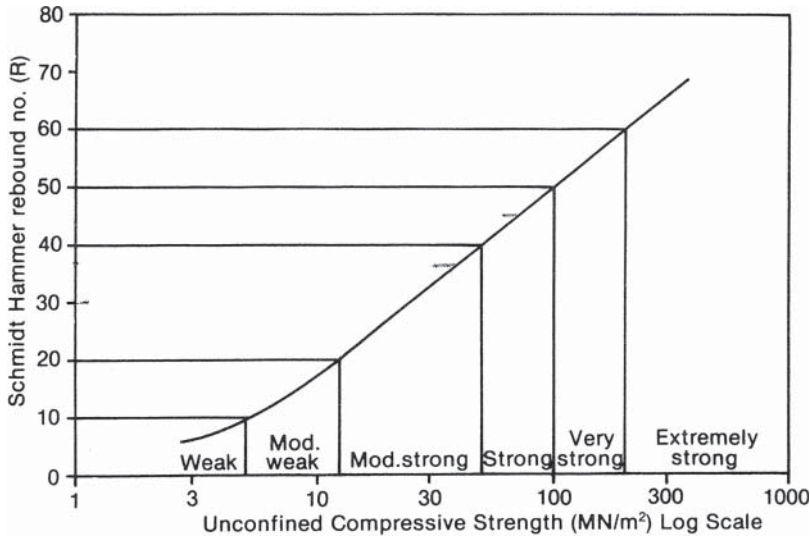


(a)



(b)

**Figure 6.25** (a) Schmidt hammer (West 1991); (b) Point load tester (West 1991).



**Figure 6.26** Correlation curve for type N19 Schmidt hammer used with Carboniferous rocks (West 1991).

of the apparatus and a load is applied by a hand-operated hydraulic pump. The hydraulic pressure at failure of the sample is indicated on a pressure gauge, which indicates the maximum reading. Failure load  $P$  is obtained by multiplying the pressure gauge reading by the cross-sectional area of the ram, the value of which is engraved on the apparatus. Point load strength  $I = P/D^2$ , where  $D$  is the distance between the platens at the end of the test and corresponds to the thickness of the rock tested. The value of  $D$  can be read off the apparatus on a scale provided (West 1991).

Field guidelines to the assessment of material strength by minimal inspection and handling are given in Table 6.2.

#### 6.4.2 Weathering

The degree of weathering is an important element of the full description of the rock material and should always be included. Omission of any reference to weathering should not be taken as an implication of fresh material; this is particularly true of coals.

Weathering is important because it has a direct effect upon the strength of the rock. It may indicate the movement and chemical action of ground water, either through the rock fabric or along open discontinuities. The

presence of weathering in the rock profile will indicate the likelihood of oxidation in any coal seam within this sequence. In old mine workings, the degree of weathering will help to indicate the state of the rock mass around existing or closed voids.

In the examination of rock types, the terms given in Table 6.3 can be used to describe the degree of weathering.

#### 6.4.3 Texture and Structure

Reference to the bedding spacing will be sufficient to describe the texture and structure of the rock mass (Table 6.4).

In rocks that are heavily fractured, sheared, or faulted, mention of this could draw attention to a particularly weak mass strength. For example, strong (intact strength), weak (mass strength), slightly weathered, indistinctly thinly bedded, heavily sheared, light grey, fine sandstone with some soft clay associated with shears. Note that the term 'heavily sheared' is not a standard term but serves to draw attention to the weak rock mass.

#### 6.4.4 Colour

Colour is the most subjective of any observation made during the logging of lithotypes,



**Table 6.2** Terms used to assess material strength in the field.

Description		Characteristics of rock
<b>Strength of rock material</b>		
R7	Extremely strong	Great difficulty in breaking with hammer, hammer rings.
R6	Very strong	Requires several hammer blows to break.
R5	Strong	Requires one hammer blow on hand-held sample to break.
R4	Moderately strong	Hammer pick indents c. 5 mm. Cannot be cut with a knife.
R3	Moderately weak	Hammer pick indents deeply; difficult to cut with a knife.
R2	Weak rock	Rock crumbles under hammer blows; cuts easily with a knife.
R1	Very weak	Broken by hand with difficulty.
<b>Cohesive soil and clay</b>		
C4	Stiff	Can be indented by thumbnail; cannot be moulded with fingers.
C3	Firm	Can be moulded by strong finger pressure.
C2	Soft	Easily moulded with fingers.
C1	Very soft	Exudes between fingers when squeezed.
<b>Non-cohesive soils</b>		
S4	Weakly cemented	Lumps can be abraded with the thumb.
S3	Compact	Not cemented but would require pick for excavation.
S2	Loose	Could be excavated with a spade.
S1	Very loose	Hand could penetrate 'running sand'.

**Table 6.3** Terms used to describe degree of weathering in the field.

Description		Characteristics of rock
W6	Fresh	No discoloration and maximum strength.
W5	Slightly weathered	Discoloration along major discontinuity surfaces; may be some discoloration of rock material.
W4	Moderately weathered	Discoloured; discontinuities may be open with coloured surfaces; rock material is not friable but is noticeably weaker than fresh material.
W3	Highly weathered	More than half the rock material is decomposed; discoloration penetrates deeply, and the original fabric is only present as a discontinuous framework; corestones present.
W2	Completely weathered	Discoloured; decomposed and in a friable condition, but the original mass structure is visible.
W1	Residual soil	Totally changed; original fabric destroyed.

and uniformity between geologists is difficult to achieve. There are several published rock colour charts, e.g. the *Rock Color Chart* published by the Geological Society of America, and these can be used together with supplementary terms (e.g. light, dark, mottled)

and secondary descriptors (e.g. reddish, greenish). Colour is important to note as it can be an aid to correlation, and during the course of an investigation it may become apparent that a distinctive coloured horizon is of particular significance.



**Table 6.4** Terms used to describe the bedding spacing in the field.

Description	Spacing (m)
Very thick	>2.0
Thick	0.6–2.0
Medium	0.2–0.6
Thin	0.06–0.2
Very thin	0.02–0.06
Thickly laminated	0.006–0.02
Thinly laminated	<0.006

#### 6.4.5 Grain Size

Grain size is important in the description of rock and soil material; its omission can only be justified when describing mudstones, claystones, shales, and siltstones in hand specimens. In conglomerates and breccias, the sizes of clasts should be included.

Typical grain sizes are as follows: conglomerates, larger clasts in a finer grained matrix 2.0 to >20 mm, sandstones 0.06–2.0 mm, siltstone 0.002–0.06 mm, and mudstone/claystone <0.002 mm.

Grain shapes include descriptions of angularity (e.g. angular, subangular, subrounded, and rounded) and form (e.g. equidimensional, flat, elongated, flat and elongated, and irregular).

#### 6.4.6 Total Core Recovery

This is the length of core recovered, both solid stick and broken, expressed as a percentage of the full core run. It is a simple percentage figure entered onto the log. When core is lost, this will be less than 100%, but it can be greater than 100% if dropped core is overdrilled and recovered. The percentage core recovery is important as, in general, recoveries less than 95% are not accepted and a redrill may be required.

#### 6.4.7 Solid Core Recovery

This is the total length of pieces of core recovered that have a full diameter, expressed as a

percentage of the full core run. Like TCR, this can be less or more than 100%.

#### 6.4.8 Rock Quality Designation

Deere (1964) proposed a quantitative index of rock mass quality based on core recovery by diamond drilling. As a result, RQD has come to be very widely used and has been shown to be particularly useful in classifying rock mass for the selection of tunnel support systems (Hoek and Brown 1980).

RQD is calculated by taking the total length of core recovered for a length of 100 mm or longer, and at least 50 mm in diameter, expressed as a percentage of the full core run, with only core lengths terminated by natural fractures being considered.

RQD (%)

$$= \frac{100 \times \text{Length of core in pieces} > 100 \text{ mm}}{\text{Length of borehole}}$$

As with TCR and SCR, percentages can be more than 100%, but RQD cannot be greater than SCR. For a core with one large fracture along the entire length, 0% should be recorded. RQD is sometimes expressed for lithological units rather than core runs. RQD descriptions are 0–25% very poor, 25–50% poor, 50–75% fair, 75–90% good, and 90–100% excellent.

#### 6.4.9 Fracture Spacing Index

This is the number of fractures per metre of core, it is defined for lithological units and is independent of core run or recovery. If in one lithological unit there is a marked change in the fracture spacing, then the FI for sub-units should be given. An upper limit to the FI should be defined, above which the index is not calculated but is recorded as being greater than the defined limit (e.g. >25). Also, non-intact material should be recorded separately. This applies to sections of lost core, core badly broken and disorientated during drilling, and non-cohesive broken core from fault zones or old mine workings.

**Table 6.5** Terms used to describe discontinuities in the field.

No.	Depth	Type	Dip	Azimuth	Description	Aperture	Infill
1	24.82	B	20	90	Planar, smooth	T	—
2	25.31	Fr	30	90	Irregular	2.0	Clay
3	26.18	Fr	30	320	Stepped, rough	T-O(4)	Broken rock

### 6.4.10 Fracture Logging

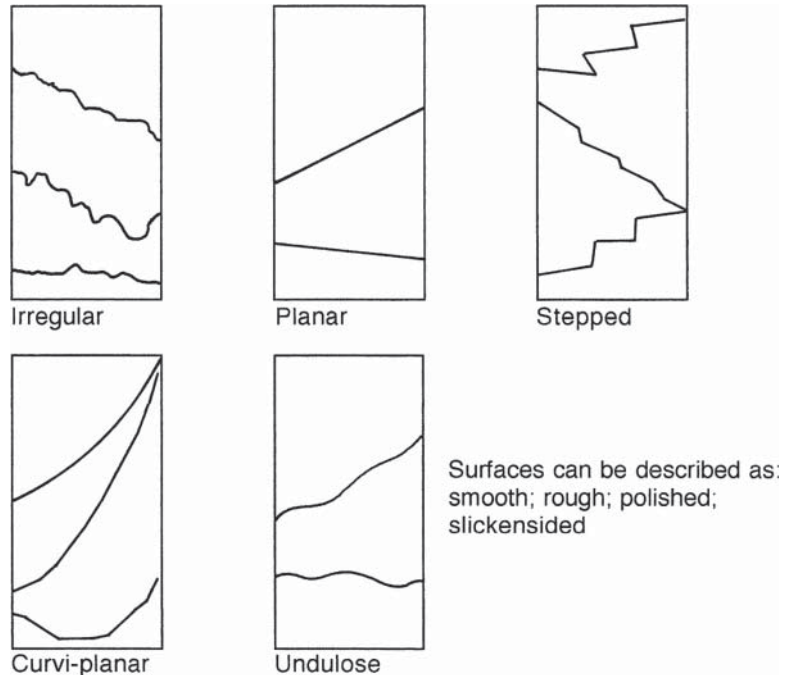
The clearest way of presenting fracture details is to draw a graphic log alongside the lithological description. The log shows the exact position of the discontinuity, which is numbered and described on the separate fracture log. It is common practice to describe each discontinuity as in Table 6.5.

In Table 6.5, the number is the reference number given on the lithology log, and type is the type of discontinuity, e.g. B is bedding, J is joint, F is fault, S is shear, Fr is fracture, FrZ is fracture zone, SZ is shear zone, and FZ is fault zone. The aperture is usually described according to its width, e.g. T is tight, Open 2.0 (with maximum opening in millimetres),

and T-O(2) is tight to open (with maximum opening in millimetres). West (1991) gives aperture widths as follows: closed, 0.1 mm; tight, 0.1–1.0 mm; open, 1–5 mm; wide > 5 mm.

The dip is the angle between the discontinuity and the plane perpendicular to the core axis, and the azimuth is the angle between the bedding and the dip of the discontinuity measured clockwise, looking from the top of the borehole. Discontinuities are commonly described as irregular, planar, stepped, curvilinear, and undulous; and if seen, the surface can be described as smooth, rough, polished, or slickensided (Figure 6.27). The aperture is usually described according to its width: closed, <0.1 mm; tight, 0.1–1.0 mm; open, 1–5 mm; and wide, >5 mm (West 1991).

**Figure 6.27** Types of discontinuities, including bedding and joints that have no displacement, and faults and shears that have a measurable or unknown displacement. *Source:* Reproduced with permission of Dargo Associates Ltd.



ANYMINE		Easting - Northing - Elevation -		Borehole 1 Sheet 4 of 4					
IN SITU TESTS	CORE RUN DEPTH	TORR	SORR	FOOR	LSI	DEPTH	DESCRIPTION OF STRATA	LEVEL	FRACTURE LOG
						6	Dark grey, thinly bedded, fresh SILTSTONE, moderately strong. Closely spaced undulating, 10° bedding fractures, smooth, tight.		
		96	96	84		31.32	grading into		
						4	Grey, fine and medium, medium bedded, fresh, quartzitic SANDSTONE, strong. Medium spaced, irregular 10° to 15° bedding fractures, rough tight and some very closely spaced irregular 80° fractures, tight to open (2mm) with a little iron staining		
	water at 22.90m. water at 16.42m.	32.60	26/5/90			9			
						9			
34.076						34.16	Basal contact 10° open		
34.16	1/2/1					8			
34.16	Sample 1/2/2	98	72	64		34.55	Black, bright and lustrous, thinly laminated, fresh COAL weak, subvertical cleat with occasional pyrite.		
34.55						8	Black, dull, thinly laminated, fresh COAL moderately strong ... 35.26 to 35.30 black carbonaceous mudstone		
34.55 to 35.92	Sample 1/2/3	35.60				NE			
35.92	1/2/4					35.92	Basal contact adheres, 10°		
36.08						9			
						4	Dark grey, thinly bedded, fresh, carbonaceous, clayey SILTSTONE, weak. Closely spaced, 10° bedding fractures, smooth tight.		
		96	88	88		37.06	Dark brownish grey, indistinctly bedded, fresh CHALYSTONE, weak with many rootlet and plant remains.		
						6	Dark grey, thinly bedded, fresh SILTSTONE, moderately strong, closely spaced, planar 10 to 15° bedding fractures, smooth tight. ... 37.41 to 37.80 irregular, 70° fractures, tight to open (2mm), iron staining.		
		38.60				38.60	End of Borehole.		

34.16 to 35.92 Seam 2  
 See sheet 1 of 1 for drilling or instrumentation details.  
 Borehole backfilled with bentonite cement grout.

Figure 6.28 Example of a geotechnical logging sheet. Source: Reproduced by courtesy of M.C. Coultas.

The infill is a description of any material filling the aperture, usually materials like clay, calcite, and broken rock.

An example of a combined geotechnical logging sheet is shown in Figure 6.28.

#### 6.4.11 Rock Mass Rating

No single method is adequate as an indicator of the complex behaviour of rock mass surrounding an underground excavation. Two classifications are generally used: one proposed by Bieniawski (1976) of the South African Council for Scientific and Industrial Research (CSIR), and one by Barton et al. (1974) of the Norwegian Geotechnical Institute (NGI). The CSIR classification uses five basic parameters: strength of intact rock material, RQD designation, spacing of joints, condition of joints, and groundwater conditions. A series of importance ratings are applied to the parameters, and the number of points or rating for each parameter are added together to give an overall rating with adjustments for joint orientation (Bieniawski 1976). The NGI proposed an index for the determination of the tunnelling quality of a rock mass (index  $Q$ ), whereby

$$Q = \frac{\text{RQD} \times J_r \times J_w}{J_n \times J_a \times \text{SRF}}$$

where RQD is Deere's rock quality designation,  $J_n$  is the joint set number,  $J_r$  is the joint roughness number,  $J_a$  is the joint alteration number,  $J_w$  is the joint water reduction number, and SRF is a stress reduction factor (Barton et al. 1974). Though rock mass rating values are usually applied to civil engineering excavations, an understanding of the various parameters used is important when planning and implementing underground excavations in underground coalmines.

## 6.5 Computer Applications

There are now a large number of software packages available that provide integrated

geological data management, analysis, and visualisation. Geological data collected in the field in the form of direct observation, sample points, borehole data, survey, and location data can all be entered into the selected software program to produce point data plans, contour plans, survey plans, 3D surface diagrams, and volumetrics, as well as producing cross-sections and vertical logs.

The use of GPS has enabled geologists to accurately position data points in the field. This is particularly important in poorly surveyed areas, such as forested or desert terrain. In the case of rainforest, it is now often possible to get fixes through the canopy (Milsom and Eriksen 2011).

The great advantage with some software programs is that additional data can be readily added or modified and new plans and diagrams produced. Large amounts of field and borehole data can be unwieldy in hard-copy form, and field records are either coded onto forms by the geologist in the field and then entered into the geological database or encoded directly into a personal computer or tablet (Figure 6.11b) carried in the field or on site. This data can then be transferred directly into the geological database for processing and analysis. One of the principal tasks is to ensure that the correct information is contained in the input and that the right type of process is selected to produce correct results.

The principal modelling processes are the gridded model, the block model, and the cross-sectional model. In coal evaluation, the gridded model is normally used, as it is particularly suited to bedded deposits. The advantage of using grids is that it facilitates the manipulation and use of the data. Structures, thicknesses, and other parameters stacked on top of each other can be added, subtracted, multiplied, divided, or compared to arrive at other derived sets of data, and only those parameters of interest need be modelled.

To estimate the value of data for a given position on the grid, i.e. a node, one can use the data points closest to the node. However,

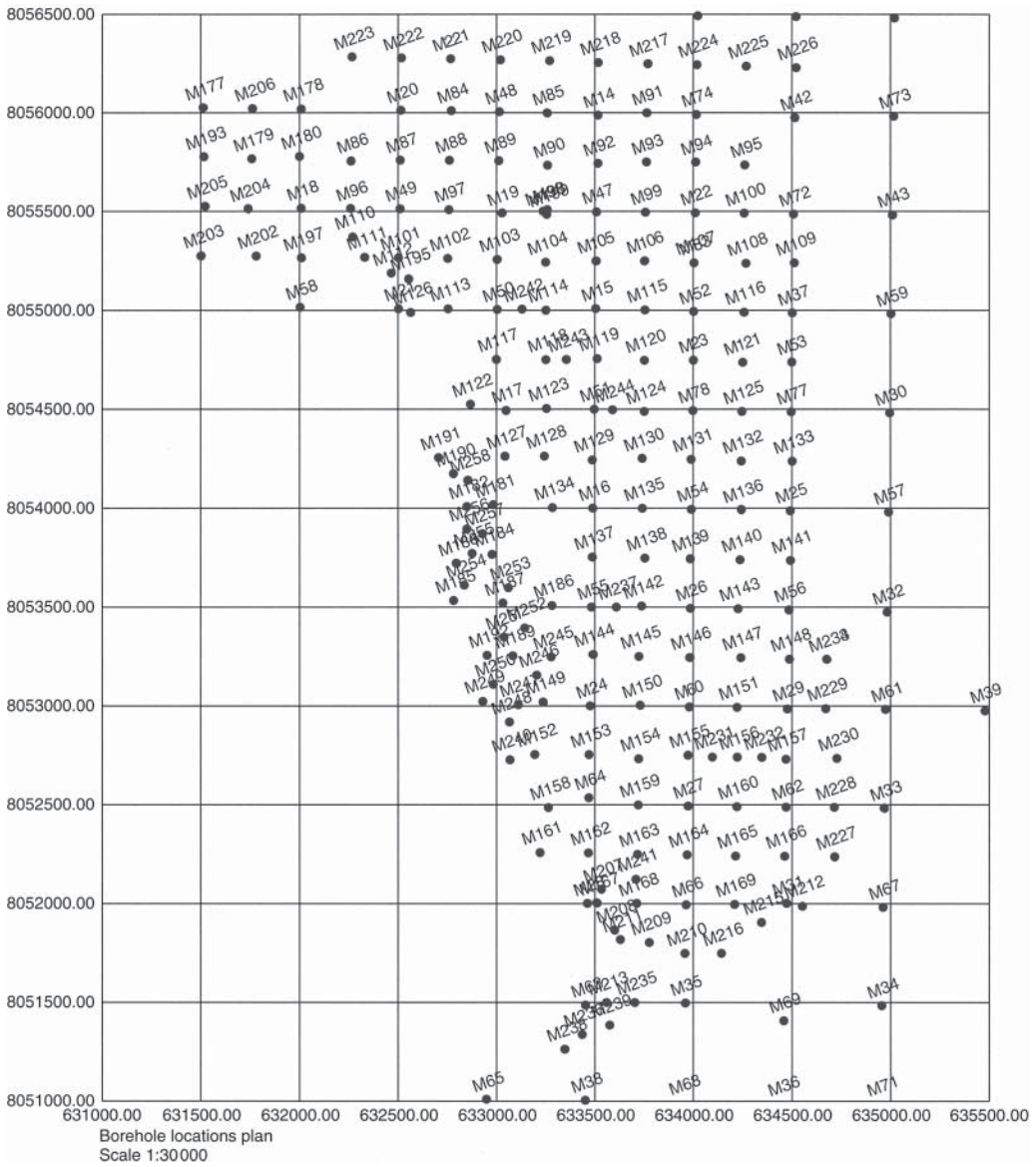


Figure 6.29 Computer-generated borehole location plan. Source: Courtesy of Dargo Associates Ltd.



this can result in the selection of data from only one set of data points, e.g. a line of boreholes or a sample traverse. To reduce the bias, many programs now require selection of data from within a specific search radius and limit the number of data points to one to three points from each quadrant, sextant, or octant. The search radius limits the number of points selected for evaluation, and this in turn reduces the computer search time. If the desired number of points is not found, then a preset increment is added to the original radius and the search is continued. This is especially useful where there is an uneven distribution of data (Hartman 1992). In selecting points for estimation, it is important to obtain a distribution of data from all directions, in order to represent more closely the actual conditions.

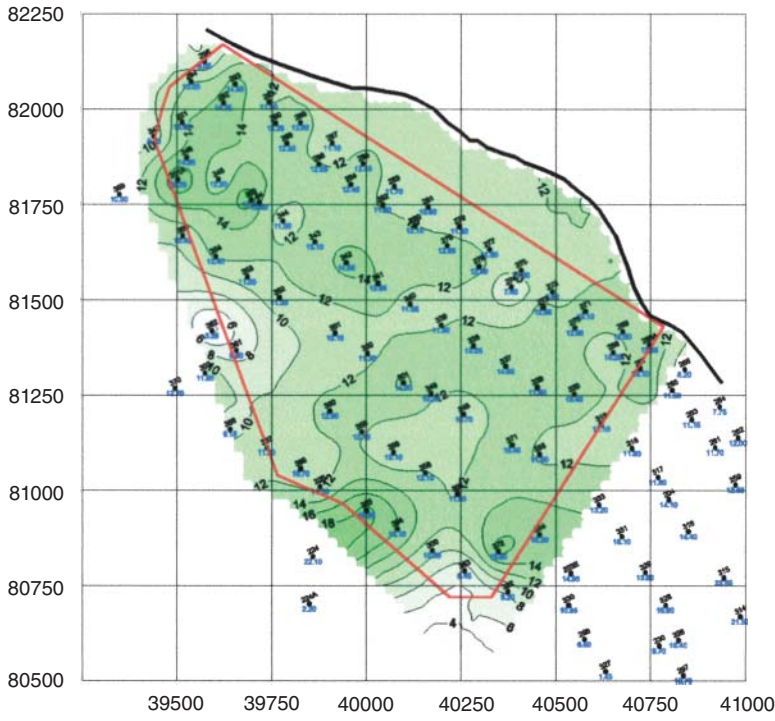
The creation of grids that exhibit the spatial relationships of outcrop, sample, and borehole locations allows the creation of contour plans. Figure 6.29 shows a borehole distribution plan with numbers for each borehole; each location is tied to an  $X$ - $Y$  coordinate. It is essential that a plan of this type has correct locations for all data points; if not, all other contour plans generated from this location pattern will compound any errors. Figure 6.30a shows the coal seam thickness of a single seam dipping to the south-west with the line of outcrop running north-west-south-east. The thickening of the coal shown by the bunching of the contours in the west and south-west is due to the coal seam splitting and including increasing thicknesses of interburden. This can be contrasted with Figure 6.30b, which shows seam thickness excluding any partings or interburden. Errors tend to show up as 'bull's eyes' on the plan. Where this occurs, the data for that location need to be verified. Structural and thickness data are retrieved and modelled first; and if the deposit has a significant dip, a necessary correction has to be made to provide for a true thickness calculation.

Once the location grid is established, it allows the production of plans to show coal seam(s) thickness variations, depth to top and

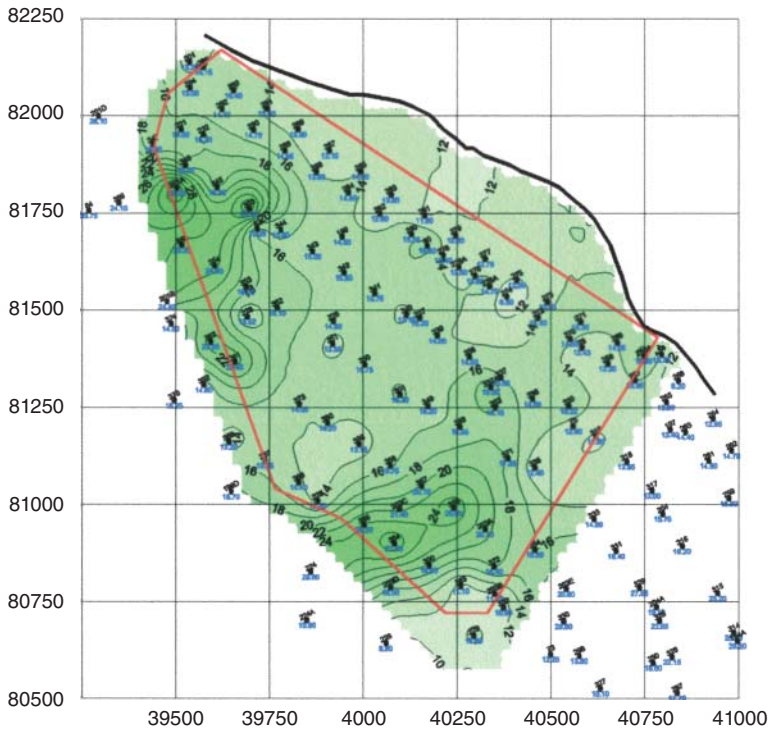
bottom of coal seam(s), interburden thickness variations, and overburden thickness variations. At the same time, coal quality data are retrieved and modelled; initially, this may be for moisture, ash, sulfur and calorific value (CV) to determine the economic potential of the coal seam(s). Figure 6.31a shows a contour plan of total sulfur for the same coal seam shown in Figure 6.30. The higher sulfur content values are concentrated along the south-eastern edge of the plan, coinciding with the increase in interburden within the seam. Figure 6.31b shows the variation in net CV for the same seam; again, the lower net CV values tend to occur where the amount of included interburden is greatest.

All of the gridded data can then be used to calculate coal resources and reserves. The planimeter used for calculating areas, as described in Section 7.3.3, is replaced by a digitizer tablet. Each contour level is traced and the volumetric program uses the contour thickness value and the area traced to calculate the volume. Tonnage calculations require a density factor to convert the volumes to tonnes. This type of calculation is equivalent to the manual method but at much faster speed. All calculations are automatically accumulated so that the results for each coal seam are rapidly available. More complex volumetric programs can be used; these are dependent on the geometry of the deposit. It is easier to use volumetric applications for formations that are flat-lying or uniform in development. Developments from the horizontal gridded technique include the production of cross-section profiles by plotting the grid values along a line or connected group of lines. Other displays include the perspective or isometric view of the gridded surface. Such displays show the surface or variable as a block diagram that can be viewed from any selected point of origin.

A simplified flow diagram illustrating the stages in the processing of geological field data to produce contour plans, profiles, and reserve calculations is shown in Figure 6.32.



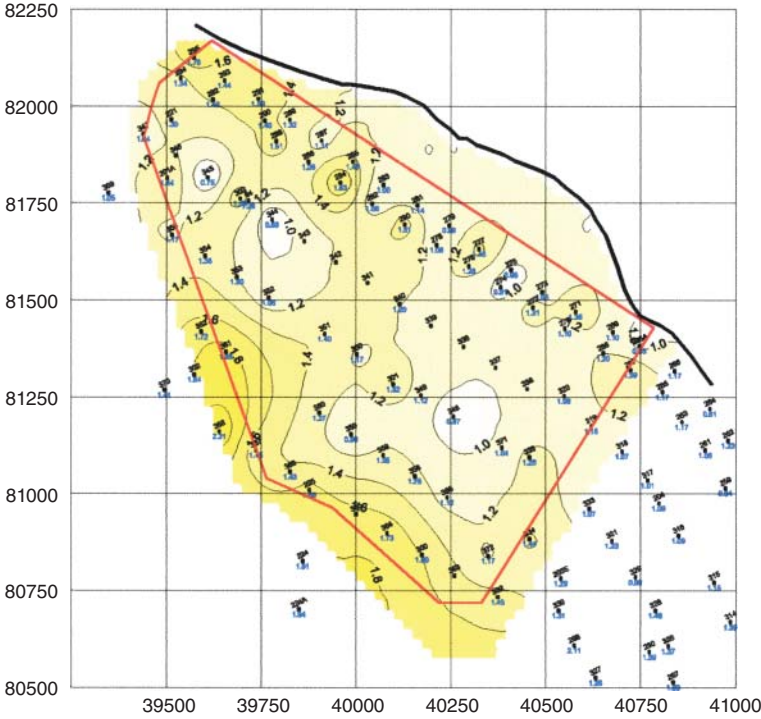
(a)



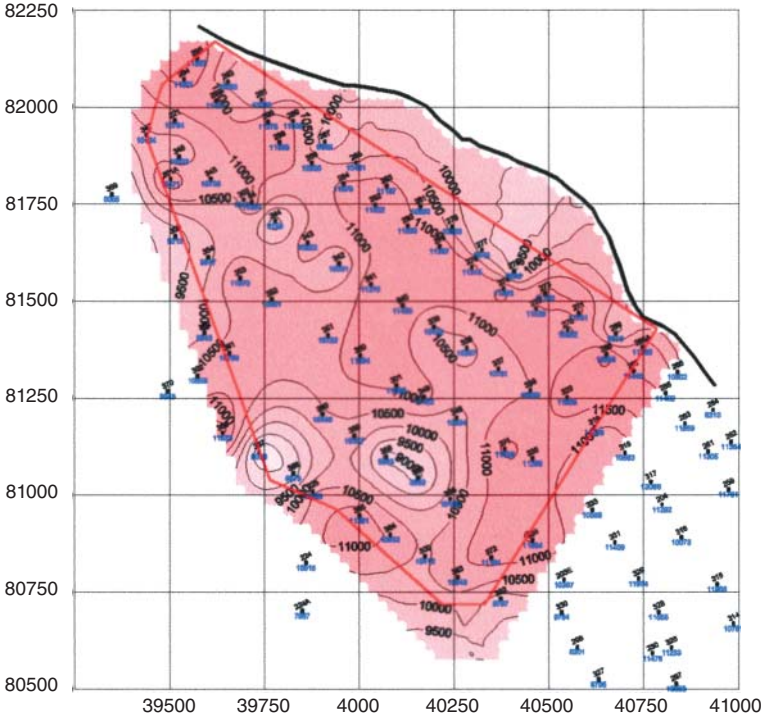
(b)

**Figure 6.30** (a) Coal thickness contour map with borehole locations. (b) Full seam thickness (including partings) contour map with borehole locations. *Source:* Reproduced with permission of Dargo Associates Ltd.

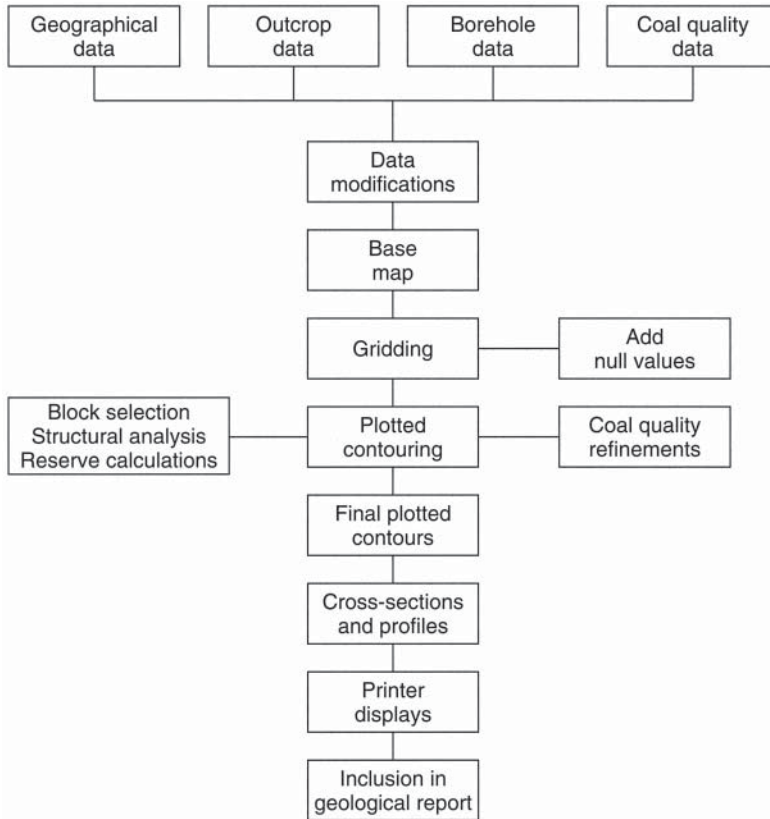
**Figure 6.31** (a) Total sulfur content contour map with borehole locations. (b) Net CV contour map with borehole locations. *Source:* Reproduced with permission of Dargo Associates Ltd.



(a)



(b)



**Figure 6.32** Simplified flow diagram of a geological contouring programme. *Source:* Reproduced with permission of Dargo Associates Ltd.

## 7

### Coal Resources and Reserves

#### 7.1 Introduction

The investigation of any coal deposit is carried out to ascertain whether coal can be mined economically and that a coal product can be obtained that will be marketable. An essential requirement of any coal investigation is that an assessment is made of the coal resources within the area of interest. Such an assessment will influence the decision of whether to develop the deposit, to extend existing mine operations, or, conversely, to curtail mining activity or even to cease development or operations altogether.

In the case of the sale of a lease or mine prospect, the coal resource assessment will play an important part in determining the success or failure of the transaction.

Resources can be divided on the basis of two points of view according to:

- (i) their degree of geologic assurance;
- (ii) their degree of economic feasibility.

There is a third subdivision that distinguishes between the coal in place and the amounts that can be technically recovered. Unavoidable losses during exploitation represent the difference between the two quantities of those in place and those recoverable.

The reliability of a coal tonnage estimate is based on the definition and expression of geologic assurance and the methods of its estimation.

Geological uncertainties pertaining to coal arise from topographical and tectonic variations in the environment at the time when peat was being deposited, and from post-depositional erosion and structural alteration. As described in Chapter 2, the geometry and morphology of coals varies according to the depositional setting in which they were formed. For example, lenticular coals with great variations in thickness will need more data points than relatively undisturbed areally extensive coals of constant thickness. Data-point spacing criteria should take into account such differences in the depositional settings and geological features specific to each coal deposit.

Coal resources categories range from the general evaluation of a coal basin to the calculation of specific reserves located within mine workings. The final result of geological investigation of a coal deposit will be to calculate all categories of coal resources, using the codes of practice adopted by the project management for the lease area under consideration.

It should be borne in mind that, for providing information related to coal supply in the short term, reserve estimates have limitations; here, one is more concerned with capacity and deliverability, whereas resource analysis is valid for longer term assessment, i.e. for 10 years hence and longer.



## 7.2 Coal Resources and Reserves Classification

A number of countries and organisations have cooperated in developing sets of definitions and methods to be used to calculate coal resources and reserves. These include the principal coal-producing countries who have devised codes for the assessment of coal resources to meet their particular requirements; these vary in complexity and degrees of scale. Invariably, all have to meet the standards and requirements for resource and reserve assessments demanded by the individual national stock exchanges and financial institutions. Many of these codes have been updated since the second edition of this book (Thomas 2013a), and reviewed in Thomas (2013b). The principal codes are outlined in the following.

The Australian Code for Reporting of Exploration Results, Mineral Resources, and Ore Reserves was set up by the Joint Ore Reserves Committee (JORC) in 1971. Revised and updated editions of the 'JORC Code' have been issued. The current version dates from 2012 and supersedes all previous editions. Since 1994, the Committee for Mineral Reserves International Reporting Standards (CRIRSCO) has worked to establish international definitions for reporting mineral resources and reserves based on the JORC definitions; these were revised in 2013. At the same time as the development of the JORC Code, a committee of the Council of Mining and Metallurgical Institutions (CMMI) from Australia, Canada, South Africa, USA, and the UK reached provisional agreement on standard definitions for reporting resources and reserves in 1997. It was agreed in 1998 that these definitions be incorporated into the United Nations Framework Classification of Fossil Energy and Mineral Resources (UNFC) developed by the United Nations Economic Commission for Energy (UNECE). The CMMI is now disbanded and the CRIRSCO has remained a separate entity. CRIRSCO's members are

national reporting organisations who are responsible for mineral reporting standards. Currently, the national reporting organisations signed up to the CRIRSCO template are Australia (JORC), Brazil (Comissão Brasileira de Recursos e Reservas), Canada (Canadian Institute for Mining, Metallurgy, and Petroleum [CIM] Standing Committee on Reserve Definitions), Chile (National Committee), Europe (Pan-European Reserves and Resources Reporting Committee [PERC]), Kazakhstan (Kazakhstan Association for Public Reporting of Exploration Results, Mineral Resource and Mineral Reserves), Mongolia (Mongolian Professional Institute of Geosciences and Mining), Russia (National Association for Mineral Resources [NAEN]), South Africa (South African Mineral Reporting Codes), and the USA (Society for Mining, Metallurgy & Exploration [SME]). These standards, codes, and guidelines are now published and adopted by the relevant professional bodies in the aforementioned countries and other countries in Europe. The use of such standards is now required as the basis for resource and reserve assessments by the stock exchanges and financial houses in these countries. The large coal-producing countries of India and China, and other major coal producers in South America and South-East Asia, are now adapting their reporting systems to be compatible with these international guidelines. However, the CRIRSCO template is non-binding and does not account for local regulatory reporting requirements. A further development has been the Australasian Code for the Public Reporting of Technical Assessments and Valuations of Mineral Assets (VALMIN Code 2015), which provides a set of fundamental principles, mandatory requirements, and supporting recommendations accepted as good professional practice to assist in the preparation of relevant public reports on any technical assessment or valuation of mineral resources and reserves. The VALMIN Code uses the same terminology as the JORC Code.

The resource and reserve standards used in the countries of the former USSR, are detailed but were set up under a state-run industry that had different economic parameters. The Russian system is now being equated with the CRIRSCO system for projects requiring international finance. Although other variations of resource/reserve classification codes are currently used and/or are proposed for international use, as described below, the JORC Code is widely used and stock exchanges and financial institutions are now fully familiar with its principles and definitions.

Documentation detailing exploration results, coal resources, and coal reserves estimates from which a public report is produced must be prepared by, or be under the direction of, and signed by a competent person or persons. A competent person must be a professional member of an approved institution or professional organisation and must have a minimum of five years relevant experience in the type of deposit under consideration and who has the responsibility of ensuring that the applicable rules, regulations, and guidelines pertaining to the particular country are adhered to. This is usually a geologist, but it can be another technical expert.

### 7.2.1 Australia

The JORC Code had been adopted by the Australasian Institute of Mining and Metallurgy and the Australian Institute of Geosciences and has been incorporated in the listing rules of the Australian Securities Exchange and the New Zealand Stock Exchange. The main principles governing the application of the JORC Code (2012) are clarity of report, inclusion of all relevant data pertaining to the judgement of the status of mineral resources and ore reserves, and competency (i.e. carried out by suitably qualified and experienced persons approved by a recognised professional body). The JORC Code is applicable to all solid minerals, and for the purposes of public reporting the requirements for coal are generally similar

to those for other commodities. The JORC Code uses the following definitions:

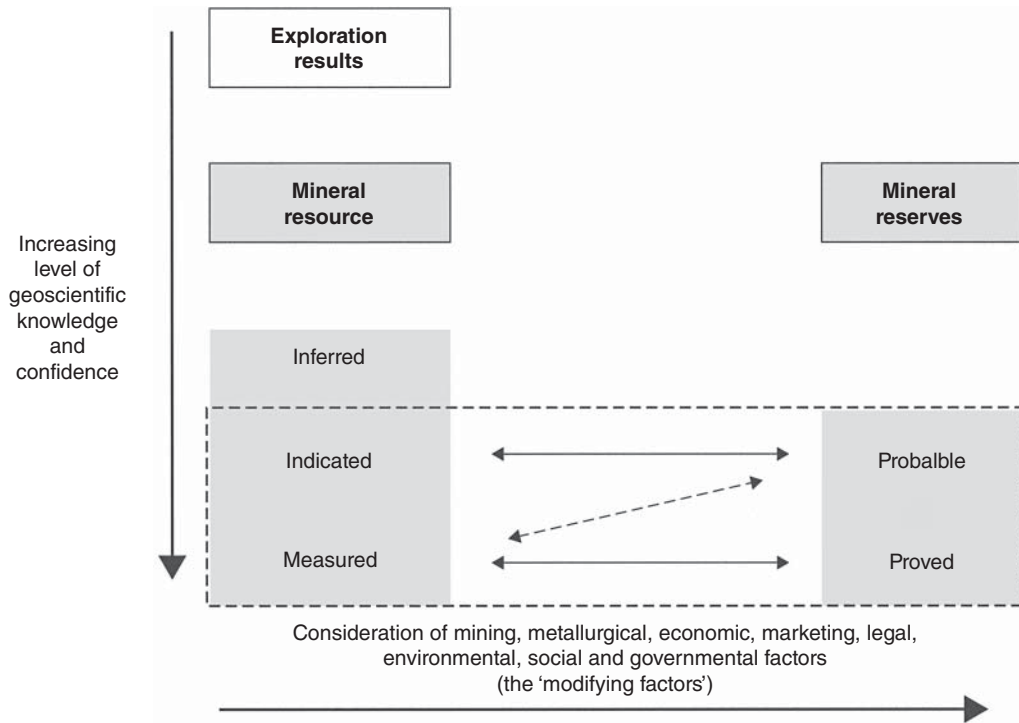
- (i) *Points of observation* are intersections of coal-bearing strata at known locations that provide information to varying degrees of confidence about the coal by observation and measurement of surface or underground exposures, borehole cores, open-hole cuttings, and downhole geophysical logs. Points of observation for coal quality are obtained from exposure and/or borehole core sampling, the latter having an acceptable level of recovery (95% linear recovery).
- (ii) *Interpretive data* are observations supporting the existence of coal and include results from mapping and geophysical surveys.
- (iii) *Exploration target* is a statement or estimate of the exploration potential of a coal deposit in a defined geological setting quoted as quantities and/or qualities for which there has been insufficient exploration to estimate coal resources.
- (iv) *Exploration results* are reports of coal occurrences that, owing to insufficient data, cannot be given a resource value or quality parameters but might be of use to investors.
- (v) *Modifying factors* are considerations used to convert coal resources to coal reserves, which include mining, processing, quality, marketing, environmental, legal, social, economics, and governmental factors.

The relationship between mineral resources and mineral reserves is shown in Figure 7.1

The JORC Code (2012) classification is given in the following two subsections.

#### 7.2.1.1 Coal Resources

- (i) A *resource* is an occurrence of coal in such form, quality, and quantity that there are reasonable prospects for eventual economic extraction.
- (ii) An *inferred resource* is that part of a coal resource for which tonnage and quality



**Figure 7.1** Relationship between exploration results, mineral resources, and mineral reserves (PERC 2008).

can be estimated at a low level of confidence using outcrops, pits, workings, and boreholes. The number and distribution of points of observation plus interpretive data, if available, should provide sufficient understanding of the geology to estimate continuity of coal seams, range of coal thickness, and coal quality. Inferred resources can be upgraded to indicated resources with continued exploration.

- (iii) An *indicated resource* is a coal resource with a higher confidence level. The points of observation plus interpretive data are sufficient to allow a realistic estimate of average coal thickness, areal extent, depth range, quality, and in-situ quantity. Such a level of confidence will be sufficient to generate mine plans and estimate the quantity of product coal and the economic viability of the deposit. An indicated resource has a lower level of confidence than that of a measured

resource and may only be converted to a probable coal reserve.

- (iv) A *measured resource* is a coal resource where the points of observation, which may be supplemented by interpretive data, are sufficient to allow a reliable estimate of average coal thickness, areal extent, depth range, quality, and in-situ quantity. This will provide a level of confidence sufficient to generate detailed mine plans and determine mining and coal beneficiation costs plus the specification for a marketable product and the final evaluation of the economic viability of the deposit. Distance between points of observation may be extended if the competent person considers that any variation to the estimate would be unlikely to significantly affect potential economic viability, and should take into consideration cut-off parameters such as depth or coal quality.

Coal resource estimates are not precise calculations, as they are dependent upon the amount of geological data and the interpretation of such information on the extent, continuity, and thickness of coal seams and their quality. Where the geological character of a deposit is more complex, then an increase in the number and closer proximity of points of observation will be required.

### 7.2.1.2 Coal Reserves

- (i) A *coal reserve* is the economically mineable part of a measured and/or indicated resource. It includes diluting materials and allowances for mining losses defined by studies at pre-feasibility or feasibility level that demonstrate coalmining can be justified.
- (ii) A *probable coal reserve* is the economically mineable part of an indicated, and in some circumstances a measured, coal resource. Allowances are made for losses in mining and dilution. The confidence in the modifying factors (such as mining method, marketing, legal, environmental, social and governmental) applied to probable coal reserves is lower than that applied to proved coal reserves.
- (iii) A *proved coal reserve* is the economically mineable part of a measured coal resource. Allowances are made for losses in mining, and there is a high level of confidence in the modifying factors given for probable coal reserves. Proved coal reserves have been assessed to demonstrate that, at the time of reporting, coalmining can be justified. A proved coal reserve is the highest confidence category of coal reserve estimates.
- (iv) A *marketable coal reserve* represents tonnages of coal that will be available for sale, either as raw coal or beneficiated or otherwise enhanced coal product, where modifications to mining, dilution, and processing have been considered. These may be reported with, but not instead of, coal reserves. The basis on which such

figures have been calculated must be stated.

Coal reserve estimates are not precise calculations; resultant tonnage and quality figures should reflect this by rounding off to appropriately significant figures.

### 7.2.2 Canada

The CIM approved the CIM Standards on Mineral Resources and Reserves – Definitions and Guidelines in 2000. This was updated in 2010 and again in 2014 to reflect the more detailed guidance available and to maintain consistency with current regulations (CIM Standing Committee on Reserve Definitions, 1995).

The mineral resource and reserve categories laid down are similar to those in the CRIRSCO Standard Definitions (CRIRSCO 2012) and JORC Code (2012) and require the same level of competent person affiliated to an approved professional institution. As in JORC, resources are reported in ascending order of technical certainty as *inferred*, *indicated*, and *measured resources*. Reserves are also reported as *probable* and *proven reserves* using the same criteria as in JORC (2012) (Figure 7.1). The CIM guidelines are slightly more rigorous in terms of what a reasonable level of confidence entails. Spacing must be close enough for geological and quality continuity to be *reasonably* assessed.

All technical reports that disclose information about exploration or other mining properties to the public are governed by a number of regulations in Canada. In 1997, the Ontario Securities Commission and the Toronto Stock Exchange established the Mining Standards Task Force, which in 1999 recommended the adoption of the CIM standards through National Instrument (NI) 43-101.

For coalmining and other mineral properties it is NI 43-101, developed by the Canadian Securities Administrators in 2001 using the *Guide for Engineers, Geologists, and Prospectors Submitting Reports on Mining Properties to*

*Canadian Provincial Securities Administrators* (National Policy Statement No. 2) and the *Use of Information and Opinion Re Mining by Registrants and Others* (National Policy Statement No. 22). In 2011, NI 43-101FI provided standards of disclosure for mineral projects and a summary of scientific and technical information concerning mineral exploration, development, and production activities on a mineral property (Ontario Securities Commission 2011). Guidelines are also set out in Hughes et al. (1989), but these must be in accordance with the CIM definition categories for public reporting.

Other international guidelines imply that there can be no fixed definition for the term ‘economic’, but that it is expected that companies will attempt to achieve an acceptable return on capital invested. In particular, CIM guidelines consider that a comprehensive study of the viability of the coal project must have been performed, which includes mining method, pit configuration, coal preparation (if required), and demonstrates that economic extraction is justified.

### 7.2.3 Europe (Including the UK)

The PERC Code was first published in 2008 and set out the minimum standards, recommendations, and guidelines for public reporting of exploration results, mineral resources, and mineral reserves in the UK, Ireland, and Europe (PERC 2008). The committee comprised personnel from the Institute of Materials, Minerals and Mining, the Geological Society of London, the European Federation of Geologists and the Institute of Geologists of Ireland. This was updated in 2017 and reissued as the PERC Standard and is consistent with the CRIRSCO International Reporting Template and the national codes and standards from which it is derived (PERC 2017). The definitions in the 2017 PERC Standard are identical to the CRIRSCO Standard Definitions (CRIRSCO 2012), and are either identical to or not materially different from

those international definitions; the relationship of coal resources and coal reserves again conform; as shown in Figure 7.1. The 2017 edition supersedes all previous editions and standards; including previous PERC codes, IMM Code, and the Recommended Rules for Public Reporting of Exploration Results, Surveys, Feasibility Studies and Estimates of Mineral Resources and Reserves in Sweden, Finland, and Norway.

Prior to the PERC Code, countries such as the UK and Germany used classifications developed by their publicly owned coal industries, chiefly for the underground mining of black coal (Cook and Harris 1998). Now in Europe, state-run coal companies have either disappeared or now compete with the private sector. All, however, need to work to CRIRSCO Standard Definitions in order to raise finance.

As in the case of the JORC Code, coal resources are subdivided in order of increasing geological confidence, i.e. inferred, indicated, and measured categories. Specific distances between points of observation are not given, but in the case of indicated coal resources the locations are too widely or inappropriately spaced to confirm geological and/or quality continuity but are spaced closely enough for continuity to be assumed. In the case of measured coal resources, the points of observation are spaced closely enough to confirm geological and quality continuity. In deciding between measured coal resources and indicated coal resources, the competent person may consider that any variation from the estimate would be unlikely to significantly affect potential economic viability.

Coal reserves are defined as the economically mineable part of a measured and/or indicated coal resource. They include dilution of materials and losses occurring when coal is mined. Coal reserves are subdivided in order of increasing confidence into probable coal reserves and proved coal reserves. When reporting coal reserves, a clear distinction must



be made between reserves where mining losses have been taken into account (known as recoverable reserves or run of mine) and saleable product, where both the mining and processing losses have been included (known as marketable reserves). The bases used to measure coal quality should also be clearly reported.

#### 7.2.4 South Africa

In South Africa, the South African Code for Reporting of Exploration Results, Mineral Resources, and Mineral Reserves, the SAMREC Code, was drawn up under the joint auspices of the South African Institute of Mining and Metallurgy and the Geological Society of South Africa. This consisted of a working group made up of South African technical institutions, government departments, and legal and financial organisations. First issued in 2000, it was updated in 2007 and 2009. The latest edition (SAMREC 2016) supersedes all previous editions. The definitions and standards used are identical to, or not materially different from, the CRIRSCO Standard Definitions (CRIRSCO 2012) and are applicable to all minerals for which public reporting of exploration results, mineral resources, and mineral reserves is to be made. This is required by the Johannesburg Stock Exchange. The SAMREC Code is virtually identical to the JORC and CIM codes. In addition, the South African Code for the Reporting of Mineral Asset Valuation (SAMVAL Code 2016) sets out minimum standards and guidelines for reporting mineral asset evaluation in South Africa.

#### 7.2.5 United Nations

In 2007, the Committee for Sustainable Energy of the Economic Commission for Europe (ECE) directed the Expert Group on Resource Classification (previously the Ad Hoc Group of Experts on Harmonization of Fossil Energy and Mineral Resources Terminology) to develop a revised UNFC that would have worldwide application. This resulted in the development

of the *United Nations Framework Classification for Fossil Energy and Mineral Reserves and Resources 2009* (UNFC-2009), published as ECE (2010). This was followed in 2013 by publication of the specifications for the application of UNFC-2009 (UNECE 2013). In 2017, the UNFC changed its name to UNFC Classification for Resources, and in 2019 it produced an update intended to satisfy the requirements of different resource sectors and applications without changing the classification system (UNECE 2020). The UNFC-2009 classification is considered to be universally acceptable and internationally applicable for the classification and reporting of fossil energy and mineral resources and reserves, and the ECE believe that this is the only current classification in the world to do so. Additional guidelines for the application of UNFC-2009 for mineral resources in Finland, Sweden, and Norway were drafted in 2017 but are unlikely to impact on the coal sector.

UNFC-2009 applies to fossil energy and mineral resources and reserves located on or below the Earth's surface. It has been designed to meet the needs of applications pertaining to energy and mineral studies, resource management functions, corporate business processes, and financial reporting standards. UNFC-2009 is a generic principle-based system in which quantities are classified on the basis of the three fundamental criteria of economic and social viability (E), field project status and feasibility (F), and geological knowledge (G), using a numerical coding system. Combinations of these criteria enable a 3D system to be constructed (Figure 7.2).

The E category indicates the level of acceptability of social and economic conditions in order to establish the commercial viability of the project, taking into consideration legal, regulatory, and environmental conditions together with market prices. This is subdivided as follows:

$E_1$  category, indicating the extraction and sale of product has been confirmed to be economically viable;

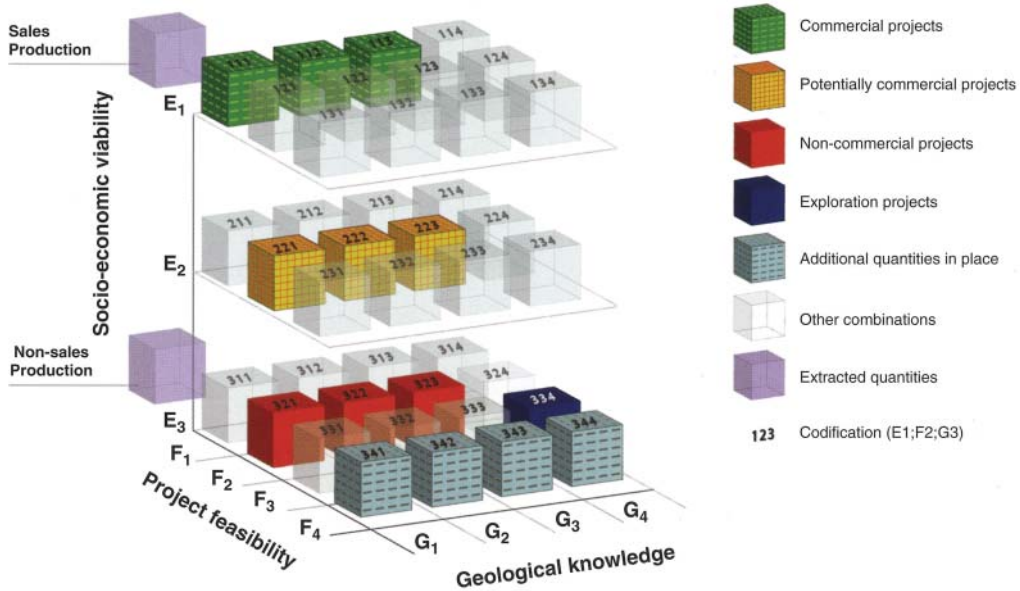


Figure 7.2 UNFC-2009 resource and reserve categories and examples of classes (ECE 2010).

E<sub>2</sub> category, indicating extraction and sale of product is expected to become economically viable in the foreseeable future;

E<sub>3</sub> category, indicating extraction and sale of product is not expected to become economically viable in the foreseeable future or evaluation is at too early a stage to determine economic viability.

The F category designates the level of investigation necessary to develop projects and produce mining plans. These range from early exploration through to a project that is mining and selling a product such as coal. Again, this is subdivided as follows:

F<sub>1</sub> category, indicating the feasibility of extraction by a defined development project or mining operation has been confirmed;

F<sub>2</sub> category, indicating the feasibility of extraction by a defined development project or mining operation is subject to further evaluation;

F<sub>3</sub> category, indicating the feasibility of extraction by a defined development project or

mining operation cannot be evaluated due to limited technical data.

The G category indicates the level of confidence in the geological knowledge and recoverability of the mineral, in this case coal. Each discrete estimate reflects the geological knowledge and confidence associated with a specific part of a deposit. There is always a degree of uncertainty associated with mineral estimates. This uncertainty is communicated by quoting discrete quantities of decreasing levels of confidence (high, moderate, and low). The G category is subdivided as follows:

G<sub>1</sub> category is quantities and qualities associated with a known deposit that can be estimated with a high level of confidence;

G<sub>2</sub> category is quantities and qualities associated with a known deposit that can be estimated with a moderate level of confidence;

G<sub>3</sub> category is quantities and qualities associated with a known deposit that can be estimated with a low level of confidence;

G<sub>4</sub> category is for estimated quantities and qualities associated with a potential deposit, based primarily on indirect evidence.

In addition to the E, F, and G categories, subcategories are represented by a numerical code, with the lowest numbers indicating the highest level of confidence. As the categories are always quoted in the same order, only the numerical codes are used, as these are understood universally. Figure 7.2 shows the relationship of the three categories and

their numerical notation indicating increasing levels of confidence. These can be assigned classes and subclasses to indicate the status of any project; these are summarised in Table 7.1.

The mapping of the CRIRSCO template to UNFC-2009 classes and categories is shown in Table 7.3. The term ‘inventory’ is not a definition in the CRIRSCO template, referring to the situation where the assessment of modifying factors indicates that the project is not viable in the foreseeable future, i.e. does not have ‘reasonable prospects for eventual economic

**Table 7.1** UNFC-2009 classes and sub-classes defined by subcategories (UNECE 2009).

UNFC classes defined by categories and subcategories						
Total commodity initially in place	Extracted	Sales production				
		Non-sales production				
	Class	Sub-class	Categories			
			E	F	G	
Known deposit	Commercial projects	On production	1	1.1	1, 2, 3	
		Approved for development	1	1.2	1, 2, 3	
		Justified for development	1	1.3	1, 2, 3	
	Potentially commercial projects	Development pending	2	2.1	1, 2, 3	
		Development on hold	2	2.2	1, 2, 3	
	Non-commercial projects	Development unclarified	3.2	2.2	1, 2, 3	
		Development not viable	3.3	2.3	1, 2, 3	
	Additional quantities in place		3.3	4	1, 2, 3	
	Potential deposit	Exploration projects	[No sub-classes defined]	3.2	3	4
		Additional quantities in place		3.3	4	4

extraction'. Such quantities may not be disclosed in a public report but for other purposes would generally be classified in UNFC-2009.

The CRIRSCO template is independent from UNFC-2009 and may be mandatory for reporting purposes in some jurisdictions or in particular circumstances. The UNFC-2009 specifications document has no bearing on such mandatory reporting requirements or on the independent application of these codes and standards. It is stated that it is a voluntary system and does not impose rules regarding the categories of resources that should be disclosed; such disclosure of resource quantities is entirely at the discretion of the reporter.

### 7.2.6 United States of America

In 1988, the SME set up a working party to develop guidelines for the public reporting of exploration results, mineral resources, and mineral reserves. These guidelines were published in 1992 and updated in 2014.

The definitions given are in broad agreement with the JORC Code; however, there are differences between the content of the SME (2014) guide and the requirements of the US Securities and Exchange Commission, which regulates the reporting of exploration results, mineral resources, and mineral reserves by organisations, individuals, or companies. Dialogue between the two organisations continues. It is important to note that, prior to the international rationalisation of the reporting of mineral resources and reserves, a coal resource classification system was published by the United States Geological Survey (USGS) (Wood et al. 1983). The system determines how coal is classified into resource/reserve base/reserve categories on the basis of the geological certainty of the existence of those categories and on the economic feasibility of their recovery. Categories are also provided that take into account legal, environmental, and technological constraints. This system can be seen to be the basis on which the later

international codes were developed and as such is outlined here.

Geological certainty is related to the distance from points where coal is measured or sampled, thickness of coal and overburden, knowledge of rank, quality, depositional history, areal extent, correlations of coal seams and associated strata, and structural history.

The economic feasibility of coal recovery is affected not only by geological factors but also by economic variables, such as the price of coal against mining costs, coal preparation costs, transport costs and taxes, environmental constraints, and changes in the demand for coal.

The term resource is defined as naturally occurring deposits of coal in the Earth's crust in such forms and amounts that economic extraction is currently or potentially feasible.

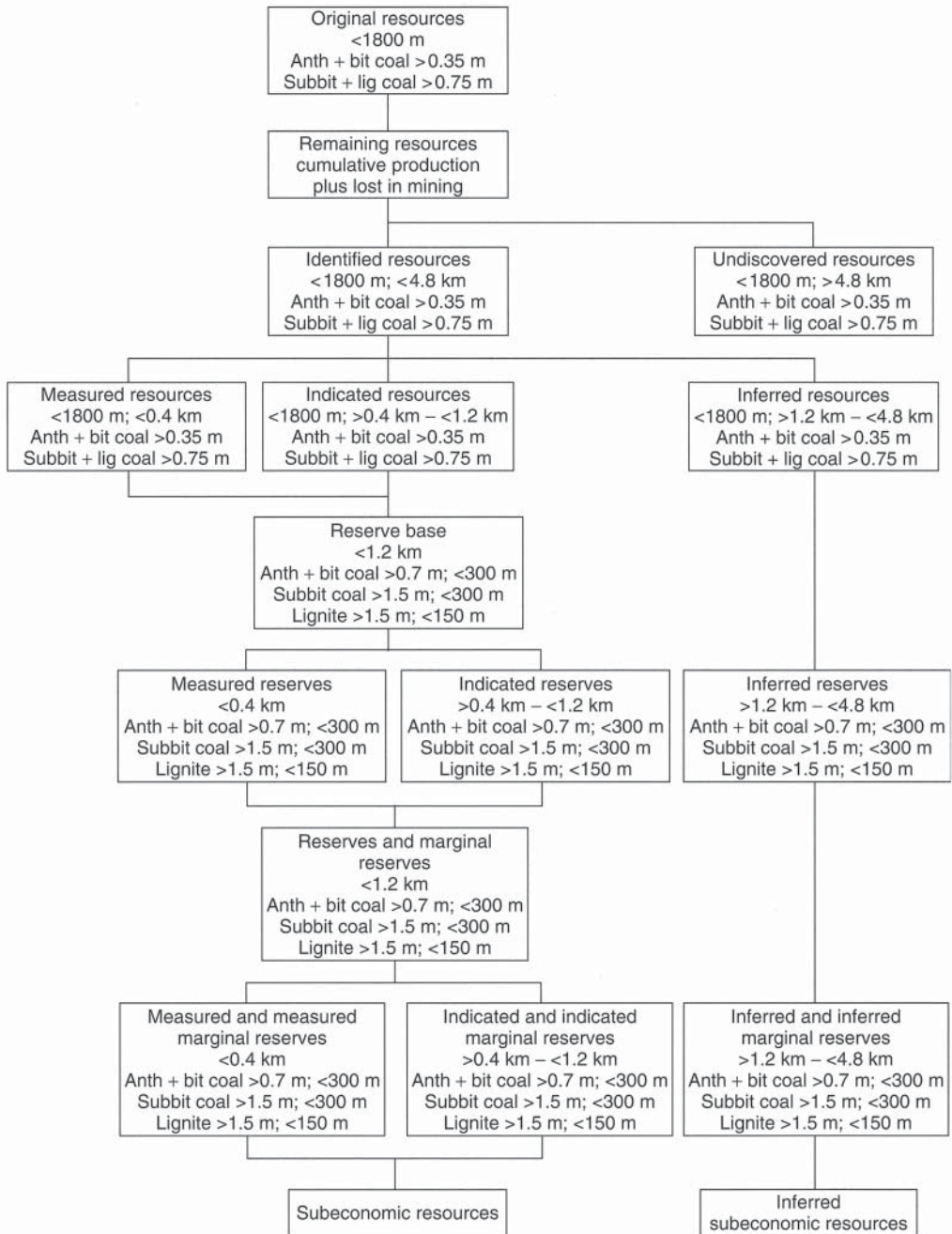
The hierarchy of coal resources/reserves categories outlined by the USGS is given in Figure 7.3, and the application of the reliability categories based on distance from points of measurement, i.e. coal outcrops and boreholes, by the USGS is demonstrated in Figure 7.4.

*Original resources* are the amount of coal in place before production; the total of original resources is the sum of the identified and undiscovered resources plus the coal produced and coal lost in mining.

*Remaining resources* include all coal after coal produced and coal lost in mining is deducted.

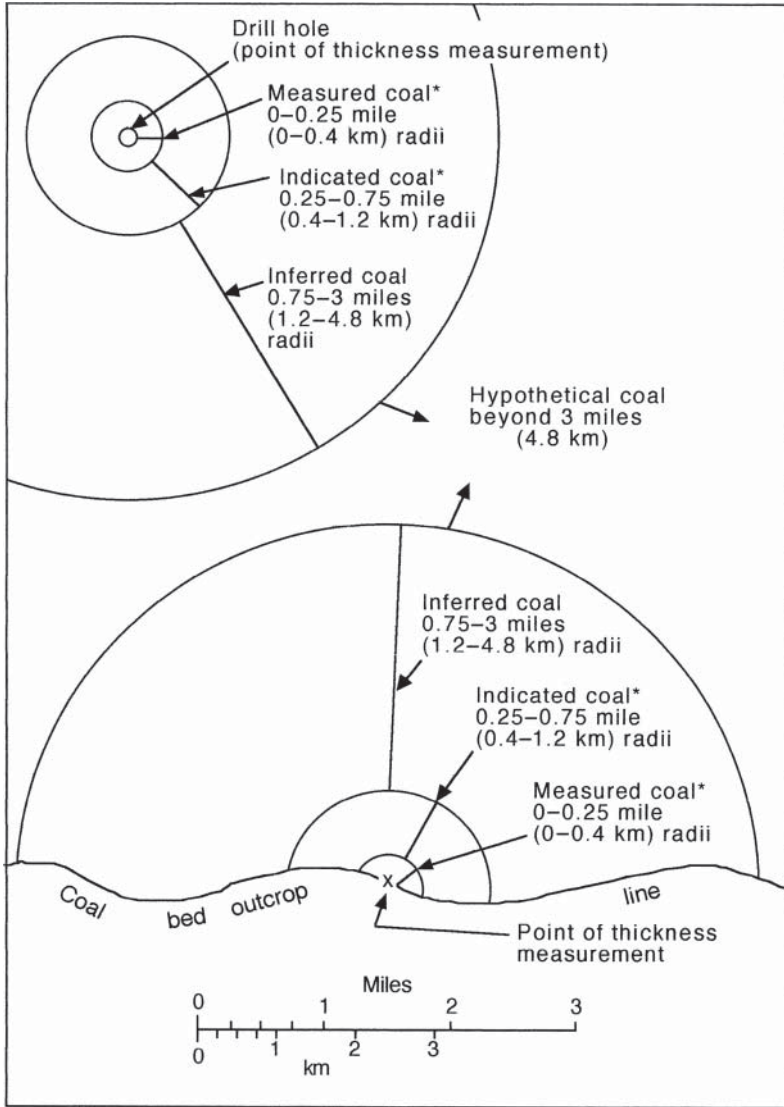
*Identified resources* are those resources whose locations, rank, quality, and quantity are known or estimated from specific geological evidence. The levels of control or reliability can be subdivided into inferred, indicated, and measured resources.

These subdivisions are determined by projecting the thickness of coal, rank, and quality data from points of measurement and sampling on the basis of geological knowledge.



**Figure 7.3** Criteria for distinguishing coal resource categories, adapted from USGS hierarchy of coal resources (Wood et al. 1983). Anth, anthracite; bit, bituminous; subbit, subbituminous; lig, lignite.





**Figure 7.4** Diagram showing reliability categories based solely on distance from points of measurement. *Source:* From Wood et al. (1983). \* Measured and indicated coal can be summed to demonstrated coal.

*Inferred resources* are assigned to individual points of measurement that are bounded by measured and indicated coal for 1.2 km, succeeded by 4.8 km of inferred coal. Inferred resources include anthracite and bituminous coal 0.35 m or more in thickness and subbituminous coal and lignite 0.75 m or more in thickness to depths of not more than 1800 m. Coal resources outside these limits are deemed hypothetical in nature.

*Indicated resources* are assigned to individual points of measurement bounded by

measured coal for 0.4 km succeeded by 0.8 km of indicated coal. Indicated resources have the same thickness and depth limits as inferred resources.

*Measured resources* are determined by the projection of the thickness of coal, rank, and quality data for a radius of 0.4 km from a point of measurement. Measured resources also have the same thickness and depth limits as indicated and inferred resources.

The *reserve base* is identified coal defined only by physical and chemical criteria as

determined by the geologist. The concept of the reserve base is to define a quantity of in-place or in-situ coal, any part of which is, or may become, economic. This will depend upon the method of mining and the economic assumptions that will be used.

The reserve base includes coal categories based on the same distance parameters given for coal resources but further defining the coal thickness and depth criteria, i.e. anthracite and bituminous coal to be 0.7 m or more, subbituminous coal to be 1.5 m or more, all to occur at depths not more than 300 m, lignite to be 1.5 m or more at depths not greater than 150 m.

*Inferred reserves* include all coal conforming to the thickness and depth limits defined in the reserve base and bounded by the same distance limits as given for inferred resources.

*Indicated reserves* include all coal conforming to the thickness and depth limits defined in the reserve base and bounded by the same distance limits as given for indicated resources.

*Measured reserves* include all coal conforming to the thickness and depth limits defined in the reserve base and bounded by the same distance limits as given for measured resources.

*Marginal reserves* are those reserves that border on being economic, i.e. they have potential if there is a favourable change in circumstances, mining restrictions are lifted, quality requirements are changed, lease areas become available, or there is a newly created demand for the type of coal held in this reserve category.

*Subeconomic resources* are those in which the coal has been lost in mining, is too deeply buried, or seam thickness becomes too thin, and/or the coal quality deteriorates to unacceptable limits.

### 7.2.7 Russian Federation

In 2011, the Russian Code for the Public Reporting of Exploration Results, Mineral

Resources, and Mineral Reserves (the NAEN Code) was prepared by the National Association for Subsoil Examination, members of which represent leading mining companies, industry research centres, and regional centres for the subsoil survey of Russia, plus the Society of Russian Experts on Subsoil Use, a member of the European Federation of Geologists (EFG), together with CRIRSCO and PERC representatives (McCombe 2011). The NAEN Code has been developed in accordance with general criteria adopted by the world's mining community, taking into account the Russian State System of Subsoil (Subsurface) Use management, classification, and accounting of solid minerals. The NAEN Code is based on the CRIRSCO template and the Guidelines on Alignment of Russian Minerals Reporting Standards and the CRIRSCO template agreed by the Russian Federal Government Agency State Commission on Mineral Reserves and CRIRSCO in 2010 (Dixon 2010). In 2014, CRIRSCO summarised the integration of Russia into the international reporting system (Malukhin 2014).

The traditional Russian system for reporting estimates of coal resources and reserves had a different objective from the CRIRSCO template. The Russian system comprised detailed documentation presented in a set format according to Russian Federation law, the primary purpose of which is the estimation and recording of the country's coal assets. This contrasted with the CRIRSCO International Reporting Template, which provides a standard terminology for use in assessing assets of projects and mining companies for disclosure to stock markets and financial institutions. Although the NAEN Code is in operation, many existing coal companies still have their coal resources and reserves calculated in the traditional format. The Russian classification system and its relationship to the various levels of exploration and investigation is shown in Table 7.2 (Dixon 2010). According to the level of geological knowledge, the Russian system identifies four levels of resource. These are,

**Table 7.2** Russian resource/reserves classification (Dixon 2010).

	Exploration stages	Final documentation of stage completion	Resource/reserves, categories:		Prognostic resources, categories:		
			A, B, C <sub>1</sub>	C <sub>2</sub>	P <sub>1</sub> ,	P <sub>2</sub>	P <sub>3</sub>
↑ Increasing level of technical and economic knowledge	Detailed exploration work	TEO of 'permanent conditions', Russian reserves estimation report	'Fully explored' deposit		Not considered at this stage		
			1. Balance (economic) 2. Off-balance (potentially economic)	1. Balance (economic) 2. Off-balance (potentially economic)			
	Deposit delineation (estimation) work	TEO 'provisional conditions', Russian reserves estimation report	'Estimated' deposit		Not considered at this stage		
			1. Balance (economic) 2. Off-balance (potentially economic)	1. Balance (economic) 2. Off-balance (potentially economic)			
	Prospecting work	Preliminary estimation (TES (scoping study) on geologically justified concepts on the sizes and characteristics of known mineralisation)	Not considered at this stage	Not considered at this stage	Mineral occurrence	Not considered at this stage	
Regional geological studies	Geological report on exploration results	Not considered at this stage	Not considered at this stage	Not considered at this stage	Ore clusters, mineralisation fields, etc.		
← Increasing level of geological knowledge and confidence							

in order of decreasing geological knowledge, A, B, C<sub>1</sub>, and C<sub>2</sub>; these correspond to coal resource categories in the CRIRSCO International Reporting Template. The CRIRSCO measured and indicated resources are equivalent in terms of definition to Russian resource categories A, B, C<sub>1</sub>, and C<sub>2</sub> based only on the level of geological knowledge. The Russian system also has additional categories of ‘prognostic resources’ designated P<sub>1</sub>, P<sub>2</sub>, and P<sub>3</sub>. These include resources less well known than C<sub>2</sub> resources in descending order of knowledge and can represent those resources at a deeper level or outside the boundaries of the deposit under examination. The Russian classification also groups coal resources according to their geological complexity from first (simplest) to fourth (extremely complex). Category A resources are only in the 1st level of geological complexity, category B resources are in first and second levels of geological complexity, category C<sub>1</sub> can include first, second, and third levels of geological complexity, and where areas have been studied in detail can include the fourth level of geological complexity. C<sub>2</sub> category includes all four levels of geological complexity.

Two levels of modifying factors are also determined; these represent a lower level of detail required for a technical–economic justification (TEO) of provisional conditions (i.e. a pre-feasibility study) and a higher level of detail required for a TEO of permanent conditions (i.e. a full feasibility study). The CRIRSCO and NAEN Code documents equate to an estimated coal deposit and a fully explored coal deposit. In addition, resources are classified into two categories according to their economic significance. These are ‘balanced’ resources, which are economically exploitable, and ‘off balanced’ resources, which are only potentially economic. Balanced resources represent tonnage and qualities before any dilution or mining losses are applied. Once such losses are applied, these are then referred to as exploitation reserves (or industrial reserves); these correspond to the probable and proved coal reserves categories in the CRIRSCO International Reporting Template.

Figure 7.5 summarises the relationship between the Russian classification and the CRIRSCO International Reporting Template (McCombe 2011).

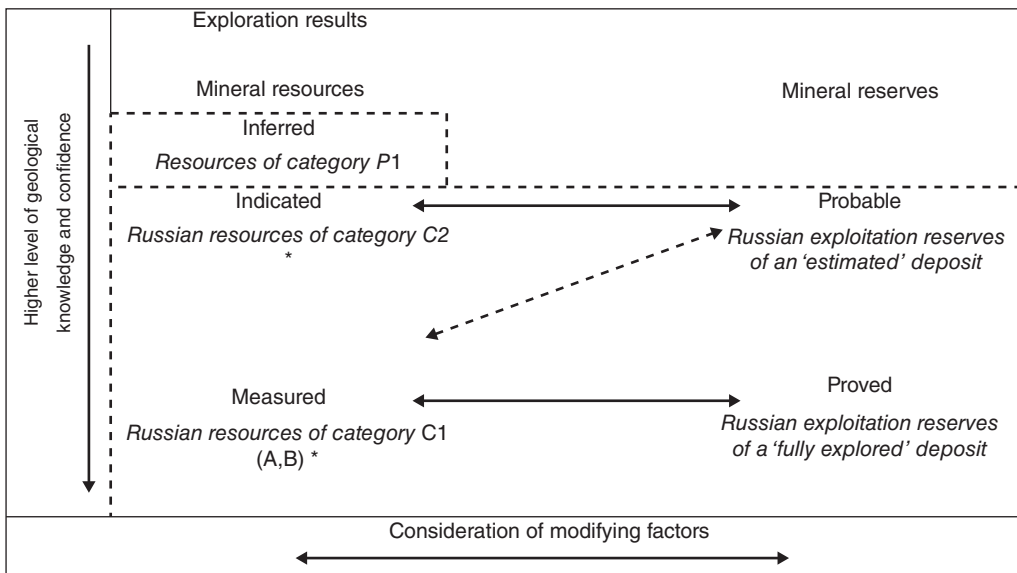


Figure 7.5 Comparison of Russian and CRIRSCO classifications (Dixon 2010).

The conversion of coal resources/reserves estimated according to the Russian system as equivalents in the CRIRSCO International Reporting Template, e.g. the JORC Code, again requires the signature of a competent person.

### 7.2.8 People's Republic of China

The traditional Chinese resource classification was developed from the former Soviet system and is now phased out. This classification used categories A to F based on decreasing levels of geological confidence. This was replaced with the 1999 Chinese Mineral Resource/Reserve Classification, which was based in part on UNFC 1999, which included the incorporation of CRIRSCO definitions (in 1998). This classification was reviewed in 2007–2008, as the then current Chinese Classification System was not regarded as particularly useful for reporting in a market economy. It was not easily comparable with the CRIRSCO group of codes, such as the JORC Code (Stoker 2009). The updated 2004 Chinese Mineral Resource/Reserve Classification classified resources/reserves on the basis of geological knowledge and interpreted continuity (the UNFC 'G' category), and on the basis of project economics and feasibility study status (UNFC 'F' and 'E' categories). The system contained 16 categories that are referenced by a three-digit number in the same EFG notation as outlined in the UNFC classification (see Section 7.2.5). Table 7.3 (Ilieva 2016) illustrates this classification and includes the term 'basic reserve' (suffix 'b'), which is the total quantity of in-situ reserve that forms the basis for the recoverable reserve. The EFG system had, in addition, the letter 'M' put after the E category for marginally economic reserves, and the letter 'S' after the E category for sub-marginally economic reserves. Historically, there had also been a category of 'undiscovered resources' that has no equivalent in the CRIRSCO codes. In 2015, the Mineral Resources and Reserves Evaluation Centre of the Ministry of Lands and Resources produced an update of the above classification. The essential difference being

that, although still based on the UNFC-1997 with elements of USGS Circular 891, the concept of project status had been approved. Table 7.4 shows the updated format (Li 2015) relating to project status, which brings the Chinese classification closer to the CRIRSCO codes. In 2017, the UNFC changed its name to the United Nations Framework Classification for Resources, and in 2018 it produced a bridging document between the National Standard of the People's Republic of China (PRC) 'The Classification of Resources/Reserves of Solid Fuels and Mineral Commodities' (GB/T 17766-1999) and UNFC-2009, the mapping of which is shown in Table 7.5.

The requirement of a competent person to sign off on resource/reserve reporting is not applicable in the PRC. However, where foreign investors are involved, a competent person must provide a comparison between resource estimates that are non-compliant with CRIRSCO codes and CRIRSCO codes such as the JORC Code. The PRC is not a member of CRIRSCO, but its adoption of the UNFC-2009 guidelines, together with updates, does begin to equate with the CRIRSCO template.

### 7.2.9 India

The Committee on Assessment of Resources under Coal Council of India drew up the Indian Standard Procedure (ISP) for coal reserve estimation in 1957. The Geological Survey of India locates coal-bearing areas and has assessed and classified coal resources into categories based on distances between points of observation – inferred (1–2 km apart), indicated (200 m<sup>-1</sup> km apart), and proven (<200 m apart) – and represents volumes in the ground without any reference to coal quality, mineability, or economics. The Central Mine Planning and Design Institute then converts such 'indicated' reserves into the 'proven' category through detailed exploration. It has included coal resources that are unavailable for exploitation or areas sterilised by fire, and so on.



**Table 7.3** Comparisons between CRIRSCO and other resource and reserve classifications.

CRIRSCO (JORC/CIM/ PERC/NAEN)	Resource		Measured	Indicated			Inferred		Exploration potential or exploration target
	Reserve		Proven	Probable					
Russian state classification	Reserve		A, B, and C <sub>1</sub>	B, C <sub>1</sub> , and C <sub>2</sub>			C <sub>2</sub> and P <sub>1</sub>		P <sub>2</sub> and P <sub>3</sub>
UNFC-2009 (Chinese and Indian classification)	‘E’	Economic evaluation (100)	Designed mining with loss	Recoverable reserve (111)	Probable recoverable reserve (121)		Probable recoverable reserve (122)		
			Designed mining without loss						
		Marginal economic (2M00)	Basic reserve (M1)	Basic reserve (2M21)		Basic reserve (122b)			
		Sub-economic (2500)	Resource (2S11)	Resource (2S11)		Resource (2S22)			
		Intrinsically economic (300)			Resource (331)		Resource (332)	Resource (333)	Resource (334)
	‘F’	Feasibility evaluation	Feasibility (101)	Pre-feasibility (020)	Scoping (030)	Pre-feasibility (020)	Scoping (030)	Scoping (030)	Scoping (030)
‘G’	Geological evaluation	Measured (001)			Indicated (002)		Inferred (003)	Predicted (004)	

Source: From Ilieva (2016).

**Table 7.4** Update of Chinese coal classification codes (Li 2015).

Classification codes						
Project status	Economic status		Geological confidence level			
			Measured	Indicated	Inferred	Reconnaissance
Commercial projects	Economic	Extractable reserves	Feasibility study			
			(111)	(122)		
		Reserves base	Feasibility study		Parts of (333)	
(111B)	(122B)					
Potentially commercial projects	Intrinsic economic		Geological study			
			(331)	(332)	(333)	(334)?

**Table 7.5** Mapping of GB/T 17766-1999 to UNFC-2009 classes and categories (UNECE 2018).

CCMR-1999 class		CCMR-1999 categories				UNFC-2009 class	UNFC-2009 'minimum' categories		
Economic	Reserves	(111)				Commercial projects	E <sub>1</sub>	F <sub>1</sub>	G <sub>1</sub> ,G <sub>2</sub>
		(121)	(122)						
Marginal economic	Basic reserves	(111b)				Not defined in UNFC-2009			
		(121b)	(122b)			Potentially commercial projects	E <sub>2</sub>	F <sub>2</sub>	G <sub>1</sub> ,G <sub>2</sub> ,G <sub>3</sub>
		(2M11)							
		(2M21)	(2M22)						
Sub-marginal	Mineral resources	(2S11)							
		(2S21)	(2S22)						
Intrinsic economic		(331)	(332)	(333)					
Economic -interest undefined	Undiscovered resources				(334)?	Exploration projects	E <sub>3</sub>	F <sub>3</sub>	G <sub>4</sub>

In 2001, the government of India decided to adopt the UNFC classification for its mineral resources and reserves, and Coal India Ltd has since codified its resources/reserves as per UNFC-2004. The ISP resource categories and their equivalents in UNFC are shown in Table 7.3. The G, F, and E categories are the same as shown in Section 7.2.5 and the numerical codes as in Figure 7.2.

Chand and Sarkar (2006) stated that India is to continue to review and follow the UNFC classification format so as to include economically mineable and technically feasible parts of a measured or indicated coal resource. This was reinforced in 2011 by The

Energy Research Institute (TERI), which stated that the total coal inventory is not so important, what is important is how technically feasible is it to mine the coal. TERI suggested that India should adopt the UNFC-2009 Code for resource/reserve classification (Kulkarni 2011) that would give a more realistic assessment of India's coal resources/reserves related to economic and technical viability.

### 7.2.10 Other Countries

Other countries have adopted the CRIRSCO template, with Brazil, Chile, Kazakhstan, and Mongolia listed as CRIRSCO members.

Other coal-producing countries are adopting the CRIRSCO template or, as in the case of Eastern European countries, which have previously used the Russian classification, are now reviewing their classifications, and some are recommending the adoption of the UNFC-2009 Code, e.g. Serbia (Ilic et al. 2009).

## 7.3 Reporting of Resources/Reserves

The reporting of resources and reserves as part of a public document can be at three levels. The following definitions are from the CRIRSCO standard definitions (2012).

- (i) A *scoping study* is the study of the potential viability of coal resources that includes assessments of realistically assumed modifying factors together with any other operational factors that are necessary to demonstrate at the time of reporting that progress to a pre-feasibility study can be reasonably justified.
- (ii) A *pre-feasibility study* is a comprehensive study of a range of options for the technical and economic viability of a coal project that has advanced to where a preferred mining method, whether underground or open pit, is established. It includes a financial analysis based on reasonable assumptions on the modifying factors and any other relevant information sufficient for a competent person to determine if all or part of the coal resource may be converted to coal reserve at the time of reporting. A pre-feasibility study is at a lower confidence level than a feasibility study.
- (iii) A *feasibility study* is a comprehensive technical and economic study of the selected development option for a coal project that includes detailed assessments of applicable modifying factors together with any other operational factors and detailed financial analysis that are necessary to

demonstrate at the time of reporting that extraction is reasonably justified, i.e. economically mineable. The results of the study may reasonably serve as the basis for a final decision by a proponent or financial institution to proceed with, or finance, the development of the project. The confidence level of the study will be higher than that of a pre-feasibility study.

All factors used to limit resources and reserves that are necessary to verify the calculations must be stated explicitly. These will include the points of measurement, e.g. open-hole, cored boreholes, and outcrops. The relative density value that is selected for the calculation of the coal tonnage should be stated, together with the reasons for its selection.

### 7.3.1 Coal Resources and Reserves

To report resources and reserves, the required data will be based on the following:

- 1) Determination of legal tenure to any coal lease area.
- 2) Details on each coal seam within the lease area.
- 3) On a depth basis, in regular depth increments if sufficient information is available.
- 4) On a seam thickness basis, the minimum coal thickness used and the maximum thickness of included non-coal bands should also be stated. Normally, where a seam contains a non-coal band thicker than 0.25 m the two coal splits can be regarded as separate seams, and tonnages should be reported for each. The limits for non-coal bands in brown coal sequences may be greater, e.g. 1.0 m.
- 5) On a quality basis, maximum raw coal ash should be stated; and for marketable reserves, only that coal that can be used or beneficiated at an acceptable yield (which should be stated) should be included in the estimate. Other raw coal parameters, particularly those that affect utilisation, should be given, e.g. total sulfur and calorific value.

Subdivisions of the resources may be made for areas of oxidised coal and heat-affected coal.

A summary of all the factors relevant in determining the categories of coal resources/reserves assessment are shown in Figure 7.6. In the diagram, coal product value and yield are plotted against ease of mining, coal recovery, and mining costs, and against geological certainty.

### 7.3.2 Coal Resources and Reserves Maps

Any report of resources/reserves should be accompanied by maps and plans at appropriate scales showing all the relevant data. Such maps and plans should show those areas assigned to each category of resources/reserves, seam

depth contours, seam isopachs, quality contours for each seam, and all areas not to be mined.

The geological information required for resource/reserve assessments is based on the points of measurement. Both for quantity and quality assessments it is important that the recorded information at the points of measurement is correctly compiled and is therefore reliable. If this is not the case, then making resource/reserve assessments is a useless exercise. If any point of measurement has a doubt against its reliability, this should be taken into account when the assessment is made.

Extrapolation from points of measurement up to the distance limits imposed by the resource/reserve category is based on the judgement and knowledge of the local geology. Geological hazards that need to be taken into

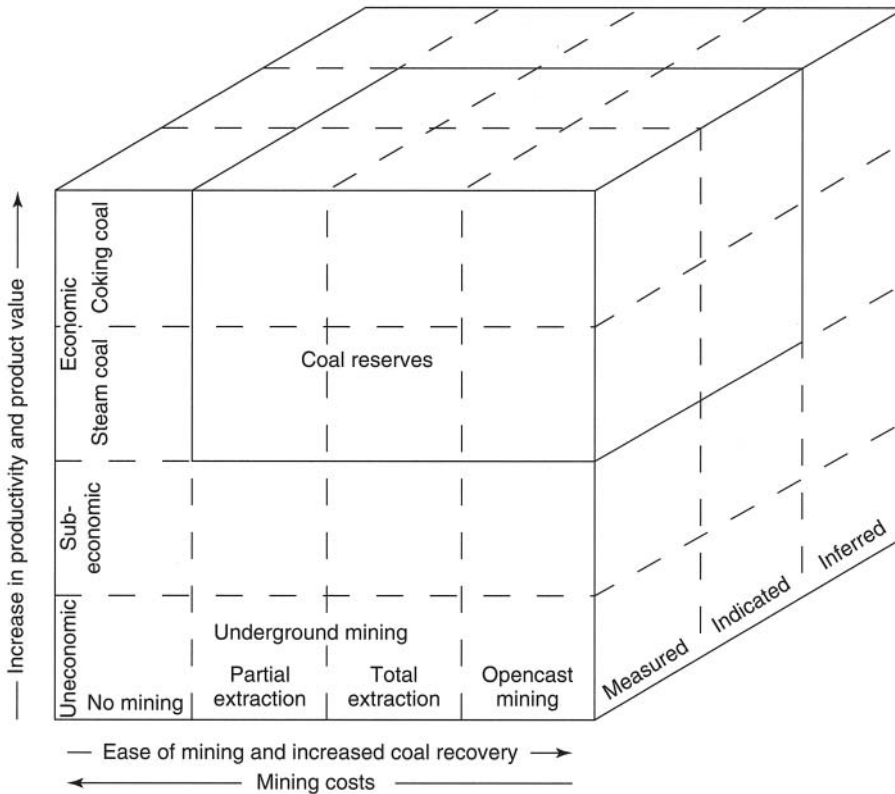


Figure 7.6 Variable factors in coal reserves assessment. Source: Adapted from Ward (1984).

account are faulting, seam thickness variations (particularly rapid thinning and splitting of seams), washouts, sharp changes in dip, and the presence of igneous intrusions.

The distances between points of measurement used for the different resource/reserve categories is theoretically the same for underground and opencast coal. If the geology is similar between two points the same distance apart at depth or near the surface, then the confidence level must be similar. However, shallow drilling is relatively cheaper, and this allows for more holes to be drilled, and therefore the confidence level will be greater. Also, in coal occurring near the surface, there are the additional problems of oxidation and, in some cases, zones of burning that have to be delineated.

Deep boreholes are costly, so that at the exploration stage a lower level of confidence may be achieved; however, geophysical surveys are used to supplement the drilling to ascertain any major structural disturbances and changes in thicknesses in the coal-bearing sequence.

### 7.3.3 Calculation of Coal Resources

Modern coal projects use computer programs to calculate coal resources; however, it is essential that the coal geologist understands the methodology of calculating coal resources. He or she may need to do a rapid assessment of a sample block of resources to verify the resource tonnage given by a report as part of a due diligence on the project. Therefore, it is important for the coal geologist to be able to resource calculations by the traditional methods as well as using computer programs.

#### 7.3.3.1 In-Situ Tonnage Calculations

The basic formula to calculate coal resources is

$$\begin{aligned} &\text{Coal thickness (m)} \times \text{Area (m}^2) \\ &\quad \times \text{Density (t m}^{-3}) = \text{Total tonnes} \\ &\text{(for each defined resource/reserve block)} \end{aligned}$$

Coal thickness is determined at each point of measurement.



**Figure 7.7** Digital planimeter being used to calculate coal reserve areas within a working mine. Source: Photograph by M.C. Coultas.

The area of each resource/reserve block is measured on the map or plan as follows:

- (i) In the traditional way, by using a planimeter. The boundaries of each block area are traced and the area is calculated automatically and given as a reading, Figure 7.7 shows such a planimeter being used in this fashion.
- (ii) The coordinates of each block area are entered into a computer program specified for the purpose and an areal calculation obtained.

The relative density is normally taken from a total seam section, i.e. that section of the seam to be mined, not a density of the cleanest portion of the seam. In opencast mines this will usually be a whole seam section, whereas underground this is not always so as coal quality constraints or mining difficulties may mean mining only part of the seam.

If no density determination is available, an estimated density (or relative density) can be adopted dependent upon the known average ash content of the seam; this would be in the order of  $1.3\text{--}1.4\text{ t m}^{-3}$  in coals with reasonably low ash contents.

This means that an area of  $1\text{ km}^2$  ( $1 \times 10^6\text{ m}^2$ ) underlain by a coal with a thickness of  $1.0\text{ m}$  and with a density of  $1.4\text{ t m}^{-3}$  will contain

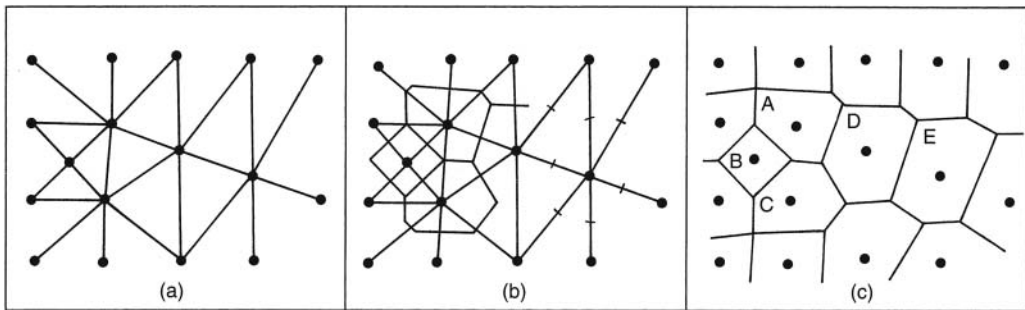


1.4 × 10<sup>6</sup> t of coal. Resource tonnages are usually quoted in millions of tonnes and are usually rounded off to the nearest hundred thousand, ten thousand, or one thousand tonnes, depending on the degree of accuracy required.

A common method of calculation of coal resources has been the polygon or area of influence technique. This method assigns an area to a point of measurement that is a function of the distance to the immediate neighbouring points of measurement. A polygon is formed from joining the midpoints between the point

of measurement and those surrounding it, resulting in the original point of measurement forming the centre of the polygon. Figure 7.8 shows the construction of polygons for a series of points of measurement together with the calculation of reserve tonnages for selected polygons for a hypothetical example.

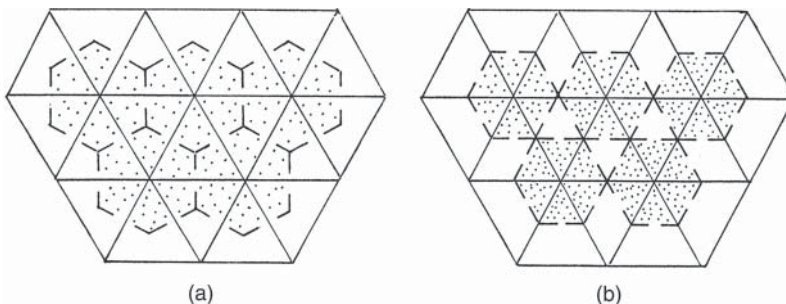
In constructing polygons of influence for the estimation of coal resources, mistakes can occur; for example, in the construction of blocks of influence for a triangular grid, Figure 7.9a shows the correct selection of areas of interest, whereas Figure 7.9b illustrates



Calculation of reserves for polygons A – E:

	Area (m <sup>2</sup> )	Coal thickness (m)	Relative density	In situ tonnage
A	211 500	2.20	1.4	651 400
B	108 000	2.16	1.4	326 600
C	225 000	1.86	1.4	585 900
D	289 800	2.40	1.4	973 700
E	351 900	2.46	1.4	1 211 900

**Figure 7.8** Polygon method for calculation of in-situ coal resources/reserves. (a) Link all points of measurement. (b) At the midpoint between points of measurement, draw lines at right angles and join to form polygonal areas around each point of measurement. (c) Complete all polygons, e.g. polygons A–E can now be measured to calculate reserves using the central point of measurement as control for each polygon. Total in-situ tonnage is 3 749 500 t (3.75 Mt).



**Figure 7.9** Construction of polygons (blocks) of influence for an equiangular triangular grid: (a) correct construction of elementary blocks; (b) incorrect construction of elementary blocks. Source: From Tasker (1985).

how such areas have been selected incorrectly (Tasker 1985).

The weaknesses of using the polygon method are that if the drill holes in a deposit are widely and irregularly spaced, it is possible that some points of measurement will give undue emphasis in the calculation of resources; also, widely spaced drill holes leave uncertainty as to the continuity of coal seams. Polygons based on widely spaced data points give no indication of the accuracy of the results; it is possible that the actual and calculated resources may differ by significant amounts.

A practical development of the polygon method is to outline resource zones containing several points of measurement, calculate the area of the zone, and use an average or weighted mean seam thickness for each zone.

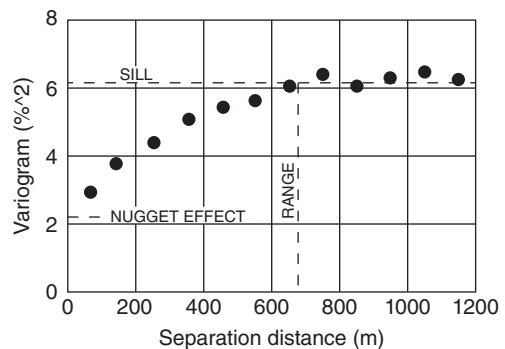
Many areas are insufficiently well delineated, either by outcrops or drill holes, so that some boundary of arbitrary width needs to be placed around the drilled portion of the deposit. If the deposit is large, a subjective selection of the boundary zone width is satisfactory, because a moderate change in width will not substantially alter the total area of the deposit. However, if the deposit is small, a significant proportion of the resources may lie within the boundary zone. The calculated resources, therefore, will be unreliable because of their dependence on the subjective selection of the boundary zone.

In an area that has been drilled extensively, detailed contouring of the coal seams can enable resource estimates to be made as a further definition of the estimated resource figure; this method is also useful to highlight areas of geological hazard, such as washouts or faulting.

### 7.3.3.2 Geostatistics and Computer Modelling

The use of geostatistics in coal resources estimation considers the estimation of coal and overburden volumes and coal quality parameters. The most common geostatistical

estimation procedure is kriging. The confidence limits based on these estimations will determine where to add additional points of measurement, i.e. boreholes, plus spacing of such boreholes to allow classification of resources into inferred, indicated, or measured (Larkin 2009). Kriging uses information on spacing of points of measurement referred to as spatial variation and is able to sort out geological continuity and redundancy together with providing a measure of confidence in the estimate. Geostatisticians use a graphical summary called a variogram to analyse and understand spatial variation. Figure 7.10 shows an example of a variogram calculated for percentage ash data from a Powder River coal deposit (Srivastava 2013). The variogram plots variation (i.e. how much data values differ from each other) as a function of distance. Data samples that are close together will have data values that are more similar to each other than data values that are further apart. The variogram is a plot of the average squared difference between the pairs of data values. In practice, the minimum required to calculate a variogram requires at least a dozen data points, which when paired with each other create several hundred pairs with different spacings. In Figure 7.10, the sill is usually close to the overall variance of the data values; in Figure 7.10 the ash data have a variance of 6.1. The range is the distance at which the variogram reaches its



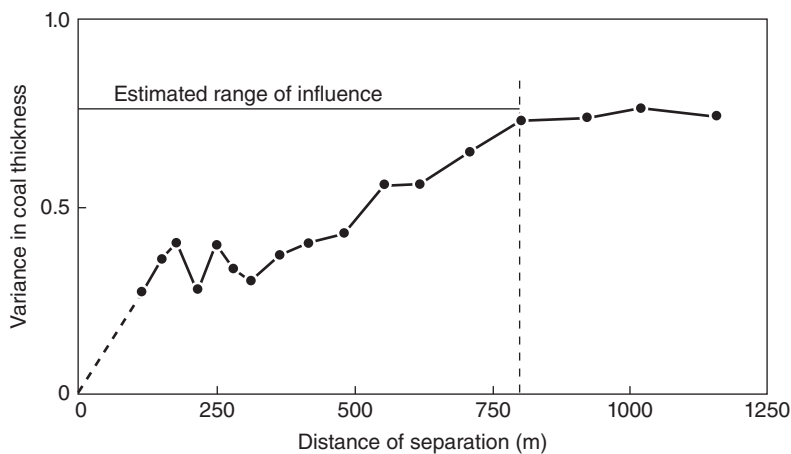
**Figure 7.10** Variogram of ash content over distance. *Source:* From Srivastava (2013).

plateau and corresponds closely to our knowledge of the range of geological correlation. The nugget effect is the apparent y-intercept in the plot of the variogram. The nugget effect quantifies the variability observed at extremely short distances. This can be influenced by in-situ variability and operator error in data measurement, age and quality of data, and a combination of old and new data. When kriging variance is high due to lack of data and/or where the variogram model describes a lot of variability at the scale of distance to the nearest data points, then the correlation coefficient is low and there is little confidence in the estimate. Where the kriging variance is low using well-correlated data points, the correlation coefficient is high and there is more confidence in the estimate (Srivastava 2013).

Kriging can be used to improve the reliability of the boundary width for a deposit in estimating the average range of influence of the points of measurement by means of a variogram. The variance in thickness is calculated for pairs of points of measurement, where the pairs have a common distance of separation (as on a grid pattern). In general, as the distance of separation increases so does the variance, until a maximum variance is reached, as shown in Figure 7.11 (Ventner 1976). The distance corresponding to this maximum may be taken as

a measure of the range of influence for points of measurement within the deposit. Thus, the areal extent of a deposit may be defined by assuming the deposit to extend (geological continuity permitting) a distance equal to the range of influence in all directions beyond the points of measurement. The boundary zone width is therefore dependent on the data available; if this is insufficient or the points of measurement are irregularly sited then this method is not applicable.

Conditional simulation is another method for building models that uses all of the available borehole data and fills in the gaps between the boreholes with a plausible prediction. Where the spatial variability of coal seam thickness and quality characteristics have a significant impact on how the coal is mined and processed, geostatistical simulation may have advantages over kriging methods. Pardo-Igúzquiza et al. (2013) considered that a more realistic assessment of risk is obtained when the uncertainty of variogram model parameters is taken into account. The difference between simulation and kriging is in the way they fill in the gaps in the data: kriging produces a model that may be smoother than reality, whereas simulation aims for predictions that mimic the true in-situ variability.

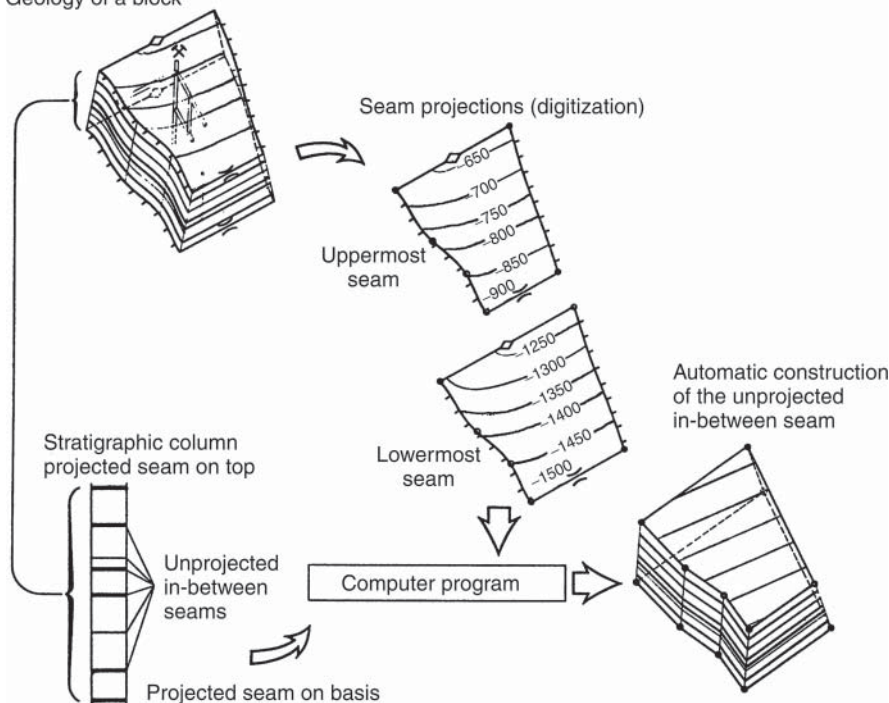


**Figure 7.11** Variogram of coal thickness. Source: From Ventner (1976).

Research by Atalay and Tercan (2017) showed that the kriging method is based on the weighted average of local data values that are not involved in the determination of the weights. They applied an estimation method based on a Gaussian copula that combines sample configuration and local data values (copulas are alternative ways of modelling the dependence between variables). This method used lower calorific values from coals in the Thrace region of north-west Turkey. These estimates, when compared with kriging estimates, show the Gaussian copula identifies local variability, with estimates being more accurate than by kriging. Application of such analysis may influence future mine planning. However, such a method is complex, and its use may not be justified when upgrading or adding resources to an asset base.

Computer programs allow the input of geological data into a geologically significant computer model of the coal deposit. In this way, coal deposits can be subdivided into major block units, each equating to the proposed working district of a coalmine. Within these units, smaller structurally delineated blocks provide the basic geological data that is transformed into computer data. Within each of these smaller units the coal seams provide the basic data for the whole assessment, i.e. depth, thickness, dip, and size of the area. For example, the minimum coal thickness to be evaluated could be taken as 0.30 m, and the minimum fault displacement to delimit a block could be 10.0 m. Such blocks consist of a series of vertically neighbouring coal seams, and the projected stratigraphic column represents the series of coal seams within the block (Figure 7.12; Juch et al. 1983).

Geology of a block



**Figure 7.12** Schematic diagram of the calculation of sectional areas using the block model for coal resource/reserve calculations. *Source:* From Juch et al. (1983).

From these data, the spatial position, form, and area of all intermediate coal seams are calculated using a mathematical model. The top and bottom of a block are digitised to include regularly distributed depth values. On all the intermediate planes, sectional areas are defined by closed polygons. Data input of these sectional areas is done by digitising seam projections at 1:10 000 scale. Straight lines joining corresponding points on the top and bottom planes define the lateral delineation of the block on all intermediate planes and also all in-between sectional areas, as shown in Figure 7.12.

In areas previously worked, the percentage of the worked areas is estimated for each sectional area and has to be subtracted in the process of calculation.

In similar fashion, other coal seam data can be built into the computer model, such as seam thickness and coal quality data.

Experience has shown that, for a reliable experimental variogram, it is necessary to have at least 100 data points per coal seam in a major block unit, and at least 25 data points per coal seam for the estimation of one sectional area. Such data points need to be less than 1.0 km apart to assure a level of reliability acceptable to implement the computed resource/reserve calculation.

It can be seen that computers now provide a means whereby statistical confidence criteria can be provided on a regular basis. The main limitation to these methods is the amount and quality of data required, and the reliability of geological and geophysical interpretation. Most methods require data points relatively close together and evenly spaced, i.e. spatial variation to be low; such conditions are only likely to occur in fairly well-defined coal deposits.

It should be noted that the use of geostatistics provides many benefits when calculating resource estimates, but each of these benefits comes at a cost. For a typical coal resource calculation of just tonnages of coal and cubic metres of overburden, the use of geostatistics

probably outweighs any benefits. The value of using geostatistics is where one is dealing with more complex variables in the data, such as coal quality variations. There may also be benefits such as better definition of selected future borehole spacing and determination of whether increased drilling will yield substantial additional information (Larkin 2009).

### 7.3.3.3 Opencast Coal Mining

**7.3.3.3.1 Stripping Ratio Calculation** In opencast mining operations, overburden to coal ratios are often quoted on a volumetric basis, i.e. bank (in-situ) cubic metres (bcm) of overburden per tonne of coal in situ. The calculation of the stripping ratio (SR) is

$$SR = \frac{\text{Overburden cubic metres (bcm)}}{\text{Coal cubic metres} \times \text{coal RD}}$$

For coal deposits where the relative density of the coal is essentially constant, SR is expressed simply as the ratio of the thickness of overburden to that of the total workable coal section; however, the basis of the ratio must always be stated clearly. The most realistic results are achieved when the overburden thickness is calculated from the difference between the topographic surface and the structure contours of a seam at the selected data points within the area of interest. Where numerous data points exist, the SRs are most conveniently calculated by computer and this data can later be plotted as SR contour plans.

In the UK, overburden and SRs are calculated somewhat differently, and can be considered as either

$$\begin{aligned} &\text{Overburden ratio (without batters)} \\ &= \text{In-situ vertical ratio} \end{aligned}$$

or

$$\begin{aligned} &\text{Actual SR (including batters)} \\ &= \text{Working ratio} \end{aligned}$$

with batters being the amount of overburden that must remain in the pit as part of the angled walls to ensure pit stability.

The overburden ratio is calculated as follows:

$$\begin{aligned} & \text{Overburden (in-situ vertical) ratio} \\ &= \frac{\text{In-situ overburden thickness} \\ & \quad (\text{including coal thickness}) - \text{Coal thickness}}{\text{In-situ coal thickness}} \end{aligned}$$

The overburden ratio can also be calculated on a volumetric basis:

$$\begin{aligned} & \text{Overburden (in-situ vertical) ratio} \\ &= \frac{\text{In-situ overburden volume} \\ & \quad (\text{including coal volume}) - \text{Coal volume}}{\text{In-situ coal volume}} \end{aligned}$$

For mining a multiple coal seam sequence, the interburden (i.e. the thickness of non-coal between coal seams) is treated as overburden for the coal seams below the highest coal seam. Figure 7.13 shows the effects of topography and geological structure on the overburden or SR. The SR selected for any coal deposit is based on the economics of removing and rehandling overburden, coal quality, and geotechnical limitations.

**7.3.3.3.2 Depth of Planned Opencast Mining** In the UK, a useful equation is used for the rapid estimation of the highwall depth:

$$\begin{aligned} \text{Highwall depth} &= 2[(\text{Overburden ratio} \\ & \quad \times \text{In-situ thickness}) + \text{In-situ thickness}] \\ & \quad - \text{Low-wall depth} \end{aligned}$$

And for estimating the average depth of the coal excavation area:

$$\begin{aligned} \text{Coal excavation area average depth} \\ &= \frac{\text{Highwall depth} + \text{Low-wall depth}}{2} \end{aligned}$$

### 7.3.3.4 Geological Losses

The calculation of recoverable or extractable reserves requires the identification of those geological constraints that are likely to inhibit mining. Such constraints include the identification and positioning of fault zones, changes in dip, washouts, seam splitting and thinning, losses in quality, and igneous intrusions. In

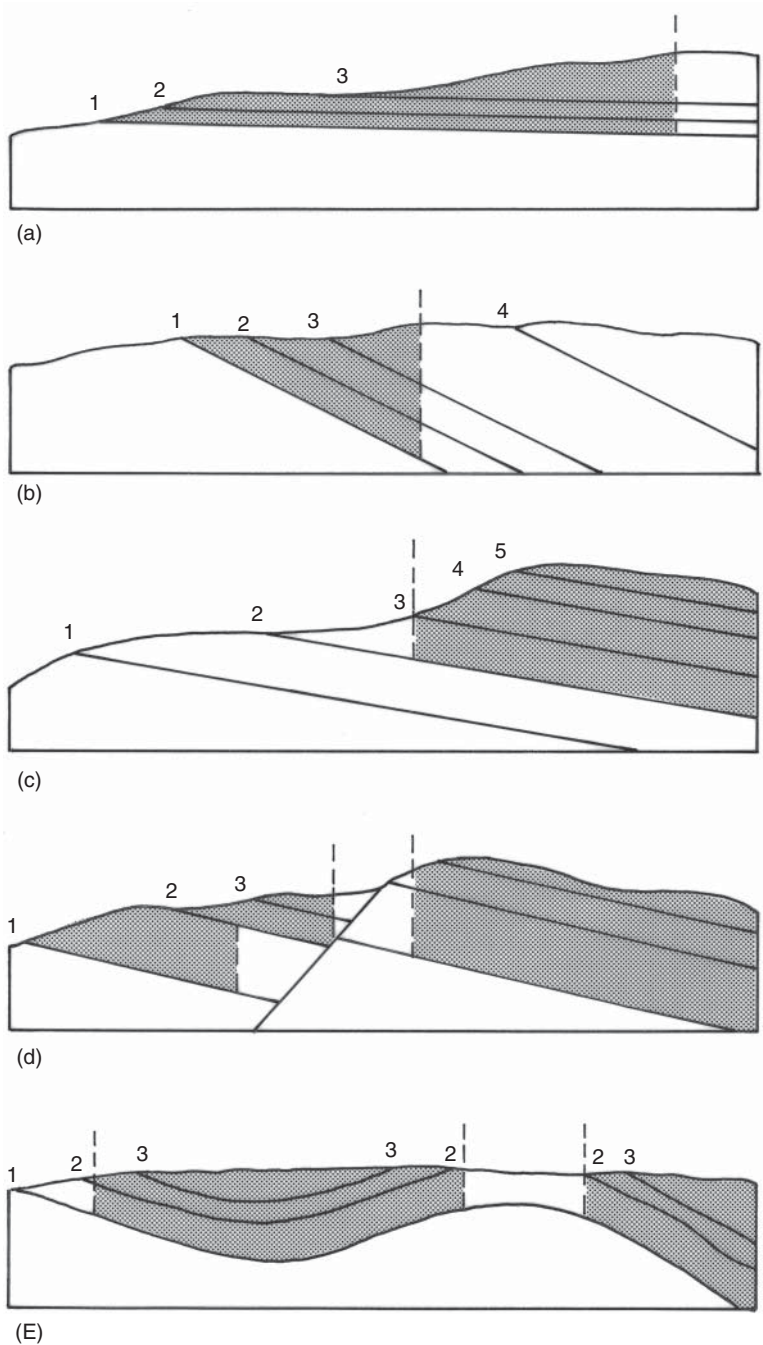
opencast workings, reserves will be affected by deleterious changes in the SR. All these factors contribute to a reduction in the mineable in-situ reserve figure and are known collectively as geological losses.

In underground workings, the method of mining will influence those geological losses deducted from the mineable in-situ reserve figure. If the method to be used is longwall mining, a larger geological loss will occur, due to the fact that longwall operations need hazard-free runs in a designated panel of coal. All faulted areas will need to be discounted if their amounts of throw displace the coal seam to be mined out of line with the preset coal shearer. If the coal panels between faults are too small then the whole block may be discounted. If the bord-and-pillar (equal to room-and-pillar) method is adopted, small faults can often be worked through, and in some cases igneous intrusions can be worked around, as is the case in some South African coalmines. This method also allows for small blocks of coal to still be taken. All methods of mining are affected when coal is lost through washouts, and changes in seam thickness or seam quality.

Geological losses will vary considerably from mine to mine, but in general a 10% geological loss can be expected in opencast operations, whereas geological losses of 25–50% may occur in underground operations.

In addition, losses other than geological may need to be accounted for at this stage. Areas close to lease boundaries may be discounted, as well as reserve areas that run beneath railways, motorways, and critical buildings and installations. Those reserves deemed recoverable may, in certain circumstances, have a minimum depth limit imposed. This is often the case where reserves are accessed by drivages in from the base of the highwall in opencast workings. Such limits will be determined according to the nature of the particular surface area and are intended to reduce the effects of subsidence, particularly close to areas of urban population.





**Figure 7.13** Effects of geological structure and topography on stripping ratios. For this example, the SR cut-off value is taken at 10 : 1 bcm; only shaded area(s) will be considered for mining. Coal seams are numbered in ascending order. (a) Effects of shallow dips. (b) Effects of steep dips. (c) Effects of topography. (d) Effects of faulting. (e) Effects of folding.

In opencast mines, the effects of depth can produce an increase in the SR, which will become uneconomic at some point, as well as imposing geotechnical constraints on deeper excavation.

### 7.3.3.5 Reserves Reporting

Using the above calculations, the objective is to report, first, the mineable in-situ reserves. To do this, the following information is required:

- 1) An outline of the proposed mining method, together with a conceptual mine plan.
- 2) In underground mines, the physical criteria limiting mining, such as maximum and minimum working section thickness, minimum separation of seams, the maximum dip at which the coal can be mined by the stated method, geological structure.
- 3) Overburden ratio or SR in opencast operations.
- 4) Quality restrictions, maximum and minimum levels for ash, sulfur, volatile matter, etc. In the case of coals that have quality problems and need to be beneficiated, the predicted yield needs to be given.
- 5) Depth limits, imposed by either physical or economic constraints or both.

Added to these are mining losses such as

- 1) Areas where coal may not be mined (e.g. beneath motorways).
- 2) Stress fields in the coal seams, which may require the re-orientation of reserve panels, and therefore loss of reserve.
- 3) Roof stability, affecting the thickness of pillars to be left in the mine, again producing loss of reserve.

These and other factors, once deducted from the measured in-situ reserve, enable the measured recoverable (or extractable) reserves to be calculated. These are the reserves required by investors when considering any mine's potential.

### 7.3.3.6 Reserve Economics

Where the mine configuration is known and the production costs and sales figures are also known, it is possible to apply computer methods to determine pit areas of equal value, e.g. in an opencast pit whose walls earn a constant value on the investments made. Such analysis has been applied to metalliferous mines but is equally applicable to modern opencast operations. Such an analysis does require considerable details of the coals mined, and the condition of the mine itself, e.g. whether it is a series of discrete open pits, or separate parts of a mine producing from different seams.

From this it is possible to understand the following relationships:

- (a) Physical (coal quantity and quality), and in the case of low-rank brown coals, the reserves need to be extensive in both thickness and lateral extent to be economic (Waghorne and Heizmann 2001).
- (b) Rate-dependent relationships (quantity of coal A per quantity of coal B).
- (c) Economic relationships and potential (cash flow against pit size).

Additional information such as changes in mine access and need for pit backfilling and restoration all have an influence, as will primary drivers such as market price, market sustainability, and environmental constraints.

## 7.4 World Coal Reserves and Production

### 7.4.1 World Coal Reserves

Estimates of proven coal reserves for black (i.e. for anthracite and bituminous coals) and brown coal (i.e. sub-bituminous coal and lignite) are given in Table 7.6 These figures are taken from BGR (2016) and BP (2017). These proven reserves are those that can be regarded with reasonable certainty to be recoverable

**Table 7.6** World coal reserves (million tonnes).

Region	Bituminous + anthracite	Subbituminous + lignite	Total	R/P ratio
USA	221 400	30 182	251 582	381
Canada	4 346	2 236	6 582	109
Mexico	1 160	51	1 211	151
<b>Total North America</b>	<b>226 906</b>	<b>32 469</b>	<b>259 375</b>	<b>356</b>
Brazil	1 547	5 049	6 596	<sup>a</sup>
Colombia	4 881	—	4 881	54
Venezuela	731	—	731	<sup>a</sup>
Other South and Central America	1 784	24	1 808	<sup>a</sup>
<b>Total South and Central America</b>	<b>8 943</b>	<b>5 073</b>	<b>14 016</b>	<b>138</b>
Bulgaria	192	2 174	2 366	75
Czech Republic	1 103	2 573	3 676	80
Germany	12	36 200	36 212	206
Greece	—	2 876	2 876	87
Hungary	276	2 633	2 909	311
Kazakhstan	25 605	—	25 605	250
Poland	18 700	5 461	24 161	184
Romania	11	280	291	13
Russian Federation	69 634	90 730	160 364	417
Serbia	402	7 112	7 514	196
Spain	868	319	1 187	<sup>a</sup>
Turkey	378	10 975	11 353	163
Ukraine	32 039	2 336	34 375	<sup>a</sup>
UK	70	—	70	17
Uzbekistan	1 375	—	1 375	355
Other Europe and Eurasia	2 618	5 172	7 790	201
<b>Total Europe and Eurasia</b>	<b>153 283</b>	<b>169 841</b>	<b>322 124</b>	<b>284</b>
South Africa	9 893	—	9 893	39
Zimbabwe	502	—	502	186
Other Africa	2 756	66	2 822	276
Middle East	1 203	—	1 203	<sup>a</sup>
<b>Total Africa and Middle East</b>	<b>14 354</b>	<b>66</b>	<b>14 420</b>	<b>54</b>
Australia	68 310	76 508	144 818	294
China	230 004	14 006	244 010	72
India	89 782	4 987	94 769	137

(Continued)

**Table 7.6** (Continued)

Region	Bituminous + anthracite	Subbituminous + lignite	Total	R/P ratio
Indonesia	17 326	8 247	25 573	59
Japan	340	10	350	261
Mongolia	1 170	1 350	2 520	66
New Zealand	825	6 750	7 575	<sup>a</sup>
Pakistan	207	2 857	3 064	<sup>a</sup>
South Korea	326	—	326	189
Thailand	—	1 063	1 063	63
Vietnam	3 116	244	3 360	85
Other Asia Pacific	1 322	646	1 968	29
<b>Total Asia Pacific</b>	<b>412 728</b>	<b>116 668</b>	<b>529 396</b>	<b>102</b>
<b>Total world</b>	<b>816 214</b>	<b>323 117</b>	<b>1 139 331</b>	<b>153</b>

a) More than five hundred years.

Source: Based on *BP Statistical Review of World Energy 2017* (BP 2017).

from known deposits under existing economic and operating conditions; they do not include those large resources of coal that are currently being evaluated or will be in the future.

Table 7.6 gives an overall total of 1 139 331 Mt for those known deposits in the world today. Such a figure in itself is meaningless, but the regional totals do give an indication as to the geographical distribution of the bulk of the world's black coal (816 214 Mt) and brown coal (323 117 Mt) resources. Those countries with one million tonnes or less are omitted from the table.

In North America, South and Central America, Australia, and China, reserves of black and brown coal are evenly divided. In Africa, reserves are predominantly black coal, and in Europe and Eurasia there are greater reserves of brown coal (Table 7.6). The distribution of proved reserves shows little change over the last 25 years, the chief increases being in the Asia Pacific region. The wide distribution of coal is demonstrated in Table 7.6, with major coal deposits existing on every continent. Coal resources are often estimated to be four to

five times greater than estimated reserves, providing potential to upgrade coal resources in the future. Furthermore, reserve figures do not consider other ways of accessing energy from the coal resources (see Chapter 11).

## 7.4.2 World Coal Production

### 7.4.2.1 Coal Production Statistics

Production figures for black and brown coals are shown in Table 7.7 for the six world regions for the years 2013–2016. Total world coal production, based on 2016 figures, is 3,656.4 Mtoe (million tonnes oil equivalent), based on BP (2017). Countries with very minor production levels are excluded. Black coals in the form of thermal and coking coal make up all internationally traded coal in the world. The PRC is the largest producer of black coal, and Germany produces the greatest tonnage of brown coal.

In 2016, global coal production decreased by 6.2%, with the decline in coking coal being higher than for thermal coal. This is primarily due to the weakening of economic growth,

**Table 7.7** World coal production (million tonnes oil equivalent).

Region	2013	2014	2015	2016
USA	500.9	507.7	449.3	364.8
Canada	36.4	35.6	31.9	31.4
Mexico	7.2	7.3	6.9	4.5
<b>Total North America</b>	<b>544.5</b>	<b>550.5</b>	<b>488.1</b>	<b>400.7</b>
Brazil	3.7	3.4	3.5	3.5
Colombia	59.0	61.1	59.0	62.5
Venezuela	0.9	0.6	0.6	0.2
Other South and Central America	1.7	2.4	1.9	1.5
<b>Total South and Central America</b>	<b>65.3</b>	<b>67.5</b>	<b>64.9</b>	<b>67.6</b>
Bulgaria	4.8	5.1	5.8	5.1
Czech Republic	17.7	16.8	16.8	16.3
Germany	45.1	44.1	42.9	39.9
Greece	6.7	6.4	5.7	4.1
Hungary	1.6	1.6	1.5	1.5
Kazakhstan	51.4	48.9	46.2	44.1
Poland	57.2	54.0	53.0	52.3
Romania	4.7	4.4	4.7	4.3
Russian Federation	173.1	176.6	186.4	192.8
Serbia	7.7	5.7	7.2	7.4
Spain	1.8	1.6	1.2	0.7
Turkey	15.5	16.4	12.8	15.2
Ukraine	36.6	25.9	16.4	17.1
UK	8.0	7.3	5.4	2.6
Uzbekistan	1.1	1.2	1.1	1.1
Other Europe and Eurasia	18.0	17.0	15.3	14.9
<b>Total Europe and Eurasia</b>	<b>450.9</b>	<b>433.2</b>	<b>422.5</b>	<b>419.4</b>
<b>Total Middle East</b>	<b>0.7</b>	<b>0.6</b>	<b>0.7</b>	<b>0.7</b>
South Africa	145.3	148.2	142.9	142.4
Zimbabwe	2.0	3.7	2.8	1.7
Other Africa	5.1	5.5	6.0	6.3
<b>Total Africa</b>	<b>152.3</b>	<b>157.5</b>	<b>151.7</b>	<b>150.5</b>
Australia	285.8	305.7	305.8	299.3
China	1894.6	1864.2	1825.6	1685.7
India	255.7	269.5	280.9	288.5
Indonesia	279.7	269.9	272.0	255.7
Japan	0.7	0.7	0.6	0.7
Mongolia	18.0	14.8	14.5	22.8
New Zealand	2.8	2.5	2.0	1.7
Pakistan	1.3	1.5	1.5	1.8
South Korea	0.8	0.8	0.8	0.8
Thailand	4.9	4.8	3.9	4.3
Vietnam	23.0	23.0	23.2	22.0
Other Asia Pacific	25.1	25.7	28.6	33.9
<b>Total Asia Pacific</b>	<b>2792.5</b>	<b>2783.1</b>	<b>2759.4</b>	<b>2617.4</b>
<b>Total world</b>	<b>4006.1</b>	<b>3992.4</b>	<b>3887.3</b>	<b>3656.4</b>

Source: Based on BP Statistical Review of World Energy 2017 (BP 2017).

**Table 7.8** World coal consumption (million tonnes oil equivalent).

Region	2013	2014	2015	2016
USA	454.6	453.5	391.8	358.4
Canada	20.8	19.7	19.6	18.7
Mexico	12.7	12.7	12.7	9.8
<b>Total North America</b>	<b>488.1</b>	<b>486.0</b>	<b>424.2</b>	<b>386.9</b>
Argentina	1.3	1.5	1.4	1.1
Brazil	16.5	17.5	17.7	16.5
Chile	7.5	7.6	7.3	8.2
Colombia	5.0	5.2	5.3	4.6
Peru	0.9	0.9	0.8	0.8
Venezuela	0.2	0.2	0.2	0.1
Other South. and Central America	2.9	3.2	3.2	3.4
<b>Total South and Central America</b>	<b>34.2</b>	<b>36.1</b>	<b>35.9</b>	<b>34.7</b>
Austria	3.3	3.0	3.2	3.2
Belarus	0.9	0.8	0.7	0.8
Belgium and Luxembourg	3.3	3.3	3.2	3.0
Bulgaria	5.9	6.4	6.6	5.7
Czech Republic	17.2	16.0	16.6	16.9
Denmark	3.2	2.6	1.7	2.1
Finland	5.0	4.5	3.8	4.1
France	11.6	8.6	8.4	8.3
Germany	82.8	79.6	78.5	75.3
Greece	7.0	6.7	5.6	4.7
Hungary	2.3	2.2	2.4	2.3
Republic of Ireland	2.0	2.0	2.2	2.2
Italy	13.5	13.1	12.3	10.9
Kazakhstan	36.3	41.0	35.8	35.6
Lithuania	0.3	0.2	0.2	0.2
Netherlands	8.2	9.1	11.0	10.3
Norway	0.8	0.9	0.8	0.8
Poland	53.4	49.4	48.7	48.8
Portugal	2.7	2.7	3.3	2.9
Romania	5.8	5.7	5.9	5.4
Russian Federation	90.5	87.6	92.2	87.3
Slovakia	3.5	3.4	3.3	3.1
Spain	11.4	11.6	13.7	10.4
Sweden	2.2	2.1	2.1	2.2
Switzerland	0.1	0.1	0.1	0.1

*(Continued)*



Table 7.8 (Continued)

Region	2013	2014	2015	2016
Turkey	31.6	36.1	34.7	38.4
Ukraine	41.6	35.6	27.3	31.5
UK	36.8	29.7	23.0	11.0
Uzbekistan	1.1	1.2	1.1	1.0
Other Europe and Eurasia	23.8	21.9	23.0	23.0
<b>Total Europe and Eurasia</b>	<b>508.1</b>	<b>487.3</b>	<b>471.3</b>	<b>451.6</b>
Iran	1.4	1.6	1.6	1.7
Israel	7.4	6.9	6.7	5.7
Saudi Arabia	0.1	0.1	0.1	0.1
United Arab Emirates	1.4	1.5	1.3	1.3
Other Middle East	0.5	0.7	0.5	0.5
<b>Total Middle East</b>	<b>10.9</b>	<b>10.8</b>	<b>10.2</b>	<b>9.3</b>
Algeria	0.2	0.2	0.1	0.1
Egypt	0.4	0.4	0.4	0.4
South Africa	88.6	89.8	83.4	85.1
Other Africa	8.3	11.9	11.4	10.3
<b>Total Africa</b>	<b>97.5</b>	<b>102.3</b>	<b>95.3</b>	<b>95.9</b>
Australia	43.0	42.6	44.1	43.8
Bangladesh	1.0	0.8	0.7	0.8
China	1969.1	1954.5	1913.6	1887.6
China Hong Kong SAR	7.8	8.1	6.7	6.7
India	352.8	387.5	396.6	411.9
Indonesia	57.0	45.1	51.2	62.7
Japan	121.2	119.1	119.9	119.9
Malaysia	15.1	15.4	16.9	19.9
New Zealand	1.5	1.5	1.4	1.2
Pakistan	3.2	4.7	4.7	5.4
Philippines	10.0	10.6	11.6	13.5
Singapore	0.3	0.4	0.4	0.4
South Korea	81.9	84.6	85.5	81.6
Taiwan	38.6	39.0	37.8	38.6
Thailand	16.3	17.9	17.6	17.7
Vietnam	15.8	18.9	22.3	21.3
Other Asia Pacific	13.8	16.0	16.9	20.6
<b>Total Asia Pacific</b>	<b>2748.3</b>	<b>2767.0</b>	<b>2747.7</b>	<b>2753.6</b>
<b>Total world</b>	<b>3887.0</b>	<b>3889.4</b>	<b>3784.7</b>	<b>3732.0</b>

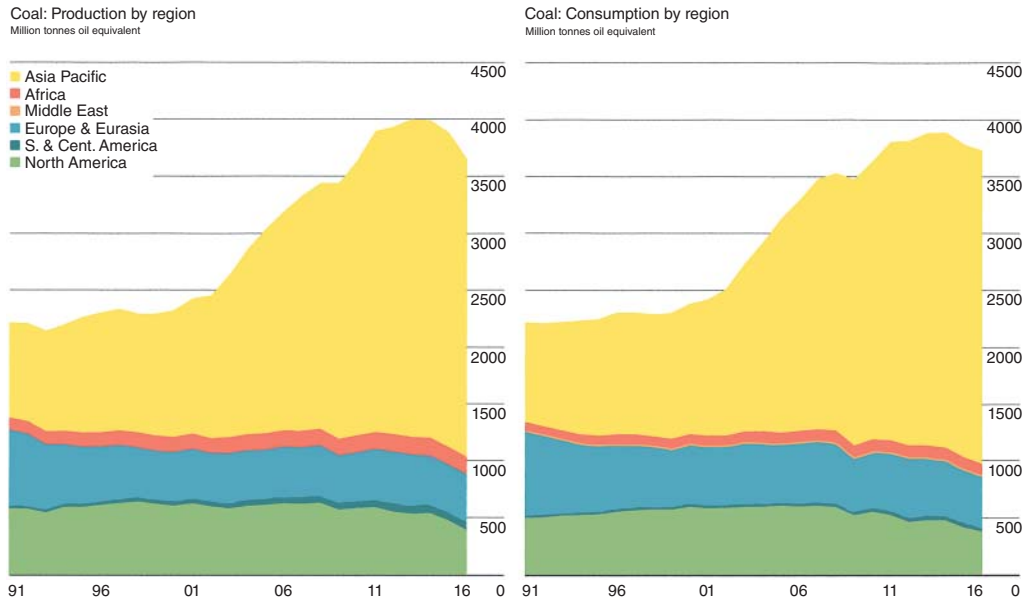
Source: Based on BP Statistical Review of World Energy 2017 (BP 2017).

competitiveness of natural gas and renewables, together with government and social pressure to shift towards cleaner, lower carbon fuels. Changes in coal prices and increases in demand, particularly electricity generation, in countries with little choice in fuel options will continue to affect coal production; however, the big coal producers, such as the PRC, the USA, India, Australia, and Indonesia, will still have the strongest influence on coal production figures.

#### 7.4.2.2 Regional Production and Consumption

Table 7.8 shows the principal areas of coal consumption, with total world consumption being 3732 Mtoe in 2016. This is a decline of 1.7% compared with 2015. The biggest reductions have occurred in the PRC (1.6%), the USA (8.8%), and the UK (52.5%). This can be

compared with Table 7.7 to illustrate that the bulk of all coal mined is actually consumed in the area in which it is produced. Figure 7.14 shows a comparison of coal consumed against coal produced worldwide for the period 1991–2016, with the biggest increase occurring in the Asia-Pacific region. Those countries in which coal production is greater than consumption and where the coal quality is internationally marketable are coal exporters, e.g. the USA, Australia, and South Africa. Smaller producers, such as Colombia, export a large percentage of their total production, whereas others, such as the PRC and North Korea, produce large tonnages but use almost all domestically. Those developing producers, such as Mongolia and Indonesia, also produce more coal than they will consume; this production is intended for the export market.



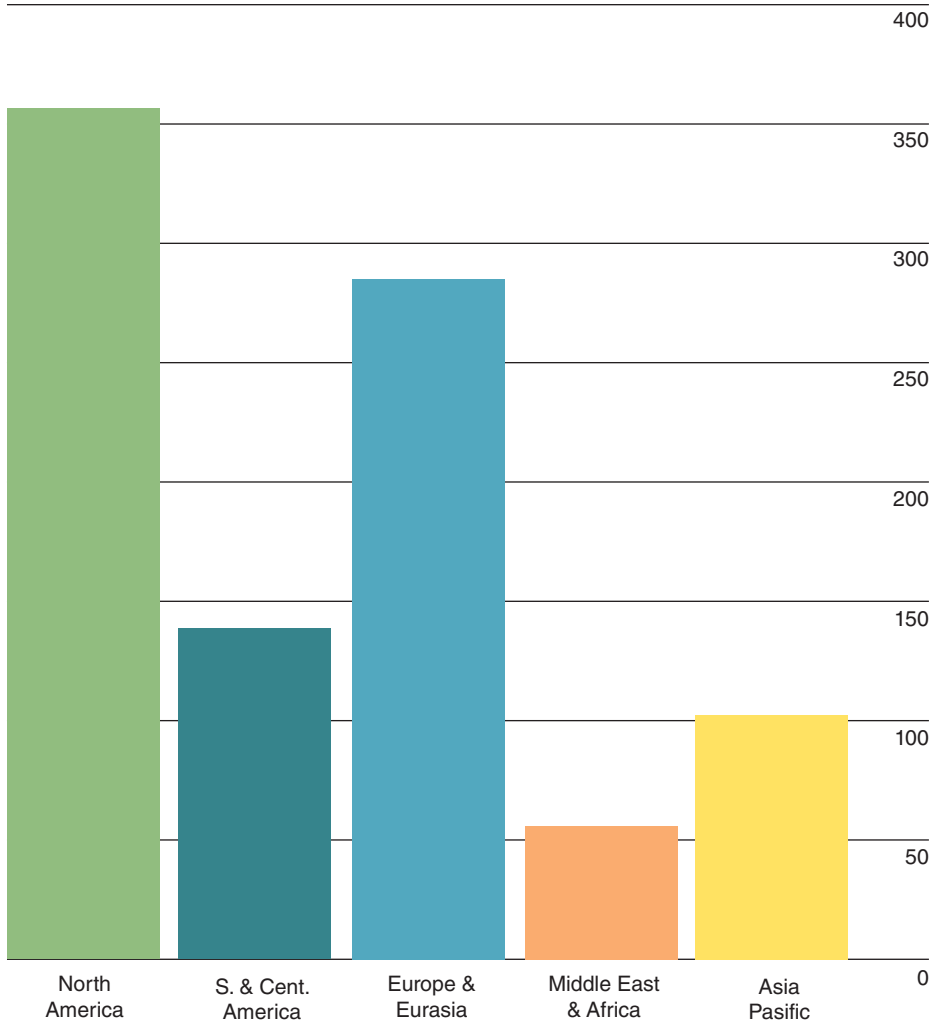
**Figure 7.14** Coal production and consumption from 1985 to 2010. *Source: BP Statistical Review of World Energy 2011 (BP 2011).*

### 7.4.2.3 Reserves/Production Ratio

The reserves/production ( $R/P$ ) ratio is based on those reserves remaining at the end of any year divided by the production in that year. The result is the length of time that those remaining reserves would last if production were to

continue at the present level. World proved reserves of coal in 2016 are sufficient to meet 153 years of global production, roughly three times the  $R/P$  rate for oil and gas. Figure 7.15 illustrates  $R/P$  ratios by region.

#### 2016 by region



**Figure 7.15** Fossil fuel  $R/P$  ratios at end 2010. Source: *BP Statistical Review of World Energy 2011* (BP 2011).

## 8

# Geophysics of Coal

## 8.1 Introduction

The use of geophysics in the exploration for coal basins and in the delineation of coal seams and geological structures in coal deposits is now well established.

Since the oil crisis in 1973, there has been an enormous increase in the use of geophysical methods to identify coal deposits and to further determine their economic potential. Several of the techniques have been used initially in the exploration for oil and gas and adapted where applicable to coal exploration.

Large-scale studies make use of regional gravity, deep seismic, and aeromagnetic surveys to determine the sedimentary and structural framework of the area under consideration. On the smaller scale, more detailed examination of coal deposits utilises shallow seismic, ground magnetic, electrical resistivity, and microgravity methods coupled with the geophysical logging of all boreholes, which in turn involves the use of density, electrical, electromagnetic (EM), and radiometric techniques.

In established coal mining areas, the combination of geophysical logging with high-resolution seismic and ground magnetic surveys contributes significantly in the delineation of economic mining areas, both for opencast and underground operations. In underground operations, the use of the in-seam seismic (ISS) method is now used as a tactical tool in planning the orientation

of mining areas, particularly in the siting of longwall panels.

This combination is used on both large- and small-scale investigations; the drawback is that it can be an expensive exercise to carry out such investigations. When planning an exploration programme, any use of geophysics, whether as a field survey or in borehole logging, will be a high-cost item on the exploration budget. The benefits of using such techniques, such as whether the amount of drilling required will be reduced, will be set against such costs.

The background principles of the physics and mathematics governing the various geophysical techniques employed in coal exploration and development are not covered in this book as they are available in standard geophysical texts, such as Buchanan and Jackson (1986), and others covering more specific aspects of coal geophysics exploration, such as the ISS method (Dresen and Ruter 1994). Coal geophysics has benefited from the recent use of computer methods for geophysical data collection, processing, modelling, and accurate interpretation. These recent developments are covered in modern applied geophysical books, such as Milsom and Eriksen (2011) and Reynolds (2011), and summarised in Hatherly (2013). In this chapter, only a simple outline is given of the basic physical properties of coal-bearing sequences together with an outline of the field methods used to locate and quantify coal deposits.

## 8.2 Physical Properties of Coal-Bearing Sequences

Coal as a lithology responds well to most geophysical methods, in that its physical properties contrast with those of other lithologies commonly found in coal-bearing sequences. Coal has, in general, a lower density, a lower seismic velocity, a lower magnetic susceptibility, a higher electrical resistivity, and low radioactivity compared with surrounding rocks in typical coal-bearing sequences.

### 8.2.1 Density

Density measurements of rocks are not usually measured in situ, but in the laboratory on some small outcrop or drill core samples. Such results rarely give the true bulk density because samples may be weathered or dehydrated; consequently, density is not often well known in specific field situations. Table 8.1 gives saturated density ranges and averages for coals, sediments in coal-bearing sequences, and igneous and metamorphic rocks that

may be associated with coal basins either as underlying basement or as intrusives into the coal-bearing strata.

Low- and high-rank coals ( $1.1\text{--}1.8\text{ g cm}^{-3}$ ) are less dense than the surrounding sediments ( $1.6\text{--}2.9\text{ g cm}^{-3}$ ), which in turn are less dense than igneous and metamorphic rocks ( $2.1\text{--}3.1\text{ g cm}^{-3}$ ). In sedimentary rocks, the wide range of density is due to variations in porosity, the nature of pore fluids, age, and depth of burial, as well as mineralogical composition. Some igneous rocks, such as volcanics, have high porosities, and therefore lower density; for example, pumice can have a density less than  $1.0\text{ g cm}^{-3}$ . Density also increases with the degree of metamorphism, as recrystallisation reduces pore space to form a denser rock as well as converts some minerals to denser forms.

### 8.2.2 Seismic Velocity

The seismic velocity of a rock is the velocity at which a wave motion propagates through the rock media. As shown in Table 8.1, the seismic velocity of coal is in the range  $1.8\text{--}2.8\text{ km s}^{-1}$ ;

**Table 8.1** Table of physical properties of coals and associated sedimentary and igneous rocks.

Lithology	Density, wet ( $\text{g cm}^{-3}$ )		Seismic velocity ( $\text{km s}^{-1}$ )	Magnetic susceptibility ( $\times 10$ SI units)		Electrical resistivity ( $\Omega\text{ m}$ )
	Range	Average		Range	Average	
Sandstone	1.61–2.76	2.35	3.6	0–20	0.4	$1\text{--}6.4 \times 10^8$
Shale	1.77–3.20	2.40	2.8	0.01–15	0.6	$20\text{--}2 \times 10^3$
Limestone	1.93–2.90	2.55	5.5	0–3	0.3	$50\text{--}1 \times 10^7$
Lignite	1.10–1.25	1.19				9–200
Bituminous coal	1.20–1.80	1.32	1.8–2.8	—	0.02	$0.6\text{--}1 \times 10^5$
Anthracite	1.34–1.80	1.50				$1 \times 10^{-3}\text{--}2 \times 10^5$
Acid igneous rock	2.30–3.11	2.61	4.0–5.5	0–80	8.0	$4.5 \times 10^3$ (wet granite) $1.3 \times 10^6$ (dry granite)
Basic igneous rock	2.09–3.17	2.79	4.0–7.0	0.5–100	25.0	$20\text{--}5 \times 10^7$ (dolerite)
Metamorphic rock	2.40–3.10	2.74	5.0–7.0	0–70	4.2	$20\text{--}1 \times 10^4$ (schist)

Source: Based on Telford et al. (1990).

mudrocks, such as shales, have similar values. Sandstones have a higher value, which increases with increased quartz content, whereas dense limestones and igneous and metamorphic rocks have much higher velocities of 4.0–7.0 km s<sup>-1</sup>.

### 8.2.3 Seismic Reflection Coefficients

The seismic reflection coefficient determines whether an interface gives a reflection, and it depends upon the density and the seismic velocity. Coal seams with a low density and low seismic velocity often have high reflection coefficients and can be picked up well on seismic sections.

### 8.2.4 Magnetic Susceptibility

The magnetic susceptibility of a rock depends primarily on its magnetite content. Weathering generally reduces susceptibility because of the oxidation of magnetite to haematite. As in the case of rock density, measurements of magnetic susceptibility in the field do not necessarily give a bulk susceptibility of the formation; however, outcrop magnetic susceptibility measurements by portable instruments have led to improved bulk susceptibility measurements. Although there is great variation in magnetic susceptibility, even for a particular lithology, and wide overlap between different types, sedimentary rocks generally have the lowest average susceptibilities, with coals having amongst the lowest susceptibility within the sedimentary suite (see Table 8.1). Basic igneous rocks have high susceptibility values. In every case, the susceptibility depends on the amount of ferromagnetic minerals present, mainly magnetite, titanomagnetite, or pyrrhotite. It is worth noting that sulfide minerals, such as pyrite, which is a common mineral in coals and associated sediments, have a low susceptibility value; like many of the sulfide minerals, it is almost non-magnetic. Table 8.1 gives the range and average values in rationalised SI units for those rocks associated with coal.

### 8.2.5 Electrical Conductivity

Electrical prospecting involves the detection of surface effects produced by electric current flow in the ground. It is the enormous variation in electrical conductivity found in different rocks and minerals that requires a greater variety of techniques to be used than in the other prospecting methods.

Several electrical properties of rocks and minerals are significant in electrical prospecting. Of these, by far the most important in coal prospecting is electrical conductivity or the inverse electrical resistivity, which is expressed in ohm-metres ( $\Omega$  m), the others being of less significance. As most rocks are poor conductors, their resistivities would be extremely large were it not for the fact that they are usually porous and the pores are filled with fluids, mainly water. The conductivity of a porous rock varies with the volume and arrangement of the pores, and the conductivity and amount of contained water. Water conductivity varies considerably depending on the amount and conductivity of dissolved chlorides, sulfates, and other minerals present, but the principle influence is usually the sodium chloride or salt content.

### 8.2.6 Radiometric Properties

Trace quantities of radioactive material are found in all rocks. Small amounts of cosmic radiation passing through the atmosphere produce a continuous background reading that may vary from place to place. In general, the radioactivity in sedimentary rocks and metamorphosed sediments is higher than that in igneous and other metamorphic types, with the exception of potassium-rich granites.

In coal-bearing sequences, the contrasts in natural radioactivity in coals and surrounding sediments has led to the development of the use of nuclear well logging instruments for measuring radioactivity of formations encountered in boreholes.

Coals have very low radioactivity, as do clean sandstones, sandstones with high contents of rock fragments, and clay matrices; siltstones



and non-marine shales have low to intermediate values, whereas marine shale and bentonite (tonstein) have high radioactivity due to the presence of uranium/thorium minerals in the shale and potassium in the bentonite.

## 8.3 Surface Geophysical Methods

The petroleum industry has used various seismic geophysical methods for a number of years as an aid in the exploration for geological structures suitable for hydrocarbon entrapment. In order to locate sedimentary basins, electrical, EM, gravity, and magnetic surveys together with reflection and refraction seismic surveys are used. These are usually large-scale operations involving a great deal of equipment, manpower, and finance. Although of use in broad regional investigations, they are little used in the examination of coal-bearing sequences for small selected areas.

In the investigation of mine lease areas, high-resolution seismic reflection surveys are the most effectively employed. Other methods used are cross-borehole seismic techniques and seismic refraction, which are particularly useful in opencast mine development.

### 8.3.1 Seismic Surveys

Exploration for mineable coal is generally concerned with the top 1.5 km of strata; because of this shallow nature of the investigations, high-resolution seismic profiling is required to detect relatively thin coal seams. The recording system is designed to retain as much of the high-frequency reflections as possible.

The use of satellites to determine position by using global positioning systems allows accuracy of 1–2 m, with an even greater accuracy being achieved after post-processing of the received data (Figure 8.1). Such positioning accuracy has greatly increased the efficiency of surface seismic reflection surveying, which has steadily improved over the last thirty-five

years, and it is now applied to coal mining with increased accuracy and confidence.

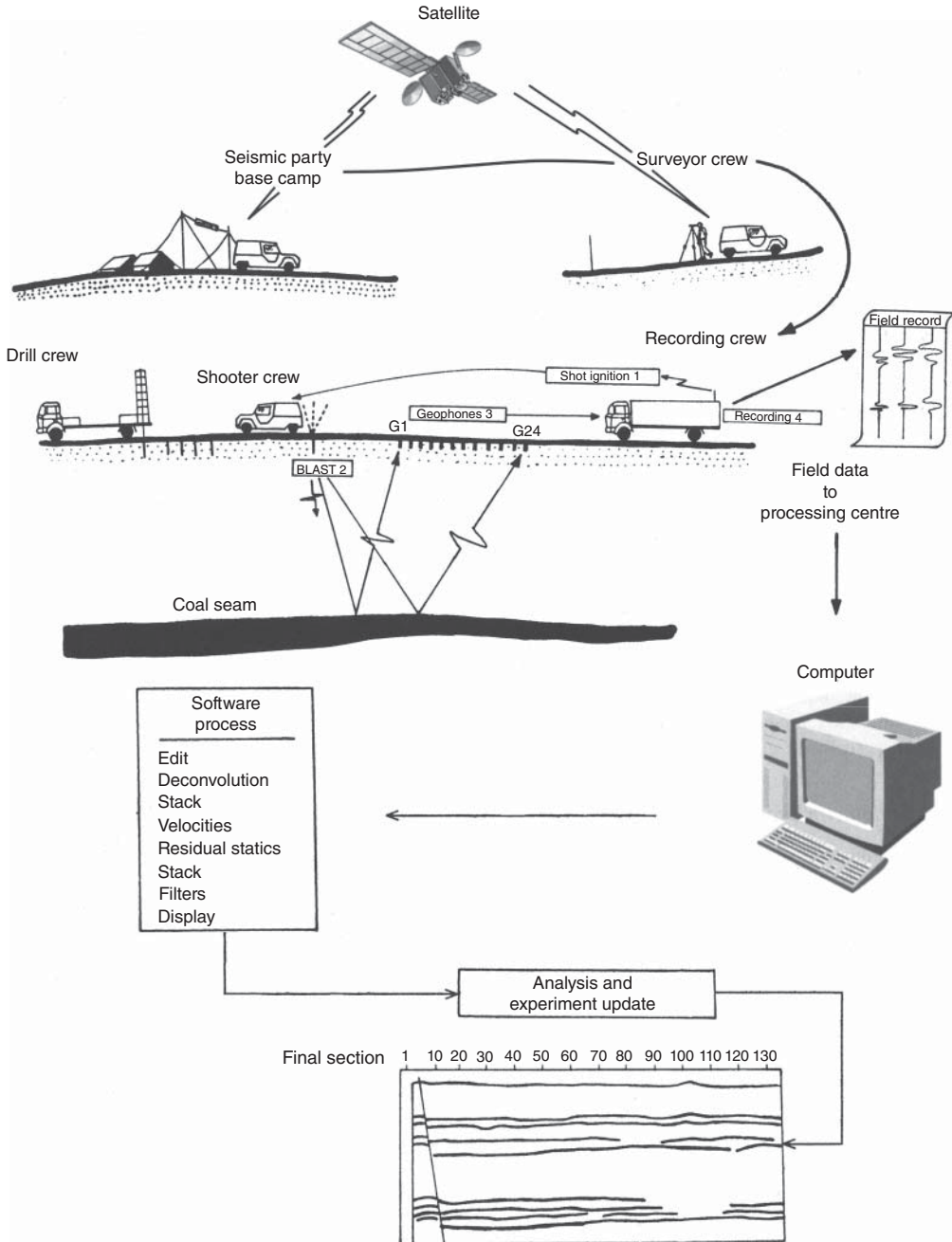
#### 8.3.1.1 Seismic Reflection Surveys

The principle of seismic reflection is that an acoustic signal or seismic wave produced by an explosion or other impulse source is introduced into the ground at selected points, and this signal radiates through the ground. The velocity at which the signal travels depends upon the rock type encountered. Typical velocities are  $5.5 \text{ km s}^{-1}$  in hard, dense limestone, with a range between  $1.8$  and  $2.8 \text{ km s}^{-1}$  for most types of hard coals. These and other velocities of typical lithologies encountered in coal-bearing successions are given in Table 8.1. The velocity of the seismic wave is a function of the lithology through which it passes; when the wave reaches a boundary marking a lithological change, a reflected and a refracted ray result. When the changes in velocities and density at a boundary are large, there is a large reflection coefficient and a strong reflection is generated; this reflection is detected by receivers or geophones, which produce an electrical signal that is recorded, as shown in Figure 8.1.

The instant the signal is generated it is also recorded, and by recording the time it took for the signal to reach the reflection point and return, referred to as two-way travel time (TWT), the depth of the reflection point can be determined, providing the velocities of the traversed lithologies are known.

The physical property of coal and its surrounding strata that makes seismic surveys feasible is its acoustic impedance, which is defined as the product of its density and seismic velocity. Coal has a much lower density and velocity relative to other sedimentary rocks normally encountered in coal-bearing sequences; the contrast in acoustic impedance between coal and often sediments may be between 35% and 50%. This produces large reflection coefficients (Hughes and Kennett 1983).

In seismic surveys of coal deposits, reflections from most features of interest return to the surface during the first second after the



**Figure 8.1** Seismic reflection survey, showing field data acquisition and seismic data processing. *Source:* Adapted from Peace (1979).

seismic wave has been generated, i.e. the TWT. This would give a maximum reflection depth of 1.5 km if the overlying rocks have an average seismic velocity of  $3.0 \text{ km s}^{-1}$ .

The advance in the identification of smaller and smaller disturbances in coal seams is mainly due to the increased power of computers, which has enabled interactive

data processing, in particular, the opportunity to use combinations of selected parameters, with the capacity to refine the data along each seismic section and between sections. This technique has been successfully carried out in the UK, where surface seismic exploration has been combined with ISS methods (see Section 8.4.1). The latter is able to map smaller faults that are below the resolution of surface seismic surveys (Carpenter and Robson 2000).

The ability to identify faults, folds, washouts, seam splits, and thickness changes by the use of seismic reflection techniques is an effective method of pinpointing potential geological hazards that can then be built in to mine planning and design.

The source commonly used in seismic reflection surveys for coal is either the detonation of an explosive charge, such as dynamite (but this produces environmental problems in populated areas), or a lower energy impulse produced by an earth compactor known as the Mini-SOSIE (Pinchin et al. 1982). Another energy source used for shallow reflection surveys is a 'gun' firing blank ammunition into the ground, but this is only suitable for investigation to around 150 m in rocks with good transmission properties.

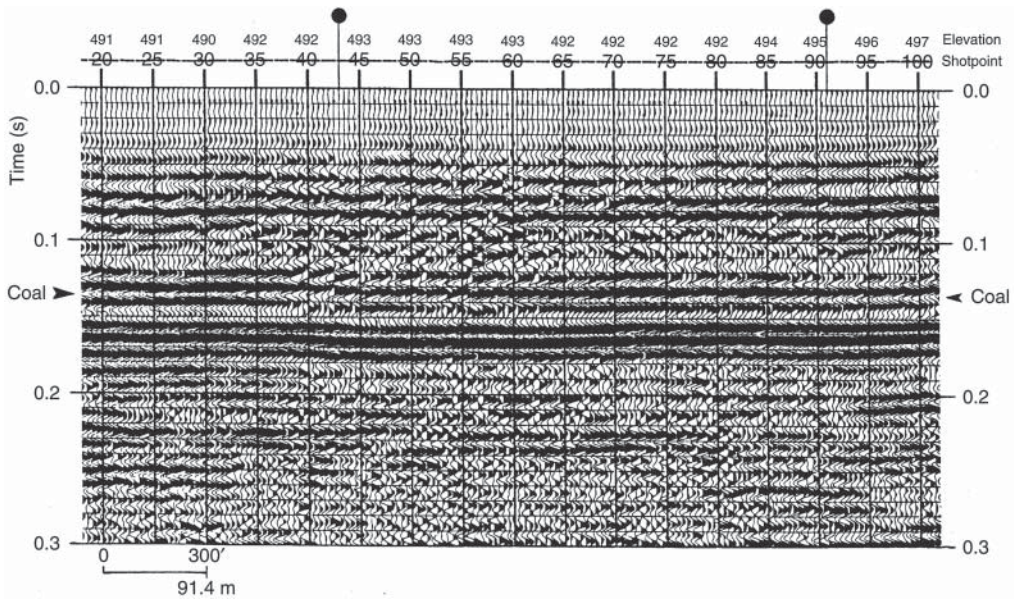
One of the main problems with any shallow seismic survey is the effect of the total travel time when the waves have to pass through a low-velocity weathered zone near the ground surface. Variations in the thickness and velocity characteristics of the weathered zone produce variations in the arrival times of the wave reflections. This can be partly overcome by placing the shot point and, if possible, the geophone in holes drilled to below the low-velocity zone, although this is often not feasible due to the extent and depth of weathering. This can be further complicated by the presence of superficial deposits masking the rockhead. Weathering is a particular problem in subtropical and tropical countries, such as Africa and Australia, whereas superficial deposits are a common problem in Europe and North America.

The ease of processing and interpreting the data from surface seismic surveys is naturally influenced by the local geology. Where coal deposits are geologically uncomplicated (i.e. thick seams with low dips, little faulting, and close to the surface with little weathering effects), interpretation will be relatively easy. Contrast this with other geological scenarios where coal seams exhibit complexities in splitting and variation in thickness, have a high incidence of faulting or the presence of washouts in the coal seams, and perhaps lie at depths up to 1.0 km, then the interpretation of seismic data from surveys in such areas is a lot more difficult.

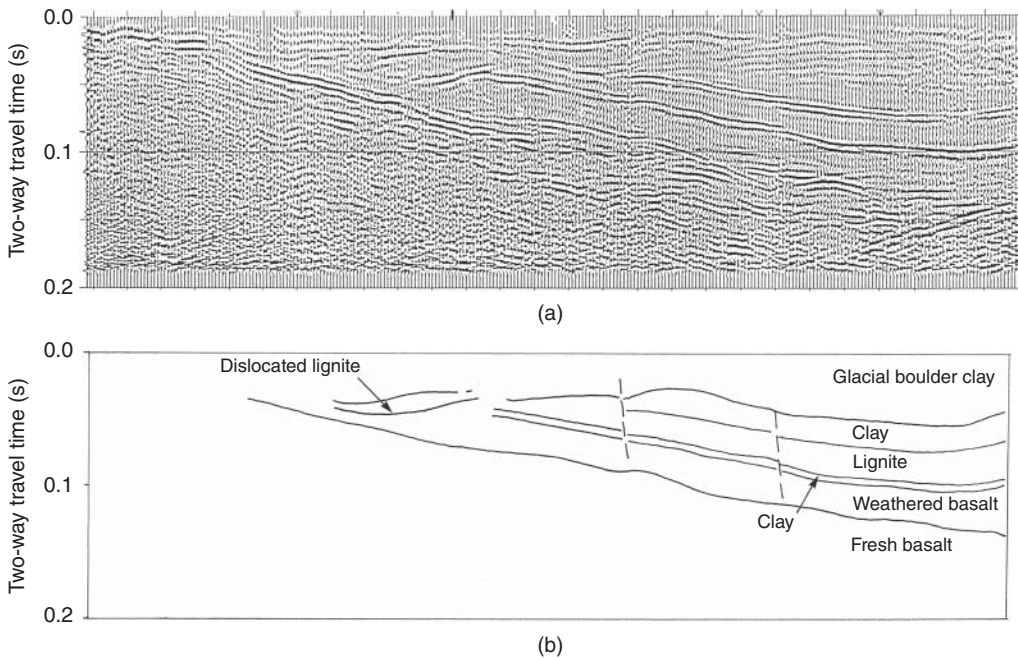
Figure 8.2 illustrates a seismic reflection profile across a coalfield in the USA. The section shows the coal seam reflection to be robust and continuous from shotpoints 20 to 100, indicating uniform coal seam thickness across the entire section.

Figure 8.3a shows a seismic reflection profile across a lignite deposit in Northern Ireland, UK; this is a product of a shallow reflection seismic survey using a 'gun' energy source. In this survey, in order to provide depth resolution, the higher frequencies of seismic waves were used. The sequence consisted predominantly of saturated Palaeogene–Neogene clays, which have a very low attenuation for seismic energy; this suggests an expectation of good high-frequency transmission. This is also dependent upon the effects of the superficial deposits in which the energy source is coupled. The geology of the area consists of superficial heterogeneous glacial deposits overlying clays and lignite; these, in turn, lie on a zone of weathered basalt with fresh basalt at depth. In this instance there is little seismic velocity contrast between the clays and lignites but, because there is a large density contrast, the acoustic impedance is sufficiently large to produce a large reflection coefficient and a detectable reflection.

The interpretation of Figure 8.3a is shown in Figure 8.3b. The good reflectors at about 0.07 and 0.1 s (TWT) on the south-east margin



**Figure 8.2** Seismic section showing a robust and continuous coal seam reflection. This indicates uniform coal seam thickness with no detectable geological disturbance. *Source:* From Gochioco (1991).



**Figure 8.3** (a) Shallow seismic reflection survey, Northern Ireland, UK. (b) Interpretation of seismic section in (a), clearly showing interbedded lignite and underlying basalt. *Source:* Unpublished figure reproduced with permission of Antrim Coal Company Ltd.

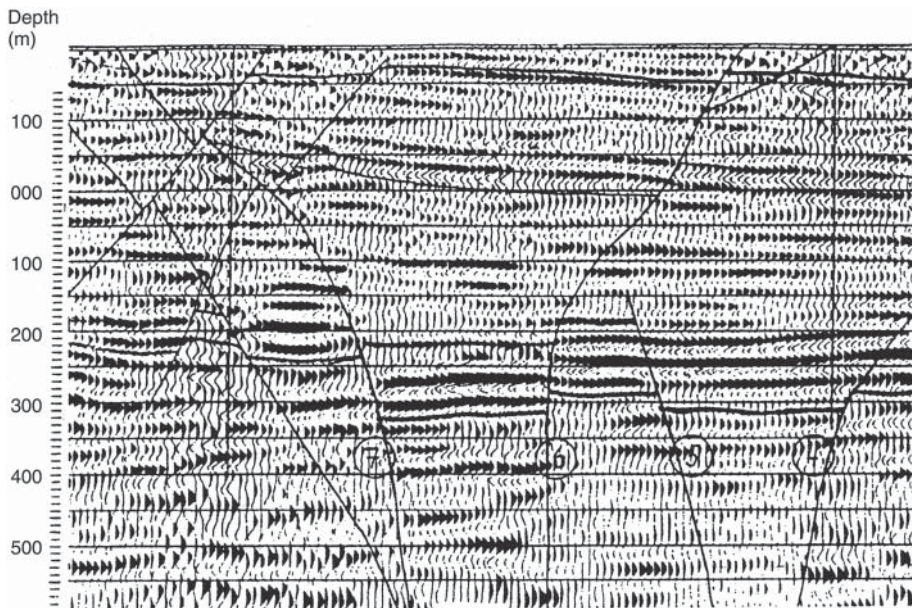


mark the top and bottom of the lignite. The weathered–fresh basalt interface is seen on the south-east margin at about 0.13 s (TWT) and can be seen clearly to shallow towards the north-west. In addition, two faults with downthrows to the south-east can be detected. The irregularly shaped body on the left of the section is considered to be a raft of lignite that has become detached from the main lignite due to frost action during the Quaternary glacial period.

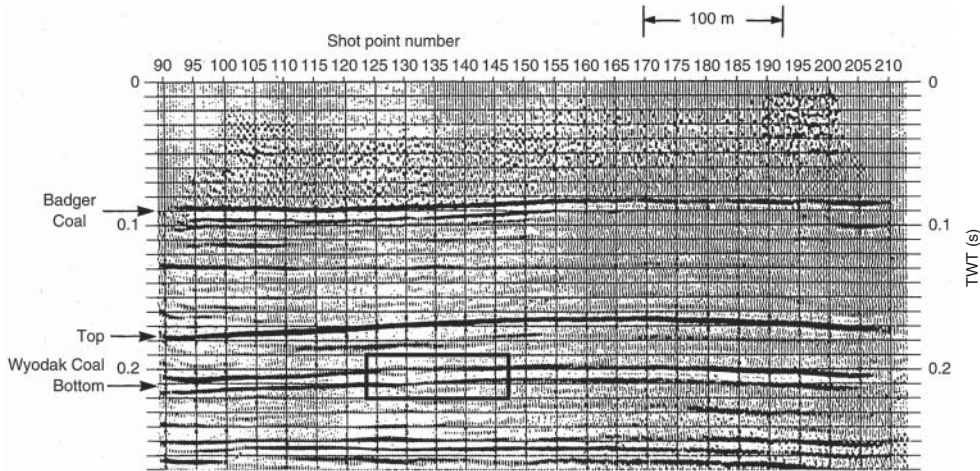
Surface reflection seismic surveys have been carried out in the Kladno Coalfield in Bohemia. Oplustil et al. (1997) described the results of seismic measurements across the Kladno Basin, and Figure 8.4 shows the depth seismic profiles and the interpreted faulting. Up to 75% of seismic indications correspond with observations in underground mines. The number of normal faults in the seismic profiles exceeds the number of observed faults, and in one case a large fault with a downthrow of tens of metres has been interpreted with no equivalent in any mine. There is also some suggestion of faults being the result of synsedimentary movements in the deposit.

In Wyoming, USA, high-resolution reflection profiles were recorded over a prospective underground coal gasification test site. The target seam was the Wyodak coal some 180 m below the surface with a thickness of 30 m. Seismic reflection profiling was considered the most effective on technical and financial grounds. This survey used a dynamite energy source, and one profile line is shown in Figure 8.5. The top and bottom of the Wyodak coal give strong reflections, as does the overlying Badger coal (3 m thick). All the sections show a gentle anticlinal structure in the sequence, particularly in the top of the seam. This may be due to differential compaction of the coal seam or movement within the coal seam as a response to gravity in a down dip direction before coalification occurred. No faulting was observed in the profile. The base of the Wyodak coal has a zone of anomalous amplitude (Figure 8.5), interpreted as a washout in the basal section of the seam; later drilling proved this to be the case.

Figure 8.6 shows a distinct washout structure in the roof of a lignite in the Texas Gulf Coast region. The edges of the strata surrounding

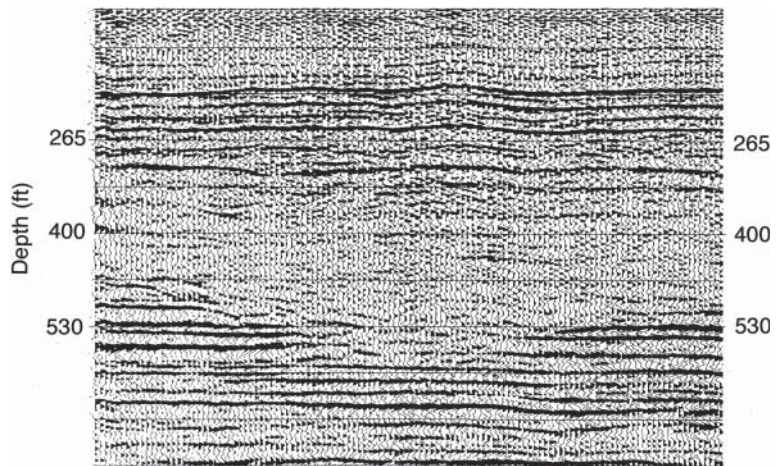


**Figure 8.4** Depth seismic profile with interpreted faults. *Source:* From Oplustil et al. (1997).



**Figure 8.5** Seismic profile showing anticline structure of Wyodak coal and the good reflection of Badger coal. Seismic anomaly (boxed area) is interpreted as a channel cut into the base of the Wyodak coal, Wyoming, USA. *Source:* From Greaves (1985).

**Figure 8.6** Seismic reflection profile showing a buried channel at a depth of 152.4 m (500 ft), Texas Gulf Coast, USA. *Source:* From Peace (1979).



the washout are quite distinct; from this, the dimensions of the channel can be measured and located within any proposed mine development. In this example, the washout is approximately 35 m (115 ft) thick, extending from 141 m (465 ft) to 176 m (580 ft) on the section.

High-resolution seismic methods were applied to the Domeniko coal basin in central Greece to successfully detect low-angle thrusts (Tselentis and Paraskevopoulos 2002). Using geostatistical methods to combine the seismic results with borehole data revealed

the three-dimensional (3D) model of this coal basin. High-resolution seismic methods have also been applied to the exploration and development of coal deposits in the Upper Silesian and Lublin coal basins in Poland in various geological and mining conditions to determine commercial coal seams, locate non-coal areas and faults, and solve some problems associated with mining conditions. These surveys enabled improved designs of new mines and/or expanded existing ones to provide more productive and safer coalmines (Pietsch and Slusarczyk 1992).

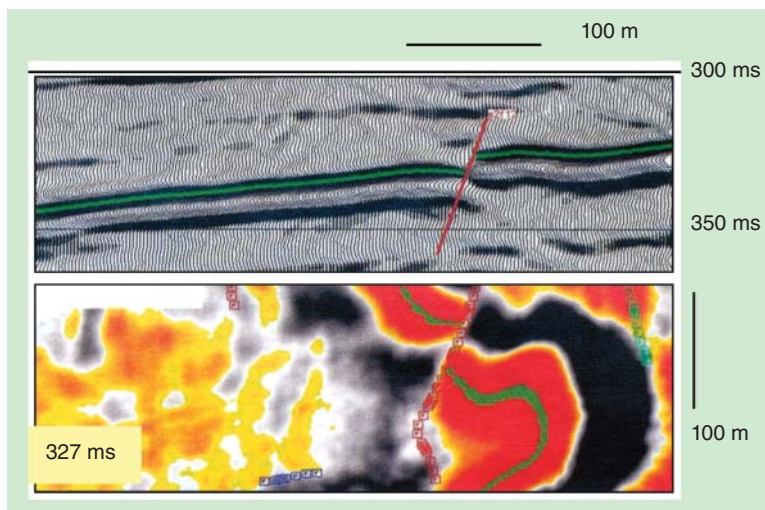


For the past two decades all new coalmines in China have been required to be preceded by a seismic survey before approval of the mine plan and commencement of coal production (Zhao and Wu 2005). An early example of the success of seismic surveys is in the Chensilou coalmine, where the geological survey only detected a couple of faults but a subsequent seismic survey revealed 24 faults. The workforce and coal production plan were completely altered, with an estimated saving of US\$ 20 million. Initially, these surveys were all two-dimensional (2D); subsequently, they were replaced by 3D seismic surveys, which formed 95% of all seismic surveys in 2003, when a total of 350 surveys had been completed. These 3D seismic surveys not only allow display of colour seismic sections but also of colour time and horizontal slices (Figure 8.7), thereby helping to detect the areas without coal, old coal workings, and especially coal seam faults. The fault detection success quoted by Zhao and Wu (2005) is 85% of all faults with displacement of 5 m or more and 60% of faults with displacements of 3–5 m, which provides an important structural guide for coalmine planning and production.

Quan-Ming et al. (2008) had considerable success in identifying many collapsed breccias before mining in certain northern Chinese

coalfields, from 3D seismic survey results, by studying seismic interval attributes. Collapsed breccia pipes are created by chimney subsidence of the overburden into dissolution voids in limestone or evaporates. The features are very serious safety hazards as they threaten rapid flooding underground if not detected in advance.

Yuan et al. (2011) developed new data processing techniques for 3D seismic surveys in areas of complex coal geology. In particular, the use of pre-stack Kirchhoff time migration has, in comparison with previous post-stack time migration, produced better images of coal seam reflections. These recent modern 3D seismic surveys in Chinese coalfield studies have helped to increase Chinese coal production significantly. In Australia, 3D seismic surveys have provided useful information on faulting and overall coal seam structure. Such surveys are not large, as underground mining only covers relatively small areas. However, 3D surveys are viewed as an expensive exploration tool, so cheaper options using high-frequency vibrators and accelerated weight drops are often preferred (Hatherly 2013). The detection of faults with small throws, as well as thin dykes and fractured zones, is still a challenge. The use of 3D seismic surveys with quality data and with modern interpretation software



**Figure 8.7** Seismic exploration showing a fault in section (upper) and time slice (lower). *Source:* From Zhao and Wu (2005).

have allowed small faults with throws of 3 m to be identified.

In the Bowen Basin, Australia, 3D seismic exploration has been effectively integrated into standard work practices. Typically, accurate delineation of structure is the primary objective for the 3D seismic survey. In addition, stratigraphic interpretation of the 3D seismic data has contributed to predicting roof and floor conditions that impact mining operations (Peters 2005). Zhou et al. (2017) pointed out that the frequency of seismic waves is too low, or the period and wavelength of the seismic waves is too long, to allow minor structural features to have a discrete expression on conventionally processed seismic reflection sections. Seismic interpreters sought to identify diffraction patterns as indicators of small faults and other discontinuities. In recent years, there has been renewed interest in diffraction imaging from the petroleum industry in the oil shale and coal seam gas sector. Diffraction has also been used for detection of tunnels, mines, and local heterogeneities, and it could have an important role in identification of faults, dykes, and other structures affecting coal seams (Zhou et al. 2017). Diffraction analysis should not be used as a sole indicator for the presence of faults and dykes on seismic reflection sections, but rather should be considered as one of the tools available to seismic interpreters in establishing underground structures.

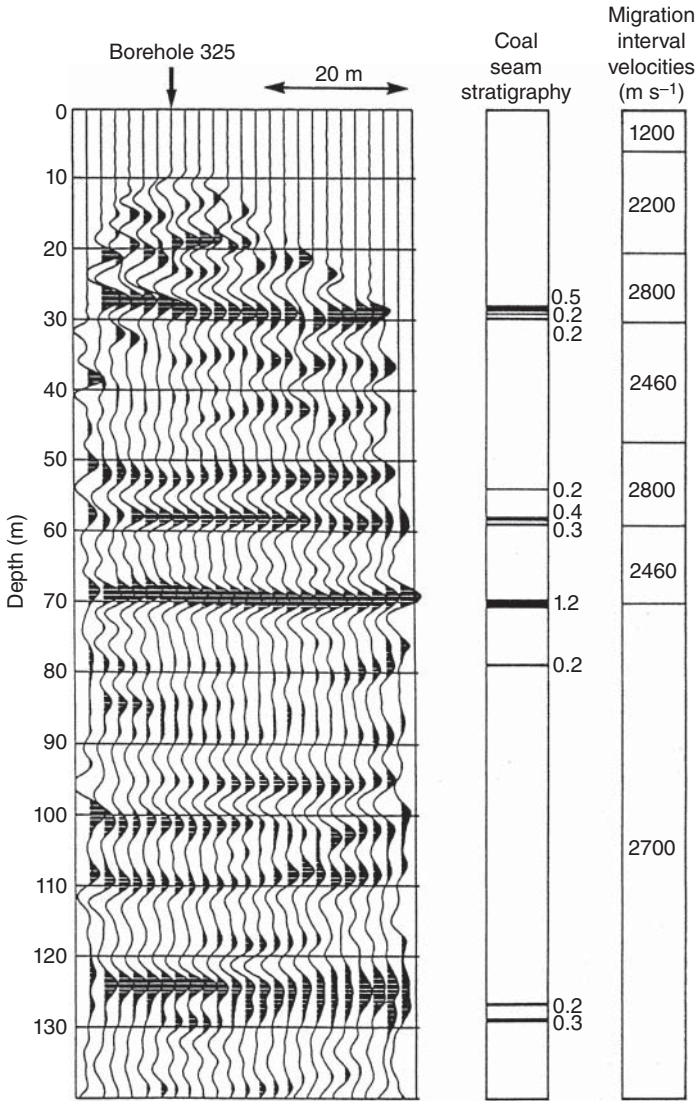
The dynamite shot hole method has been widely used, and although it gives higher frequency content and better resolution, such surveys are expensive and environmentally problematic. The cheaper Mini-SOSIE technique has been applied and can provide data of equal quality in the right circumstances. The method uses a similar recording spread as for dynamite, and the Mini-SOSIE whackers can put 800 to 1500 impulses into the ground for each record. Mini-SOSIE has been widely used, with particular success in Australia. However, difficulties have been encountered in using the technique in dissected hilly terrain combined with deep weathering profiles.

A number of vibrating sources have been developed to optimise target resolution and depth penetration. Vibroseis has been developed for deep imaging below 1 km but with the refinement to provide clear imaging of near-surface and smaller targets.

In the UK, seismic reflection techniques using an energy source situated at depth in a borehole have been developed for use in investigating opencast coal operations.

The hole-to-surface reflection data are processed with standard vertical seismic profiling (VSP) techniques. Small down-hole shots are fired at 2.0 m spacing below the water table in the borehole and geophones with a spacing of around 4.0 m are deployed at the surface along a line intersecting the top of the borehole. The travel time along a seismic ray path is independent of the direction of travel. Using this method, the rays are traversed in opposite directions from conventional reflection surveys and can be processed accordingly. A seismic depth section obtained from such a survey is shown in Figure 8.8, which shows the seismic depth section, coal seam stratigraphy, and the velocity profile used for migration. The coal seams at 30, 54, 58, 70, and 128 m all give a good reflection, but the thin seam at 80 m is hardly seen at all. Other reflections at 100 and 110 m may be weaker reflections from differing lithotypes in the sequence.

This technique has been used to correlate between boreholes, Figure 8.9 shows the stratigraphy and the combined seismic section from three hole-to-surface surveys with the migration interval velocities used in seismic data processing. The coal seam at 20 m has a good reflection, and the worked-out seam at 50 m has a very good reflection because the air-filled void in the old workings produces a very large reflection coefficient. This worked-out seam also shows two small faults with downthrows of 1–2 m to the right. The weaker reflections below 50 m are interpreted as a sandstone–mudstone interface at 70 m and a coal seam at 85 m; the fault in this seam is inferred from the borehole data.



**Figure 8.8** Seismic depth section obtained from the hole-to-surface survey in boreholes at a UK opencast site, with the coal seam stratigraphy and velocity profile used for migration. *Source:* From Kragh et al. (1991).

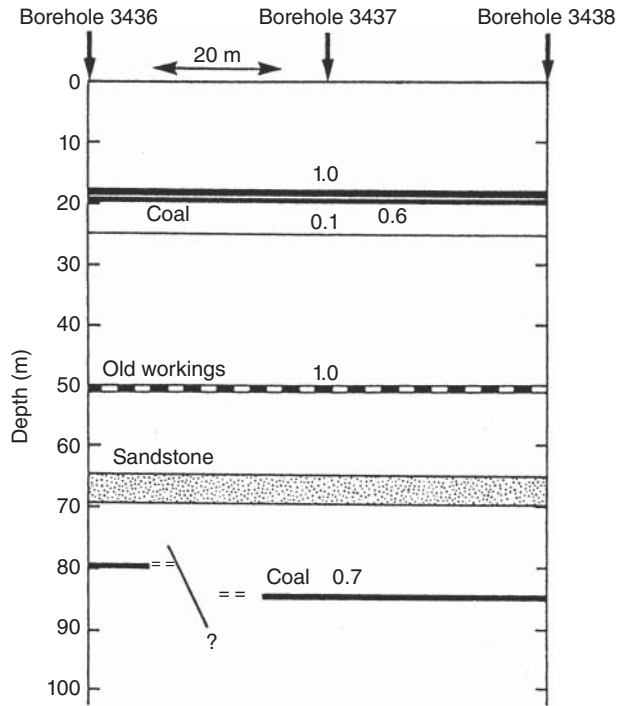
Cross-hole seismic reflection surveys are possible in certain opencast exploration sites that have closely spaced boreholes (Goultly et al. 1990; Kragh et al. 1991). This method involves down-hole seismic sources and detectors sited below the water table (hydrophones); this provides better resolution than surface seismic surveys and even VSP do, but it requires the availability of numerous boreholes.

These seismic methods are particularly useful in identifying old mine workings and

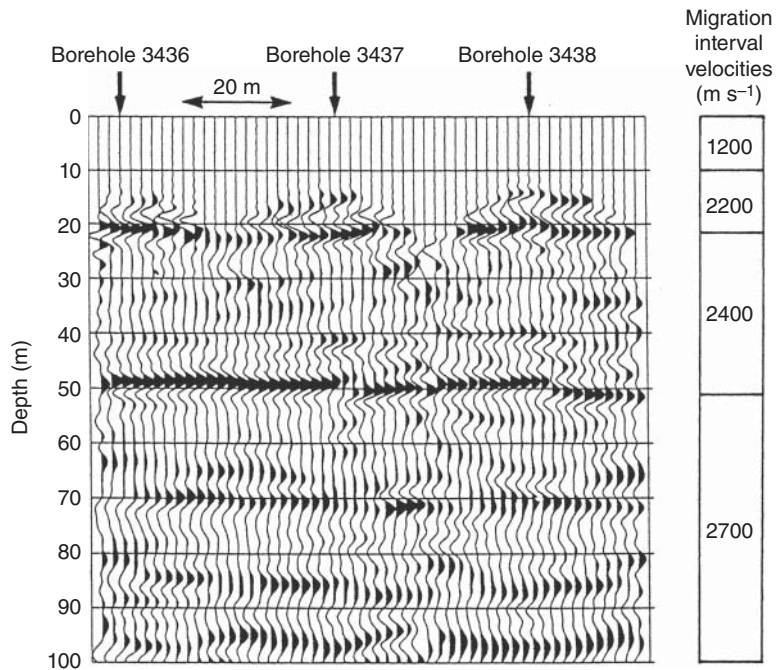
worked-out coal seams as well as illustrating the geology and structure of the area. In areas of the UK that have a long mining history, the position and extent of shallow underground workings have not always been recorded. In planning opencast operations, hole-to-surface seismic reflection surveys and cross-hole seismic reflection methods can help to identify such potential problems before detailed mine planning begins.

In underground mines, the planning of the mining of new reserve blocks of coal will lead

**Figure 8.9** (a) Coal seam stratigraphy for three boreholes at an opencast site, UK. (b) Combined seismic section from the three hole-to-surface surveys conducted in these boreholes with the velocity profile used for migration. The section is true scale, zero phase, and normal polarity, and an equal energy trace normalisation has been applied. *Source:* From Kragh et al. (1991).



(a)



(b)

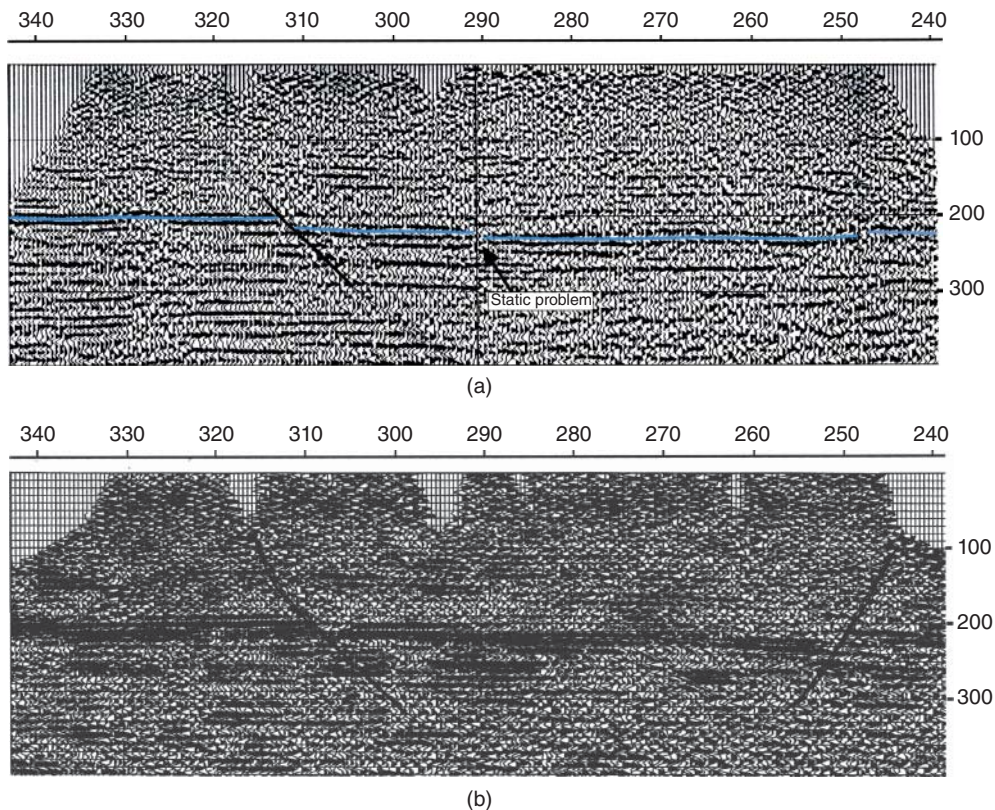


to a reassessment of the extent and quality of any existing seismic data. This will then result in either the requirement for additional high-resolution seismic data over the planned mine area or/and the reprocessing of existing seismic data using modern processing techniques. The reprocessing can yield an improvement of the seismic sections and often identifies structure previously undetected. This is particularly important where the geology and surface conditions vary considerably over a short distance. Figure 8.10 shows part of a seismic section in which the original processing is compared with reprocessing. The coal seam reflector is seen to be more continuous and identifiable. The static is removed and a medium-sized fault appears on the right-hand side of the section that was not detected on the original section (Carpenter and Robson 2000).

### 8.3.1.2 Seismic Refraction Surveys

The energy input to the ground must be stronger for refraction shooting; consequently, explosives continue to be the dominant energy source, although other sources are also used, such as falling weights for shallow studies. Refracted waves differ from reflected waves in that the principal portion of the refracted wave path is along an approximately horizontal interface between two rock layers before refraction to the detectors at the surface.

Refraction methods have been used in opencast coal exploration to locate previous workings (Gouly and Brabham 1984), to determine the variation in thickness of glacial deposits overlying coal-bearing sequences, and to locate faults at shallow depths. Refraction methods have limitations when applied to the location of subsurface positions of coal



**Figure 8.10** Reprocessed seismic data line showing stronger reflections and structure hitherto undetected. Source: From Carpenter and Robson (2000).

seams, but it is successful in locating previous workings because of the contrast in seismic velocity between backfilled mine workings and unworked areas.

### 8.3.1.3 Passive Seismic Surveys

Mining activity itself can induce seismic movements, especially along fault zones and in areas with a long mining history. In areas where this is an established phenomenon, seismic monitoring may be carried out. Mining-induced seismic events may be intense rockbursts or seismic activity induced by mining equipment and/or blasting. Kalab (1997) and Holub (1997) described such seismic activity in the Ostrava–Karvina Coalfield in the Czech Republic. Redmayne et al. (1998) describe similar seismic effects in the Midlothian Coalfield, UK.

Passive seismic studies by static seismometers involve comparing recorded observations over a period of time of only natural seismic activity, with no seismic energy added by the investigator.

The Polish Mining Institute has for over half a century monitored the Upper Silesian Coal Basin, which is one of the most seismically active in the world. Stec (2007) noted that over 50 000 tremors were recorded in this basin in the 30 years to 2004 and separated events of low and moderate magnitude associated with mining from regional higher magnitude events occurring in fault zones. These seismic activities create hazards such as rock outbursts and/or gas-and-rock outbursts in mine workings, as well as damaging surface structures. Styles et al. (1997) undertook surface and borehole microseismic monitoring of mining-induced seismicity in Britain and related fracturing around longwall mining faces with implications for roof control, subsidence, and gas migration. Gochioco (2008) recommended more passive seismic technologies for coalmine safety in US coalmines.

In addition to 3D seismic reflection surveys, the use of time-lapse seismic surveys, referred to as four-dimensional (4D) seismic surveys,

is being used increasingly to determine underground mining subsidence (Goultly 2008), and to determine underground gas migration and to identify geological conditions pertaining to hydrocarbon reservoirs and the suitability for CO<sub>2</sub> sequestration.

The surface detection and identification, from other background noises in mines, of microseismic signals from specific underground sources, such as hammering of trapped miners, water flow, and even small fault movement, would be a great advantage to miner safety and safe mine development. Shaw (2011) described the development of a new portable seismic instrument with this capability, stating that studies have been carried out over mines in the UK and the demonstration of the detection and identification of surface signals specifically from hammer pounding (to simulate trapped miners) at around 300 m below the surface in a working coalmine in West Virginia, USA.

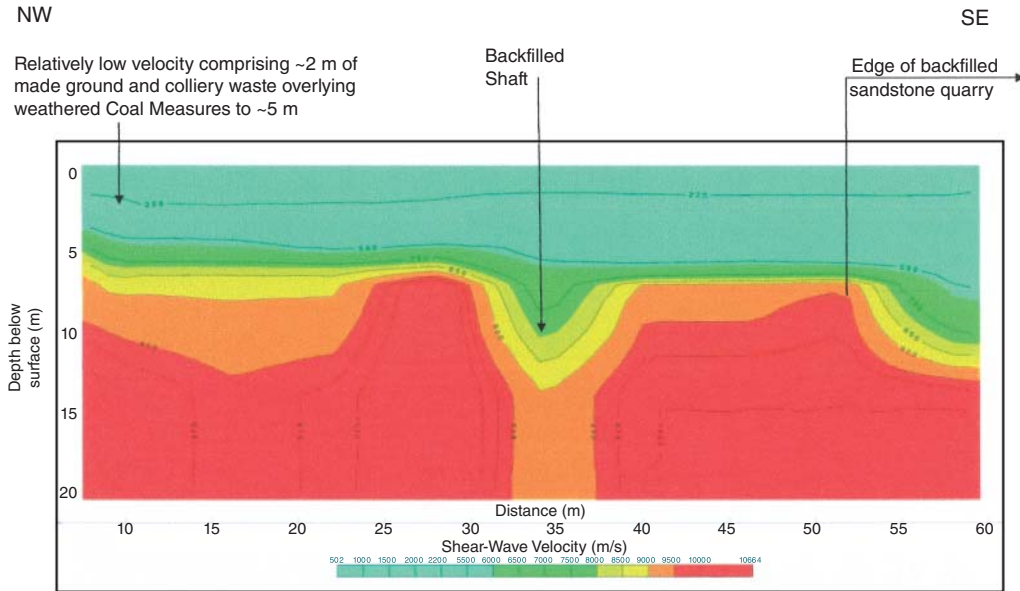
Passive refraction microtremor surveys utilise standard field seismic refraction recording equipment to record ambient background noise produced by microtremors caused by natural and anthropogenic activities. The production of 2D shear wave velocity sections has allowed the identification of reworked ground to a depth of 5 m below ground level, deeper strata at depths below 10 m, and backfilled mineshafts and quarries at depths below 7 m. Figure 8.11 shows a pseudo-section of contoured shear wave velocity data (Raines et al. 2011).

### 8.3.2 Gravity Surveys

The distribution of rock masses of different densities in the Earth's crust gives rise to local and regional variations in the Earth's gravitational field.

Gravity measurements are made using a gravimeter, taking readings at stations whose spacing may vary from 1 m to 20 km. The station interval is usually selected on the basis of assumed depth and size of the anomalous bodies sought.





**Figure 8.11** A 2D pseudo-section of contoured shear-wave velocity data showing made ground, a backfilled shaft, and backfilled quarry. *Source:* From Raines et al. (2011).

Areas with an anomalously high Bouguer gravity value (a positive anomaly) can indicate relatively dense rock, such as crystalline basement. A low Bouguer gravity value (a negative anomaly) is associated with the presence of less dense material, such as a thick succession of sedimentary strata. The magnitude and form of anomaly is related to the shape, orientation, and depth of the feature, together with the contrast in density between the different rock types involved.

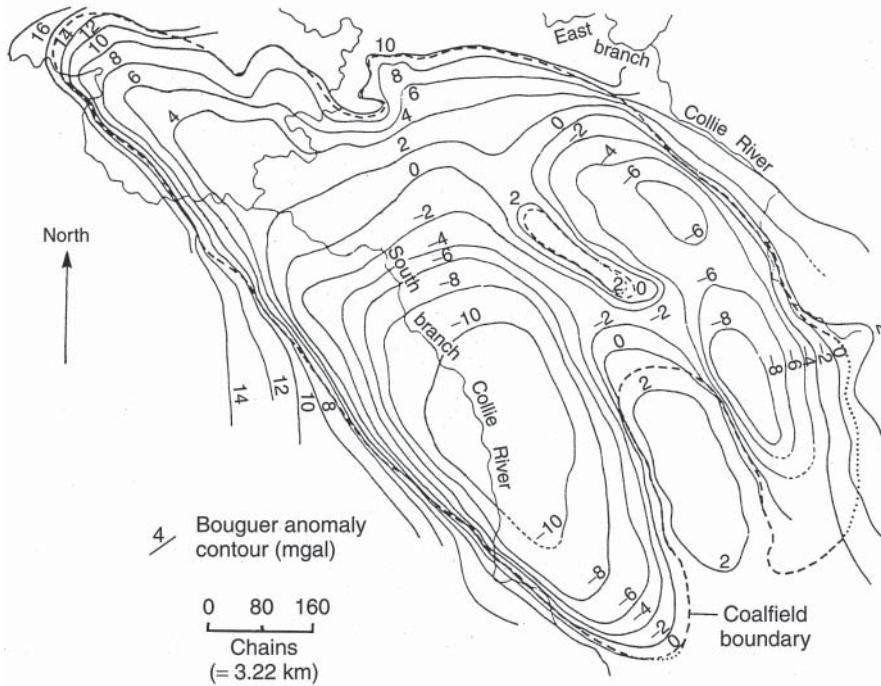
Gravity surveys are important, particularly on a regional scale, and are used in coalfield exploration both to detect the presence of sedimentary basins and to provide information on the overall structure of individual sedimentary basins. The results of a gravity survey are usually supported by additional geological data, such as density determinations on rock samples and field mapping results. Typical densities of coal and coal-bearing sediments, together with igneous and metamorphic rocks, are given in Table 8.1.

The major use of gravity surveys has been in the location of sedimentary basins that could

be coal bearing and which may be concealed by younger strata.

Those areas containing Gondwana coal-fields (i.e. Australia, South Africa, India, and Brazil) are especially suited to the use of gravity surveys. Most Gondwana sediments are preserved in basins lying directly on crystalline basement. These produce negative Bouguer anomalies that contrast with the surrounding positive values over areas of crystalline rocks.

In Western Australia, the Collie Coal Measures occur in a basin of Gondwana age, extending for about 180 km<sup>2</sup>. The basin is a remnant of a once extensive deposit preserved by downfaulting or folding in the Precambrian basement. The surrounding granites and the coal measures themselves are almost wholly covered by laterite and Pleistocene or Recent lacustrine deposits. Figure 8.12 shows the negative Bouguer gravity anomaly map of the Collie Coal Basin (Parasnis 1986). The principal feature is the Bouguer gravity anomaly low representing the less dense sedimentary coal-bearing sequence. The boundary of the coal sediments is indicated by a large Bouguer



**Figure 8.12** Bouguer anomaly map of the Collie Coalfield, Western Australia. *Source:* From Parasnis (1986).

gravity anomaly gradient where anomaly values increase as they pass across from the lighter sediments to the denser granite basement. The gravity survey indicates that the Collie Basin is divided into two main troughs, separated by a basement ridge extending from the south-east end, through the Bouguer gravity anomaly high to the northern boundary (Figure 8.12). A drilling programme has confirmed these results and discovered a new coal-bearing sequence covering approximately 25 km<sup>2</sup> in the eastern trough, containing a coal seam 10.0 m in thickness, which is now being mined.

In India and Bangladesh, similar coalfield basins have been identified. In north-west Bangladesh, Gondwana coal-bearing sediments are present in a series of small basins, now preserved as graben structures in the underlying crystalline basement, with the whole area concealed beneath Palaeogene–Neogene sediments. Gravity (and magnetic) surveying has identified these areas, and subsequent drilling has confirmed

the presence of thick Gondwana coals down to depths of 300 m.

Recent advances in airborne gravity instrumentation, accurate altitude, and precise position fixing have enabled more reliable gravity measurements to be made from aeroplanes (Studinger et al. 2008). Reliable regional gravity surveys can now be usefully made over large coalfields.

### 8.3.3 Magnetic Surveys

The magnetic properties of rocks may differ by several orders of magnitude rather than by a few tens of per cent. Typical values for coal and coal-bearing sediments, together with igneous and metamorphic rocks, are given in Table 8.1.

Coal-bearing sequences have relatively low magnetic susceptibilities, in contrast to the higher magnetic properties of basement igneous and metamorphic rocks. Magnetic surveys are used to delineate the broad structural framework of a coal-bearing area. Such surveys

do not detect coal, but they help in locating sedimentary sequences likely to contain coals at accessible depths (Evans and Greenwood 1988).

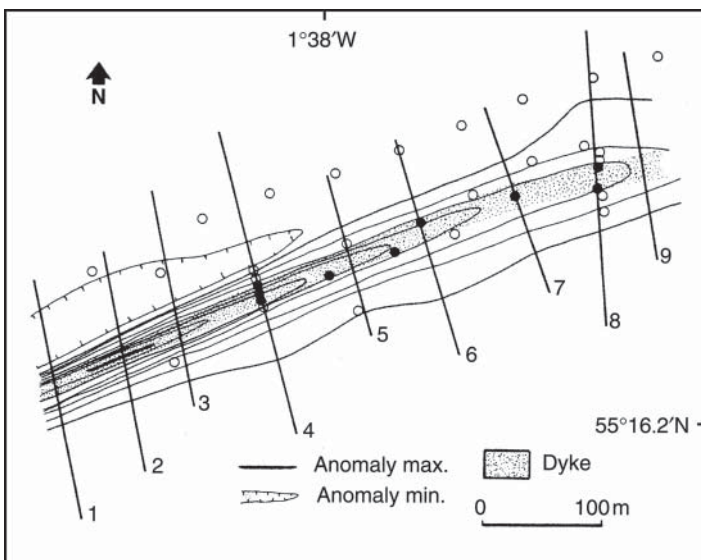
In north-west Bangladesh, aeromagnetic surveys have indicated in some areas that the depth to crystalline basement is less than 250 m (Busby and Evans 1988). The aeromagnetic survey together with subsequent drilling has delineated coal-bearing sediments preserved in graben structures in the basement. This has enabled those areas identified as accessible by mining to be targeted, and further drilling has identified sediments of Gondwana age containing a number of thick bituminous coals.

Such regional aeromagnetic survey results are often combined with regional gravity data to confirm the presence of sedimentary basins.

Distinct from large-scale aeromagnetic surveys, detailed ground magnetic surveys are used to locate the presence of basic igneous (dolerite) dykes in mine areas, and also to detect the limits of burnt coal seams.

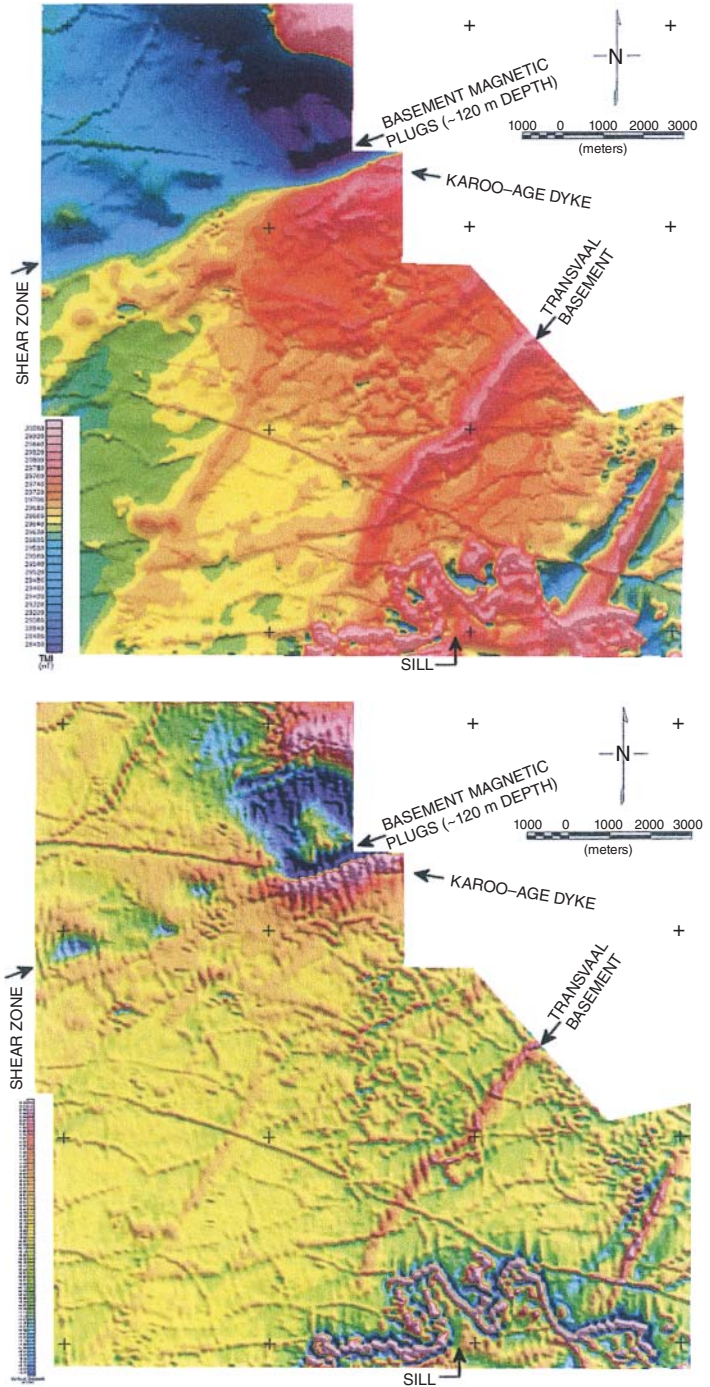
To locate dolerite dykes, a series of profiles are surveyed and plotted approximately perpendicular to the strike of each dyke and magnetic readings are taken every few metres. An example of a lenticular-shaped magnetic anomaly from Causey Park, north-east

England, showing the configuration of a dolerite dyke is shown in Figure 8.13 (Goulty et al. 1984). The anomaly contours are expressed in 100 nT intervals, a nanotesla being a unit of magnetic field strength. The location of dykes of significant size is important, particularly in opencast mining, coals that are in contact or close proximity to dykes will have undergone devolatilisation due to baking and, therefore, many have deteriorated (or in some instances improved) in rank and quality. This phenomenon is particularly important in South Africa, where dyke and sill swarms are intruded into the coal-bearing Karoo sediments. Undetected dykes cause problems in underground mining by making tunnelling through them difficult and expensive and by affecting coal quality. High-resolution aeromagnetic mapping and interpretation by modern image processing, and other data enhancement techniques, together with palaeomagnetic studies have enabled magnetic dykes and weathered basic intrusions to be confidently predicted at depth in underground coal workings in Mpumalanga (formerly the Eastern Transvaal). Figure 8.14 shows a high-resolution magnetic survey over an Eastern Transvaal coalfield, highlighting Karoo-age dykes and sills and also



**Figure 8.13** Magnetic anomaly map for part of the Causey Park dyke, UK. Contours are at 100 nT intervals. Boreholes that encountered dolerite at rockhead are shown as solid circles. *Source:* From Goulty et al. (1984).

**Figure 8.14** High-resolution aeromagnetic survey over an Eastern Transvaal coalfield, mapping outcropping Karoo-age dykes and sills and Precambrian basement features: (a) total magnetic field image; (b) vertical gradient.





earlier basement features (Campbell 2006). Aeromagnetic surveys can be more effective than ground surveys, particularly if there are magnetic surface boulders and cultural magnetic interference, as the airborne survey can choose the most appropriate survey height and spacing for deep dyke detection and minimise surface magnetic effects.

In the delineation of burnt zones in coal seams, the magnetic susceptibility of unbaked sedimentary rocks is quite low, whereas the magnetic susceptibility of the baked rocks is variable (Hooper 1987). Most of the maghaematite and magnetite in the baked rocks is derived from the thermal alteration of sedimentary minerals. Some baked areas have undergone iron enrichment, as iron is mobile during thermal metamorphism and can be redeposited in the baked rocks. On heating, shales and siltstones undergo significant reductions in volume; however, shales tend to separate into small pieces, so exposing a greater surface area available for iron enrichment. The sediments around the edges of a burnt seam may contain appreciably more magnetite if the coal fire is extinguished due to the lack of oxygen, which reduces more iron oxides and hydroxides to magnetite. In these cases, larger magnetic anomalies may be expected along the margins of the baked zones. Figure 8.15 shows a magnetic profile over a burnt coal zone in East Kalimantan, Indonesia; the zone of burnt coal is some 160 m wide, extending in from the outcrop. The magnetic profile shows a distinct magnetic anomaly of over 1000 nT amplitude when passing across the burnt mudstones.

Similar anomalies produced by baked sediments over burnt coal seams in the Southland District of New Zealand are shown in Figure 8.16 (Lindqvist et al. 1985).

By carrying out a series of traverses perpendicular to the strike of the coal, a zone of burnt coal can be accurately determined and the loss of coal within the area of investigation can be calculated.

### 8.3.4 Electrical Methods

#### 8.3.4.1 Electrical Resistivity Methods

Resistivity values for coal and coal-bearing sediments are given in Table 8.1.

Electrical resistivity mapping is primarily used for detecting local, relatively shallow inhomogeneities and is employed typically in delineating geological boundaries, fractures, and cavities. Coal resistivity decreases exponentially with increasing pressure and increasing temperature, so that the sensitivity of coal resistivity to the confining pressure is reduced when the temperature is higher (Wang et al. 2016).

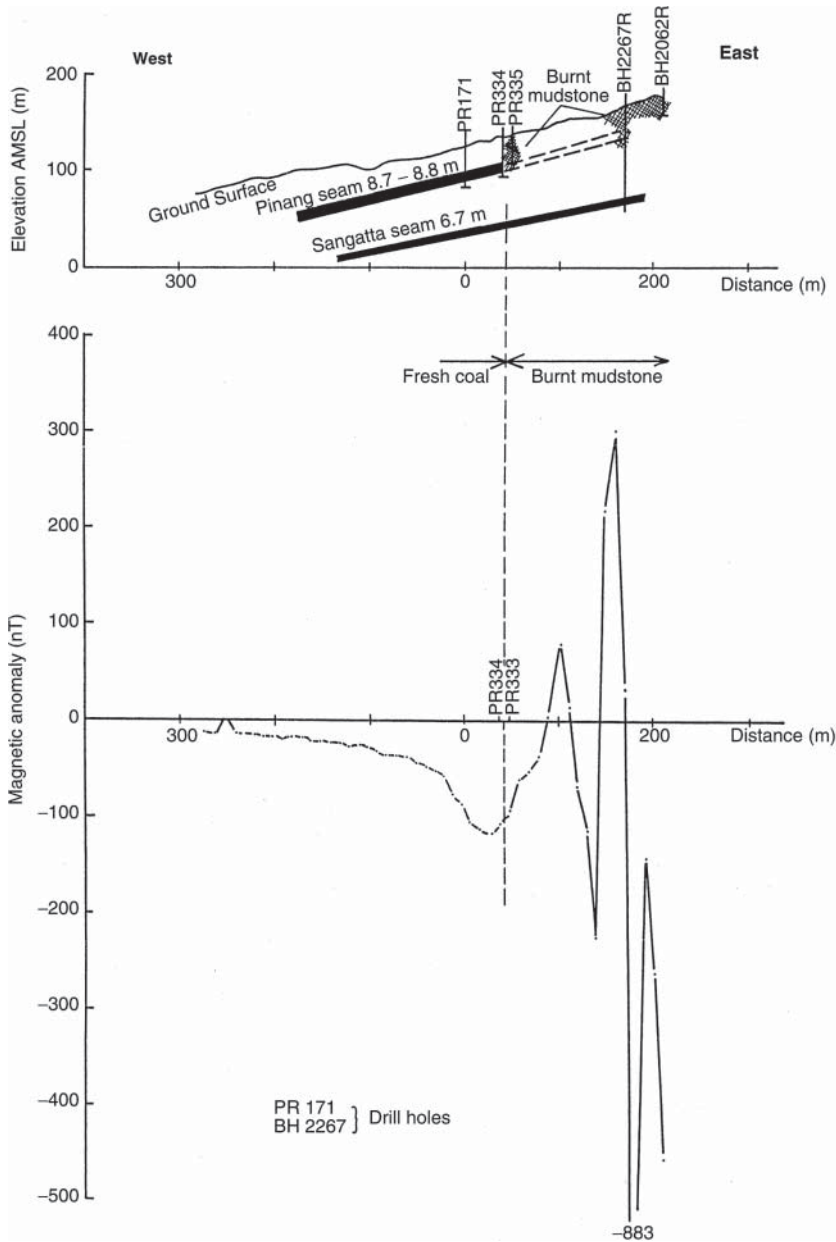
In the Raniganj Coalfield, India, a combination of resistivity depth sounding and surface electrical mapping has been used to identify lithological contacts and the presence of faults and dykes under tropical weathering conditions.

Electrical resistivity to locate coal seams has also been applied to the Pennsylvanian anthracite field and in Wyoming, USA, where it has detected the splitting of a coal seam into two, separated by a sand body; it has also been used in Australia. Electrical resistivity has been used to locate old, concealed mine shafts and other mining cavities in the UK. Multielectrode resistivity measurements have been used in the USA to map shallow coalmine workings (Johnson 2003), and electrical resistivity tomography has been used to delineate mine galleries in the Raniganj Coalfield, India (Nath and Chakraborty 2004).

#### 8.3.4.2 Ground-Penetrating Radar Methods

Ground-penetrating radar (GPR; also known as ground-probing radar) methods have proliferated over the past decade as more advanced instruments and 3D surveys have been employed. The new GPR systems have been able to detect features at depths of tens of metres deeper than previously possible. Franke and Utsi (2009) identified a coal seam at greater than 30 m depth which was detected, as well as voids, in an East Asian coal sequence. GPR has





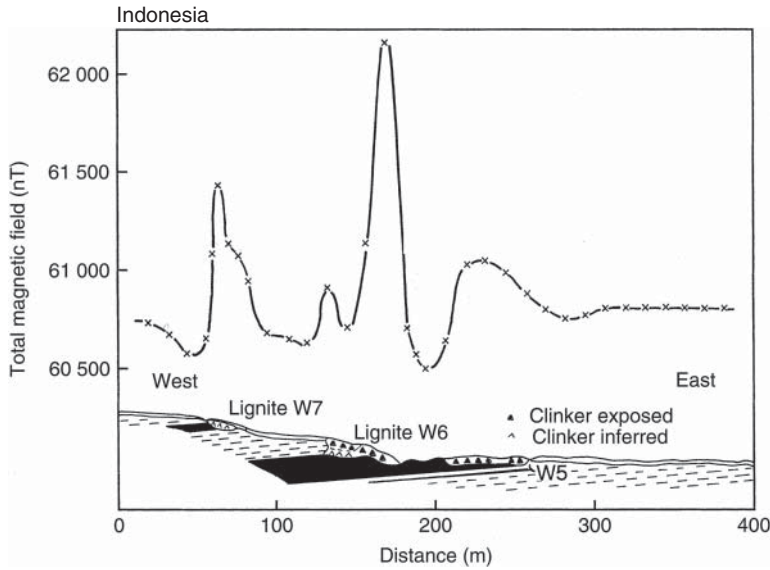
**Figure 8.15** Magnetic profile over burnt coal and geological cross-section, East Kalimantan, Indonesia.

yet to become a significant geophysical tool for use in coal mining.

#### 8.3.4.3 Electromagnetic Surveys

In the USA, helicopter-borne and ground EM surveys have been useful in the studies

associated with coal-bed methane in the Powder River Basin in Wyoming (Lipinski et al. 2008) and using thermal infrared imagery and helicopter EM conductivity to examine the abandoned underground coal mining workings in Pennsylvania (Love et al. 2005).



**Figure 8.16** Total magnetic field profile and geological cross-section, southern end of New Vale Coal Co., Eastern Southland, New Zealand. *Source:* From Lindqvist et al. (1985).

### 8.3.5 Radioactive Methods

In coalfields, radioactive surveys can be used to trace marine shales that are high in radioactivity. This can be useful as an indirect method of mapping coal seams when the position of the radioactive shale is known to occur in the vertical sequence, and also its position in relation to known coal seams.

Topographic irregularities, dispersion of radioactive materials due to weathering, and 'background radiation' affect instrument readings.

## 8.4 Underground Geophysical Methods

The use of geophysical techniques underground encounters difficulties. Space restrictions and safety requirements, particularly in the use of some electrical equipment, limit the use of certain geophysical methods underground. Nevertheless, ISS and pulse radar methods have been used underground, gravity measurements have been made in coalmine galleries to detect faults and voids in underground coalmines, provided appropriate bulk density values are obtained and the gallery corrections are applied (Casten and Gram 1989)

### 8.4.1 In-Seam Seismic Surveys

These surveys involve the use of channel waves propagating in the coal seam to detect discontinuities in advance of mining. ISS surveys use seismic waves that travel parallel to the bedding planes of sedimentary strata. They are restricted to travelling along beds within which the seismic velocities are lower than in the stratigraphic units above and below; and since they travel with the bulk of their energy confined to the low seismic velocity layer, they are referred to as 'channel' waves. Coal seams are excellent mediums for channel waves as they invariably have lower seismic velocities than the sediments that surround them. Table 8.1 shows the seismic velocity of coal to be about half that of sandstone and limestone. The targets for ISS surveys are reflections from obstructions within the wave channel, namely faults, dykes, and washouts. Such discontinuities are of vital importance to the economics of longwall mining.

An explosive source is normally used for seismic surveys, but for underground operations alternative sources are required, so mechanical devices such as Vibroseis, Mini-SOSIE, and land airgun techniques may be used. In the UK, a pneumatic piston impactor has

been tested for this purpose in mines where the use of roof and rib bolting as a roadway support acts as anchor points for the geophones. Additional tests have used surface seismic sources to indirectly produce channel waves propagation in underground coal seams in order to identify faults. These tests have shown some success in locating such seam discontinuities.

As the seismic recorder is not flameproof, it is located either on the surface or in an intake roadway in which methane concentration is less than 0.25%. In Germany, a flameproof digital recording unit has been developed that will help to overcome this problem.

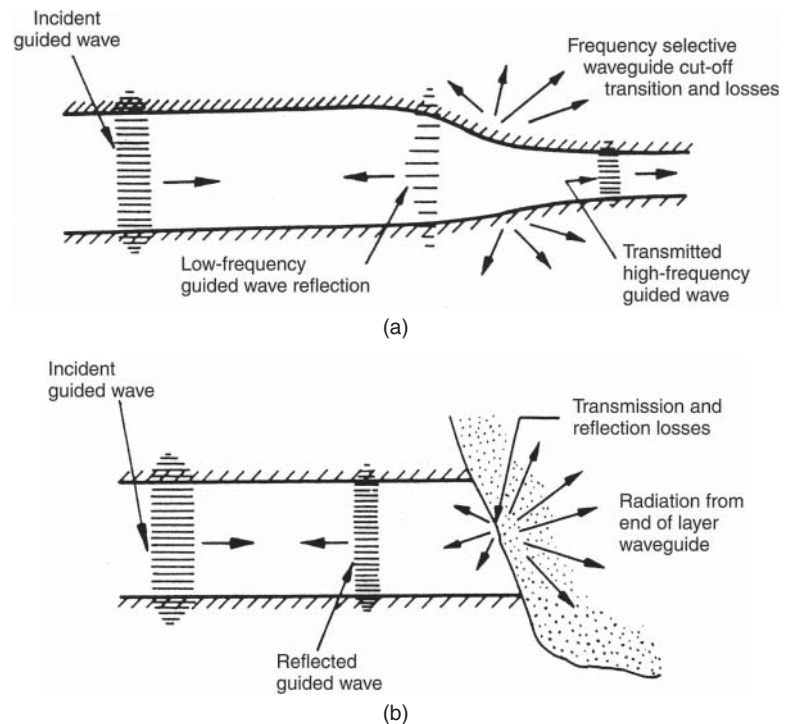
The ISS method can be used as shown in Figure 8.17 (Jackson 1981), which illustrates the behaviour of channel waves reflected from or transmitted through a variety of discontinuities. Depending upon the target structure, shot holes and receivers can be located in the same or different roadways. The recorded seismic data are then processed in

order to construct a map of the distribution of faults or washouts in the coal seam.

Gochioco (2000, 2004) described a 3D seismic survey conducted in advance of coalmine development in the Illinois Basin to better define a geological structure with the potential to adversely affect longwall mining conditions. The 3D seismic data indicated an abrupt change in coal seam elevation referred to as a roll. This was seen to trend south and then south-east in the coal reserve area. Figure 8.18 shows four in-line seismic sections 240 ft (c. 73 m) apart; these indicate that the steep western flank of the roll decreases as the roll trends to the south-east (Gochioco 2004).

The ability to detect small-scale faulting by ISS surveys is dependent on the particular attenuation characteristics of the coal seam concerned. ISS surveys can detect faults that are below the resolution of surface seismic surveys, and can improve the positioning of faults interpreted from the surface seismic surveys, both of which have economic significance,

**Figure 8.17** Conceptual behaviour of channel waves on encountering (a) a coal seam pinch-out, (b) a channel sand cut-out, (c) a fault with a throw less than the seam thickness, and (d) a fault with a throw greater than the seam thickness. *Source:* From Jackson (1981). Reproduced with permission of IEA Coal Research.



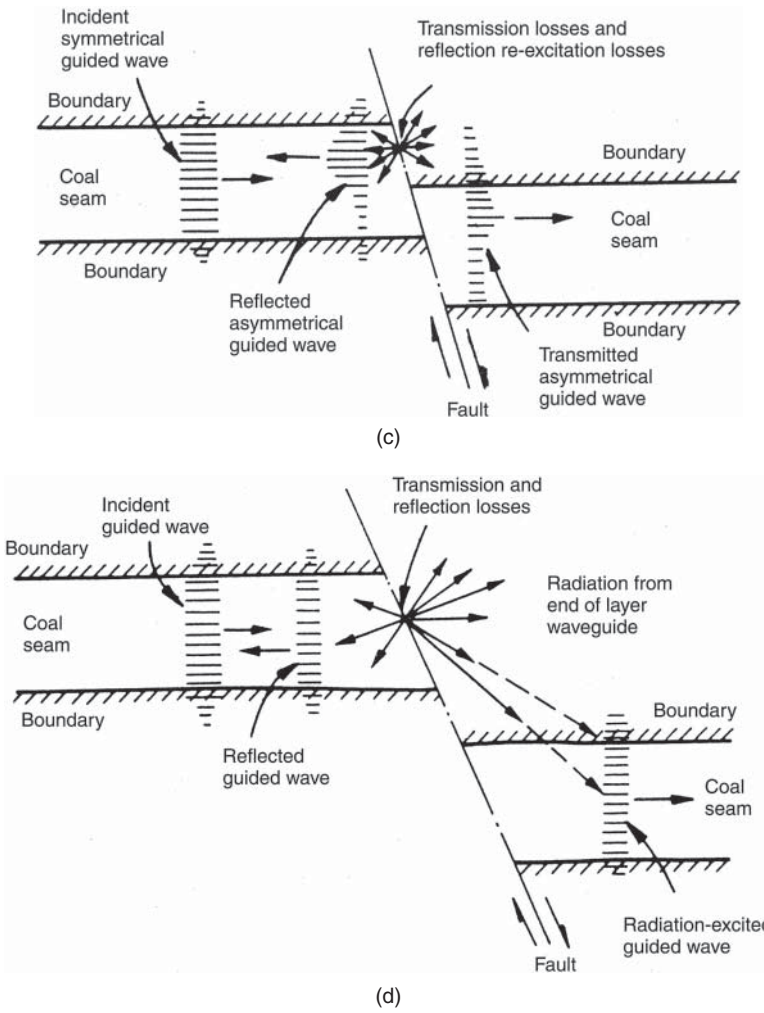
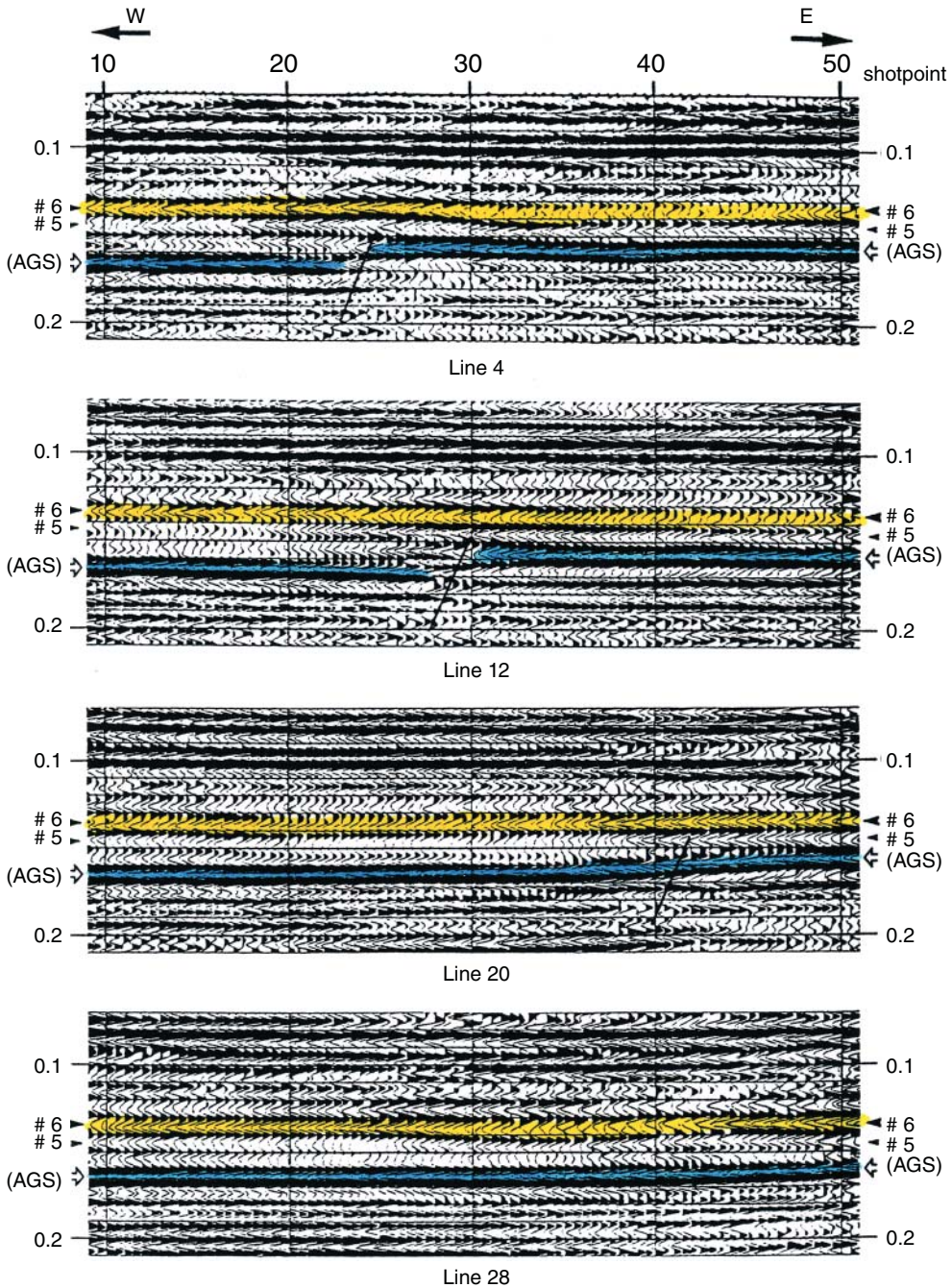


Figure 8.17 (Continued)

and faults that disrupt the coal seam can be detected within 200–300 m. Enhancement of the ISS technique is particularly useful where longwall mining techniques are employed. Where advancing longwall faces are used, it is essential to have prior knowledge of the nature of the coal seam through which the advance will be made. The high financial investment in establishing longwall faces is at

risk due to loss of production if structurally affected ground is encountered. The prediction of faults in a panel of coal is important in maintaining the lifetime of existing longwall faces. Surveys from the face carried out at regular intervals, dictated by the rate of advance, can give early warning of impending dangers. For example, a longwall advance face was expected to terminate due to the constraint





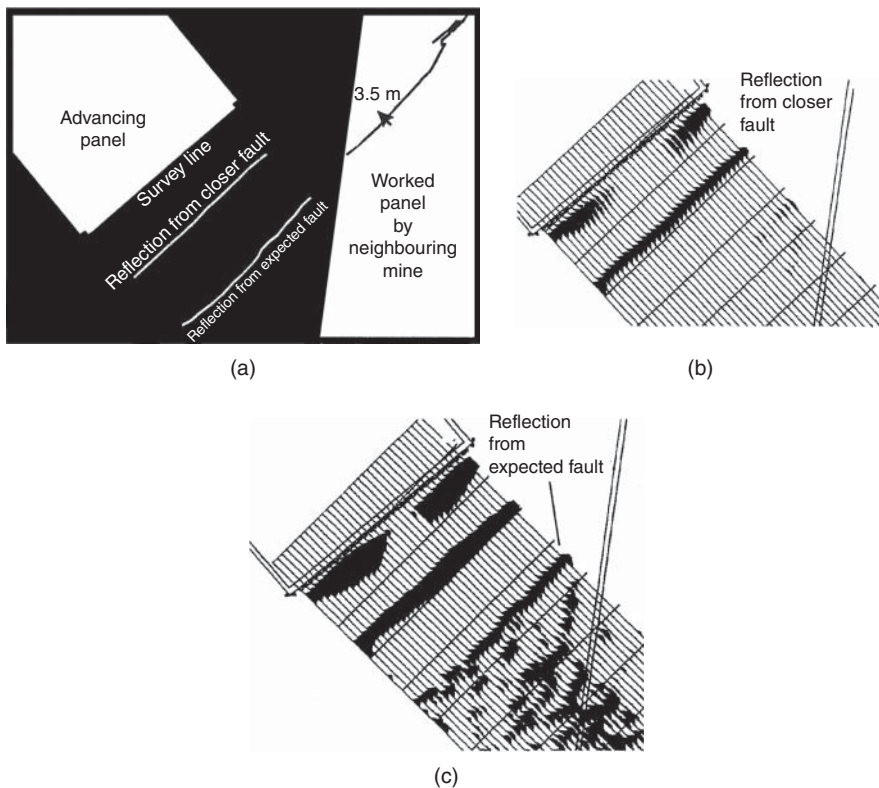
**Figure 8.18** Four line seismic sections, 240 ft (c. 73 m) apart, showing roll feature trending to the south-east in the mine reserve area. *Source:* From Gochioco (2004).



of a 3.5 m fault encountered in a neighbouring worked-out panel (Figure 8.19a). An ISS reflection survey was undertaken to locate the exact position of the fault before the face came into contact with it. The ISS survey located the 3.5 m fault successfully, but also imaged a closer fault that was hitherto undetected (Figure 8.19b and c). On this evidence, the face was stopped short to avoid damage to working equipment.

A microseismic monitoring system was established in the Xin Zhuangzi mine, Huainan Province, People's Republic of China. This system monitored the fault activation process and development trend in mining excavation activities. It was observed that the fault activation builds up and the outburst danger increases as the working face approached a

known fault. Once excavation ceases, fault activation tends to be stable. Such monitoring is considered accurate in terms of detecting tectonic activities and enables effective countermeasures to be taken to prevent and reduce outburst events (Liu et al. 2017). Another example of microseismic monitoring is from the Pranhita–Godavari Coal Basin, India, where three mine entries were planned for the development of a new underground mine from the existing open-pit highwall. Vinoth and Kumar (2014) employed a microseismic monitoring system to understand the response of rock mass to disturbance from underground mining involving blasting and use of continuous miners, and to monitor slope stabilities in the overlying open-pit mine. In this instance, no serious stability issues have occurred along



**Figure 8.19** ISS reflection survey used to detect faulting in advance of longwall mining. (a) Position of longwall panel indicating interpreted faulting in advance of mining. (b) Reflection pattern from fault close to longwall panel face. (c) Reflection from expected fault some distance from longwall panel face. *Source:* Reproduced with permission of IMC Geophysics Ltd (1997).

the highwall slope from seismic events generated by underground activity. The rock mass was found to be stable during the entire monitoring period and no sign of failure was observed.

### 8.4.2 Underground Gravity Surveys

Underground gravity surveys have been made in coalmine galleries to detect faults and voids in underground coalmines, with the provision that appropriate bulk density values are obtained and the gallery corrections (i.e. free air correction, Bouguer, and terrain corrections) are applied (Casten and Gram 1989)

### 8.4.3 Ground-Penetrating Radar Techniques

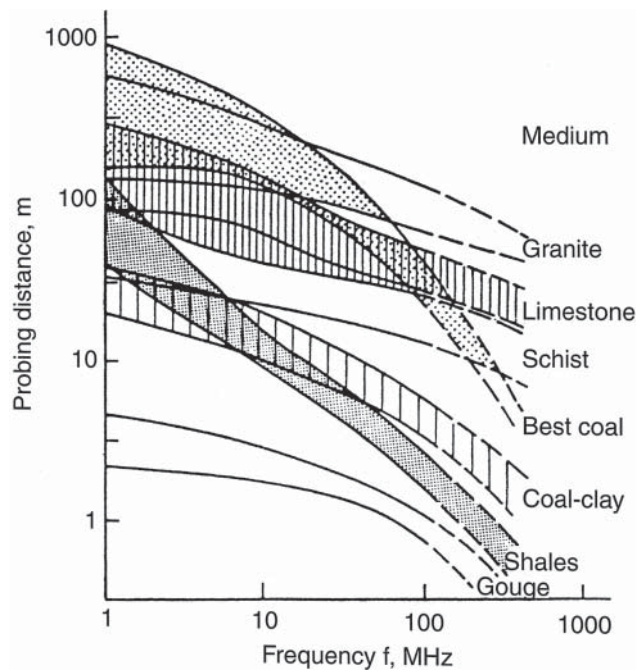
GPR or pulse radar work in underground mine roadways is used to detect hazards in advance of mining, to measure the thickness of the coal layer remaining in the roof of mine roadways, and to detect geological discontinuities in the coal panel to be worked. This is important in longwall face operations in controlling the

cutting position of the continuous coal-cutting machines, and for safety reasons. Pulse radar methods have been employed in underground workings in the UK and USA to determine coal thickness. In the USA, pulse radar has been used to determine mine roof stratigraphy, enabling clay and shale layers within the coal to be identified. Cook (1975) identified the relationship of probing distance as a function of different frequencies for a variety of geological lithologies and showed that good coals have a better penetration distance than for other lithologies. Figure 8.20 shows this relationship as illustrated in Reynolds (2011), adapted from Cook (1975). This will ensure that GPR will continue to be used in underground coal mining operations to detect coal seam discontinuities such as faults.

## 8.5 Geophysical Borehole Logging

Geophysical logging (geologs) is the measurement of the variation with depth of particular physical properties of surrounding rocks with

**Figure 8.20** Probing distances as a function of frequency for different geological lithologies. *Source:* From Reynolds (2011) based on Cook (1975).





**Figure 8.21** Mobile geophysical logging unit. *Source:* Reproduced with permission of Robertson Geologging Ltd.

geophysical measuring tools (sondes) located in boreholes. Measurements are made by lowering a sonde attached to the end of a cable to the bottom of the borehole and then raising the sonde back out of the borehole at a constant rate to record the geolog. It is easier to maintain a constant rate by raising rather than lowering the sonde, which is important for data quality. Figure 8.21 shows a logging unit in operation.

The sonde is connected by electrical conductors within the cable to recording and control instrumentation situated on the surface. This instrumentation is referred to as the logging unit, which also contains the powered winch used to lower and raise the sonde. In coal exploration, such units are small and portable. The logging unit makes a permanent record of the log data on a paper chart and on magnetic tape or disc, the latter being suitable for future computer analysis.

The objective of geophysical logging is to determine in situ the rock type and other properties, such as porosity, fluid content, and ash content, which may characterise the sedimentary lithotypes and igneous rocks intersected in boreholes.

All exploration and development drilling programmes will have a logging unit ready to log cored and open-holes within the area of interest. The depth and thickness estimates on the drilling lithological logs will be reconciled with the corrected depths recorded on the geolog. All features of interest on the logs will be used to site additional exploratory boreholes, and to site additional boreholes alongside selected logged boreholes (pilot holes) in order to take coal cores for quality analysis.

In coal exploration and mining, it is necessary to measure and identify one or more of the physical properties of the coal-bearing sequence. The appropriate geologs are selected to obtain the required geological information. The geological information sought includes: the identification of coal; the identification of depths to coal seam roof and floor, and the coal seam thickness; the identification of partings within the coal seams, and quality variations within the seam; the determination of geological features, such as faulting, jointing, washouts, and thick sandstones and igneous intrusions; and

the determination of hydrogeological and geotechnical characteristics.

Once this information has been obtained, it can be built into the geological database, so that assessment of coal quality and geotechnical properties can be made, together with coal resource/reserves calculations with geological losses (see Chapter 7). This information will then be incorporated into the mine planning programme.

Geolog interpretation is essentially a three-phase exercise: (i) log calibration, converting the measured log units into either standardised log units or recognised physical properties; (ii) basic interpretation, locating and measuring the bed boundaries, the depths and thicknesses, and an average value of log units for the formation; (iii) log analysis, relating the standard log units or physical properties to formation characteristics.

The logs most useful and most used to identify coal and coal-bearing sequences are gamma ray, density, neutron, calliper, sonic, and resistivity. Of these, the first two, gamma ray and density, are usually sufficient to identify coal horizons, and other common lithotypes in coal-bearing sequences. Additional logs are used to identify the structural attitude of coal-bearing strata and their inherent stress field orientation. These are dipmeter and acoustic scanning and image processing techniques.

### 8.5.1 Radiation Logs

Gamma ray, density, and neutron logs measure nuclear radiation emitted from naturally occurring sources within geological formations or emitted from sources carried on the logging tool. Unlike electric logs, radiation will work in the absence of a borehole fluid (air-filled boreholes) and through casing.

In coal exploration, where boreholes tend to be shallow (less than 350 m), narrow, and often dry, with walls in poor condition, radiation logs are often the only tools available for coal identification.

#### 8.5.1.1 Gamma-Ray Log

The gamma-ray log measures the naturally occurring radiation in geological formations. The principal source of radioactivity in rocks is usually the potassium-40 isotope associated with clay minerals, and therefore found more abundantly in mudstones and clay-rich siltstones.

Conversely, good-quality coals and clean sandstones have a very low level of natural radiation. As the amount of clay material included increases, in the form of clay partings in coal and as clay clasts and clay matrix in sandstones, so the natural radiation increases. In the case of marine mudstones, higher levels of potassium, together with other radioactive isotopes in the form of uranium and thorium, may be preserved. This causes the natural radiation levels to be much higher than in the more typical non-marine mudstones.

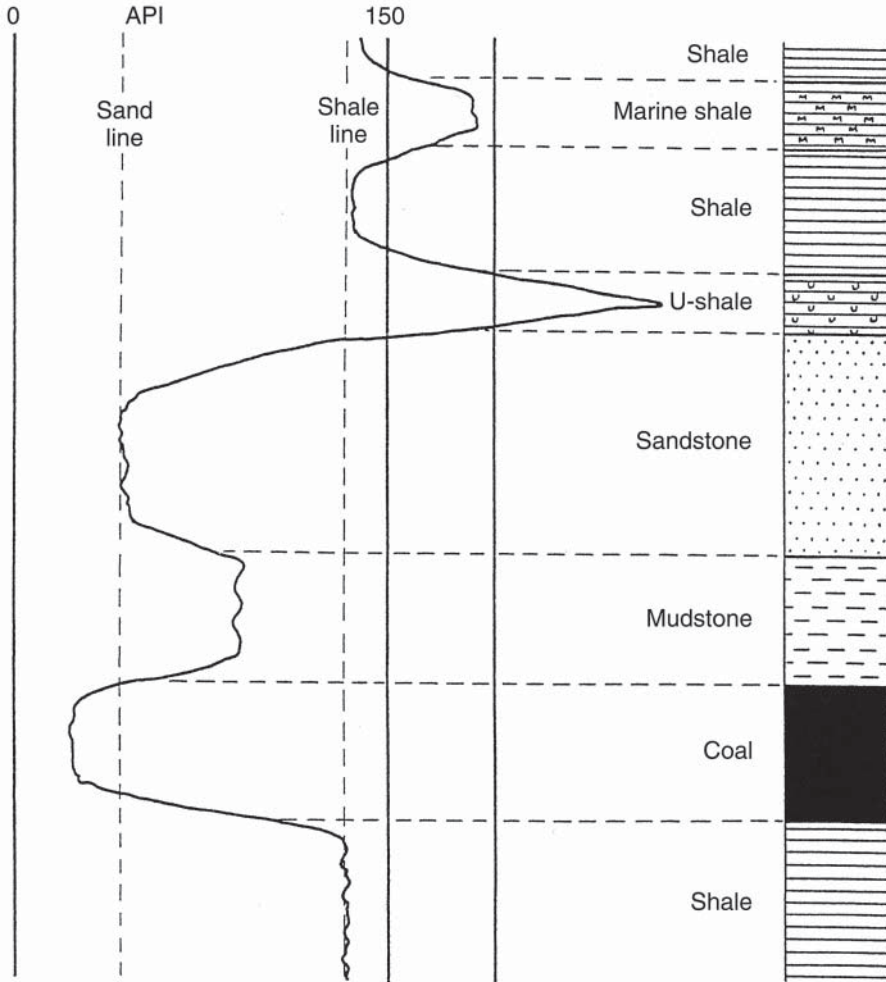
The occurrence of horizons exhibiting high levels of radiation is extremely useful for correlation purposes; this is also the case for very low radiation levels in clean coal. Figure 8.22 shows the relationship of the gamma rays to selected lithotypes found in coal-bearing sequences.

The use of natural radiation to determine the ash content of coals is an unreliable technique, as coals with the same ash content may emit differing amounts of natural radiation, due to the make-up of the mineral content of the ash fraction in the coal.

The gamma-ray log is not wholly diagnostic, and in normal practice it is used in conjunction with other geophysical logs in order to fully distinguish formations.

Gamma-ray logs can be displayed as counts per second, but they are often calibrated according to the American Petroleum Institute (API) Standards pit in the USA and adjusted so that the log gives values expressed in API units.

Gamma-ray logs have relatively poor vertical resolution, as the gamma-ray tool 'senses or sees' a fairly large area (up to 40 cm vertically). As adsorption increases as density increases, the depth of investigation becomes lower in



**Figure 8.22** Coal-bearing sequence lithotypes and gamma-ray log response. *Source:* Reproduced with permission of Reeves Oilfield Services Ltd.

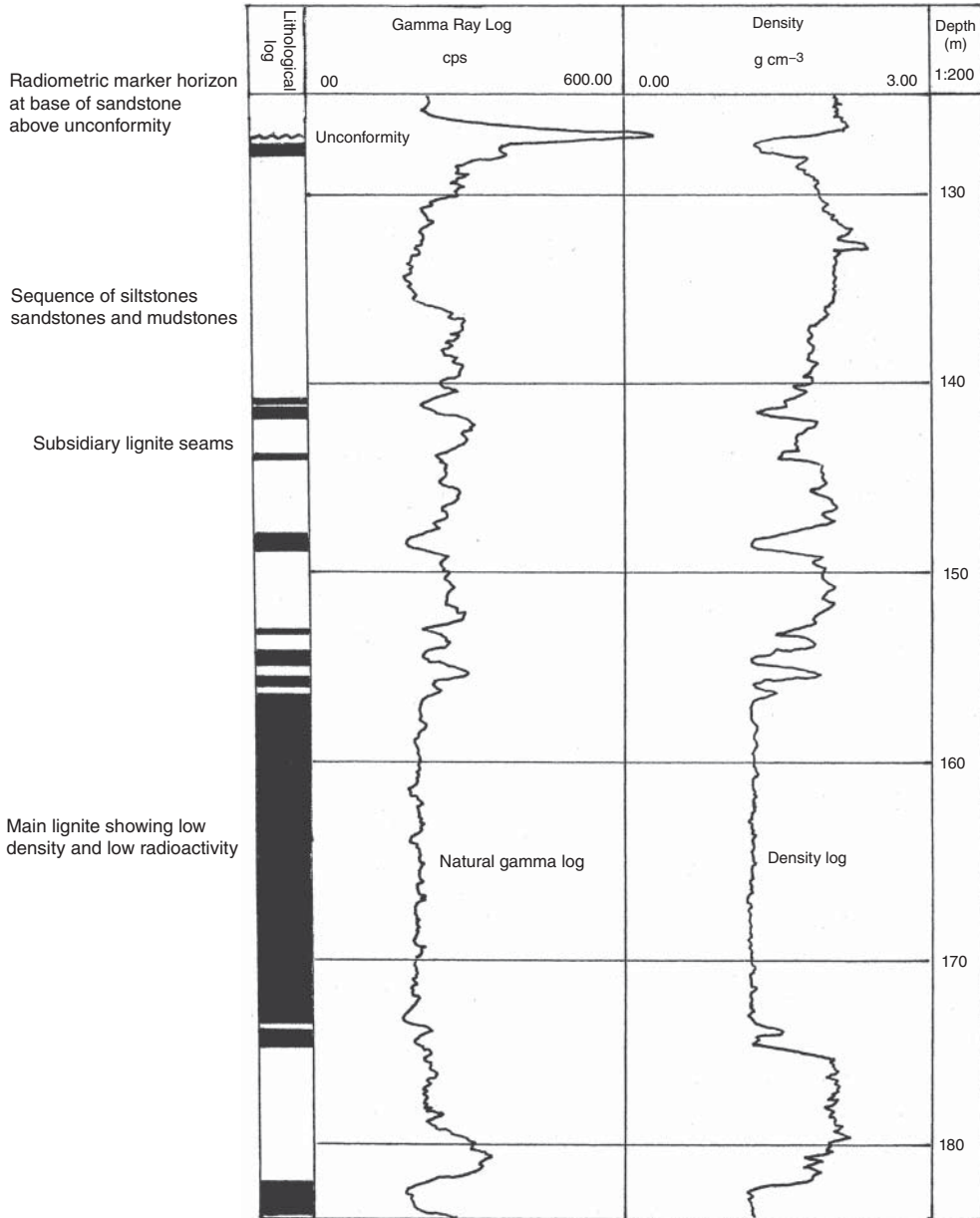
high-density formations, such as basic igneous rocks. As a guide, coal seam thickness can be interpreted by taking the point on the gamma-ray curve one-third down from the base of a typical mudstone. Such interpretations are asymmetrical, as gamma rays travel further in the less dense coal medium.

Gamma-ray performance is not impaired by borehole caving or loss of borehole fluids, as air, water, and mud are not high absorbers of gamma rays. In addition, gamma rays can be run through casing, as casing tends only to

attenuate the radiation received; the shape of the log is preserved, although the base level is altered. For accuracy, however, an adjustment to allow for casing has to be made.

As illustrated in Figure 8.22, the trace of the gamma-ray log identifies a sharp geological boundary between two formations as a curve with a vertical height equal to the average vertical resolution of the gamma-ray detector and a horizontal length equal to the difference between the gamma-ray count of the formations either side of the geological boundary.





**Figure 8.23** Section showing gamma, density, and resistivity logs over a principal lignite seam, Thar Coalfield, Pakistan. *Source:* Reproduced with permission of Oracle Coalfields plc.

Figure 8.23 shows the relationship of the gamma-ray log to lithology: the lignite has low radioactivity whereas a sand containing a high percentage of detrital orthoclase feldspar rich in potassium has a high radioactivity.

### 8.5.1.2 Density Log

In coal exploration, the density log is used as a principal means of identifying coal, principally because coal has a uniquely low density compared with the rest of the coal-bearing

sequence (see Table 8.1). In certain circumstances, there is the additional benefit of an approximate linear relationship between ash content and density for a given coal seam in a given area.

In the density sonde, two detectors are used to measure gamma rays passed into the formation from the source and reflected to the detector by scattering. The long and short spacing logs are shielded from direct radiation from the source used and measure the gamma rays that have been reflected, or backscattered, from the rock.

Induced gamma rays in the energy range used for density logging are usually scattered in a forward direction. This means that density logs only respond to the formations between the source and the detector. The gamma rays can be considered to have 'diffused' through the material between the source and detector, and the density logs will be equally affected by all this material. The vertical resolution of the density logs is thus approximately equal to the spacing of the source and detector.

The density log is calibrated by measuring sonde output in homogeneous blocks of material of known density and plotting a calibration curve, the results of which are applied automatically by a surface logging unit.

The density log will respond not only to the formation but also to the fluid in the borehole. As the borehole diameter increases, so the effect of the borehole fluid increases. This adverse effect is removed by designing the sonde so that the measurement system is focused to give a narrow beam directed into the formation and forced against the borehole wall by a spring-loaded arm (calliper), so that it is always in contact with the formation. This removes the effects of borehole fluid except where irregularities in the borehole diameter occur during drilling due to material being washed out (caving) or deviations in borehole diameter. The problem of caving is significant for density logging in coal exploration, as a short-spaced density log can produce a response in a caved mudstone that resembles a

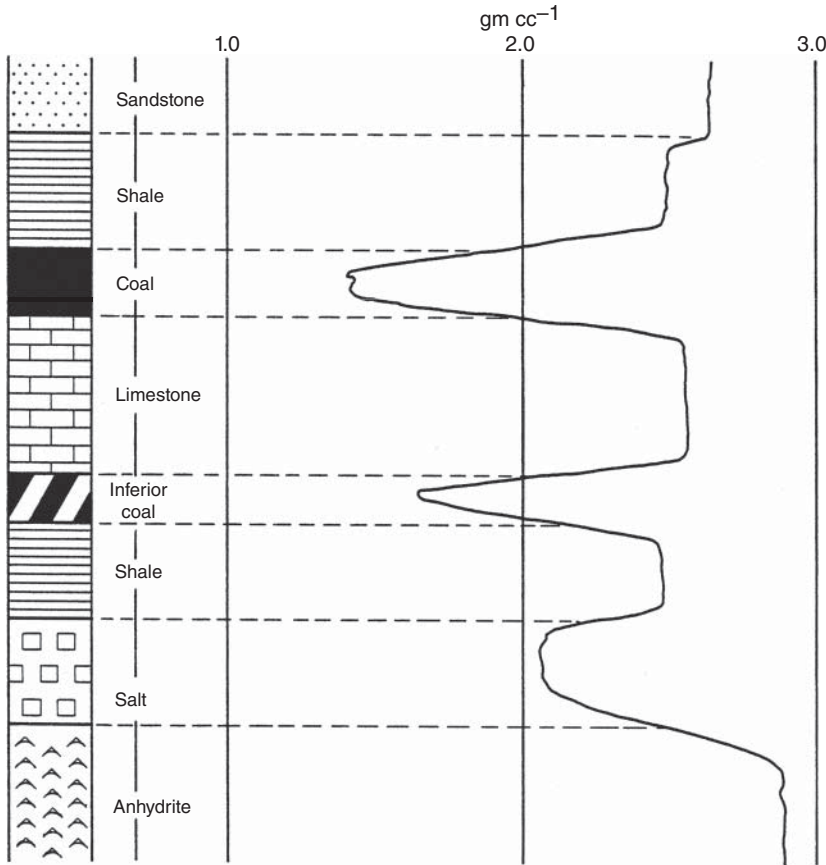
response from a coal seam. Usually, reference to the three-arm calliper log printout should highlight such anomalies.

A density log will show a sharp boundary between two formations as a curve with a vertical height approximately equal to the source to detector spacing  $S$  of the log and a horizontal length equal to the difference in densities between the formations either side of the boundary. Figure 8.24 shows the response of short- and long-spaced density logs. If the relationship between the gamma-ray intensity measured at the detector and the formation density is non-linear (long-spaced density log), the boundary point can be read off using the log calibration scale in grams per cubic centimetre on the log; or if count rate only is available, it is assumed to be two-thirds along the curve from the high count rate value, as shown in Figure 8.24. If the response is linear (short-spaced density log) the boundary is taken as the halfway point along the curve. Figure 8.23 illustrates a low-density reading for a thick lignite horizon, with higher densities characterising the mudstone and siltstone horizons. In coal seams with thicknesses less than  $S$  (source to detector spacing), the full log value of the thin bed is not recorded. Accurate thickness and log values for thin beds are more difficult to evaluate; this is particularly so for coal horizons with multiple thin partings.

With the development of a parallel-sided drill rod, it has become possible to run geophysical logs through the rod. This has resulted in gamma-ray and density logs being used in this way. This has minimised the likelihood of a radioactive source being lost down a borehole, with serious financial and environmental consequences. If there are such problems, the drill rod can be removed, bringing the tool up with it.

### 8.5.1.3 Neutron Log

Neutron logs respond primarily to the hydrogen content of saturated rocks. The neutron log consists of a source that provides a continuous



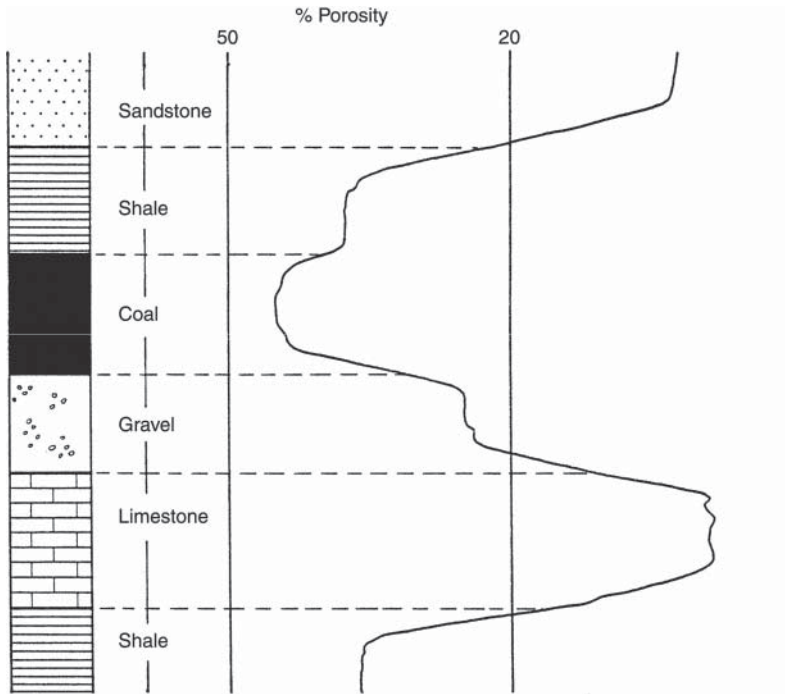
**Figure 8.24** Long-spaced density log response to coal. *Source:* Reproduced with permission of Reeves Oilfield Services Ltd.

spectrum of high-energy neutrons and a detector sensitive only to low-energy (thermal) neutrons. Hydrogen is the most effective element in the slowing down or moderation of neutrons. Once slowed down, the neutrons diffuse away from the source and are gradually captured. Therefore, as one moves away from the source, the thermal neutron population first increases as more fast neutrons are moderated and then decreases as they are absorbed.

Hydrogen is found in the rock matrix itself, in water chemically bound to the rock molecules and also in the fluid in the pore spaces of the rock. The last of these is a measure of the porosity of the rock and the amount of fluid it contains.

In sandstone, the neutron response is logarithmic with porosity, such that at low porosities it is sensitive but at high porosities it is less so. Coal gives a response of around 60% effective porosity due to its structure of hydrogen and carbon; any change in count rate can reflect changes in calorific values (CVs), which on an ash-free basis can be considered as a coal rank parameter. Where moisture is relatively constant, the neutron log can give an approximate guide to the amount of volatile matter present.

The response of a neutron log over a formation boundary is not as simple as a density log response, and it cannot be used for such accurate interpretation of thickness. The same approach as used with density logs must



**Figure 8.25** Response of neutron log over a coal seam. *Source:* Reproduced with permission of Reeves Oilfield Services Ltd.

be taken to arrive at average formation log values for use in quantitative work. From experience, bed boundary location can be approximately interpreted at about one-fifth from the high count rate value of the curve for a coal-sandstone interface, as shown in Figure 8.25.

Neutron logs have been used to synthesise rock quality indices that can be directly related to mining problems. The hydrogen held in micro- and macrofractures in the borehole and bound to rock molecules is measured (known as the hydrogen index), and empirical relationships are established between it and the observed fracture density for those rock types most commonly encountered in coal-bearing sequences. This technique has been applied, particularly in Europe and North America, for geotechnical studies.

#### 8.5.1.4 Gamma Spectrometry

Experiments to determine coal seam sulfur contents using well logging methods have been largely unsuccessful. However, Gregor

and Tezky (1997) described the measurement of sulfur content in brown coal by means of well logging equipment. It is based on spectral analysis of prompt gamma radiation generated by the capture of thermal neutrons using a spectrometric logging probe. The probe can also detect other elements, such as iron, silicon, aluminium, manganese, titanium, zinc, potassium, and calcium. This development is experimental, but if successful and made available to operate with the established log suites it would be a valuable tool in future coal exploration and mine planning.

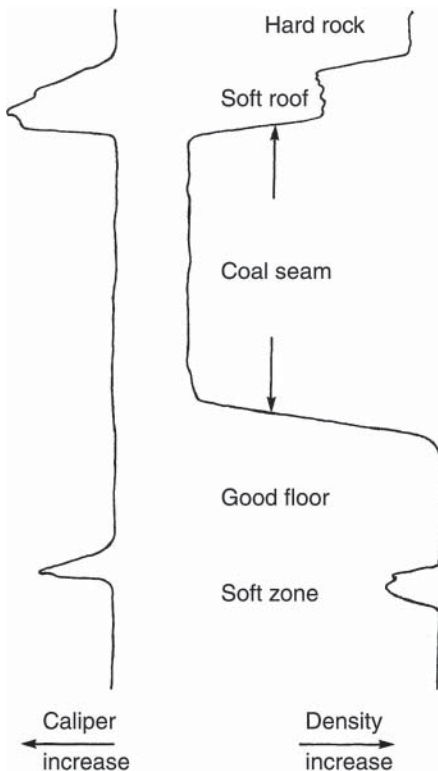
#### 8.5.2 Calliper Log

The calliper log measures the borehole diameter, and its main use is for correcting long- and short-spaced density readings. The calliper measuring system can either be part of the density logging tool, where it is a single arm used to force the logging tool against the side of the borehole, or as an individual tool with three arms at equal spacing. It is calibrated by

measuring the log output at arm extensions fixed in place using a calibration plate marked out in borehole diameters.

The three-arm calliper gives an average of borehole diameter measured at three points. Difficulties arise when using the single-arm tool where the hole size is enlarged due to caving in front of the density logging face but not on the side of the borehole where the calliper is travelling.

The calliper log in association with the density log can be used to indicate rock strength; Figure 8.26 shows the response of calliper and density logs across a coal seam that has a good sound floor but a soft roof. Such coal seam profiles are of importance to the mining engineer in estimating the mining conditions likely to be encountered in underground workings.



**Figure 8.26** Use of the combination of calliper and density logs to determine coal seam roof and floor characteristics. *Source:* Reproduced with permission of Reeves Oilfield Services Ltd.

### 8.5.3 Electric Logs

In the examination of coal-bearing sequences, electric logs may be used to support radioactive logs, but they are rarely used alone. This can be illustrated by the fact that coal possesses high resistivity, but in boreholes coal is difficult to distinguish from many rock types. Single-point resistance log and self-potential logs have been superseded by the gamma-ray log due to the latter's reliability in a variety of borehole conditions. Ideally, shales give a low reading and all other lithologies give a high reading, but both coal and sandstone can read low and therefore cause serious errors.

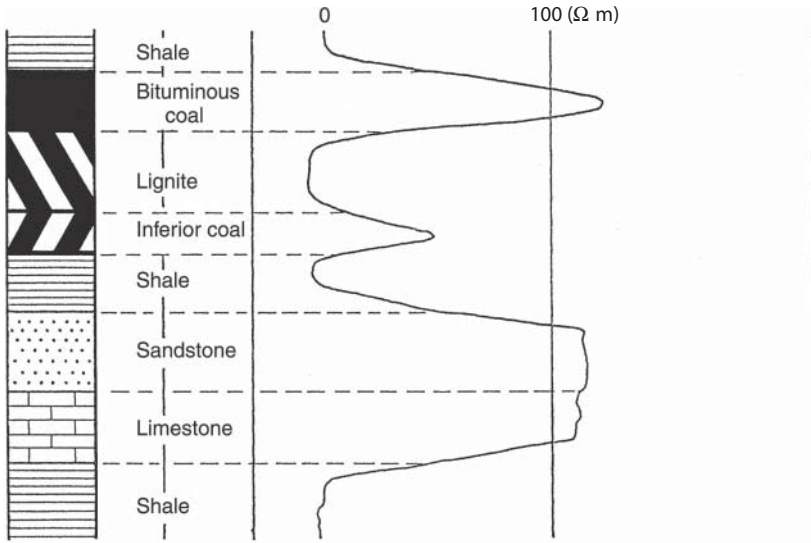
Coals in the form of lignite and anthracite give very low resistivity readings, whereas subbituminous and bituminous coal can vary from low to high. A typical response of the resistivity log in a coal-bearing sequence is shown in Figure 8.27. In addition, resistivity logs are sensitive to the volume and salinity of groundwater, and also to the clay mineral content of lithologies encountered.

Resistivity logs can distinguish coal that is burnt close to an intrusion or that is oxidised due to weathering. Burning has the effect of reducing resistivity close to an intrusion, and Figure 8.28 shows such a resistivity response across a burnt coal section.

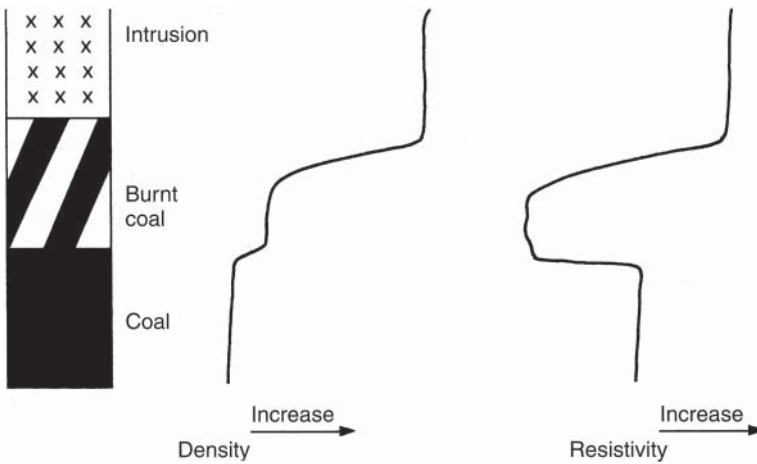
### 8.5.4 Dipmeter Log

Dipmeters make high-resolution microresistivity measurements around the borehole circumference that are correlated to produce apparent dip information. This is combined with tool orientation data to provide formation dips. Dipmeters are in two sections: a lower calliper arm holds the dipmeter pads containing the microresistivity electrodes against the borehole wall; the upper part contains the magnetometers and level cells needed to define the orientation of the tool in three dimensions. A minimum of three circumferential measurements is needed to define a plane, so that dipmeters have three arms 120° apart, and the intervals measured are overlapped





**Figure 8.27** Response of resistivity log to coal-bearing lithotypes. *Source:* Reproduced with permission of Reeves Oilfield Services Ltd.



**Figure 8.28** Response of density and resistivity logs over a burnt coal zone. *Source:* Reproduced with permission of Reeves Oilfield Services Ltd.

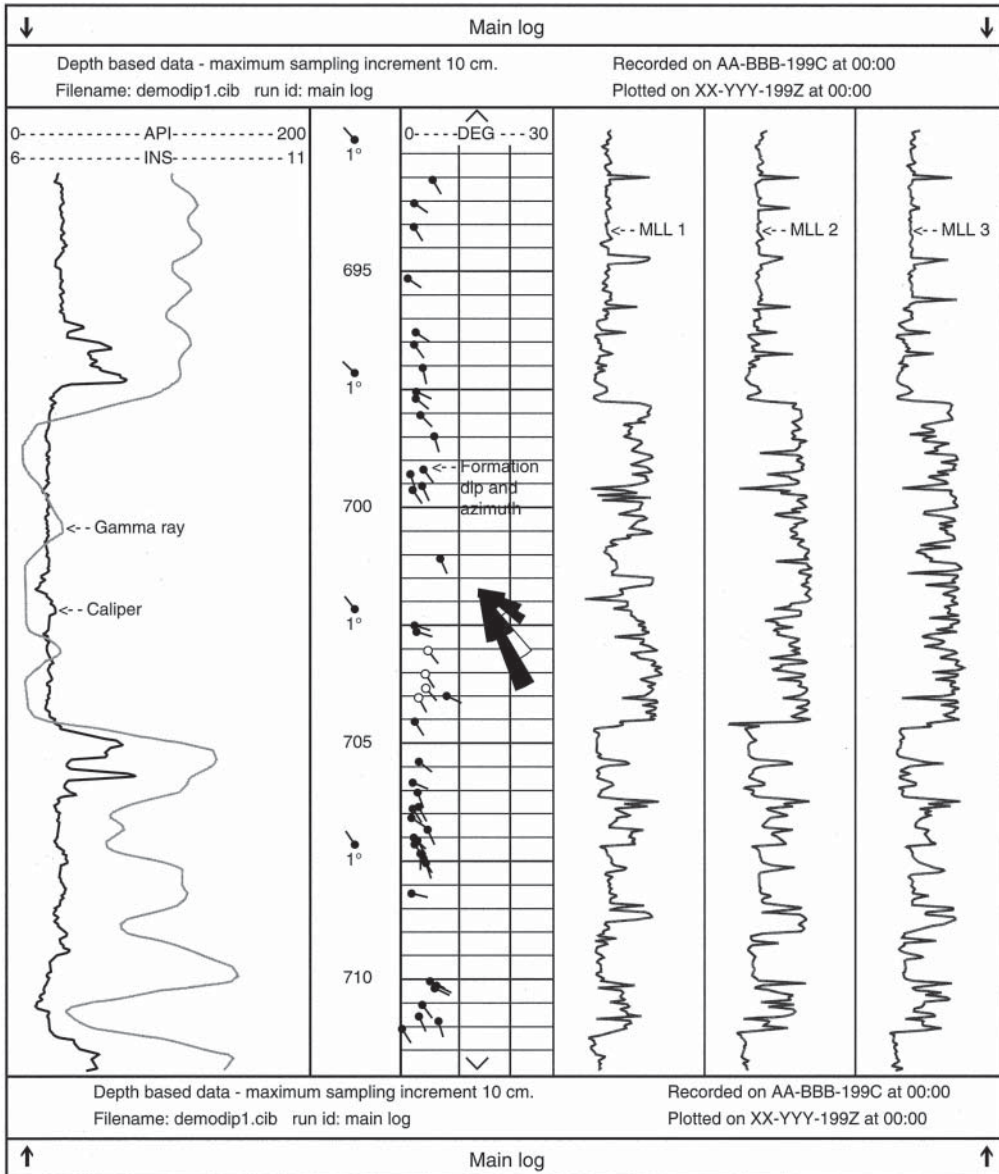
by up to 50%. Dipmeters are used not only to calculate the structural dip of the strata in the borehole, but also dip patterns at the time of deposition, by subtracting the structural dip from the observed dip in the borehole. Figure 8.29 shows dipmeter tadpoles plots for a shallow dipping coal seam in the UK.

### 8.5.5 Sonic Log

The sonic log has a similar response to the density log, as a result of the close relationship

between compaction and density. In lithological interpretations it is not better than a density log, and it is rarely run as a simple lithology log. The response of a sonic log in a typical coal-bearing sequence is shown in Figure 8.30.

The operational disadvantage of the sonic log is that it requires an open, fluid-filled hole; in addition, it is adversely affected by caving. Nevertheless, the sonic log is a useful indicator of rock strength. The log interprets the velocity of sound waves in different lithotypes, which

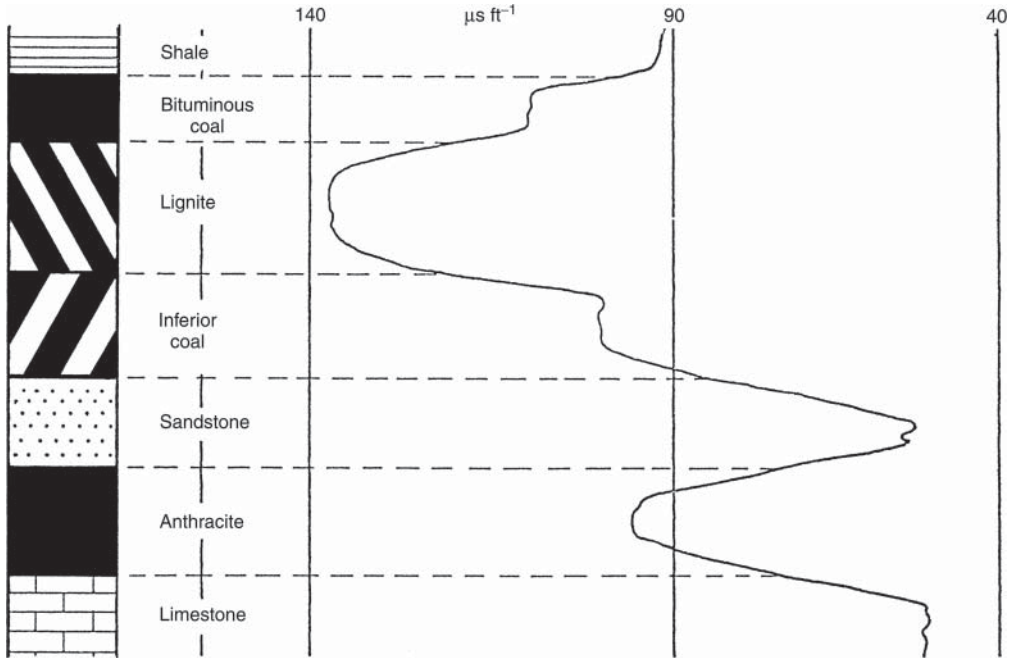


**Figure 8.29** Dipmeter tadpole plot at the target seam depth, Fillongly Hall borehole, UK. *Source:* From Firth (1999). Reproduced with permission of Reeves Oilfield Services Ltd.

is of great value in the processing and interpretation of seismic data. As the velocity is related to the geomechanical properties of the rocks, the sonic log may also be used to predict the engineering characteristics of the strata for mine planning purposes.

### 8.5.6 Acoustic Scanning Tools

Acoustic scanning tools contain a rapidly rotating transducer that emits repeated short bursts of sound energy. Each burst produces a borehole wall reflection whose amplitude and travel time characteristics are measured



**Figure 8.30** Response of sonic log to coal-bearing lithotypes. *Source:* Reproduced with permission of Reeves Oilfield Services Ltd.

by the tool and recorded at the surface. As the tool traverses the borehole, a continuous helical scan is made. This is transferred into a series of circumferential scan lines that are then rotated into a common frame of reference to remove the effects of tool orientation and borehole trajectory (Firth 1999). Continuous false-colour images are constructed by adding successive scan lines above one another on a plotter or display screen.

The maximum diameter of the tool is 57 mm, and it is important that the tool is properly centralised in the borehole. The tool is equipped with a pair of optical in-line centralisers, each centraliser comprising three articulating polished steel arms. The arms expand and contract as the borehole changes shape, thereby maintaining centralisation, even when tilted. The acoustic transducer is mounted in a rotating head assembly where it is exposed directly to the borehole fluid. A magnetometer adjacent to the transducer provides the azimuth information needed to orientate the image in a vertical

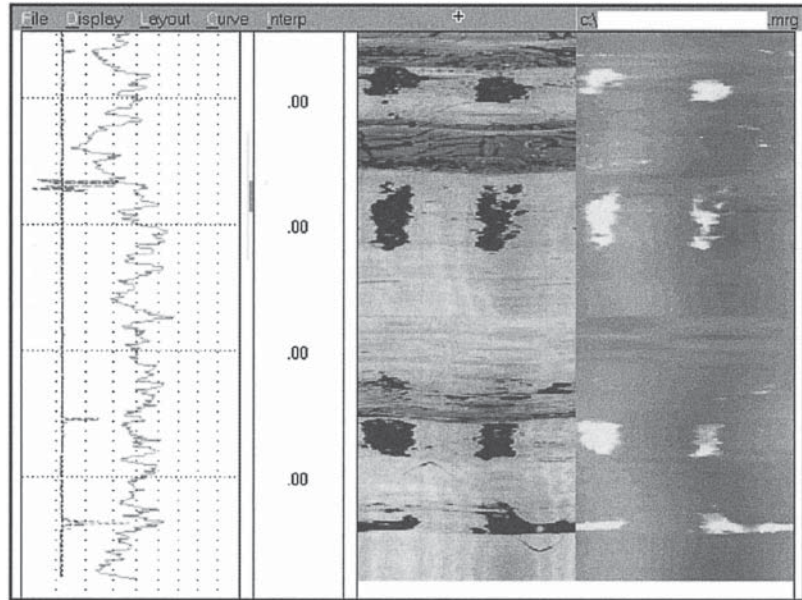
borehole. Two level cells allow the tool to be orientated in the case of an inclined borehole. The process also contains a natural gamma-ray measurement that facilitates depth correlation to core data and other open-hole logs.

Acoustic scanning tools are used in the identification of stress fields in coal-bearing strata, by portraying breakouts as dark patches on the amplitude image. In Figure 8.31 (Firth 1999), the 360° calliper shows that the borehole has caved in a particular orientation. This corresponds to the direction of minimum horizontal stress. The plotting of stress directions in a series of boreholes will determine the final orientation of underground working areas by maximising roof and wall stability (see Section 10.2.2.3).

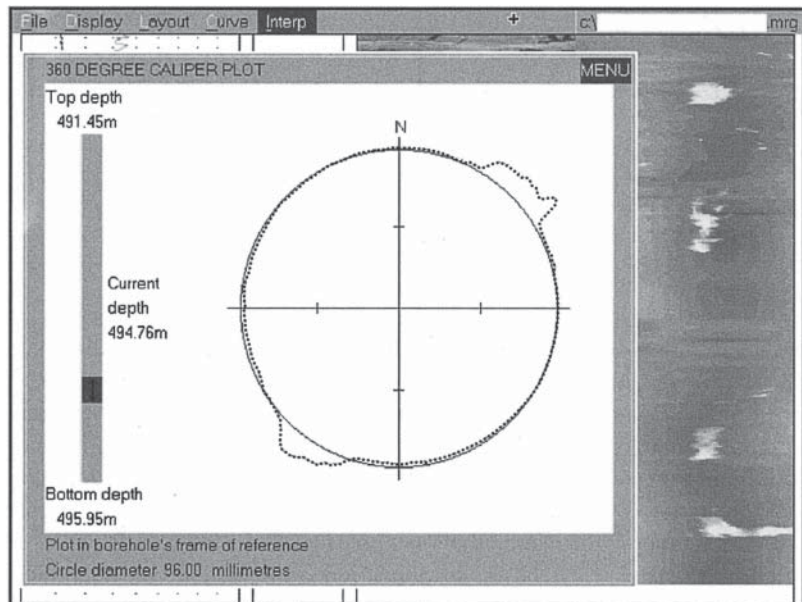
### 8.5.7 Temperature Log

The standard bottom-hole temperature is important in planning underground mining operations, and ventilation systems in particular. Changes in the temperature log can also

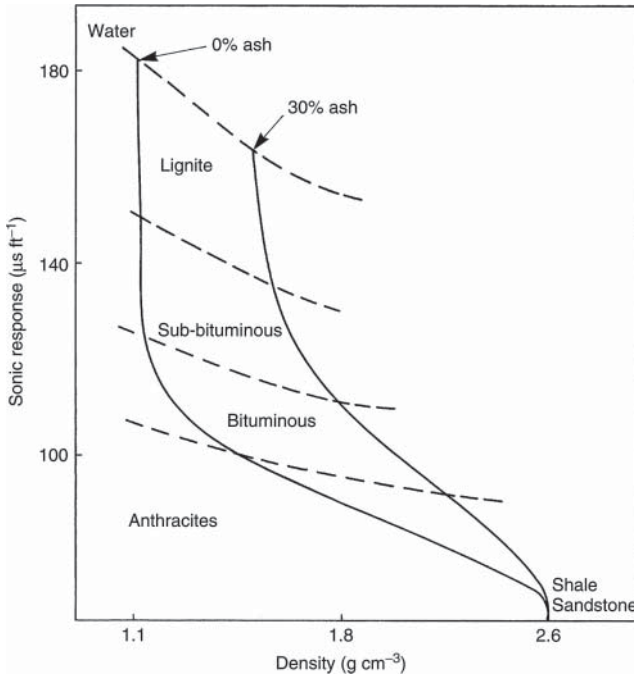
**Figure 8.31** (a) Breakout as it appears on amplitude (left) and transit time (right) images, and (b) its portrayal on a 360° caliper plot. *Source:* Reproduced with permission of Reeves Oilfield Services Ltd.



(a)



(b)



**Figure 8.32** Interpretation of coal rank from sonic and density logs. *Source:* Reproduced with permission of Reeves Oilfield Services Ltd.

indicate the levels in the borehole at which significant groundwater inflow occurs.

### 8.5.8 Advanced Interpretation

Further information on the lithology of the strata and the characteristics of the coal can be obtained by combining data from several different types of logs. For example, sonic and density logs can be combined for interpretation of coal rank, as illustrated in Figure 8.32.

Attempts have been made to develop a simple method of measuring in-situ moisture content in coal, but moisture levels in coal deposits are usually too high and render conventional neutron tools insensitive.

The combination of gamma-ray and density logs with lithological logs from boreholes is used for correlation within coal deposits. Figure 8.33 shows a density analysis for a coal seam with bulk density, raw ash, and raw CV calculations together with density and calliper

logs. A series of such printouts for each seam in a lease area can be used for correlation and for estimations of likely raw coal quality across the area. Also, where horizons with high gamma readings are present, they will readily show up on gamma-ray logs, as will coal seams and clean sandstones, which have low gamma radiation, all of which facilitates correlation. Figure 8.34 shows a combination of lithological, sonic, and gamma-ray logs for the coal-bearing Gidgealpa Group, Cooper Basin, Australia. Here, the purpose is to identify not only the coal horizons but also the sandstones as possible hydrocarbon reservoir rocks. Frequency plots using gamma and density data can be produced. Computer software calculates matrix, shale, and porosity volumetrics for every depth-matched point of gamma, density, and calliper; the density end point is determined by the desired matrix type, i.e.  $2.65 \text{ g cm}^{-3}$  for a sandstone matrix. Figure 8.35 shows the cross-plot of



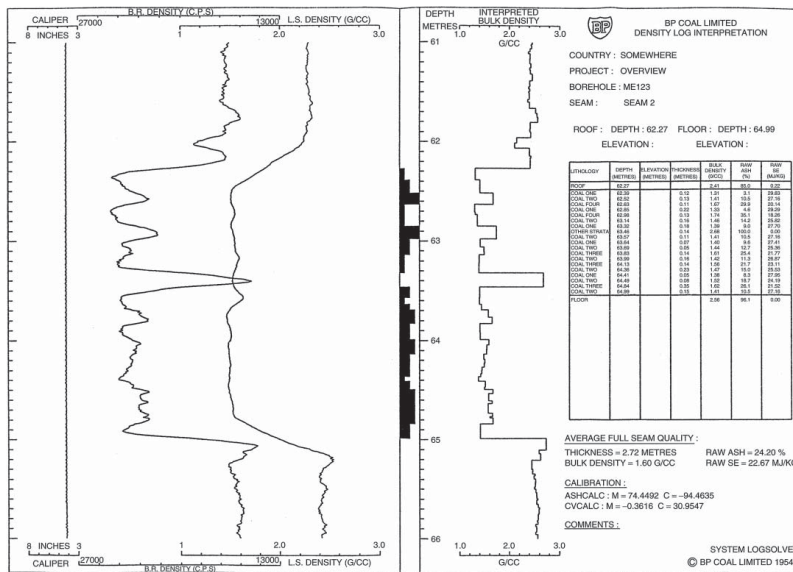
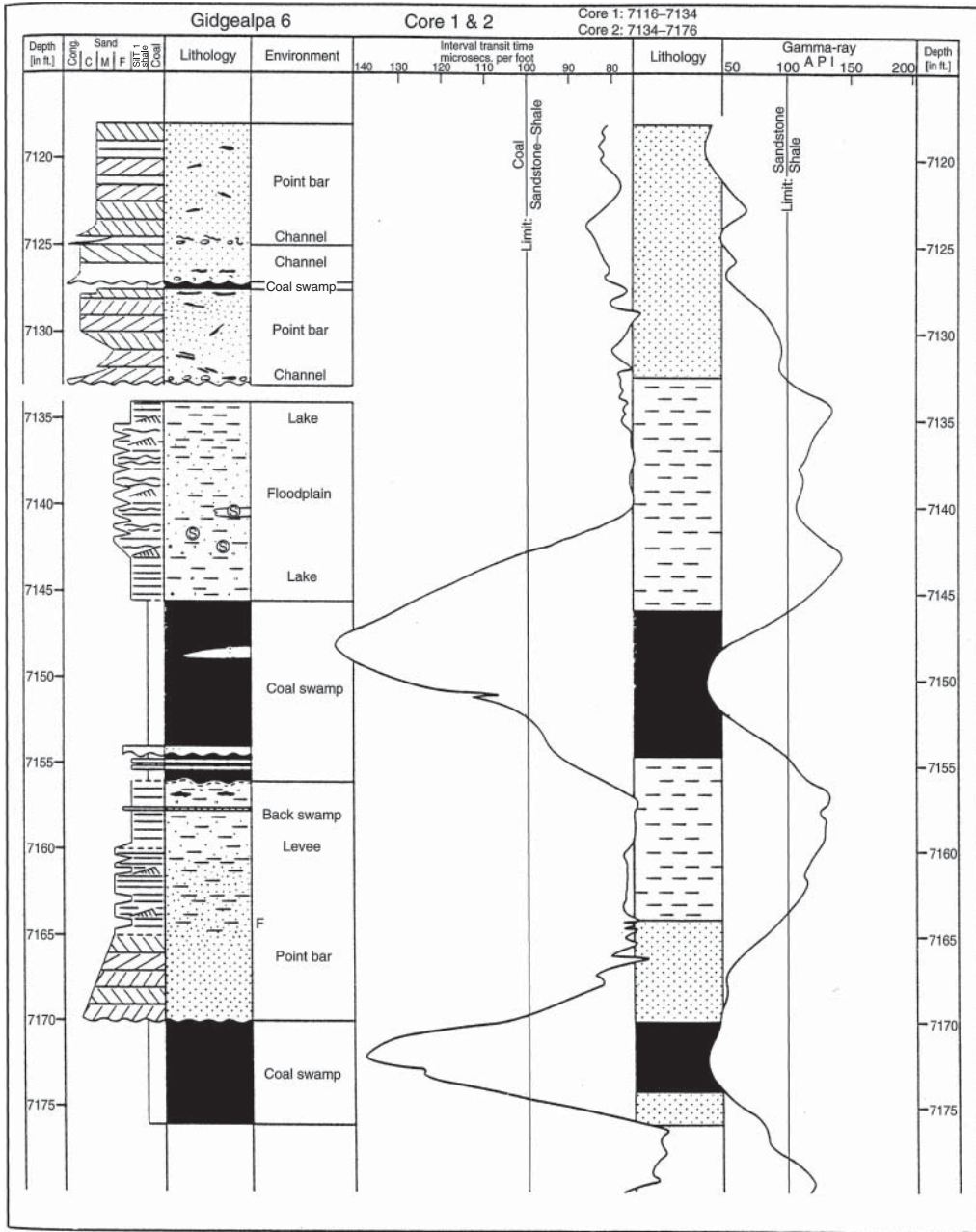
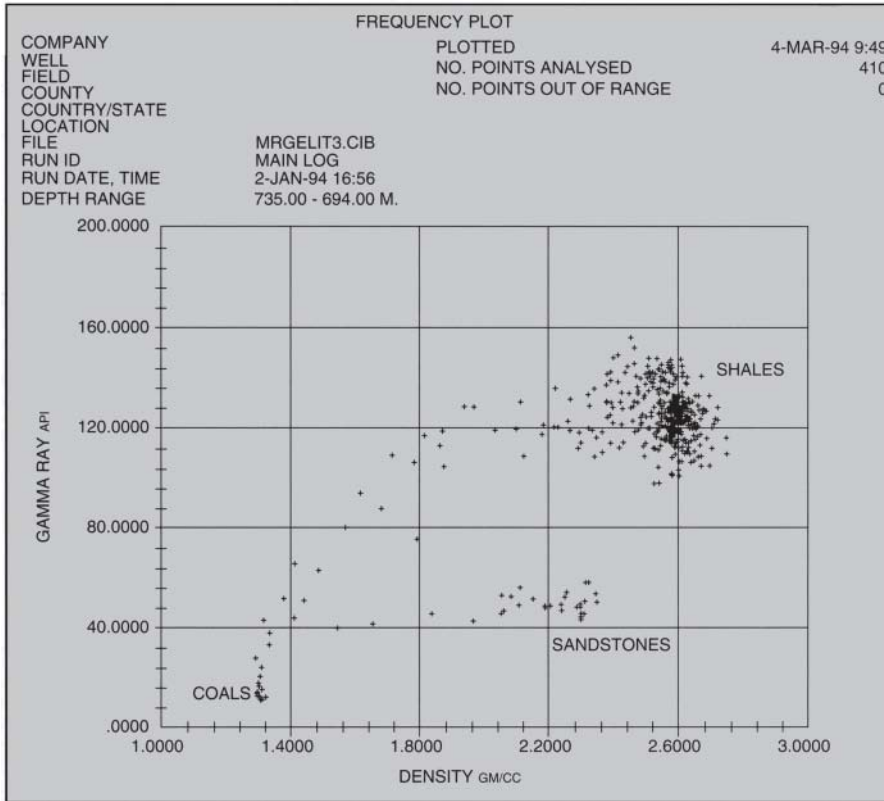


Figure 8.33 Density log interpretation together with interpreted coal seam bulk densities and raw ash content and raw CV calculations together with density and calliper logs.



**Figure 8.34** Combination of lithological, sonic, and gamma logs to identify coal seams and sandstones as potential hydrocarbon reservoirs. *Source:* From Thornton (1979).



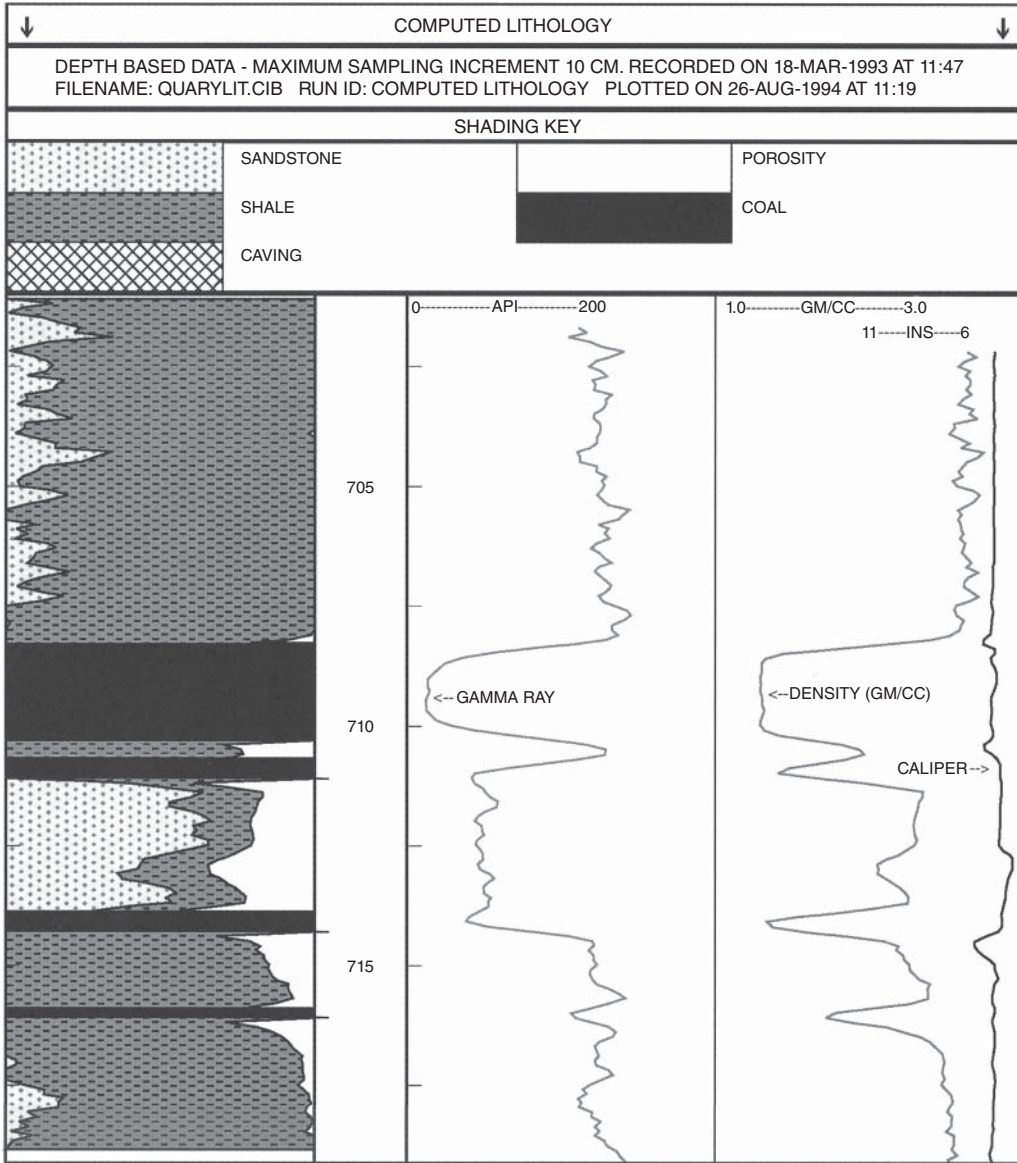
**Figure 8.35** Cross-plot of depth-matched gamma-ray and density data. *Source:* Reproduced with permission of Reeves Oilfield Services Ltd.

depth-matched gamma-ray and density data; sandstones and shales are indicated, and coals can be identified owing to their unique combination of low gamma ray and low density. Coal is plotted where densities of less than  $1.9 \text{ g cm}^{-3}$  are found; calliper values of less than 10 cm have been selected so that low densities due to caving will not be interpreted as coal. Once the end points and coals have been identified, the final lithological analysis can be produced; for example, the computed lithology analysis as shown in Figure 8.36 (Firth 1999).

This style of correlation is carried out during the exploration drilling phase of any project, together with ensuring the reconciliation of

coal seam roof and floor depths on the geophysical logs with those on the lithological logs, the latter based on the open and cored borehole records.

The use of geophysical logs in coal exploration is now established practice, essential for any reserve assessment of a coal deposit. Using the various logs described, the required information can be summarised as follows: correlation of coal seams and other horizons across a deposit, accurate seam depths and thicknesses, coal seam structure details, control of drill core sampling, assessment of core recovery percentages, and the indication of coal quality and quality variation across the deposit. A summary of log



**Figure 8.36** Computed lithology analysis derived from data shown in Figure 8.34. *Source:* Reproduced with permission of Reeves Oilfield Services Ltd.

responses in a variety of lithologies is shown in Figure 8.37.

More sophisticated geophysical logging techniques can provide the following additional information: lithology interpretation, assessment of geotechnical properties of formations

(principally rock strength), the orientation (verticality and direction of deviation) of boreholes, and the measurement of stratigraphic dip of formations for use in the determination of geologic structure.

		Gamma ray		Density		Sonic		Porosity		Resistivity $\Omega$ m							
		0	API	150	1.0	$g\ cm^{-3}$	3.0	140	$\mu S\ ft^{-1}$	40	50	SST	PU	0	10	100	1000
Shale	Marine	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█
	Non-marine	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█
Coal	Bituminous	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█
	Inferior	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█
	Lignite	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█
	Anthracite	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█
Sandstone	Porous	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█
	Tight	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█
Siltstone		█	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█
Evaporites	Gypsum	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█
	Salt	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█
	Anhydrite	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█
Limestone	Porous	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█
	Tight	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█	█

**Figure 8.37** Summary of log responses in various lithologies. *Source:* Reproduced with permission of Reeves Oilfield Services Ltd.



## 9

# Hydrogeology of Coal

## 9.1 Introduction

The influence that water plays in the mining of coal is significant, both in terms of surface water and groundwater movement. Water poses some of the major problems in mining in opencast and underground coal operations, and the presence of water, or sometimes the lack of it, in planning exploration and development programmes needs to be carefully assessed.

Field operations are strongly influenced in this way; for example, high and low water levels in stream and river sections will decrease or increase exposure. An abundant supply of water is required for water and mud flush drilling, but not necessary for air flush drilling, which is inhibited by a large head of water. Fundamental logistics such as transport are affected: dirt roads within the field area can become waterlogged and impassable, and very dry roads can create dust problems.

There are also environmental concerns on how mine waters are treated and disposed of.

There are numerous texts on how surface water and groundwater will affect mine operations; for example, the National Coal Board Mining Department (1982) publication *Technical Management of Water in the Coal Mining Industry*. A summary of the basic properties of groundwater and those aspects of hydrogeology directly related to coal-bearing sequences and coalmining are described in this chapter.

## 9.2 The Nature of Groundwater and Surface Flow

The hydrological cycle begins and ends with the oceans: water is evaporated from the ocean, which then condenses to form clouds. Water is precipitated from clouds, some of which falls onto the land surface and collects to form streams, rivers, and lakes, and eventually flow overland back to the oceans. A portion of the rainfall passes through the soil to reach the water table and so becomes groundwater.

### 9.2.1 Surface Water

The area of land that drains to a river is called the catchment area. These areas are separated by high ground called a watershed or divide. Streams that flow throughout the year are termed perennial streams, those that flow only occasionally are termed ephemeral streams, and those that flow only after the wet season are known as intermittent streams.

The discharge of a river or stream is the volume of water flowing past a given point in a unit of time, i.e. it equals the cross-sectional area of the flow section times the speed at which the water is flowing. Most useful is the record of flow plotted against time, which is shown on a discharge hydrograph.

Two components contribute to river flow: a baseflow component consisting of groundwater flow and slow interflow, and a quickflow

component derived from rapid interflow, surface runoff, and any rain that falls directly onto the river or stream.

Surface water flow measurements are usually made during the geotechnical studies stage following the geological exploration stage, particularly where it is likely that water-courses will need to be rerouted during the construction and working life of the mine.

### 9.2.2 Groundwater

The upper part through which percolation occurs is known as the vadose zone or zone of aeration, and water movement is primarily under the influence of gravity. The phreatic zone or zone of saturation is below the water table in which pore spaces within the rock are filled with water. Water movement is primarily under the influence of hydrostatic and hydrodynamic pressures. These two zones are separated by the groundwater table, which will vary in position as changes in the groundwater level occur. These changes can be negative, resulting from groundwater movement and discharge, or positive, resulting from groundwater recharge by percolating water from the vadose zone.

Rocks that contain groundwater and allow it to flow through them in significant quantities are termed aquifers. Under normal circumstances water flows to a natural discharge point, such as springs and seepages. This process can be interrupted if wells are sunk into the aquifer and water is abstracted.

This ability to allow water to flow through the aquifer is termed permeability and is controlled largely by geological factors. When the properties of the fluid are considered, the permeability or the ease with which the water can move through the rock is referred to as the hydraulic conductivity, expressed in metres per day. The changes in height or head that the water can attain naturally are known as the hydraulic gradient; the steeper the gradient, the faster the flow of water.

Groundwater may be contained in and move through pore spaces between individual grains in sedimentary rocks. Where rocks are fractured, this can significantly increase the hydraulic conductivity of the rock. The ratio of the volume of voids in the rock to the total volume of the rock is termed porosity. Some lithotypes do not allow the passage of fluids through them at significant rates or may allow only small quantities to pass through. These are termed aquicludes and aquitards respectively, but they are more commonly referred to as confining or impermeable horizons.

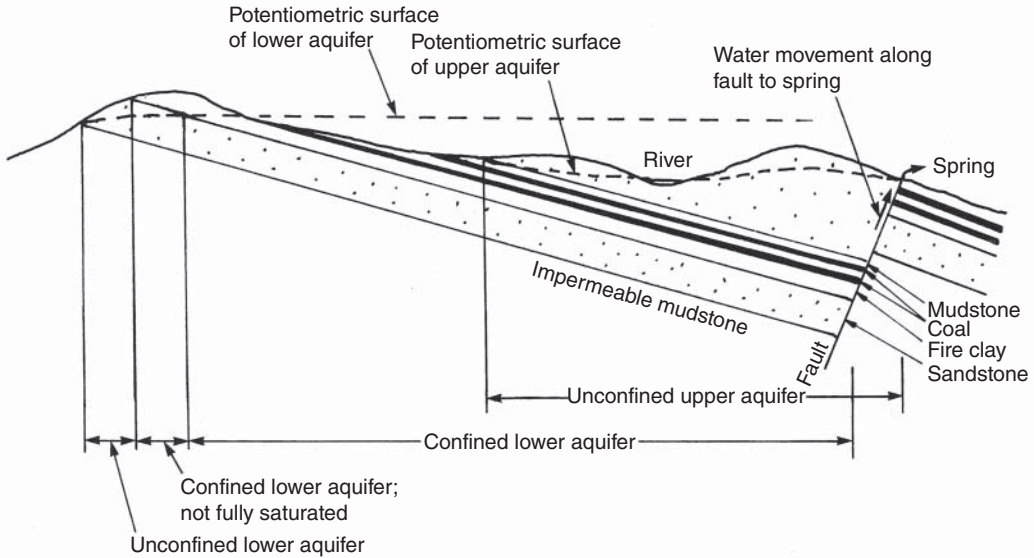
Groundwater usually flows under a hydraulic gradient, i.e. the water table. Where an aquifer is overlain by impermeable rocks, the pressure of groundwater may be such that the rest level of water would normally be well above the base of the impermeable layer. In such circumstances, the aquifer is said to be confined, and the surface is known as the potentiometric or piezometric surface.

The relationship of permeable and impermeable strata in confined and unconfined conditions in a coal-bearing sequence is illustrated in Figure 9.1.

In studying coal-bearing sequences, it is essential to identify the horizons that will act as aquifers and those that will remain impermeable. In the case of aquifers, it will be necessary to calculate how much water passes through in a given time. To study this phenomenon, Darcy's law is used, first proposed by Darcy in 1856; this states that a fluid will flow through a porous medium at a rate  $Q$  that is proportional to the product of the cross-sectional area  $A$  through which flow can occur, the hydraulic gradient  $i$ , and hydraulic conductivity  $K$ :

$$Q = KiA$$

An aquifer's effectiveness to transmit water, as calculated by Darcy's law, is known as its transmissivity, usually expressed in square metres per day.



**Figure 9.1** Groundwater conditions in a coal-bearing sequence, showing an upper unconfined aquifer and a lower confined aquifer with an intervening coal–mudstone sequence.

In the case of an unconfined aquifer, the slope of the water table is a measure of the hydraulic gradient; in this case, the transmissivity is the product of the hydraulic conductivity and saturated bed thickness, the latter not being a constant feature. Because the water table is sloping, water flow is not purely horizontal, which means that the hydraulic gradient has a vertical and a horizontal component.

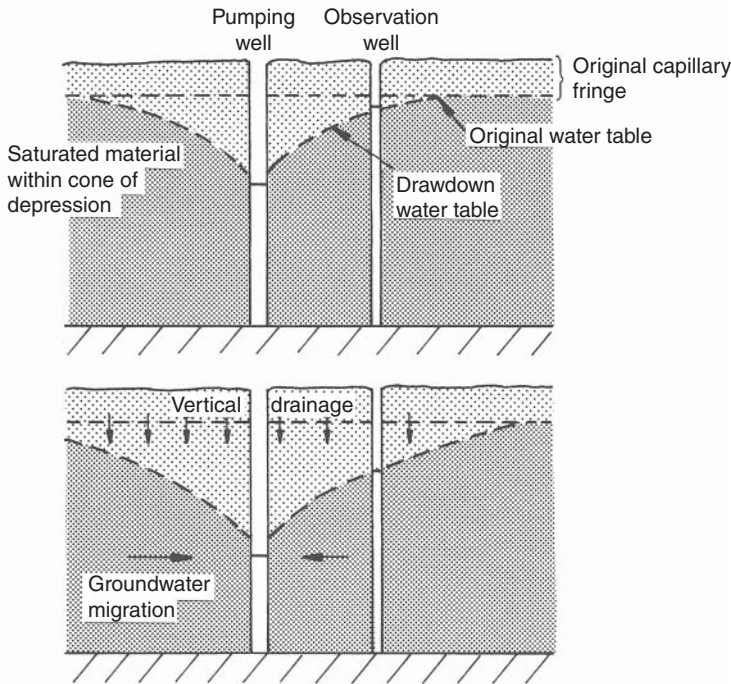
In the case of a confined aquifer, the aquifer remains fully saturated. When water is removed from the pore spaces, the water pressure is lowered and downward pressure on the pore spaces within the aquifer causes slight compression. This reduction in pressure also causes a slight expansion of the water. The volume of water released from or taken into storage per unit surface area of the aquifer for each unit change of head is known as the storage coefficient.

In practical terms, changes in water levels within the designated area of interest must be monitored and recorded. This is accomplished by installing piezometers and observation and pumping wells (boreholes). Boreholes that are

sealed throughout much of their depth, so that they can measure the head at a particular depth in the aquifer, are known as piezometers. The information they supply forms an essential part of the geotechnical investigations carried out prior to the mine feasibility study.

Boreholes that pass through the sequence to be mined act as monitoring holes from which the water table levels within the area can be measured on a regular basis.

The action of pumping water from a borehole causes a reduction in pressure around the pump. This creates a head difference between water in the borehole and that in the aquifer. Water then flows from the aquifer towards the borehole to replace the water pumped out. Gradually, water flows towards the borehole from further and further out in the aquifer; this has the effect of lowering the hydraulic gradient, so that around the borehole the hydraulic gradient becomes steeper, forming a characteristic lowering of the water table. This is known as the cone of depression; the reduction in head or lowering of the water table at the borehole itself is called the drawdown. These features are illustrated in Figure 9.2.



**Figure 9.2** Drawdown of the water table due to pumping in an unconfined aquifer. Source: From Price (1996).

### 9.3 Hydrogeological Characteristics of Coals and Coal-Bearing Sequences

The majority of coal-bearing sequences contain sandstones, siltstones, mudstones, fireclays, and coals. Of these, sandstones have the greatest potential for storing and transmitting groundwater; the other lithotypes have characteristically low permeabilities.

In older coal-bearing sequences, such as Carboniferous–Permian strata, the sediments are well indurated with low permeabilities; sandstones are often well cemented, so reducing their porosity and permeability. In younger formations, such as in the Palaeogene–Neogene, sandstones may still be only partially cemented or totally uncemented and therefore have the potential to hold and transmit large amounts of groundwater by connected intergranular flow, i.e. they have primary permeability.

This is not to say that coals and coal-bearing strata do not allow the passage of groundwater.

Most sequences are tectonically disturbed and contain numerous discontinuities, such as joints and faults; if open, these will hold and allow groundwater flow, i.e. they have secondary permeability. In addition, it is a common feature for groundwater to flow along inclined bedding surfaces and appear in workings as a series of seepages, often staining the underlying strata with mineral precipitates.

The permeability of coal in general can be regarded as highly stress dependent, decreasing as the level of stress is increased. Coals react differently to stress due to their composition and rank; coals with a high degree of elasticity and no apparent fractures are usually relatively unaffected by fluctuations in stresses exerted upon them. On the other hand, highly fissured and/or low mechanical strength friable coals tend to microfracture under stress. In the case of the latter, subsequent release of stress will leave the coal permanently microfractured, and this then creates an increase in the overall permeability of the coal. There is also a

relationship in the compressibility of coal compared with the volatile matter present. There seems to exist an increase in compressibility with increasing volatile content up to around 36%, and then a decrease towards the lower rank coals.

Coals can hold significant amounts of water, which upon being breached by mine workings is released and can cause mining difficulties; this mainly occurs in underground workings, as opencast operations tend to be dewatered prior to mining.

Fireclays or seatearths and other clay-rich sediments have the ability to hold water, both in those clay minerals that have expanding lattice structures and as adsorbed water on the individual clay particles. This relatively high porosity does not, however, result in high permeability; the water is retained around the clay particles, held by surface tension, a phenomenon known as specific retention. Although such sediments have low permeability, their water content is important as they have geotechnical significance as horizons of weakness when subjected to increased stress or sudden depressurisation.

Table 9.1 shows a list of indicative porosities and hydraulic conductivities for selected unconsolidated and consolidated sediments found in coal-bearing sequences (Brassington 1988). In the case of peats and brown and black coals, it is difficult to give such indicative values. Porosity values for peat will be high, whereas brown, and more particularly black, coal will have low porosity values due to the increasing effects of compaction and coalification. Permeability values are difficult to quantify owing to the fact that coals are dominated by discontinuities that may or may not allow the passage of water. High-ash coals are known to have porosities of around 20%, with permeability values of less than 1 m per day.

Studies of the hydraulic conductivity of peats have indicated that estimates showed time dependence, but that highly humified peat does not appear to transmit water strictly in accordance with Darcy's law; this may be due to air entrapment, whereas low-humified peats tend to conform to Darcy's law.

In low-rank coals, all the water does not reside in pores alone; some water must actually be included in the organic structure.

**Table 9.1** Indicative porosities and hydraulic conductivities for unconsolidated and consolidated sediments which characterise coal-bearing sequences.

Lithotype	Grain size (mm)	Porosity (%)	Hydraulic conductivity $K$ ( $\text{m day}^{-1}$ )
<b>Unconsolidated sediments</b>			
Clay	0.0005–0.002	45–60	$<10^{-2}$
Silt	0.002–0.06	40–50	$10^{-2}$ –1
Sand	0.06–2.0	30–40	1–500
Gravel	2.0–64.0	25–35	500–10 000
<b>Consolidated sediments</b>			
Shale/mudstone	$<0.002$	5–15	$5 \times 10^{-8}$ – $5 \times 10^{-6}$
Siltstone	0.002–0.06	5–15	$5 \times 10^{-8}$ – $5 \times 10^{-4}$
Sandstone	0.06–2.0	5–30	$10^{-4}$ – $10^b$
Limestone	Variable	0.1–30 <sup>a</sup>	$10^{-5}$ – $10^b$

a) Secondary porosity.

b) Secondary permeability.

Source: From Brassington (1988).



Experimentally, pore size distribution can be determined by forcing mercury into coals at increasing pressures and measuring the volume of mercury intrusion (mercury porosimetry). However, corrections have to be made for the compressibility of coals; also, the high pressure may open/close pore space, which can be ascertained by measuring helium density of the coal samples before and after mercury intrusion.

The experimental work of Gan et al. (1972) can be used to illustrate this. Twelve coals were tested by mercury porosimetry, and the results are shown in Table 9.2. The pore volume distributions are given for the following pore ranges:

- (i) Total pore volume  $V_T$  accessible to helium as estimated from helium and mercury densities.
- (ii) Pore volume  $V_1$  contained in pores  $>300 \text{ \AA}$  in diameter.
- (iii) Pore volume  $V_2$  contained in pores  $300\text{--}12 \text{ \AA}$  in diameter.
- (iv) Pore volume  $V_3$  contained in pores  $<12 \text{ \AA}$ ,  $V_3 = V_T - (V_1 + V_2)$ .

The proportion of  $V_3$  is significant for all coals; its value is a maximum for the anthracite sample (PSOC-80) and a minimum for the lignite sample (PSOC-89). From Table 9.2, it can be concluded that:

- (i) Porosity in coals with carbon content  $<75\%$  is predominantly due to macropores.
- (ii) Porosity in coals with carbon content  $85\text{--}91\%$  is predominantly due to micropores.
- (iii) Porosity in coals with carbon content  $75\text{--}84\%$  is associated with significant proportions of macro-, meso-, and microporosity.

On a microscopic level, the influence of microporosity of coals by maceral types has been the focus of a number of recent studies, chiefly linked with the retention and desorption of methane (Karayığit et al. 2017; Teng et al. 2017). Although an integral part of the properties of any coal, it has little influence on groundwater flow in the context of mining operations.

**Table 9.2** Gross open-pore distributions<sup>a</sup> in coals.

Sample	Rank <sup>b</sup>	Carbon (% daf)	$V_T$ ( $\text{cm}^3 \text{g}^{-1}$ )	$V_1$ ( $\text{cm}^3 \text{g}^{-1}$ )	$V_2$ ( $\text{cm}^3 \text{g}^{-1}$ )	$V_3$ ( $\text{cm}^3 \text{g}^{-1}$ )	$V_3$ (%)	$V_2$ (%)	$V_1$ (%)
PSOC-80	Anthracite	90.8	0.076	0.009	0.010	0.057	75.0	13.1	11.9
PSOC-127	lv	89.5	0.052	0.014	0.000	0.038	73.0	0	27.0
PSOC-135	mv	88.3	0.042	0.016	0.000	0.026	61.9	0	38.1
PSOC-4	hvA	83.8	0.033	0.017	0.000	0.016	48.5	0	51.5
PSOC-105A	hvB	81.3	0.144	0.036	0.065	0.043	29.9	45.1	25.0
Rand	hvC	79.9	0.083	0.017	0.027	0.039	47.0	32.5	20.5
PSOC-26	hvC	77.2	0.158	0.031	0.061	0.066	41.8	38.6	19.6
PSOC-197	hvB	76.5	0.105	0.022	0.013	0.070	66.7	12.4	20.9
PSOC-190	hvC	75.5	0.232	0.040	0.122	0.070	30.2	52.6	17.2
PSOC-141	Lignite	71.7	0.114	0.088	0.004	0.022	19.3	3.5	77.2
PSOC-87	Lignite	71.2	0.105	0.062	0.000	0.043	40.9	0	59.1
PSOC-89	Lignite	63.3	0.073	0.064	0.000	0.009	12.3	0	87.7

a) See text for definitions of pore volume distributions.

b) Bituminous coals: lv, low volatile; mv, medium volatile; hv, high volatile.

Source: From Gan et al. (1972).

## 9.4 Collection and Handling of Hydrogeological Data

During any mining operation it is important to minimise any disturbance of the surface hydrology or groundwater regimes. These regimes comprise the dynamic equilibrium relationships between precipitation, run off, evaporation, and changes in the groundwater and surface water store, and they can be extended to include erosion, sedimentation, and water quality variations. It is necessary, therefore, to know the pre-mining conditions that exist in the area of interest. The intensity of investigations will be influenced by the particular circumstances existing in the area of interest; for example, the rainfall characteristics, drainage characteristics, presence or absence of aquifers, and the geological and structural character of the area.

The collection of data relating to (i) surface water flow and (ii) groundwater flow together with water quality analysis will enable a hydrogeological model to be constructed that will form an integral part of the development studies prior to mine design. The field techniques required to measure both surface water and groundwater are outlined in detail in Brassington (1988).

### 9.4.1 Surface Water

In the majority of countries in which coal is mined, the bulk of precipitation is in the form of rainfall. To measure rainfall over the designated area, a network of rain gauges is sited and monitored on a regular basis, in order to build up a detailed record of rainfall for the area (expressed as millilitres).

The flow of most springs is measured by filling a calibrated vessel in a given period of time. This can be a difficult operation in tropical terrain, where the area around the spring may need to be cleared of vegetation prior to any measurement being taken.

The flows of rivers and streams are calculated by measuring the water velocity and

the river cross-sectional area, or by installing a weir – the former being more suitable for rivers and the latter for streams – where the flow is measured in litres per second.

### 9.4.2 Groundwater

In order to ascertain the potential groundwater problems that may be encountered during mining (particularly opencast operations), it will be necessary to determine the groundwater characteristics of the area, in particular the groundwater flow patterns, flow rates, and depth to the water table.

Flow patterns will be affected by lithotypes, their disposition, relative permeabilities, and the presence of faults, joints, and open bedding planes. For a full understanding of the groundwater for a proposed mining area, a site-specific hydrogeological model would need to be developed.

In order to achieve this, a system of monitoring boreholes or wells will need to be constructed, their siting to be based on all the information gathered on the geology of the area plus all surface water occurrences. This information will be used to determine the expected direction and flow rate of groundwater and will enable the monitoring boreholes to be sited favourably. Once the set of boreholes or wells has been constructed, hydraulic testing can be carried out.

These boreholes will provide information on the position of the water table, which in turn will define a baseline condition of any seasonal or climatic fluctuations against which the impact of mining can be assessed. In addition, samples taken from them will indicate general water quality. This monitoring programme is particularly important if aquifers are present in the designated mining area.

Boreholes used as monitoring points may be of different types. Those boreholes that are sealed throughout much of their depth in such a way that they measure the head at a particular depth in the horizon selected are known as piezometers. Piezometers are installed in

areas that have opencast mining potential; they monitor the groundwater conditions in both shallow formations, such as superficial deposits, and formations likely to influence surface mining operations. Piezometers can also be drilled at an angle to intercept vertical fractures in less permeable strata. Piezometers are usually sited where they can function throughout the life of the mine; Figure 9.3 shows piezometers set to measure levels in two formations in an opencast site.

Boreholes that are used as observation wells will include those that have already been drilled in the area for stratigraphical purposes and have been kept open for such a use, plus new site-specific boreholes. Of these, at least one observation borehole should be placed intersecting each aquifer both upstream and downstream of the mine site; further boreholes should be placed around the periphery and



**Figure 9.3** Piezometer group measuring water levels in two formations in an opencast working. *Source:* Photograph by LPT.

within the mine site, to ensure that a reliable estimation of the water level in each aquifer can be determined; and additional boreholes should be located at any geological discontinuities that may affect groundwater flow, such as faults, folds, and abrupt changes in aquifer thickness.

Field permeability tests (in situ tests) to determine the coefficient of permeability  $k$  may be undertaken in open boreholes, piezometers, or in sections of cored boreholes isolated by inflated packers. British Coal (1993) outlined the following tests for soils and rock in potential opencast mine prospects.

Falling- and rising-head tests require that the groundwater level is above the base of the borehole(s), and the basis of the tests is that the water level in the borehole or piezometer is either raised by adding water or reduced by extracting water, and the rate at which the level returns to the pre-test position is measured. If the water level has been raised, then measurements are taken of the fall with time (falling-head test) until equilibrium is reached. If the water level has been lowered, measurements are taken at intervals until the water level rises to its equilibrium point (rising-head test). Constant-head tests are performed where opencast mining is envisaged, and usually applied to soils. The rate of flow required to maintain a constant head of water within the borehole is measured. This is chiefly used where stress changes will result in significant consolidation or swelling (i.e. clays). Constant-head tests are likely to give more accurate results than falling- and rising-head tests as they should not be regarded as accurate determinations of absolute permeability for borehole yield or groundwater inflow. Packer tests are carried out in boreholes and are equivalent to the constant head for soils. Particular sections of a borehole are isolated using inflatable packers, water is then introduced to the test section in pressure stages, and measurements are made of water pressure and flow. Permeability can be calculated from the rate of flow into the borehole and the

length and radius of the test section. Practical problems can arise with effective sealing of the packers and nature of the strata selected for the test. Pumping tests are generally preferred to the aforementioned tests; these consist of groundwater being pumped at a steady rate from a borehole and the drawdown measured within the borehole and in observation boreholes at a distance from the test borehole. The groundwater level around the test borehole will fall as pumping proceeds, forming a cone of depression. The shape of the cone of depression and rate of water level lowering may be used to determine the hydraulic properties of any aquifer present and the hydraulic interconnection and hydraulic gradients within these aquifers. Figure 9.4 shows a pumping test in operation in the Thar Coalfield, Pakistan. Pumping tests will be influenced by the presence of any structural barriers, such as faults or local recharge from surface drainage. The coefficients of permeability,

transmissivity, and storage from observation borehole data can be determined and the results used to assess dewatering requirements for the mine.

Observation boreholes and piezometers are regularly placed to determine groundwater levels and the pressurisation field around the proposed position of a highwall in an opencast pit. Figure 9.5 shows the siting of such boreholes in a proposed opencast mine.

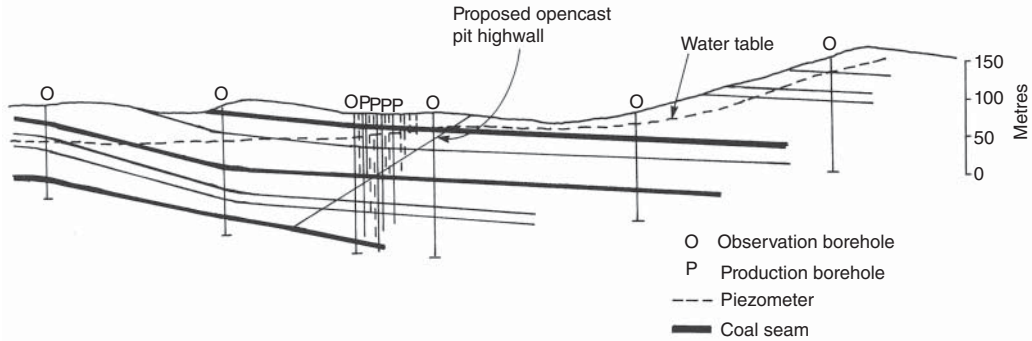
Water levels will be measured and recorded in all boreholes and piezometers on a regular basis. If any borehole is used to pump water, then the drawdown and rest levels in the pumped borehole together with rest levels of water in surrounding observation boreholes will be recorded.

Once the network of piezometers and boreholes has been established in the designated area, regular monitoring and recording of field data will become an important routine operation.



**Figure 9.4** Pumping test in operation in the Thar Coalfield, Pakistan. *Source:* Photograph by Dargo Associates Ltd.





**Figure 9.5** Section across proposed opencast pit showing position of observation boreholes, production boreholes, and piezometers, placed to determine groundwater levels and pressurisation field around proposed position of highwall in opencast pit.

Data obtained from piezometers and observation boreholes can be used to construct groundwater contour maps and flow nets. Groundwater contours are constructed from groundwater field levels related to a common datum plotted on a scale plan. Points of equal height are joined to form contours; flow lines are drawn at right angles from each contour. These give a plan view only, whereas in an actual cross-section the flow paths curve towards a discharge point, such as a spring, stream, or a pumping well.

The groundwater contour map may represent a water table surface or a potentiometric surface, which is derived from the geological and well information. The spacing of contours gives an indication of aquifer permeability values: when they are close together, it is indicative of low permeabilities, as a steep hydraulic gradient is needed to impel the water through the aquifer, whereas widely spaced contours indicate a more permeable aquifer. Flow lines indicate the overall direction of groundwater flow and where such flow is concentrating.

Current investigations utilise these hydrogeological data to produce computer models of the groundwater movement patterns likely to exist in a proposed mining site. This has the advantage that the model can be modified as more geological and hydrogeological data can be input into the system.

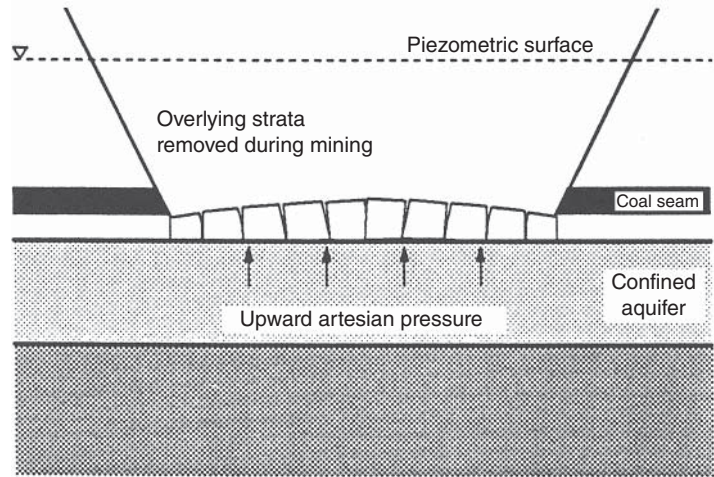
## 9.5 Groundwater Inflows in Mines

When an aquifer is excavated into during both open-pit and underground mine operations, groundwater may enter the mine. Some aquifers have a finite flow, and others are more persistent, particularly if the aquifer is being constantly recharged. Small aquifers may have insignificant water inflows, but in the case of open-pit operations, can seriously affect slope stability during dewatering and mining. Geological discontinuities, such as faults, joints, and bedding planes, provide either pathways for groundwater flow or, conversely, act as hydrogeological barriers. The inflow of water can be controlled and prevented by dewatering the mine. This dewatering will also reduce the inflow of groundwater from prolific recharge zones, such as rivers or lakes.

In open-pit mines, water may enter through the pit floor, caused by the upward pressure in a confined aquifer fracturing overlying rocks that have become thin due to the deepening of the pit (Figure 9.6). In underground mines, a similar phenomenon can occur where substantial stress relief has occurred. Floor heave is a serious problem, as it can result in the sudden flooding of the mine and cause disruption or even cessation of coal production. In old underground workings, flooding may have occurred that can produce a large



**Figure 9.6** Floor heave resulting from upward artesian pressure. *Source:* From Clarke (1995). Reproduced with permission of IEA Coal Research.



volume of water which may act unpredictably if the configuration of the workings is not fully known due to poor mining records.

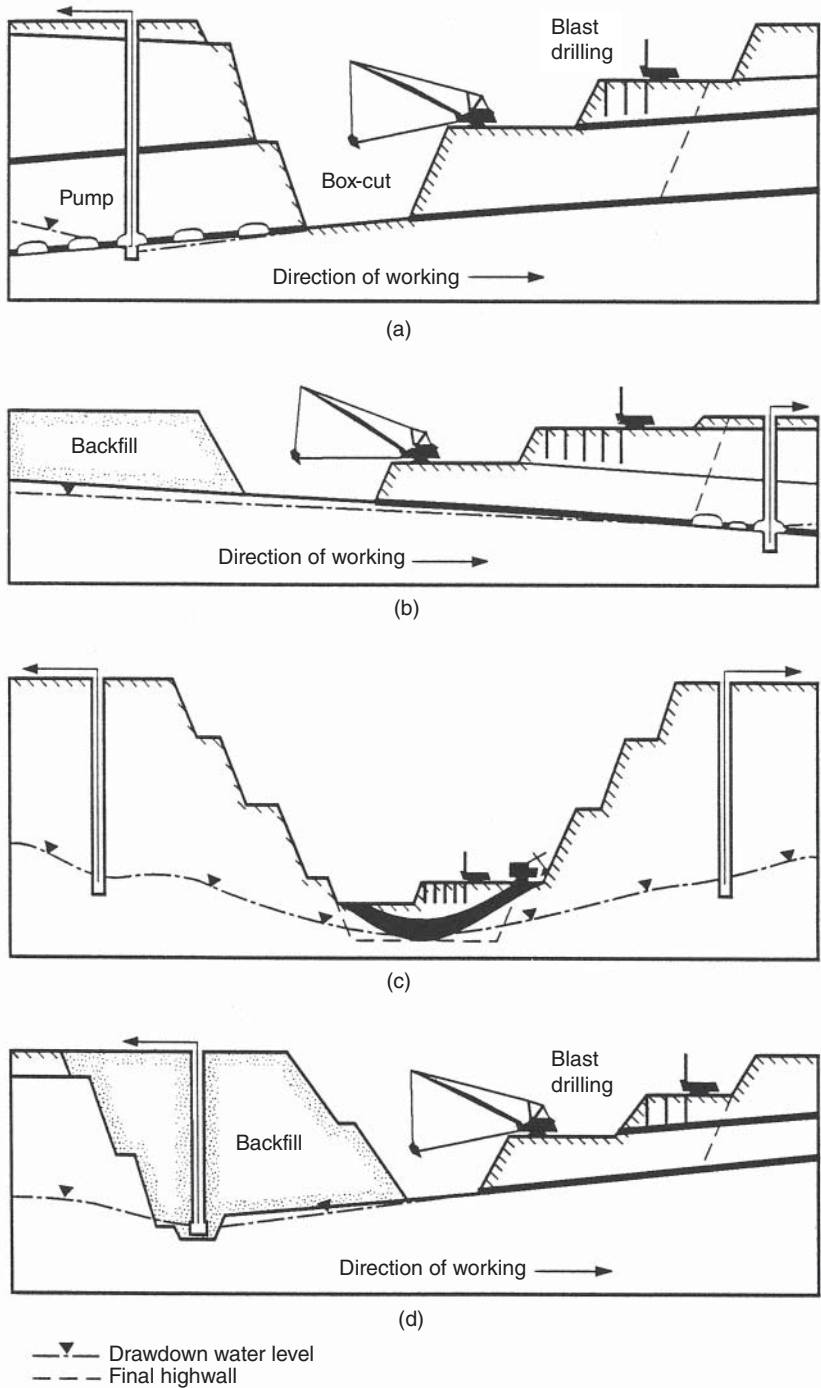
### 9.5.1 Dewatering of Open-pit Mines

Open-pit mines where aquifers are known to be present in the overburden, interburden, or immediately below the target depth of the pit have to be dewatered in order for the mine to operate successfully and safely. Dewatering is designed to stop water inflows into the pit, to maintain slope stability, and to protect groundwater for abstraction in the area around the mine workings. The choice of dewatering method is dependent on the geology and hydrology of the mine site.

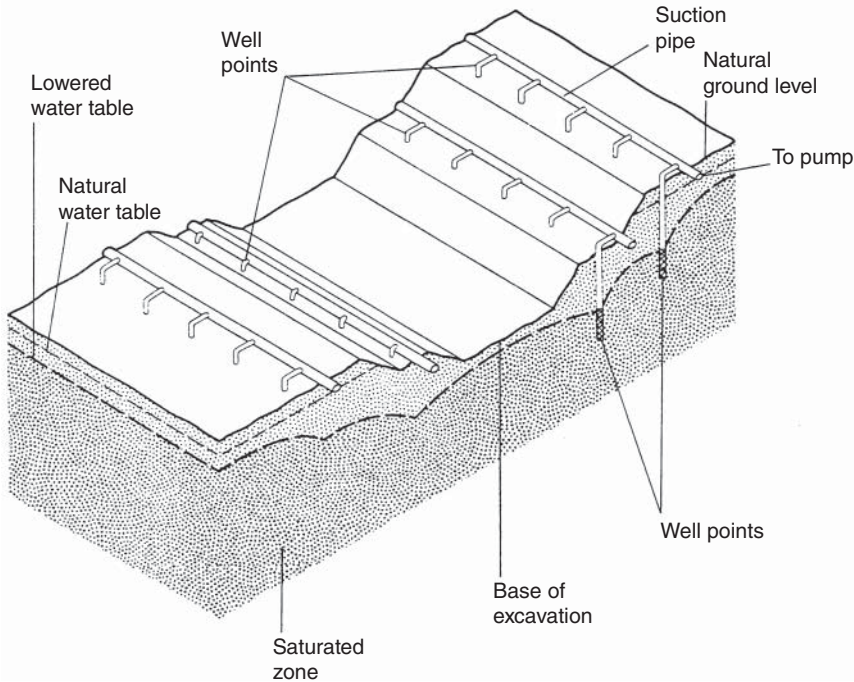
The standard situation is where the mine will intersect and excavate below the water table. This means that the groundwater level around the mine needs to be depressed to avoid flooding. This can be achieved by using pumps installed in a sump at pit bottom, but this does not allow for dewatering in advance of mining. Vertical wells are used extensively for dewatering, the exact pattern of wells being dependent upon the site-specific hydrogeological characteristics. Water can be removed from an aquifer by gravity or by pumping using submersible pumps in boreholes. Gravity wells drain water from an upper aquifer into a

lower one below the level of the pit bottom. Pumping wells raise water from the aquifer to the ground surface to be disposed of. The water is pumped at a rate that maintains a steady cone of depression, the level of which is constantly monitored by piezometers. The cone of depression will extend beyond the mine boundary into the surrounding area. Figure 9.7 shows the dewatering of open-pit mines using several methods of excavation (Clarke 1995); the wells are installed through the overburden, coals, and footwall sequence, and also in the backfilled box-cut area when up-dip mining. A series of interconnected well points may be used to lower the water table to a level below the proposed base of the excavations. In Figure 9.8, a two-stage dewatering scheme is depicted that is designed to lower the water table below the two levels of excavation in the site (Price 1996).

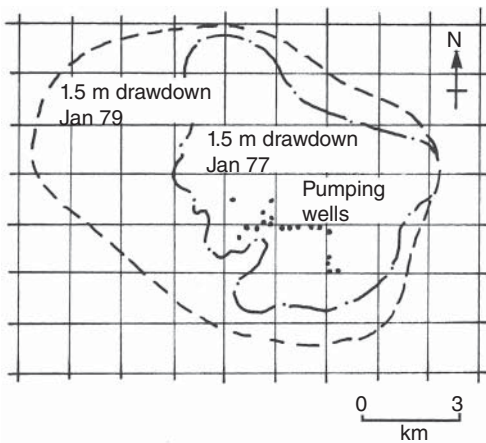
In Saskatchewan, Canada, dewatering is required in order to opencast mine brown coals overlain by 15–35 m of overburden. Here, the coal itself plus sands occurring both above and below the coal are the principal aquifers, with the coal acting as the major aquifer conducting water from the overburden sediments by means of joint and fissure flow. Dewatering tests included pumping from the coal and measuring the response in the overburden, and the measurement of the response



**Figure 9.7** Types of opencast mining in which advance dewatering methods are used: (a) Box-cut and retreat (up-dip) mining; (b) shallow down-dip mining; (c) open-pit mining; (d) up-dip mining, pumps installed in backfilled box-cut area. *Source:* From Clarke (1995). Reproduced with permission of IEA Coal Research.



**Figure 9.8** Site dewatering. A two-stage dewatering scheme using well points to lower the water table below the base of an excavation. *Source:* From Price (1996).



**Figure 9.9** Migration of the cone of depression due to pumping. *Source:* From Clifton (1987).

of potentiometric levels in the overburden during excavation of test pits. The water levels in the coal were rapidly drawn down by pumping from structural lows in the coal seam. Figure 9.9 shows the migration of the 5.0 m

drawdown contour from the pumping centres within the mine area. Pumping and test pit excavations caused relatively rapid reductions in potentiometric levels in the overburden. These tests showed that the overburden could be dewatered by directly pumping from the coal, with the secondary permeability being sufficient to provide drainage from the overburden if enough lead time was provided prior to mining. A lowering of potentiometric pressure by further dewatering would be required in the sands below the coal to eliminate floor heave in the pit (Clifton 1987).

Recent studies have concentrated on groundwater management in opencast brown-coal mines, chiefly in Europe and the former USSR.

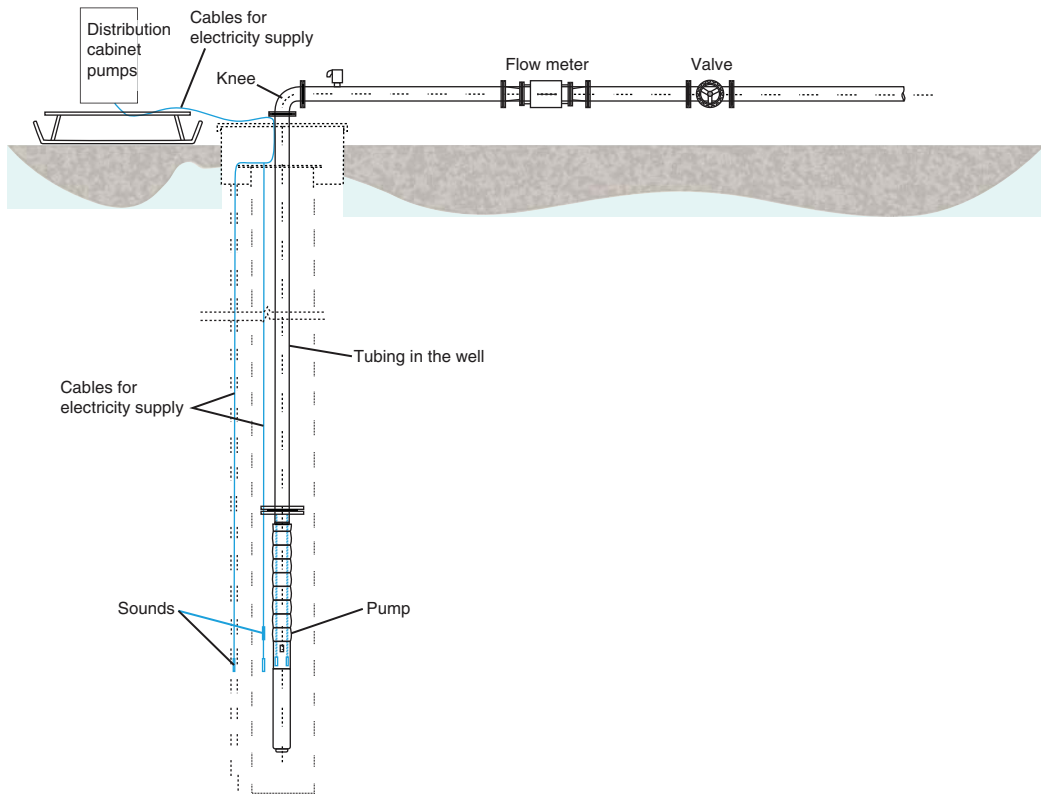
The opencast Drmno lignite mine in Serbia is located close to the confluence of the Danube and Mlava rivers. The lignite-bearing sequence is characterised by Quaternary and Palaeogene–Neogene sands and clays with lignite seams of varying thicknesses. The lignites

exhibit numerous discontinuities that increase their porosity. The rivers are in hydraulic continuity with the permeable sections of the sequence, and the regional water table is at around 50 m depth. As a consequence, these factors pose a serious hydrogeological risk to the mining operation (Pavlovic et al. 2008, 2012).

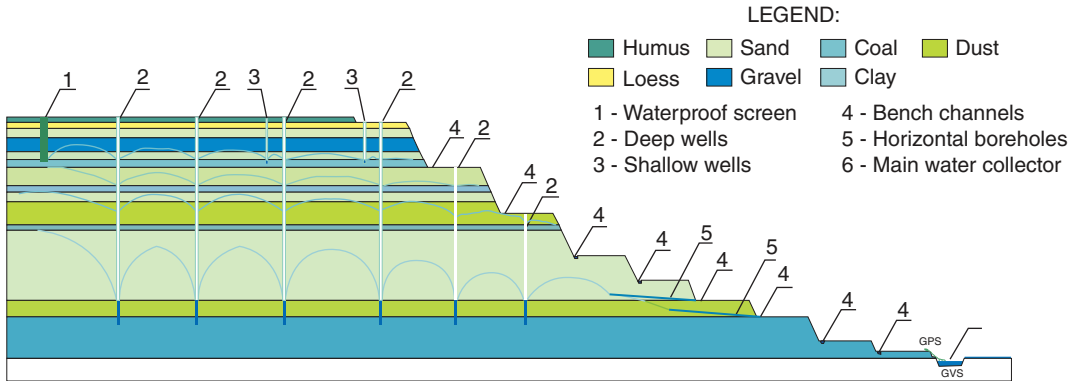
A groundwater model was developed and a piezometer layout for the main aquifer and lignite roof and floor strata was devised. Using the model and piezometer observations, the Drmno mine has developed a dewatering and water management scheme that consists of drainage wells (Figure 9.10), sited as the mine advances, a waterproof screen, and bench and drainage channel systems within the lignite workings (Figure 9.11). The waterproof screen installation is located between the mine and the River Danube and reaches up

to 60 m depth and is 1 m wide, filled with a bentonite–cement mixture. These measures are designed to ensure safe mining conditions for the life of the mine and for the area subsequent to lignite mining. Figure 9.12 shows the final layout of the dewatering and water management measures required to safeguard the mine from water incursion (Pavlovic et al. 2008, 2012).

In similar fashion, protection of the Radlievo opencast brown-coal mine, also in Serbia, against the inflow of groundwater from two aquifers consists of drainage wells, gravity pipelines, bench and drainage channels, water collectors, and pump stations. Lines of drainage wells will be constructed across the site as the mining operation progresses (Pavlovic 2012). In some cases, a large area may need to be dewatered for very large open-pit mines to create a large cone of depression in



**Figure 9.10** Schematic view of equipment for a dewatering well (Pavlovic 2012).



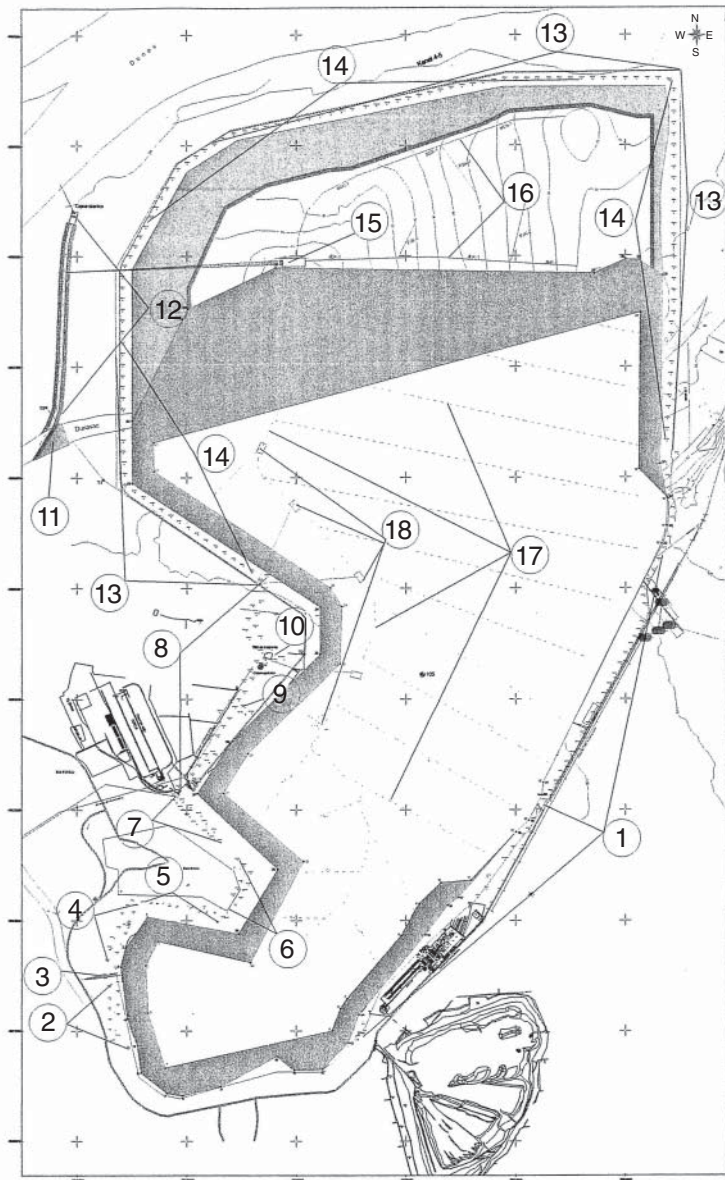
**Figure 9.11** Designed dewatering facilities for groundwater control at Drmno opencast mine, Serbia (Pavlovic 2012).

the piezometric surface. The brown-coal mines of Germany produce over 60 Mt per year and operate at depths up to 400 m. At Hambach, the pit area is up to 20 km<sup>2</sup> and the depth is up to 400 m. From this mine alone,  $350 \times 10^6$  m<sup>3</sup> of water has to be pumped each year to maintain the piezometric surface below a depth of 500 m. This has produced a cone of depression that extends for up to 30 km from the mine (Clarke 1995). The abstracted water is used for drinking water, cooling water in local power stations, industrial water supplies, irrigation, dust suppression within the mine, and to protect the wetland areas close to the mine. Any remaining water is reintroduced into local streams and rivers. The abstraction of groundwater in this fashion can cause significant water management problems. The resultant cone of depression can result in reduced flow from springs and rivers and lower levels in local wells, producing dry wells in some cases. The removal of water in the overburden can also produce settlement that can affect buildings, underground conduits, and roads and railways in the area. Such changes can have strong influences on local populations, e.g. in India, where there are thousands of villages dependent upon shallow wells for water supply. Local open-pit mining has to be conscious of this and provide alternative supplies and/or compensation. Figure 9.13 shows the predicted effect of dewatering an

area proposed for opencast mining in India. Figure 9.13a shows the existing water table, which slopes westwards towards a major water course. Figure 9.13b shows the probable effect of dewatering the proposed site after 10 days, which has resulted in the cone of depression lowering the level of the water table by a maximum of 22 m. After two years of pumping, the likely effect is shown in Figure 9.13c, where a maximum lowering of the water table is 64 m. The lowering of the water table will affect an area of 10–12 km<sup>2</sup> and will cause the local shallow wells to be dry. In this instance, the mining company will have to provide alternative sources of water for the local population.

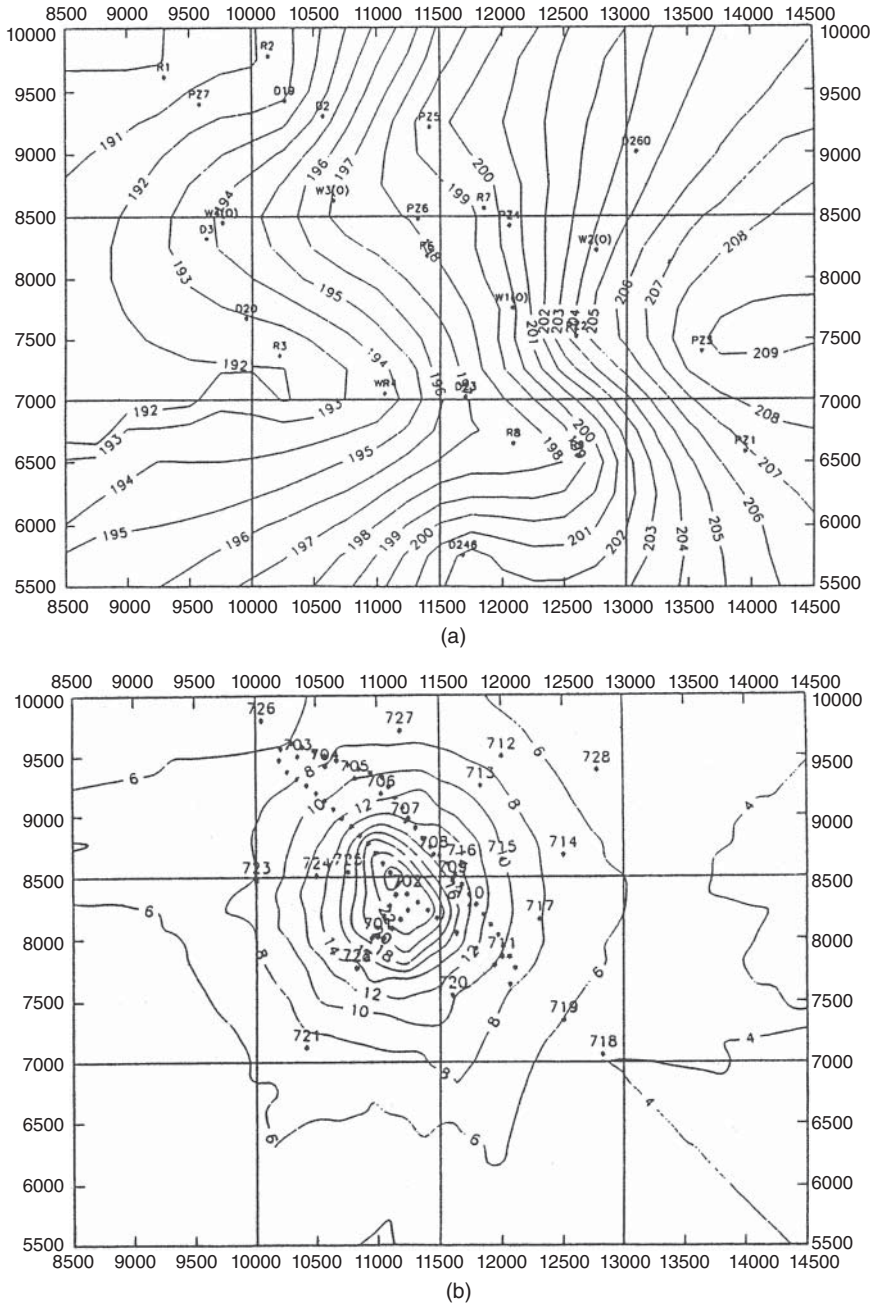
In mines where strata are steeply dipping, horizontal or inclined drains are used for dewatering. In areas where the geology is more complex, wells may be concentrated in areas with potential water problems; for example, in close proximity to a large fault. This will result in a local depression of the potentiometric surface, whereas the rest of the site may be served by a more general drawdown achieved by a grid of pump wells. Wells can also be used as depressurisation wells, to reduce hydraulic pressure in an aquifer, perhaps to improve slope stability or prevent floor heaving. A more specialised technique is the use of sealing walls, which involves the construction of an impermeable barrier between a prolific source of groundwater and the pit (see Figure 9.11).



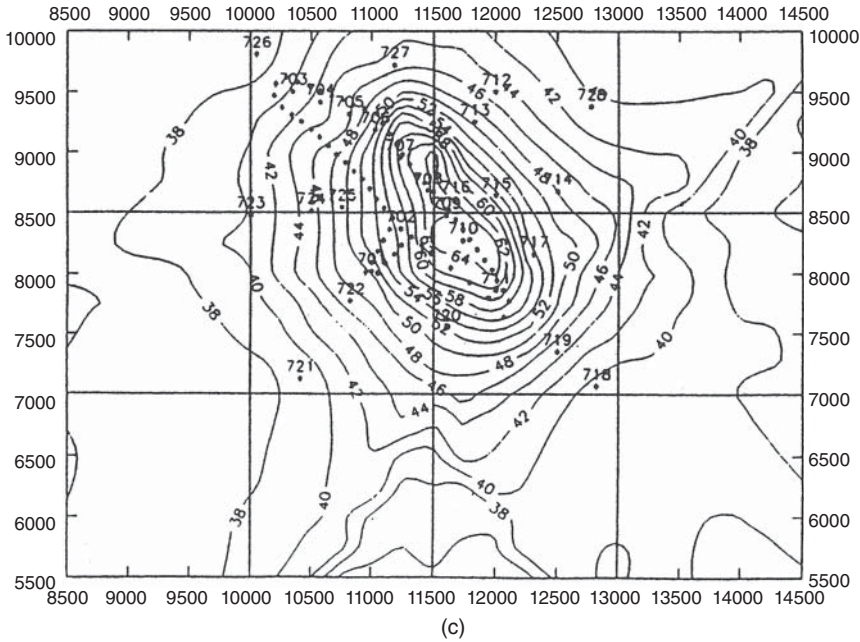


- |  |   |
|--|---|
| 1 - Barrage wells ŠLA with drain line OLŠLA      | 2 - Barrage wells LB-II with drain line OLB-II        |
| 3 - Flow of discharging line to Mlava            | 4 - Barrage wells LB-III with drain line OLB-III      |
| 5 - Barrage wells LC-4 with drain line OLC-4     | 6 - Barrage wells LB-IV with drain line OLB-IV        |
| 7 - Barrage wells LC-V with drain line OLC-V     | 8 - Barrage wells LB-V with drain line OLB-V          |
| 9 - Barrage wells LC-VII with drain line OLC-VII | 10 - Barrage wells LC-VIII' with drain line OLC-VIII' |
| 11 - Clay cap                                    | 12 - New Dunavac riverbed with pump station           |
| 13 - Waterproof screen                           | 14 - Wells line LEB with the screen                   |
| 15 - Water collector with pump station           | 16 - Bench channels                                   |
| 17 - Drainage channels                           | 18 - Drainage water collectors DV                     |

**Figure 9.12** System for protecting Drmno open-pit mine against water incursion at the end of mining.  
 Source: From Pavlovic et al. (2008). Reproduced with permission.



**Figure 9.13** Theoretical study to predict the decline in water level for a proposed opencast mine in India after selected periods of pumping: (a) original water table; (b) water table after pumping for 10 days; (c) water table after pumping for two years. Drawdown levels are predicted to increase to 64 m after pumping for two years.



**Figure 9.13** (Continued)

They are designed to intercept groundwater and prevent it discharging into the pit or cone of depression. Sealing walls do not actively remove groundwater but can reduce the pumping requirements needed to maintain the cone of depression.

Large-scale dewatering projects can produce land subsidence: unconsolidated or partially unconsolidated aquifers, or parts of aquifers that have been dewatered, are subject to lower pressures, which can cause an increase in effective stress within the sediments; this permits greater consolidation, and thus subsidence, particularly in the floor of the opencast pit.

Conversely, dewatering will be required to depressurise the pit floor, the highwall, and end wall of an opencast pit. This is carried out not only to reduce inflows of groundwater, but also to relieve pressure on the pit walls that may otherwise be subject to failure, particularly when shear zones and other discontinuities have been identified.

All of these measures are designed to control the groundwater conditions surrounding and

within opencast coalmine operations, and they are now a major factor in assessing the technical and financial viability of such mines.

### 9.5.2 Dewatering of Underground Mines

Numerous underground mining operations require the removal of water from the mine workings. Many mines, especially in old mining districts, are connected underground, so that pumping in one mine can control water levels in neighbouring mines. Water entering underground coal operations can be from several sources: from groundwater flow, from natural precipitation at the surface, and from abstractions from, for example, rivers and wells. The latter two occurrences are from infiltration of the workings via shafts and adits. Water may also enter from old abandoned workings situated in close proximity to the current mine.

In order to achieve effective mine drainage, detailed knowledge of the groundwater flow pattern within and around the mine

is desirable. To achieve this can be a problem, in that the changes in permeability of the coal-bearing sequence with depth, and the presence of unforeseen structural discontinuities, can make quantitative analysis difficult. In shallow mines (<100 m depth), rainfall, particularly tropical storms, can quickly affect water levels in the mine, as well as recharging groundwater that may enter mines at greater depths. The latter effect occurs after a time lag dependent on the permeability of the strata and rate of recharge.

The planning and design of effective mine drainage systems requires the best possible knowledge of underground water flow patterns and reliable forecasts of future yields. It is important that comprehensive records of mining be maintained and coordinated. All mine areas to be abandoned must be sealed off and clearly recorded on the mine plans; the locations of all mine interconnections, boreholes, shafts, and adits must also be plotted. In addition to this information, the multiplicity of seams, faults, and interseam connections must be recorded. Effective use of all of this data will allow long-term planning of underground water control to be implemented.

The amount of water that may be removed from mining areas can be considerable. In the UK, mine waters pumped from all active underground mines and adjacent abandoned workings has totalled up to  $1 \times 10^6 \text{ m}^3$  per day, and include the Nottinghamshire Coalfield totalling  $14 \times 10^6 \text{ m}^3$  in 1991 and the Durham Coalfield totalling  $70 \times 10^6 \text{ m}^3$  in 1994. Similarly, in Germany, the Ruhr Coalfield pumped  $114 \times 10^6 \text{ m}^3$  in 1994 (Clarke 1995). It is an ongoing operation and expense to drain water from underground workings, whether to facilitate working conditions or to minimise the potential hazards that water build-up can create.

### 9.5.3 Water Quality

During the mining operation it is necessary to monitor the quantities and qualities of

the waters flowing into the workings. Water pumped out of the mine should be utilised to the greatest practicable extent, but groundwater pumped from deep mine workings is often acidic or saline in nature and unsuitable for immediate discharge at the surface. Therefore, it may be necessary to improve the quality by treatment (see Section 12.2.1).

In opencast workings, surface waters and shallow groundwaters do not have the concentration of elements found in deep groundwaters. This is not to say that water quality can be taken for granted; quality measurements are just as important an exercise as for deep mines, it is just that the problems of water quality are usually more acute in underground workings.

The groundwater that flows into mine workings is normally free from suspended particles, but it always contains dissolved substances in concentrations related to the depth and hydrogeological conditions. However, such waters can become contaminated with fine-grained particles of coal and other lithotypes through contact with underground working operations.

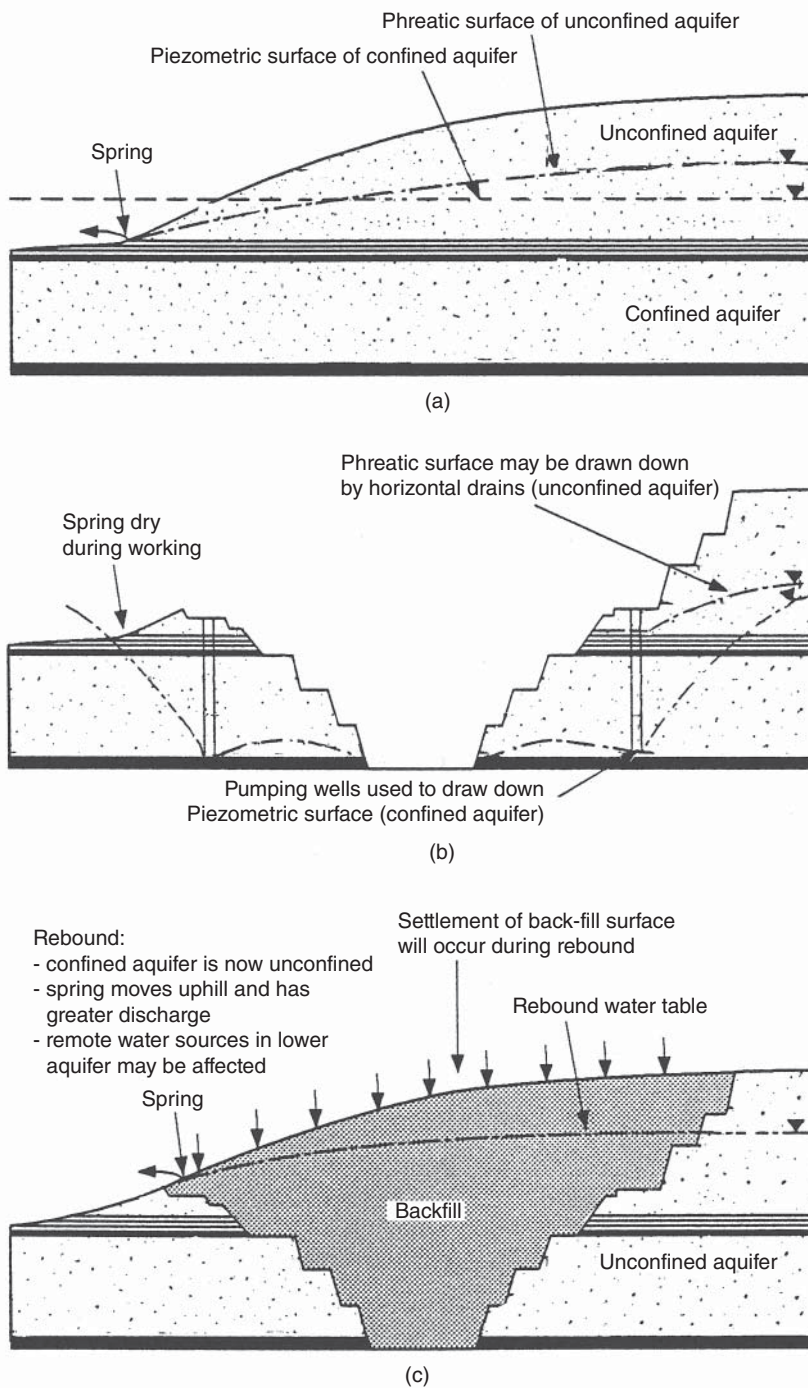
It is the contaminated groundwater that creates the most problems, and its treatment is an additional cost to the mining operation. Methods used are given in Section 12.2.1.

Oxygenated groundwater, when in contact with coal, results in the oxidation of the organic and inorganic constituents of the coal. Water is adsorbed onto the cleat faces, and if it is held there for a period of time it will oxidise that coal immediately in contact with it. This has the effect of reducing the coal quality (see Section 4.3.4).

## 9.6 Groundwater Rebound

In some traditional mining areas, regional dewatering of underground mines may have been in operation for over 100 years. Any change to this regime will have a profound effect on the water levels in the region. Figure 9.14 shows the effect of dewatering





**Figure 9.14** Rebound effect on the water table after cessation of opencast mining, showing changes to the local aquifer system: (a) before mining; (b) during mining; (c) after mining. *Source:* From Clarke (1995). Reproduced with permission of IEA Coal Research.



a mine and the resultant water table rebound effect once pumping has ceased. The original situation in Figure 9.14a shows an upper unconfined aquifer and a lower confined aquifer. Their respective piezometric surfaces were depressed by pumping during the mining operation, as shown in Figure 9.14b. However, after the mine has ceased working and been backfilled and pumping stopped, both aquifers are now unconfined and a water table rebound occurs, as shown in Figure 9.14c. The time taken for this to occur will depend upon the rate of recharge to the system and on the nature and permeability of the backfill material. Closure of deep mines in Yorkshire, UK, and the cessation of associated dewatering have led to concerns about the possible future pollution of groundwater and surface water resources once groundwater rebound is

complete. Burke and Younger (2000) used a computer model to predict the rate of groundwater recovery in the abandoned workings and the timing and flow rates of future surface discharges. Similar studies will be required in those mining areas that have now ceased to operate, or which are about to do so, that have been subject to dewatering over long periods of time. Cessation of pumping on a regional scale can mean the movement of original spring lines and variations in the flow rates at springs and abstraction points. In areas where mining has existed for a long period of time, it is not unknown for buildings erected ostensibly on 'dry' ground during the pumping period to be flooded at a later date when pumping has ceased and the water table has reached equilibrium at a higher elevation than that maintained during mining operations.

## 10

# Geology and Coal Mining

## 10.1 Introduction

The mining of coal and the methods used are well accounted for in the literature. Ward (1984) and Hartman (1992) give details of the technology used for both underground and opencast or surface mining. Only a brief outline is given here to highlight the importance of geological knowledge in mine planning and in mining operations.

The ability to mine coal is the direct result of the evaluation of geological data collected during the exploration phase of the project. The provision of values for coal reserves, coal qualities, and detailed geological conditions is essential for mine planning and process design.

Coalmining takes place in the geological medium; consequently, geological conditions have a profound influence throughout the life of a mining operation. In this context, coalmining geology is a concentrated extension of the exploration process (outlined in Chapter 6), in that greater detail is required covering a small area. In addition, all the geological support studies assume a greater significance, e.g. engineering geology and geotechnics, hydrogeology, coal analysis, and detailed geophysics.

The ultimate aim is to maintain the coalmine's production; one of the main reasons for failure is due to unreliable coal reserve estimates or unforeseen structural limitations.

The modern mining process itself is a highly mechanised operation, and the selection of the mine layout to optimise equipment use is based on the mine design, to which geology is a major input. The most cost-effective mine operation will influence the financial return made from the mine and, in the development stage, encourage investment from interested parties.

The mining of coal has taken place in Europe for over seven hundred years and in most other major coal-producing countries for over one hundred and fifty years, using a number of methods to extract coal from the ground. In modern-day mining there are two basic methods of mining coal: (i) underground mining, where coal seams occur at depth beneath the ground surface and are accessed by tunnels and/or shafts; and (ii) opencast or surface mining, where coal seams are close to the ground surface and can be accessed by direct excavation of the land surface.

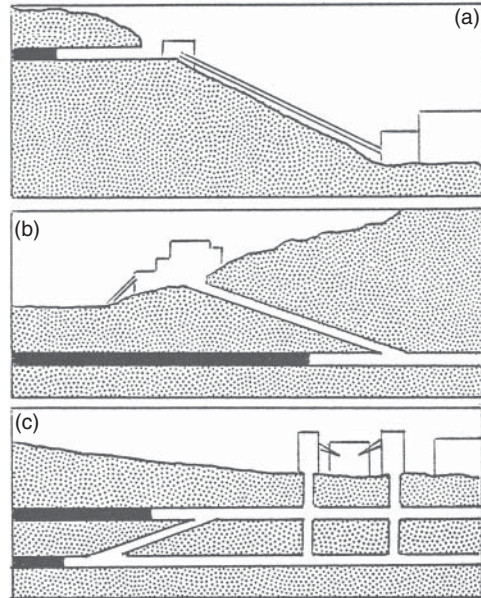
Coalmining developed as a result of the Industrial Revolution, when the demand for coal for industrial use, power, and shipping was suddenly accelerated. Those areas where this phenomenon first took place were Europe, North America, and the former USSR. Coals were mined by underground methods as the coals in these areas were situated at depth. Despite the fact that underground mining is practised worldwide and that the bulk of

the world's coal resources lie at depths only mineable by underground methods, the modern trend has been to increase the mining of coal by opencast methods. This is due to an increase in geological knowledge, cheaper methods of operation, and the ability to utilise coals of all ranks and qualities in the industrial process.

## 10.2 Underground Mining

To develop an underground coalmine, four main operations have to be carried out: (i) a shaft sinking and/or an adit drivage to reach the target coal seam(s) beneath the surface; (ii) the drivage of underground roadways, either within the coal seam or to access a coal seam; (iii) the excavation of working faces in order to extract coal; and (iv) to make provision for temporary underground storage of materials.

Once a mine design plan is complete, the first obstacle to be overcome is to access the target coal seam. This may be strongly influenced by surface topography: if the coal seam is exposed on a hillside, an adit, or tunnel, may be driven directly into the coal; if the coal seam is at depth beneath the hillside, an inclined adit may be driven down at an angle to intersect the coal seam. In areas where the topography is either flat or is a valley bottom and the coal seams are at considerable depth, a vertical shaft is sunk to access the coal (see Figure 10.1). The cost of sinking shafts to the level of the coal seam (or just below, to provide a water drainage sump) is often the largest single cost in developing an underground mine. Sinking vertical shafts over 700 m deep may take two years or more; the deepest shafts in the UK (comparable to coalmines anywhere) were approximately 1000 m deep. Additional costs are incurred when the shafts are sunk through porous sandstones or running sands; in such cases, the ground is first solidified by injecting cement through vertical boreholes (cementation method) or by circulating saline



**Figure 10.1** Methods of entry for underground mines: (a) in-seam adit; (b) inclined shaft; (c) vertical shafts. *Source:* Adapted from Ward (1984).

water at temperatures below the freezing point of fresh water through vertical boreholes to freeze the ground before shaft sinking (freezing method). In this case, the shaft walls are often reinforced by cast iron tubing to prevent water inflows when sinking has been completed and after the ice mass has thawed. Underground mines must have at least two points of entry; one is used for the intake of fresh air to the ventilation system, known as the 'downcast shaft', and the other is used for expelling the returned air, and is the 'upcast shaft'. The downcast shaft is also used for coal haulage and other access. The upcast shaft's primary function is for ventilation, but it also is an emergency exit. Shafts and adits are the most important capital item when opening underground mines. They provide all services for underground operations; these include fresh air, transportation of equipment and supplies, personnel traffic, power, communications, water supply and drainage, and, not least, the transportation and removal of coal from the mine. Depending on the depth of the mine,

shaft sinking may take up to 60% of the development time (Unrug 1992). Adits are chosen where possible in preference because of the lower cost and shorter construction time. In a number of mines, a combination of shaft and adit is used. The shaft diameter and hoisting depths need to be selected to accommodate the maximum design use of the mine. It is better to overdesign in the first stage of a mine than face a bottleneck in future years. In using adits equipped with belt transporters and using a maximum slope of 15.5°, the coal transportation system is uninterrupted all the way to the surface. However, there are economic limits to the adit length, and shafts can be less expensive for depths exceeding 350 m. Roadways in the mine may be excavated within the coal seam, as is the case in some mines in the USA and People's Republic of China (PRC); this makes development quicker and yields a coal tonnage at the same time, but it is only possible when the coal is 2–3 m thick and has a strong roof.

### 10.2.1 Geological Factors

Ideally, mine access should be central to the planned extraction area. In siting a shaft, all geological details of rock types, structure, and hydrogeology should be determined. In the case of adits into hillsides, hazardous sites where rock falls, landslides, or flooding can occur should be avoided.

Geological investigations for underground mines must include: (i) the identification of any mining hazards or breaks in coal seam(s) continuity, such as faults, igneous intrusions, washouts, and seam splitting, and any areas containing these features should be identified on the mine plan; (ii) the drawing up of plans showing thickness variations in the target seam(s) and thickness of strata between coal seams, together with coal quality trends obtained from borehole and mine sampling; and (iii) the geotechnical characteristics of the roof and floor strata for all seams that are planned to be extracted. The geotechnical behaviour of the roof and floor strata will

influence the type of support to be used, e.g. whether roof bolting will be possible by having a strong cohesive rock above the coal.

The incidence of faults in coal seams has a significant effect on the selection of mining systems and on the productivity. Major faults, with throws greater than 20 m, very often delineate mine boundaries or may cut out the coal entirely. Major faults are often associated with numerous minor faults, running roughly parallel. The effect of these faults is far more serious in underground mining than in surface mining operations. A fault of 1–5 m may be inconsequential in a surface mine but be a serious impediment in underground mining; for example, a fault with a displacement of 2 m can 'lose' a coal seam of 1.5 m. When this occurs, a new coal face has to be established at the new level after roadways or connections are made between the two seam levels. This may take several days or weeks, with a consequent loss in production. The problem is compounded if a series of faults is encountered and may result in the abandonment of working the seam in the problem area. Highly faulted coalfields often reflect a structural and metamorphic history that has increased the rank of the coal, as seen in the anthracite coalfields in South Wales, UK, North Vietnam, south-west China, and western Siberia. Igneous intrusions are found in a number of coalfields throughout the world. Where the intrusions are vertical, they are known as dykes and can be of various widths; dykes are often doleritic, and their effect is relatively local. They are difficult to locate underground and have a direct effect on coal quality, in that hot molten rock reduces the volatile content and 'cinders' the coal seam. Igneous intrusions in the horizontal plane above the coal seam or cutting through the coal seams are known as sills. The effect of sills on coal workings can be more subtle and dangerous. Sills are often several metres in thickness, very hard, and competent; such a roof may not 'cave' or subside regularly in long-wall workings, and thereby cause problems. In room-and-pillar workings, the strong roof has

led to the design of undersized pillars for roof support, which have eventually collapsed. In the 1960s, in a mine in South Africa, the whole of the underground workings collapsed virtually instantaneously when all the undersized pillars collapsed under a massive dolerite sill, with a catastrophic loss of life. Research following the disaster has led to empirical tables being available to mining engineers designing room-and-pillar workings for any combination of coal seam depth and thickness.

Other studies include the hydrogeological regime of the planned mine area and whether the mine is likely to be gassy. These studies will be part of the mine planning stage, but they will also extend as part of an ongoing programme during mine development and then continue during coalmining to ensure a continuing accessible coal reserve. The understanding of the geological parameters is essential to ensure successful high-productivity coalmining.

Technical development in underground mining has concentrated principally on coalmining by two systems: longwall mining and room-and-pillar mining. Both methods have developed to meet the criteria of reductions in the workforce and increased productivity, leading to reduced operating costs. The preference for either system relates to a number of factors: depth of mining, geological conditions, size of reserve area, availability of equipment, mining regulations, and, most importantly, availability of investment capital.

Suitable geological and geotechnical conditions are necessary for high-productivity mining, whichever of the two methods is chosen. If this is not the case, even with the best equipment and large reserves the mine will prove uneconomic to mine. Geotechnical conditions need to be such that development roadways and mining faces can be opened up rapidly.

## 10.2.2 Mining Methods

### 10.2.2.1 Longwall Mining

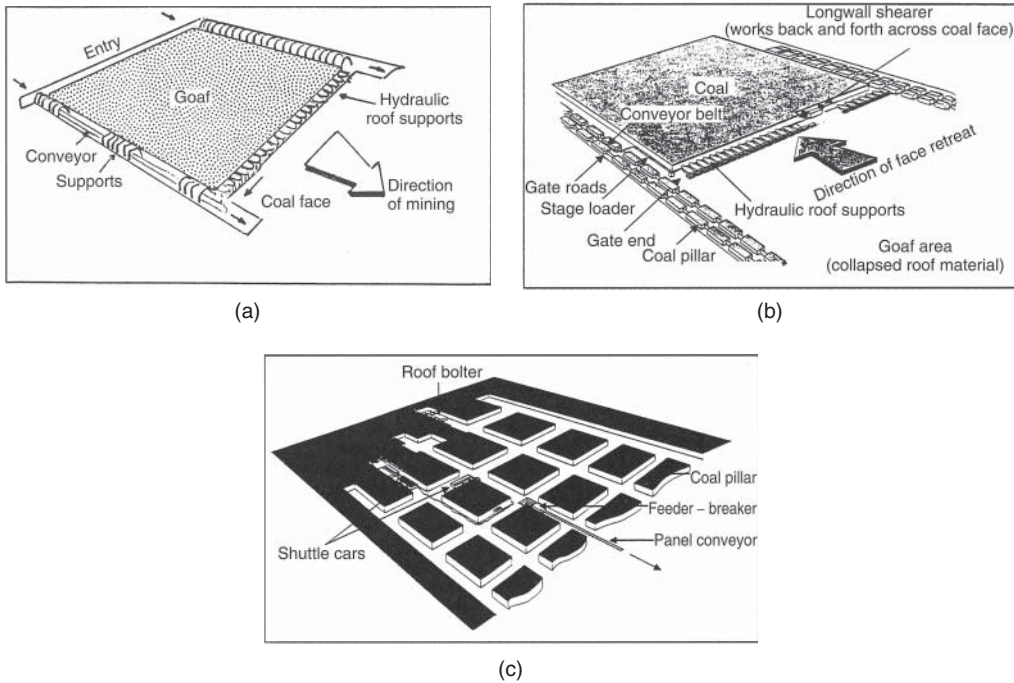
Modern longwall coalmining has developed over the last 60 years. It has a simple system

layout and is designed to be fully automatic and to provide continuous production. The planned area to be mined is divided into a series of elongate panels accessed from an entry roadway. Mining involves the removal of coal from a single face representing the width of the panel. The workable coal seam thickness is usually 1.5 m up to 4 m, depending on the size of the hydraulic supports. The coal is mined by either longwall advance, where the face is moved forwards into the coal away from the entry roadway (Figure 10.2a), or longwall retreat, where drivages are made around the edges of the selected panel area and the face is then worked back towards the entry roadway. The working area is protected by moveable hydraulic supports and overhead shields, which, as the equipment advances, protect the workforce from the roof, which collapses behind as the support is removed. Figure 10.2b shows the basic layout of the longwall retreat working model. In both cases the coal is cut by either a rotary drum shearer (Figure 10.3a) or a coal plough (Figure 10.3b), which is attached to the front of the roof supports and runs along a flexible chain conveyor. The use of coal ploughs in preference to shearers is more effective in thin coals, as they can cut different areas of the face at different cutting depths.

Longwall advance is used to mine thinner seams or while a retreat panel is being established. Longwall retreat is the more widely used method, with the advantages of collapsing the roof towards the main entry, which must be kept open. Then, when the panel is completely mined, the equipment can be transported more easily to the next panel. Another advantage is that, by driving access tunnels along the sides of the panel, any change in the predicted geology can be ascertained. This can limit the mineable length of the panel if the coal seam is severely dislocated by previously undetected faulting.

The panel width and length will depend on the geological conditions and on the capacities of the transportation, ventilation, and power equipment that can be supplied and installed.





**Figure 10.2** Methods of mining in underground mines: (a) longwall advance mining; (b) longwall retreat mining; (c) room-and-pillar mining. *Source:* b, c from Hunt and Bigby (1999).

In the USA, panel width is usually 120–293 m, and in UK it was 200–250 m; and panel lengths can be 600–4000 m. If the panel width is, for example, 50 m and the length less than 500 m, then this is referred to as shortwall mining, and in application it is intermediate between longwall and room-and-pillar mining.

From the point of view of economics, increasing panel width reduces the number of panels in a mine reserve area, which results in a reduction in development costs for panel entry drivages, an increase in the recovery level of coal due to fewer pillars, and an increase in the production of coal. However, if the panel width exceeds 300 m, further increases have less effect on coal production as the coal may have to be moved over longer distances. Increased panel width also increases the roof exposure time, creating the potential for roof fall between the face and the overhead shields.

Longwall mining of seams thicker than the height of the available supports will mean either leaving a portion of the seam behind

or mining the whole seam by two longwall faces progressively staggered, as shown in Figure 10.4a; alternatively, one longwall face can be used and the remaining coal above is collapsed and collected through sliding gates at the back of the powered supports (Figure 10.4b). Problems arise when the coal seam attenuates or splits. Because the face is set at a fixed height, any change to the seam configuration could mean cutting non-coal material with the coal. This is not desirable, particularly if the saleable product is run-of-mine coal. Also, the shearer can emit sparks if a quartzose sandstone is encountered; this is to be avoided in gassy mines. Figure 10.5 shows three possible instances of coal seam change in front of a longwall face; other changes may be small-scale faulting or washouts. Because these features are extremely local, the exploration geology may not have detected such changes. The use of in-seam seismic techniques has helped to minimise the loss of longwall faces by identifying conditions



(a)



(b)

**Figure 10.3** Longwall mining equipment. (a) Rotary drum shearer, Huabei mine, PRC. *Source:* Photograph by Dargo Associates Ltd. (b) Coal plough, Friedrich-Heinrich mine, Germany.

in advance of the longwall face, thus reducing failure rates.

Longwall mining is used extensively in the USA and western Europe, and has now become established in Australia, South Africa, the PRC, eastern Europe, India, and the Commonwealth of Independent States (CIS).

#### 10.2.2.2 Room-and-Pillar Mining

The room-and-pillar method is a type of open stoping used in near-horizontal strata in reasonably competent rock. The roof is supported

by pillars, and coal is extracted from square- or rectangular-shaped rooms or entries in the coal seam, leaving coal between the entries as pillars to support the roof (Figure 10.2c). The pillars are usually arranged in a regular pattern to simplify planning and operation. The rooms or entries are normally around 5 m wide, and the roof is supported by either steel or timber beams or by long metal rock bolts.

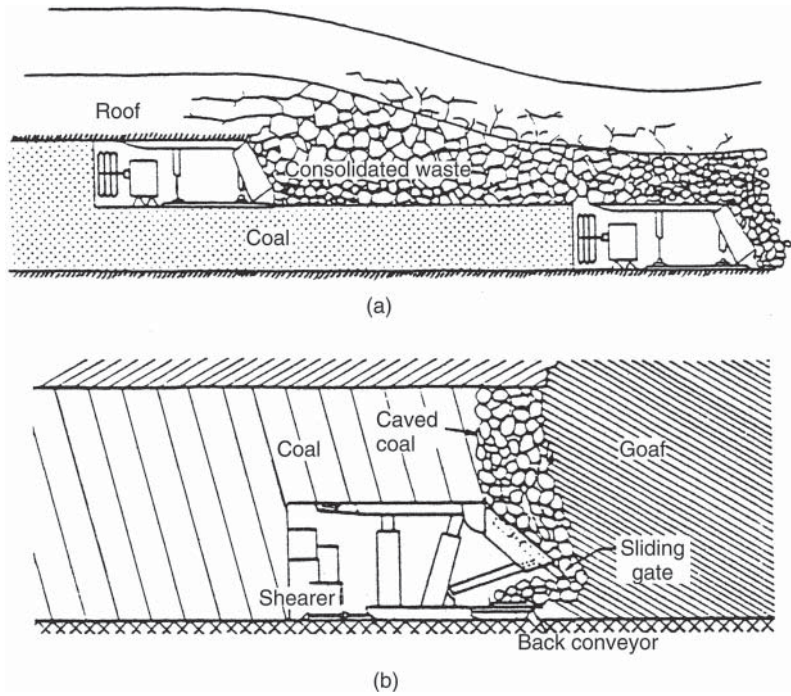
Coal is extracted by drilling and blasting the coal face, a system called ‘conventional mining’, or by mechanically cutting and loading the coal using a ‘continuous mining’ system.

In conventional mining, a mobile loader collects the broken coal after blasting and loads it onto a conveyor or shuttle car to be transferred to the main mine haulage system. In continuous mining, a single machine with a cutting head cuts the coal without requiring blasting (Figure 10.6). This machine also collects and moves the coal to a shuttle car or directly onto a conveyor for transfer to the main mine haulage system. The continuous miner provides a higher production rate than conventional mining and is the most widely used method in modern room-and-pillar coalmines. Room-and-pillar mining is most suitable for thick, shallow seams with strong roof and floor strata. The thickness of the seam is critical. Usually, seam thicknesses of 1–3 m are worked; however, the largest continuous miners can cut seams up to 4.8 m.

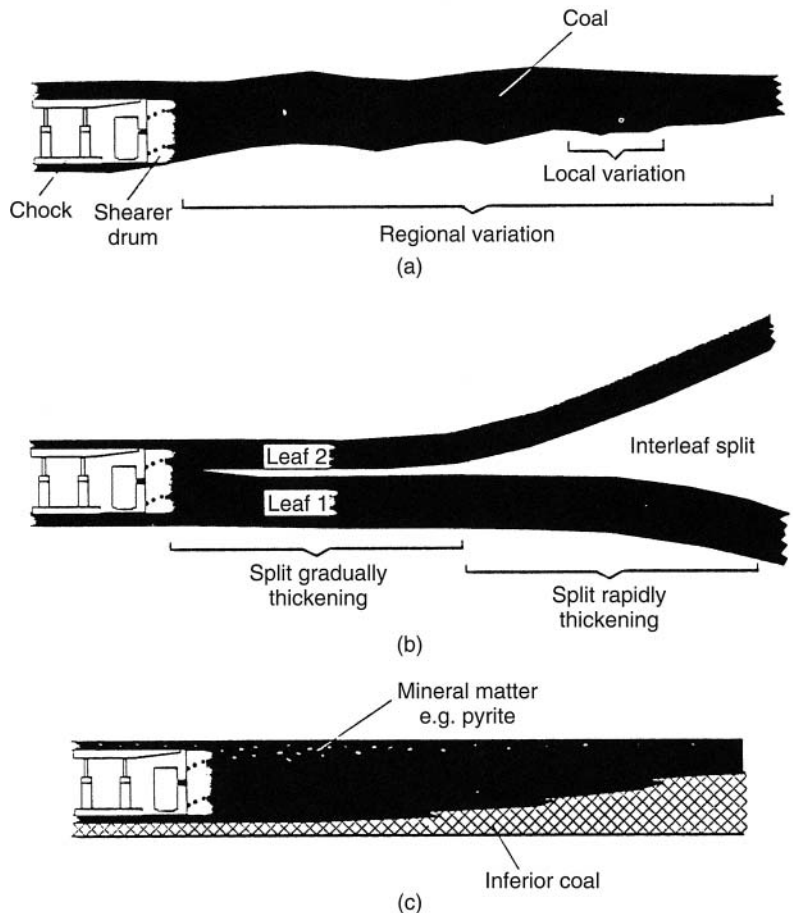
In shallow mines, up to 60% of the seam may be mined without pillar extraction, but larger pillars are necessary as the depth of the coal increases, and the percentage recovery is reduced. Once a district has been developed and access through it is no longer required, coal is then taken from the pillar areas. The roof is allowed to collapse into the abandoned area. Pillar extraction requires little additional equipment movement, such as conveyors, ventilation, and electrical cables.

In order to operate a productive mine, it is important that the prevailing geological conditions facilitate rapid development in seam roadways with the most economic level

**Figure 10.4** Methods for longwall mining in thick seams: (a) multiple slicing, using two longwall systems; (b) single longwall system combined with sub-level caving.



**Figure 10.5** Effects of changes in coal seam development on longwall mining. (a) Local thickness variation affects working of individual faces. (b) Coal seam splits affect quality of face product; rapid thickening may terminate mining. (c) Change in mineral content affects face product and increases wear on mining equipment. Source: From Fulton et al. (1995).







**Figure 10.6** EIMCO Dash 3 continuous miner in operation in Alabama, USA. *Source:* Photograph by courtesy of EIMCO, Bluefield, WV, USA.

of support. The ideal is for the mine to have small pillars and good roadway conditions. In modern room-and-pillar mines, roof support is by roof bolting with or without additional support, such as mesh or bench bars placed behind the anchor-bearing plates into which the bolt is threaded. Several types of bolt can be used. The column resin roof bolt is used to spread the bearing load over the whole length of the roof bolt. Together with the mechanical point anchor bolt, these are the most commonly used roof support. Bolts are usually 1.0–1.8 m long, of which 40–50% should penetrate a geotechnically sound anchor rock. Bolt spacing varies according to roof stability but is based on a 1 m square grid. The best development rates are where roof bolting is undertaken in good roof conditions, i.e. the continuous miner can advance 2–5 m without stopping to roof bolt, or alternatively, as with the most modern continuous miners, coal cutting can proceed simultaneously with on-board roof bolting.

Mechanised room-and-pillar mining has long been established in the USA, Australia, and South Africa, but it has rarely been used in Europe. In the past in the UK, underground coal workings that extended offshore were worked by the room-and-pillar method as longwall shearers had high levels of vibration. Room-and-pillar mining is also widespread in

Indian and Chinese coalmines with varying levels of mechanisation. In northern China, new fully mechanised underground mines use a combination of longwall and room and pillar. In India, mines operate modern longwall faces in one part of a mine and conventional room-and-pillar mining in another.

### 10.2.2.3 Stress Fields

The choice of mining support system will be determined by the type of rock strata and the loads acting on it, i.e. the stress field. The nature of the coal-bearing sequence is a complex mixture of mudstone, siltstone, sandstone, and coal, with occasional limestone or igneous intrusions, all of which are subject to varying amounts of stress. This is borne out by the presence of interbed shearing and major and minor structural discontinuities.

In shallow mines (<200 m deep) and in good mining conditions, the superincumbent strata do not impose strata control problems. However, strata control problems are not uncommon in deeper mines. The vertical component of stress is dependent on the depth of mining. In the UK, Germany, and Poland, mining depths are greater than 600 m, whereas in Australia, South Africa, and the USA they are usually less than 300 m (Hunt and Bigby 1999). Mining becomes progressively more difficult and expensive at greater depths. Larger areas of coal (pillars) have to be left between longwall panels or in room and pillar districts.

Horizontal stress is a critical factor affecting roof stability in underground mines. Mark and Gadde (2010) stated that the stress regimes encountered in underground coalmines are closely linked to those that exist deep in the Earth's crust. Rock mechanics research has shown that horizontal stresses can be up to three times greater than vertical stress, and the deeper the mine the greater the horizontal stress (Hoek and Brown 1980). The horizontal components of stress are the result of the regional tectonic framework. From studies of plate tectonics, geophysicists created the *World Stress Map* (Reinecker et al.



**Figure 10.7** World stress maps of USA, Australia, and northern Europe showing principal stress directions together with stress measurements taken in specific coalfield areas (Mark and Gadde 2010).

2005), which can be used to indicate regional stress fields. Mark and Gadde (2010) evaluated global trends in coalmine horizontal stress measurements and illustrated stress orientations in the USA, Australia, and northern Europe together with stress measurements taken in specific coalfield areas (Figure 10.7). The stress orientations measured in these coalfields closely reflect the regional stress trends. In the UK and Europe, a major horizontal stress component exists, orientated north-west–south-east. In Australia, however, horizontal stress orientations in the southern coalfields of New South Wales can vary from north to south in some areas and from east

to west in others; and the dominant stress direction in Western Australia is east–west. In the eastern USA, the greatest horizontal stress is east–north–east–west–south–west, whereas in the western USA the horizontal stress directions have a wide variation in direction across the region. The Mark and Gadde (2010) research findings indicated that the calculated depth gradient for the eastern USA was 1.6 times the vertical stress, in the Bowen Basin, Australia, it was 1.4 times the vertical stress, and for northern Europe (UK and Germany) it was 0.9 times the vertical stress. It is essential, therefore, that the direction of maximum horizontal stress is determined at an early stage,



as it has a profound effect on the orientation of longwall panels and road entry directions in room-and-pillar mines. In northern England, longwall panels aligned east–west were unsuccessful, but when changed to a north–south orientation they exhibited greatly improved conditions. Here, horizontal stress is redirected about the goaf rather than wholly transferred through cracked and caved ground. Vertical stresses are redistributed within the solid coal pillars and within the goaf, depending on extraction geometry. The stress distribution along one such longwall retreat panel is shown in Figure 10.8. The stress fields influencing a roadway can therefore vary over time due to mining activity and can be directly related to the geometry of the mining layout (Siddall and Gale 1992).

Another method of horizontal stress relief is the creation of ‘sacrificial roadways’. In a UK mine, a longwall panel was aligned  $110^\circ$  to the assumed major horizontal stress and suffered from severe floor heave and poor roof conditions, but once a new face line was driven, leaving a 4.5 m pillar between, conditions improved with little roof movement and floor lift. It could be seen that the failed roadway provided stress relief to the new roadway (Siddall and Gale 1992). In Australia, the selection of longwall panel locations was severely affected in a change to the orientation of the principal horizontal stress field. The change from north–south to east–west caused severe damage in the installation roadways and existing longwall panels. This was relieved by driving a stress-relief roadway parallel to the proposed installation roadway. During the drivage, bad roof conditions exhibiting shear failures were encountered due to the high lateral stress fields. This roadway was caved; when the new installation roadway was driven, excellent roof conditions prevailed. The roof remained intact during the whole longwall installation period, proving the success of the technique. In recent years, horizontal stress problems have been recognised in a number of mines in the USA and South Africa, which has increased the

importance of identifying the horizontal stress field as early as possible. Figure 10.9 shows an optimised longwall layout and working sequence for high horizontal-stress conditions (Hunt and Bigby 1999).

The stress field is also affected by multi-seam mining, where workings are closely spaced, and seams currently worked may overlie or underlie seams already worked out. Interaction effects may be severe and act as a constraint on further development.

In Australia, South Africa, and the USA, room-and-pillar mining has become fully mechanised with the use of continuous miners and roof bolting machines. The ability to rapidly develop in-seam roadways with limited support requires suitable geotechnical conditions. Rock stress conditions are therefore important, and the orientation of the planned room-and-pillar district should take the direction of horizontal stress into consideration. It may be necessary to change the configuration of the pillars to rectangular or even diamond-shaped, rather than remain square, in order to maintain favourable roadway orientation (Figure 10.10). The selection of pillar size is determined by the depth of working (vertical stress) and the position of shear zones around the coal seam. Roof conditions may determine the minimum pillar size rather than pillar strength (Hunt and Bigby 1999).

Horizontal stress can be measured by drilling into the seam roof, installing a measuring device, and then drilling a larger hole around the first one. The second hole relieves the stress and allows the rock to expand, the amount of expansion allowing the original stress field to be measured. Other methods include pressurising a drill hole with fluid until the rock fractures, the fracture pressure and fracture orientation are then measured. Indications of horizontal stress can be simply observed by checking shifts in, for example, bolt holes and fixed equipment.

The use of downhole geophysics has led to the realisation that horizontal stress regimes

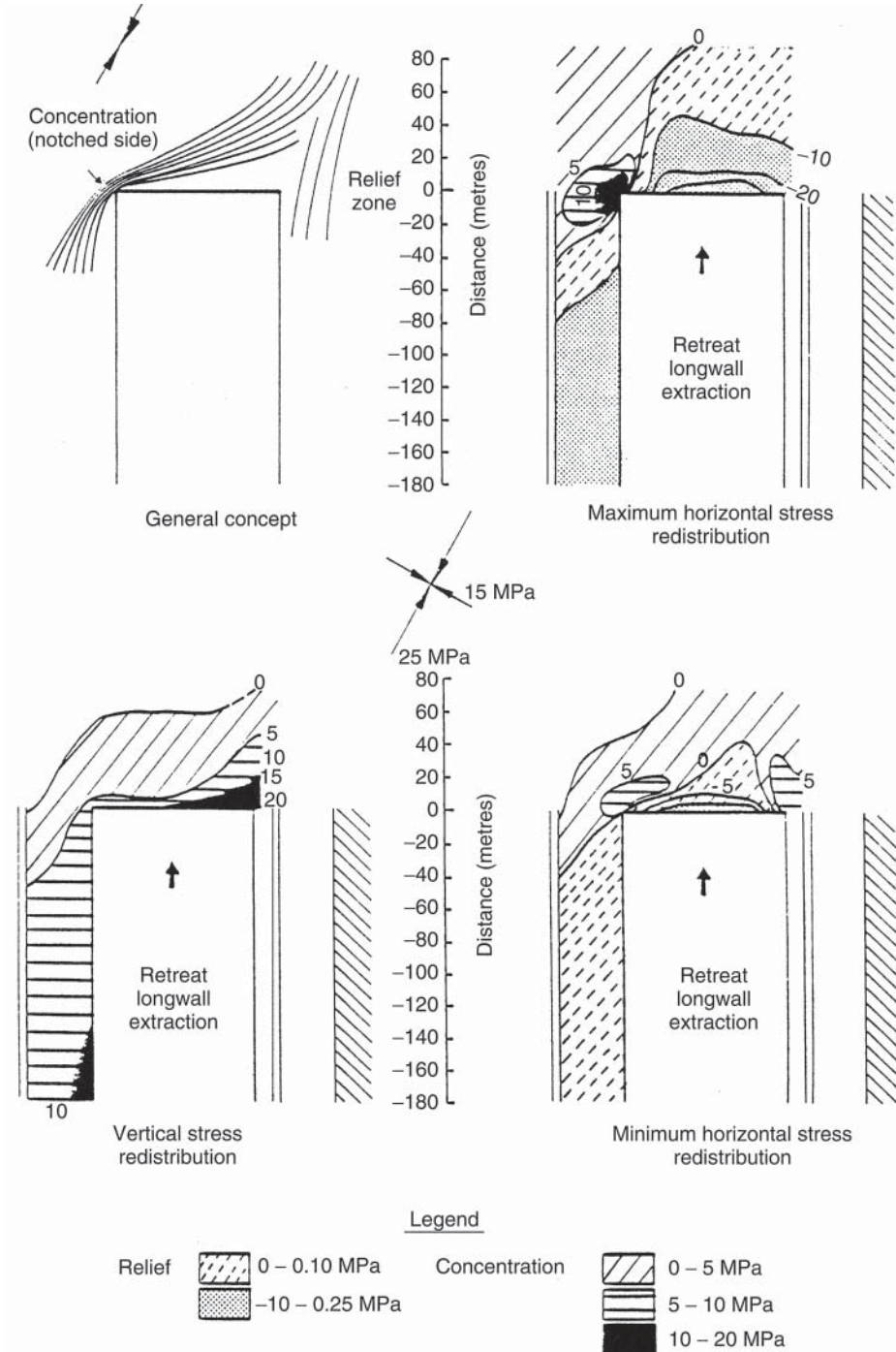
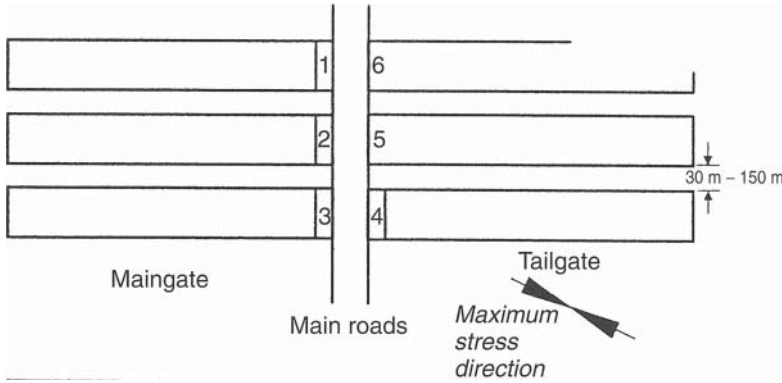
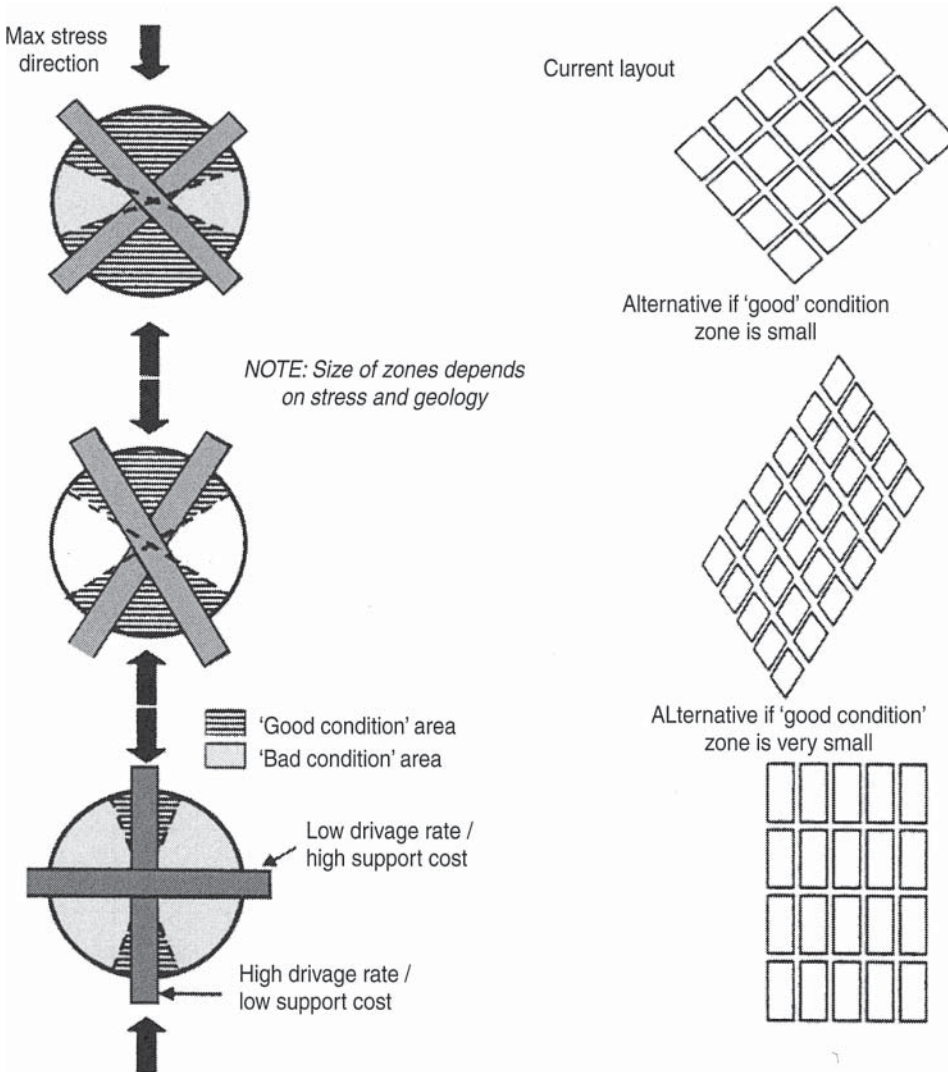


Figure 10.8 Stress redistribution about a retreating longwall panel. Source: From Siddall and Gale (1992).



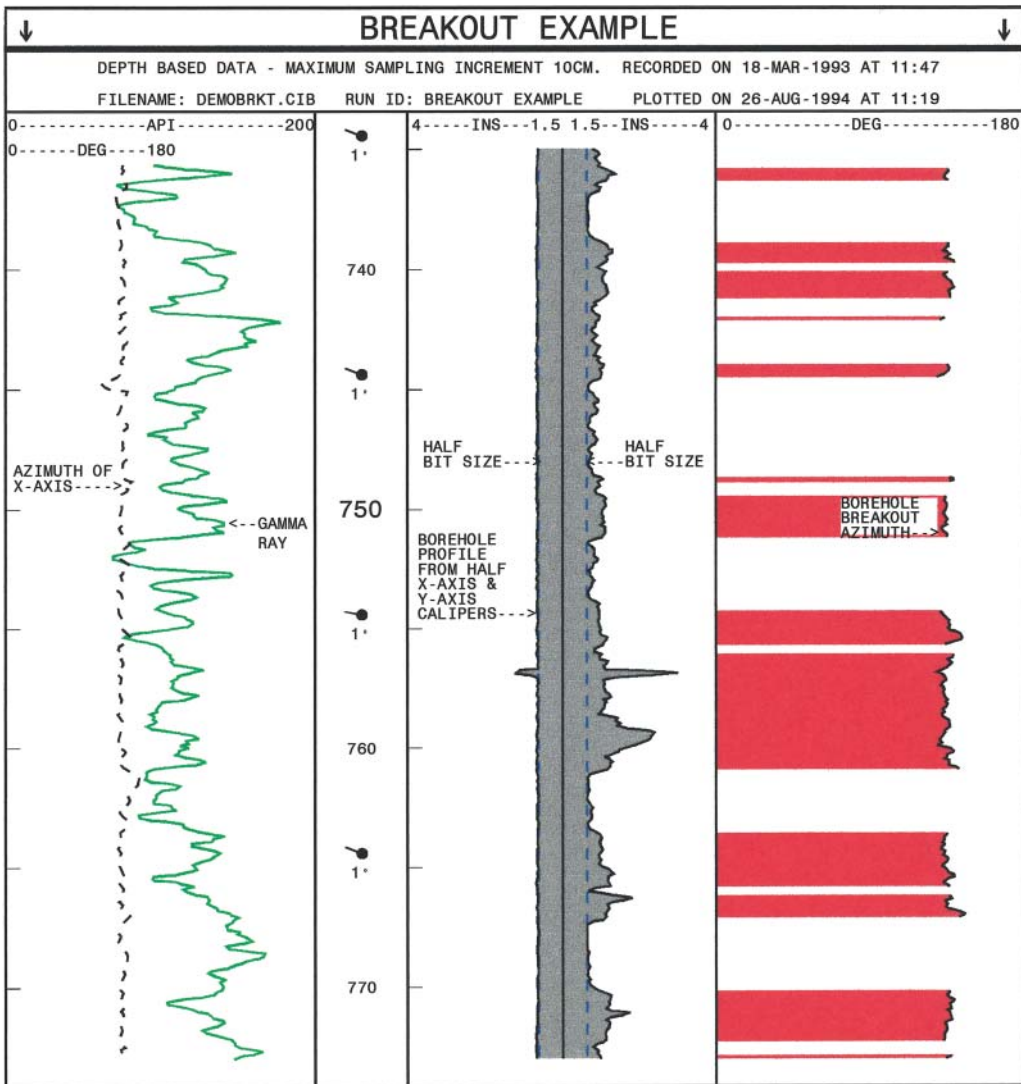
**Figure 10.9** Optimised longwall layout and working sequence for high horizontal-stress conditions. *Source:* From Hunt and Bigby (1999).



**Figure 10.10** Possible layouts to minimise horizontal-stress effects for a room-and-pillar district. *Source:* From Hunt and Bigby (1999).

may be recognised and their orientations measured from the nature of associated breakouts. Breakouts are indicated by increases in borehole diameter along one axis. Boreholes elongate in a direction perpendicular to the maximum horizontal stress orientation and are measured using  $x$ -axis and  $y$ -axis calipers together with borehole verticality data. Figure 10.11 is a borehole breakout log showing minimum and maximum callipers. In

recent years, the use of acoustic scanning tools has produced high-resolution formation images in boreholes, and breakouts can be identified and plotted using this technique (see Figure 8.31). In the UK, an average breakout orientation of  $54^{\circ}/234^{\circ}$  has been identified in conjunction with minimum stress orientation measurements from other techniques such as hydrofracturing and overcoring (Brereton and Evans 1987).



**Figure 10.11** Borehole breakout log showing minimum and maximum callipers. Breakout is identified by rock spalling or an increase in borehole diameter. *Source:* Reproduced with permission of Reeves Oilfield Services Ltd.

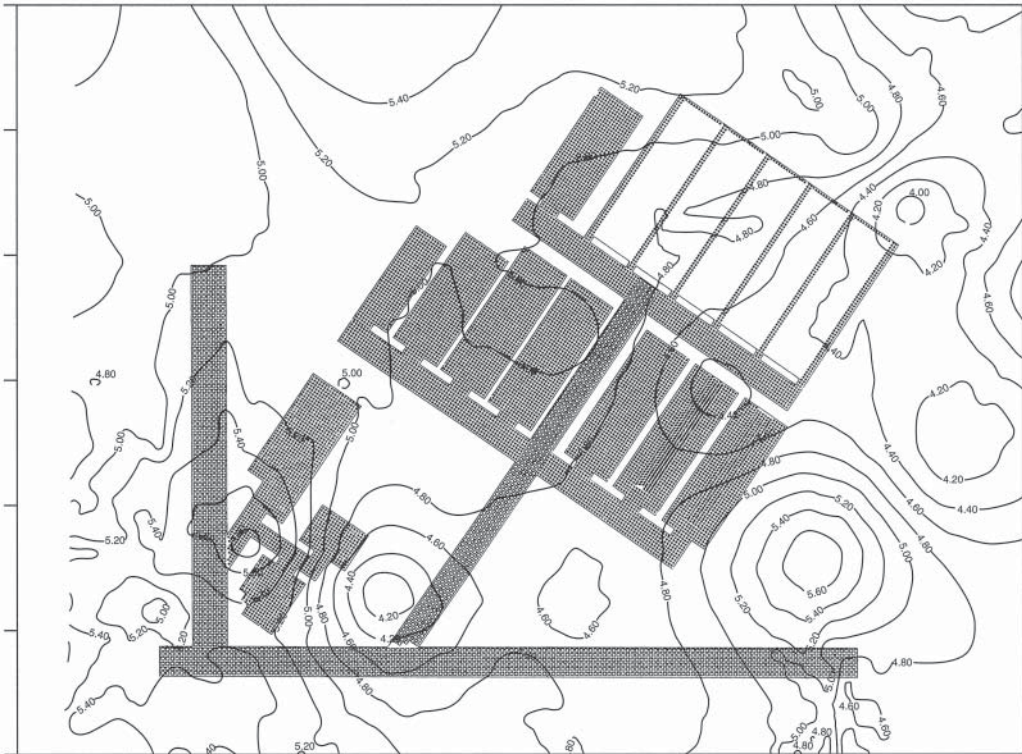


Underground mine planning may involve building upon previously developed mine layouts or designing a new mine area. Existing data are inputted to create a panel design, the object being to create a panel design that can be constructed, saved, copied, and modified as many times as required; this is stored in a panel library (Hartman 1992). Panel designs will differ for room-and-pillar mining and longwall mining. Room-and-pillar mines require parameters such as the pillar configuration, headings, and cross-cut dimensions in order to generate the panel design. Longwall mines require the dimensions of the longwall block, pillar configuration for the gate roads, and the dimensions of the barrier pillar. The panel can then be modified using interactive graphics.

The mine layout design is achieved by combining the digitised existing mine plan with the interactive graphics design. Once a panel is

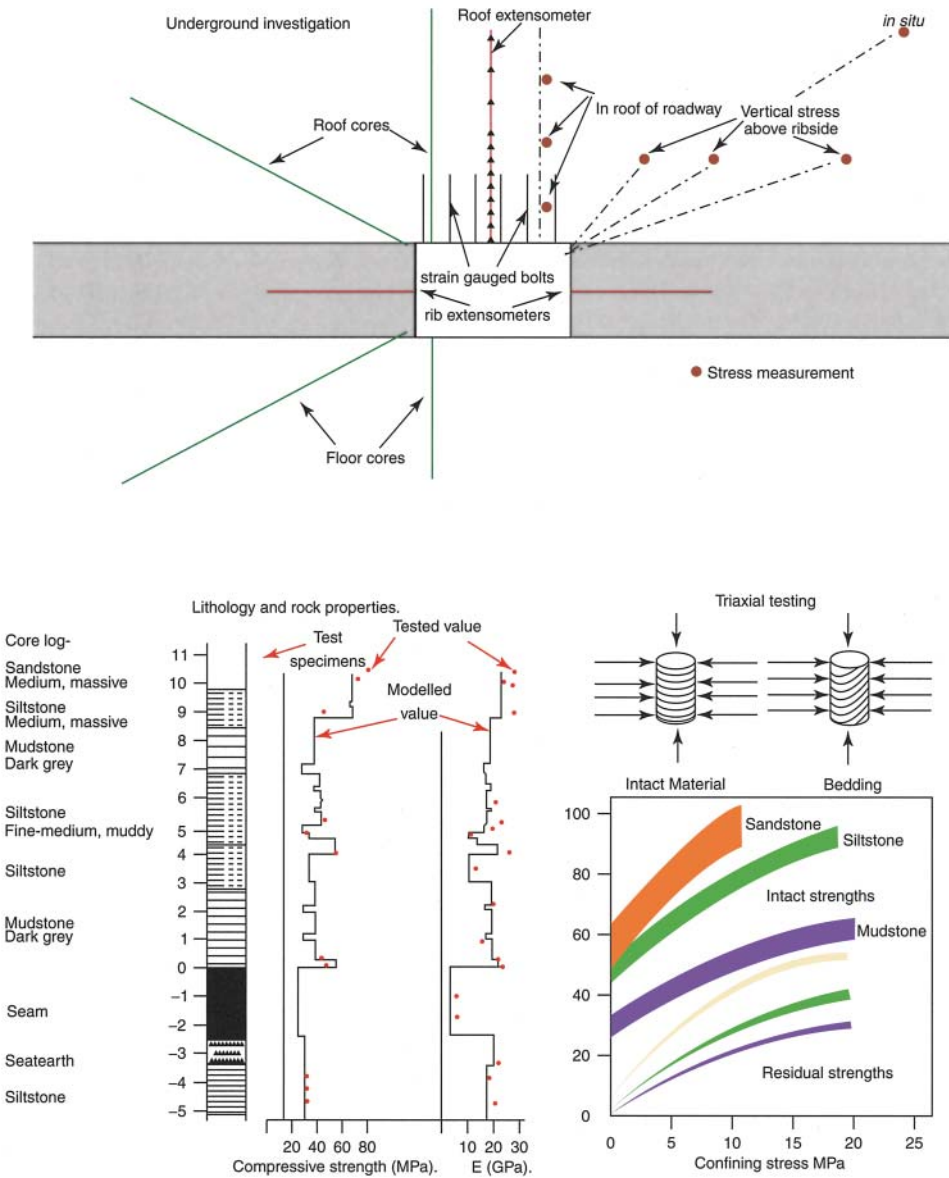
selected from the panel library, it can be placed on the layout at any orientation to other panel designs. In existing operations, the relevant portions of the current mine plan are digitised, pillar configuration is entered using an interactive menu program, and the pillars are automatically generated. Figure 10.12 shows a computer-generated final layout design showing selected pillar design for room-and-pillar and longwall panels.

Computer modelling is also used to carry out strata control and reinforcement design studies. The necessary data on the rock properties and in-situ stresses has to be compiled and fed into the computer model. The range of underground measurements and laboratory tests normally undertaken for model generation is shown in Figure 10.13. The residual strength properties of each rock unit are determined; these include strength properties for intact



**Figure 10.12** Final layout design for an underground mine development, with superimposed contours. Source: Reproduced with permission of Society for Mining, Metallurgy & Exploration Inc. ([www.smenet.org](http://www.smenet.org)).





**Figure 10.13** The range of underground measurements and laboratory rock tests normally undertaken for model generation. *Source:* Reproduced from Garratt (1999).

rock and rock showing discontinuities. These are assigned to each modelled strata unit, and the model is built up in layers. The in-situ stresses in the model are initialized on the basis of the expected cover load for a given depth and the results from the in-situ stress measurement. The nature and magnitude

of roadway displacements – as indicated by the use of extensometers and by how the stresses around the roadway have changed from pre-mining levels, indicated by stress measurement results – will be characteristic of the deformation mechanisms around the roadway. When using computer simulation to select

a reinforcement system such as rock bolts, it is important to consider the effects on roadway behaviour and the likely loads to which the system will be subjected. The modelled reinforcement behaviour is verified against strain-gauged bolt data obtained underground to ensure proper verification of the simulation.

It is now possible to model the predicted and actual behaviour of coal-bearing strata, to assess both longwall panel and roadway orientation, and to identify support requirements and optimum pillar size. This improves the accuracy of interpreting coalmine geotechnical properties; this will reduce the risk factor in underground coalmining, with obvious commercial advantages.

Such information is used to plot progress on the mine plan. These programs are capable of maintaining records of actual mining operations. They can perform day-to-day monitoring of operations, to include use of personnel and equipment as well as production and maintenance.

#### 10.2.2.4 Coal Bursts

One of the negative effects of stress in underground workings is the occurrence of coal bursts. Increases in depth in mining operations (600–1000 m) has resulted in a significant number of coal bursts. These have occurred principally in Australia, the USA, the PRC, Poland, the Czech Republic, Kazakhstan, and the UK. The termination of major underground coalmining in the UK has meant that coal bursts are no longer a hazard there.

Coal bursts are dynamic rock failure involving a sudden release of energy stored in the rock mass due to the disturbance of an unstable state of equilibrium (Zhang et al. 2017). This phenomenon has led to safety and production risks in mining operations. The energy levels and velocity of rock and coal material involved in a coal burst can cause a high level of damage to, or even destruction of, conventional ground support elements such as bolts, mesh, and steel arch supports, as well as endangering the lives of mining personnel.

Coal bursts have occurred in a variety of mining systems, most commonly in development, pillar extraction, and during longwall retreat. Geological factors include depth of mining, structural elements in surrounding strata, coal roof and floor characteristics, and presence and behaviour of gas. Studies of the coal gas content and its relationship to coal bursts have been carried out in the PRC (Xue et al. 2014; Lei et al. 2015) and in Australia (Black 2018), where geological structure and high gas content have contributed to coal bursts being a serious threat to mining. Zhang et al. (2017), in their review of coal bursts both in Australia and internationally, identified strong, rigid roof and floor strata surrounding a coal seam as a leading contributor, providing more likely conditions for coal bursts than coals in a sequence of weak strata. Depth of mining is also a common factor, producing greater increases in vertical stress on pillars and faces. Coal bursts are the result of the relationship between the highest principal stress (vertical or horizontal) and the geometry of the roadways and working areas. Orientation of the principal stress relative to the face strike or pillar long axis has been shown to be a major factor in the incidence of coal bursts, plus the fact that underground excavations cause the redistribution and concentration of the in-situ stress on the working faces and pillars. Therefore, monitoring and control of coal bursts in high-risk mines is essential for mining safety. Zhang et al. (2017) described two types of prevention techniques: (i) passive prevention by design, and (ii) active prevention by operational controls. The aim is to forecast timing, location, and scale of coal burst events. Passive prevention focuses on gate road design, pillar design, and mine layout. In some countries (e.g. the PRC and Europe), artificial roadways are constructed at the edge of longwall panels; the removal of large pillars reduces coal burst risk. Another method is to mine another coal seam above or below the target seam, as this reduces the stress in the target seam and surrounding area; this method is widely

used in the PRC for multi-seam mining. Active prevention includes fracturing, drilling, and blasting to destress the coal face; however, if there is potential for gas, those areas have to be defined and gas drainage applied. In Australia, gas content is used to assess coal burst risk; however, as mines develop those areas of increased gas content and reduced permeability, the effectiveness of gas drainage in lowering the gas content of coals is reduced (Black 2018). Other measures taken are to strengthen the mine support systems.

In the USA, the use of empirical or numerical computer models, referred to as analysis of multiple seam stability (AMSS), can identify potentially high stress zones due to multiple seam mining. Overall risk levels can be assigned to future mining areas: low risk, green zones; higher risk, yellow zones; and greatest risk, orange zones. Table 10.1 shows the risk factors and level of risk for longwall mines in the USA (Mark and Gauna 2016).

It is clear that coal bursts are produced by the combination of geology, stress, and mining conditions and that there is no one set of conditions that characterise this phenomenon.

#### 10.2.2.5 Strata and Air Temperatures

In underground mines, the virgin rock temperature increases with depth, which can make mining conditions uncomfortable. A typical temperature gradient is 30 °C per 1000 m depth (as in the UK). In deep coalmines, the temperature is kept at reasonable levels by increased air flows (ventilation) of colder air from the surface. As underground workings get further from the ventilation shafts, there is more time for heat transfer from the surrounding strata into the airways. Hence, the longer the air has to travel to the coal face, the nearer the air temperature will approach that of the virgin rock temperature. Deep mines may employ booster fans underground to increase the speed of ventilating air and reduce the temperature increase.

**Table 10.1** Coal burst risk analysis for longwall mining in the USA.

Risk factor	Level of factor		
	Low	Moderate	High
Depth of cover	<360 m	360–600 m	>600 m
Pillar design	Development only, meets National Institute for Occupational Safety and Health or other criteria	Longwall mines should use yield, abutment-yield, or interpanel barrier pillars as appropriate for depth and geology	
Multiple seam interaction	AMSS condition: 'green'	AMSS condition: 'yellow'	Inadequate maps or remnant surrounded by gob (AMSS condition: 'red')
Roof condition	Weak shale or similar, no massive strata within 15 m	Typical Western USA or Central Appalachian stratigraphy	Strong, thick, and massive strata near the seam
Floor condition	Claystone or similar, no massive strata within 15 m	Typical Western USA or Central Appalachian stratigraphy	Strong, thick, and massive strata near the seam
Other geological factors			Sandstone channels, faults or fracture zones, seam dips, rapid topographic changes
Past history of bursts	No burst history in the seam	Burst history in the seam	Burst history in the mine

Source: From Mark and Gauna (2015).

### 10.2.2.6 Spontaneous Combustion

Coal seams that are exposed to air for long periods of time, e.g. in main roadways, will oxidise, assisted by the air flow system in the mine. Oxygen provided in this way accelerates oxidation of the coal (see Section 4.3.4) and increases the propensity for spontaneous combustion, particularly in lower rank coals. In underground workings, extinguishment of coal seam spontaneous combustion is difficult as it is not easy to remove and/or dissipate the large amount of heat generated. Methods such as grouting and sealing off oxygen inflows by means of muds, gels, and inorganic foams can be applied, but mines are notoriously difficult to seal, particularly where there are large areas of goaf and disused workings. Liu et al. (2017) listed the application of these methods for controlling spontaneous combustion in multi-seam operations in the PRC, which have met with success in a number of coalfield areas. There are some coalfields, such as Jharia in India, where underground fires have been burning for over 75 years and estimated to have consumed  $40 \times 10^6$  t of coal; and in the PRC, fires have consumed up to  $200 \times 10^6$  t of coal with no prospect of containing the fires.

## 10.3 Surface Mining

Surface mining (also referred to as opencast, open-pit, or open-cut mining) describes the accessing of a coal seam or seams from the ground surface by excavating all of the material above, between, and including the coal seam(s).

Surface mining has a number of advantages over underground mining: a higher degree of geological certainty, lower capital costs, lower operating costs, and a safer mining environment for personnel. The major disadvantages are the restriction of depth due to cost and geotechnical limits, surface water and ground-water influences, the direct effect of climate, and the commitment to restore the land to meet environmental requirements.

Ward (1984) divided surface mining into two types:

*Strip mining*, where the material above the coal, known as overburden, is excavated and deposited in one operation adjacent to the working face. This method is usually employed along the outcrop of a coal seam or a number of seams. The strip mine will extend along the strike for long distances, but only a short distance down dip.

*Opencast mining*, where the overburden is taken away from the working face and dumped elsewhere on the mine property. Opencast mining is the best suited for mining thick seams or a series of seams. The mine configuration is less elongate and is sometimes referred to as 'area' mining.

The majority of new coalmine developments in the world are for surface mines, and both black and brown coals are mined worldwide by this method. The large volume of black coal exported from Colombia, Indonesia, and Venezuela is from large surface mines, and the large black-coal mining operations in the Powder River and Green River basins in the USA are also surface mining operations. Brown-coal mines are virtually all surface mines, many of which are large-scale operations, as at Belchatów in Poland, Anjaliang in the PRC, and in the Latrobe Valley in Victoria, Australia.

### 10.3.1 Geological Factors

A surface mine will be designed by:

- (i) Assessing the basic geological data of the area, together with geographical and economic constraints. This will determine whether further investigation is warranted or should cease.
- (ii) The assessment of reserves from more detailed geological data followed by hydrogeological and geotechnical studies, to test the viability of a mine. From this, a decision is given on whether to commit

finance and other resources to develop a mine design.

- (iii) The refinement of the geological data and selection of the mining method is completed and a final mine design is produced. From this point, construction of the mine can commence.

As can be seen, each stage is based on an increase in the amount of geological knowledge. The first stage consists of exploration work with some limited drilling, so that the thickness, mining depth, and extent of the target coal seam(s) is known, together with the structural framework of the mine area, which includes identifying all major faults and changes in dip or strike of the strata. Samples for coal quality will be taken from outcrops and boreholes. Again, if the geological conditions are unfavourable or the coal quality is unsuitable, the project will be terminated at the end of the first stage.

During the second stage, a planned drilling programme will identify the weathering and hardness of the overburden, changes in coal seam thickness, seam splitting, washouts, small-scale faulting, and the nature of the non-coal interbeds between coal seams. The groundwater regime and all water levels should be ascertained, and a flow net plan should be plotted for the proposed mine site. Additional drilling will be carried out to collect coal samples for more detailed analysis and cores taken to determine the geotechnical nature of the overburden (see Section 6.4).

The major factors that influence stability are the existence of discontinuities, such as bedding, weak contact zones, jointing, faulting, steeply dipping beds, and the existence of old workings and fire zones. Stress relief due to the excavation of material can result in the opening of steeply dipping joints or development of tension cracks behind the working face. If these cracks become filled with water, then high hydrostatic pressures can develop that can result in block failures in the highwall.

Therefore, understanding the hydrogeological regime and rock strength characteristics is

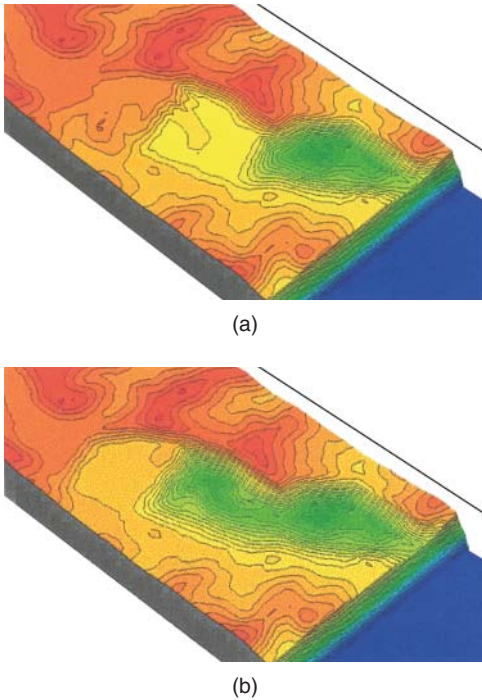
essential prior to further advancement of any surface mine project.

The configuration of the pit will be defined by limits of the lease area and outcrop and depth of the coal, together with any large physical features that will curtail mining, such as a large river or a major fault boundary. Computer-generated limits will include stripping ratio (SR) limits; that is, economic limits. Once the configuration of the pit area is fixed, the computer software can define the ground slope angle for each bench using the geotechnical data relating to the physical strength and competency of the strata.

The actual benches and blocks to be mined and the sequence in which they are to be mined are based on the type of equipment to be used. In opencast mining, the selection of equipment – such as bucketwheel excavators (BWEs), draglines, truck, and shovel, or combinations of these – together with the size of equipment selected, will influence the width and height of cuts to remove material and the successive advances of the mine.

The 3D model is also able to illustrate the volumetric calculation, coal quality variations, scheduling, and production sequencing, using preselected parameters relating to coal seam mineable thickness, quality, SR, and depth cut-off limits, which have been built into the model. Figure 10.14a and b shows 3D contour maps of the SR schedule for a selected area to be mined after 3 years and 4.5 years respectively (Datamine, personal communication, 2002). Such analysis allows schedules optimised for SR to be calculated for selected time periods. Additional considerations will be the hydrogeological regime within and surrounding the mine area and the geotechnical characteristics of the strata. For example, at Hazelwood Mine in Victoria, Australia, geological and coal quality modelling together with aquifer modelling have enabled short-, mid-, and long-term mine planning to be achieved, which should ensure the mine's profitability. The software allows bench plans and cross-sections to be generated efficiently;





**Figure 10.14** A 3D contour map showing SR schedule after (a) three years and (b) 4.5 years. *Source:* Reproduced with permission of Datamine International.

and with the block model intercepts projected onto the mine topography, the stratigraphy of any part of the current or future pit face can be indicated. The block modelling also allows effective overburden management, by creating solid models of three levels of dumps, and the development of new dump areas for the future (Maxwell 2000).

Operation of the mine depends on the scheduled operating shifts, the number of hours per shift, the equipment fleet, the assignment of each piece of equipment to a given bench area, the scheduled down time, holidays, and other items (Hartman 1992). Production scheduling is based on tonnage of coal produced, waste material removed, the production capacity of a specific piece of equipment, and the quality and recovery of the coal product.

The great value in being able to use using a block model format is its usefulness in pit

optimisation programmes. The ultimate pit configuration may already be defined by the mine owners or by physical constraints, but of greater value is the ability to model a number of alternative extraction strategies in liaison with the mining engineers. The interactive graphic capacity of the computer program will enable the mining engineer to study and refine the design as the process proceeds. Such strategies will take into consideration production targets over time, and targets to maximise net product value, cashflow, SR, and coal quality limits.

### 10.3.2 Mining Equipment

In surface mining, the excavation of overburden is by dragline, electric or hydraulic shovels, or BWEs. Coal excavation is usually by shovel (black or brown coal) and BWEs (brown coal).

#### 10.3.2.1 Dragline

The large walking dragline is a mainstay for large surface coalmines, particularly in the USA. Draglines are used to remove overburden in both brown (lignite) and black (bituminous) coal surface mines, and it can move large amounts of overburden at a lower cost than other mining techniques. More than 90% of all overburden removal by draglines is handled by large machines with 30 m<sup>3</sup> and larger buckets (Table 10.2). In the USA, a single dragline of this size can move 7 × 10<sup>6</sup> bcm per year, and the largest draglines (85–122 m<sup>3</sup>) can move over 20 × 10<sup>6</sup> bcm per year, and can excavate down to coal seams 45 m below the surface. Such equipment is only useable in the largest mines; elsewhere, dragline capacity may be lower (10–20 m<sup>3</sup>) when the mining operation is smaller, e.g. in India.

As the coal reserves in surface mines have become deeper and less accessible, walking draglines have been designed with longer booms and with extended digging and dumping ranges. Draglines currently in use with such modifications are shown in Table 10.2.

Figure 10.15 shows a large dragline removing overburden and dumping adjacent to the excavation area. The average cost of a walking

**Table 10.2** Dragline and BWE capacities currently in use.

Make	Dragline		BWEs		
	Boom length (m)	Bucket capacity (m <sup>3</sup> )	Manufacturer	Make	Mining capacity
Cat 8000	75–101	24–34	TAKRAF	Bagger 293	76 460 m <sup>3</sup> d <sup>-1</sup>
Cat 8750	109–132	76–129	TAKRAF	SR 1050	4800 m <sup>3</sup> h <sup>-1</sup>
P&H 9020XPC	100–130	85–122	ThyssenKrupp	L1355R	7310 m <sup>3</sup> h <sup>-1</sup>
			ThyssenKrupp	D1456	3860 m <sup>3</sup> h <sup>-1</sup>

Source: Adapted from manufacturers' websites.



**Figure 10.15** Dragline removing overburden in open-pit mine, USA.

dragline with a 10 m<sup>3</sup> bucket is US\$7.7 million, and with a 6 m<sup>3</sup> it is US\$3.3 million (Pavlovic, personal communication, 2020). Draglines mounted on crawlers are smaller capacity; the largest, with a 69 m boom and 17 m<sup>3</sup> bucket, is operating in New South Wales, Australia. A crawler dragline with 7.6 m<sup>3</sup> bucket and 61 m boom is US\$5 million. All of these are major capital items to equip a surface coalmine.

### 10.3.2.2 Powered Shovels

The modern trend in surface mining is one of large-scale operations that produce in excess of 1.0 Mt per year. Most of these operations utilise large loading equipment together with

large-capacity trucks; this style of operation is referred to as 'truck and shovel'. There are three types of loading equipment: electric mining shovels, hydraulic excavators, available as either front shovel or 'backhoe' type, and wheel loaders.

Wherever overburden cannot be economically handled by draglines or BWEs, it has been traditionally removed by electric mining shovels working in tandem with large trucks. In recent years, these have been challenged by the developing hydraulic excavator industry. The number of electric shovels has declined as unit size has increased, and as smaller models in smaller mines are replaced by competitive products. Electric shovels have remained the dominant loaders when size is considered, but new large hydraulic excavators are now being developed with a bucket capacity of 43.6 m<sup>3</sup>. All are larger than the typical wheel loader.

The USA is currently the largest market for all three types of loader, with large electric shovels and large trucks dominating the larger coalmines.

The manufacture of such equipment is very specialised, and there are a number of major manufacturers of electric shovels, hydraulic excavators, and wheel loaders. Table 10.3 shows the size and capacity of a selection of electric and hydraulic excavators and wheel loaders that are currently in use, and Figure 10.16 shows examples of all three kinds of equipment.

Surface coalmines producing 10 Mt per year or more will likely select the biggest trucks and shovels, as is the case in USA and Australia.

**Table 10.3** Size and capacity of a selection of electric and hydraulic excavators, wheel loaders, and trucks that are currently in use.

	Make	Bucket capacity (m <sup>3</sup> )	Load capacity (t)
Electric excavator (shovel)	CAT 7182	6.9–17.6	
	CAT 7395	20–55	
	CAT 7495HD	27–60	
	CAT 7495HF	30.6–61.2	
	P&H 2300XPC	19–36	
	P&H 4100C	30.6–61.2	
Hydraulic excavator (shovel)	CAT 6015	8	
	CAT 6040	22	
	CAT 6090	52	
	KOM PC850-8EO	3–4	
	KOM PC3000-6	15–16	
	KOM PC8000-6	42	
	Liebherr R9250	15	
	Liebherr R996B	34	
Wheel loaders	Liebherr R9800	42	
	CAT 988H	6.4–7.7	
	CAT 994K	14–36	
	KOM WA150-5	1.5	
	KOM WA600-6R	6.4	
Trucks	Liebherr L528	2–4	
	CAT 777F		100
	CAT 793F		226
	CAT 797F		363
	Liebherr T236		100
	Liebherr T284		363
	BeLAZ 7557		90
BeLAZ 75601		320–360	

CAT, Caterpillar; KOM, Komatsu.

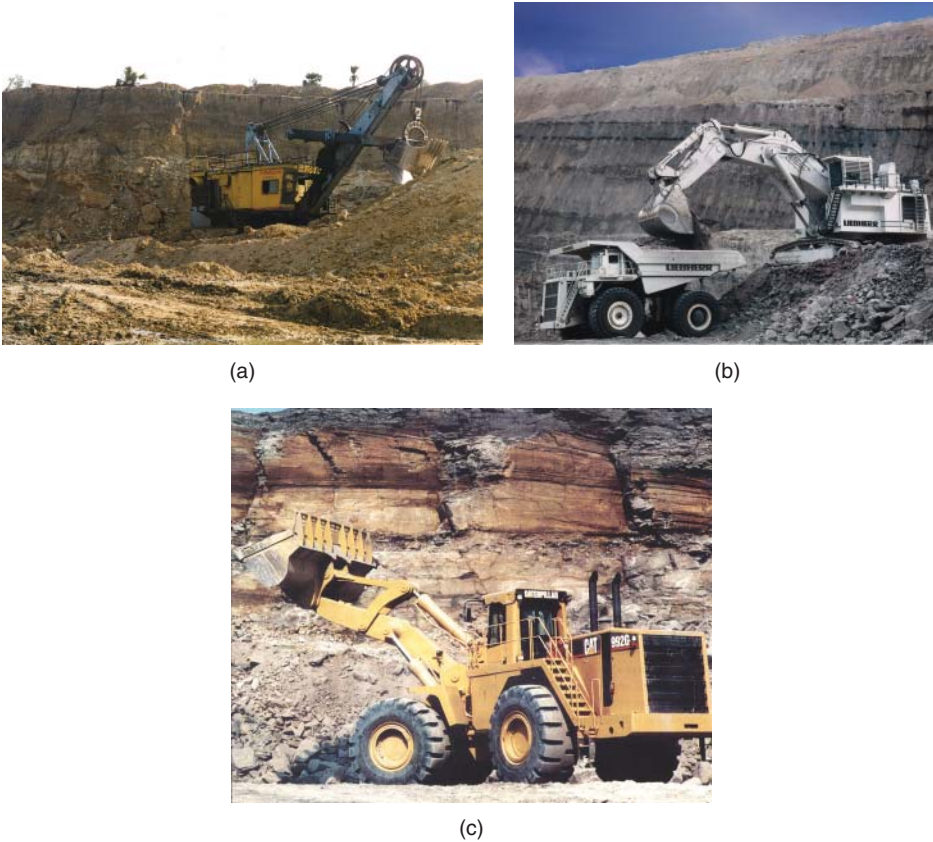
Source: Adapted from manufacturers' websites.

Smaller mines will require smaller units, as used in the UK and India. Other factors influencing the choice of equipment can be the necessity for mobility in an area, or electric power may be impractical in another.

For loading coal, it is usual to use smaller-size hydraulic excavators with either a front bucket or 'backhoe' configuration; these load coal into either road trucks or dump trucks.

Hydraulic excavators are preferred in Europe, whereas wheel loaders are rarely used.

This is in contrast to Australia, where hydraulic excavators and wheel loaders dominate mining. In India, coalmining has relied on smaller machines, chiefly because surface mining has only recently begun to increase in scale; the older machines are electric, and these are being replaced by hydraulic models. The new coal-exporting countries of Colombia, Indonesia, and Venezuela use mainly hydraulic excavators, and South Africa has changed to similar equipment in recent years. The



**Figure 10.16** (a) Electric shovel removing overburden, in central India. This type of shovel is still commonly in use, but it has been superseded by larger capacity models in the larger mining operations in the USA and Australia. *Source:* Photograph by courtesy of Dargo Associates Ltd. (b) Hydraulic shovel loading overburden in Spain. Modern shovel capacities are 20–30 m<sup>3</sup>. *Source:* Photograph by courtesy of Liebherr Mining Equipment Co. (c) Wheel loader removing overburden. Photograph by courtesy of World Coal, Palladian Publications.

PRC, the USA, and the CIS utilise both large and small electric shovels and are the chief manufacturers.

In considering electric shovels and hydraulic excavators, haul or dump trucks are utilised for both overburden and coal removal. For overburden, large dump trucks are used in the larger mining operations to complement the larger sized shovels. Dump truck sizes range from 35 to 100 t for smaller shovels and 100 to 363 t for the large shovels (Figure 10.17); most are rear dump models. Figure 10.18 shows truck fleets carrying overburden from the exaction area to the dump in a moderate-sized opencast mine. Bottom dump trucks, usually

100–190 t, and articulated dump trucks are also in operation in some countries. Table 10.3 gives examples of the types of trucks in operation at the present time.

To equip a surface mine with a truck and shovel fleet, the capital cost for electric shovels is US\$3.6 million, for hydraulic excavators it is US\$1 million–2 million, and dump trucks are US\$1.8 million–3 million (Pavlovic, personal communication, 2020). In addition, ancillary equipment such as dozers, graders, and backhoe excavators will also be required. Depending on the mine size, SR, and production scheduling, the capitalisation is a major item, but has the advantage of flexibility in that





**Figure 10.17** Large dump truck (280t) being loaded at Fording coalmine, Canada. *Source:* Photograph by courtesy of Komatsu Mining Systems.

the capital cost may be phased in as the mine increases in size over time. Most truck and shovel equipment has a mine life of around seven years, and capital must be available to replace worn-out equipment at several stages throughout the life of the mine.

### 10.3.2.3 Bucketwheel Excavators

In large-scale surface mines, with thick coal seam sections, as found in the large lignite mines in eastern Europe and Australia, BWEs (or dredgers) are used. These machines consist of a boom with a rotating wheel at one end

around which a series of buckets or scoops with a cutting edge can excavate relatively soft lignite or soft overburden. The excavated material is fed onto a series of conveyors that then load onto a main belt conveyor or into trams for transport out of the mine. The normal capacity of these machines is  $420\text{--}2300\text{ m}^3\text{ h}^{-1}$  ( $550\text{--}3000\text{ yd}^3\text{ h}^{-1}$ ), and they are particularly effective in excavating soft overburden in flat-lying strata. Figure 10.19a shows a BWE cutting overburden in a mine in Bosnia-Herzegovina, and Figure 10.19b shows a large BWE in operation in the Berezovsky mine in the Kansk–Achinsk Basin, Russia. Disadvantages are the inability to cut hard overburden or overburden containing boulders or large consolidated rock masses that typify glacial deposits. The system of fixed conveyors makes the use of BWEs less flexible than truck and shovel operations, and BWEs are not suitable in small, confined mines.

The cost of BWEs varies according to capacity: a  $4800\text{ m}^3\text{ h}^{-1}$  machine has a capital cost of around US\$22 million, plus accompanying conveyors at US\$7.5 million and spreaders at US\$8 million (Pavlovic, personal communication, 2020). As large mining operations may have between two and four BWEs operating, they make up a very large initial capital cost and ongoing maintenance costs. They do, however, have a long mine life. The manufacture and capacity of a selection of BWEs in current use is shown in Table 10.2.



(a)



(b)

**Figure 10.18** (a, b) Truck fleets transporting overburden from shovel to dump.





(a)



(b)

**Figure 10.19** (a) BWE cutting overburden in Gacko lignite mine, Bosnia-Herzegovina. *Source:* Photograph by courtesy of Dargo Associates Ltd. (b) Large BWE and conveyor system in operation in Berezovsky opencast mine, Kansk–Achinsk Basin, Russian Federation. *Source:* Photograph by courtesy of Dargo Associates Ltd.

### 10.3.3 Surface Mining Methods

The method of mining and the equipment used in surface mining are dependent upon the size, configuration, and depth of the planned mine, together with the ability to excavate hard or soft strata to access the coal seam(s). The width of surface mines is dependent on the geology, and the deepest level of excavation can be up to 150 m. Working faces can have an angle of  $50^{\circ}$ – $90^{\circ}$ , and mine batters and spoil tips have angles that range from  $30^{\circ}$  to  $45^{\circ}$ , dependent on the geotechnical properties of the overburden, with the lower angles being most common.

A brief outline is now given of the principal mining methods currently used in surface mining.

#### 10.3.3.1 Strip Mining

Strip mining usually commences close to where the coal seam crops out at the surface. If there is a significant weathering profile, then the initial box cut may be located down dip to expose the coal seam. The overburden is excavated directly or, if hard, is blasted before excavation. Overburden removal is by means of large electric or hydraulic shovels, and/or a dragline. The shovel stands on the top of the coal and excavates overburden from the highwall, whilst the dragline is situated on the top of the overburden and excavates down to

the coal. The working face is advanced along the strike of the coal; this leaves the coal seam exposed in the floor of the pit. The coal, which may or may not need to be blasted, is then excavated by a smaller shovel and loaded into trucks. The overburden from the first box cut is placed up dip or below the outcrop of the coal. Once the first cut has been completed, the second strip of overburden is removed down dip and parallel to the first. The overburden from the second strip is placed in the area left after removing the coal from the first strip, and successive strips are cut in this manner. Excavation is continued until the thickness of overburden becomes too great, because the SR is too high, and/or the excavation equipment has reached its maximum working depth. With this method of mining, land restoration is commenced early on in the mining schedule: the spoil area is landscaped, and the top soil is then replaced and prepared for appropriate land use. Such large-scale strip mines are operating in the Powder River Basin, Wyoming, USA; e.g. Jacobs Ranch mine produces 13 Mt per year of subbituminous, low-sulfur coal for electricity generation (Hartman 1992).

#### 10.3.3.2 Opencast or Open-pit Mining

The term opencast mining strictly refers to the excavation of the material above the coal seam or overburden and deposited in an area

immediately adjacent to the working face. This process is referred to as ‘back casting’ (Ward 1984) and involves the use of dragline excavators.

Open-pit mining refers to the removal of overburden from above the coal seam to a site some distance away (which may be within the excavated pit area or to an external dump area) by means of a selected haulage or transportation, such as a truck fleet or conveyor system.

However, the term opencast is widely used to describe both types of excavation.

Open-pit mining is more complex than strip mining, particularly when a very thick coal is to be extracted or a series of coals are targeted in one mine. In these circumstances, a series of benches will be developed and coal extracted from each bench; this can be on a very large scale, as at Anjaliang, PRC (Figure 10.20a), or be of more modest size, as seen at the

Pljevlja mine in Montenegro (Figure 10.20b) and Stanari mine in Bosnia-Herzegovina (Figure 10.20c). The use of explosives may be required to break up resistant rock in the overburden. As the mine develops, benches are constructed at succeeding lower levels. This means that all non-coal material (i.e. overburden and interburden) has to be removed and dumped away from the working bench areas. To achieve this, electric and hydraulic shovels with truck fleets will be used, particularly when the overburden is hard and requires blasting, when there are restrictions of space for equipment, and when the pit reaches lower and lower levels. Where large volumes of relatively soft overburden or thick brown coal (lignite) have to be excavated, BWEs may be used. If the pit is relatively shallow and wide, BWEs with their associated conveyor systems may be most appropriate. Black coal may be



(a)



(b)



(c)

**Figure 10.20** (a) Large-scale benched mining operation, Anjaliang opencast mine, China. *Source:* Photograph by courtesy of Dargo Associates Ltd. (b) Smaller scale benched mining operation, Pljevlja brown-coal mine, Montenegro. *Source:* Photograph by courtesy of Dargo Associates Ltd. (c) Stanari lignite mine, Bosnia-Herzegovina.



(a)



(b)

**Figure 10.21** (a) Truck and shovel operation, Shotton opencast mine, Northumberland, UK. *Source:* Photograph by courtesy of M.C. Coultas. (b) Brown coal being loaded, Stanari lignite mine, Bosnia-Herzegovina.

**Figure 10.22**  
Wirtgen surface strip miner in operation in Gacko mine, Bosnia-Herzegovina. *Source:* Photograph by courtesy of Dargo Associates Ltd.



blasted, if necessary, and shovelled directly into waiting trucks by small shovels or wheel loaders. Figure 10.21a illustrates such an operation at Shotton mine in Northumberland, UK, and Figure 10.21b shows brown coal being loaded at Stanari mine in Bosnia-Herzegovina. Brown coal will be cut by BWEs and conveyed or shovelled directly into trucks. In the case of exposed horizontal brown or soft coal, it can be cut and loaded by means of a surface miner such as a Wirtgen machine. This is used to strip off a thin layer of coal and to load simultaneously into a truck (Figure 10.22). This method of coal stripping has been used for selective

mining of coals to leave non-coal partings out of the run-of-mine product. A Wirtgen 3500 machine produces around  $500 \text{ bcm h}^{-1}$ . Overburden is removed to a designated dumping area and remains there until a void area in the mine is available. This is then filled in and the land restored. Coal may be taken to a loading area for transport away from the mine or stockpiled for use in the area adjacent to the mine. Examples of large black coal opencast mines are Antaibao and Haerwusu in the PRC, El Cerrejon Norte in Colombia, Guasare in Venezuela, Grootegeluk in South Africa, and numerous similar operations in Australia and



the USA. Smaller black coal opencast mines can be found in East Kalimantan, Indonesia, in India, and in the UK. Brown-coal mines dominate central and eastern Europe, who produce 50% of the world's total; there are also significant producers in the Ekibastuz coalfield, Kazakhstan, in the Latrobe Valley, Victoria, Australia, at Hambach, Germany, at Belchatow, Poland, and in the Big Brown mine in Texas, USA.

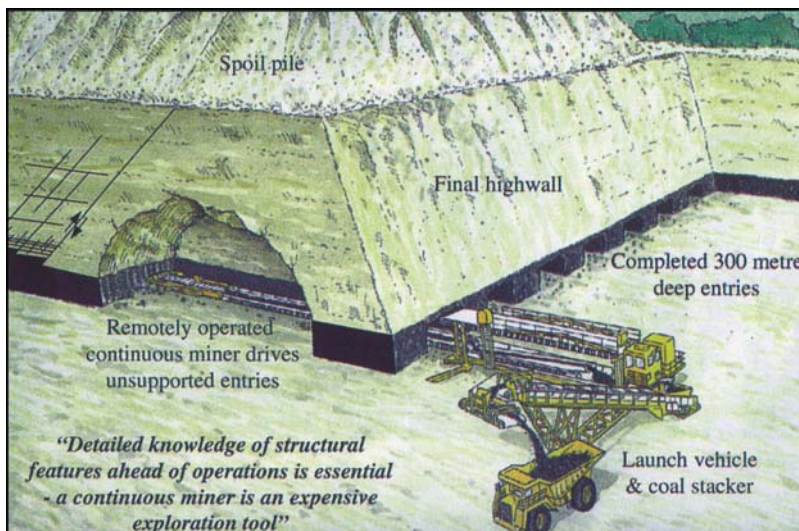
Contour mining is carried out in rugged topography, characterised by hills, ridges, and V-shaped valleys. The coal is extracted by removing the soil and rock overlying the coal, and is often referred to as the mountain removal method. The location is then restored to its approximate original contour. This procedure is followed along the outcrop of the coal seam as successive cuts are made. Contour mining has been practised successfully in the Appalachian coalfield, eastern USA, particularly in Kentucky and West Virginia.

### 10.3.3.3 Highwall Mining

Highwall mining is a remotely controlled mining method that extracts coal from the base of an exposed highwall, usually in a series of parallel entries driven to a shallow depth within

the coal (Shen and Fama 2001). This method enables coal to be mined that otherwise would remain in the ground. The arrival at the final highwall position may be due to an uneconomic SR or be in an area of the mine that had effectively sterilised the coal due to overlying mine infrastructure. Highwall mining is reliant upon the self-supporting capacity of the ground, because there will be no artificial support in the entries. It is essential that the nature of the highwall and the ground behind it is fully understood, so that a mining layout can be designed. Two types of highwall mining systems are used: the continuous highwall mining system, which uses a continuous miner to cut rectangular entries approximately 3.5 m wide (Figure 10.23), and the auger system, which creates individual or twin circular holes of various diameters depending on whether a single- or twin-auger system is used. Figure 10.24 shows auger mining in UK; the auger holes have a diameter of 0.45 m.

The continuous highwall mining system has been more widely used than the auger system because of its higher productivity and recovery rate. However, the auger system can better tolerate changing and difficult geological conditions, such as unstable seam roof and floor and seam thickness variations. In



**Figure 10.23** Schematic image of a highwall mining operation. *Source:* From Shen and Fama (2001). Reproduced with permission.

**Figure 10.24** Auger mining in Bottom Busty seam, Shotton opencast mine, Northumberland, UK. Auger holes are 0.45 m in diameter. *Source:* Photograph by courtesy of M.C. Coultas.



Australia, continuous highwall miners have reached 500 m penetration and 124 000 t per month, and the auger system has reached 200 m penetration and 60 000 t per month (Shen and Fama 2001). In the USA, auger systems have produced over 16 Mt of coal from the Appalachian coalfields from coal seams ranging from 0.6 to 4.9 m in thickness (Walker 1997). The final highwall can be 40–60 m deep with a face angle of 70°. One of the chief concerns is the stability of the highwall face. Instability can result from subsidence, discontinuities in the rock, either parallel to the face or intersecting at an angle to produce rock wedges, and from fracturing due to previous blasting. Both highwall mining systems have a protective shield above the working area, but this is only effective against small rock falls. In highwall mining, pillars are left between the entries. The success of the technique depends upon the stability of the pillars, because if they collapse during mining this may lead to the loss of mining equipment. It is essential that the unsupported spans are sufficiently stable to remain for days or even weeks. This is particularly true for continuous highwall mining with wide spans. Auger mining creates an arched roof with an effective span of around 1.0 m. Highwall mining is still a relatively new concept, having begun commercially in

the USA in the 1970s and in Australia in the 1980s. It has now been introduced in South Africa and the UK, and is a means of accessing hitherto considered unmineable coal reserves in opencast mines.

## 10.4 Coal Production

In most coal-producing countries, underground coalmining is no longer the principal mining method. Table 10.4 shows the proportion of underground to surface mining in the chief coal-producing countries. Of these, only the PRC still produces more coal from underground operations. Global coal consumption increased by 64% from 2000 to 2014, after which the first annual decrease occurred in coal production since 1990. Both the PRC and the USA saw a fall in production in 2016, of 9% (320 Mt) and 19% (85 Mt) respectively; and there was a drop in world production by 458 Mt. Elsewhere, production rates have fluctuated (see Section 7.4.2.1). Rising costs, overproduction, the price of natural gas, and environmental pressures have all contributed to changes in production levels. However, the increased demand for energy, particularly electricity, has meant that the demand for coal is still increasing.



**Table 10.4** Percentage of underground/opencast coal production in world's five largest coal producers.

Country	Underground production (%)	Opencast production (%)	Total production (Mt oil equivalent)
Australia	20	80	299
India	15	85	288
Indonesia	2	98	255
PRC	85	15	1685
USA	33	67	364

### 10.4.1 Underground Coal Production

A comparison of production between longwall and room-and-pillar operations shows the highest face production is achieved in longwall mines, e.g. 1.0–2.5 Mt per year, dependent on seam thickness, compared with 0.25–1.0 Mt per year for continuous miners.

Typical continuous-miner section production per shift for one continuous miner and two or three shuttle cars is 300–500 t for a 1.5–2.0 m thick coal seam, 500–700 t for a 2.0–2.5 m thick coal seam, and 700–900 t for a 2.5–3.0 m thick coal seam. Production will increase if two continuous miners are used per section with three or four shuttle cars.

In the USA, longwall production had increased from 5000 t per employee-year in 1989 to over 10 000 t per employee-year in 1999 (Sabo 2000), with a 10 hour shift producing up to 25 000 t in large longwall systems.

In Australia, the overall productivity of longwall mines (including surface and coal preparation plants) has averaged 9000 t per employee-year, compared with 6000 t per employee-year for non-longwall mines.

Where high production is essential to remain competitive, e.g. in the USA and Australia (where wages are a high proportion of the total cost), where good management and planning prevail, together with skilled operators in good mining conditions, then a modern longwall represents the best model, i.e. outputs of >2 Mt per year with less than 200 employees underground. Longwall mining is able to work under a greater range of geological conditions and is

preferred for deeper mining, for thinner coal seams, and for poor roof conditions. However, any interruption to production can cause serious problems. The relocation of longwall equipment to a new panel or to overcome an unforeseen obstacle is a high-cost process.

Room-and-pillar mining is more flexible than longwall mining in meeting variable underground conditions. It is less capital intensive, and the risk of losing the working faces is a lot less. Planning and design are simpler than for longwall mining, and training and organisation is also simpler. Because of the depth limitations, room-and-pillar mines are not generally considered for depths beyond 200–300 m.

Apart from geological considerations, finance can influence which mining method is selected. In addition to providing underground access (shaft or adit), and then excavating development roadways equipped with conveyors or underground railways, the cost to equip two longwall faces will be £35 million–£45.0 million (US\$60 million–US\$80 million) and a room-and-pillar continuous mining system is nearer £12 million (US\$18 million). For the investment in coalmining in developing countries, room-and-pillar mining is more flexible in changing geological conditions than longwall systems is, although the latter is more productive.

### 10.4.2 Surface Coal Production

A major part of world coal production comes from surface coalmines, Gilewicz (1999)

**Table 10.5** World's top 10 brown-coal producers.

Country	Production, 2015 (Mt)
Germany	178
PRC	140
Russia	73
USA	65
Poland	63
Australia	63
Indonesia	60
Turkey	50
Greece	46
India	44

quoted 1.3 billion tonnes produced from the world's major surface mines; this includes all significant brown coal production. Surface black-coal mines produce, on average, 1.0–15.0 Mt per year, whereas lignite mines can be much larger, producing up to 40 Mt per year.

Table 10.5 shows the top 10 brown-coal producers in 2015, with the PRC and Germany

being the principal suppliers, almost all providing fuel for the domestic coal market. In surface mining, the capital cost savings when compared with underground mining are large, although offset by the higher rehabilitation costs incurred when land restoration is required.

Surface mining is the preferred mining option where environmentally possible, and many of the major coal-producing countries have invested heavily in surface mines. This trend is likely to continue in the future. Transportation of coal within and from coalmines is by truck, rail, and/or conveyor. Well-established surface mines have integrated rail or conveyor systems connected to either nearby consumers, such as power plants, or connect to the surrounding rail network. Figure 10.25 shows a multiple rail system in Ekibastuz mine, Kazakhstan, where coal is moved along rails from different levels within the mine. In more remote areas, trucks and conveyors together with river transport may be the preferred option.

**Figure 10.25**

Multiple rail system in Ekibastuz mine, Kazakhstan. *Source:* Photograph by courtesy of Dargo Associates Ltd.



# 11

## Coal as an Alternative Energy Source

### 11.1 Introduction

The essential property that distinguishes coal from other rock types is that it is a combustible material. In the normal course of events, coal is burnt to provide warmth as a domestic fuel, to generate electricity as a power station feed stock, or as a part of the industrial process to create products such as steel and cement.

Coal, however, is more versatile than this and has been, and still is, able to provide alternative forms of energy. This may be from its by-products such as gas, or through chemical treatment to become liquid fuel, and by in-situ combustion to convert coal to liquid and gaseous products.

The development of these energy alternatives is important, particularly in those areas where coals are too deep for exploitation or where underground mining has ceased for economic reasons. Those coalfield areas once thought to be exhausted can still provide large amounts of energy through the use of modern technology. In addition, the understanding of the origins of oil and natural gas show coal to be a contributory source rock.

Although the bulk of coal utilisation is, and will continue to be, by direct handling and combustion, the alternative uses of energy from coal are increasing in significance and are being developed in all the major coal-producing countries.

### 11.2 Gas in Coal

Bituminous coals contain a number of gases, including methane ( $\text{CH}_4$ ), carbon dioxide ( $\text{CO}_2$ ), carbon monoxide (CO), nitrogen ( $\text{N}_2$ ), and ethane. The amount of gas retained and held by a coal depends on various factors, such as pressure, temperature, pyrite content, and the structure of the coal. Fresh coal contains more gas than coal that has been subject to oxidation. Large volumes of gas can be accommodated on the internal surfaces of the coal as a result of adsorption. It is released by the removal of pressure, usually by mining or drilling. The gas may migrate into associated strata, such as porous sandstones, which release the gas into openings, such as boreholes and mine excavations.

The association of gases with coal has been a constant problem in mine workings since underground coalmining first began. In underground workings,  $\text{CH}_4$  is released from coal exposed at the coal face, plus from the broken coal being transported through the mine.  $\text{CH}_4$  is a flammable gas and is explosive between a lower limit of c. 5% and an upper limit of 15% when mixed with fresh air. This highly combustible gas is known as 'firedamp'. The faster the coal is mined, the larger the amount of  $\text{CH}_4$  released into the workings, so that is essential that an adequate ventilation system is in operation. A danger is that of  $\text{CH}_4$  collecting

in roof pockets and in the upper parts of 'man-holes' or cuts in the roadway sidewalls where the rock sequence may still be exposed, as well as in goaf areas where coal has been extracted.

The safety lamp, invented by Sir Humphrey Davey in 1815, was the greatest single invention in the cause of safety, since it enabled coalminers to measure the concentration of  $\text{CH}_4$  in the mine ventilation system. The safety lamp could detect firedamp levels as low as 1.25% on a lowered flame. Since that time, the statutory maximum limit for  $\text{CH}_4$  content has been 1.25% for the use of electrical power. The use of locomotives and shotfiring, i.e. using explosives underground, must be discontinued if the  $\text{CH}_4$  exceeds this limit. At 2%  $\text{CH}_4$ , labour must be withdrawn from the workings until the  $\text{CH}_4$  content is diluted to within the statutory limit. Other coalmining countries used the same method of detection as the UK until hand-held monitors and continuous recording monitors were introduced in the 1970s and 1980s, so that gas emanating from the coal face can be monitored by keeping a methanometer in close proximity to working personnel. This allowed much lower concentrations of firedamp to be detected, and therefore allowed lower statutory limits to be introduced.

In the UK, the current statutory limits are 0.25% of  $\text{CH}_4$  in air for air entering the working area and 0.5% for  $\text{CH}_4$  in air for air returning from the workings.

In France, the maximum percentage of firedamp is 1.0–1.5%, and in Germany 1.0% is the normal limit, but this has been increased to 1.5% in certain longwall installations.

In Australia, intake airways are kept to below 0.25%  $\text{CH}_4$  and up to 2.0% in return airways. Above 1.25%, electrical power must be switched off, and persons are not allowed to travel in roadways with 2.0%  $\text{CH}_4$ . Continuous mining equipment may be required by the inspectorate to be equipped with automatic  $\text{CH}_4$  monitors, which emit audible signals at 1.0% at the cutting head, and the power is

automatically tripped at 2.0%. Similarly, on longwall faces, power is cut off at 1.25%  $\text{CH}_4$ .

In New Zealand, the limit for  $\text{CH}_4$  in the general body of the air in a coalmine is 1.25%. An inspector can call for a ventilation survey of the mine if this figure is exceeded.

In the USA, electrical shutdowns are required at 1.0%, and labour withdrawn at 1.5%.

Carbon dioxide is more common in brown coal than in bituminous coal workings. However, bituminous coals that have a high pyrite content contain higher amounts of  $\text{CO}_2$ , due to the fact that coals rich in pyrite absorb more oxygen ( $\text{O}_2$ ) when moist, and this absorbed  $\text{O}_2$  produces not only water by combination with hydrogen ( $\text{H}_2$ ), but also  $\text{CO}_2$  by combination with carbon (C).  $\text{CO}_2$ , also known as 'blackdamp', is a colourless gas and is heavier than air. It therefore tends to accumulate in the lower parts of mine workings.

Carbon monoxide originates from the incomplete oxidation of coal, especially after  $\text{CH}_4$  explosions and underground fires. The gas is combustible and poisonous.

Only a small proportion of the  $\text{N}_2$  found in coal gases has its origin in the  $\text{N}_2$  present in the coal material; the bulk of the  $\text{N}_2$  originates from the surrounding air.

Free  $\text{H}_2$  occurs in small amounts associated with  $\text{CH}_4$ , but it is not usually found in any great amounts.

Ethane is more prominent in gases derived from oxidised coals; cannel coal contains ethane in its pore structure.

Radon is a naturally occurring radioactive gas and, as such, is distinguished from the other gases present in coal; it does have significance in posing a health hazard to humans (see Chapter 12).

The  $\text{CH}_4$  content of the coal can, however, be regarded as a significant source of energy and is the subject of a large amount of research and development.  $\text{CH}_4$  is usually referred to as coal-bed  $\text{CH}_4$  (CBM) in most literature; however, in Australia, CBM is known as coal seam gas.

## 11.2.1 Coal-bed Methane

### 11.2.1.1 Coal-bed Methane Generation

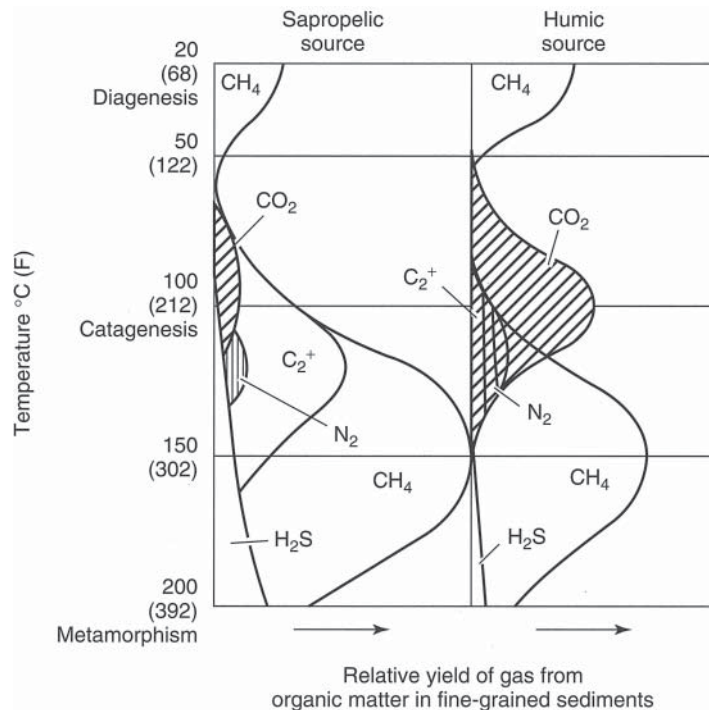
The process by which plant material is progressively altered through peat, lignite, subbituminous, bituminous, to anthracite coal is termed 'coalification' (see Chapter 4). As the organic material is altered through the effects of temperature and pressure, both physical and chemical changes take place. Diagenetic change occurs up to the lignite–subbituminous boundary, depending on time–temperature relationships. Above subbituminous rank, changes can be equated to metamorphic alteration.

The major products of the coalification process are CBM,  $\text{CO}_2$ ,  $\text{N}_2$ , and water. CBM is generated in two ways. First, during the early stages of coalification at temperatures below  $50^\circ\text{C}$ ; this is biogenic  $\text{CH}_4$  and is formed by decomposition of the organic material, and where biological activity induces reducing conditions, which remove  $\text{O}_2$  and sulfate. Where subsidence and burial are rapid, biogenic CBM may be trapped in shallow gas reservoirs (Rightmire 1984).

Second, CBM is generated by means of catagenesis, the process by which organic material is altered as a result of the effect of increasing temperature. CBM generated at temperatures in excess of  $50^\circ\text{C}$  will be due to this process and is referred to as thermogenic  $\text{CH}_4$ . The relative volumes of CBM generated by biogenic and thermogenic mechanisms are shown in Figure 11.1.

During coalification, more than twice as much  $\text{CO}_2$  as  $\text{CH}_4$  is generated up to the high-volatile bituminous–medium-volatile bituminous coal boundary. CBM volumes generated increase rapidly above this point, with the CBM generation peak occurring at about  $150^\circ\text{C}$ , or at the medium-volatile bituminous–low-volatile bituminous coal boundary (Figure 11.1). The two gases associated with CBM are  $\text{CO}_2$  and  $\text{N}_2$ ; the latter is only found as a minor constituent of thermogenic gas as it migrates readily from the system due to its small molecular size.  $\text{CO}_2$  is a principal constituent of early thermogenic gas (Figure 11.1), but it is only a relatively minor

**Figure 11.1** Generation of gases with depth,  $\text{C}_{2+}$  represents hydrocarbons heavier than  $\text{CH}_4$  in gas phase.  $\text{N}_2$  is generated initially as ammonia. *Source:* From Rightmire 1984 (based on Hunt 1979).





and extremely variable constituent in the gas produced at high temperatures. It is highly soluble in water, and this facilitates its mobility from the system. Analysis of gas produced from coal-beds shows that 95% is CBM, <3% is CO<sub>2</sub> and N<sub>2</sub>, and there are trace amounts of higher hydrocarbons (e.g. ethane, propane).

#### 11.2.1.2 Coal-bed Methane Retention

CBM is retained in coals in three ways: first, as a free gas within the pore space or fractures in the coal; second, as adsorbed molecules on the organic surface of the coal; and third, dissolved in groundwater within the coal. Porosity in coals occurs as fracture porosity and matrix porosity. The latter is more significant when considering the CBM retention potential of coals.

Gas generated in excess of that which can be adsorbed on the coal surfaces will be 'free' gas within the porosity of the coal, most notably in the fracture porosity. This gas is available to be dissolved in groundwater moving through the coal. The CBM saturation of large volumes of water can remove large volumes of gas from the coal seam(s), which will be lost to the system and possibly vented to the atmosphere. The fracture porosity in coal is primarily due to the formation of fractures called cleat. Cleat is a joint or set of joints perpendicular to the top and bottom of the coal seam. Usually, there are two cleat sets developed in an orthogonal pattern (see Section 2.3.2.1). Cleat is a major control on the directional permeability of coals. Cleat fracture porosity in coal is estimated to be between 0.5% and 2.5% (Laubach et al. 1998). Owing to the increasing importance of coals as gas reservoirs, geologists are re-examining the characteristics and origins of cleat. For CBM extraction, knowledge of the properties of natural fractures is essential for planning exploration and development, due to their influence on the recovery of CH<sub>4</sub>, plus the understanding of the local and regional flow of hydrocarbons and water.

Although small amounts of free gas may exist in coal fracture systems, CBM is adsorbed

on the large internal surface area of the impermeable coal matrix and fracture surfaces. A number of fractures influence the ability to utilise pore surface for CBM adsorption. The size of the aperture accessing the pore surfaces, the variation in moisture content, and/or the degree of coalification may alter the surface areas of a coal. Surface area is also related to carbon (C) content; studies have shown that coals with C contents of <76% and >83% generally have surface areas <1 m<sup>2</sup> g<sup>-1</sup>, whereas coals within that range have areas >10 m<sup>2</sup> g<sup>-1</sup>. An exception to this is anthracite with >92% C, which also has high areas of 5–8 m<sup>2</sup> g<sup>-1</sup> (Gan et al. 1972). Studies of open pore distribution in the coal rank series have been carried out to support this (Table 11.1). Porosity may also depend on the maceral content in high-volatile bituminous coals. Vitrinite has fine porosity, with pores of 20–200 Å diameter; inertinite is the most porous, with pore diameters of 50–500 Å (Rightmire 1984). The adsorptive capacity of coal appears to increase with increasing rank. The maximum amounts of CBM that can be adsorbed onto the internal surfaces of coals according to coal rank and depth are shown in Figure 11.2. This adsorption of gas molecules on a solid surface is related to the gas pressure around the surface at a fixed temperature. The CH<sub>4</sub> desorption process follows a curve of gas content against reservoir pressure; this model was developed by Irvine Langmuir in 1916 and is called a Langmuir isotherm. This is used for most models of adsorption, and Figure 11.3 shows a typical CBM isotherm characteristic of a San Juan Basin coal. These are properties of the coal and vary widely; coals may have different Langmuir parameters despite having other similar coal properties.

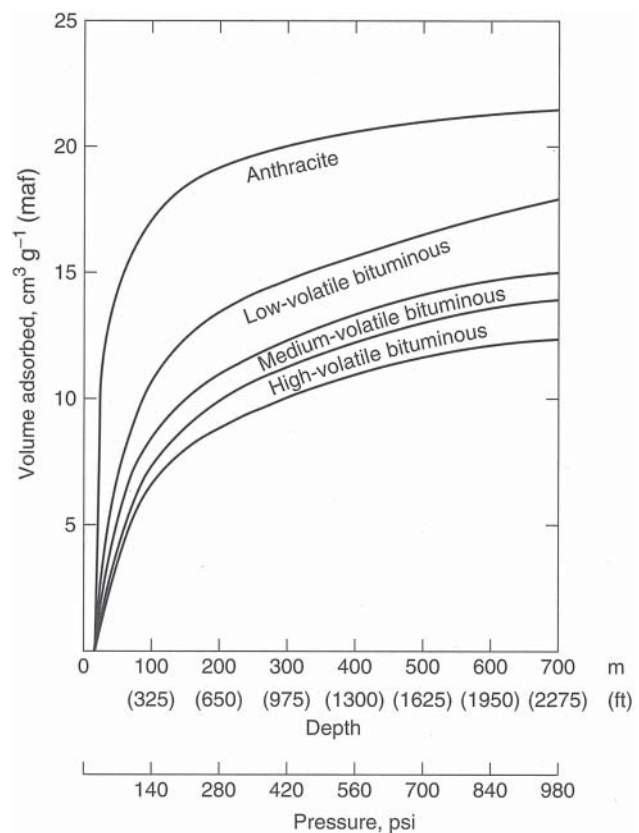
Numerous studies of coal cleat formation, orientation, and genesis together with coal permeability have been carried out in recent years. These are a response to the development of CBM extraction as a large-scale commercial enterprise. Huy et al. (2010) studied permeability in relation to fracture width and

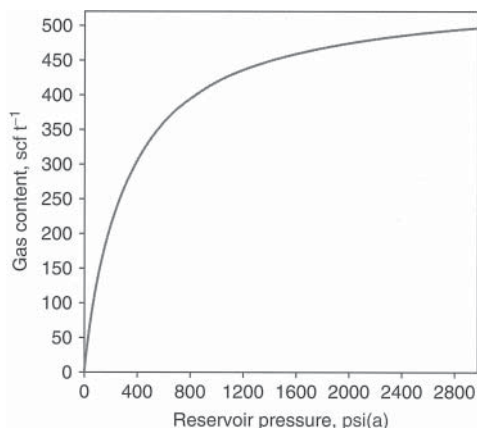
**Table 11.1** Gross open-pore distribution in coals.

Rank	C (% daf)	Porosity distribution (%)		
		<12 Å	12–300 Å	>300 Å
Anthracite	90.8	75.0	13.1	11.9
Low-volatile bituminous	89.5	73.0	0	27.0
Medium-volatile bituminous	88.3	61.9	0	38.1
High-volatile A bituminous	83.8	48.5	0	52.0
High-volatile B bituminous	81.3	29.9	45.1	25.0
High-volatile C bituminous	79.9	47.0	32.5	20.5
High-volatile C bituminous	77.2	41.8	38.6	19.6
High-volatile B bituminous	76.5	66.7	12.4	20.9
High-volatile C bituminous	75.5	30.2	52.6	17.2
Lignite	71.7	19.3	3.5	77.2
Lignite	71.2	40.9	0	59.1
Lignite	63.3	12.3	0	87.7

Source: Based on Rightmire (1984) and Gan et al. (1972).

**Figure 11.2** Adsorptive capacity of coal as a function of rank and depth. Source: From Rightmire 1984 (based on Kim 1977).





**Figure 11.3** Typical CBM isotherm characteristic of San Juan Basin coal (Baimukhametov et al. 2009).

determined that fracture permeability can be estimated for measurements of gas flow rates in coal core samples.

Coal-bed permeability through the cleated network is sensitive to both fracture aperture and fracture length distribution. Permeability of coal increases with cleat density and cleat aperture size, and high cleat density in coal seams is favourable for higher fluid flow in CBM reservoirs (Paul and Chatterjee 2011). Detailed studies of Queensland Permian coals to determine the relationship between cleat spacing, cleat height, and coal banding texture for coals of different rank was undertaken by Dawson and Esterle (2010) (see Section 2.3.2.1). They distinguished four major classes of cleats and concluded that narrow-spaced cleats exist at all ranks but the distribution of cleat spacing with cleat height varies for specific cleat classes. Other researchers have found that cleat spacing decreases from lignites to medium-volatile bituminous coal and increasing in anthracite coal. Fracture permeability in most coals found in the USA lies in the range of 0.1–50 md (millidarcies; Wikipedia 2011).

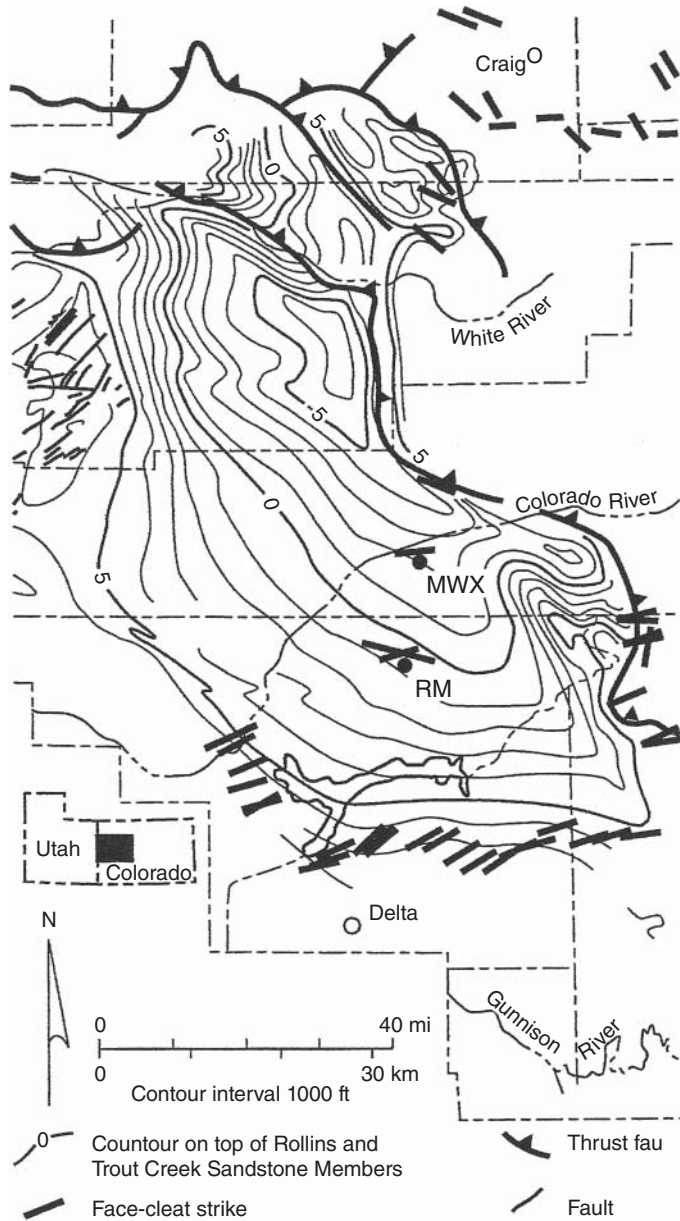
Cleat patterns on a regional scale are often better known than fractures in non-coal strata, because the dominant cleat type in an outcrop can be readily identified. Maps of cleat

orientation have indicated domains of uniform and variable cleat orientation. However, conflicting interpretations of cleat domains have occurred. This may be due to development of cleat in areas where stress orientations differ or superimposed episodes of cleat development are associated with changes in stress directions during cleat formation at different times (Laubach et al. 1998). Transitions between domains of uniform orientation can vary from gradual to abrupt. Figure 11.4 shows the orientation of the cleat system plotted in the Piceance Basin, Colorado, USA. Here, a domain of east-north-east-striking face cleats in the south in Cretaceous Mesaverde Group coals is replaced in the northern part of the basin by west-north-west-striking cleats. Study of the thermal history showed that the northern coals reached higher thermal maturity and possibly the beginning of cleat development at  $R_o$  (vitrinite reflectance) values between 0.3% and 0.5%, later than coals in the southern part of the basin. Coalification in the southern area occurred more rapidly, and  $R_o = 0.5\%$  was reached approximately twenty-two million years after deposition, whereas in the north  $R_o = 0.5\%$  was not reached until thirty-one million years after deposition (Laubach et al. 1998). If cleat development took place at  $R_o = 0.5\%$ , the two cleat domains could represent a shift in palaeostress directions. Evidence for this includes different face cleat orientations in Cretaceous and Palaeogene–Neogene coals in the same area.

In the UK, Figure 11.5 shows the dominant cleat orientation in Westphalian Coal Measures to be north-west–south-east in northern England, swinging round to a north–south orientation to the south. Less regular cleat orientations are present in Kent and South Wales. Cross-cleat development is intermittent and locally complex; the mean angle between cleats is  $85^\circ$  (Jones et al. 2004).

Mapping of the cleat patterns of coal seams that are CBM reservoirs is necessary in order to estimate the behaviour and potential flow directions in such a CBM reservoir. Clear

**Figure 11.4** Map of cleat system in the Piceance Basin, Colorado, USA (Laubach et al. 1998).

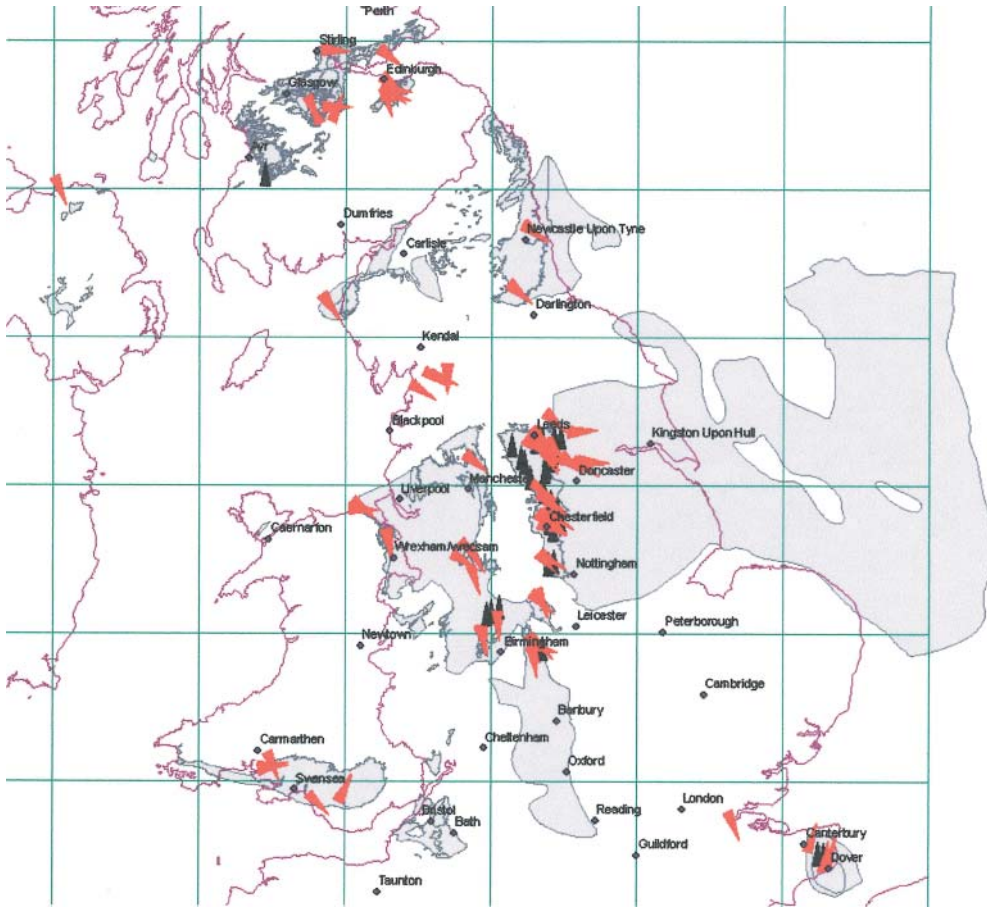


understanding of cleat systems together with their relationship to stresses adjacent to and within wells will be a key factor in the selection of well completion technology in order to optimise production from CBM wells (Paul and Chatterjee 2011). As a consequence, studies of regional patterns of cleat orientation are forming a significant part of CBM exploration

together with the analysis of larger scale discontinuities in coal seams targeted for CBM exploitation.

### 11.2.1.3 Coal-bed Methane Production

The production of CBM from underground sources is either by draining old and current mine workings (coalmine  $\text{CH}_4$ , CMM) or by



**Figure 11.5** Map of UK selected cleat directions (red) and cleats without recorded orientation (black). The symbols outside the Westphalian Coal Measures relate to Palaeogene–Neogene and older coals (Department of Energy & Climate Change [DECC] 2013).

production from wells sunk into virgin or unmined coal seams (CBM) (Figure 11.6).

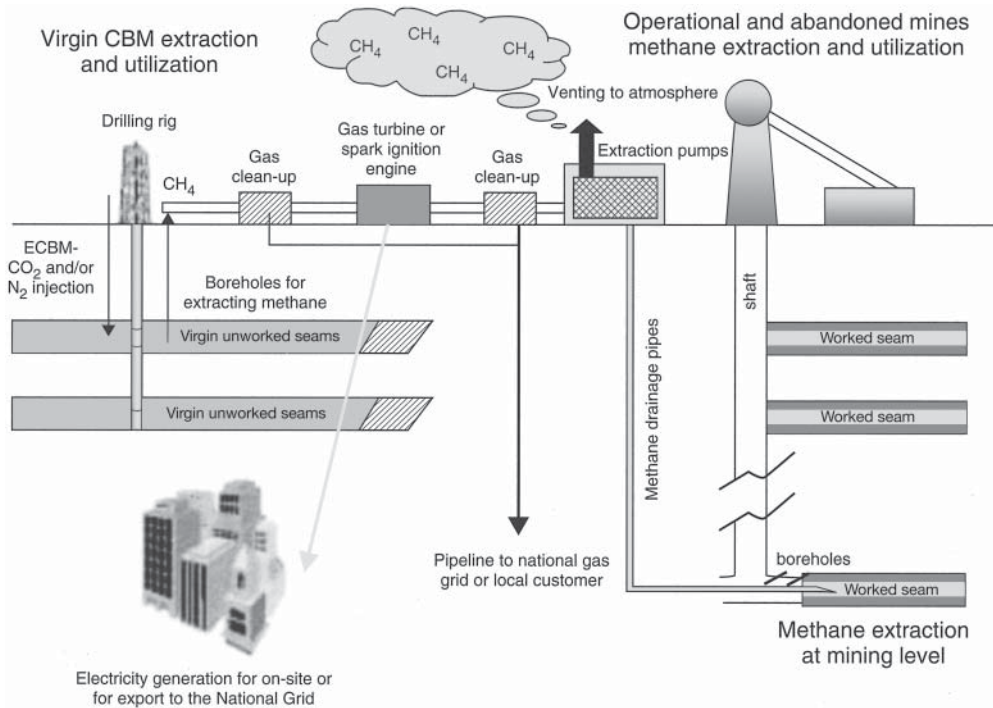
#### 11.2.1.3.1 Coalmine (Active and Abandoned) Methane

**Active Coalmine Methane**  $\text{CH}_4$  is released as a direct result of the process of coalmining; in many mines, the preferred mining method is by longwall extraction, and this, as with other underground mining techniques, releases  $\text{CH}_4$  previously trapped within the coal seam into the air supply of the mine, thus creating a safety hazard. In addition,  $\text{CH}_4$  emissions arise from the collapse of the surrounding strata after the coal seam has been mined and the

roof supports are advanced as mining progresses through the selected panel area. The collapsed rock section is known as goaf or gob, and this also releases  $\text{CH}_4$  or gob gas into the mine.

In many coal-producing countries, underground mining operations are becoming deeper and deeper due to the exhaustion of shallower seams and improvements in mining technology. The increasing depth of coal seams usually equates with higher  $\text{CH}_4$  content, this puts pressure on mine ventilation systems which have to be increased, as worldwide safety standards require that mines cease operation when  $\text{CH}_4$ -in-air levels exceed a



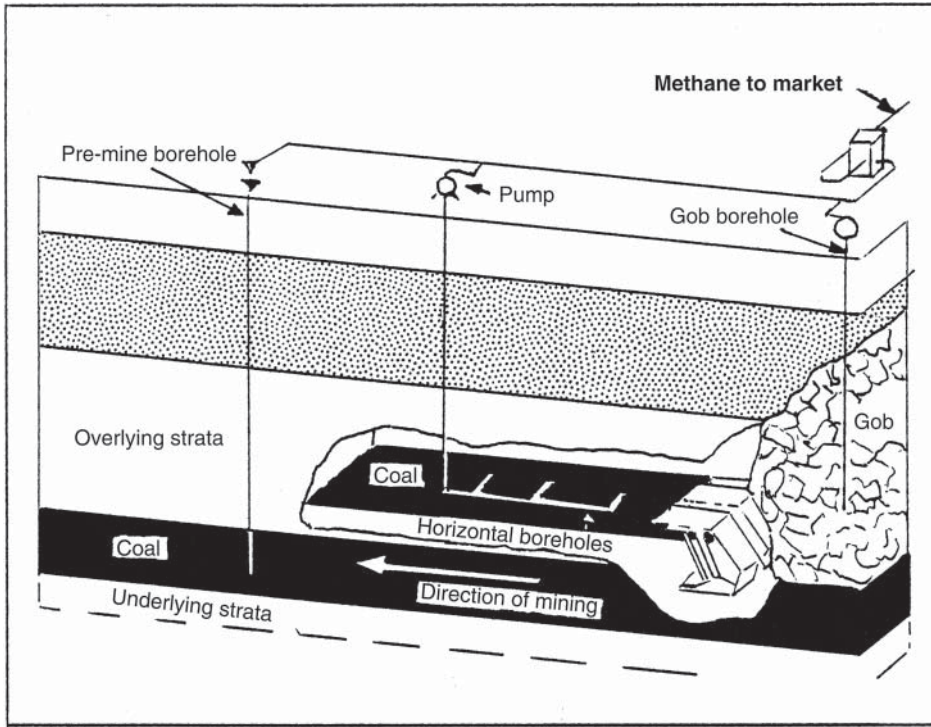


**Figure 11.6** CBM and CMM options for  $\text{CH}_4$  extraction and utilisation (Department of Trade and Industry [DTI] Report 2001).

pre-determined percentage. Lost production due to excessive  $\text{CH}_4$ -in-air content can have serious economic implications. When faced with this phenomenon, mines should recover and use the  $\text{CH}_4$ . In the USA, those mines with high gas contents vent up to 75% of all gas liberated. Worldwide, mines vent 1.23 trillion cubic feet (TCF; 35 billion cubic metres [BCM]) to the atmosphere and use around 0.07 TCF (2 BCM) (Schultz 1997) – for all unit conversions see Appendix E. Depending on the drainage method selected, a mine can remove between 20% and 70% of  $\text{CH}_4$  in a coal seam. This will provide significant relief to the ventilation system, as well as producing gas of suitable quality for the commercial market. The mine ventilation systems move the diluted  $\text{CH}_4$  out of the working areas of the mine into shafts leading to the surface; this technique is known as ventilation air  $\text{CH}_4$  (VAM). The VAM is released through ventilation shafts and can then be destroyed or captured for utilisation

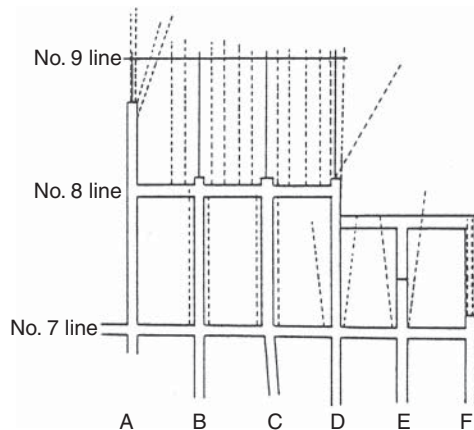
rather than allowing it to be released directly into the atmosphere. VAM has the lowest concentration of  $\text{CH}_4$  from coal seams due to its high exposure to air, often displaying levels of 0.05–0.8%. This technique is applicable in mines that are still either in operation or are still maintained on a care and maintenance level.

Drainage methods include vertical wells (vertical pre-mine), gob wells (vertical gob), long hole horizontal boreholes and horizontal and cross-measure boreholes (Figure 11.7). The vertical wells are drilled from the surface to drain gas from coal situated in advance of the mine workings; these wells produce almost pure  $\text{CH}_4$ . Similarly, gob wells are drilled from the surface to drain gas from the mined-out areas of the mine, such as the mined-out voids behind longwall faces or disused or collapsed areas of room-and-pillar districts. Gob wells produce a  $\text{CH}_4$ -air mixture, but as gob gas is exposed to lower volumes of air than VAM,



**Figure 11.7** CH<sub>4</sub> extraction from active mine workings. *Source:* Adapted from Schultz (1997).

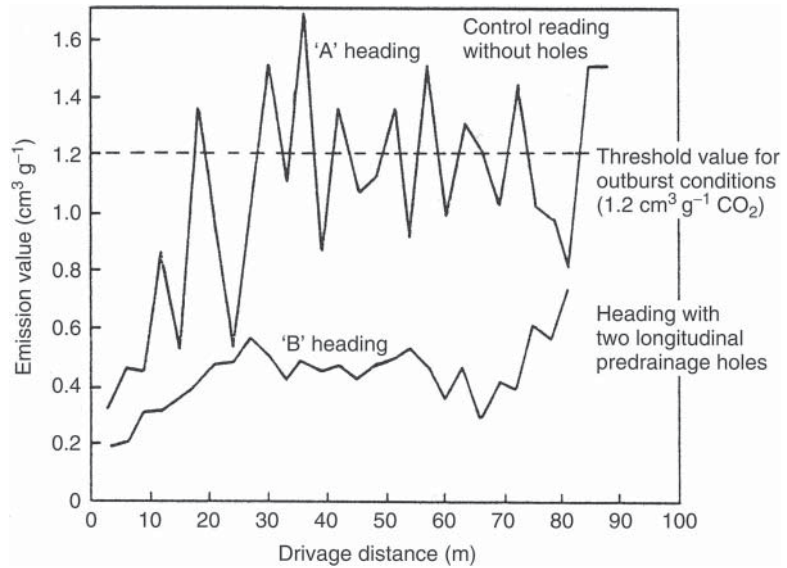
it displays much higher CH<sub>4</sub> concentration levels, typically between 35% and 75%. Horizontal and cross-measure boreholes are drilled within the mine to drain CH<sub>4</sub> from seams in production and from surrounding gob areas. Again, the holes in-seam produce pure CH<sub>4</sub> and the holes in the gob areas produce a CH<sub>4</sub>-air mixture. In the Sydney Basin, CBM drainage has been carried out by drilling a series of boreholes both vertically from the surface into the coal, and by drilling horizontal boreholes into the seam ahead of the working panels. In Figure 11.8, a pattern of horizontal boreholes has been drilled into the panels ahead of heading B, each borehole ranging from 50 to 100 mm in diameter. Figure 11.9 shows a comparison of the resultant gas emissions in headings A and B; the gas emissions in heading B are reduced to less than half those of the undrained heading A, demonstrating the effectiveness of this method.



**Figure 11.8** Pattern of drainage holes ahead of a panel front, Metropolitan Colliery, Sydney Basin, Australia. *Source:* From Hargreaves and Lunarzewski (1985).

**Abandoned-Mine Methane** Abandoned-mine CH<sub>4</sub> (AMM) can be recovered from previously worked by now disused underground coalmines. Such mines are either sealed

**Figure 11.9** Reduction of gassiness resulting from pre-drainage.  
 Source: From Hargreaves and Lunarzewski (1985).



(i.e. any entrances into the mine have been sealed or vented where ventilation shafts have been left unsealed), or flooded by groundwater influx into the old mine workings. Of these, well-sealed abandoned mines afford the greater opportunity for CH<sub>4</sub> extraction than do vented and flooded mines. Vertical gob gas wells can be drilled and CBM recovery will be similar to that applied to working mines. Vented mines allow CBM recovery via existing ventilation shafts; this can be a low-cost option for CBM recovery, whereas flooded mines incur a higher cost as mine dewatering must take place before CBM recovery can take place. In all cases, the greatest benefits occur if the CBM recovery takes place in the first two years post-mine closure. The UK, USA, and Germany have been leaders in AMM projects, but there is huge potential for AMM in the People's Republic of China (PRC) and other countries with coalfield areas now no longer worked.

**Coalmine Methane/Abandoned-Mine Methane Production** In Australia, there has been a significant development in the field of coalmine gas drainage and utilisation. In 2008, of the 29 underground mines operating, 12 employed gas drainage, where gas content of the coal

exceeded 6–8 m<sup>3</sup> t<sup>-1</sup> (Black and Aziz 2009). In 2010, Australia's emissions totalled 1.7 BCM from active mines, whereas emission levels from abandoned mines remained low. In 2015, there were 25 CMM projects registered at 14 active underground and AMM projects at five abandoned mines, of which nine projects used CMM to generate electricity.

The largest source of CMM is the dilute CH<sub>4</sub> emitted from mine ventilation shafts (VAM); it is estimated that more than 55% of all CMM emissions originate from mine shafts (Black and Aziz 2009). West Cliff colliery uses plant known as WestVAMP VAM to generate 6 MW<sub>e</sub> of electricity from the dilute CH<sub>4</sub> in the ventilation air.

The PRC, where 95% of the coal mined comes from underground workings, has become a world leader in CMM recovery and use: by 2011 there were 1047 mines with CH<sub>4</sub> drainage systems, draining c. 12.6 BCM of gas, of which 4.5 BCM was used (US Environmental Protection Agency [EPA] 2015). The concentration of CH<sub>4</sub> in many Chinese coalmines is low; usually, there is a concentration of 10–35%, with a maximum concentration of 60%, with 70% of the recovered CMM having a concentration of less than 30% (International

Energy Agency [IEA] 2009). The principal type of CMM project in the PRC is for power generation: in 2013, 12.6 BCM of CH<sub>4</sub> was drained, and 4.25 BCM had been used to generate 1500 MW by the end of 2011 (US EPA 2015). Although the PRC has an enormous potential for CMM recovery, most Chinese coalmines have drainage systems that produce low-quality CMM; in particular, they have the problem of the transport of gas in or near the explosive range of CH<sub>4</sub> close to underground workings. Improvements in mine degasification will help to avoid potential risks. In 2011, Chinese mines emitted more than 19 BCM of VAM. As the PRC gains experience with VAM systems and gas recovery at coalmines, utilisation of CMM will increase. In many parts of the PRC, coalmines are located in isolated rural areas where the transportation of CH<sub>4</sub> to potential users is difficult and not financially viable. The construction of small power-generating units in these areas will be a major consideration for the PRC in the future.

CMM in Russia is concentrated in the Kuzbass, Pechora, and Donbass regions. In 2011, CMM from underground mines totalled 2.02 BCM (US EPA 2015). The Kuzbass region accounts for 65% of CMM, with 47 active underground mines, and the Pechora Basin contributes 19%. In recent years, the rate of CH<sub>4</sub> recovery from CMM drainage in active mines has been 25–30% on average (US EPA 2015), and abandoned coalmines are monitored for CH<sub>4</sub> concentrations. To improve this general situation, a number of projects have been set up to study the utilisation of CMM from both active and abandoned mines. In the Kuzbass, 32 abandoned coalmines have high levels of CH<sub>4</sub> that are currently monitored for future AMM drainage.

In the adjoining Ukraine, there are 31 CMM projects registered, including 16 underground, 2 opencast mines, and 1 abandoned mine. Traditionally, 95% of the CH<sub>4</sub> produced from underground coalmines was vented directly to the atmosphere. Projects designed to utilise CMM are ongoing at the Bazhanov and

South Donbass #3 coalmines. CMM emissions reductions from degasification of the mines is anticipated to be c. 46 billion cubic feet (BCF; 1.32 BCM) per year.

In Poland there are three active mine CMM projects and four more proposed. In 2010, emissions were 480 Mm<sup>3</sup>, of which 31% was captured through degasification of the mines in 2013 (US EPA 2015). CMM from active mines has decreased with mine closures. In France, with the closure of all major underground mines, AMM has focused on the Pas de Calais Basin. There are three CH<sub>4</sub> recovery projects at abandoned mines, with potential for similar projects in Lorraine in north-east France and Gardanne in southern France.

In Germany, 43 CMM projects have been identified: 34 are located at abandoned mines and 9 at active underground mines. The CH<sub>4</sub> from 30 projects is utilised for power generation, and 13 projects use it for heating and power. The total CMM emissions are estimated at 194 MCM in 2010 (US EPA 2015). In addition, there are 37 individual AMM projects to be used for power generation.

In the Czech Republic, CH<sub>4</sub> emissions totalled 290 Mm<sup>3</sup> in 2015. The OKR mining basin has accounted for 99.8% of CMM from Czech active mines. In 1998, 4.2 TCF (0.12 TCM) of CH<sub>4</sub> was drained from the mines; in addition, AMM is produced from 10 abandoned shafts and four wells totalling 880 MCF (25 MCM) per year (US EPA 2015). AMM degasification figures will increase with significant mine closures.

In the Karaganda Basin in Kazakhstan, the coalmines are extremely gassy. The average CH<sub>4</sub> content of coal seams is 18–24 m<sup>3</sup> t<sup>-1</sup> at depths up to 700 m (Baimukhametov 2009). This can result in the release of gas at 150 m<sup>3</sup> min<sup>-1</sup> on a coalface producing 5000 t d<sup>-1</sup>. The gas emissions are 70% from worked areas (goaf) and 30% from the coal-bed being mined. Efficiency of coal-bed degassing is dependent upon the permeability of the coal-beds, which decreases with increasing mining depth. In Karaganda, the reduction of

coal-bed gas content prior to coal extraction is by degassing the unmined coal by directional hydraulic fracturing of the coal-bed from the surface and by degassing coals in previous mine workings (Baimukhametov 2006). In addition, the undermining of the coal seam by working at a lower level, but within 30 m, has the result of producing a sharp decrease in the gas content of the higher seam, so that further degassing is not required. Implementation of these integrated degassing methods allows a 46–55% degassing efficiency to be reached on working faces. Emissions are approximately 30 Mm<sup>3</sup>, of which 25 Mm<sup>3</sup> is utilised for heating in the aboveground mine facilities (Baimukhametov 2006). Measurements of CH<sub>4</sub> flow and concentration are in progress, particularly in abandoned mines, where the quality of sealing produces varying CH<sub>4</sub> release rates.

In the UK, conventional CH<sub>4</sub> drainage has concentrated on boreholes which give access to CH<sub>4</sub> held in gas sands and carbonaceous material above and below the coal seam being extracted. Trials have been carried out in known gassy mines to improve the accessing of CH<sub>4</sub> from the coal itself. Mixed results were obtained due to technical and geological difficulties in drilling near-horizontal wells in the confined space underground. Following the trials, a comprehensive underground drilling programme was considered impossible due to the technical difficulties and prohibitive cost. Studies were also carried out into improving ventilation systems in rapidly advancing drivages, i.e. where continuous miners are in operation. It was shown that approximately 20% of the gas quantity ultimately released from the roadway sides enters the workings during the first hour after the coal is cut. After this initial high flow, the coal continued to degas for a period up to seven months (International Mining Consultancy 1997b). Underground mining has now ceased in the UK, with only a small number of surface mines in operation.

The large volumes of gas in abandoned mines not only cause surface emissions, but

also lead to migration into adjacent operating mines, resulting in high gas levels, particularly during falls in barometric pressure (International Mining Consultancy 1997a). The redevelopment of mine sites adds to the number of locations where mine gas emissions may collect to form hazards. Such redevelopment may also disturb the naturally occurring gas seals, such as glacial clays, and so exacerbate the problem. Currently, at least 25 AMM drainage schemes are in operation in the UK from abandoned underground mines. These are privately run and sell gas to generate electricity, each site producing 1–10 MW<sub>e</sub>. New AMM sites are being investigated, so an increase in the amount of CMM drainage in the UK is likely over the next few years.

The USA has been a leader in CMM recovery: in 2013, there were 22 projects at 16 active underground mines and 17 projects at 37 abandoned mines. Most of the projects involved upgrading CMM/AMM for injection into a commercial pipeline; other uses include electricity generation, use in boilers, and two VAM projects. All of the mines are located east of the Mississippi River except for one in Colorado and one in Utah. In 2013, CMM/AMM emissions totalled 5.3 Bm<sup>3</sup>, and recovery and utilisation from coalmine degasification systems has averaged 83% since 2000 (US EPA 2015).

A number of other countries are implementing CMM/AMM gas drainage schemes, including Mexico and Nigeria, but large coal producers, such as India, still have no commercial gas recovery projects.

Table 11.2 summarises a number of CMM/AMM projects worldwide.

#### **11.2.1.3.2 Coal-bed Methane Production in Unmined Areas**

##### **Coal-bed Methane Well-Production Methods**

The production of CBM from surface wells penetrating virgin or unmined coal has experienced phenomenal growth worldwide over the past twenty years. Studies of the major coal-bearing basins in the world suggest that more than 50% of the estimated in-situ



**Table 11.2** Estimated CMM and AMM projects and utilisation.

Country	CMM projects	AMM projects	Utilisation
Australia	125	5	Power generation, VAM oxidation
PRC	67	0	Town gas, power generation, industrial use, vehicle fuel, pipeline injection, VAM oxidation
Czech Republic	1	4	Pipeline injection
France	0	3	Pipeline injection, industrial use
Germany	9	37	Heat and power generation
Kazakhstan	1	0	Boiler fuel
Mexico	3	1	Boiler fuel, power generation
Nigeria	3	1	Boiler fuel, power generation
Poland	3	1	Power generation, coal drying, industrial use, boiler fuel
Russia	7	0	Power generation, boiler fuel
Ukraine	4	1	Power generation, heating, industrial use
UK	0	25	Power generation, boiler fuel, pipeline injection
USA	22	17	Power generation, coal drying, heating, pipeline injection
Total	145	95	

Source: From various sources, including US EPA (2015) and UK Coal Authority (2011).

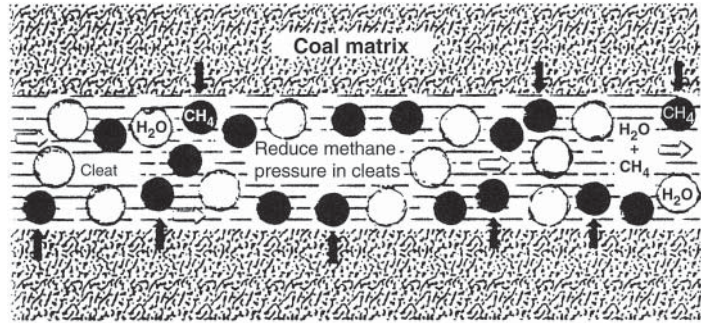
CBM resources is found in coals at depths below 1500 m (5000 ft). Drilling in deep, low-permeability reservoirs has demonstrated that open fractures can exist at depths of 2000–3000 m (7000–10 000 ft) (Myal and Frohne 1991). Of major concerns are, first, the effect of horizontal and vertical stress components on deep-lying coal beds; here, tests have shown that CBM can be produced at economic rates from coals below 1500 m under low to moderate stress conditions (Murray 1996). Second, the effect of gas and water saturation of coal on CBM production; the ideal would be ‘dry’ coal (no mobile water and free gas in the cleats and fractures) and the rapid desorption of CBM as the formation pressure is lowered, combined with low-stress conditions. It has been demonstrated that CBM wells have not deteriorated over time. In the USA, in the San Juan Basin, New Mexico, a CBM well drilled in 1953 has produced 150–180 MCF d<sup>-1</sup> (4.2–5.1 MCM d<sup>-1</sup>) for thirty years. This well has shown that CBM under conditions of favourable geology and reservoir

characteristics can produce economically over a long period of time (Murray 1996).

Over time, production of gas from a coal reservoir results in changes in pressure, which influences the permeability of the coal. As gas is desorbed, pressure exerted by gas inside the pores decreases, causing them to shrink in size and restricting gas flow through the coal. As the pores shrink, the overall matrix shrinks as well, which may increase the space the gas can travel through the cleats, so increasing gas flow.

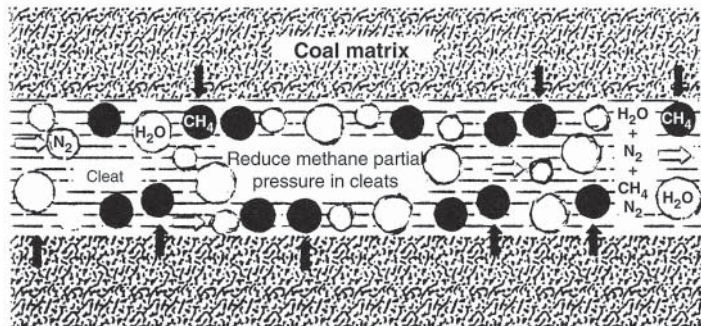
There are a number of methods by which CBM is produced from wells. The standard means for CBM production is by reservoir pressure depletion (Figure 11.10a). Reservoir pressure is reduced by dewatering the coal-bed, and gas then desorbs from the matrix and micropores of the coal by a process of diffusion. The desorbed gas then flows to the well bore via the coal cleat and fracture system, along with any groundwater still present in the fractures. However, this method does not recover much more than 50% of the CBM in

**Figure 11.10** (a) Coal-bed gas recovery by reservoir pressure depletion. *Source:* From Murray (1996), based on Puri and Yee (1990). (b) Enhanced coal-bed gas recovery by use of  $N_2$  injection. *Source:* From Murray (1996), based on Puri and Yee (1990).



- Reduce cleat pressure by producing water
- Methane desorbs from matrix and diffuses to cleats
- Methane and water flow to wellbore

(a)



- Inject nitrogen into cleats
- Keep total cleat pressure high
- Reduce partial pressure of methane
- Methane desorbs from matrix and diffuses to cleats
- Methane, nitrogen and water flow to wellbore

(b)

place. To overcome such low recovery rates, a series of experiments have been carried out in the USA on reservoir enhancement. One method is the injection of an inert gas, such as  $N_2$ , into the coal; this lowers the partial pressure in the coal and allows a greater percentage of CBM to be recovered (Figure 11.10b). Alternatively,  $CO_2$  is injected in similar fashion into the coal-bed reservoir to release a greater percentage of CBM. This has been used successfully in the San Juan Basin, USA, and has been tested in Alberta, Canada. The use of dynamic openhole cavity completion techniques has also been developed in the USA. In this type of operation, perforated casing is run to total well depth and the target

coal-beds are subjected to various types of fracture inducement or cavity completion. This is designed to create numerous fractures of varying orientation linking the reservoir to the well. Tests have shown that adequate reservoir permeability, reservoir overpressuring, and thermal maturity to at least high-volatile A bituminous rank are required for successful cavity completions. This technique has been pioneered in the San Juan Basin and helped to make it the most prolific CBM-producing basin in the world (Murray 1996). The USA leads the field in such experimentation, driven by the fact that CBM has become an increasingly important energy resource available to large energy consumers.

**Table 11.3** Major coal resources (BP Statistical Review 2011) and estimated CBM resources in the world.

Country	Coal resource (10 <sup>6</sup> t)	CH <sub>4</sub> resource (TCF <sup>a</sup> , in place)
Russia	166 364	2955
PRC	244 010	1100
USA	251 582	700
Canada	6582	500
Australia	144 818	33
Germany	36 212	100
UK	70	102
Kazakhstan	25 605	32
Poland	24 161	200
India	94 769	120
South Africa	9893	5–10
Zimbabwe	502	40
Ukraine	34 375	63
Indonesia	25 573	453
France	No longer exploited	15
Total	106 449	6423

a) Trillion cubic feet.

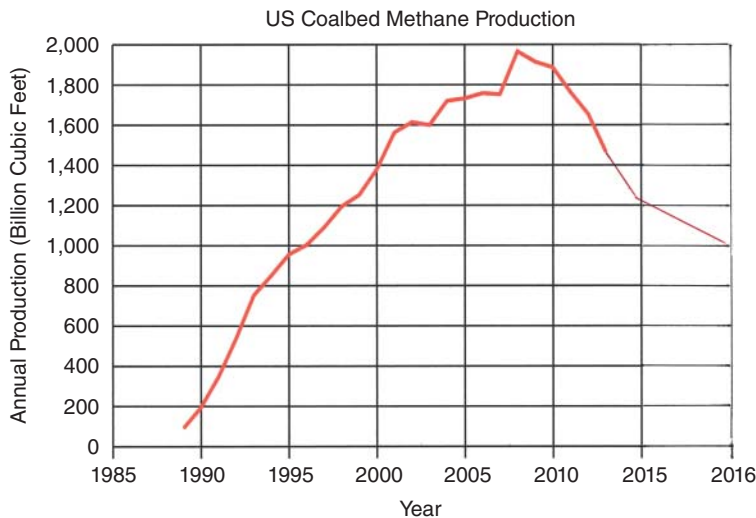
Source: From multiple sources 2016.

**World Coal-bed Methane Resources and Production** There are widely differing estimates of global in-place CBM resources, ranging from 275 to 11 000 TCF (78–320 TCM),

of which 30–60% is deemed recoverable. Those countries with the largest CBM resource potential are shown in Table 11.3. Of these, the PRC, Russia, and the USA have the largest resources, with other significant resources in Australia, Canada, Europe, India, Indonesia, Kazakhstan, southern Africa, and Ukraine.

Over the last thirty years, the USA has led the world in developing CBM production on a large scale. Figure 11.11 shows the growth of CBM production in the USA from its beginnings in 1985 to 1379 BCF in 2000 (US Energy Information Administration [EIA] 2018); this increased to more than 1900 BCF in 2008 before falling to 1020 BCF in 2018 (US EIA 2018). Nearly 50 000 wells have been completed, giving an estimated reserve of 10 585 BCF. The principal US producing areas are Virginia (Central Appalachians), Alabama (Black Warrior Basin), New Mexico (San Juan Basin), Colorado (San Juan and Piceance basins), and Wyoming and Montana (Powder River Basin). CBM reserves and production figures are given in Table 11.4. Figure 11.12 shows the geographical distribution of the principal CBM resources in the USA.

In the Western United States, the San Juan and Piceance basins cover an area of 32 000 km<sup>2</sup>, deriving CBM from low- to high-volatile bituminous coals with gas

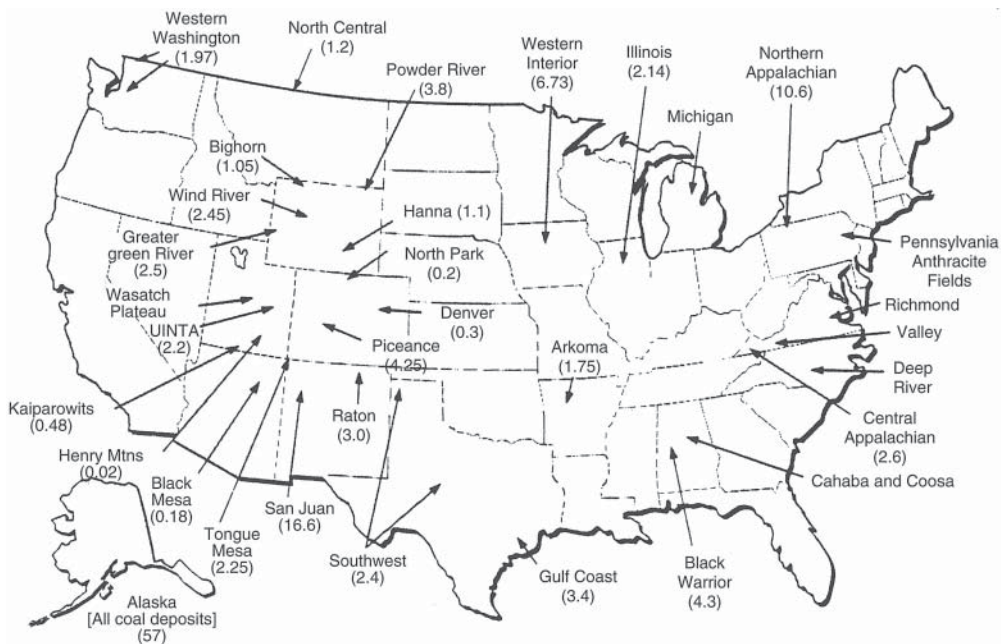


**Figure 11.11** US CBM production 1985–2016 (www.wikipedia.org 2017).

**Table 11.4** CBM proved reserves and production from principal US producing areas.

Area	CBM proved reserves 2016 (BCF)	CBM production 2016 (BCF)
Alabama (Black Warrior Basin)	985	45
Colorado (Piceance, San Juan)	3 265	352
Kansas	55	11
New Mexico (San Juan)	2 210	253
Oklahoma	320	33
Pennsylvania (N Appalachians)	206	10
Texas	84	11
Utah (Uinta Basin)	332	39
Virginia (C Appalachians)	2 117	102
West Virginia (N Appalachians)	99	9
Wyoming (Powder River Basin)	882	143
Others	30	12
Total USA	10 585	1 020

Source: US Energy Information Administration (2018).



**Figure 11.12** Coal-bed gas resources of the USA (in TCF); includes probable, possible, and speculative potentially producible supply. Source: From Murray (1996).

contents of  $8.5\text{--}17\text{ m}^3\text{ t}^{-1}$ . The Powder River Basin covers over  $40\,000\text{ km}^2$  and contains low-rank subbituminous coal to lignite with a gas content of  $2\text{ m}^3\text{ t}^{-1}$  but with high permeabilities. In the Eastern United

States, the northern Appalachian Basin covers  $72\,000\text{ km}^2$  with coals of low to high bituminous rank. Gas contents range from  $2.8$  to  $7\text{ m}^3\text{ t}^{-1}$ , and gas production is from surface and in-mine workings. The Black Warrior

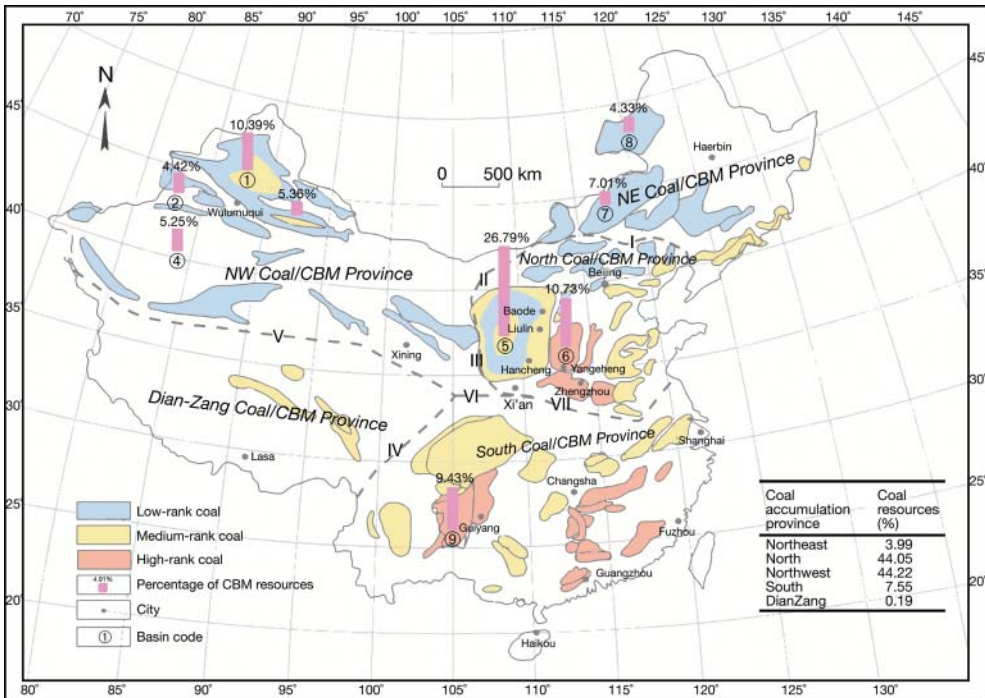


Basin is also large, some 70 000 km<sup>2</sup>, with low- and high-volatile bituminous coals having gas contents of 8.5–19.8 m<sup>3</sup> t<sup>-1</sup> (Thakur 2016).

In Canada, it is estimated that there are up to 500 TCF of CBM in the province of Alberta, with additional resources in British Columbia. Coals are highly inclined low-volatile bituminous with a gas content of 8.5 m<sup>3</sup> t<sup>-1</sup>. Development of CBM in Canada has lagged behind the USA, in part due to lack of permeability in coals and a number of environmental issues (Stanley 2014).

The PRC has large CBM resources, estimated at 1100 TCF (30 TCM), of which the estimated recoverable CBM resources are 350 TCF (10 TCM), considered the second largest CBM resource in the world. A review of the resources and geology of CBM in the PRC has been carried out by Qin et al. (2018); this gives a good overall picture of CBM potential in the PRC. Research on the CBM potential

in the PRC has grown, first to investigate and verify CBM resources, and second to carry out specific studies on those areas identified with the greatest potential. The first evaluation on CBM resources was submitted in 1985 as 633 TCF (17.93 TCM) for depths less than 2000 m. In 2004–2006 the Ministry of Land and Resources estimated CBM resources from 42 basins as 1300 TCF (36.81 TCM), whereas the Chinese Academy of Engineering indicated a total of 882 TCF (25 TCM). At greater depths of 2000–3000 m, an additional 670 TCF (19 TCM) may be present, but this will be difficult to extract commercially. The bulk of the resources is in the large- to medium-sized basins of Qinshui, Ordos, Junggar, Erenhot, Hailar, and western Guizhou (Figure 11.13). Geological conditions are more structurally complex than the commercial areas in the USA. CBM production has increased from 0.35 BCF (0.01 BCM) to 158.8 BCF (4.5 BCM) in 2016



**Figure 11.13** Coal/CBM accumulation provinces of PRC. Orogenic belts: I Yinshan, II Helanshan, III Liupanshan, IV Longmenshan, V Kunlunshan, VI Qinling, VII Dabieshan. Basins: 1 Junggar basin, 2 Tianshan basin, 3 Tuha basin, 4 Santangshan basin, 5 Ordos basin, 6 Qinshui basin, 7 Erenhot basin, 8 Hailar basin, 9 Qianxi basin. From Qin et al. (2018).



(Qin et al. 2018). The Qinshui Basin is located in the south-west of Shanxi Province with an area of 27 000 km<sup>2</sup> (Figure 11.13) and has been the largest CBM producer since 2003. CBM is sourced from Permo-Carboniferous coals of medium- to low-volatile bituminous rank. Gas in coals ranges from 4 to 22 m<sup>3</sup> t<sup>-1</sup>, being higher in the southern part of the basin where coals are deeper. Qin et al. (2018) reported that, by 2014, the basin had more than 6300 CBM wells and accounted for 74% of all Chinese CBM production. The Ordos Basin (Figure 11.13) has the second largest CBM production in the PRC. The basin covers 20 000 km<sup>2</sup>, and in 2015 there were 2677 wells producing CBM from Permo-Carboniferous medium- to low-volatile coals. Gas contents ranged from 3 to 27 m<sup>3</sup> t<sup>-1</sup> (Qin et al. 2018). Again, CBM potential is governed by geological structure, permeability, and groundwater characteristics. The Junggar Basin in north-west PRC (Figure 11.13) contains CBM resources of 135 TCF (3.83 TCM) (Che et al. 2008). The basin contains Lower and Middle Jurassic coals, and coal rank increases with increasing depth, with the higher rank coals being medium-volatile bituminous. Gas contents range from 2 to 10 m<sup>3</sup> t<sup>-1</sup> with localised highs of 15 m<sup>3</sup> t<sup>-1</sup> (Qin et al. 2018). In south-west PRC, the western Guizhou Basin (Figure 11.13) covers 28 000 km<sup>2</sup> and has the highest potential in South PRC with an estimated resource of 74 TCF (2.09 TCM) (Qin et al. 2018). Coals of Permian age range from low-volatile bituminous to anthracite, with variable gas contents of 0.2–29.2 m<sup>3</sup> t<sup>-1</sup>.

Russia has estimated CBM resources of 2955 TCF (83.7 TCM), with the Kuzbass Basin providing one of the largest CBM resource development opportunities with resources of 462 TCF (13 TCM) at 1800–2000 m depth. The Pechora Basin has a CBM resource of 68 TCF (1.9 TCM), but this area's harsh climate may limit exploitation of this resource. In the Kuzbass Basin, all of the underground coalmines are gassy; more than 1.2 TCM was liberated in 1991 (Marshall et al. 1996), but only 17% was removed by mine drainage

systems. In 2003, Gazprom launched a project to determine the potential of CBM production in Kuzbass (Gazprom 2011). In the Taldinskaye region, a CBM resource of 3.35 TCF (95.3 BCM) has been estimated, of which 88 BCF (2.4 BCM) is at depths less than 600 m, and 175 BCF (4.9 BCM) is at depths of 600–1200 m. Gazprom produced 20 000 m<sup>3</sup> of CBM from this area in 2011.

The Eastern Donbass Basin has a resource of 3.42 TCF (0.097 TCM); the larger part of the CBM resources present in this basin are in Ukraine. Figure 11.14 shows the geographical distribution of basins with CBM potential.

In Ukraine, the Donbass Basin has estimated CBM resources of 60 TCF (1.7 TCM); the Bazahnov and South Donbass mining areas have multiple coal seams with gas contents ranging from 15 to 20 m<sup>3</sup> t<sup>-1</sup>. Development in these areas has been slow due to economic and political factors.

Kazakhstan has large resources of CBM centred in four principal areas: the Karaganda Basin with 19.4–26.4 TCF (550–750 BCM), Ekibastuz Basin with 2.64–3.53 TCF (75–100 BCM), Zaviador Field with 515–593 BCF (14.6–16.8 BCM), and Samarskiy Fields with 388–500 BCF (11.0–14.2 BCM); development is still at an early stage.

Poland has an estimated CBM resource of 200 TCF (5.8 TCM) present in the Lower and Upper Silesian basins and Lublin Basin. The Upper Silesian Basin has the best prospect, where economic resources of 3.3 TCF (95.9 BCM) were estimated in 2016 (<http://geoportal.pgi.gov.pl/surowce/energetyczne/mpw>).

With the closure of the Lower Silesian coalmines, and rationalisation of the mines in the Upper Silesian Basin, production of CBM will be critical to the maintenance of energy supplies in these industrialised areas.

Australia has been the most active in development of CBM outside of the USA. CBM recovery has focused on Queensland and New South Wales, with 97% of CBM production occurring in Queensland and 3% in the



**Figure 11.14** Russian Coal Basins with CBM potential (adapted from Gazprom 2011).

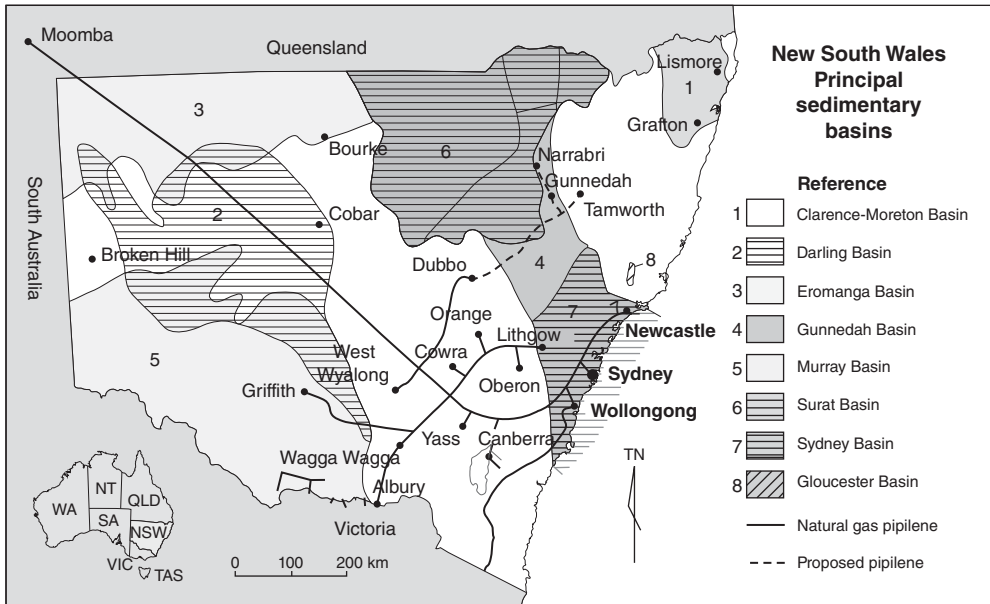
Sydney Basin in New South Wales. Commercial CBM production began in Queensland in 1996, providing gas to three coastal cities. Proved and probable reserves estimates rose to 33 TCF (934 BCM) in 2011, with a total CBM resource of 203 TCF (5.75 TCM) in 2012. During 2012–2013, CBM exploration continued, with 1315 CBM wells drilled. The Bowen, Galilee, and Surat basins continue to be the main focus, whereas the Sydney, Gunnedah, Gloucester, and Clarence–Morton basins are also being targeted (see Figure 11.15).

In India, the CBM resource is estimated as 120 TCF (3.4 TCM) in 44 major coal and lignite fields. Of these, the Raniganj, Jharia, and Singrauli coalfields have the best prospects. The coal is high rank with gas contents of  $7.3\text{--}23.8\text{ m}^3\text{ t}^{-1}$  down to a depth of 1200 m. The government has put up 26 blocks for bidding, and 200 test wells and nearly 300 production wells have been drilled. Production has commenced in the Raniganj block, and other areas are expected to follow suit (Mendhe et al. 2017).

CBM production is operational in the Lorraine Coalfield in north-east France, where, in 2011, CBM resources of 8.5 TCF (240 BCM) were estimated by La Francaise

de l’Energie (<https://www.francaisedelenergie.fr/en/home/>). In Germany there has been little development of CBM resources, estimated at 100 TCF (3 TCM). There is interest in developing the Ruhr Coalfield.

In the UK, the last decade has seen a significant increase in exploration for CBM. Initial targeting of CBM prospects was governed by the extent of underground mine workings, the depth of the target sequence (preferably <1500 m), the volumes of coal present in situ, and the coal rank and measured gas content (Bailey et al. 1995). Figure 11.16 shows the extent of the Westphalian Coal Measures and CBM developments in the UK (DECC 2013). Jones et al. (2004) estimated the CBM resource in the UK to be 102 TCF (2.9 TCM); although this CBM resource is large, commercial production has yet to produce, leaving a greater degree of uncertainty in the viability of the resource. The South Wales Coalfield has high-rank coal combined with significant confining pressure, and areas such as Cheshire and Nottinghamshire have coal preserved at greater depth. These areas appear the most attractive for CBM exploration. The coalfield areas of the UK vary in rank, in confining pressure, and in



**Figure 11.15** Principal sedimentary basins with oil and gas potential, in New South Wales, Australia. *Source:* Reproduced with permission from MINFO®.

gas content; this, with low coal seam permeability, will constrain CBM production in many parts of the UK (Creedy 1999). Because of low seam permeability and low gas content, as well as other technical and logistical constraints, Jones et al. (2004) estimated that as little as 1% of the resource could be recovered; but in view of improved recovery in similar geological conditions in the USA, together with the latest drilling technology, greater recovery may be feasible. A number of licences have been granted for both conventional CBM wells and also for gob gas and mine drainage wells. CBM capture from former mine workings will have the added advantage of reducing the amount of CBM currently escaping to the atmosphere. In Scotland, a pilot electricity generation scheme is planned that will use CBM from four wells drilled in unusually permeable virgin seams.

Indonesia has one of the largest CBM resource in the world, with a potential 453 TCF (12.8 TCM). The South Sumatra Basin contains 183 TCF (5.2 TCM) and the Kutei Basin (Kalimantan) contains 80 TCF (2.2 TCM). Exploration drilling began in South

Sumatra in 2009, and there are interests in block areas in Kalimantan. Progress has been slow due to environmental concerns and lack of investment.

In Zimbabwe, 40 TCF (1.1 TCM) of potentially recoverable CBM is estimated to be in place in the Lupane-Lubimbi area. Recent financing has renewed interest in developing the resource. In Brazil, the Charqueadas-Santa Rita Basin, Rio Grande do Sul, has been under intensive study (Martins et al. 2010).

The extraction of CBM from low-rank coals is now being considered, which would extend the global use of CBM and enable countries with low-rank coal deposits to benefit from this energy source.

Other countries that have known coal deposits will assess their CBM potential, the viability of which has increased with improved drilling technology but which will require significant investment in the future. It is clear that, in the future, the utilisation of both CBM and CMM will become a major contributor to the energy needs of numerous countries worldwide.

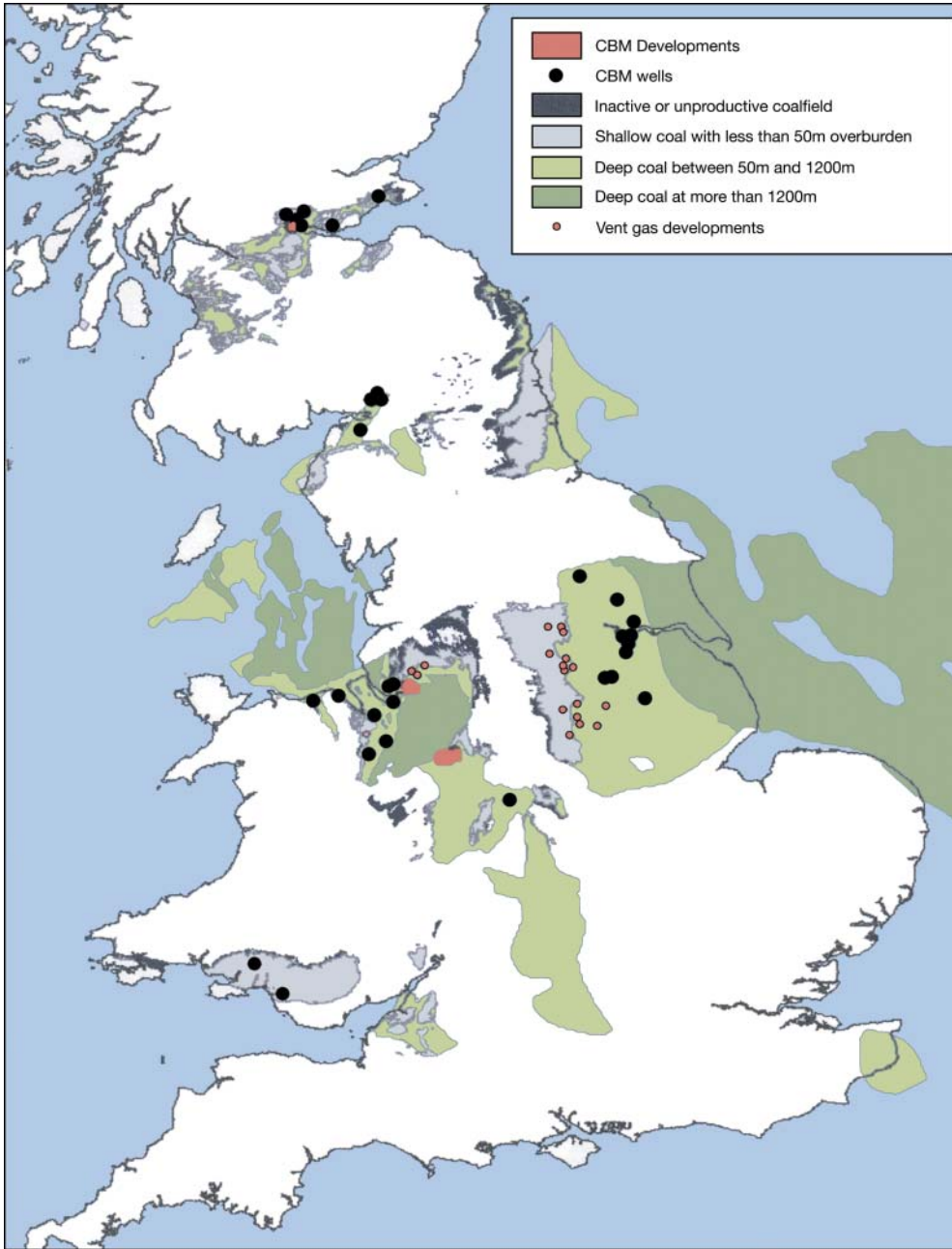


Figure 11.16 Extent of Westphalian Coal Measures and CBM development in the UK (DECC 2013).

## 11.3 Underground Coal Gasification

The concept of underground coal gasification (UCG) of coal was first envisaged by Sir William Siemens (1868) and Mendeleev (1888). A patent was issued in the UK in 1909, but there was no follow-up. It was not until the 1930s that tests were carried out in the former USSR, where a number of field stations were established for the purpose of developing a workable underground gasification technology. This led to the establishment of a number of large industrial installations that have supplied a low calorific value (CV) gas to power stations and other industrial consumers. Improvements have been made to the process since the 1970s, largely through the extensive field and laboratory testing conducted in the USA and Europe during the 1970–1980 energy crisis. Commercialisation never took place due to a twenty-year depression in crude oil/natural gas prices. In recent years, successful demonstrations have been carried out at depths of 250 m. Trials have also taken place in Europe, but at depths of 500 m as the coals are typically at deeper levels. These trials are designed to enable the technology to access unmined deep coals (>1000 m), such as those in Belgium, Poland, and the UK. Around 50 trials have been undertaken principally in Australia, PCR, Europe, Russia, and the United States. The result has been that, since 2010, 10 UCG projects are in operation or in advanced stages of development, together with 31 projects identified for potential development (Lavis et al. 2013).

### 11.3.1 Underground Coal Gasification: The Case For and Against

The case for developing UCG has advantages and disadvantages. Burton et al. (2007) outlined the numerous advantages of UCG over conventional underground or opencast mining and surface gasification as follows:

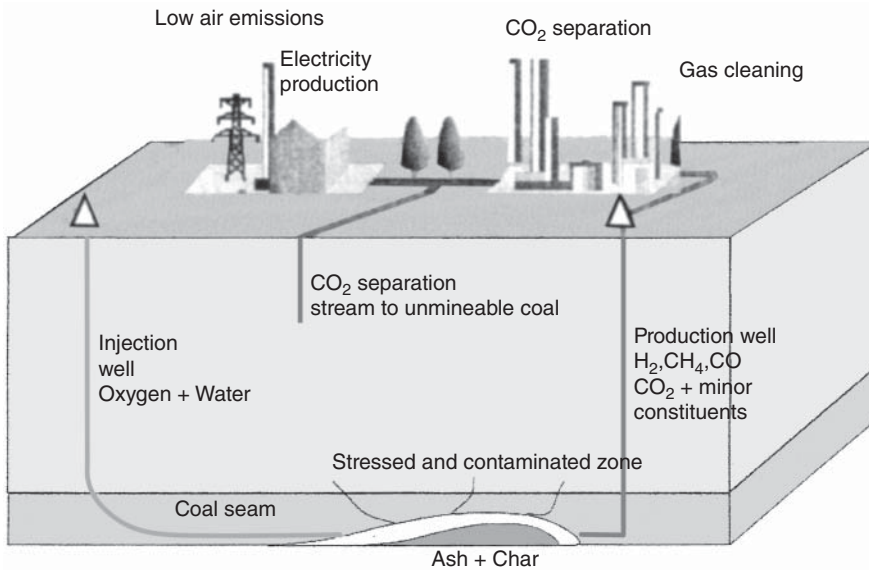
- (i) Conventional coalmining is eliminated with UCG, reducing operating costs and surface damage and eliminating mine safety issues.
- (ii) Coals that are unmineable (too deep, low quality, too thin) are exploitable by UCG, thereby increasing domestic resource availability.
- (iii) Surface transportation of coal is eliminated, thus reducing cost, coal stockpiling, and shipping.
- (iv) No surface gasification facilities are required, reducing capital costs.
- (v) Ash in coal remains underground, avoiding ash disposal at the surface.
- (vi) A reduction in sulfur oxides, nitrogen oxides, and other pollutants.
- (vii) UCG produces less GHGs than conventional mining and surface combustion do. The well infrastructure for UCG can be used subsequently for CO<sub>2</sub> sequestration operations.
- (viii) Coal seam gas technology can extract 3–5% of the energy (as CH<sub>4</sub>) in the coal, whereas UCG extracts more than 80% of the coal's energy.

These advantages are illustrated in Figure 11.17, which shows the layout of a commercial UCG site for power generation (UCG Engineering Ltd 2006).

UCG technology is still not perfect, and there are a number of limitations; namely:

- (i) UCG can have serious environmental implications, particularly aquifer contamination and ground subsidence. Site selection and operation need to be carefully assessed.
- (ii) Even when UCG may be technically feasible, the selection of a number of coal deposits may be limited due to geological and hydrogeological factors that increase environmental risks to unacceptable levels.
- (iii) UCG operations cannot be controlled to the same extent as surface gasifiers can. Water influx, distribution of reactants





**Figure 11.17** The main components of a commercial UCG site for power generation (UCG Engineering 2006).

in the gasification zone, and the growth of the cavity can only be estimated from measurements of temperatures and product gas quality and quantity.

- (iv) Until a reasonable number of UCG-based power plants are built and operated, the economics of UCG has major uncertainties.
- (v) UCG is inherently an unsteady-state process, and both the flow rate and heating value of the product gas will vary over time. Any operating plant must take this factor into consideration.

### 11.3.2 Underground Coal Gasification Technology

#### 11.3.2.1 Coal Gasification Reactions

The chemistry involved in UCG is complex, but essentially the C gasification reactions, as outlined by the Energy Technology Support Unit (ETSU 1993), Burton et al. (2007), and Paul Ahner (personal communication, 2011; pfaconsulting@sbcglobal.net), are as follows:

- (a) The primary oxidation reaction. When  $O_2$  is passed over/through hot coal it combines

with C to form CO, which is a flammable gas. This is an exothermic reaction (i.e. it produces heat). If too much  $O_2$  is supplied, then  $CO_2$  is produced which is inert, therefore careful regulation is required.

- (b) The steam-char reaction. When water or steam is passed over/through heated coal, the  $O_2$  in it combines with C to form CO and releases the  $H_2$  content. This mixture of CO and  $H_2$  produces a high-CV gas. However, this reaction is endothermic (i.e. requires the addition of heat to sustain it). The conventional gasification method is to alternate the two reactions. However, in UCG, the two would be carried out simultaneously, the reaction temperature being regulated by adjusting the  $O_2$ /water ratio.
- (c) The  $CO_2$  reduction reaction (reverse Boudard reaction), when  $CO_2 + C$  produces CO.
- (d) The water-gas shift reaction, when  $CO + \text{water}$  produces  $H_2$  and  $CO_2$ .
- (e) The  $CH_4$  synthesis reaction. At the high pressures achievable in the gasification of deep coal,  $H_2$  combines with C to form

$\text{CH}_4$ . This reaction is beneficial, in that it increases the CV of the product; and since it is exothermic, it reduces the amount of input  $\text{O}_2$  required to gasify a unit mass of coal.

- (f) Pyrolysis: when coal + heat produces  $\text{CH}_4 + \text{CO} + \text{H}_2 + \text{light hydrocarbons}$ .

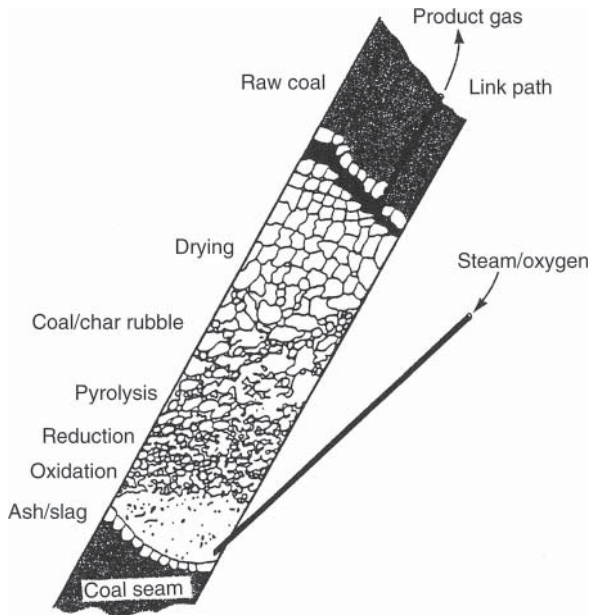
Since the desirable reactions require high temperatures and excess C (from coal), process efficiency is increased by confining the heat from the oxidation to the coal seam as much as possible, providing good gas/hot C contact and decreasing excessive water influx into the reactor. To keep the potential contaminants inside the reactor, some groundwater influx is needed; they can then be destroyed in the gasification process or produced with the product gas. Inorganic components from the coal ash, which remains underground, can leach into the groundwater to make it more saline, which is one reason to operate in saline, unusable aquifers (P. Ahner, personal communication, 2011).

Before gasification can proceed, some coal seam preparation is required. The permeability of the coal between boreholes (wells) must be enhanced; this is normally done through reverse combustion linking or directionally drilled linking, or a combination of both.

In reverse combustion linking, two wells are drilled 30–100 m apart into the coal seam; coal at the base of one well is then ignited (the ignition well) and high-pressure air or steam is injected down the other well (the injection well). Air permeates through the coal from the injection well and intersects the combustion zone at the base of the ignition well. This zone will expand and follow the air source back to the injection well, creating a high-permeability channel between the two wells. In directionally drilled linking, a borehole is drilled between the vertically drilled well pair. This requires directional drilling techniques, whereby an initially vertical borehole can be deflected at an angle to coincide with the dip of the coal seam. This is the preferred method for establishing the link because the inclined borehole

can be placed near to the base of the coal seam, thereby increasing resource recovery. It is usual for reverse combustion linking to then be used after directional drilling to complete the mechanical link. To further enhance gas circulation through the coal seam, it may undergo ‘hydrofracking’, a process that applies pulsating hydraulic pressure to produce fracturing of the coal seam.

Other configurations have been developed and tested. In steeply dipping coal seams, an inclined production well is drilled near to the base of the coal seam and an inclined injection well is then drilled below the coal seam and only enters the seam at the area to be gasified where it intersects the production well (Figure 11.18). Another method, developed by the Lawrence Livermore National Laboratory in the USA, is termed the controlled retracting injection point (CRIP). A horizontal injection well is drilled at the base of the coal seam into which is also drilled a vertical and a horizontal production well. The vertical ignition/production well is used during process start-up, after which the horizontal production well is used. A liner is inserted in the injection well and a mobile igniter-burner is placed inside the liner. The igniter-burner can be retracted and used to burn off the liner, exposing unburnt coal to the process. Figure 11.19 illustrates an example of a commercial UCG before and during gasification. A third vertical borehole can be drilled, offset by at least 50 m from the line of the two original boreholes. Water and  $\text{O}_2$  can be injected via this borehole to achieve a link between this and the initial reaction area, which should have increased the natural permeability of the coal seam (Figure 11.20). This will allow a lateral expansion of the reaction zone, and the process can be repeated to further increase the area of production (Oliver and Dana 1991). Observations made during tests near Hanna, Wyoming, in 1987, suggested that groundwater has a profound influence on the UCG system. Not only is the volume of water used substantial, but the resulting impacts on the hydrogeological



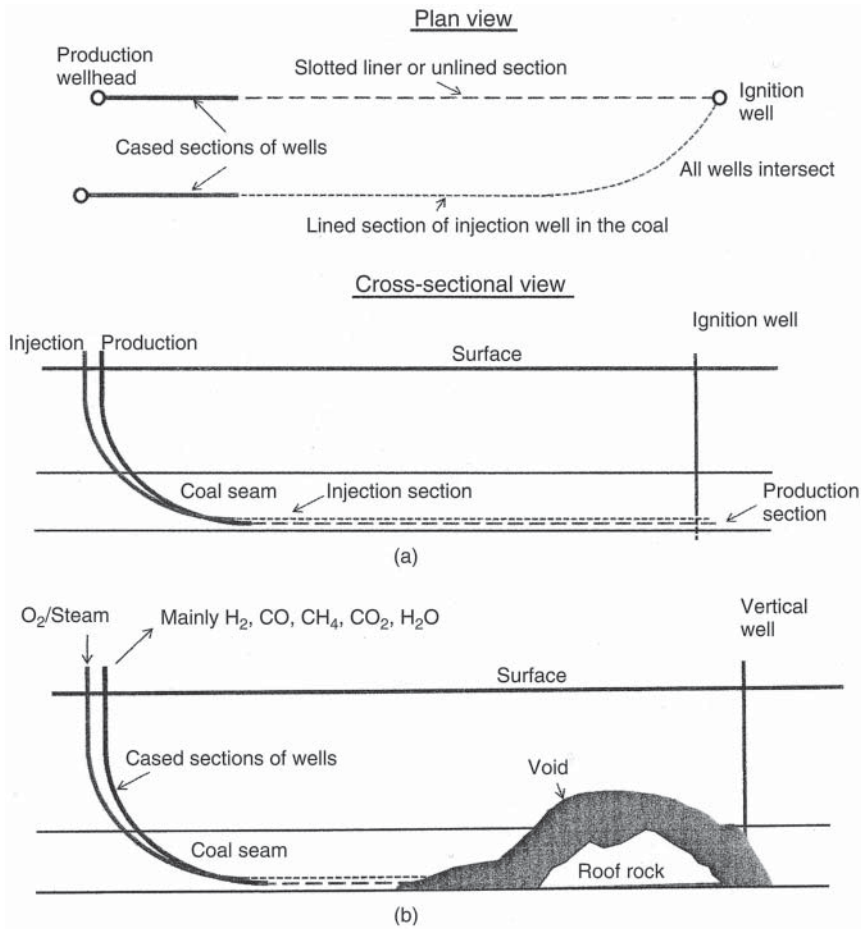
**Figure 11.18** Typical configuration for steeply dipping bed gasification.  
 Source: From Oliver and Dana (1991).

system in the fractured coal medium were felt over a substantial area. During the test it was estimated that 2% of the water available in the coal seam was consumed during the test. This substantially lowered the hydraulic head on the site and also affected head beyond the site area. The relationship of hydrogeological conditions under high- and low-pressure operating conditions are seen in Figure 11.21 (Beaver et al. 1991). When the cavity pressure was less than the hydrostatic pressure, groundwater moved towards the cavity, where it was converted to steam, resulting in a loss of head. High-pressure conditions at the gasification centre initiated movement of product gas up dip, and the gas, being less dense than water, displaced the water at the top of the coal seam; and as migration progressed, gas was detected in the surrounding monitoring wells. Mao (2016) made comparisons between composition and the CV of gas products from a number of projects in Australia, the PRC, Russia, and the USA (Table 11.5). The CV of the gas produced is linked to the water and  $O_2$  content of the gasification agent injected. A lower CV of the gas results when air is injected, but a higher CV is achieved when using ‘ $O_2$ -rich

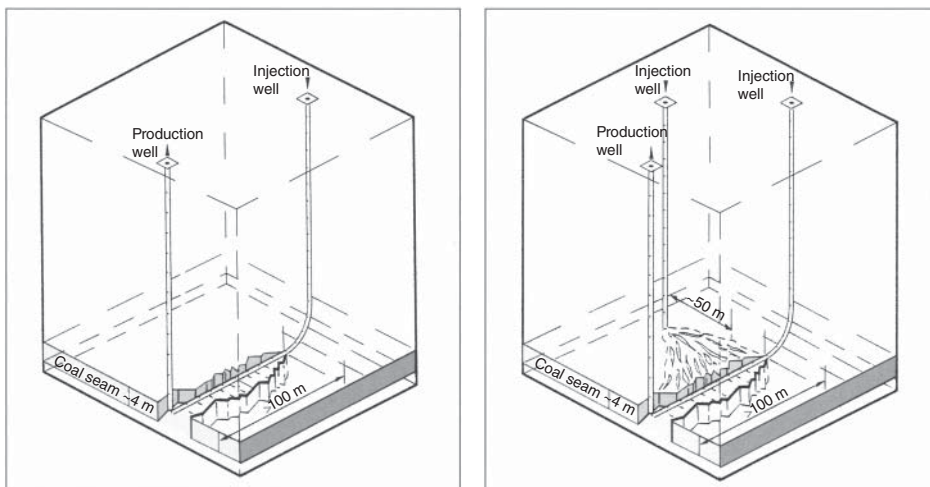
vapour’ ( $O_2$ -rich water gas). Comparisons of gas compositions are listed in Table 11.5 (Mao 2016).

Therefore, in order to prepare an area for UCG, a detailed knowledge of the geology and hydrogeology is required. An example of the desired underground gasification site characteristics is given in Table 11.6 (P. Ahner, personal communication, 2011). These include the depth, thickness, quality, structural condition, and hydrogeological character of the target coal seam, together with knowledge of the properties of the overlying strata. Vertical boreholes are drilled in the conventional way, but curved or ‘deviated’ boreholes require detailed knowledge of the depth and dip of the coal seam. Because of the problems associated with drilling the more difficult directed injection boreholes, it is preferable, where possible, to drill it first and adjust the position of the production well(s) to match rather than the other way around.

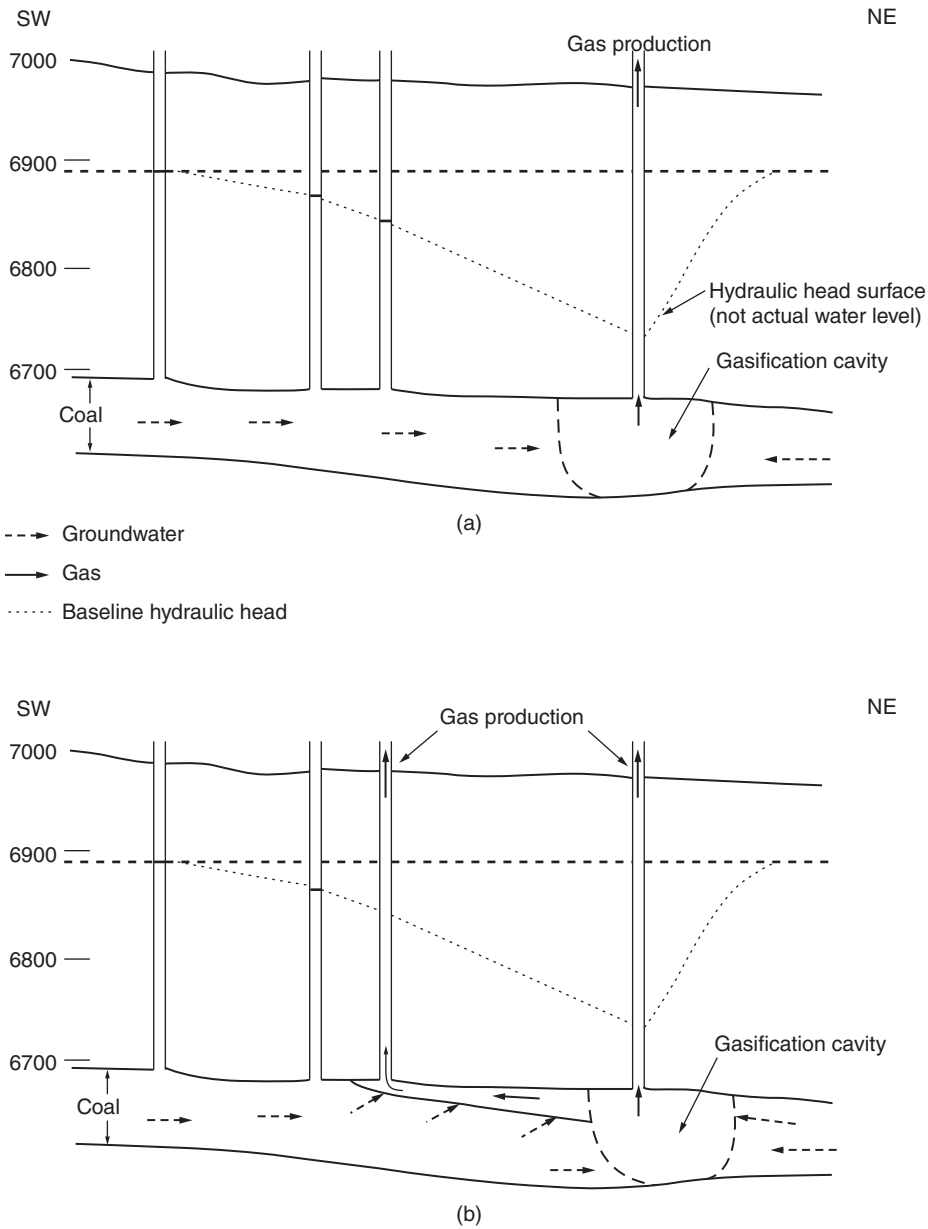
As described above, the first objective of UCG is to achieve ignition of the coal, by producing a gasified channel by high-pressure injection; the second phase is then to extend the area of combustion by drilling additional



**Figure 11.19** (a) Plan and cross-section views of a UCG panel before gasification (P. Ahner, personal communication, 2011). (b) Cross-section view of an operating UCG panel showing roof fall (P. Ahner, personal communication, 2011).



**Figure 11.20** (a) Injection and production wells using the CRIP technique. (b) Addition of an offset well designed to extend the combustion zone laterally. Source: From ETSU Report (1993).



**Figure 11.21** Schematic diagram of hydrogeologic relationships under high- and low-pressure operating conditions during the Rocky Mountain 1 UCG test. *Source:* From Beaver et al. (1991). (a). Under typical operating conditions (cavity pressure less than hydrostatic), groundwater moves towards the cavity where it is consumed in the process and converted to steam, resulting in loss of head. (b). Under elevated operating pressures, product gas migrated up-dip along fractures and was detected in groundwater monitoring wells in the south-west part of the site.



Table 11.5 Parameters of coal gas products of some representative UCG projects (Mao 2016).

Country	Project name	Technique	Agent injected	Gas composition (%)						Net CV (MJ m <sup>-3</sup> )	
				CH <sub>4</sub>	H <sub>2</sub>	CO	C <sub>m</sub> H <sub>n</sub>	CO <sub>2</sub>	N <sub>2</sub>		
Soviet Union	Ergo exergy	Soviet Union UCG	Air	1.8–1.9	17.6–19.7	4.5–7.5	0.2–0.3	17.2–20.0	42.7–58.0	3.18–3.56	
	Geer Krumlov		Air	2.8	29.4	19.5		18.3	27.8	6.67	
	Lehizensk		Shaftless	Air	2.38	15.8	7.62	0.19	32.2	38.33	3.67
	Moscow suburbs		Air	1.9	35	15.3	0.4	28.1	16.2	6.71	
USA	Maas mine	CRIP	Air	6.3	37	25.8		28.8	1.4	9.55	
	High Vegas		Air	4.1	24.5	21.2	0.3	47.8		7.04	
Australia	North Wyoming	Shaftless UCG shaftless	Air	4	20	10		16	50	3.77–5.44	
	Chinchilla, Queensland		Air	18	32	17				5	
PRC	Xinhe, Xuzhou		Air	2.91	23.58	7.78			52.25	5.02	
			O <sub>2</sub> -enriched vapour	11.05	59.9	12.64			2.84	13.65	
			Air	2–4	10–20	5–25			40–65	4.18–5.86	
	Liuzhuang, Hebei	LLTS-UCGP	O <sub>2</sub> -enriched vapour	7.8–14.1	40.6–53.1	11.2–28.1			5.1–17.9	13.78	
	Hebi, Henan		Air	2	13	20			59	4.73	
	Feicheng, Shandong		Shaft	3–5	15–25	5–8			40–65	3.76–5.02	
	Xiyang, Shanxi			Air	2–4	15–20	5–10			40–65	4.18–5.02
				O <sub>2</sub> -enriched vapour	7–10	45–55	10–15			10–15	11–13
				Air	3.04	24.59	3.37			52.81	4.22
	Suncun, Xinwen			O <sub>2</sub> -enriched vapour	9.28	52.86	9.32			5.6	11.36
				Air	1.51	16.34	12.73	0.25	17.51	51.66	4.1
	Huating, Shanxi	Reverse, shaft		O <sub>2</sub> -enriched vapour	3.52	36.19	29.61	0.49	25.73	4.46	9.27

LLTS-UCGP, long passage, large cross-section, two stage, positive and negative direction and reverse combustion gasification.

**Table 11.6** Desired underground coal gasification site characteristics.

Characteristic	Comments
Coal below water table	Required to form a water-tight seal to contain the process
Minimum coal seam thickness 3 m	A thicker coal is more economical to gasify because both the coal access per well and the thermal efficiency increase with the coal thickness
Coal must be 'gas tight'	Fractures that extend to the surface and proximity to mine workings could prevent the coal from being extinguished due to air infiltration after the process is shut down. These fractures could also permit product gas loss during operation
Minimum depth 150–200 m	The deeper the coal the greater chance that (i) it is not being used for groundwater or near an aquifer that is being used, (ii) the water is too saline to be useful, (iii) immediate roof fall above the gasified zone will not cause intercommunication between fresh and saline aquifers, and (iv) surface subsidence will not occur
Maximum depth is variable	The primary limit on depth is dictated by economic reasons. Depending on energy prices, there will be a limit where increased drilling costs will make UCG uneconomic. The greater operating pressure generally required with depth can also increase the capital and operating expense, but this increased pressure can be offset by savings on the gas clean-up side
Coal must be a confined aquifer	A coal bounded by non-permeable strata will (i) decrease the chance for excessive groundwater influx into the UCG reactor, (ii) limit the amount of regional water table drawdown, (iii) provide better pressure maintenance in the reactor for a more efficient operation, (iv) contain the potential contaminants within the reactor zone to allow their destruction in the reactor or their safe production with the product gas, and (v) decrease the produced water rate during operations, which reduces surface water treatment costs
Coal hydraulic conductivity (permeability)	Coal permeability below 2 md is desirable but not strictly necessary since the primary reactor confining mechanism is the hydrostatic head on the coal. However, increasing permeability generally increases the product water rate and water handling costs
Resource size	The larger the resource the greater the economies of scale. A 100 Mt resource could supply a 500 MW power plant for 25 years, assuming that 62% of the coal is gasified and power generation efficiency is 43%
Coal rank	Lignite to bituminous coals are amenable to UCG. A swelling bituminous coal with an FSI > 3 can present plugging problems that will require special consideration. The folded and faulted nature of anthracite coals precludes it from UCG
Coal structure	Steeply dipping strata at 63° have been very successfully gasified in the USA. The increased cost of drilling as the process moves further down dip must be considered. UCG in coal seams dipping <5° is more economic for this reason. Steeply dipping coal seams will be more fractured and leaky where the transition from the horizontal to the vertical orientation occurs. Sufficient distances from faults and structural disturbance must be maintained to ensure leak-tight operations
Ash and moisture content <50%	There are no inherent problems performing UCG on high-ash or high-moisture-content coals, but the economics of the process will suffer since the amount of energy recovered per kilogram of coal via UCG is proportional to the CV of the coal
Bounding strata competency	Roof fall is inevitable in UCG. The overburden should be strong enough to form a stable arch over the gasified void. This arch has been observed to be as high as two to three times the coal thickness. Field test data, strength data from cores, and mining experience should allow estimates of roof fall to be made
Coal seam proximity	If roof fall creates communication between an upper coal seam and the target coal seam, the effects of potential simultaneous gasification of both coal seams must be considered. This situation can possibly improve the economics by using the same well to access multiple coal seams

Source: From P. Ahner (personal communication, 2011).

**Table 11.7** Efficiencies from typical large US UCG pilot test (P. Ahner, personal communication, 2011).

	Rocky Mountain 1	Rawlins 2	Hanna	Hoe Creek
Tons of coal gasified <sup>a</sup>	11 227	8560	14 182 (sum of five tests)	3251
Percentage energy in dry gas (%) <sup>b</sup>	79.3	81.9	79.3 (average from five tests)	73
Percentage energy in tars (%)	4.9	1.8	—	—
Percentage energy in sensible heat (%)	3.7	5.5	—	—
Percentage energy in steam produced (%)	6.1	5.3	—	—
Percentage energy lost to formation (includes product gas) (%)	6	5.5	—	—

a) All public US tests that gasified more than 2000 tons of coal.

b) Percentage basis: total energy produced/total energy in the coal gasified, determined from a C balance.

injection wells. The final phase of the UCG operation is to extinguish the fire. This is done by injecting N<sub>2</sub> into the reaction area, and then after a time lapse of several days the underground cavities are filled with water.

The high cost associated with UCG field trials requires an efficient control and monitoring system. This will include the development of a series of boreholes that will indicate the extent of the burn zone by detecting any temperature rise, by the low flow of gas as the burn zone nears the monitoring boreholes, and when 'breakthrough' occurs.

Well-documented larger US field tests yielded the thermal efficiencies shown in Table 11.7. These efficiencies, which are comparable to surface gasifiers, strictly compare the energy in the total coal gasified versus the energy produced. They do not represent the total energy required for the total process, as energy is required for well drilling, gas clean-up, etc.

### 11.3.3 Global Development of Underground Coal Gasification

A number of UCG stations were built in the former USSR in the 1950s, at Tula, Yushno–Abinsk, Shatsky, and Angren; all were

designed to burn lignite. In addition, UCG installations were constructed at Lisichansk and Kamanskaya in bituminous and anthracite coals. These installations produced gas at varying CVs (Table 11.8), but only the stations at Yushno–Abinsk and Angren remained in operation after 1980 (Douchanov and Minkova 1997).

In the USA, tests at the Energy Center in Wyoming showed that hydrofracturing of coal at 120 m depth enhanced the permeability and in 1973 produced 8.5 MCF (0.24 MCM) gas per day, and the Morgantown Energy Center has produced gas at 3.5 MCF (0.1 MCM) per day with a CV of 1100 kcal m<sup>-3</sup> from coal seams

**Table 11.8** UCG production in the former USSR.

Installation	Production 10 <sup>6</sup> m <sup>3</sup> gas yr <sup>-1</sup>	CV (kcal m <sup>-3</sup> )
Tula (Russia)	400	750–850
Yushno–Abinsk (Kuzbass Basin)	100	1000
Shatsky (Russia)	200	800
Kamenskaya (Donbass Basin)	730	900
Lisichansk (Donbass Basin)	120	850

Source: Based on Douchanov and Minkova (1997).

275 m deep in West Virginia (Douchanov and Minkova 1997). The Lawrence Livermore Laboratory in Wyoming has produced gas at 1.8 MCF (50 000 m<sup>3</sup>) per day from coal seams 150–900 m deep and has carried out UCG tests at the Hoe Creek UCG site in Wyoming, targeting a 30 m coal seam at c. 300 m depth. The field demonstration of UCG was situated below the water table so that uncontrolled burns could be prevented by stopping the injection. However, active gasification introduced toxic volatile organic compounds into the aquifers as a result of migration of contaminants derived from gasification by-products formed by the pyrolytic breakdown of the coal. These problems were further exacerbated by subsidence and collapse of the cavity roof, which resulted in the interconnection of, and the contamination of, the three aquifers. Groundwater contamination occurred in only 2 of the 30 US field tests, and lessons were learnt.

At the same time, tests have been carried out in Europe, the UK, France, Czech Republic, Italy, Hungary, and Spain. The coal seams in Europe are characterised by their greater depth and relatively thin development. This means that, to achieve the successful development of any UCG operation, a number of technical difficulties will have to be overcome, such as the effective linking in the coal seam, the control of the gasification front, and overall control of the multistage gasification process.

In 1988, six member states of the European Union formed a European Working Group on UCG. To demonstrate the commercial feasibility of UCG, field trials were carried out together with the development of a semi-commercial plant. The trials were carried out in the Teruel region of Spain at El Tremedal, and two coal seams 1.9–7.0 m in thickness were targeted at a depth of 600 m (Figure 11.20). The coal was subbituminous in rank and had low permeability, and the test area was at least 200 m from any significant faults. As demonstrated in the USA, the test used the CRIP method to control the enlargement of the gasifier; this was seen to be successful. The influx of groundwater

was sufficient to meet the requirements of the chemical reactions of gasification, but during the test was uncontrolled, which created the risk of quenching the reactor. The trial also experienced significant gas losses from the gasification zone to the surrounding strata; this was partly due to permeable strata above the coal and a working pressure above hydrostatic. However, overall, the trial was a success and demonstrated that the production gas had a quality and heating value consistent with the theoretical estimates and would appear suitable for industrial use (Green 1999).

As a result of the Spanish trial results, the DTI in the UK identified UCG as one of the potential technologies for the development of the UK's large unmined coal resources. Detailed work has been carried out on the geological and hydrogeological criteria required for UCG (ETSU 1993).

In Australia, the Chinchilla UCG project in Queensland was run from 1997 to 2003. The long-term goals of the project were power production and liquid fuels production using gas-to-liquids technology. Nine process wells were drilled and produced syngas from a 10 m thick coal seam at a depth of 140 m at a rate of 80 000 Nm<sup>3</sup> h<sup>-1</sup> (Nm<sup>3</sup>: normal cubic metre) at a lower heating value of 5.0 MJ Nm<sup>-3</sup> at a pressure of 10 barg and temperature of 300 °C. Other UCG testing is being carried out at nearby Bloodwood Creek, Queensland, to develop Australia's first commercial power generation from UCG syngas.

The PRC has had the largest UCG programme and includes 30 projects in different phases of development. These include the Xinhe #2 mine test, the industrial trial at Liuzhuang mine in Tangshan, and Xin-Wen's tests at Suncun in Shandong and the Caozhuang mine in Feicheng. The method utilises the abandoned galleries of coalmines for the gasification. Vertical boreholes are drilled into the galleries to act as injection and production wells, and a system of alternating air and steam injection is used to improve the production of H<sub>2</sub>. The PRC also developed an

industrial pilot UCG project in the Northern Inner Mongolia Autonomous Region at Gonggou mine, targeting a coal seam at 200 m depth. This was expected to produce 1.5 MCM per day of syngas and generate  $32.4 \times 10^6$  kWh yr<sup>-1</sup> of power by 2010.

In South Africa, Sasol and Eskom have UCG pilot facilities that have been in operation for some time. Other countries are undertaking UCG field trials, including Canada, India, Kazakhstan, New Zealand, Slovakia, Slovenia, and Vietnam.

Although a number of countries would like to benefit by obtaining energy from their deep-lying coal resources, the question of the cost of producing this energy by UCG still places it in a minor role as an energy provider from coal sources. However, in the right circumstances, UCG appears to be commercially viable; however, several key scientific and technical gaps remain that further research should address. One important possibility is that the challenge of managing CO<sub>2</sub> emissions can link UCG with carbon capture and storage (CCS).

The advance of UCG technology and the placing of UCG into a major global energy industry will mean that regulators will have to adapt their regulatory and licensing regimes to allow UCG to take place. This will include the development of an industry-wide standard for UCG resource assessment. The Underground Coal Gasification Association is engaged in the development of such a new formal resource assessment method (Lavis et al. 2013).

## 11.4 Coal as a Liquid Fuel

### 11.4.1 Petroleum Potential of Coal

Coal-bearing sequences are essentially non-marine in nature and have been estimated to account for less than 10% of the world's oil, and much of this non-marine contribution is derived from lacustrine source rocks, accounting for 85–95% of the oil in areas, such as Brazil, the PRC, and Indonesia (Fleet

and Scott 1994). However, in spite of being a minor contributor to the world's oil resources, oil-prone coal sequences are considered as significant oil-source rocks in South-East Asia and Australasia, whereas the role of coal sequences in providing oil in North America and western Europe is much more debatable. The identification of oil derived from coals has important implications; recognition of such oils in a basin can indicate the presence of coals not previously identified. The oil-generating and -expelling potential of humic coals is volumetrically more significant than that of sapropelic coals.

Geological processes acting on coal-bearing sequences have produced hydrocarbon reserves, which may have been preserved as oil, condensate, and wet and dry gas.

Littke and Leythaeuser (1993) listed properties that could have been important for petroleum generation from coals:

- (i) The H<sub>2</sub>-rich and hydrocarbon-generating maceral group liptinite may be present in significant quantities in some coals.
- (ii) Coals produce pyrolysis products when heated.
- (iii) Coals are a major source of CH<sub>4</sub>.
- (iv) Bitumen is contained in bituminous coals, which can be extracted using solvents.
- (v) The coalification process produces a loss of volatiles, including hydrocarbons.

### 11.4.2 Coal Properties as an Oil-Source Rock

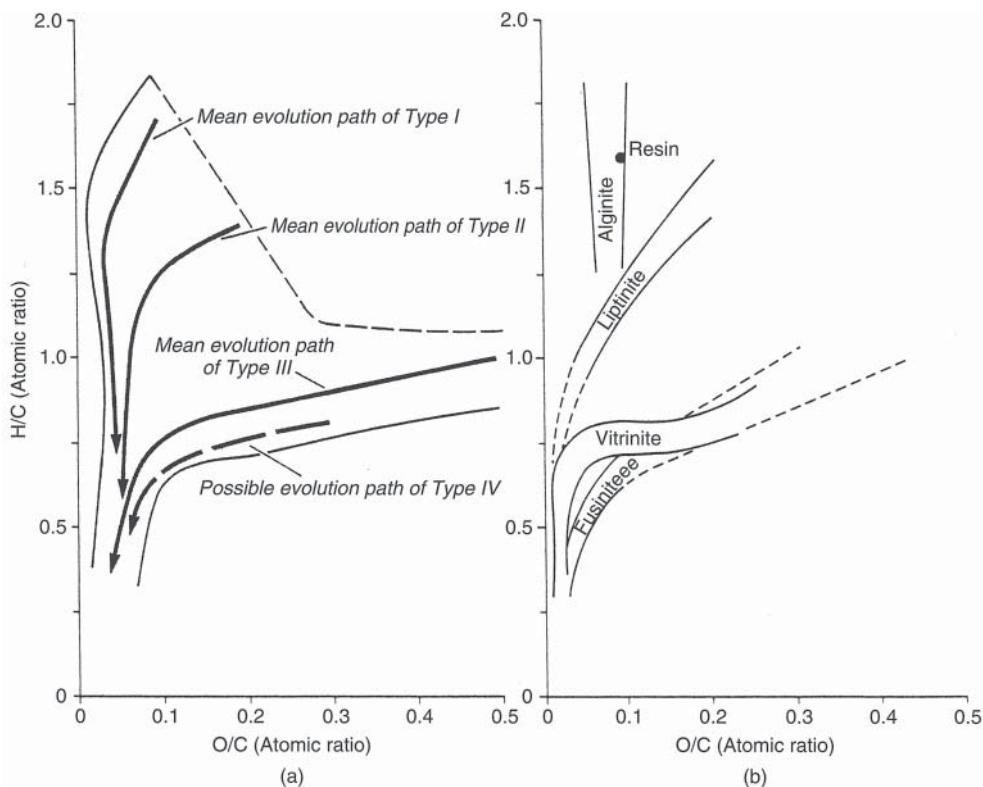
In order to comprehend how coals or coal-bearing sequences have the ability to expel petroleum in the liquid phase, it is necessary to understand the quantity and quality of liquids and gases that coal-bearing sequences can expel in response to their thermal and structural history. Laboratory techniques have been used to try to distinguish oil-prone coals from other coals; these indicate that oil-prone coals are richer in H relative to C. It is generally accepted that sediments must



contain moderate to high concentrations of H-rich kerogens in order to have significant oil-source potential. The progenitors of H-rich kerogens are derived from vascular plants, lacustrine algae, photosynthetic bacteria, and in-sediment bacteria. To have oil potential there has to be a combination of sedimentary and environmental processes that have enhanced the production and preservation of the organic constituents (Thompson et al. 1994).

The composition of macerals reflects the original composition of the plant precursors, although having been substantially modified during the biochemical and early thermal stages of coalification. Liptinite (or exinite) is richer in H than vitrinite is, which in turn is richer than inertinite is. These maceral groups occupy different coalification pathways with thermal maturation, and are

similar to the pathways for Types I–IV organic matter (kerogen) defined for sedimentary rocks in general (Figure 11.22). The maceral composition of oil-prone vitrinite-rich and liptinite-poor New Zealand coals shows that liptinite macerals are not the only, nor necessarily the most important, source of liquid hydrocarbons in certain coals. Rank and maturity of coals must be determined before an accurate thermal history can be identified. Vitrinite reflectance has been a good indicator of rank, but the identification of vitrinite reflectance suppression has caused problems. Vitrinite reflectance suppression is caused by variation in the H content of vitrinite macerals whereby anomalously low reflectance is shown by H-rich vitrinites. To avoid this problem, oil-prone New Zealand Cretaceous and Palaeogene–Neogene coals are classified using Suggate rank *S* (see Figure 4.9b), which is



**Figure 11.22** Comparison of kerogen types, evolution paths, and petrographic components of coal based on atomic ratios. *Source:* From Powell and Boreham (1994).

based on volatile matter versus the CV of coals and has proved useful in petroleum potential studies of New Zealand coals (Suggate 2000).

Whatever the age, some coals contain oil-prone macerals of one sort or another. Of these, Mesozoic and Palaeogene–Neogene sequences are considered the more likely as source rocks. The complex relationship of maceral types and their composition is comprehensively reviewed by Wilkins and George (2002).

Studies of organic matter or kerogen types suggest that, in order for a source rock to have hydrocarbon potential, 10–20% of its organic matter must equate with Type I organic matter, or 20–30% must equate with Type II organic matter. The bulk H/C ratios would therefore be in the range 0.8–0.9, or H indices in Rock-Eval analysis would be above 220–300 mg hydrocarbon per gram of C before oil expulsion is considered (Powell and Boreham 1994).

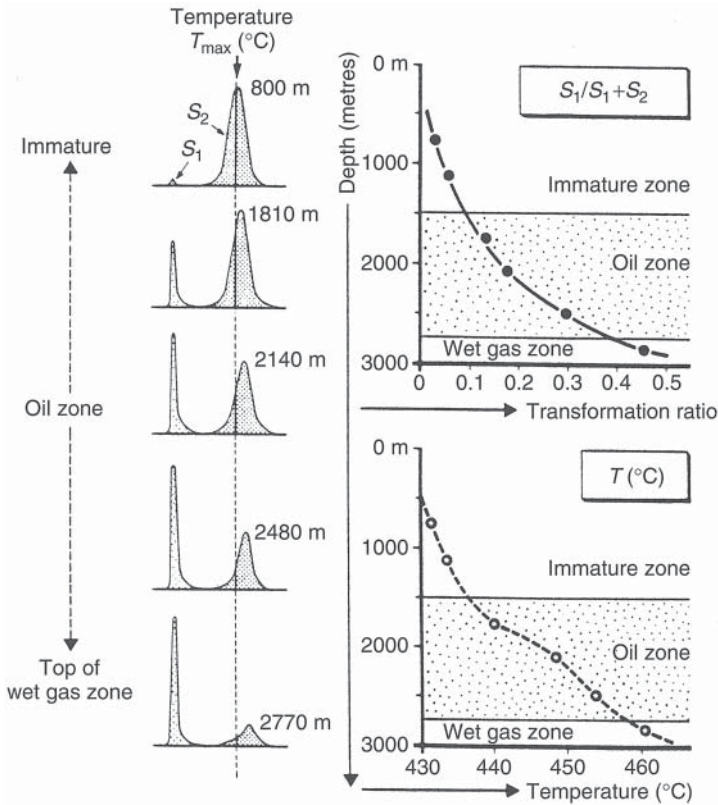
The use of the petrographic composition of coal as an indicator can have limitations, and it is possibly the association of macerals, the microlithotype, rather than the macerals themselves, that controls the expulsion of liquid petroleum (Fleet and Scott 1994). Clues to this type of source can be high wax and low sulfur content in the oil (Hedberg 1968). Biomarker molecules in the oils, which can be linked to land plant communities, can add further evidence to an origin for the oil. The character of the vegetation component in coal deposition will have changed through geological time as plant communities have evolved, being influenced by climate and environmental change. The Jurassic coals of Australia have a dominance of conifers in swamp floras, whereas the Late Cretaceous coals contain angiosperm flora. Both of these provide an abundant amount of potentially oil-prone material that has been preserved as exinite. The Palaeogene–Neogene oil-prone coal sequences of South-East Asia have resulted from deposition of oil-prone detritus in coastal plain environments under wet tropical conditions (Fleet and Scott 1994). These two sets of conditions have led to

the suggestion that the significant oil-prone coal-bearing sequences are restricted to Late Jurassic–Palaeogene–Neogene basins of Australasia and the tropical Palaeogene–Neogene basins of South-East Asia (Macgregor 1994).

Because H is the significant factor in the generation of hydrocarbons from sedimentary organic matter, it is suggested that the hydrocarbon potential of terrigenous organic material may be expressed as a ratio of H-poor and H-rich components. Rock-Eval pyrolysis records the release of hydrocarbons and CO<sub>2</sub> with increasing temperature and determines the temperature of maximum hydrocarbon generation  $T_{\max}$ . Free hydrocarbons ( $S_1$ ) already present in the rock are liberated at low temperatures, whereas newly generated hydrocarbons ( $S_2$ ) are given off at higher temperatures. The transformation ratio  $S_1/(S_1 + S_2)$  and  $T_{\max}$  both increase with increasing maturity, as shown in Figure 11.23, which illustrates the characterisation of source rock maturity by pyrolysis methods (Taylor et al. 1998). The ratio  $S_2/\text{TOC}$  (total organic C, expressed in weight per cent) or H index correlates with the atomic H/C ratio measured by elemental analysis on kerogen.

Isotopes can be used to characterise the total C or bulk fraction of a kerogen, oil, or gas. Isotope analysis is considered most useful for characterising gases, and therefore is important in studying the petroleum generated from coal-bearing sequences, which, although containing liquid products, also contain a high proportion of gas.

Coal quality requirements vary according to the method used for coal liquefaction. In both pyrolysis and hydrogenation, the use of low-rank coals with high H contents enhance the liquid yields. The required high H/C ratio is closely linked to rank and petrographic composition; the latter requires a high proportion of reactive components in the coal, such as vitrinite and liptinite. Liptinite remains highly reactive over a large range of rank, whereas vitrinite first increases and then decreases with increasing rank. Inertinite shows varying



**Figure 11.23** Characterisation of source rock maturity by pyrolysis methods. Transformation ratio and/or peak temperature  $T_{\max}$  may be used as indicators of thermal evolution. *Source:* From Taylor et al. (1998).

degrees of reactivity in coal liquefaction (Taylor et al. 1998).

In hydrogenation, the use of coals containing high amounts of  $O_2$ ,  $N_2$ , and sulfur means that H is consumed in the removal of these heteroatoms. Because the supply of H is expensive, this constitutes a financial loss in the process. Inorganic impurities, such as the mineral matter content of the coal, have been found to influence the liquefaction behaviour of coals. A high ash content can lower the reaction throughout and increase problems of solid and liquid separation, and it may deactivate any catalyst. Some inorganic material can have catalytic effects of their own, e.g. pyrite has favourable catalytic properties (Taylor et al. 1998).

The gasification processes are comparatively insensitive to coal properties and can utilise coals that would be unsuitable for other processes. For example, the Fischer–Tropsch

(FT) synthesis used by SASOL in South Africa uses inertinite-rich, high-volatile bituminous coal with a high ash content, a typical Gondwana-type coal.

Taylor et al. (1998) summarised those characteristics favourable for coal hydrogenation as follows:

Vitrinite reflectance	<0.8%
H/C atomic ratio	>0.75%
Vitrinite + liptinite	>60%
Volatile matter (daf)	>35%

Low concentration of heteroatoms

#### 11.4.3 Coal Liquefaction Technology

The world's coal resources are greater than the known oil resources; because of this, it is likely that the liquefaction of coal will be necessary to

provide synthetic fuel as a substitute for crude oil once oil sources begin to run out.

The conversion of coal to oil has been developed commercially since the 1920s, whenever oil supplies became unavailable. This has usually been due to physical and technical production constraints or for political reasons. Commercial production of coal-derived synthetic liquid fuels is still limited, the prime reason being the high cost of current coal-to-oil processes. Future crises in the oil industry may stimulate further development of coal-to-oil production.

There are various methods of coal liquefaction, the main problem being the deficiency of H in coal compared with liquid fuels. This can be overcome by adding H to the coal by a number of processes.

- (a) direct liquefaction by hydrogenation;
- (b) indirect liquefaction by the FT synthesis;
- (c) removal of part of the C content from the coal by pyrolysis.

The three processes differ in technology and in the yield of liquid and solid products.

The most direct method is by hydrogenation; to overcome the H deficiency, the H/C ratio is increased by adding an H donor. Coal is dispersed in a thermally stable 'solvent' and/or 'H donor' and passed into a pressurised autothermal reactor at temperatures between 400 and 500 °C. If additional H is not supplied to the reactor, the H-depleted solvent oil is itself rehydrogenated at a later stage. The reaction products are filtered and distilled to separate the solvent from the coal extract, which is subjected to vacuum distillation to produce distillate oil (Taylor et al. 1998).

Experiments have shown that liquid yields equivalent to 4 bbl per tonne of dry coal have been obtained by this method (Taylor et al. 1998). In practice, these figures would be lower, but still considerably higher than those from coal without an H additive.

FT synthesis was developed in Germany in the 1920s and formed the basis for the production of oil from coal in Sasolburg, South Africa,

the name SASOL having become synonymous with the process. The development of the SASOL plants was motivated by South Africa's long political isolation and attendant oil embargo. This overrode any financial considerations, and up to 60% of transportation fuel was supplied in this way.

FT synthesis involves the gasification of the coal, carried out in a Lurgi-gasifier to produce the synthesis gases CO, H<sub>2</sub>, and CH<sub>4</sub>. The CH<sub>4</sub> is treated in a gas reformer and synthesised in a Kellogg reactor. The CO and H<sub>2</sub> are subjected to fixed-bed FT synthesis by passing them through ovens containing circulating water with an iron or cobalt catalyst. Gasoline and other products can then be obtained by cracking the resulting synthetic crude oil. Around 34 Mt per year is used by SASOL in plants designed to produce 50 000 bbl per day of gasoline and other products for chemical feedstocks from the processing of 30 000 t per day of coal (Sage and Payne 1999).

Pyrolysis involves the heating of pulverised coal extremely rapidly in a vacuum, known as 'flash pyrolysis'. The feed coal passes through a plastic stage during which the macerals soften and decompose into gas, char, and tarry liquids. The tars are hydrogenated to produce heavy or light oil as required.

#### 11.4.4 Future Development of Coal Liquefaction

A large number of studies have been carried out to produce liquid hydrocarbons from coal. This has been triggered in recent times by the oil crises in the 1970s and early 1980s; since that time, much of the development has been put on hold.

The front runners in coal liquefaction development have been the USA, South Africa, Germany, Japan, and the UK. Other major coal producers have the potential to consider upgrading coal to supplement their own oil reserves and/or to reduce their oil imports, e.g. the PRC, India, Indonesia and Poland. Additional smaller coal producers, such as Turkey,

Greece, Romania, and Spain, could consider coal liquefaction if the coal market price and the cost of coal liquefaction were to be such that the process was economically viable.

In the UK, research and development into coal liquefaction has concentrated on the direct liquefaction process known as liquid solvent extraction, developed by British Coal, and no process has been demonstrated at a commercial scale (Robinson 1994). The model plant is designed to produce approximately 50 000 bbl per day of liquid transport fuels from 17 000 t per day of coal. In the first stage, bituminous coal is digested in an H-donating solvent in the absence of H<sub>2</sub>, at a low pressure. The resulting mixture is filtered to produce a low ash extract solution. In the second stage, the extract solution is catalytically hydrocracked in the presence of hydrogen to upgrade the products (Barraza et al. 1997).

In the USA, a variation on the pyrolysis technique is SGI International's Liquids-from-Coal (LFC<sup>®</sup>) process (Weber and Krottnerus 2000). Its development was influenced by the electricity generators in the USA changing from high-sulfur coals from the eastern USA to low-sulfur coals from the western USA. This change was in order to meet the sulfur dioxide emission standards set up by the US Clean Air Act. However, because the western coals are high in moisture content, and therefore expensive to transport, the LFC process was designed to overcome these problems. The coals used are from the Powder River Basin; they are high-moisture coals (25–32%) with relatively low heating values (7900–8800 Btu lb<sup>-1</sup>). The coal is dried to almost zero moisture content; it is then mildly pyrolysed and approximately 60% of the volatile matter and most of the organic sulfur is removed. The coal char is then cooled and has controlled amounts of moisture and O<sub>2</sub> added to produce a stable solid fuel. The volatile matter driven off during pyrolysis is partially condensed in a multi-step operation to produce crude hydrocarbon liquid. It has been shown that this process can produce approximately 0.5 t of solid fuel

and 0.5 bbl of crude hydrocarbon liquid from each short ton of raw coal feed. The liquid is low-sulfur heavy liquid hydrocarbon, which can be further processed to produce other chemical and industrial products.

In Serbia, studies into the conversion of low-rank brown coals into liquid products have been undertaken by using the direct catalytic hydrogenation process. Results indicated that the yield of particular liquid products varied markedly depending on temperature and residence time. A high degree of conversion (84%) was observed; this was confirmed by petrographic analysis, which showed that there was no unreacted coal in the solid residues. It was also noted that the petrographic composition of the residues depended on the reaction conditions (Aleksic et al. 1997).

Laboratory tests have been carried out on high-volatile bituminous coal from the Asturias Coal Basin in north-west Spain to determine the exploitable gas and hydrocarbon properties of the coal. The thermal kinetic model takes into account the hydrocarbon production potential of the coal by considering the reconstructed activation energies and the maceral composition (Piedal-Sanchez et al. 2005).

In Australia, the Linc Energy Chinchilla Gas to Liquids Project has made significant progress in demonstrating the combination of its UCG and Gas to Liquids (GTL) technologies. The GTL demonstration plant has shown that it can clean UCG synthesis gas to the levels required for FT synthesis, and in 2009 it produced high-quality hydrocarbon products. The company is confident that it can use this gas source to make synthetic liquid hydrocarbons.

In the past, the greatest drawback to coal liquefaction has been the high cost of production, particularly when oil, natural gas, and coal prices have been low. It has always been cheaper to obtain coal supplies from either indigenous sources or as imports than to invest in coal liquefaction plants. However, in times of oil price increases, the concerns over gas supplies, and the resistance to the development



of nuclear energy, then the greater availability of coal could provide the ideal economic scenario for future development of coal liquefaction technology.

#### 11.4.5 Coal-Sourced Oil and Gas Occurrences

The principal coal-bearing-sequence-sourced oil basins are listed in Table 11.9. The Gippsland Basin in south-east Australia and the Kutei Basin in East Kalimantan, Indonesia,

are the largest, most fully documented cases of coal-bearing-sequence-sourced oil provinces (Macgregor 1994). Some 80% of Australian oil is attributed to the Gippsland Basin, where oils are considered derived from non-marine Upper Cretaceous coals and carbonaceous shales. The Cooper Basin is a Permo-Triassic intercratonic basin situated in south central Australia. During the Permian the basin was filled by a series of lacustrine and fluvial deposits; the coals are characterised by high concentrations of inertinite and low levels of liptinite, a common

**Table 11.9** Examples of case studies of petroleum systems derived from terrigenous sediments i.e. coal or terrestrially-sourced organic matter.

Basin/province	Source rock		
	Age	H index <sup>a</sup>	Reference
<b>Australia</b>			
Gippsland Basin	Late Cretaceous–Palaeogene–Neogene	200–350	Moore et al. (1992)
Cooper/Eromanga Basin	Permian Jurassic	150–300 200–400	Vincent et al. (1985)
Bowen/Surat Basin	Permian	150–250	C.J. Boreham (unpublished results, 1994)
<b>Canada</b>			
Beaufort–Mackenzie Basin	Eocene–Palaeocene	130–250	Issler and Snowdon (1990)
<b>PRC</b>			
Turpan Basin	Jurassic	200–500	Zhao et al. (1997)
<b>Indonesia</b>			
Ardjuna Sub-basin, Java	Late Oligocene	250–400	Noble et al. (1991)
Kutei Basin, Kalimantan	Middle Miocene	200–350	Durand and Oudin (1979)
<b>New Zealand</b>			
Taranaki Basin	Late Cretaceous–Palaeogene–Neogene	230–360	Curry et al. (1994)
<b>Nigeria</b>			
Niger Delta	Late Cretaceous–Palaeogene–Neogene	<200	Bustin (1988)
<b>Norway</b>			
Haltenbanken area, North Sea	Jurassic	275	Forbes et al. (1991)

a) H indices are measured on immature samples.  
Source: Part based on Powell and Boreham (1994).

feature of Gondwana coals. However, in spite of this, the H/C ratio of the inertinite is 0.5 and the non-inertinitic material is 1.0, and the H index values for coals in the Cooper Basin range from 116 to 300 mg of hydrocarbons per gram of C, and considered to be derived from terrestrial organic matter co-deposited with algal-derived organic material (Curry et al. 1994; Powell and Boreham 1994).

In New Zealand, oils in the Taranaki Basin are considered derived from Cretaceous, Palaeogene, and Eocene terrestrial source rocks. Here, coals, shaly coals, and carbonaceous mudstones are all considered contributors.

In Indonesia, the Mahakam delta in the Kutei Basin, East Kalimantan, contains Miocene–Pliocene deltaic sediments with several oil, gas, and condensate fields. The Ardjuna Basin (north-west Java) oils are considered sourced from Oligocene deltaic sediments containing coals and interdistributary bay marine-influenced shales; the latter complicates the implication that the coals generated the oil.

Offshore Norway contains Late Jurassic marine shales and Early Jurassic coals and shales. The H index for the Jurassic coals is 100–350 mg of hydrocarbons per gram of C, and they have the potential for oil and condensate generation.

In north-west PRC, a thick sequence of Jurassic sediments is located in a number of tectonic basins along the flanks of the Tianshan–Qilan Mountains; one such basin is the Turpan Depression, which contains 4500 m of sediments. The sequence is characterised by rhythmic transgressions and regressions of lake and swamp facies over a large area. Deltaic sand bodies are associated with the lacustrine–swamp transition zone and act as hydrocarbon reservoirs. The organic material is primarily composed of liptinite and vitrinite, classified as Type I and Type II organic matter. The H index has a range of 200–500 mg of hydrocarbons per gram of C, and the depth of hydrocarbon maturation and expulsion

is given as 3000–4000 m (Zhao et al. 1997). These hydrocarbons from coal-bearing source rocks are of major importance in the PRC, and large reserves of oil and gas have been identified.

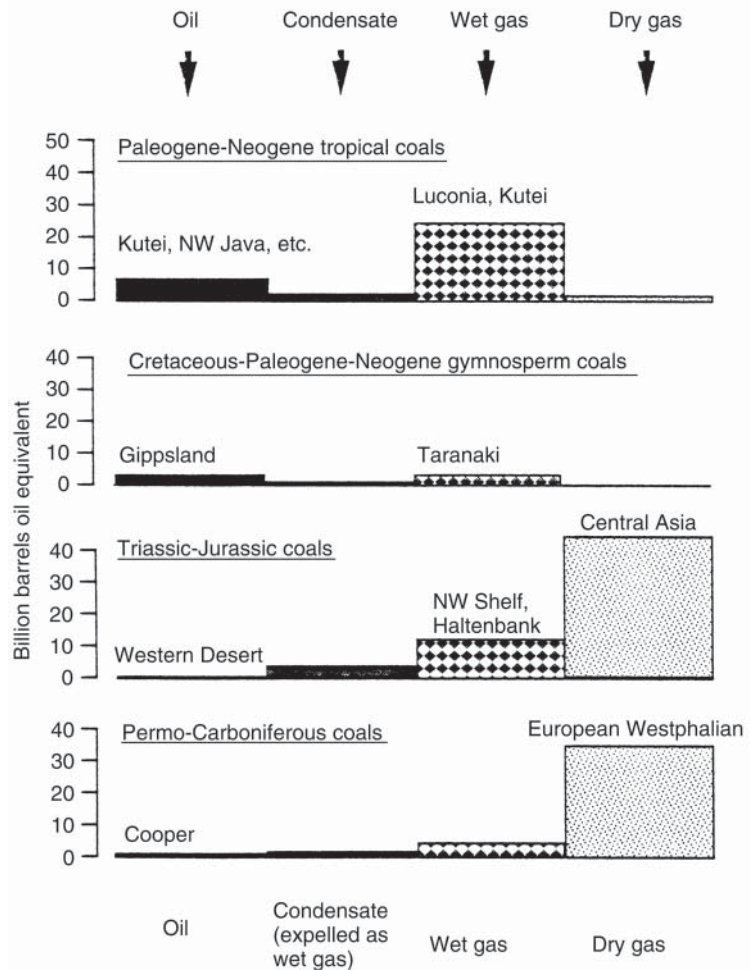
In South-East Asia, the proportion of oil reserves sourced from coal-bearing sequences is estimated at 10–30%. Other significant contributions of coal sourced are from Middle Jurassic coals in China and Egypt. Minor reserves of oil may be present sourced from Palaeogene–Neogene coals in Venezuela. Dry gas-prone coals become increasingly significant with increasing geological age. This is represented by the European Westphalian coals, which seem to have expelled only dry gas, although these coals are well within the oil-producing window. Figure 11.24 shows the hydrocarbon reserves in relation to geological age, and in particular the change from gas-prone coals in pre-Cretaceous times followed thereafter by oil-prone coals (Macgregor 1994).

It is significant that coal-bearing sources have not been considered for any of the world's 30 largest oil provinces, and that less than 1% of the world's known oil reserves are sourced from coal-bearing sequences (Macgregor 1994).

There are numerous occurrences of coal worldwide that are not tied to either oil or gas reserves due to the geological history of the deposits. For example, the Palaeogene–Neogene coals in the Philippines are similar to those in nearby Indonesia, yet no significant oil discoveries have been made other than those clearly derived from marine source rocks. Clearly, the oil potential of coals must vary across the region.

Coal-bearing source rock sequences are more confined in space and geological time than other oil-source rocks. Botanical controls and the environment will define the likelihood of oil availability within any coal-bearing sequence. Lacustrine margin coals appear to be more favourable than marine margin coals (Macgregor 1994).

**Figure 11.24** Hydrocarbon reserves tied to coal-bearing source sequences by age. Volumes of different hydrocarbon phases are plotted in billion barrels oil equivalent. Oil-prone coals plot to the left; gas-prone coals plot to the right. In pre-Cretaceous times, gas-prone coals predominate; later times see the appearance of significant oil-prone coals. *Source:* Part based on Powell and Boreham (1994).



The study and understanding of coal-bearing sequences as source rocks for oil and gas is still in its infancy, but it is clear that such studies will need to consider the sedimentological, palaeobotanical, and geochemical characteristics of the coals in each individual

sequence, as well as identifying suitable reservoir rocks. Those areas containing coal but hitherto considered not a good prospect may be reassessed in the future due to the increasing need for hydrocarbon reserves.

## 12

### Coal Use and the Environment

#### 12.1 Introduction

Since the last edition, the relationship of past and current coalmining operations with the environment has not changed. Thus, the following synopsis still holds validity.

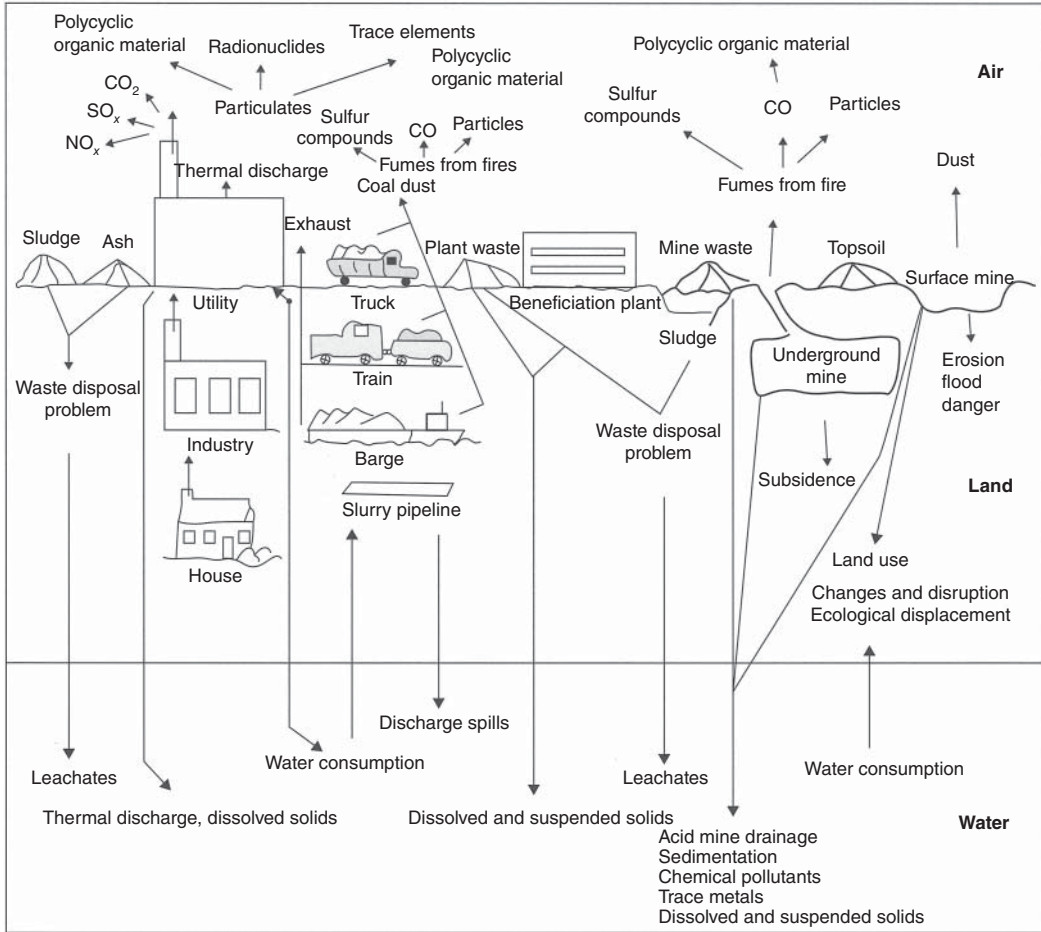
In the last forty-five years, public awareness has increased regarding local, national, and international environmental issues. This has resulted in the concentration of political attention by means of statutory regulations covering the majority of industries in every industrialised nation. Developing nations are being asked to conform to environmental standards conceived in the industrialised nations without which aid, financial support, and trading facilities will not be forthcoming. However, there is a concern that environmental standards can be imposed without due consideration of scientific and technical evidence together with the economic welfare of the community.

It is clear that no industry has attracted greater attention than the coal industry. The mining and use of coal remain emotive issues with environmentalists and their political supporters despite the tremendous improvements in mine rehabilitation and coal-fired power station emissions. Historically, the coal industry had left a legacy of both land and atmospheric pollution; indeed, the traditional image of dirty mines and industrial chimneys belching smoke is still too easily evoked in irrational discussions about the coal industry,

even though it bears no relation to the modern coal industry. The media are sympathetic to the environmentalists and have painted a picture that depicts the mining and use of coal as a threat to human health and therefore must be strongly regulated as a result. Such regulations serve to increase the cost of coal production and use; but in spite of this, coal remains a low-cost, abundant, and secure source of energy. In addition, coal is a democratic fuel, in that it is widespread globally and therefore less sensitive to political instability.

Because of the close attention given to it by environmentalists, the coal industry has had to address numerous environmental issues. This has included remedying previous environmental damage and to prevent future occurrences. Planned current and future environmental regulation as a result of international agreements will ensure that the coal industry will continue to improve both its working practice and its public profile.

However, it is undeniable that the mining and use of coal does have pronounced effects on the environment, and are principal causes of environmental concern. Figure 12.1 is a well-known summary of the effects of the use of coal on the environment. These have a direct influence upon the geological investigation and exploration for coal. Coals that are environmentally disadvantaged, e.g. high-sulfur coals, are unlikely to be the prime target of mining companies, who know their limitations in the coal sales market.



**Figure 12.1** Physical and chemical effects on the environment due to coalmining, transportation, and combustion.

Greenhouse gas (GHG) emissions from active and abandoned coalmines and coal-fired power plants are being reduced by capture and use of methane ( $\text{CH}_4$ ). Carbon capture and storage (CCS) in the form of carbon dioxide ( $\text{CO}_2$ ) produced primarily from power generation is being actively investigated with a number of projects operating worldwide.

## 12.2 Coalmining

The increasingly complex regulatory regimes imposed by governments has brought environmental planning to the forefront in the mine

planning and development process. This has resulted in changes in the methods of working, in types and utilisation of equipment, and in coal preparation techniques. In addition, changes in mine planning and operation have had to be developed, but at the same time both complying with the regulations and remaining cost effective and competitive.

Once the results of the exploration phase are known, and that these indicate that a viable mining operation is possible, it is normal practice to prepare a preliminary environmental report. This report, plus the final details of the coalmine planning, will serve as support documents in the preparation of the official



environmental impact and social assessment. All potential mine developments require an environmental impact and social assessment in order to obtain the necessary legal permits and concessions in order to mine coal. Although each mine can have different environmental effects, there are a number of factors that strongly influence the environment. Not only will the mine be assessed, but also the effect on the surrounding landscape, water courses, and native flora and fauna, as well as social effects on the local community.

In 2008, the US Environmental Protection Agency (EPA) using the Code of Federal Regulations (40 CFR 434), produced a report titled *Coal Mining Detailed Study* listing the coalmining effluent limitations guidelines and standards (US EPA 2008). This study outlines all types of acid mine drainage (AMD) and its remediation. In the UK, the Environment Agency is working with the Coal Authority to identify and remediate AMD; other countries in which coalmining has or is being operated now have regulatory authorities that monitor and implement AMD remediation.

## 12.2.1 Effects on Water Supply

### 12.2.1.1 Surface Water

During surface mining operations, it is possible that certain drainage diversions may be required, particularly for the larger scale operations. All surface water originating upstream of a mining site should be diverted around the excavation and spoil areas to avoid contamination of the water and to reduce other problems within the pit. Diversions should be hydraulically efficient and designed and constructed to control erosion and sediment load. Many countries have published regulations to establish design and performance standards for such diversions. It is necessary to determine the design of the channel section that will carry the diverted flow, and also to ensure that the flow velocities will remain below that which can be tolerated by the chosen channel design.

In the case of perennial streams, increased flow velocities and sediment loads in channel diversions may not be conducive to freshwater fauna and flora. The diversion ideally should include shallows and deeps and some meander pattern to suit the local ecological regime. Mining operations must ensure that no contamination occurs where water courses are used for public water supply. Local and regional governments impose substantial penalties on companies who in any way affect public water supplies through their mining operations.

### 12.2.1.2 Underground Water

Groundwater is used globally for public water supply, and the protection of aquifers that supply water to both urban and rural communities is essential. In some cases, potable groundwater has to be removed in order to operate open-pit mines; this water has to be captured in reservoirs or recharged back into the aquifer some distance away from the mine. Again, contamination is to be avoided. Some groundwater has a high concentration of total dissolved solids and is not usable for drinking purposes or for industrial usage. Such groundwater will need to be disposed of or treated; both scenarios are a cost to the mining operation. Recent advances in underground coal gasification have shown that contamination of aquifer waters is a major concern and safeguarded against.

## 12.2.2 Contamination of Mine Waters

One of the most serious effects of underground coalmining has been the escape of polluted water from both old and current mine workings. Before the days of environmental regulation, such acid and alkaline mine waters had been allowed to pollute waterways and surrounding land, rendering the area unusable and sterile. Old industrialised countries, such as in western and eastern Europe and the USA, exhibited industrial wastelands as a consequence of coalmining. Such practices are now

long gone, but the potential for underground mine waters to escape and enter the surface water regime is still a real one in some locations.

AMD is the principal cause of contaminated water arising from coalmining. It results from the exposure of sulfide minerals, particularly pyrite, to water and oxygen ( $O_2$ ) during and after mining or in piles of mine waste. Many underground mines have to pump to remove water from the mine workings. This can be a major problem in old mining areas where old mining districts are often connected underground. Water entering the workings from near-surface aquifers is usually of reasonable quality, but mine workings in deep seams are likely to encounter more saline waters. The minerals iron pyrite and marcasite (both iron sulfide) are commonly present in coals and coal-bearing sequences, and these are reactive to atmospheric  $O_2$ . The initial products of oxidation are ferrous and ferric sulfates, sulfuric acid, and hydrated ferric oxide. With the exception of ferric oxide, these products are soluble in water, and in turn react with clays and carbonate minerals to form aluminium, calcium, magnesium, and other sulfates. Ferruginous waters that flow in the presence of air in mine workings precipitate ferric oxide; this produces the extensive red/orange staining of walls and equipment that characterises many underground workings. Water flooding into abandoned mine areas containing large quantities of sulfide minerals will rapidly become contaminated. Problems arise when mining ceases and pumping is stopped; then, all the connected workings become flooded. The water level will rise using old shafts and workings as conduits, and can result in mine water discharge into the surface water regime. The initial breakout of water is the most acidic, and contains the largest quantities of iron and other dissolved metals; this is due to the fact that the greatest potential for the oxidation of pyrite is in a humid atmosphere where there is free  $O_2$ , as in the case of old workings. However, this is greatly reduced in saturated

or flooded conditions when the presence of free  $O_2$  is removed. Consequently, once waters flowing through the workings have flushed out all the oxidised material, little additional oxidation occurs and the contamination of the mine water decreases with time, provided the water levels in the workings do not fluctuate and allow oxidation to recommence.

In the eastern USA, AMD associated with coalmining has caused severe problems. Anthracite mines in eastern Pennsylvania were abandoned and allowed to flood, and in the 1960s the initial water discharge had a pH of 3.3–5.6. As water continued to flush through the mine workings, the pH improved to 5.8–6.2 by the late 1970s. The cessation of mining and the circulation of groundwater has led to the improvement in the quality of mine water discharge in the region. In the active mines, improvement of water quality has been achieved by chemical neutralisation of the mine water before discharge. The estimated cost of such chemical treatment of mine waters in the USA has been given as >\$1 million per day (Clarke 1995).

Table 12.1 shows the relationship between depth from the surface and the concentration of selected dissolved compounds found in underground mines. Elements such as sodium, calcium, and chlorine increase in concentration with increasing depth, whereas sulfate and hydrogen carbonate compounds decrease with depth. Waters from shallow workings contain sulfates, chlorides, bicarbonates, calcium, magnesium, and sodium salts. Waters from slightly deeper workings become more heavily mineralised with calcium and magnesium salts, and at great depths concentrations of barium, strontium, and ammonium chlorides are characteristic. Saline waters from deep coalmines contain high amounts of chlorides, e.g.  $61\,240\text{ mg l}^{-1}\text{ Cl}^-$  in waters from deep coal workings in Nottinghamshire, UK (Downing et al. 1970), together with high amounts of ammoniacal nitrogen. Other contaminated mine waters produced within deeper workings may also contain diffused  $CH_4$  gas. Such saline

**Table 12.1** Changes in groundwater quality with depth.

Depth from surface (m)	Concentrations of dissolved compounds (mg l <sup>-1</sup> )									
	Na <sup>+</sup>	Ca <sup>2+</sup>	Mg <sup>2+</sup>	Ba <sup>2+</sup>	Sr <sup>2+</sup>	NH <sub>4</sub> <sup>+</sup>	Mn <sup>2+</sup>	Cl <sup>-</sup>	SO <sub>4</sub> <sup>2+</sup>	HCO <sub>3</sub> <sup>-</sup>
30	40	60	40	< 2	< 1	< 0.1	< 0.1	50	200	200
300	10 000	800	260	60	25	12	0.3	18 000	< 5	200
900	41 500	11 700	2000	550	400	70	3.0	90 000	< 5	80

waters can be harmful to crops, and suffer corrosion to metallic machinery, and have proved to be a problem in a number of areas. In the Upper Silesian Basin, Poland, salinity increases with depth, and chemical analyses have shown salinity levels of more than 250 000 mg l<sup>-1</sup> (Clarke 1995). In addition, Polish saline waters contain natural radioactive isotopes, mainly radium-226 (<sup>226</sup>Ra) from the uranium series and radium-228 (<sup>228</sup>Ra) from the thorium series. Up to 40% of the total amount of radium remains in the ground, but up to 225 MBq of <sup>226</sup>Ra and 400 MBq of <sup>228</sup>Ra are released daily into rivers along with other mine effluents. To counteract this, technical measures, such as induced precipitation in gob areas, have been undertaken in several mines, and the results have shown that the total amount of radium released to the surface waters has diminished by 60% in the last 10 years (Chalupnik et al. 2001).

Table 12.2 shows some selected analyses of saline and acidic waters from deep coal workings in the UK. Of note is the acidic iron-rich nature of the initial mine drainage water from Bentinck Mine compared with the pumped water after equilibrium is reached from Moorgreen and Pye Hill mines (after Banks et al. 1997).

In open-pit mines, the exposure of rock (with its content of sulfide minerals) to the atmosphere and the hydrological cycle can produce acidic mine waters. Piles of removed overburden and interburden, whether as infill or as spoil heaps, together with all surface mine waste and spoil heaps from underground workings, can produce contaminated water.

In new mine development, a detailed hydrogeological investigation is essential to understand the movement of groundwater around and within any proposed mine workings. The management of saline waters from coalmines operating in arid regions can pose difficulties. In the Hunter Valley, New South Wales, Australia, coal seams carry saline groundwater that drains naturally into the Hunter River. During normal rainfall years, the mine waters are diluted by better quality surface flows, but during extended drought conditions the saline groundwater makes up a larger proportion of the river recharge water. During such periods, it is essential that carefully controlled discharge of saline mine waters is maintained. In the Thar Desert of south-east Pakistan, proposed open-pit mining operations will have to pump out large amounts of groundwater from three separated aquifers in order to commence mining. In such an arid area, the water will be needed for drinking water to replace local supplies disrupted by pumping, to supply the mine operation, and for industrial use such as cooling water for the proposed new power plants in the area. Water that is not treated or utilised may be recharged back into the groundwater system.

In underground mines, this is never as straightforward as in open-pit mines. The transmissivity of water at changing depths and structural intensities makes it difficult to anticipate groundwater behaviour. Nevertheless, such studies will assist in the planning of groundwater removal from the mine (see Section 9.5) and reduce the potential for groundwater contamination.

**Table 12.2** Selected analyses (in mg l<sup>-1</sup>) of deep waters from coal measures strata in the UK, illustrating compositions of saline brines. Note the acidic iron-rich nature of the first drainage (i.e. non-equilibrium) water from Bentinck Colliery when compared with the pumped (equilibrium) saline water from the nearby Moorgreen and Pye Hill collieries.

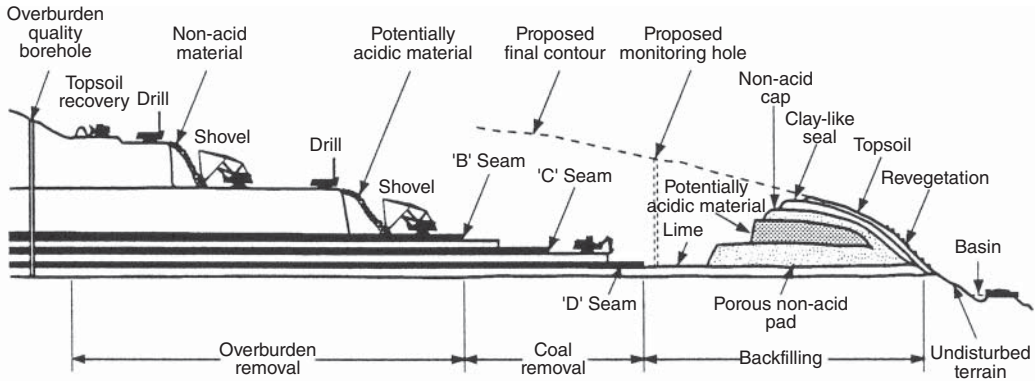
	<b>Eakring 8 Crawshaw sandstone</b>	<b>Glentworth 5 Lower coal measures sandstones</b>	<b>Plungar 4 Crawshaw sandstone</b>	<b>Moorgreen Piper Colliery Pumped water</b>	<b>Pye Hill No. 2 Colliery Pumped water</b>	<b>Bentinck Colliery Initial drainage water</b>
Na <sup>+</sup>	8 079	7 005	7 900	—	—	—
K <sup>+</sup>	96	9	31	—	—	—
Ca <sup>2+</sup>	792	1 552	822	—	—	—
Mg <sup>2+</sup>	218	192	556	—	—	—
Cl <sup>-</sup>	14 555	11 786	14 910	3 600–10 800	1 100–3 900	31 400
SO <sub>4</sub> <sup>2-</sup>	0	2 718	342	—	—	—
HCO <sub>3</sub>	73	549	220	—	—	—
Total dissolved solids	23 776	23 532	24 669	—	—	—
pH	7.1	7.4	7.8	6.9–7.9	7.3–8.0	5.7
Fe (total)	—	—	—	<0.1–7	1–9	150

Source: Adapted from Banks et al. (1997). Reproduced by permission of the Geological Society.

The problem of contaminated mine drainage is best dealt with by preventing polluted drainage from occurring, or by collecting and treating it before it is discharged. In most cases, the reprocessing of spoil heaps is too expensive, and the removal of underground sources of contamination is not feasible other than to flood completely the area from which the contamination is generated. If the source cannot be removed, then the alternative is to keep water away from spoil heaps and within the mine. A number of barrier methods are used, particularly in relation to spoil heaps. Compaction and revegetation are two ways of inhibiting water passage through spoil heaps, but more effective isolation of spoil material from percolating waters is to use low-permeability barriers, such as clay or plastic membranes (Clarke 1995). In the USA, a common method used to isolate pyritic spoil from groundwater flow, percolating surface waters, and oxygenation is the *high and dry placement* method. Figure 12.2 shows the segregation of acid-prone material to reduce exposure to water and O<sub>2</sub>. The acid-prone material overlies

porous material, which allows groundwater to pass through, and is itself compacted to restrict surface water infiltration. This reduces the contamination but may not achieve the levels demanded by modern legislation. Also, the method is mainly applicable to spoil in open-pit operations. Spoil from underground mining is much more difficult to assess and it is difficult to segregate. Clay and bentonite-rich liners have been used as clay caps to prevent water entering the spoil, but they have been prone to cracking in dry conditions, and they may also be attacked and breached by certain mineral-rich mine waters. Plastic membranes can be used, preferably in ongoing mining operations where they can be integrated into the reclamation programme; however, the high cost of their use is likely to restrict them to the most acid-producing sites. Effective prevention of rainwater inflow is achieved by using asphalt or concrete caps to cover spoil materials.

The treatment of AMD has included the use of bacteria, principally *Thiobacillus*



**Figure 12.2** Selective handling and placement of mine spoil to prevent the formation of AMD at a mine in West Virginia, USA. *Source:* From Clarke (1995). Reproduced with permission of IEA.

*ferrooxidans*, which catalyses the oxidation of pyrite. *T. ferrooxidans* is also sensitive to heavy metals, but due to the low levels present in mine waters the use of surfactants has proved more effective in AMD treatment. Commercially produced slow-release surfactants are now available in the form of spray or pellets and have an effective life of two to seven years (Clarke 1995), and they have been applied to sites in Pennsylvania and Kentucky in the USA. The cost of bactericide is offset by savings in reductions in the amount of topsoil required or in the use of other remedial methods.

There has been recent research into the construction of wetlands as an effective method for the treatment of drainage from abandoned mines. The treatment of mine drainage requires the use of aerobic wetland processes; these are designed to encourage the oxidation process and are consequently of shallow depth (0.3 m). This will remove iron in the wetland by the precipitation of ferric hydroxide, which in turn lowers the pH of the water, which will reduce the oxidation rate. This is then compensated by growing plants such as reeds that pass  $O_2$  through their root systems, causing aeration of the substrate. Wetland treatment studies have been carried out in Kentucky, USA, and in South Wales, UK (Robinson 1998).

### 12.2.3 Other Water Pollution

Pollution of surface waters can occur from the use of drilling muds and additives. In both greenfield and developed areas, drilling programmes must avoid the pollution of streams and rivers by drilling fluids being allowed to flow into them. Discoloration of the water, though not necessarily toxic, is not desired by urban and rural peoples alike. The building of a sealed circulation pit and the monitoring of flow rates into and out of a borehole should avoid this situation. Similarly, in wells used for abstraction of drinking water, as well as surrounding streams, the leakage of diesel, kerosene, and other industrial fluids must be avoided.

### 12.2.4 Run-off, Erosion, and Sedimentation

Run-off results from precipitation and is the major cause of erosion in mining areas, particularly in regions of concentrated heavy rainfall, as is the case in tropical countries. Attempts to combat soil erosion are aimed at controlling run-off, reducing the erodibility of the soil itself, and removing any sediment from the run-off that does occur.

Deforestation and the stripping of vegetation cover need to be kept to a minimum, and exposure of the required area of land



should be for as short a term as possible. This requires effective mine planning and scheduling the sequential stages of vegetation removal, overburden stripping, mining, and reclamation. Seasonal climatic variations may play an important role in this scheduling. Such planning should include the siting of haul roads and any banking, as these are the sites of much of the run-off and erosion. Diversion structures, such as terraces and ditches, can be sited to intercept run-off on long steep slopes, together with keeping topsoil loose to aid infiltration and by using new vegetation types to stabilise slopes.

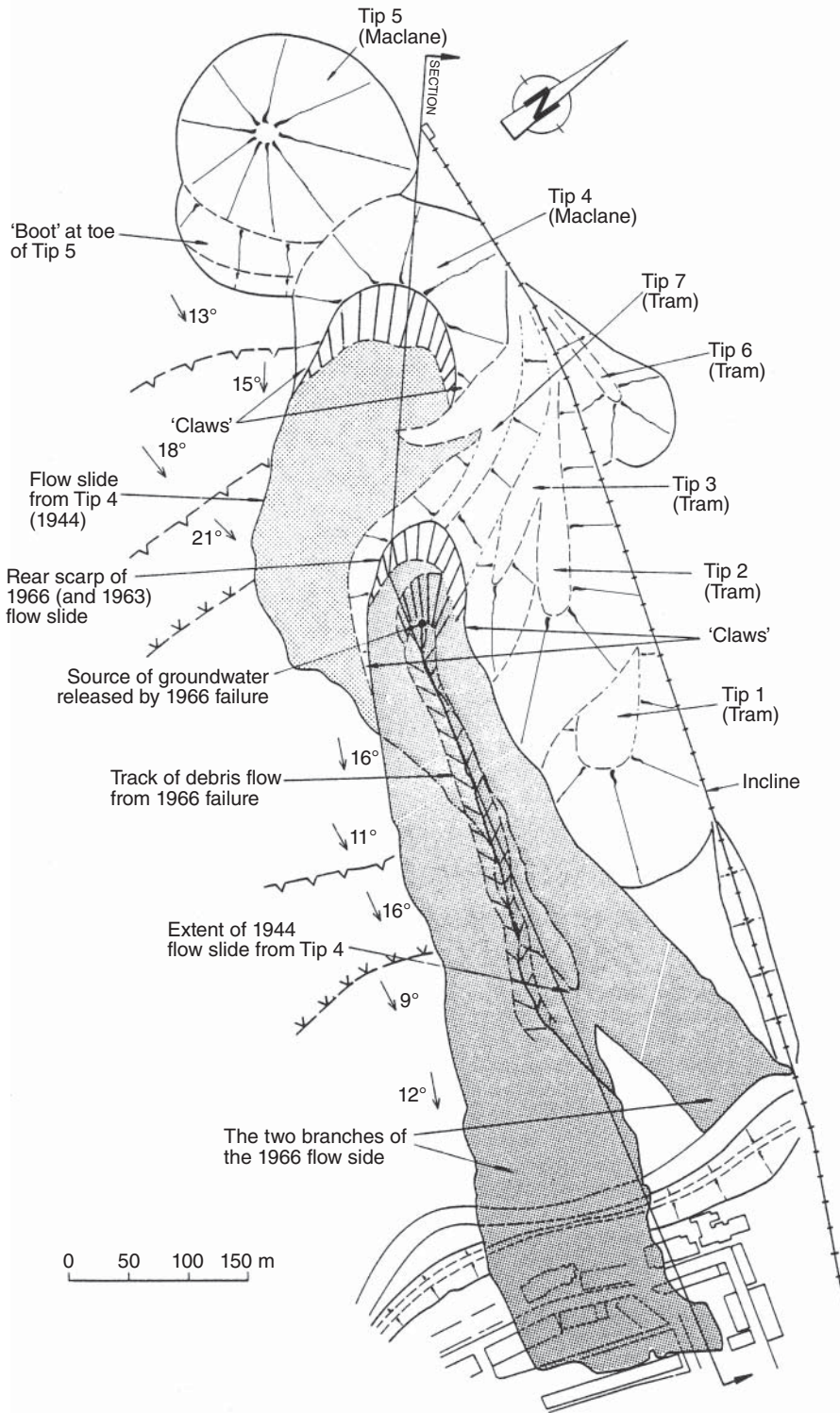
Concave slopes are least affected by erosion, yield the least sediment, and change shape slower than other profiles. Convex slopes erode most rapidly, yield the most sediment, and change shape quickly. Uniform and complex slopes are affected to an intermediate degree, but they can still be severely eroded in a single storm. It is recommended, therefore, that slopes should be produced with as low a gradient as possible and be concave where possible.

The loss of soil and land due to erosion can result in the degradation of streams and lakes as a result of increased sediment loads. As most run-off and erosion prevention measures are not 100% successful, all run-off originating within the mined area should be routed through a sedimentation pond, the primary purpose of which is to trap sediment movement from the mined area. Suspended sediment concentrations in waters draining from surface mining areas can be very high, with concentrations of 10 000 up to 100 000 mg l<sup>-1</sup>. Such a sedimentation pond should be of sufficient size to store the sediment load without having the need for frequent removal of settled material, and to be of a size so that inflowing water has a sufficiently long detention period and low velocity to allow suspended sediment to settle out.

### 12.2.5 Spoil Dumping

Environmentally, the dumping of spoil material is considered one of the least desirable surface manifestations of coalmining. Historically, the dumping of spoil was not regulated, and old underground mining areas were easily identified by the characteristic skyline of conical and elongate spoil tips with their attendant dumping systems, usually by tram railway or overhead ropeways. Modern underground mines still have the problem of where to place rock waste, but this has been reduced by the widespread introduction of longwall mining and by repacking waste material in abandoned districts in room-and-pillar mines.

Apart from the undesirable visual effects, spoil heaps can be a problematic mining legacy. Spoil heaps or tips can be extremely large and be a product of a number of different mining episodes. The materials in tips can vary enormously; apart from non-coal rock waste, machinery, wood, ropes, boiler ash, and general rubbish can all end up in spoil tips. For example, in the UK, at Cilfynydd in South Wales, a large tip fed by an aerial ropeway received up to 500 tons per day, and the tip was used for over 50 years (Bentley et al. 1998). In areas of mountainous and dissected topography, tips have been constructed on steep slopes, and in some cases across spring lines. Load pressures in these circumstances combined with high annual precipitation rates have led to tip failures. Failure of such tips was not uncommon in old mining areas, but none captured the headlines more than the failure of the Merthyr Vale Colliery spoil tip No. 7 at Aberfan in October 1966 (Figure 12.3; Siddle et al. 1996). Failure had been recorded in other tips at Aberfan in 1944 and 1963, but tip No. 7 is remembered for the high loss of life (144 people) caused when the tip failed. Rotational slides of spoil disintegrated to flow slides that ran downhill for 600 m. These flows



**Figure 12.3** Plan of Merthyr Vale (Aberfan) colliery tip flow slides of 1944, 1963, and 1966. *Source:* From Siddle et al. (1996).



**Figure 12.4** Opencast mine in Bosnia-Herzegovina showing large-scale overburden removal. Note small-scale failures of the highwall caused by surface water action during periods of mining inactivity. *Source:* Photograph by courtesy of Dargo Associates Ltd.

released groundwater from a fault zone in the underlying sandstone that then caused a secondary debris flow (Figure 12.3). Since the Aberfan disaster, improved tipping practices and rigorous inspections have ensured that no rapid tip failures have occurred in the UK since 1967.

It is a different picture in open-pit operations. The final restoration of the site has to be included in the overall mine planning; and in the interim period, when the mine is in full production, provision has to be made for dumping of topsoil and then overburden and interburden waste material. In modern-day mining, the amount of material removed is enormous. For example, in the USA alone, more than  $1.5 \times 10^9 \text{ m}^3$  per year is moved by dragline; add to this figure the amount moved by shovels, and it is clear that very large amounts of spoil are moved. Even in small open-pit operations, such as in central Europe,

over 7.4 million bcm of overburden will be moved to mine 1.8 Mt of coal. Such overburden dumps can cover large areas, as seen in Figure 12.4. Certain methods of open-pit mining ease the problem by backfilling the mining void as the mine progresses (see Section 10.3) and then restoring the topsoil once mining is completed. In many open pits, the overburden is tipped onto areas that will not be mined as near to the working operation as possible; long waste haulage with dump trucks is expensive and time consuming. Slope stability studies determine the size and shape of the spoil dump, and the natural drainage is piped or diverted. Many mines leave the spoil dumps as a permanent feature, but modern-day restoration demands that the dumps are contoured, covered with top soil, have adequate drainage, and are revegetated with selected local plant species. In some cases, areas of tip waste material have been contoured and sculpted; e.g. in



the UK, the sculpture Northumberlandia, or Lady of the North, is now a tourist attraction. Some open-pit mine areas now serve as wildlife conservation areas. In areas where flooding from rivers is a reality, a bund of overburden may be placed between the water course and the mine working area (Figure 12.5). Old dumps or tips where no restoration was carried out are now being retreated or removed, and some are being reworked for their discard coal content. This not only removes the dump altogether but also can yield coal otherwise lost. Tip reclamation operations are well established in the USA (e.g. in southern Illinois, Indiana, and western Kentucky) and in UK (e.g. in South Wales, where old tips [pre-World War II] are recycled for coal). The percentage of coal is likely to be higher in these old tips as coal preparation (if any) was less efficient and mining methods less precise.

Reject spoil material from coal preparation plants is disposed of by dumping or sent to landfill. Fine waste is usually disposed of as slurry in specially constructed lagoons or ponds, or else is dewatered and dumped with coarser spoil. The fine-grained slurry or tailings are dumped in the lagoon or settling pond where the solids settle out, the clean water is then discharged or reused in the mine or coal preparation plant. Abandoned tailings ponds are potential sources of water pollution, as is the coarser coal waste. Rainfall and groundwater passing through spoil dumps that contain pyrite and other potential sources of contamination can generate AMD. This can be controlled by providing adequate drainage channels within and underneath spoil dumps. If the water is contaminated, it can be treated before discharge. One problem with old mine waste is that imprecise



**Figure 12.5** Opencast mine in Maharashtra State, India, showing protective bund of overburden to prevent flooding from nearby river. *Source:* Photograph by courtesy of Dargo Associates Ltd.

records on the location of old settling ponds can prevent modern treatment of the waste to avoid future water pollution.

Reclamation of open-pit areas with back-filled spoil covered with topsoil must be accompanied by the regular testing of soil and water samples to ensure that the soil profile remains uncontaminated and suitable for revegetation. Some restored areas have failed to support revegetation because the soil has become acid rich from waters percolating through the backfill material. Modern mine reclamation has to assess the nature of the backfill, its potential for contamination, and the remedial measures necessary to ensure successful land reclamation.

Many mines now leave the area in better soil condition than before mining commenced, but it is a cost to the mining operation that has to be assessed prior to mining.

### 12.2.6 Spontaneous Combustion

The propensity for spontaneous combustion is related to the rank, moisture content, and size of the coal. In addition, mining and ventilation practices and geological conditions can also be contributory factors.

O<sub>2</sub> is adsorbed onto the surface of the coal in an exothermic reaction, which is the start of oxidation (see Section 4.3.4). If the amount of free O<sub>2</sub> is small, then the reaction is slow with little rise in temperature; but where the quantity of O<sub>2</sub> passing over the coal is much larger, any heat produced will be dissipated, the temperature will not rise again, and oxidation will proceed at a low level. Between these two conditions exists the situation where the quantity of oxygen is sufficient to promote oxidation but not sufficient to dissipate the heat. This increases the rate of oxidation and eventually ignition will occur. Therefore, the coal's capacity to adsorb O<sub>2</sub> determines its propensity to spontaneously combust. Another contributory factor to spontaneous combustion is the presence of pyrite in the coal. Pyrite can easily be oxidised; on doing so, it swells and

exposes more coal surface to O<sub>2</sub> and therefore assists in the oxidation process. Solid coal presents less risk of spontaneous combustion; but when it is shattered by mining or broken by structural dislocation, the surface area of the coal is greatly increased.

Lower rank coals with high moisture contents are most susceptible to spontaneous combustion, both in opencast and underground mines. Stockpiles or cargoes of such coal are also vulnerable to spontaneous combustion.

In underground mines, areas of coal such as in sidewalls or pillars are subject to oxidation. Regular monitoring of the mine atmosphere and exposed coal areas is essential to avoid the hazard of underground coal fires. Some mining areas have a history of coal fires, the most notorious being India, and in particular the Jharia Coalfield, which contains the largest complex of surface and underground coal fires in the world. The coalfield contains 40 coal seams, of which 70% are 3.5 m or greater in thickness and are high-volatile bituminous coal of medium coking quality. Mining is by a mix of underground and opencast operations, and the first mine fire was reported in 1916 and has spread over the ensuing 100 years. At the present time, numerous fires are still burning from 15 to 140 m below the ground surface. They have produced uncontrolled subsidence, devastated land, ill health, and death to the indigenous population. Throughout the 1990s, up to 40 Mt of coal had been lost through fires and a further 1500 Mt isolated from further development (World Coal 1997). The big problem is how to eliminate the fires. This requires elimination of one or more of the components needed to sustain them; namely, fuel, O<sub>2</sub>, and heat. Methods such as excavation (removes fuel), smothering (removes O<sub>2</sub>), and quenching (lowers fuel temperature) are currently being applied. Of these, only excavation is the certain means to extinguish the fire. However, in the case of Jharia, this would require  $500 \times 10^6$  m<sup>3</sup> of material from 60 open-pit excavations; apart from the prohibitive cost, this would take a very



long time. This option, therefore, is not viable except on a case-by-case basis (Michalski and Gray 1997). Though Jharia is an extreme case, underground fires occur elsewhere, e.g. in underground brown-coal mines in Turkey. All are hazardous, producing danger to life and loss of revenue.

In open-pit mines, the problem is less easy to isolate, and fires start either at the coal face, coal stockpiles, or in out-of-pit dumps where enough combustible material is present.

Coals can also catch fire at outcrop, either by spontaneous combustion or as a result of events such as forest fires. Figure 12.6a shows such an example of a thick coal seam burning in East Kalimantan, Indonesia, and Figure 12.6b shows coal burning in the high-wall of an Indian opencast mine. In the case of stockpiles, remedies are available: first, avoid having large ranges of coal particle size, which allows a higher rate of oxidation; second, the stockpile can be compacted to exclude oxygen

from circulating, and therefore decrease the oxidation rate, and the stockpile can be sealed with a layer of clay or bitumen. It is important to prevent the practice of spraying water on stockpiles to ostensibly cool them down; this has the opposite effect of causing increases in temperature as the water reacts with the coal surfaces and can even start fires. The stockpile temperature should be regularly tested to ensure no ‘hot-spots’ develop; this is also true for coals prone to spontaneous combustion during transport, especially on ocean-going vessels.

### 12.2.7 Dust Suppression

In underground mines, dust control is maintained by good ventilation systems and water sprays at the coalface where coal is being cut. The quantity of air and the distance from the end of the airline to the coal face are critical. Because of leakage through and



**Figure 12.6** (a) Coal burning as a result of ignition by forest fires in Kalimantan, Indonesia. *Source:* Photograph by LPT. (b) Spontaneous combustion of coal in the highwall of an Indian mine. *Source:* Photograph by courtesy of Dargo Associates Ltd.



**Figure 12.6** (Continued)

around working districts, more air must be forced into a mine than would otherwise be used for ventilation. Such leakage adds to the mines' operating costs, so efficient sealing of mine areas is necessary. Water spray systems using non-clogging water nozzles are fitted as standard on power shearers and continuous mining equipment. Coal travelling on conveyors is damped down to further reduce dust in the mine atmosphere. Cutting rock produces more dust than cutting coal, so in development areas where rock and coal have to be removed, face-mask respirators may be worn.

In open-pit operations, the use of wider haul roads and larger haul trucks has put greater emphasis on road conditions and dust control. If not controlled, excessive dust can raise equipment operation and maintenance costs and shorten the life of vehicle components and systems. Heavy dust is also a sign that the haul road surface is degrading. Data collection on air quality are taken by using dust deposition gauges and are operated throughout the life

of the mine. Studies of dust levels can provide valuable information in the future planning of the mine. Water may appear the cheapest form of dust control but can contribute to road surface deterioration. Dust can be controlled by applications of diluted suppressant that can enhance water penetration into the road surface, lengthening the time it takes for the water to evaporate. A major factor is the nature of the climate; for example, in areas with a definite rainfall season, such as the monsoon season in India, dust is a problem for the dry period; in other areas, like Indonesia, a tropical rainfall climate of hot sun and frequent heavy showers produces a dust–mud cycle, both of which can inhibit trafficability within the mine.

Dust suppression is also necessary in coal preparation plants, particularly where the coal is only crushed and screened, and in coal loading facilities. Automatic coal loading is easier to control for dust emissions, but wagon loading in rail yards can be a problem in dry, windy conditions. The usual method is to dampen down the coal, but this can create a problem if the coal is prone to spontaneous combustion.

Environmental legislation, particularly in areas with indigenous populations, means that mining companies are under pressure to minimise dust pollution, and most mines use dust suppressants to combat this.

### 12.2.8 Subsidence

Subsidence is a consequence of underground mining. It may be localised or extend over large areas, and it may be immediate or delayed for many years. When a cavity is created underground, the stress field in the surrounding strata is disturbed. These stress changes produce deformation and displacement of the strata, the scale of which is dependent upon the magnitude of the stress and the cavity dimensions. Over a period of time, mine roof and sidewall supports deteriorate and can result in instability. Roof collapse induces strata above to move into the void; these

movements move up to the ground surface and appear as a depression or a series of depressions. In mines, when the void left by coal extraction is of larger size, the collapsed strata fall into the excavation; this process continues until a height of three to six times the mined seam thickness is reached. When the cavity is filled with broken rock, the debris offers some support to the adjacent strata. As these strata settle or sag, bed separation may occur because of the tendency for lower strata to subside more than the higher beds (Figure 12.7). Overall, as the strata settle or subside, they sag, rather than break, and produce a dish- or trough-shaped depression on the ground surface (Figure 12.8) known as trough or sag subsidence. The *critical width* of the workings is the minimum width that needs to be mined before the maximum possible subsidence is observed at the centre of the trough. If the mined area is less than critical, it is termed *subcritical*, and the amount of subsidence that occurs will be less than the maximum. If a *supercritical* width is extracted, the central portion of the trough will attain maximum subsidence and a flat-bottomed depression will be produced. Such flat-bottomed depressions are a feature in many of the coalfields in the People's Republic of China (PRC), where in some circumstances they have been flooded and used for fish farming (Figure 12.9). Trough subsidence may occur due to mining at any depth; the overall movements of the ground around the mine cavity are shown in Figure 12.10. The direction of motion can be seen not only to be vertically downward but also horizontal and, in some locations, upward (Singh 1992).

Old shallow mines have produced small surface subsidence features; in modern longwall mining, however, the collapse of pillars in room-and-pillar mines and the large network of underground roadways can produce widespread subsidence, and all countries with a long history of underground mining have experienced subsidence problems. Subsidence has caused the collapse and distortion

of buildings, roads, and railways and is an ongoing legacy in areas where coalmining has long ceased.

Subsidence can be reduced substantially if large pillars of coal are left in place, or if rock waste from other mining districts is packed into the mined void. Where full coal extraction takes place, as is the case in longwall mining, the maximum subsidence is usually around 80% of the thickness of coal removed (Ward 1984).

In both underground and opencast mines, subsidence can be a result of the lowering of the water table as part of a mine dewatering scheme. This has a small effect on agricultural land but can seriously affect buildings and conduits close to the mine. Usually, such subsidence is caused by the compaction of sands and gravels in superficial deposits, such as river or glacial deposits overlying the coal-bearing strata. In central Europe, the dewatering of large areas in order to mine shallow brown coal deposits has produced such subsidence. The land surface will not return to its original level once mining has ceased even though the water table will return to a higher level.

As well as subsidence, a legacy of old mining areas is the numerous old shafts used to access the coal. Many of these are not shown on modern plans, and their locations are largely now unknown. For example, in the UK, this has become a problem in the old coalfield areas, where a number of shafts have been built over and have subsequently opened up as the capping material has deteriorated over time, exacerbated by the weight of material above and/or by the action of groundwater erosion. Figure 12.11 illustrates such a shaft that has opened up in a housing complex in Scotland, UK.

Mining subsidence and deterioration in the capping of old mine shafts can cause the problem of gas seepage from old workings, particularly  $\text{CH}_4$ , which is less dense than air. As shown in Figure 12.12, gas from old mine workings is able to migrate upwards via fractures caused by rock collapse into voids

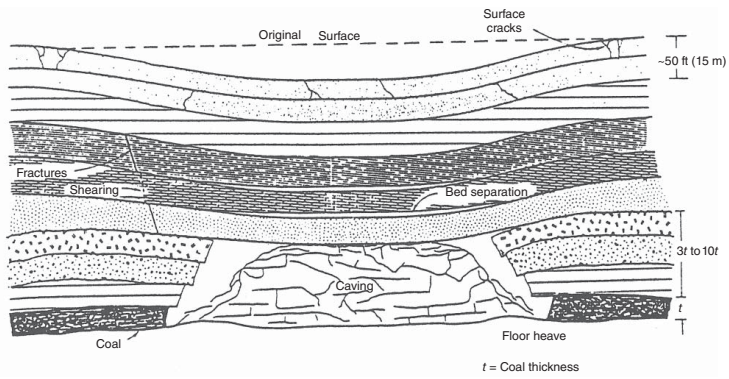
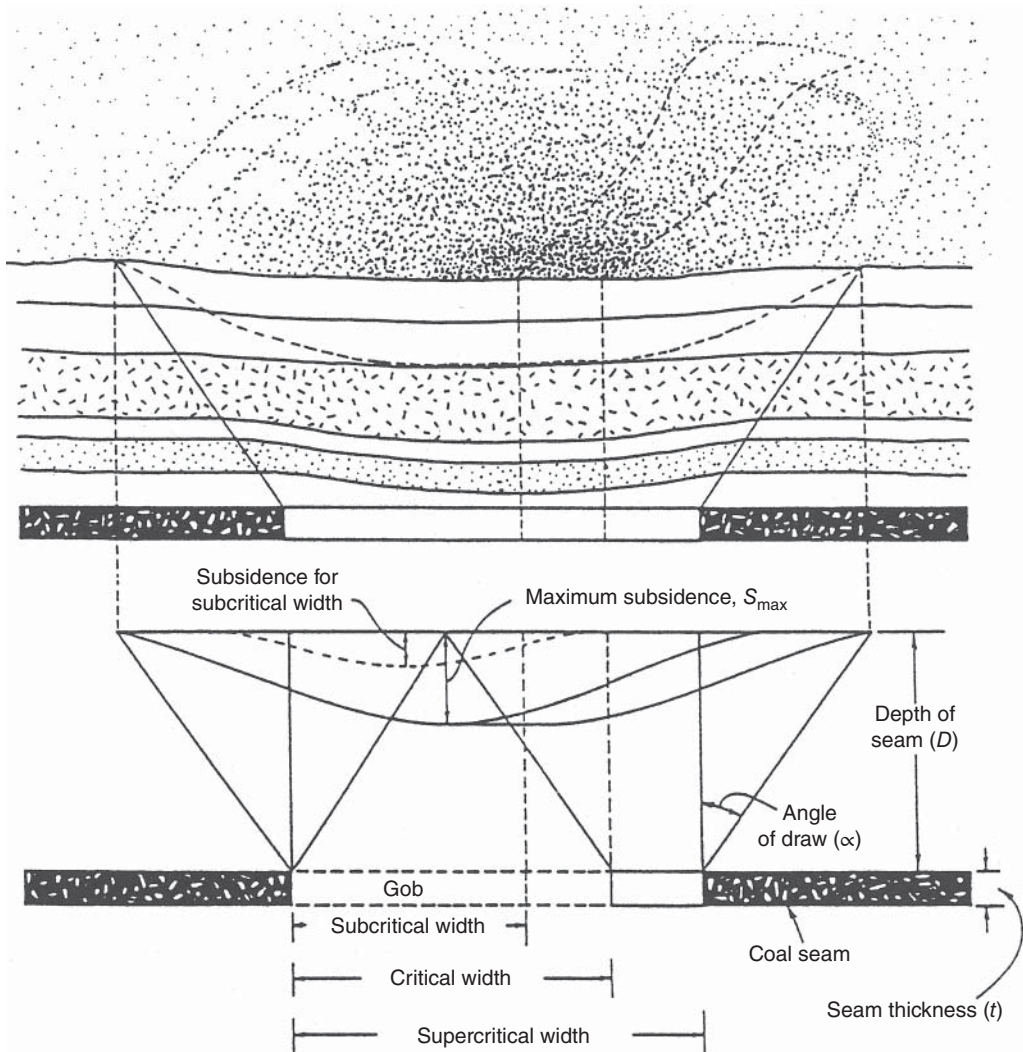


Figure 12.7 Strata disturbance and subsidence caused by mining. Source: From Hartman (1992). Reproduced with permission of SME.





**Figure 12.8** Influence of extraction width on subsidence. *Source:* From Hartman (1992). Reproduced with permission of SME.

left between coal pillars. Gas will also migrate upwards in old shafts, and if these shafts are inadequately sealed the gas will reach the surface and escape. In Figure 12.12, buildings at the surface will be vulnerable to gas invasion. House A is protected by an underlying layer of clay that prevents gas from reaching the surface, whereas house B has no protection. Fatalities have occurred from gas seepage from old mine workings; as a result, careful

investigations should now be made before buildings are erected in old mining areas.

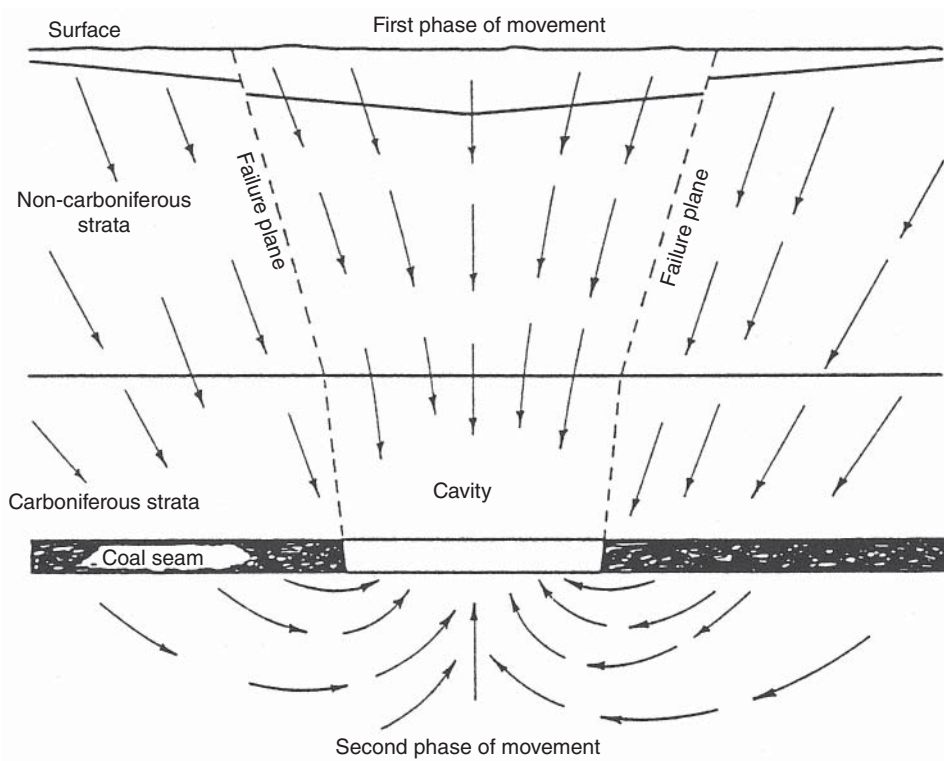
## 12.3 Coal Use

Coal is a versatile fuel, and has long been used for heating, industrial processes, and in power generation. Internationally traded coals have predominantly been for use in coke-making





**Figure 12.9** Subsided land overlying underground workings, the PRC, now flooded and used for fish farming. *Source:* Photograph by courtesy of Dargo Associates Ltd.

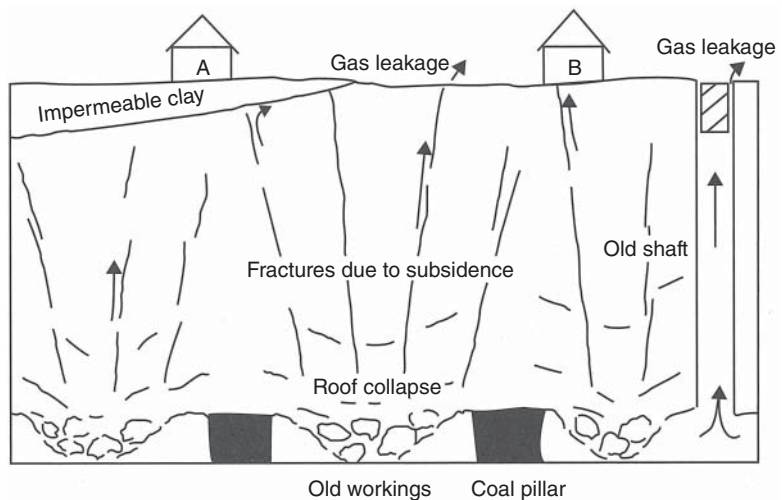


**Figure 12.10** Schematic representation of ground movements due to subsidence. *Source:* From Hartman (1992). Reproduced with permission of SME.

**Figure 12.11** Collapse of capping material over an old concealed shaft in a modern urban area, UK. *Source:* Photograph by courtesy of IPR/25-36c British Geological Survey. © NERC. All rights reserved.



**Figure 12.12** Schematic view of the upward migration of mine gas by means of fractures due to subsidence and from an inadequately sealed shaft. House A is protected by underlying impermeable clay, whereas house B is vulnerable to gas invasion.



in the iron and steel industry, and in the electricity generation sector. Coal provides around 28.1% of global primary energy needs and generates about 39.3% of the world's electricity, amounting to some 9538 TWh in 2015. Of this figure, the three leading producers of coal-fired electricity were the PRC with

4109 TWh, the USA with 1471 TWh, and India with 1042 TWh (IEA 2017). In addition, 14% of the world's total black coal production is currently utilised by the steel industry, 60% of which is dependent on coal. Recent years have seen an increase in international traded coal to a total of 1333.5 Mt in 2016, and in particular

the trading of steam coals for electricity generation. In 2015, 1045.0 Mt of steam coal and 282.1 Mt of coking coal was traded internationally (IEA 2017). Coal consumption in the iron and steel industry is moving towards the use of lower quality coking coal (soft and semi-soft) in blends with high-quality coking coal. The heating market, including other areas of industry, domestic, and miscellaneous consumption, will continue to contract.

### 12.3.1 Electricity Generation

Although this volume is essentially concerned with coal and its character, its uses cannot be ignored, especially as the market forces of supply and demand will determine where and how much coal will be mined in the future. Because electricity generation is the biggest single user of coal, its environmental effects and technologies to control pollution are briefly discussed here.

Electricity generation is singled out as one of the largest causes of pollution of the atmosphere. The rapid growth of the demand for electricity has led to large increases in production, and therefore large increases in emissions, which in turn has brought attendant environmental problems (Figure 12.13).

Nowadays, boilers are designed to burn a range of coals from lignite to anthracite. The

majority of stations burn coals in the middle of this range; these are selected according to their ash, sulfur, moisture, and volatile matter contents, together with their heating value (calorific value [CV]), grindability, and ash fusion temperature. The performance of different coals in terms of their specifications and their pre-combustion, combustion, and post-combustion performances is well documented elsewhere. In the context of this account, it is the result of burning coal in power plants and the direct contribution this makes to the environment in terms of waste products, both solid and gaseous, that is of concern here. Environmental legislation governs the limits of emissions and waste products from power plants, and coals that are likely to cause problems are not likely to be utilised in the future. This can have serious repercussions on coalmining. For example, the low sulfur requirements for coals to be used for power generation in the USA has seen a decline in the mining of high-sulfur coals in the traditional coalfields in eastern USA, in Illinois, Indiana, and western Kentucky, and the enormous expansion in mining low-sulfur coal in the western coalfields of Wyoming and Colorado.

As a guide, Table 12.3 gives the coal specifications that are normally used in coal-fired boilers, although coals outside the given ranges can be burned.



**Figure 12.13** Modern coal-fired power station, Inner Mongolia, PRC. *Source:* Photograph courtesy of Dargo Associates Ltd.

**Table 12.3** Normal range coal specifications for pf-fired boilers.

Parameter	Range	Comment
Total moisture	Max. 15% (ar)	If high, creates handling problems. The limits are higher for lignites and low-rank coals. Reduces net CV
Ash	Max. 20% (ad)	If high, creates fly ash problem. Reduces net CV
Volatile matter	Min. 20–25% (daf)	For conventional pf burners
CV	High	Almost any CV fuel can be used, the higher the better
Sulfur	Max. 0.8–1.0% (ad)	Maximum value dependent on local emission regulations
Nitrogen	Max. 1.5–2.0% (daf)	Various limits apply in some countries because of NO <sub>x</sub> emissions
Chlorine	Max. 0.2–0.3% (ad)	Causes ash fouling problems in boiler
Hardgrove grindability index (HGI)	Min. 45–50	Lower HGI values require larger grinding capacity and more energy
Ash fusion temperature	Various	Dry bottom boiler, initial deformation temperature >1200 °C Wet bottom boiler, flow temperature <1300 °C
Maximum size	Max. 40–50 mm	Dependent on capacity of grinding equipment
Fines content (–3 mm)	Max. 25–30%	High fines can increase moisture content and create handling problems

Source: BP Coal Ltd (1987).

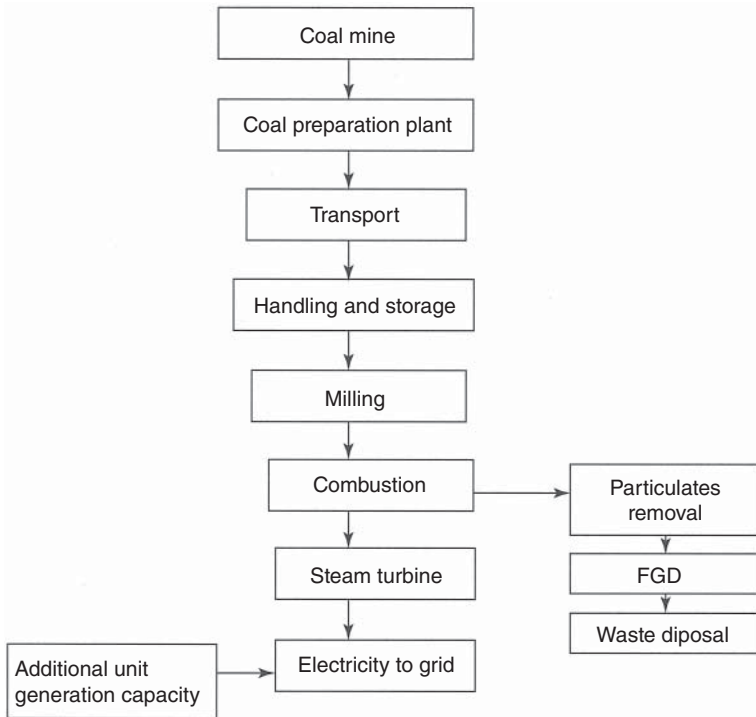
In simple terms, as shown in Figure 12.14, coal is brought from the mine to the power plant, the coal having a quality determined and agreed to by both parties; it is then stockpiled and, when required, fed through the mills and then into the combustion chamber. It is at this stage that waste products are generated. Exhaust gases contain particulates, oxides of sulfur and nitrogen, and volatile organic compounds. Fly ash removed from the exhaust gases can make up 60–85% of the coal ash residue in pulverised coal boilers. Bottom ash includes slag and coarse heavier particles than fly ash. The volume of solid waste may be substantially higher if environmental measures such as flue gas desulfurisation (FGD) are adopted and the residues are not reused in other industries. Steam turbines also require large quantities of water for cooling, including steam condensation. Water is also required for auxiliary station equipment, ash handling, and FGD systems. Water contamination arises from demineralisation, lubricating

and auxiliary fuel oils, and chlorine, together with any other chemicals used to control the water quality in the cooling system, which also serves to increase the water temperature.

#### 12.3.1.1 Emissions

In the industrialised countries, although the problem of emissions is of serious concern, the ability and desire to diminish the harmful elements in emissions is not consistent. Reasons are primarily financial, as control of harmful emissions requires the replacement or modification of existing equipment and/or the addition of new technology.

From an environmental point of view, attention is focused on emissions of particulate matter (PM) less than 10 μm in size, sulfur dioxide (SO<sub>2</sub>), on nitrogen oxides (NO<sub>x</sub>), and fly ash. In addition, CO<sub>2</sub> and dioxins have also attracted a great deal of attention in deciding on the effects of these pollutants on the atmosphere and global ecology.



**Figure 12.14** Flow diagram showing stages from coal delivery to electricity generation in a modern power station.

From 2005 to 2015, coal-fired power generation grew 34%, whereas total power sector emissions of  $\text{SO}_2$ ,  $\text{NO}_x$ , and PM decreased by 55%, 34%, and 32% respectively (International Energy Agency [IEA] 2016). This is principally due to the introduction of emission standards for coal-fired power plants, which has led to the use of low-sulfur coal or to the installation of pollution control technologies. Regulation for  $\text{SO}_2$ ,  $\text{NO}_x$ , and PM emissions from coal-fired plants has been introduced in many countries, Table 12.4 gives the emission limits for existing and new power plants in major generating countries or regions, based on IEA (2016). In industry, coal accounts for more than 75% of combustion-related  $\text{SO}_2$  emissions, 38% of  $\text{NO}_x$ , and 37% of PM emissions.

To estimate the amount of  $\text{SO}_2$  produced daily from the power station stack, it is necessary to know the CV, the hydrogen and moisture content of the coal, and the station heat rate. For example, a  $500 \text{ MW}_e$  plant using coal with 2.5% sulfur, 16% ash, and a CV of  $30 \text{ MJ kg}^{-1}$  ( $12\,898 \text{ Btu lb}^{-1}$ ) will emit

200 t of  $\text{SO}_2$ , 70 t of  $\text{NO}_2$ , 500 t of fly ash, and 500 t of solid waste and have 17 GWh of thermal discharge (Ackermann et al. 1999, pp. 413–421).

$\text{SO}_2$  is one of the principal gaseous pollutants; it can be a hazard to human health and damage both the natural and man-made environment. Total global emissions of  $\text{SO}_2$  from the combustion of coal, oil and oil-derived fuels, refining, and smelting amounted to 80 Mt per year in 2015. To control the emissions of  $\text{SO}_2$ , it may be possible to use a fuel that has a lower sulfur content or has a sulfur content that can be removed before use. For example, the inorganic sulfur fraction in coal is usually in the form of iron pyrite ( $\text{FeS}_2$ ) and this can be removed in a coal preparation plant, and has been effective in reducing sulfur content in traded coals, e.g. in South Africa. However, the organic sulfur and sulfate fractions will remain within the coal. The alternative is to remove the sulfur during use and the most efficient means of controlling  $\text{SO}_2$  emissions is to remove the  $\text{SO}_2$  from



**Table 12.4** Emission limits for existing and new power plants in selected major generating countries/regions.

Region	Policy	Emission limits (mg m <sup>-3</sup> )					
		SO <sub>2</sub>		NO <sub>x</sub>		PM	
		Existing	New	Existing	New	Existing	New
PRC	Emission standard of air pollutants for thermal power plants	200–400	100	200	100	30	30
EU	Industrial Emissions Directive	200–400	150–400	200–450	150–400	20–30	10–20
USA	New Source Performance Standards	160–640	160	117–640	117	23	23
India	Environmental (Protection) Amendment Rules 2015	200–600	100	300–600	100	50–100	30
Indonesia	MOE decree no. 212008	750	750	850	750	150	100
South Africa	Minimum emissions standards published by government	3500	500	1100	750	100	50

'Existing' refers to the emission limit for currently operating power plants. 'New' refers to the limit for planned or proposed plants.

Source: Based on IEA (2016).

the flue gases before they are released to the atmosphere (see Section 12.3.1.2).

NO<sub>x</sub> emissions are produced by the reaction of the nitrogenous compounds in the coal with oxygen. It is considered that nitrogen is released during devolatilisation and enters the gas phase as hydrogen cyanide or ammonia, where it reacts with air to form NO<sub>x</sub> in a complex series of chemical reactions. High temperatures and rapid heating rates maximise the yield of the volatile nitrogen species and the presence of free O<sub>2</sub> favours the formation of NO<sub>x</sub>. Among the NO<sub>x</sub> emissions from power plants is nitrous oxide (N<sub>2</sub>O); this gas has a significant effect on the atmosphere, it being a strong absorber of infrared radiation and considered as a major contributor (20%) to ozone depletion. Global NO<sub>x</sub> emissions in 2015 were 107 Mt, of which industry contributed 28% and power 14%.

Dioxins is the general name given to a group of some 210 species consisting of 75 polychlorinated dibenzo-*para*-dioxins and 135 polychlorinated dibenzofurans. The majority of these species are considered to pose little threat to health at the levels generally found. A

small group of dioxins (17) are of great concern because of their toxicity/carcinogenicity, and these are formed as unwanted by-products in some industrial processes and in combustion processes such as waste incineration. Dioxins are present in all environmental media, and there are many natural sources of dioxins, such as forest fires. Since the incomplete combustion of any organic material can result in the formation of trace hydrocarbons in flue gases, and coal contains chlorine, the combustion of coal has been implicated as a significant contributor to the release of dioxins to the atmosphere. Research carried out on dioxin emissions from coal in Europe has recorded low concentrations of dioxins in flue gases from coal-fired plants. The highest dioxin emissions were from domestic combustion appliances (Dorrington et al. 1995).

The concentration of trace elements in ash is dependent upon particle size. Increasing concentrations are correlated with decreasing particle size. In coals from Indiana, USA, it has been demonstrated that concentrations of lead, thallium, antimony, cadmium, selenium, arsenic, zinc, nickel, chromium, and sulfur

**Table 12.5** Distribution of elements among bottom ash, fly ash, and flue gas (Valkovic 1983).

Element	Bottom ash (22.2%)	Fly ash (77.1%)	Flue gas (0.7%)
Aluminium	20.5	78.8	0.7
Antimony	2.7	93.4	3.9
Arsenic	0.8	99.1	0.05
Barium	16.0	83.9	0.09
Beryllium	16.9	81.0	2.0
Boron	12.1	83.2	4.7
Cadmium	15.7	80.5	3.8
Calcium	18.5	80.7	0.8
Chlorine	16.0	3.8	80.2
Chromium	13.9	73.7	12.4
Cobalt	15.6	82.9	1.5
Copper	12.7	86.5	0.8
Fluorine	1.1	91.3	7.6
Iron	27.9	71.3	0.8
Lead	10.3	82.2	7.5
Magnesium	17.2	82.0	0.8
Manganese	17.3	81.5	1.2
Mercury	2.1	0	97.9
Molybdenum	12.8	77.8	9.4
Nickel	13.6	68.2	18.2
Selenium	1.4	60.9	27.7
Silver	3.2	95.5	1.3
Sulfur	3.4	8.8	87.8
Titanium	21.1	78.3	0.6
Uranium	18.0	80.6	1.5
Vanadium	15.3	82.3	2.4
Zinc	29.4	68.0	2.6

were markedly increased in the size range 0.65–74  $\mu\text{m}$ . In Australia, a threefold increase in concentrations of gallium, germanium, mercury, and lead have been observed between the coarse (>50  $\mu\text{m}$ ) and fine (<2  $\mu\text{m}$ ) fly ash fractions. Table 12.5 shows the distribution of elements between the main coal residues in a power plant, i.e. bottom ash, fly ash, and flue gas, taken from an example in the USA. The trace elements present in high percentages in the flue gas fraction can be seen to be chlorine,

chromium, mercury, nickel, selenium, and sulfur. Particulate emissions will contaminate surrounding soil areas and, if inhaled, have serious health effects on the local population.

To control the emission of the fine particulate fraction and its undesirable trace element content, emission level limits have been imposed by most countries. Current permitted particulate emission levels are to be substantially reduced. For example, in the UK, emissions levels for  $\text{NO}_x$  and  $\text{SO}_2$  were 1714 kt and 773 kt respectively in 2005, with a commitment to reduce these by 2020 by 55% to 771 kt and 59% to 317 kt respectively, and to further reduce emissions by 73% to 463 kt and 88% to 93 kt respectively (Department of Environment, Food and Rural Affairs 2019).

Power plants burning high-ash coals, as is the case with most Gondwana coals, are faced with the problem of large amounts of fly ash disposal. Large plants using coal with ash contents of  $\geq 50\%$  can produce over 1 Mt of fly ash per year. If the fly ash is suitable for industrial use, then it is simply removed from site. If, however, the amount of fly ash produced far exceeds any industrial requirement, then the fly ash has to be disposed of. In the case of a mine-mouth power station, and if the mine is an open-pit operation, fly ash is returned and used to fill the mined-out void. Stockpiling of fly ash has proved a problem, particularly in dry, windy conditions, e.g. during the dry season in India.

Where fly ash is considered for use in other industries, it has been classified based on the source coal and specified major element oxide contents. The American Society for Testing and Materials (ASTM) has differentiated two types of fly ash (Class F and C). Class F fly ashes are derived from anthracite and bituminous coals and should contain a minimum of 70% of  $\text{SiO}_2 + \text{Al}_2\text{O}_3 + \text{Fe}_2\text{O}_3$ , and Class C fly ashes are derived from lignite and subbituminous coals and should contain a minimum of 50% of  $\text{SiO}_2 + \text{Al}_2\text{O}_3 + \text{Fe}_2\text{O}_3$ . Both classes of fly ashes should possess their own distinctive sets of chemical, physical, and engineering

properties. This classification is used as the basis for selecting fly ashes as an admixture in cement and concrete.

CO<sub>2</sub> is emitted as a product of coal combustion, and it is related to the C content of the coal burned at the time. Old and inefficient plant allows higher CO<sub>2</sub> emissions than modern installations do. In 2015, global CO<sub>2</sub> emissions reached 32.3 Gt, and coal accounted for 45% of these, principally from electricity and heat generation. However, 2015 was the first year to show a decrease in global emissions from coal combustion (IEA 2016).

### 12.3.1.2 Flue Gas Desulfurisation

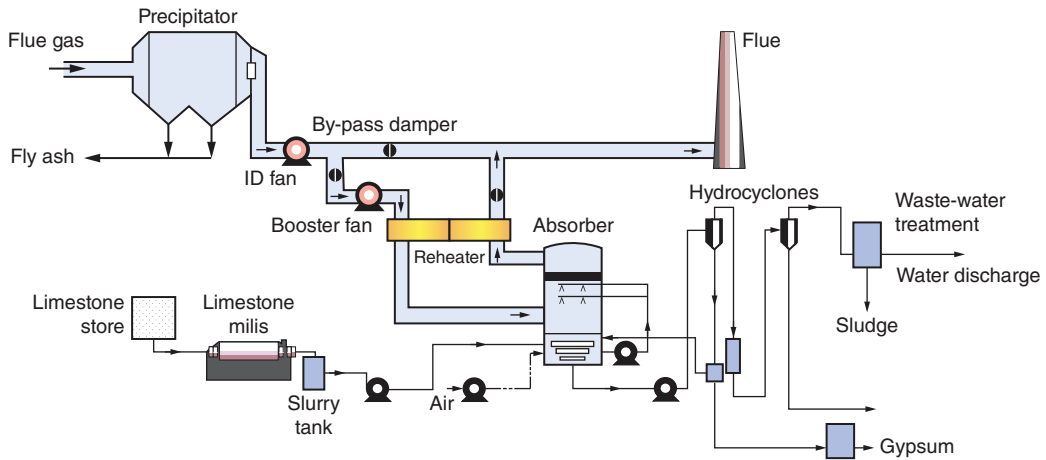
The control of SO<sub>2</sub> emissions has centred on FGD technologies (DTI Report 2000a; IEA 2016). These are now widely used to control the emissions of SO<sub>2</sub> and sulfur trioxide (SO<sub>3</sub>). Primary combustion technologies and measures tend to be preferred over end-of-pipe technologies because of their lower installation cost. An alternative to control technologies is the use of low-sulfur coal to fuel power plants. Coal blending can achieve compliance at a lower cost than a retrofit with FGD equipment. The dependency of iron and steelmaking on coal for coke-making opens up the possibility of using less metallurgical coal types with a lower SO<sub>2</sub> content. A variety of FGD processes are available; most use an alkali sorbent to recover the acidic sulfur compounds from the flue gas. The most commonly used alkaline materials are limestone (calcium carbonate), quick lime (calcium oxide), and hydrated lime (calcium hydroxide). Limestone is an abundant and relatively cheap material, and both quick lime and hydrated lime are produced from limestone by heating. Other alkalis are sometimes used; these include sodium carbonate, magnesium carbonate, and ammonia. The alkali used reacts with SO<sub>2</sub> in the flue gas to produce a mixture of sulfite and sulfate salts, and this reaction can take place either in bulk solution ('wet' FGD processes) or at the wetted surface of the solid alkali ('dry' and 'semi-dry' FGD processes). Selection of the

FGD process is made on economic grounds, i.e. the process selected will be the one with the lowest overall through-life cost. Technical considerations include the degree of desulfurisation the process can achieve and the flexibility of the process.

The most common FGD process now being installed worldwide is the limestone–gypsum wet scrubbing process, which has evolved over 50 years. Today, a plant would be designed to achieve a high-quality gypsum product, which can be used in wallboard manufacturing. There are variants in equipment arrangement, absorber type, and reheat methods, according to the client's requirements. Figure 12.15 shows one of the most common limestone–gypsum plant layouts. The FGD plant is located downstream of the electrostatic precipitator so that most of the fly ash from combustion is removed before the gas reaches the FGD plant. In a coal-fired plant, fly ash removal would be 99.5%. The gas is then scrubbed with the recirculating limestone slurry to remove the required amount of SO<sub>2</sub>. FGD plant manufacturers claim that over 95% of SO<sub>2</sub> can be removed with the absorber. This process also removes almost all of any hydrogen chloride (HCl) in the flue gas.

The calcium carbonate from the limestone reacts with the SO<sub>2</sub> and O<sub>2</sub> (from air) to produce gypsum (hydrated calcium sulfate, CaSO<sub>4</sub>·2H<sub>2</sub>O) which precipitates from solution in the sump. HCl is also dissolved in the water and neutralised to produce calcium chloride solution. The gypsum slurry is extracted from the absorber sump and treated for storage and removal to industrial users. Fresh limestone is then pumped into the absorber sump to maintain the required pH. The remaining gas is reheated and then exhausted to the stack. This process will usually offer the lowest through-life cost option for large inland coal-fired power plants with medium- to high-sulfur fuel, a high load factor, and a long residual life.

Other wet FGD processes include seawater washing, ammonia scrubbing, and the



**Figure 12.15** Schematic flow diagram of a limestone–gypsum FGD process.

Wellman–Lord process (using aqueous sodium sulfate solution). Semi-dry processes are the circulating fluidised bed, spray dry, and duct spray dry, all producing dry powdered mixtures of calcium compounds. Dry processes inject hydrated lime or sodium bicarbonate into the furnace cavity of the boiler and absorb  $\text{SO}_2$ . The spent sorbent is extracted with the fly ash as a mixture of ash and calcium or sodium compounds.

### 12.3.1.3 Other Emission Controls

The reduction of  $\text{NO}_x$  emissions requires the fitting of  $\text{NO}_x$  reduction equipment to both existing plant (retrofitting) and to new plant. Conventional pf burners are designed to achieve rapid intimate mixing of the fuel with the combustion air. Since the fuel devolatilises in high temperatures and air-rich conditions, the level of  $\text{NO}_x$  is high. To counteract this, low- $\text{NO}_x$  burners are used based on the physical separation of three air streams to the burner, the addition of each air stream helping to reduce  $\text{NO}_x$  production. The use of low- $\text{NO}_x$  burners can cause slagging problems in the furnace and can result in a loss of efficiency in the plant. Current research is seeking to overcome such problems and make  $\text{NO}_x$  reduction cost effective as well as environmentally desirable.

A post-combustion  $\text{NO}_x$  control method is known as selective catalytic reduction (SCR), in which ammonia is mixed with the flue gas and in the presence of a catalyst reacts with  $\text{NO}$  and  $\text{NO}_2$  in the gas to form molecular nitrogen and water. SCR systems can be costly and difficult to retrofit, due to their physical size, but they are widely installed in Japan and Germany.

Several combined  $\text{SO}_2$  and  $\text{NO}_x$  removal systems have been developed, and one such system is the SNOX process developed in Denmark. The flue gas is reheated and then undergoes SCR; the flue gas is then further heated and a second catalytic reactor oxidises  $\text{SO}_2$  to  $\text{SO}_3$ . The gas is then cooled to condense out the  $\text{SO}_3$  as sulfuric acid. The condenser uses glass tubes to prevent excessive acid corrosion. SNOX units have been built in Denmark, Italy, and the USA.

Regulatory limits in air pollutant emissions often require the implementation of post-treatment mitigation technologies, especially for certain pollutants such as PM.

Table 12.6 lists mitigation technologies for air pollutants in power generation and industry (IEA 2016), and Figure 12.16 illustrates typical emissions control systems for power plants. Reduction in  $\text{CO}_2$  emissions at pf plants has mainly been by chemical absorption processes,

**Table 12.6** Selected mitigation technologies for air pollutants in power generation and industry.

Pollutant	Mitigation technology	Type of technology	Abatement efficiency (%)
SO <sub>2</sub>	Wet flue-gas desulfurisation	End-of-pipe	70–98 <sup>a</sup>
	Spray-drier absorption	End-of-pipe	50–70
NO <sub>x</sub>	Low and ultralow-NO <sub>x</sub> burners	Integral to combustion process	20–30
	Selective catalytic reduction	End-of-pipe	90
	Selective non-catalytic reduction	End-of-pipe	<50
PM	Fabric filtration	End-of-pipe	>99
	Electrostatic precipitators	End-of-pipe	>99

a) Abatement efficiency based on coals with 0.3–4.8% sulfur content.

Source: From IEA (2016).

and current research is seeking to improve on these methods. CO<sub>2</sub> emissions have been linked to reduced efficiency levels in coal-fired plants, particularly in those countries not in the Organisation for Economic Co-operation and Development. This is due to a number of factors, poor quality coal, small and ageing units, poor maintenance, and obsolete technologies.

#### 12.3.1.4 Fluidised-Bed Combustion

In the electricity generating industry, fluidised-bed combustion (FBC) in its various forms offers a technology that can be designed to burn a variety of fuels efficiently and in an environmentally friendly manner. There are two main processes in use: bubbling fluidised-bed combustion (BFBC) and circulating fluidised-bed combustion (CFBC). Both of these can be either atmospheric or pressurised in operation.

Briefly, the BFBC method is when a bed of packed small particles is subjected to an upward gas flow; the bed remains static, but the pressure drop across it increases in proportion to the increasing gas flow rate. When the pressure drop across the bed particles equals the weight per unit area of the bed, the bed becomes suspended. The bed is then considered to be at minimum fluidisation. Increases in the gas flow above the minimum will produce bubbles; the upwards and

sideways coalescing movements of the bubbles provide intense agitation and mixing of the bed particles. This results in the bed particles transferring heat at very high rates from burning fuel to cooler surroundings. Fuel can be fed into and burned in the bubbling bed. In order to burn coal efficiently, and where limestone is used to successfully retain SO<sub>2</sub>, the bed particles need to be controlled in the temperature range 800–900 °C. Over the last 40 years, BFBC technology has been shown to be well suited to the utilisation of high-moisture, high-ash coals as well as low-volatile anthracite. In addition to the primary air source in the combustor, secondary air is introduced at several levels above the bed; this provides a balance of the combustion air to reduce NO<sub>x</sub> levels.

CFBCs are a development of the BFBC, whereby the velocity of the circulating air is increased, resulting in the particles being carried upwards, away from the bed surface, and the distinctive bubbling bed disappears. The combustion chamber is then filled with a turbulent cloud of particles that no longer remain in close contact with each other. The burning particles are recovered from the air flow and fed back into the lower part of the combustion chamber. A circulating fluidised bed can sustain combustion in a similar manner to a bubbling bed, and the turbulent contact between the coal particles present and the bed solids stabilises the overall temperature.



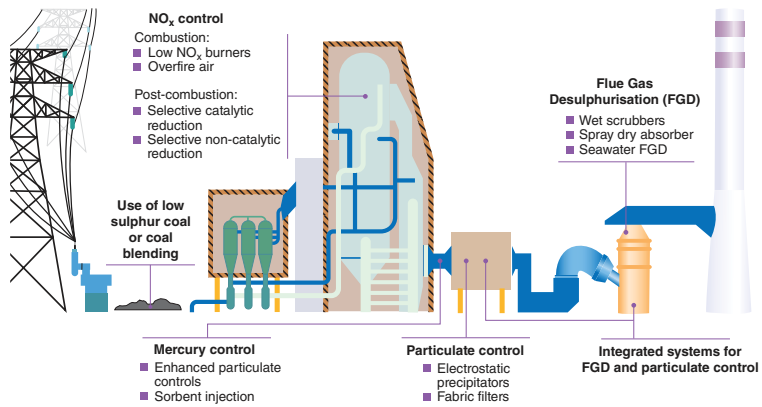


Figure 12.16 Typical emissions control systems for power plants (IEA 2016).

Again, additional combustion air is introduced at higher levels to reduce  $\text{NO}_x$  levels.

Many CFBC units proved capable of achieving lower levels of the primary pollutants,  $\text{NO}_x$ ,  $\text{SO}_2$ ,  $\text{CO}_2$ , and PM. Sorbent is added to the system to control  $\text{SO}_2$  emissions,  $\text{NO}_x$  levels are minimised by careful bed temperature control, and particulate control systems are installed. Pressurised FBC has the advantage that the hot combustion gases leave the combustor under pressure, which, if maintained, and the gases cleaned, can be fed directly into a gas turbine. Several thousand BFBC plants are in operation, predominantly in the PRC; CFBC plants are dominant in Asia, again principally in the PRC, with over 50% of installed capacity; CFBC and pressurised FBC plants are now expanding into the USA and Europe, together with a high level of activity in Japan (DTI Report 2000b).

### 12.3.2 Other Major Users

#### 12.3.2.1 Iron and Steel Production

The potential source of pollution in the iron and steelmaking process relating to coal is in the production of coke. Other emissions are part of the steelmaking process and are not considered here.

In 2016, world crude steel production was 1621 Mt, a slight drop compared with 2014. The top steel producers in 2015 were the PRC with 803 Mt, Japan with 105 Mt, India with 89 Mt, and the USA with 78 Mt. Almost 70% of global steel is produced in basic oxygen furnaces (BOFs), which require 770 kg of coal to produce 1 t of steel. A further 29% of steel is produced in electric arc furnaces, and for this process about 150 kg of coal is required to produce 1 t of steel.

Coal is heated in an oxygen-free environment until the bulk of the volatile constituents have been driven off. The solid residue is known as coke, and its principal use is to provide heat energy and to act as a reducing agent for iron ore in the blast furnace. Coke has to be a strong material, able to withstand handling

and be capable of supporting the overlying weight of coke as it moves down through the blast furnace. Coke can be produced from a single coal or a blend of selected coals. Only coals with a specific range of rank and type are capable of forming coke, and particular properties of the coal decide the nature of the coke produced (see Section 4.3.2).

Coke can be manufactured within the iron and steel plant on a large scale or produced on a small scale and transported to the plant. Traditionally, coke was produced in beehive ovens by combusting covered piles of coal until all was carbonised, usually taking three to four days. This method is still widespread in the PRC (Figure 12.17), where coke-making is still a cottage industry. This method has been replaced by carbonising a thin vertical layer of coal in less than a day; the coke oven is a standard part of any iron and steel plant. Both methods have resulted in the venting of gas by-products to the atmosphere. This has produced high levels of local atmospheric pollution as well as raising regional  $\text{CO}_2$  emissions. In 2016, global coke production was 649 Mt (www.statista 2018).

Reductions in  $\text{CO}_2$  emissions have been achieved by reducing fuel consumption of the blast furnace, making more efficient use of exhaust heat, increasing on-site electricity generation from waste gases, using expansion turbines, the partial substitution of coke through the injection of pf and heavy fuel oil, and by introducing larger blast furnace units to gain from economies of scale (World Coal Institute 2001).

In the UK, in Yorkshire, the long-established Monckton Cokeworks produced 535 t of coke and 276 000  $\text{m}^3$  of coke-oven gas per day from 790 t of coal, with an annual coke production of 200 000 t before closing in 2014. A new 11 MW<sub>e</sub> combined heat and power (CHP) plant had been developed as part of an environmental clean-up and redevelopment programme. Of the coal gas produced from the coke ovens, 40% was reused to heat the coke ovens and 60% was diverted to the new CHP plant. The



**Figure 12.17** Local coke manufacture in small 'ovens', Guizhou Province, PRC. *Source:* Photograph courtesy of Dargo Associates Ltd.

CHP plant utilised the coke oven waste gas to produce  $31 \text{ t h}^{-1}$  of superheated steam; this, in turn, produced 11 MW, with 1.4 MW required for on-site demand and the remainder sold via the local grid. This facility was able to generate more added value from the surplus high-CV gas released during the coke-making process with the added benefit of environmental improvement.

In Europe, these methods have been implemented by closing down old-style steel plants and replacing them with BOF and electric arc furnace plants. The energy consumption of basic oxygen steelmaking is very low compared with other production methods. In addition, it is possible to transform this method into a net supplier of energy by utilising the waste converter gases. The energy input and associated level of  $\text{CO}_2$  emissions per tonne of steel produced by the international steelmaking industry could be reduced by over 25% if the status of the production technology used in Europe and the USA was introduced on a global scale. Currently, the  $\text{CO}_2$  emissions per tonne of steel are three times higher in the PRC ( $3.9 \text{ t CO}_2$ ) than in western Europe ( $1.3 \text{ t CO}_2$ ).

Apart from utilising new technology, a switch to iron ores of higher quality would lower the consumption of coal and reducing agents in the blast furnaces. The main difficulty in achieving a major reduction in  $\text{CO}_2$  emissions is a financial one; it will take very large capital expenditure to achieve  $\text{CO}_2$  reductions to comply with the present C reduction requirements as defined in the Paris Agreement (see Section 12.6.8).

#### 12.3.2.2 Industrial Use

Although electricity generation and iron and steel production make up the bulk of the use of coal by the industrial sector, coal is used in a number of industries for heating. The principal effect of coal on the environment is the venting of waste gases to the atmosphere. The share of coal's contribution to this has been reduced due to the fact that modern industry has made substantial reductions in the use of coal for conventional heating, having replaced it with gas or oil. However, industries such as the manufacture of cement still utilise significant quantities of coal. The cleaning of flue gas and reducing the particulate emissions are

all contributing to the improvement of air quality.

The upgrading of low-rank coals for industrial use has become widespread, particularly in the drying of brown coals, notably lignites, to reduce the total moisture content and to increase the CV. Dried brown coal is then used as pf or compressed into briquettes or pellets for both industrial and domestic use.

### 12.3.2.3 Domestic Use

Coal as a household fuel has almost disappeared in most well-developed countries. Strict regulations on air quality in urban areas have led to the replacement of coal by gas and oil heating. The thick smogs of large cities is now a thing of the past, although photochemical smog produced by the internal combustion engine is still a reality.

In less developed countries, domestic heating using coal is still prevalent. In the Commonwealth of Independent States, the PRC, and eastern Europe coal is plentiful and oil and gas are expensive or not available, so atmospheric pollution can still reach high levels. Improvement in industrial use and the gradual replacement of coal for heating will reduce the problem, but this is likely to be a long-term prospect.

### 12.3.3 Coal Transportation

The transportation of coal is by road, rail, and conveyor on land, and by barge and ocean-going vessels on water. The effects on the environment are usually minimal but can cause local problems.

Road transport: coal is moved from the mine to the customer by lorry fleets. This means using public roads, and this can cause problems such as wear and tear, traffic congestion, and dust from coal loads. In countries such as the PRC and India, where villages alongside main roads are numerous, coal lorries do cause degradation of roads and village streets; this, coupled with poor maintenance, produces bad road conditions

and slow delivery schedules. Roads built specifically for the purpose of transporting coal do not impinge on the local transport system; e.g. in East Kalimantan, Indonesia, private coal roads transport coal to the loading areas on the major rivers or ports.

Rail transport: the overland transport of large shipments of coal by rail is the established means throughout the world. Rail transport has little effect environmentally other than dust and noise at the loading/unloading areas. Where coal is loaded/unloaded automatically, such effects are minimised.

Conveyor: overland conveyors are used to transport coal from the mine to the stockyard. Conveyors are usually covered and have no adverse effect on the environment.

Water transport: coal transported by barge or ocean-going vessel has only a dust problem on loading/unloading, and some coals have the propensity for spontaneous combustion (see Section 12.2.6).

## 12.4 Health

Worldwide, coalmining remains a growth industry, and coalmining, particularly underground mining, is still perceived to be a dangerous occupation. A great amount of time and money has been invested in mining health and safety research, and the industry is much safer as a result.

Most of the problems associated with mining are common to all countries, i.e. strata collapse, fires and explosions, dust, fumes and heat, noise, and water. Improvements in mining practices and in the development of equipment with high safety standards have helped to drastically reduce accidents in both underground and open-pit mines. Improved ventilation and sound proofing have made underground conditions less hazardous, and a good understanding of the groundwater conditions can prevent unexpected water inflows in mines. Apart from accidents, coalmining, and in particular underground mining, has

historically been associated with lung diseases caused by the breathing in of coal and stone dust over a number of years. Pneumococcosis and silicosis were the legacies of the coalminer, together with newer complaints such as vibration white finger.

Coalminers working in deep mines with high virgin strata temperatures, high use of machinery, intake air passing over machinery, and standing and sprayed water can be affected by the heat and humidity. The human thermoregulatory system tries to regulate the body temperature at 37°C, but this control is not maintained when working in hot and humid conditions; the body temperature starts to rise and produces various physiological effects, such as heat rash, fainting, heat exhaustion, and cramps, culminating in heat stroke when the body temperature exceeds 41°C (Leeming and Fifoot 2001). To combat these effects, it is essential to maintain good air quality to the working areas and to prevent unwanted heat being picked up by the air stream. A well-maintained ventilation system is also important to disperse diesel fumes where such equipment is in use.

In open-pit mines, air quality is maintained by dust control on in-pit roadways, and protection is worn to exclude noise. Local populations do have problems with respiratory complaints such as asthma that they attribute to mining, and these are often used as claims for compensation.

One hazard that is now receiving greater attention is the presence or absence of radon in mine waters and mine atmospheres. Though harmless externally, radon gas, if breathed in on dust and moisture particles, remains in the respiratory system. The major health hazard from radon is thought to be an increased risk of lung cancer. Radon gas is soluble in water, and it may be carried for great distances. When such groundwater discharges into mine workings, there is a pressure release of gas into the mine atmosphere. The occurrence of radon in coalmines is now closely monitored, and hydrogeological studies are essential to

anticipate whether any radon-rich water is likely to inflow into a mine.

Also associated with coalmining is the release of naturally occurring toxic organic compounds into the groundwater. In Bosnia, Bulgaria, Croatia, Romania, and Serbia, researchers investigated a link between organic compounds leached by groundwater from shallow lignite deposits and the disease known as Balkan endemic nephropathy. In the 1990s and 2000s, geochemical analyses of well water from villages located on Pliocene lignite deposits showed the presence of toxic organic compounds not observed in areas not underlain by lignite bearing strata. Finkelman et al. (1991) and Orem et al. (1999, 2007) have investigated this and the implications on health.

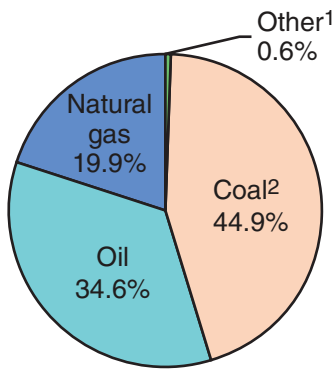
## 12.5 Carbon Capture and Storage

To address the challenge of climate change, while still meeting the increasing need for affordable energy, will require the implementation of energy-efficient and low-C technologies.

Capturing CO<sub>2</sub> that would otherwise be released into the atmosphere by injecting it into deep geological formations will reduce GHG emissions from coal use while still permitting this energy source to be used effectively. Capturing the CO<sub>2</sub> emitted by the use of coal and other fossil fuels and injecting it for storage in deep geological formations is the only currently available technological solution that will reduce GHG emissions whilst maintaining the growth of global energy demands. In 2015, 32 294 Mt of CO<sub>2</sub> was emitted. The fuel shares of CO<sub>2</sub> emissions from fuel combustion for 2015 are shown in Figure 12.18, where coal makes up 44.9% of the total (IEA 2017).

CCS is not a new technology; there have been decades of operational experience from industrial-scale CCS projects, such as used in the oil industry for enhanced oil recovery. In addition, there have been numerous research





32 294 Mt of CO<sub>2</sub>

**Figure 12.18** The 2015 fuel shares of CO<sub>2</sub> emissions from fuel combustion (IEA 2017). 1: includes industrial waste; 2: includes peat and oil shale.

CCS projects. The current rate of CCS is behind the desired reduction in GHG emissions, and there has been limited action to accelerate CCS during the Kyoto Protocol's first commitment period (2008–2012). The provision in the Kyoto Protocol to the United Nations Framework Convention on Climate Change (UNFCCC) set the foundations for the development of CCS as part of the challenge of meeting the agreed CO<sub>2</sub> reduction targets. These are included in the provisions of the Kyoto Protocol's Clean Development Mechanism (CDM). In 2011, CCS was included within the list of activities eligible under the CDM; but before CCS projects can be eligible for credit under the CDM there must be national legislation in force in the nation hosting the project (Bell 2013). The EU Directive (2009) on the geological storage of CO<sub>2</sub> gives a common framework for member states and is a key element of the EU's climate change measures. The directive deals primarily with storage, its location, the operator and management, plus monitoring for leaks and management irregularities. CCS is not a mandatory requirement, but the directive provides a framework for CCS implementation

within the EU (Bell 2013). The directive's aim is to store CO<sub>2</sub> in a safe environment without risk to the environment and human health. Such storage sites can be onshore and offshore within the territory of member states. In addition, the directive imposes a requirement that new combustion plants with capacities >300 MW should be designed so as to allow the retrofitting of CO<sub>2</sub> capture technology subject to availability of suitable storage sites and that CO<sub>2</sub> capture is technically and economically feasible (Bell 2013).

Storage sites will be determined by geological features, and on geographical location. In northern Europe, storage under the North Sea is the preferred option, where onshore storage is not being considered. In North America, onshore storage is likely to be used, partly due to the logistics of CO<sub>2</sub> transport to offshore locations. This is a similar scenario for the PRC. In Australia, both onshore and offshore locations have been considered (Freund 2013).

The coal industry is committed to minimising its GHG emissions, and the improvement of coal combustion efficiency is an important factor: the replacement of older coal-fired power plants with larger, more efficient plants could reduce GHG emissions by 5.5%, which can be compared well with the intended effect of all the measures included in the Kyoto Protocol of climate change of 5% (World Coal Institute 2010).

Integrated gasification combined cycle (IGCC) power plants are more efficient than burning coal directly and therefore less polluting, as the process facilitates C capture and environmental controls. However, owing to the high cost of construction, only around eight IGCC plants are operating worldwide, and up to 18 projects have been cancelled since 2011, although a possible 20 projects are still under consideration, including 10 in the PRC. IGCC plants of 300 MW capacity have been built in the USA and Europe, with mixed success; for example, the 335 MW plant at Puertollano in Spain was commissioned in 1998 but closed in 2016 due to technical and financial difficulties.

GHGs such as CH<sub>4</sub> can and are being recovered from active and abandoned coalmines (Section 11.2.1.3.1), and CO<sub>2</sub> injection into deep unmineable coal seams or used for enhanced CBM extraction will provide a minor contribution to world CO<sub>2</sub> storage capacity but may be a significant storage option for certain countries. It is likely that only deep disused coalmines or deep unmineable coal deposits would be considered as suitable for CCS. The use of coal seams as storage for CO<sub>2</sub> raises a number of issues as to their suitability. The low level of permeability of coals, their ability to swell, and the sterilisation of coals from future mining are to be taken into account. There is also the possibility of coal seams being discontinuous laterally, as is common in basinal coal deposits; this may represent leakage risks if CO<sub>2</sub> is injected.

In Europe, injection into coal seams may provide a technical option for CO<sub>2</sub> storage, but injection rates will not match industrial supply rates, particularly from power generation. A significant environmental issue will be that, in order to obtain the highest permeability in coals, injection targets need to be at shallow depth and therefore can be in the fresh water groundwater zone (<500 m), and so the potential contamination of the groundwater system would need to be assessed. Countries such as the USA have established a prevention programme to protect the underground sources of drinking water from potential endangerment caused by CO<sub>2</sub> injection wells (Jackson 2010).

The storage of CO<sub>2</sub> in abandoned mine workings also has the risk of leakage through fissures and fractures produced during and subsequent to mining. Only deep mines with an overlying sequence containing a reliable seal could be considered, thus ensuring that groundwater will not be affected.

Therefore, it is considered that CO<sub>2</sub> may best be stored in depleted oil and gas reservoirs, deep saline aquifers, or unmineable coal seams. The storage efficiency for CO<sub>2</sub> in coal-beds is greater than in saline aquifers, due to the adsorption of CO<sub>2</sub>; however, the total

storage capacity is low, having only 1–2% of the storage capacity of saline aquifers (Pickup 2013). Models using numerical, analytical, and semi-analytical solution methods have been used to simulate the injection of CO<sub>2</sub> into subsurface formations (Bacon 2013). Long-term modelling of CO<sub>2</sub> storage and sequestration is an integral part of site selection.

Ongoing projects include the longest running industrial-scale storage project at Sleipner in the Norwegian sector of the North Sea. It commenced in 1996, injecting 1 Mt of CO<sub>2</sub> per year into the Utsira Sand, a large saline aquifer 900 m below the seabed. By 2011, more than 13 Mt of CO<sub>2</sub> had been securely stored. The project is intensely monitored using geophysics; 3D seismic surveys have been repeated every two years in a comprehensive time-lapse monitoring programme known as 4D seismic surveys (Chadwick 2011; Chadwick and Eiken 2013). Other large-scale CO<sub>2</sub> storage projects include the In-Salah Gas Project in Algeria, where 0.75 Mt of CO<sub>2</sub> per year is injected into sandstone at depths of 1800–1900 m. The CO2CRC Otway Project in Australia stores CO<sub>2</sub> and CH<sub>4</sub> in a depleted gas field at a depth of 200 m. The Ketzin Pilot site in Germany is the first European onshore CO<sub>2</sub> storage project, where more than 67 kt of CO<sub>2</sub> have been injected into Upper Triassic sandstones since 2008 (Leibschner et al. 2013). The use of CCS technology for coal-related projects is still in its infancy, but legislation and technical advancements will promote CCS as a significant contribution to the reduction of CO<sub>2</sub> into the atmosphere.

## 12.6 Environmental Regulations

### 12.6.1 Introduction

The effects of mining and utilising coal on the environment are closely monitored and regulated. Regulations and codes of practice are implemented at national, regional, and

at local government levels in the majority of industrialised countries.

In recent years, prominence has been given to the harmful effects of GHGs on the Earth's atmosphere; these gases comprise CO<sub>2</sub>, CH<sub>4</sub>, N<sub>2</sub>O, hydrofluorocarbons, perfluorocarbons, and sulfur hexafluoride. In addition, the emission of SO<sub>2</sub> and PM has given rise to concern.

Emission regulations are many and varied all over the world; some countries do not have national emission standards, although emission limits may be issued as guidelines by health councils, e.g. Australia. In some countries, the responsibility for emission standards is divided between the state ministry and local government, e.g. Italy and Republic of (South) Korea. Others have emission standards that are the sole responsibility of central government, as in the case of the PRC, France, Germany, and the UK. The USA involves both federal and state governments in the regulation of air pollution. Most industrialised countries follow one of these models for air quality legislation. Emission limits of SO<sub>2</sub>, NO<sub>x</sub>, and PM for new and existing power plants are given in Table 12.5. Emissions into the atmosphere may traverse individual country boundaries, and because of the international nature of air pollution this has led to a number of initiatives to control trans-boundary pollution. Agreements between countries have been made, e.g. between USA and Canada, the United Nations Economic Commission for Europe Conventions (UNECE), the EU environmental legislation, the Kyoto Protocol, the Bali Action Plan, the Copenhagen Accord, and the Paris Agreement. In addition, the World Bank environmental guidelines are regularly implemented when projects seek international finance.

### 12.6.2 United Nations Economic Commission for Europe Conventions

The UNECE Convention of Long Range Trans-Boundary Air Pollution (LRTAP) was

signed in 1979 by 33 countries, including USA and Canada. Now, 42 countries are party to the convention. The convention outlines the responsibility of governments to minimise trans-boundary air pollution and it came into force in 1983. Two protocols have been made under the LRTAP Convention dealing with sulfur emissions. The first, the 'Helsinki Protocol', signed in 1985, required sulfur emissions to be reduced by 30% on 1980 levels by 1993. Twenty-one countries agreed, and in fact achieved a 48% reduction. The Second Sulphur Protocol was signed in 1994 by 27 European countries, the then European Community, and Canada. The Second Sulphur Protocol is based on a concept of critical loads. Critical loads are quantitative estimates of pollutant deposition below which plants and ecosystems are not adversely affected. This protocol's objective was to reduce the gap between deposition and the critical load by 60%. All signatories were allocated targets to be achieved by the year 2000; however, the protocol was due to come into force only after it had been ratified by 16 countries. By 1997, only eight countries had ratified the protocol (McConville 1997).

The protocol on NO<sub>x</sub>, the 'Sofia Protocol', was signed in 1988 and came into force in 1991, signed by 23 countries and ratified by 16. The protocol required that emissions of NO<sub>x</sub> be frozen at 1987 levels by the end of 1994 and then maintained.

A protocol on the control of VOC emissions was signed in 1991 and came into force in 1997. VOCs are defined as all organic compounds of anthropogenic nature, other than CH<sub>4</sub>, that are capable of producing photochemical oxidants by reactions with NO<sub>x</sub> in the presence of sunlight (McConville 1997).

Protocols are in place on persistent organic pollutants and heavy metals. These are detailed in the Stockholm Convention on Persistent Organic Pollutants in 2004 and updated by EU Regulation 2019/1021 (European Union 2019), and in UK by the Persistent Organic Pollutants Regulations 2007. Since the introduction of these protocols, the UNFCCC has reached

agreement to reduce global temperature and to assist developing countries achieve their goals (see Section 12.6.7).

### 12.6.3 European Union

The first EU-wide environmental policy, the Environmental Action Programme, was introduced in 1972. This was a general statement of objectives and principles for a community environmental policy. Since then there have been multiple environmental action programmes. EU legislation is in the form of regulations and directives. Once regulations are approved, they are directly applicable and binding on member states. Directives establish targets to be achieved. The Industrial Emissions Directive (2010/75/EU) lays out procedures for a wide range of industrial activities; in particular, coal-fired combustion plants with a rated thermal output  $>50\text{ MW}_e$  and the medium combustion plants  $1\text{--}50\text{ MW}_e$ , regulating  $\text{SO}_2$ ,  $\text{NO}_x$ , and PM emissions. Emission limits for existing and new power plants are given in Table 12.4.

### 12.6.4 World Bank

Environmental guidelines have been developed by the World Bank that are to be followed in all the projects that the World Bank funds. These guidelines are commonly used by other financial institutions when lending to projects in developing countries. The basis of the World Bank's environmental policy is Operational Directive OD4.00, 1989, updated in 1991 as OD 4.01. These establish maximum emission levels for all fossil-fuel-based thermal power plants with a capacity of  $50\text{ MW}_e$  or larger. The guidelines focus on emissions of particulates less than  $10\ \mu\text{m}$  in size,  $\text{SO}_2$ , and  $\text{NO}_x$ .

Information on health concerns and damage caused by these pollutants together with alternative methods of emission control are provided in the guidelines. Requirements are of two kinds: first, the specific requirements for the power plant itself, focusing on issues

to be addressed in arriving at project-specific emission standards; and second, requirements that relate to the operation of the power system as a whole. These are the concern of national or regional authorities with the responsibility for setting the overall policy framework for the development of the power sector (Ackermann et al. 1999, pp. 413–421).

The International Finance Corporation's (2007) *Environmental, Health and Safety General Guidelines* are applied and tailored to the hazards and risks established for each project on the basis of the results of an environment assessment in which site-specific variables, such as host country context, assimilative capacity of the environment, and other project factors, are taken into account. Such assessment is carried out consistent with OD 4.01.

### 12.6.5 Kyoto Protocol

At the Rio Earth Summit, parties to the Framework Convention on Climate Change (FCCC) agreed to stabilise emissions of GHGs at 1990 levels by the year 2000. Following this, an agreement to cut emissions of GHGs was agreed in December 1997 in Kyoto, Japan, at the third Conference of Parties to the FCCC. Industrial nations agreed to reduce their collective emissions of GHGs by 5.2% from 1990 levels by the period 2008–2012. The Kyoto Protocol was endorsed by 160 countries, but would only be legally binding if at least 55 countries signed up to it, including developed nations responsible for at least 55% of GHG emissions from the industrialised world. The cut in emissions was to be achieved by differential reductions for individual countries. The EU, Switzerland, and the majority of central and eastern European nations were to deliver reductions of 8%, the USA 7%, and Japan, Hungary, Canada, and Poland 6%; New Zealand, Russia, and the Ukraine were required to stabilise their emissions, whilst Australia, Iceland, and Norway were permitted to increase slightly, although at a reduced rate to

current trends. The USA dropped out in 2001, as it was thought to affect the US economy.

The first commitment period of the protocol ended in 2012, and this was followed by the Doha Amendment, which added new emission reduction targets for the period 2012–2020 for participating countries.

### 12.6.6 Copenhagen Accord

The United Nations Climate Change Conference took place in Copenhagen in 2009, and included the 15th Conference of the Parties to the UNFCCC, and the Fifth Conference of the Parties serving as the Meeting of the Parties to the Kyoto Protocol (COP/MOP5).

This conference followed the meeting in Bali, Indonesia, in 2007, which included the long-term global goal for emission reductions. A two-year negotiating process was agreed, referred to as the Bali Action Plan, also known as the Bali Roadmap. The Copenhagen Accord was a document that the delegates present agreed to ‘take note of’. The accord was not legally binding and did not commit countries to agree to a binding successor to the Kyoto Protocol.

The accord reinforced the need for emissions reductions together with providing financial assistance to help developing countries cut C emissions. It was recognised that in order to prevent dangerous anthropogenic interference with the climate system, the increase in global temperature should be below 2 °C to combat climate change.

The Copenhagen Accord was not considered to be a wholly successful conference and came in for strong criticism. Although countries representing 80% of global emissions have engaged with the accord, the lack of commitment by developing countries to legally binding emission reductions has been seen as a major weakness of the accord (Akanle et al. 2009). A number of post-Copenhagen studies suggest that the 2 °C objective will be difficult to achieve. The United Nations Environment Programme suggests a possible ‘emissions gap’

between the voluntary pledges made in the accord and the emissions cuts necessary to have a chance of meeting the 2 °C objective in 2020, i.e. an emission level of  $44 \times 10^9$  tons. Without the accord emissions reductions it was thought this figure might reach  $50 \times 10^9$  tons by 2020.

It remains to be seen whether the political and public profile created in Copenhagen can be translated into a binding and ambitious international agreement on climate change.

### 12.6.7 Durban Platform for Enhanced Action

At the UNFCCC conference in Durban in 2011, attending governments committed to a comprehensive plan to deliver the ultimate objective to stabilise GHG concentrations in the atmosphere at a level that will prevent dangerous interference with the climate system and at the same time preserve the right to sustainable development.

Governments were asked to take such actions required to reduce emissions and smooth the way to an effective global climate change agreement in 2015.

### 12.6.8 Paris Agreement

In 2015, 195 parties to the UNFCCC reached an agreement to combat climate change and to accelerate and intensify the actions and investments needed for a sustainable low C future (UNFCCC 2015; Wikipedia 2017). By 2019, all parties had signed up to the agreement. The agreement came into force in November 2016, ratified by 186 countries. This was the first legally binding global climate deal. The central aim is to strengthen the global response to the threat of climate change by keeping a global temperature rise this century well below 2 °C above the pre-industrial levels, and to pursue efforts to limit the temperature increase still further to 1.5 °C. The agreement aims to increase the ability of countries to deal with the impacts of climate change. To



achieve this, provision of financial resources and a new technology framework are to be put in place to support both developing and vulnerable countries in line with their national objectives.

The agreement came into force in November 2016, ratified by 55 countries (accounting for 55% of global emissions), and this increased to 125 parties by 2017. However, in 2017, the USA announced its intention to withdraw from the Paris Agreement, and it is now hoped that the PRC will take a leading role in implementing the agreement having announced its intention to reduce emissions. The USA has since begun the withdrawal process, which is currently expected to be completed in November 2020.

The aim of the work programme that was launched in Paris in 2015, which is ongoing, is to develop procedures and guidelines on a broad array of issues, including the long-term temperature goal, global peaking of GHG emissions, and the commitment of all parties to communicate and maintain a nationally determined contribution to pursue domestic measures to achieve them. The Paris Agreement also reaffirms the obligations of developed countries to support the efforts of developing countries to build clean, climate-resilient futures, while for the first time encouraging voluntary contributions by other parties. Climate change education training, public awareness, and access to information are also to be enhanced under the agreement. Unlike the Kyoto Protocol, which set commitment targets that have legal force, the Paris Agreement allows for voluntary and nationally determined targets; these are politically encouraged rather than legally binding. Because of this, the Paris Agreement has been criticised for being based on promises and not firm commitments.

## 12.7 Future Implications

It is clear that the coal industry is being and will continue to be closely monitored as to its effects on the environment, both locally and globally.

Mining practice, waste disposal, and air quality will continue to receive close attention. In the case of coalmining, environmental regulations, particularly in relation to sulfur content, have meant the abandonment and closure of mines with high-sulfur coals, and the concentration of developing mines in areas of low-sulfur coal. This is not always possible, and this may be reflected in a penalty cost in the price obtained for high-sulfur coals. Surface and groundwater issues and waste disposal of mine waters, coal preparation discard, and spoil dumping are already closely regulated, and this will continue. In the case of air quality, the current efforts to further reduce emissions, particularly by the larger industrialised nations, is likely to be a long, protracted affair. The underlying reason is that strict environmental restrictions mean higher development cost, higher operating cost, and, in the case of old plant, higher refurbishment cost. The less that has to be spent on environmental improvement, the greater the margin for profit.

Modern coalmining practices and the advances in clean coal technology, together with the development of alternative uses of coal, such as CBM extraction and underground coal gasification, and the fact that coal resources are large and globally distributed, will ensure that coal will remain a viable source of fuel for the long-term future. It is also clear that environment legislation and implementation will continue to have a profound effect on the feasibility and cost of coalmining and utilisation, as well as its uses in industry.

## 13

### Coal Marketing

#### 13.1 Introduction

Although this book is primarily concerned with coal and its properties and uses, coal is above all a saleable commodity. The marketing of coal is no different from other commodities, in that it is controlled by supply and demand. Once the demand is established, the marketing of coal is dependent upon four factors:

- the quality and quantity of the product;
- the transportation of the product;
- the contractual terms of the sale and purchase of the product;
- the price of the product.

Coal is sought after, mined, prepared, and then transported as a saleable product. As described previously, a wide range of qualities of coal are available for a variety of uses. In order to enter the coal market, the intended use for the coal must be identified.

Coals are primarily used as coking coal in the steel industry, as thermal or steam coal in the electricity generating industry, and by industrial and domestic consumers. By-products from these processes and some specialised chemical processes are also commercial outlets for coal.

Coals are marketed locally, e.g. as mine mouth supplies to power stations, and/or neighbouring industrial complexes, at a distance but within the confines of the country of origin using transportation by road, rail and water to consumers, and as an export

product transported either by ocean going vessels to overseas markets or by rail overland as cross-border exports.

#### 13.2 Coal Quality

The chemical and physical properties of coal are described in Chapter 4, and their general usage in Chapter 12. The principal concerns for the buyer are the heating value of the coal (the calorific value [CV]) and the properties of coal that affect this, in particular the ash and moisture content, together with sulfur content for environmental reasons; and in the case of coals destined for the steel industry, the carbon content and coking properties of the coal.

In order for some coals to be suitable for selected markets, it may be necessary to improve the quality of the coal prior to shipment. This beneficiation of the coal or coal preparation is closely related to consumers' demands, but other factors, such as environmental constraints, play an increasing role in influencing the quality of coals that are marketed in the world today.

Details of coal preparation processes are well documented and described in detail in numerous works (e.g. Osborne 1988). Such preparation is normally designed to reduce ash and sulfur levels and improve the CV of the coal, and to establish a consistency in quality for the coal product. This is particularly true

for coals that will be exported and transported large distances.

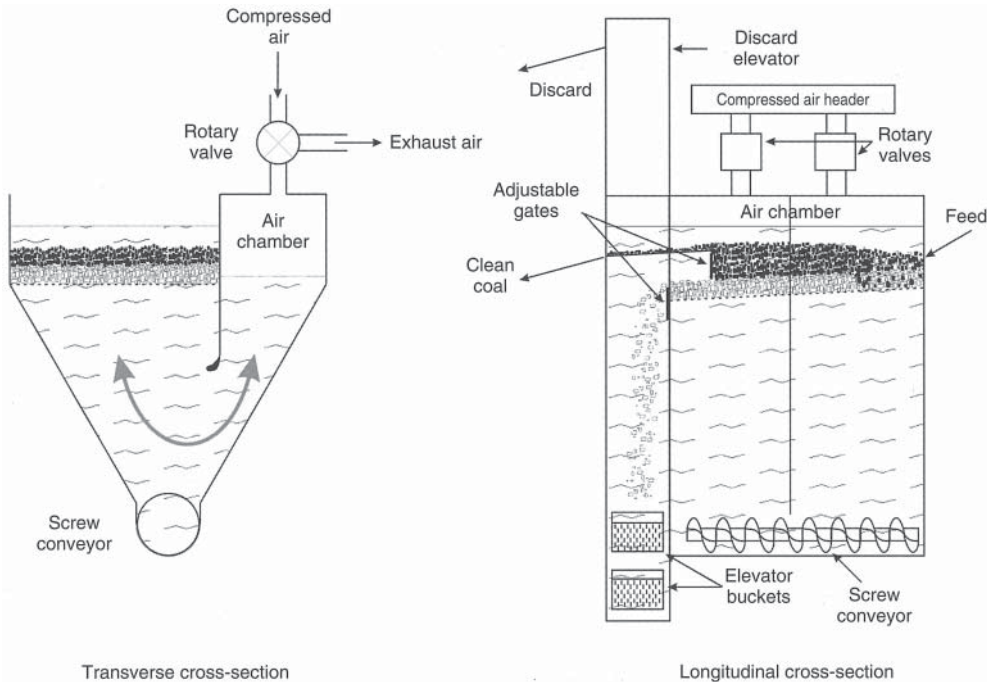
Coal preparation can simply be a screening process, whereby coal particles are separated into selected size ranges by passing through a series of screens with specified opening dimensions. However, in order to remove mineral matter from the coal, processes that separate coal particles based on their relative density are normally used. There are a number of different methods used to achieve this.

Broadly there are two types of process in general usage. Water-only-based systems are the most widely used, accounting for approximately 70% of the total tonnage treated, with the jig being the most popular. Figure 13.1 shows a section through a Baum-type jig. Coal is fed into the jig and the water therein is made to rise and fall by means of compressed air applied to one half of a U-tube. The up and down motion of the water effects a separation between the coal and discard. The other

main process is using a dense medium. In this process, coal and shale are immersed in a suspension of finely ground magnetite and water. The amount of magnetite used controls the density of the suspension, which is chosen to lie between that of coal and shale. Coal again floats and shale sinks; Section 4.3.3.5 gives an example of a typical floats and sinks analysis.

For fine coal (less than 0.5 mm in size), froth flotation is sometimes used. In this process, particles of coal, which are hydrophobic, are attached to air bubbles in water. These rise to the surface and form a froth, which is scraped off and dried. The shale stays in the water, since it is hydrophilic.

To achieve the final saleable product, it may be necessary to blend different coals together, either beneficiated coals or beneficiated and raw coals, or simply raw coals, to arrive at the required quality. Blending of coals can be carried out at the mine site or by the customer at their coal receive facilities.



**Figure 13.1** Section through a Baum-type jig. Coal is fed into the jig and the water level rises and falls by means of injection of compressed air (see transverse section). The up and down motion of the water effects a separation (see longitudinal section). *Source:* Reproduced with permission of Dargo Associates Ltd.

**Figure 13.2** Coal silos for loading coal directly to trains, the PRC. *Source:* Photograph by courtesy of Dargo Associates Ltd.



**Figure 13.3** Modern coal stockyard at port of Qinhuangdao, PRC. *Source:* Photograph by courtesy of Dargo Associates Ltd.



Many modern mines deliver coal to their dispatch area, where it is conveyed to storage silos. Each silo may contain a specific coal quality, which can then be loaded directly for transportation (Figure 13.2).

In large-scale operations, coal is stockpiled using stackers that convey coal and spread it out in layers, or in adjoining longitudinal stockpiles that form the overall stockpile. The reclaiming of blended stockpiles is by scrapers or barrel reclaimers. The coal is conveyed to hoppers for loading, or in the case of modern port facilities the coal is conveyed to the ship and loaded directly. Figure 13.3 shows the modern stockyard at Qinhuangdao, People's Republic of China (PRC), where coal stocks are reclaimed directly onto conveyors and then to ship loaders.

Coal stockpiles are monitored closely for signs of oxidation and spontaneous combustion. Coals that have a propensity for this are normally of lower rank or have a high pyrite content. As a consequence, such coals are only stockpiled for a short time.

## 13.3 Transportation

### 13.3.1 Land Transportation

Transportation of coal is a major component in the marketing of coal worldwide. The transport system must be matched to the throughput and distance of movement required. Short distances, up to 25 km, can be covered by conveyor belts, especially if the throughput is high. Longer distances may require transport

by truck or rail. The specific method chosen will depend on each location and will be a function of cost effectiveness and environmental concerns.

### 13.3.1.1 Conveyors

Overland conveyors are often used to transport coal to mine-mouth power stations, or to rail terminals for loading into special coal trains, or very occasionally direct to a ship loading terminal. An example of the latter is in East Kalimantan, Indonesia, where coal is conveyed 14 km from the Kaltim Prima Coalmine to the port at Tanjung Bara (Figure 13.4). A single-span steel-cored conventional belt conveyor carries  $1100 \text{ t h}^{-1}$  of coal to the port stockyard. Conventional belt conveyors have the disadvantage that they are best run in a straight line, but they can easily travel up to 20 km in a single stage. These conveyors run on idlers ‘troughed’ in three sections for the top or coal-conveying surface, and on flat idlers or rollers for the return or bottom belt. Another form of conveyor is the cable belt, consisting of a rubber conveying belt resting on steel cables that provide the motive force. Higher capacity or greater distance can be achieved than with a conventional belt conveyor (over 30 km), but cable belt conveyors have a higher capital and operating cost.

### 13.3.1.2 Road

Road transport is obviously very flexible and can serve any number of customers. The limitation is distance, but individual customers may be served up to 160 km by coal trucks of 10–40 t, or by heavier loads in articulated trucks and multiple trailers.

The route to be taken by trucks must be considered carefully. If the route is to use existing public roads, environmental concerns over exhaust fumes, noise, and traffic congestion may influence the choice of route. The availability of suitable vehicles will also be a factor, particularly in developing countries. India, for instance, has a ready availability of 10 t capacity road trucks (Figure 13.5), and



**Figure 13.4** Overland conveyor, 14 km in length, from Kaltim Prima Coalmine to the port of Tanjung Bara, East Kalimantan, Indonesia. *Source:* Photograph by courtesy of Dargo Associates Ltd.



**Figure 13.5** Coal transportation by small capacity 10 t truck, India. *Source:* Photograph by courtesy of Dargo Associates Ltd.

despite the intuitive conclusion that 14 t trucks would offer economies, the excessive price of the larger trucks makes them uneconomic. It is essential to load trucks as quickly as possible in order to achieve the best utilisation and reduce costs, so loading from bins (Figure 13.6) or silos is the most effective for dedicated transport





**Figure 13.6** Automatic loading of trucks from overhead bins, Orissa State, India. *Source:* Photograph by courtesy of Dargo Associates Ltd.

routes. Road transport may be used to haul coal to terminals on either rivers (e.g. in the USA and Indonesia) or to large coal terminals loading ocean-going bulk carriers. Such dedicated roads are usually limited to about 70 km in length, and throughput is about 2 Mt per year (e.g. in Venezuela and Indonesia).

### 13.3.1.3 Rail

The bulk of coal transported any long distance overland is by rail. In the extreme, this may be individual wagons to individual small customers, but this trade is diminishing. For longer distances, trains of 3000–10 000 t may be employed to carry coal in 100 t wagons using diesel or electric locomotives to export coal terminals. Coal is transported 600 km to Richards Bay in South Africa, and to Dalrymple Bay in Queensland, Australia, and in the PRC even greater distances are covered, up to 1000 km from the coalfields to the port of Qinhuangdao on the eastern seaboard.

The capacity of a rail transport system depends upon whether there is single or double track. The capacity of a double track is more than double that of a single track, the latter being constrained by the number of passing places available and the waiting time for trains travelling in the opposite direction. The speed of the train and the distance between trains are also important. Also, time is needed to negotiate busy rail junctions, and this may affect overall throughput.



**Figure 13.7** Automatic loading of trains, Alberta, Canada. Each wagon receives exact tonnage as train moves slowly through loading bay. *Source:* Howland (1998).

Large train shipments are usually loaded from fully automated train loading systems, where the train is inched through the loader at a controlled rate and the precise tonnage is released into each wagon as it passes (Figure 13.7). The trains are automatically weighed after passing through the loader, and the coal is usually sampled at this stage. In some countries, the majority of coal is still loaded with mobile equipment, such as payloaders (Figure 13.8).

Upon delivery, the coal has to be unloaded; the method of unloading is governed by



**Figure 13.8** Train being loaded by payloader, Orissa State, India. *Source:* Photograph by courtesy of Dargo Associates Ltd.



**Figure 13.9** Bottom-discharge wagons (60 t), on Eastern Railways, India. *Source:* Photograph by courtesy of Dargo Associates Ltd.



**Figure 13.10** Large train units transporting 12 000 t over 1200 km in the western USA. *Source:* Harder (1998).

the type of wagons used, i.e. they may be bottom-discharge wagons or top-discharge, tippler wagons. Bottom-discharge wagons are now the wagon of choice in most countries (Figure 13.9); they are simple to discharge and trains do not need to be uncoupled, nor do they need complex couplings. To discharge, they are simply pulled over a ground hopper and a trip mechanism opens the doors on the bottom of the wagon and the coal is discharged. Once completed, the doors are closed automatically. Very large train units transport coal in this way, notably in Canada and the USA. In the USA, such train units carry up to 12 000 t over distances of up to 1200 km from the mines for delivery for export from the Pacific Coast (Figure 13.10). Top-discharge wagons have to be inverted or tipped to empty them. Trains can be equipped with rotary couplings that permit wagons to be tipped without uncoupling. More commonly, wagons must be uncoupled



**Figure 13.11** Top-discharge wagons with drop-down doors for side unloading, Guizhou Province, PRC. *Source:* Photograph by courtesy of Dargo Associates Ltd.

and tipped separately and then recoupled. Using rotary couplings, a 60 t wagon train can be unloaded in one hour, whereas uncoupling would require three hours for the same train. Tippler wagons often have drop side doors, which enable them to be unloaded by hand (Figure 13.11).

### 13.3.2 Water Transportation

Coal is transported either by barge or by bulk carriers. Barges are defined as having no propulsion and may have a capacity up to 10 000 t. They are mainly used in sheltered waters, such as rivers, lakes, and short sea crossings. Bulk carriers are self-propelled and are usually ocean-going vessels.

#### 13.3.2.1 Barges

Barges have been used for inland water transport on canals for hundreds of years, e.g. the Grand Canal in the PRC, which runs from Shanghai to Beijing. In the UK, canals have been utilised for over 200 years, including the famous Duke of Manchester's Canal, which went underground into the coal mine for part of its length.

Barge traffic is still used in Europe on a large scale, on the Rhine and Danube rivers, and in the USA, on the Missouri and Mississippi rivers; and these barges can have capacities up to 10 000 t (Figure 13.12). Barges are also used extensively on rivers in Indonesia, such as the



**Figure 13.12** Coal barges carrying up to 10 000 t, Mississippi River, USA.

Mahakam, Barito, and Berau rivers in East Kalimantan. These barges, of 1000 t capacity, are used to take coal down the rivers to a point 2–3 km offshore where they are unloaded by floating cranes into bulk carriers. Barges are also used to transport coal on short sea crossings, such as from East Kalimantan to Java and from Sumatra to Java in Indonesia, weather permitting. Transport of coal by barge is relatively low cost, both capital and operating, but is slow.

### 13.3.2.2 Bulk Carriers

Ocean bulk carriers account for over 500 Mt per year of sea-borne trade in coal, and this continues to grow. The smallest ships are 10 000 dwt, e.g. used between the UK and Rotterdam; these increase to Handysize 20 000–37 000 dwt, Handymax 37 000–50 000 dwt, and Panamax 55 000–75 000 dwt, the largest size vessel able to negotiate the Panama Canal; Capesize vessels are >100 000 dwt, with 200 000 dwt being the largest. The larger the bulk carrier, and the longer the distance, the cheaper the freight rate per tonne-kilometre. Rates are generally very competitive between a large number of shipping companies, and rates may vary according to the availability of vessels or by competition with other commodities, such as grain and iron ore.

Loading large volumes of coal into bulk carriers requires high-capacity coal-handling systems. The major exporting coal terminals of the world can load  $10\,000\text{ t h}^{-1}$ , e.g. Richards Bay in South Africa, Dalrymple Bay



**Figure 13.13** Coal loading directly into the ship from the stockyard conveyor system, port of Qinhuangdao, PRC. *Source:* From Thomas and Frankland (1999).

in Australia, Puerto Bolivar in Colombia, and Norfolk in the USA. Major importing terminals include Krishnapatnam in India, Rotterdam in the Netherlands, and Qinhuangdao in the PRC (Figure 13.13). The throughput capacities of some of the leading coal-handling ports are listed in Table 13.1 (www.sourcewatch.org 2018). The unloading of coal is done with either grabs or continuous unloaders. Grab unloaders discharge coal into a hopper, which in turn loads coal onto conveyors for transport to storage facilities. Grab unloaders commonly unload at a rate of  $1500\text{ t h}^{-1}$ .

Continuous ship unloaders are lowered into the hold of the ship and collect coal, which is carried up a chute onto conveyors. These unloaders are less prone to dust losses and can extract more coal without help from mobile equipment than is possible with a grab unloader.

Some ships have self-discharging equipment and are referred to as ‘geared vessels’. These are ideal for discharging coal where unloading facilities are absent. The largest geared vessels are Panamax size, but this may change with a demand for geared Capesize vessels. Coals are transported around the world, and Figure 13.14 shows the long-haul sea routes taken by Capesize and Panamax vessels. Smaller Handysize vessels are usually used for shorter journeys or for ports that cannot take larger vessels, e.g. along the east coast of India.

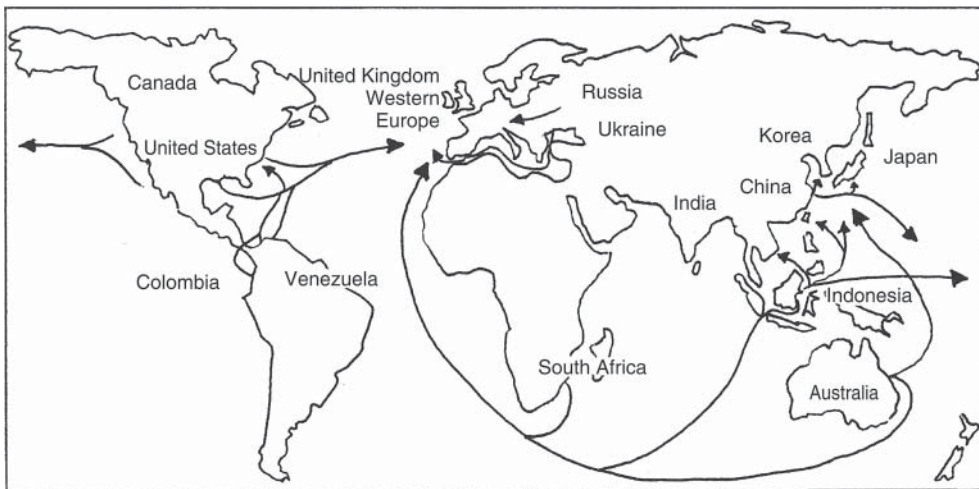
**Table 13.1** Throughput of major exporting/importing ports (www.sourcewatch.org 2018).

Port	Throughput (Mt yr <sup>-1</sup> )	Export/import
Dalrymple Bay (Australia)	85	Export
Gladstone (Australia)	75	Export
Hay Point (Australia)	55	Export
Abbott Point	50	Export
Roberts Bank (Canada)	22	Export
Qinhuangdao (PRC)	209	Import
Tianjin (PRC)	89	Import
Puerto Bolivar (Colombia)	40	Export
Krishnapatnam (India)	70	Import
Mundra (India)	60	Import
Tanjung Bara (Indonesia)	27	Export
Balikpapan (Indonesia)	15	Export
Ostochnyy (Russia)	14	Export
Murmansk (Russia)	9	Export
Richards Bay (South Africa)	91	Export
Lamberts Point (USA)	43	Export
Mobile (USA)	27	Export
Baltimore (USA)	26	Export
Norfolk (USA)	29	Export
Rotterdam (Netherlands)	29	Import

## 13.4 Coal Markets

Since 2000, global coal trade has increased by 105% (World Coal Association [WCA] 2018). The international hard (or black) coal trade totalled 1.2 Gt in 2016; of this total, the sea-borne trade comprises 833 Mt of steam coal and 271 Mt of coking coal, and 91 Mt of both coal types as cross-border trade. (HMS Bergbau Group 2017).

Table 13.2 shows the net black coal tonnage traded by the major coal exporting and importing countries (International Energy Agency [IEA] 2017). Coal is the major fuel used for generating electricity worldwide. Table 13.3 shows countries heavily dependent on coal for electricity generation (IEA 2017). Global coal-fired electricity generation in 2015 was 24 255 TWh, of which coal had 39% of the market share (Figure 13.15). This strong dependence of the electricity generators on coal will mean a long-term future for the suppliers of steam coals that have the quantity and appropriate quality of coal to satisfy the customer. The other main consumers of coal are the iron and steel industry, non-metallic minerals and chemical industries, and domestic use.

**Figure 13.14** Principal coal export routes to markets in western Europe and the Far East.



**Table 13.2** Major coal exporting and importing countries (WCA 2018).

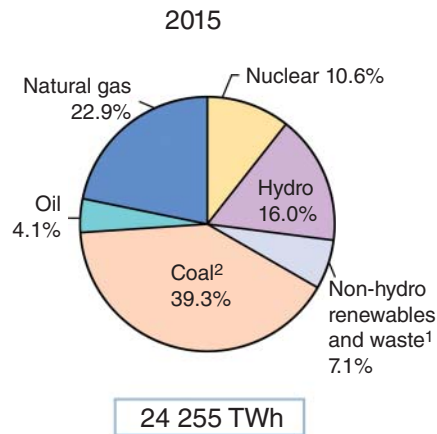
	Steam coal (Mt yr <sup>-1</sup> )	Coking coal (Mt yr <sup>-1</sup> )	Total (Mt yr <sup>-1</sup> )
<b>Coal exporters</b>			
Australia	201	188	389
Indonesia	369	1	370
Russia	149	22	171
Colombia	82	1	83
South Africa	75	1	76
<b>Coal importers</b>			
PRC	197	59	256
India	152	48	200
Japan	138	51	189
South Korea	99	35	134
Taiwan	59	7	66

**Table 13.3** Principal producers of coal-fired electricity generation 2015 (IEA 2017).

Country	TWh
PRC	4109
USA	1471
India	1042
Japan	343
Germany	284
Republic of (South) Korea	237
South Africa	229
Australia	159
Russia	159
Poland	133
Rest of the world	1372
<i>Total world</i>	9538

## 13.5 Coal Contracts

Coal contracts and sales transactions vary from straightforward individual contracts of sale to more complex indexed contracts over a long time period. In simple terms, there are three principal types of contract.

**Figure 13.15** Global electricity generation by source (IEA 2017). 1: includes industrial waste; 2: includes peat and oil shale.

### 13.5.1 Spot Purchases

This is simply the purchase of a cargo of coal offered for sale. Usually, such spot purchases are as a stop gap in coal supplies or where the purchaser is 'shopping around' for cheaper coal than they have been previously offered and is not under contract. The coal prices on the spot market will vary according to availability and changes in freight rates.

### 13.5.2 Term Contracts

This is a common form of coal contract whereby the sourcing and quality of the coal to be purchased is agreed. Such contracts usually run for one year, at which time the tonnage and price of coal to be purchased for the next year is renegotiated, with the agreed quality usually remaining the same. Term contracts are also known as perpetual contracts and run for a number of years with their annual negotiations. Such contracts are common between Japanese buyers and Australian and South African coal suppliers, and between Canada and the USA.

In negotiating successful long-term contracts, the important objective is to develop



a good relationship between coal sellers and buyers, so that when variations do occur both parties can negotiate amicably and quickly.

Coal contract terms and conditions will reflect the particular requirements of the buyer and seller. In general, however, most clauses in international coal contracts include coal type and quality, which has a significant effect on the planned use of the coal; e.g. CV for thermal coals and coking properties for coking coals, length of contract, tonnage requirements, basic coal price, escalation, bonus/penalty clauses, sampling, weighing and analysis of the cargo, payment arrangements, and currency and exchange fluctuations. The contract will also include clauses to minimise risk to either party, such as changes in law, taxes, and regulations, arbitration, force majeure, and unfair conditions (to either party). If the contract involves transportation, there will be clauses relating to the road or railway company's obligations and where responsibility lies, coal storage and handling, and/or to shipping, loading, and discharge port conditions. The latter will include port charges, demurrage, and guaranteed rates of loading.

### 13.5.3 Indexed Contracts

Indexed contracts are usually long-term contracts, such as between a coal-fired electricity generating station and a single coal supplier or group of suppliers. They are typically used where the power station is independently owned and financed by non-recourse project debt. The required tonnage and coal quality are determined by the specification of the boilers in the power station and are fixed for the duration of the contract, which can be 20 years. The contract will contain similar clauses to those for a term contract, but they may be more detailed when dealing with coal stock levels. The power plant must not break down, so a constant supply of fuel must be guaranteed. One method of dealing with a short disruption in supply is to hold a sufficient stock of fuel to operate the power plant at full load, and

clauses are included in the contract stipulating this condition. Such stockpiling of fuel does mean higher working capital requirements and likely increased cost for debt service. The failure of the coal supply, and hence failure to generate electricity, will normally be considered cause for default under the terms of the contract; therefore, it is essential that the terms of the contract must be closely linked to the conditions laid down in the power purchase agreement drawn up by the generating company and the electricity purchaser to avoid conflict between the two. Late delivery of coal may trigger penalties stipulated in the contract; in most cases, penalties under the contract are in the form of liquidated damages, which are often set so that the coal supplier has had the profit element removed but are not such that the supplier will be bankrupted. Liquidated damages set at a level of 20% of the expected coal price would be a significant incentive to the coal supplier to perform. Repeated infractions by the coal supplier may also lead to termination of the contract.

In the case of the coal supply being adjacent to the power station, local conditions and issues will decide the base price of coal together with ongoing adjustments for inflation, etc. In the case of imported coals, the power station will try to ensure that the price paid for coal is always about the current market rate in order to keep the price of electricity competitive. Provided that the specification range for the power station boilers is wide enough to accommodate a reasonable range of coals, then the power station is able to accept coal from a number of suppliers, very often from different countries, e.g. the Japanese power corporations take coal from several mines in Australia, Indonesia, and South Africa.

The price of the coal is often adjusted for quality, principally CV (as the power station is buying heat), but adjustments may occur for changes in quality that may affect power station operating costs, such as ash, moisture, sulfur, Hardgrove grindability index, and ash fusion temperature. In the case of coal imports

to a power station, the coal supplier(s) will normally have been selected by tender, sometimes open or otherwise. The original base price will be determined from the current market rate as determined by the national authority or by an independent commercial organisation, i.e. a suitable thermal coal price index.

## 13.6 Coal Price and Indexing

The basic commercial property of thermal coal is its net calorific value (NCV), and this parameter together with other properties that may affect commercial and environmental considerations will determine the price paid for the coal. For spot cargoes, this is a straight-forward transaction; and in term contracts, any changes in quality will be addressed in the regular negotiations. Actual coal prices paid are calculated using a variety of formulas. For example, a typical formula may be:

$$P_{CS} = P_B \times \frac{CV_{CS}}{CV_B}$$

where  $P_{CS}$  is the price paid for coal deliveries,  $P_B$  is the base price for coal,  $CV_{CS}$  is the NCV of coal delivered, and  $CV_B$  is the base NCV.

This is to ensure that the power station effectively buys calories or joules at a fixed price. The contract may be in terms of gross CV, in which case it is necessary to make a price adjustment for moisture content.

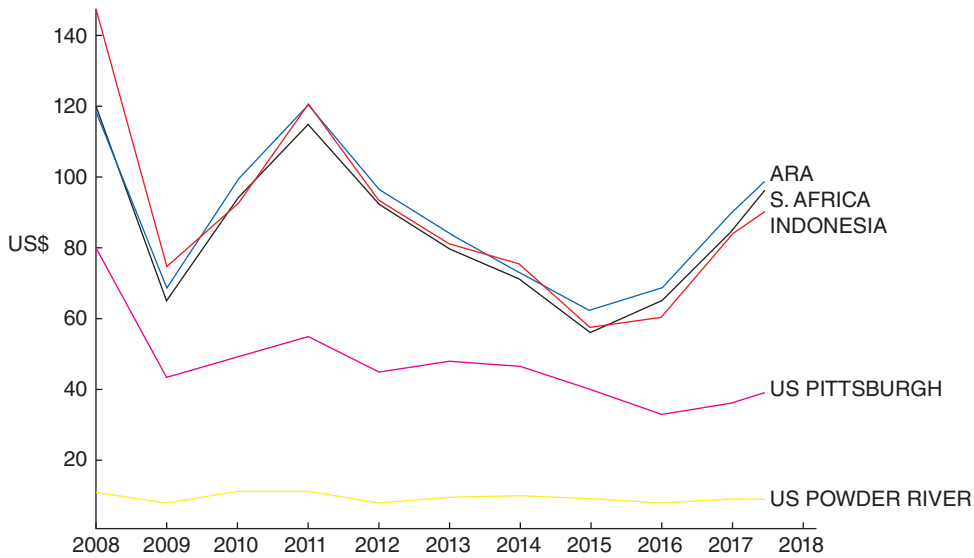
The price of coal is influenced by the world market, CV, and tonnage required, and freight and insurance rates. Cargoes may be purchased as Freight on Board (FOB) cargoes, where it is purchased when loaded at the port of embarkation; as Cost and Freight, when the cargo is purchased on arrival at the port of delivery; or as Cost, Insurance and Freight (CIF) when insurance is added. In the case of Delivered Ex-Ship, all freight and risk for loss are with the seller; most coal suppliers will not enter into this type of contract, so they are rare. Additions to these costs may be governmental charges, such as royalties, federal and local government taxes, and levies and

export/import taxes. All costs will include any freight component required to transport the coal from the mine to the port of embarkation.

The market rate for coal can either be the average price of a coal cargo from a particular country, coalfield, or coal terminal at the point of ship loading, or alternatively the average price for coal imports from any supplier at the port of entry. There are a number of sources of price data containing the current market rates that are compared with historical rates in the form of a coal price index.

Indexed contracts may include price adjustments based on one index or sometimes a 'basket' of these indices. At one time, the 'Japanese Benchmark Prices' were defined for thermal and coking coal imported into Japan by the major power corporations in consultation with Australian or American thermal coal suppliers. These were designated the market prices; however, purchasers attempted and succeeded in buying coal at a discount to the Japanese price. As a result, the Japanese Benchmark Price has now been abandoned and the Japanese are now employing a 'Fair Treatment System', which has resulted in separately negotiated prices to individual suppliers.

The Argus/McCloskey Coal Price Index Report contains price assessments for internationally traded coal on the spot market according to those specifications relating to contract basis, location, and energy value. These indexes include FOB and CIF assessments for set net as-received (NAR) CV values, for Europe, Australia, South Africa, the PRC, Colombia, and India, and also for gross as-received (GAR) CV values for Indonesia. Figure 13.16 shows the price fluctuations for examples of internationally traded thermal coal over the period 2008–2018. All of these coals follow the same trend in price changes, reflecting the large dip in coal prices in 2009 and 2015. Figure 13.16 also shows prices for some US thermal coals over the same period. These show a more consistent trend in price apart from the higher coal price that characterised 2008. These indexes are available



**Figure 13.16** Thermal coal price fluctuations 2008–2018. Amsterdam, Rotterdam, Antwerp (ARA) coal at 6000 kcal kg<sup>-1</sup> NAR monthly average USD/t CIF; South African coal at 6000 kcal kg<sup>-1</sup> NAR USD/t FOB; Indonesian coal 6500 kcal kg<sup>-1</sup> GAR USD/t FOB; US Pittsburgh Seam 6800 kcal kg<sup>-1</sup> USD/short ton FOB mine; US Powder River Basin 4600 kcal kg<sup>-1</sup> USD/short ton FOB rail. *Source:* From data provided by Argus Media Ltd (2018).

on a daily, weekly, and monthly basis ([www.argusmedia.com](http://www.argusmedia.com) 2018). S & P Global Platts also calculate indexes and make market assessments by analysing freight rates (for locational differences), quality premiums (for quality differences), movements of markets through time, and other premiums associated with the size of trader and delivery terms (S & P Global Platts 2018). Coal price indices are also produced by individual countries but are site specific and not generally used on the international market. Where indices are the average price of cargoes already sold, the indexation of prices to these indices requires a retrospective adjustment in price. Current indices include the EU based on the quarterly average CIF prices for coal imported from outside the EU and adjusted to a standard quality priced in US dollars. The CIF price, therefore, includes the cost of sea freight into Europe. The disadvantage of this index is that the data and index are

only available for publication some six months after the quarter indexed. This means that all prices for cargoes must be retrospectively adjusted, since price changes will occur each quarter.

The volatility of the various indexes is a measure of the periodicity, i.e. monthly indices vary more than indices set annually, and spot prices vary more than term contracts. It is important to select an index applicable to the particular combination of power station and supplier. It is also necessary for the power station to take into account the price paid by competing power stations when considering its own indexation. If the competing stations do not index then there may be times when they are able to purchase cheaper coal. It can be seen that, even in broad terms, the indices do not all move up or down at the same time, which can result in individual contracts paying disproportionate prices at certain times.

## Appendix A

### List of International and National Standards Used in Coal and Coke Analysis and Evaluation

The following standards are given for coal and coke as used in the UK/Europe, USA, Australia, China, India, and Russia. Other countries use these or have similar standards, and these should be referred to if requested. The list does not cover all coal standards; those covering mining, chemistry of oils, tars, etc. are not included. Latest revision dates for each standard are given where known.

British Standards are referred to BS. Figures given after the standard number indicate the year of most recent approval.

A number of the latest standards are now designated BS ISO where identical parameters are used. Some of these replace earlier versions of standards listed under BS and ISO standards. However, these are still listed as they detail the test covered.

#### A.1 British Standards Institution (BS)

Breckland, Linford Wood, Milton Keynes  
MK14 6LE.

BS ISO 540-2008	Hard coal and coke. Determination of ash fusibility
BS ISO 562-2010	Hard coal and coke. Determination of volatile matter
BS ISO 687-2010	Solid mineral fuels. Coke, determination of moisture in the general analysis test sample
BS ISO 923-2000	Coal cleaning equipment. Performance evaluation
BS ISO 1170-2008	Coal and coke. Calculation of analyses to different bases
BS ISO 1171-2010	Solid mineral fuels. Determination of ash
BS ISO 1928-2009	Solid mineral fuels. Determination of gross calorific value by the bomb calorimetric method and calculation of net calorific value
BS ISO 5068.1-2007	Brown coals and lignites. Determination of moisture content. Indirect gravimetric method for total moisture
BS ISO 5068.2-2007	Brown coals and lignites. Determination of moisture content. Indirect gravimetric method for moisture in the analysis sample
BS ISO 6127-1990	Petrographic analysis of bituminous coal and anthracite. Method of determining microlithotype, carbominerite, and minerite composition

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BS ISO 7404.2-2009	Methods for the petrographic analysis of coals. Methods of preparing coal samples
BS ISO 7402.3-2009	Methods for the petrographic analysis of coals. Method of determining maceral group composition
BS ISO 7405.5-2009	Methods for the petrographic analysis of coals. Method of determining microscopically the reflectance of vitrinite
BS ISO 11723-2004	Solid mineral fuels. Determination of arsenic and selenium. Eschka's mixture and hydride generation method
BS ISO 11726-2004	Solid mineral fuels. Guidelines for the validation of alternative methods of analysis
BS ISO 11760-2005	Classification of coals
BS ISO 13909.1-2001	Hard coal and coke. Mechanical sampling. General introduction
BS ISO 13909.2-2001	Hard coal and coke. Mechanical sampling. Coal sampling from moving streams
BS ISO 13909.3-2001	Hard coal and coke. Mechanical sampling. Sampling from stationary lots
BS ISO 13909.4-2001	Hard coal and coke. Preparation of coal test samples
BS ISO 13909.5-2001	Hard coal and coke. Mechanical sampling. Sampling coke from moving streams
BS ISO 13909.6-2001	Hard coal and coke. Mechanical sampling. Preparation of coke test samples
BS ISO 13909.7-2001	Hard coal and coke. Mechanical sampling. Methods for determining the precision of sampling, sample preparation, and testing
BS ISO 13909.8-2001	Hard coal and coke. Mechanical sampling. Methods of testing for bias
BS ISO 14180-1998	Solid mineral fuels. Guidance on sampling of coal seams
BS ISO 15237-2003	Solid mineral fuels. Determination of total mercury content of coal
BS ISO 15238-2003	Solid mineral fuels. Determination of total cadmium content of coal
BS ISO 15239-2005	Solid mineral fuels. Evaluation of the measurement performance of on-line analysers
BS ISO 18283-2006	Hard coal and coke. Manual sampling
BS ISO 19579-2006	Solid mineral fuels. Determination of sulphur by IR spectrometry
BS ISO 20904-2006	Hard coal. Sampling of slurries
BS ISO 21398-2007	Hard coal and coke. Guidance to the inspection of mechanical sampling systems
BS ISO 23380-2008	Selection methods for the determination of trace elements in coal
BS ISO 23499-2008	Coal. Determination of bulk density
BS ISO 29541-2010	Solid mineral fuels. Determination of total carbon, hydrogen, and nitrogen content. Instrumental method
BS ISO 17246-2010	Coal proximate analysis
BS 1016.1-1989	Total moisture of coal
BS 1016.8-1984	Chlorine in coal and coke
BS 1016.9-1989	Phosphorus in coal and coke
BS 1016.10-1989	Arsenic in coal and coke
BS 1016.14-1979	Analysis of coal ash and coke ash
BS 1016.21-1987	Determination of moisture holding capacity of hard coal
BS 1016.100-1994	Methods for analysis and testing of coal and coke, introduction, and methods for reporting results
BS 1016.102-2000	Determination of total moisture of coke
BS 1016.104.1-1999	Proximate analysis: determination of moisture of general analysis test sample
BS 1016.104.2-1991	Proximate analysis: determination of moisture content of general analysis sample of coke

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BS 1016.104.3-1998	Proximate analysis: determination of volatile matter content
BS 1016.104.4-1998	Proximate analysis: determination of ash content
BS 1016.105-1992	Determination of gross calorific value
BS ISO 17427-2005	Ultimate analysis of coal
BS 1016.7-1977	Ultimate analysis of coke
BS 10160.106.1.1-1996	Ultimate analysis: determination of carbon and hydrogen, high temperature combustion method
BS 1016.106.1.2-1996	Ultimate analysis: determination of carbon and hydrogen, Liebig method
BS 1016.106.2-1997	Ultimate analysis: determination of nitrogen
BS 1016.106.4.1-1993	Ultimate analysis: determination of total sulphur content, Eschka method
BS 1016.106.4.2-1996	Ultimate analysis: determination of total sulphur, high temperature combustion method
BS 1016.106.5-1996	Ultimate analysis: determination of forms of sulphur in coal
BS 1016.106.6.1-1997	Ultimate analysis: determination of chlorine content, Eschka method
BS 1016.106.7-1997	Ultimate analysis: determination of carbonate carbon content
BS 1016.107.1-1991	Caking and swelling properties of coal: determination of crucible swelling number
BS 1016.107.2-1991	Caking and swelling properties of coal: assessment of caking power by Gray-King coke test
BS 10160.107.3-1990	Caking and swelling properties of coal: determination of swelling properties using a dilatometer
BS 1016.108.1-1996	Coke tests: determination of shatter indices
BS 1016.108.2-1992	Coke tests: determination of Micum and Irsid indices
BS 1016.108.3-1995	Coke tests: determination of bulk density (small container)
BS 1016.108.4-1995	Coke tests: determination of bulk density (large container)
BS 1016.108.5-1992	Coke tests: determination of density and porosity
BS 1016.108.6-1992	Coke tests: determination of critical air blast value
BS 1016.109-1995	Size analysis of coal
BS 1016.110.1-1996	Size analysis of coke: nominal top size >20 mm
BS 1016.110.2-1996	Size analysis of coke: nominal top size 20 mm or less
BS 1016.111-1998	Determination of abrasion index of coal
BS 1016.112-1995	Determination of Hardgrove grindability index of hard coal
BS 1016.113-1995	Determination of ash fusibility
BS 1017.1-1989	Methods for sampling of coal
BS 1017.2-1994	Methods for sampling of coke
BS 3323-1992	Glossary of terms relating to sampling, testing, and analysis of solid mineral fuels
BS 3552-1994	Glossary of terms used in coal preparation
BS 5930-1999	Code of practice for site investigations
BS 6068.1 to 9-1996	Water quality glossary
BS 6127.1-1995	Petrographical analysis of bituminous coal and anthracite. Glossary of terms
BS 7022-1988	Guide for geophysical logging of boreholes for hydrological purposes
BS 7067-1990	Guide to determination and presentation of float and sink characteristics of raw coal and of products from coal preparation plants
BS 7763-1994	Method of evaluation of the performance of coal sizing equipment

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## A.2 International Organization for Standardization (ISO)

BIBC II Chemin de Blandonnet 8 CP401, 1214  
Vernier, Geneva, Switzerland

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ISO 157-1996	Hard coal – determination of forms of sulfur
ISO 331-1993	Coal – determination of moisture in the analysis sample, direct gravimetric method
ISO 332-1996	Coal – determination of nitrogen, macro Kjeldahl method
ISO 333-1997	Coal – determination of nitrogen, semi-micro Kjeldahl method
ISO 334-2013	Coal and coke – determination of total sulfur, Eschka method
ISO 335-1974	Hard coal – determination of caking power, Roga test
ISO 348-1981	Hard coal – determination of moisture in the analysis sample, direct volumetric method
ISO 349-1975	Hard coal – Audibert–Arnu dilatometer test
ISO 351-1996	Solid mineral fuels – determination of total sulfur, high temperature combustion method
ISO 352-1981	Solid mineral fuels – determination of chlorine by high temperature combustion method
ISO 501-2012	Coal – determination of the crucible swelling number
ISO 502-2015	Coal – determination of caking power, Gray–King coke test
ISO 540-2008	Solid mineral fuels – determination of fusibility of ash, high temperature tube method
ISO 561-1989	Coal preparation plant – graphical symbols
ISO 562-2010	Hard coal and coke – determination of volatile matter content
ISO 567-1995	Determination of bulk density of coke (small container)
ISO 579-2013	Determination of total moisture of coke
ISO 587-1997	Solid mineral fuels – determination of chlorine using Eschka method
ISO 589-2008	Hard coal – determination of total moisture
ISO 601-1981	Solid mineral fuels – determination of arsenic content using the standard silver diethyldithio-carbamate photometric method of ISO 2590
ISO 602-2015	Coal – determination of mineral matter
ISO 609-1996	Coal and coke – determination of carbon and hydrogen (high temperature combustion method)
ISO 616-1995	Determination of coke shatter indices
ISO 622-2016	Solid mineral fuels – determination of phosphorus content, reduced molybdophosphate photometric method
ISO 625-1975	Coal and coke – determination of carbon and hydrogen, Leibig method
ISO 647-2017	Brown coals and lignites – determination of the yields of tar, water, gas and coke residue by low temperature distillation
ISO 728-1995	Size analysis of coke, nominal top size >20 mm
ISO 923-2000	Coal cleaning equipment – performance evaluation
ISO 924-1989	Coal preparation plant – principles and conventions for flowsheets
ISO 925-1997	Solid mineral fuels – determination of carbon dioxide content, gravimetric method
ISO 975-1985	Brown coals and lignites – determination of yield of toluene-soluble extract
ISO 1013-1995	Determination of bulk density of coke (large container)

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ISO 1015-1992	Brown coals and lignites – determination of moisture content, direct volumetric method
ISO 1017-2006	Brown coals and lignites – determination of acetone-soluble material (resinous substances) in the toluene-soluble extract
ISO 1018-1975	Hard coal – determination of moisture holding capacity
ISO 1170-2013	Coal and coke – calculation of analyses to different bases
ISO 1171-2010	Solid mineral fuels – determination of ash content
ISO 1213-1993	Part 1, vocabulary of terms relating to solid mineral fuels, Part 2, terms relating to coal sampling and analysis
ISO 1928-2009	Solid mineral fuels – determination of gross calorific value by the bomb calorimeter method and calculation of net calorific value
ISO 1952-1976	Brown coals and lignites – method of extraction for the determination of sodium and potassium in dilute hydrochloric acid
ISO 1953-2015	Hard coals – size analysis
ISO 1988-1975	Hard coal – sampling
ISO 1994-1976	Hard coal – determination of oxygen content
ISO 2325-1986	Size analysis of coke, nominal top size 20 mm or less
ISO 2950-1974	Brown coals and lignites – classification by types on the basis of total moisture content and tar yield
ISO 5068-2007	Brown coals and lignites – determination of moisture content, indirect gravimetric method
ISO 5069-1983	Brown coals and lignites – principles of sampling: Part 1, sampling for determination of moisture content and for general analysis Part 2, sample preparation for determination of moisture content and for general analysis
ISO 5071-1-2013	Brown coals and lignites – determination of the volatile matter in the analysis sample. Part 1. Two furnace method
ISO 5073-2013	Brown coal and lignite – determination of humic acids
ISO 5074-2015	Hard coal – determination of Hardgrove grindability index
ISO 7404.4-2016	Methods for the petrographic analysis of bituminous coal and anthracite
ISO 7404.2-2009	Method of preparation of coal samples
ISO 7404.3-2009	Method of determining maceral group composition
ISO 7404.4-2017	Method of determining microlithotype, carbominerite and minerite composition
ISO 7404.5-2009	Method of determining microscopically the reflectance of vitrinite
ISO 7936-1992	Hard coal: determination and presentation of float and sink characteristics – apparatus and procedures
ISO 8264-1989	Hard coal – determination of the swelling properties using a dilatometer
ISO 8833-1989	Magnetite for use in coal preparation – test methods
ISO 8858.1-1990	Hard coal – froth flotation testing: Part 1. Laboratory procedure
ISO 9411-1994	Solid mineral fuels. Mechanical sampling from moving streams. Part 1. Coal
ISO 10086.1-2019	Coal: methods for evaluating flocculants for use in coal preparation – Part 1: basic parameters

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ISO 10329-2017	Coal – determination of plastic properties – Constant-torque Gieseler plastometer method
ISO 10752-1994	Coal sizing equipment – performance evaluation
ISO 10753-1994	Coal preparation plant – assessment of liability to breakdown in water of minerals associated with coal seams
ISO 11722-2013	Hard coal: determination of moisture by drying in nitrogen
ISO 11724-2016	Solid mineral fuels – determination of fluorine in coal, coke and fly ash
ISO 11726-2017	Solid mineral fuels – guidelines for the validation of alternative methods of analysis
ISO 11760-2005	Classification of coals
ISO 12900-2015	Hard coal: determination of abrasiveness
ISO 13605-2018	Solid mineral fuels. Major and minor elements in coal ash and coke ash. Wavelength dispersive x-ray fluorescence spectrometric method
ISO 13909.1-2014	Hard coal and coke – mechanical sampling – introduction
ISO 13909.2-2016	Hard coal and coke – mechanical sampling – coal: sampling from moving streams
ISO 13909.3-2016	Hard coal and coke – mechanical sampling – coal: sampling from stationary lots
ISO 13909.4-2016	Hard coal and coke – mechanical sampling – coal: preparation of test samples
ISO 13909.5-2016	Hard coal and coke – mechanical sampling – coke: sampling from moving streams
ISO 13909.6-2016	Hard coal and coke – mechanical sampling – coke: sampling from stationary lots
ISO 13909.7-2016	Hard coal and coke – mechanical sampling – methods for determining the precision of sampling, sample preparation, and testing
ISO 13909.8-2001	Hard coal and coke – mechanical sampling – methods of testing for bias
ISO 14180-1998	Guidance on the sampling of coal seams
ISO 15237-2016	Solid mineral fuels. Determination of total mercury content of coal
ISO 15238-2016	Solid mineral fuels. Determination of total cadmium in coal
ISO 15585-2006	Hard coal – determination of caking index
ISO 17246-2010	Proximate analysis of coal
ISO 17247-2013	Ultimate analysis of coal
ISO 18806-2014	Solid mineral fuels. Determination of chlorine content
ISO 18283-2006	Hard coal and coke. Manual sampling
ISO 18871-2015	Method for determining coalbed methane content
ISO 20336-2017	Solid mineral fuels. Determination of total sulfur by Coulomb titration method
ISO 23873-2010	Hard coal. Method for the measurement of the swelling of hard coal using a dilatometer
ISO 23380-2013	Selection of methods for the determination of trace elements in coal
ISO 29541-2010	Solid mineral fuels. Determination of carbon, hydrogen and nitrogen content – instrumental method

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### A.3 ASTM International, Formerly Known as American Society for Testing and Materials (ASTM)

Figures after standard number give the year of most recent re-approval.

100 Barr Harbor Drive, Conshohocken, Pennsylvania, USA

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D 121-2015	Definitions of terms relating to coal and coke
D 197-2002	Sampling and fineness test of pulverized coal
D 271-1970	Standard method of laboratory sampling and analysis of coal and coke
D 291-2012	Cubic foot weight of crushed bituminous coal
D 293-2004	Sieve analysis of coke
D 346-2004	Collection and preparation of coke samples for laboratory analysis
D 388-2017	Classification of coals by rank
D 389-1937	Specification for classification of coals by grade
D 409/ D409M-2016	Grindability of coal by the Hardgrove-machine method
D 410-1984	Method for sieve analysis of coal
D 440-2007	Drop shatter test for coal
D 441-2012	Tumbler test for coal
D 720-2015	Free swelling index of coal
D1412-2017	Equilibrium moisture of coal at 96–97% relative humidity and 30 °C
D1756-2002	Carbon dioxide in coal
D1757-2003	Sulfur in ash from coal and coke
D1812-1969	Plastic properties of coal by Gieseler plastometer
D1857-2017	Fusibility of coal and coke ash
D2013-2012	Samples, coal, preparing for analysis
D2014-1997	Expansion or contraction of coal by the sole-heated oven
D2015-2000	Gross calorific value of coal and coke by the adiabatic bomb calorimeter
D2234/ D2234M-2017	Collection of a gross sample of coal
D2361-2002	Chlorine in coal
D2492-2012	Forms of sulfur in coal
D2639-2016	Plastic properties of coal by the constant-torque Gieseler plastometer
D2796-1994	Terminology relating to megascopic description of coal and coal seams and microscopical description and analysis of coal
D2797-2007	Preparing coal samples for microscopical analysis by reflected light
D2798-2011	Microscopical determination of the reflectance of the organic components in a polished specimen of coal
D2799-2013	Microscopical determination of the maceral composition of coal
D2961-2017	Total moisture, <15% in coal reduced to No. 8 (2.36 mm) topsize
D3038-2004	Drop shatter test for coke

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D3172-2013	Proximate analysis of coal and coke
D3173-2017	Moisture in the analysis sample of coal and coke
D3174-2012	Ash in the analysis sample of coal and coke from coal
D3175-2017	Volatile matter in the analysis sample of coal and coke
D3176-2015	Ultimate analysis of coal and coke
D3177-2002	Total sulfur in the analysis sample of coal and coke
D3178-1997	Carbon and hydrogen in the analysis sample of coal and coke
D3179-2002	Nitrogen in the analysis sample of coal and coke
D3180-2015	Calculating coal and coke analyses from as-determined to different bases
D3286-1996	Gross calorific value of coal and coke by isoperibol calorimeter
D3302/ D3302M-2017	Total moisture in coal
D3402-2016	Tumbler test for coke
D3682-2013	Major and minor elements in coal and coke ash by atomic absorption
D3683-2011	Trace elements in coal and coke ash by atomic absorption
D3684-2006	Total mercury in coal by the oxygen bomb combustion/atomic absorption method
D3761-2010	Total fluorine in coal by the oxygen bomb combustion/ion selective electrode method
D4182-1997	Evaluation of laboratories using ASTM procedures in the sampling and analysis of coal and coke
D4208-2013	Total chlorine in coal by the oxygen bomb combustion/ion selective electrode method
D4239-2017	Sulfur in the analysis sample of coal and coke using high temperature tube furnace combustion methods
D4326-2013	Major and minor elements in coal by x-ray fluorescence
D4371-2012	Washability characteristics of coal
D4596-2015	Collection of channel samples of coal in the mine
D4606-2015	Determination of arsenic and selenium in coal by the hydride generation/atomic absorption method
D4621-1999	Accountability and quality control in the coal analysis laboratory
D4702-2007	Guide for quality management of mechanical sampling systems
D4749-2002	Sieve analysis for coal, performing and designating coal size
D4915-1996	Practice for sampling of coal from tops of railroad cars
D4916-1997	Practice for mechanical auger sampling
D5016-2016	Sulfur in ash from coal and coke using high temperature tube furnace combustion method with infrared absorption
D5061-2016	Microscopical determination of vol% of textural components in metallurgical coke
D5114-1998	Method for laboratory froth flotation of coal in a mechanical cell
D5142-2009	Method for proximate analysis of the analysis sample of coal and coke by instrumental procedures

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D5192-2015	Practice for collection of coal samples from core
D5263-2008	Method for determining the relative degree of oxidation in bituminous coal by alkali extraction
D5341-2017	Method for measuring coke reactivity index (CRI) and coke strength after reaction (CSR)
D5373-2016	Method for instrumental determination of carbon, hydrogen and nitrogen in laboratory samples of coal and coke
D5515-2017	Method for determination of the swelling properties of bituminous coal using a dilatometer
D5671-2001	Practice for polishing and etching coal samples for microscopical analysis by reflected light
D5865-2019	Standard test method for gross calorific value of coal and coke
D5987-2002	Method for total fluorine in coal and coke by pyrohydrolytic extraction and ion selective electrode or ion chromatograph methods
D6315-1998	Practice for manual sampling of coal from tops of barges
D6316-2009	Method for determination of total, combustible and carbonate carbon in solid residues from coal and coke
D6347/ D6347M-2005	Method for determination of bulk density coal using nuclear backscatter depth density methods
D6349-2013	Method for determination of major and minor elements in coal, coke and solid residues from combustion of coal and coke by inductively coupled plasma-atomic emission spectrometry
D6357-2011	Methods for determination of trace elements in coal and coke
D6414-1999	Methods for determination of total mercury in coal and coke combustion residues by acid extraction or wet oxidation/cold vapor atomic absorption
D6518-2007	Practice for bias testing a mechanical coal sampling system
D6542-2005	Practice for tonnage calculation of coal in a stockpile
D6543-2017	Guide to the evaluation of measurements made by on-line coal analyzers
D6609-2017	Guide for part-stream sampling of coal
D6610-2001	Standard practice for manual sampling coal from surfaces of stockpiles
D6883-2017	Standard practice for manual sampling of stationary coal from railroad cars, barges, trucks, or stockpiles
D7256/ D7256M-2008	Standard practice for mechanical collection and within system preparation of a gross sample of coal from moving streams
D7430-2016	Standard practice for mechanical sampling of coal
D7533-2009	Minimum geospatial data for abandoned mine land problem area
D7582-2015	Standard test methods for proximate analysis of coal and coke by mass thermogravimetric analysis
D7609-2010	Minimum geospatial data for an abandoned mine land project site
D7610-2010	Minimum geospatial data for abandoned mine land, keyword feature

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## A.4 Standards Association of Australia (AS)

80–86 Arthur Street, North Sydney, NSW, 2060,  
Australia.

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AS 2418-1995	Glossary of terms relating to solid mineral fuels
AS 2418.3-1982	Terms relating to brown coal
AS 2418.4-1982	Terms relating to sampling, sample preparation, analysis, testing and statistics
AS 2519-1993	Guide to the evaluation of hard coal deposits using borehole techniques
AS 2617-1996	Guide for the taking of samples from hard coal seams in situ
1038.1-2001	Total moisture in hard coal
1038.2-2006	Total moisture in coke
1038.3-2000	Proximate analysis of higher rank coal
1038.4-2006	Proximate analysis of coke
1038.5-1998	Gross specific energy of coal and coke
1038.5.1-1988	Adiabatic calorimeters
1038.5.2-1989	Automatic isothermal-type calorimeters
1038.6.1-2013	Determination of carbon and hydrogen
1038.6.2-2007	Determination of nitrogen
1038.6.3.1-2013	Determination of total sulfur (Eschka method)
1038.6.3.2-2013	Determination of total sulfur (high temperature combustion method)
1038.6.3.3-2013	Determination of total sulfur (infrared method)
1038.6.4-2005	Higher rank coal and coke. Ultimate analysis –carbon, nitrogen and hydrogen – instrumental method
1038.8.1-2013	Chlorine in coal and coke (Eschka method)
1038.8.2-2013	Chlorine in coal and coke (high temperature combustion method)
1038.9.1-2013	Phosphorus in coal and coke (ash digestion/molybdenum blue method)
1038.9.2-2013	Phosphorus in coal and coke (coal extraction/phosphomolybdovanadate method)
1038.9.3-2013	Phosphorus in coal and coke (ash digestion/phosphomolybdovanadate method)
1038.9.4-2006	Phosphorus in high rank coal (borate fusion/molybdenum blue method)
1038.10.1-2013	Determination of trace elements – determination of 11 trace elements in coal, coke and fly ash by flame absorption spectrometric method
1038.10.2-2013	Determination of arsenic and selenium in coal and coke by hydride generation method
1038.10.3-2013	Determination of trace elements – coal, coke and fly ash-determination of boron content – spectrophotometric method
1038.10.4-2013	Determination of trace elements – coal, coke, and fly ash-determination of fluorine content by pyrohydrolysis method
1038.10.5.1-2013	Determination of trace elements in coal, coke and fly-ash, determination of mercury content by tube combustion method
1038.10.5.2-2007	Determination of trace elements in coal, coke and fly ash, determination of mercury content by acid extraction method
1038.11-2013	Forms of sulfur in coal

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1038.12.1-2002	Determination of crucible swelling number of coal
1038.12.2-2013	Carbonization properties of higher rank coal, determination of Gray–King coke type
1038.12.3-2002	Determination of the dilatometer characteristics of higher rank coal
1038.12.4.1-1996	Plastic properties of higher rank coal by the Gieseler plastometer
1038.13-2013	Tests specific to coke
1038.14.1-2013	Analysis of coal ash, coke ash and mineral matter (borate fusion/flame atomic absorption spectrometric method)
1038.14.2-2013	Analysis of higher rank coal ash and coke ash (acid digestion/flame atomic absorption spectrometric method)
1038.14.3-2013	Analysis of higher rank coal ash and coke ash (wavelength dispersive x-ray fluorescence spectrometric method)
1038.15-1995	Fusibility of higher rank coal ash and coke ash
1038.16-2005	Coal and coke – assessment and reporting of results
1038.17-2013	Determination of moisture-holding capacity (equilibrium moisture) of higher rank coal
1038.18-2006	Coke – size analysis
1038.19-2013	Determination of the abrasion index of higher rank coal
1038.20-2013	Hardgrove grindability index of higher rank coal
1038.21.1.1-2008	Determination of the relative density of hard coal and coke, analysis sample – density bottle method
1038.21.1.2-2013	Determination of the relative density of hard coal and coke, analysis sample – volumetric method
1038.22-2013	Direct determination of mineral matter and water of hydration of minerals in hard coal
1038.23-2013	Determination of carbonate carbon in higher rank coal
1038.24-2013	Guide to the evaluation of measurements made by on-line coal analyzers
1038.25-2013	Durham cone handleability test
1038.26-2005	Guide for the determination of apparent relative density
AS 2434	Methods for the analysis and testing of lower rank coal and its chars
2434.1-2013	Determination of the total moisture content of lower rank coal
2434.2-2013	Determination of the volatile matter in low rank coal
2434.3-2013	Determination of the moisture-holding capacity of lower rank coals
2434.4-2013	Determination of the apparent density of dried lower rank coal and its chars (mercury displacement method)
2434.5-2013	Determination of moisture in bulk samples and in analysis samples of char from lower rank coal
2434.6.1-2013	Ultimate analysis of lower rank coal
2434.7-2013	Determination of moisture in the analysis sample of lower rank coal
2434.8-2013	Determination of ash in the analysis sample of lower rank coal
2434.9-2013	Determination of four acid-extractable inorganic ions in lower rank coal
AS 2856.1-2013	Coal petrography – preparation of samples for incident light microscopy
2856.2-2013	Maceral analysis
2856.3-2013	Microscopical determination of reflectance of coal macerals
AS 3880-1991	Bin flow properties of coal

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AS 3881-2002	Higher rank coal – size analysis
AS 3899-2013	Higher rank coal and coke – bulk density
AS 3980-2013	Guide to the determination of desorbable gas content of coal seams – direct method
AS 2096-1987	Classification and coding systems for Australian coals
AS 2916-2007	Symbols for graphical representation of coal seams and associated strata
AS4156.1-2013	Coal preparation of higher rank coal, float and sink testing
4156.2.1-2004	Coal preparation of higher rank coal, froth flotation – basic test
4156.2.2-2013	Coal preparation of higher rank coal, froth flotation – sequential procedure
4156.3-2008	Coal preparation of higher rank coal, magnetite for coal preparation plant use – test methods
4156.4-1999	Coal preparation – flowsheets and symbols
4156.6-2000	Coal preparation – determination of dust/moisture relationship for coal
4156.7-2013	Coal preparation – coal size classifying equipment – performance evaluation
4156.8-2007	Coal preparation – drop shatter test
AS4264.1-2009	Coal and coke – sampling of higher rank coal – sampling procedures
4264.2-1996	Coal and coke – sampling of coke – sampling procedures
4264.3-1996	Coal and coke – sampling of lower rank coal – sampling procedures
4264.4-1996	Coal and coke – sampling – determination of precision and bias
4264.5-1999	Coal and coke – sampling – guide to the inspection of mechanical sampling systems

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## A.5 National Standards of People's Republic of China

Figures after standard give the year of most recent re-approval

Standardization Administration of China (SAC), Beijing

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GB/T 189-1997	Classification standards for size fractions of coal
GB/T 211-2017	Determination of total moisture in coal
GB/T 212-1996	Proximate analysis of coal
GB/T 212-1996	Determination of total moisture in coal
GB/T 212-2008	Determination of inherent moisture in coal
GB/T 212-2008	Determination of ash content in coal
GB/T 212-2008	Determination of volatile matter in coal
GB/T 213-2008	Determination of calorific value in coal
GB/T 214-2007	Determination of total sulfur in coal
GB/T 215-2003	Determination of forms of sulfur in coal
GB/T 216-2003	Determination of phosphorus in coal
GB/T 217-2008	Determination of true relative density of coal

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GB/T 218-2016	Determination of carbon dioxide content in mineral carbonate associated with coal
GB/T 219-2008	Determination of ash fusion temperature in coal
GB/T 258-2012	Determination of free silica in coal
GB/T 397-2009	Technical condition of coal for metallurgical coke
GB/T 474-2008	Preparation of coal samples
GB/T 475-2008	Sampling for commercial coal
GB/T 476-2008	Determination of carbon and hydrogen in coal
GB/T 477-2008	Method for size analysis in coal
GB/T 478-2008	Method for float and sink analysis of coal
GB/T 479-2016	Determination of plastometric indices of bituminous coals
GB/T 481-1993	Sampling method of coal sample for production
GB/T 482-2008	Sampling of coal in seam
GB/T 483-1998	General rules for analytical and testing methods of coal
GB/T 1341-2007	Gray-King assay for coal
GB/T 1573-2001	Determination of thermal stability of coal
GB/T 1574-1995	Analysis of coal ash
GB/T 1575-2001	Determination of yield of benzene-soluble extract in brown coal
GB/T 2001-1991	Determination of moisture in coke
GB/T 2001-1991	Determination of ash in coke
GB/T 2001-1991	Determination of volatile matter in coke
GB/T 2087-2008	Determination of chlorine in coal. High performance liquid chromatography
GB/T 2565-2014	Determination of Hardgrove grindability index in coal
GB/T 2566-2010	Determination of transmittance for low rank coal
GB/T 2696-2010	Determination of major and minor elements in coal and coke ash. X-ray fluorescence spectrometric method
GB/T 2697-2010	Determination of sulfur, phosphorous, arsenic and chlorine in coal for import and export. X-ray fluorescence spectrometric method
GB/T 3058-2008	Determination of arsenic in coal
GB/T 3521-2013	Simultaneous determination of arsenic and mercury content in coal. Hydride generation-atomic fluorescence spectrometry
GB/T 3558-2014	Determination of chlorine in coke
GB/T 3596-2013	Test method for total fluorine in coal by the oxygen bomb combustion-ion selective electrode method
GB/T 3715-2007	Terms relating to properties and analysis of coal
GB/T 4632-2008	Determination of moisture holding capacity of coal
GB/T 4633-2014	Determination of fluorine in coal
GB/T 4634-1996	Determination of ash content in coal
GB/T 5447-2014	Determination of caking index of bituminous coals
GB/T 5448-1997	Determination of fusibility of ash in coal
GB/T 5448-1997	Caking index of coal
GB/T 5448-2014	Determination of Roga Index of bituminous coal

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GB/T 5448-2014	Determination of free swelling index (FSI)/crucible swelling number (CSN) in coal
GB/T 5449-2015	Determination of Roga index of bituminous coal
GB/T 5450-2014	Audibert–Arnu dilatometer test of bituminous coal
GB/T 5751-2009	China coal classification
GB/T 6948-1998	Microscopic determination of the reflectance of vitrinite in coal
GB/T 6949-2010	Determination of apparent relative density of coal
GB/T 7186-2008	Terms relating to coal preparation of coal
GB/T 7560-2001	Determination of mineral matter in coal
GB/T 8899-2013	Determination of maceral group composition and minerals in coal
GB/T 9649.17-2009	Terminology of classification of codes of geology and mineral resources – coal geology
GB/T 11957-2001	Determination of yield of humic acids in coal
GB/T 12937-2008	Terms relating to coal petrology
GB/T 14181-2010	Specification of anthracite for determination of caking index of bituminous coal
GB/T 15224.1-2010	Classification for ash yield of coal
GB/T 15224.2-2010	Classification for sulfur content of coal
GB/T 15224.3-2010	Classification for calorific value of coal
GB/T 15334-1994	Determination of moisture in coal – microwave drying method
GB/T 15458-2006	Determination of abrasion index of coal
GB/T 15459-2006	Determination of shatter strength of coal
GB/T 15460-2003	Determination of carbon and hydrogen in coal – colorimetric and gravimetric method
GB/T 15588-2013	Classification of macerals for bituminous coal
GB/T 15591-1995	Method of reflectance of commercial coal
GB/T 15663.1-1995	Terms relating to coal geology and prospecting
GB/T 16416-1996	Method of extraction for determination of sodium and potassium in brown coal soluble in dilute hydrochloric acid
GB/T 16417-2011	Method of evaluating the washability of coal
GB/T 16658-2007	Determination of chromium, cadmium and lead in coal
GB/T 16659-2008	Determination of mercury in coal
GB/T 16772-1997	Codification systems for Chinese coals
GB/T 16773-1997	Method of preparing coal samples for coal petrographic analysis
GB/T 17607-1998	Classification of in-seam coals
GB/T 17766-1999	Classification of resources/reserves of solid fuels and mineral commodities
GB/T 18023-2000	Classification of microlithotypes for bituminous coal
GB/T 18510-2001	Guidelines for the validation of alternative methods of analysis of coal and coke
GB/T 18511-2001	Determination of ignition temperature of coal
GB/T 19092-2003	Methods of fine coal float and sink analysis
GB/T 19224-2003	Determination of the relative degree of oxidation in bituminous coal
GB/T 19225-2003	Ash analysis in coal
GB/T 19227-2008	Determination of nitrogen in coal and coke – semi-microgasification method

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GB/T 19494.1-2004	Mechanical sampling of coal – Part 1. Method for sampling
GB/T 19494.2-2004	Mechanical sampling of coal – Part 2. Method of sample preparation
GB/T 19559-2004	Method of determination of coalbed methane content in coal
GB/T 23250-2009	Direct method for determining coalbed gas content in the mine
GB/T 25213-2010	Determination of plastic properties of coal – constant torque Gieseler plastometer method
GB/T 25214-2010	Determination of total sulfur in coal by IR spectrometry
GB/T 28753-2012	Determination of coalbed gas content-heating desorption
GB/T 28754-2012	Guidance for utilization of coalbed methane
GB/T 30047-2013	Method for evaluating the floatability of fine coal
GB/T 30049-2013	Test method for washability of coal core samples
GB/T 30732-2014	Proximate analysis of coal by instrumental method
GB/T 30733-2014	Determination of total carbon, hydrogen and nitrogen content in coal – instrumental method
GB/T 31391-2015	Ultimate analysis of coal
GB/T 31537-2015	Terms relating to coalbed methane

## A.6 Bureau of Indian Standards

Figures after standard give the year of most recent re-approval

Manak Bhavan, 9 Bahadur Shah Zafar Marg,  
New Delhi 110002, India

IS436(Part1/Sec1)-2001	Methods for sampling of coal and coke: part 1. Sampling of coal/Section 1: manual sampling (revised)
IS436(Part1/Sec2)-2000	Method for sampling of coal and coke: part 1. Sampling of coal/Section 2: mechanical sampling
IS436(Part 2)-2000	Methods of sampling of coal and coke: part 2: sampling of coke (revised)
IS437-2001	Size analysis of coal and coke for marketing (third revision)
IS439-2000	Industrial coke (third revision)
IS770-2001	Classification and codification of Indian coals and lignites (second revision)
IS1350 (Part 1)-2006	Methods of test for coal and coke: part 1. Proximate analysis (second revision)
IS1350(Part 2)-2000	Methods of test for coal and coke: part 2. Determination of calorific value (first revision)
IS1350(Part 3)-2000	Methods of test for coal and coke: part 3. Determination of sulphur (first revision)
IS1350(Part 4/Sec 1)-2000	Methods of test for coal and coke: Part 4. Ultimate analysis Section 1 – determination of carbon and hydrogen (first revision)

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IS1350(Part 4/Sec 2)-2000	Methods of test for coal and coke: part 4. Ultimate analysis Section 2 – determination of nitrogen (first revision)
IS1350(Part 5)-2001	Methods of test for coal and coke: part 5. Special impurities (first revision)
IS1353-2000	Methods of test for coal carbonization – caking index, swelling number and (LT) Gray–King assay (first revision)
IS1354-2000	Methods of test for coke special test (second revision)
IS1355-2001	Methods of determination of chemical composition of ash of coal and coke (first revision)
IS3746-2000	Graphical symbols for coal preparation plant (first revision)
IS3810(Part 1)-2002	Solid mineral fuels – vocabulary: part 1 – terms relating to coal preparation (second revision)
IS3810(Part 2)-2003	Solid mineral fuels – vocabulary: part 2 – terms relating to sample testing and analysis (first revision)
IS3810(Part 3)-2000	Glossary of terms relating to solid mineral fuels: part 3 coke
IS4023-2001	Methods for the determination of reactivity of coke (first revision)
IS4286-2000	Domestic coke (first revision)
IS4311-2000	Method for determination of mineral matter in coal
IS4433-2000	Method for the determination of the Hardgrove grindability index of coal (first revision)
IS5062(Part 1)-2000	Methods of test for brown coals and lignites: part 1. Determination of moisture content by the direct volumetric method
IS5062(Part 2)-2000	Methods of test for brown coals and lignites: part 2. Determination of ash
IS5062(Part 3)-2002	Methods of test for brown coals and lignites: part 3. Determination of the yields of tar, water, gas, and coke by low temperature distillation
IS5062(Part 4)-2004	Methods of test for brown coals and lignites: part 4. Determination of yield of benzene-soluble extract – semi automatic method
IS5062(Part 5)-2004	Methods of test for brown coals and lignites: part 5. Determination of acetone-soluble material (resinous substances) in the toluene-soluble extract
IS5209-2002	Coal preparation plant – principles and conventions for flow sheets (first revision)
IS6345-2001	Methods of sampling of coal for float and sink analysis (first revision)
IS7190(Part 1)-2004	Coke – methods of test: part 1 – determination of bulk density in small container (first revision)
IS7190(Part 2)-2004	Coke – methods of test: part 2 – determination of bulk density in large container (first revision)
IS7929-2000	Methods of determination of electrical resistivity of chemical coke
IS9127(Part 1)-2003	Methods of petrographic analysis of coal: part 1. Definition of terms relating to petrographic analysis of coal (first revision)
IS9127(Part 2)-2002	Methods of petrographic analysis of coal: part 2. Preparation of coal samples for petrographic analysis
IS9127(Part 3)-2002	Methods for the petrographic analysis of bituminous coal and anthracite: part 3. Method of determining maceral group composition
IS9127(Part 4)-2001	Method for petrographic analysis of coal: part 4. Method of determining microlithotype, carbominerite and minerite composition

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IS9127(Part 5)-2004	Methods for the petrographic analysis of coal and anthracite: part 5. Method of determining microscopically the reflectance of vitrinite (first revision)
IS9949-2000	Method of test for abrasive properties of coal and associated minerals (first revision)
IS12770-2000	Coal for cement manufacture
IS12891-2000	Method of determination of fusibility of ash of coal, coke and lignite
IS13810-2000	Code of practice for float and sink analysis of coal
IS15438-2004	Coal – determination of forms of sulphur
IS15439-2000	Hard coal – determination of oxygen content
IS15440-2004	Coal sampling of pulverized coal conveyed by gases in direct fixed coal systems

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## A.7 State Standards of Russia – GOST (GOST = Gosudarstvenny Standart)

Obtainable from: Technormativ Information Systems LLC, 19 Shossa Entuziastov, Moscow 111024, Russia

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GOST 501-2012	Determination of swelling index in the crucible
GOST 589-2012	Determination of total moisture
GOST 1186-1987	Method for determination of plastometric indices
GOST 2057-1982	Brown coals, hard coals, anthracites, combustible shales, and turf. Determination of ash fusibility
GOST 7404-5-2012	Methods for petrographic analysis of coal. Pt 5. Microscopic determination of reflection index of vitrinite
GOST 8858-1993	Brown coals, hard coals, anthracite, method for determination of moisture-holding capacity
GOST 8929-1975	Hard coal and coke. Determination of strength
GOST 9318-1991	Hard coal. Determination of caking power – Roga test
GOST 9414.1-1994	Petrographic analysis of coal: part 1. Glossary of terms
GOST 9414.2-1994	Petrographic analysis of bituminous coal and anthracite: part 2. Sample preparation
GOST 9414.3-1993	Petrographic analysis of bituminous coal and anthracite: part 3. Maceral group composition
GOST 9434-1975	Hard coal class by size
GOST 9516-1992	Moisture content of coal by gravimetric method
GOST 9521-1974	Method for determination of coking property
GOST 10089-1989	Measure of reactivity in coal and coke

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GOST 10100-1984	Determination of washability of hard coal
GOST 10175-1975	Determination of germanium in coal
GOST 10178-1975	Determination of arsenic in coal
GOST 10742-1971	Brown coals, hard coals, anthracite, and combustible shales sample preparation of coal for laboratory tests
GOST 10969-1991	Brown coals and lignites. Determination of yield of toluene-soluble extract and content of acetone-soluble materials (resinous substances)
GOST 11014-2001	Brown coals, bituminous coals, anthracite, and oil shales. Shortened methods of moisture determination
GOST 11055-1978	Brown coals, hard coals and anthracite. Radiation methods for determination of ash content
GOST 11056-1977	Hard coals. Electric method for determination of mass fraction of moisture
GOST 11223-1988	Method of sampling coal from boreholes
GOST 11760-2012	Classification of coals
GOST 11762-1987	Brown coals, hard coals and anthracite. Accuracy of weighting
GOST 12112-1978	Brown coals, method for determination of petrographic composition
GOST 13324-1994	Hard coals, method for determination of dilatometer characteristics, Audibert–Arnu dilatometer
GOST 13909-1-2012	Black coal and coke. Mechanical sampling. Pt 1. General provisions
GOST 13909-2-2012	Black coal and coke. Mechanical sampling. Pt 2. Sampling from moving streams
GOST 13909-3-2012	Black coal and coke. Mechanical sampling. Pt 3. Sampling from stationary lots
GOST 13909-4-2012	Black coal and coke. Mechanical sampling. Pt 4. Preparation of samples for testing
GOST 13909-5-2012	Black coal and coke. Mechanical sampling. Pt 5. Coke-sampling from moving streams
GOST 13909-6-2012	Black coal and coke. Mechanical sampling. Pt 6. Coke-preparation of samples for testing
GOST 14834-2014	Oxidized brown coals of the Far East. Classification
GOST 15489-2-2014	Hard coals. Determination of hardgrove grindability index
GOST 15585-2013	Hard coal. Determination of caking index
GOST 16126-1991	Hard coal. Determination of caking power – Gray-King method
GOST 17070-2014	Russian coal – Terms and definitions
GOST 17321-2015	Coal preparation – Terms and definitions
GOST 19242-1973	Brown coals, hard coals, and anthracite. Size classification
GOST 20330-1991	Hard coal and coke. Determination of crucible swelling number
GOST 21489-1976	Brown coals, hard coals, and anthracite. Classification by ranks and classes according to reflectance index of vitrinite
GOST 25543-2013	Brown coals, hard coals, and anthracite. Classification according to genetic and technological parameters
GOST 27044-1986	Hard coal and coke. Determination of carbon and hydrogen content
GOST 27588-1991	Hard coal, coke. Method for determination of total moisture content
GOST 28663-1990	Brown coals (low rank coals) codification
GOST 28743-1990	Coal. Determination of nitrogen by Kjeldahl method

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GOST 28823-1990	Methods for the petrographic analysis of bituminous coal and anthracite: Part 4. Method of determining microlithotypes, carbominerite, and mineral composition
GOST 28974-1991	Brown coals, hard coals, anthracites. Methods for determining Be, B, Mn, Ba, Cr, Ni, Co, Pb, Ga, V, Cu, Zn, Mo, Y, and La
GOST 29085-1991	Brown coals and lignites. Determination of moisture content. Indirect gravimetric method
GOST 29086-1991	Hard coal. Determination of mineral matter
GOST 30100-1993	Brown coal and lignites. Determination of moisture content. Direct volumetric method
GOST 30313-1995	Classification of coal
GOST 32247-2013	Hard coals, method of determination of the IGI-VUHIN swelling value
GOST 32558-2014	Coal. Determination of bulk density
GOST 32561-2013	Hard coal. Determination of plastic properties-Gieseler plastometer method
GOST 32619-2015	Brown coals, hard coals, anthracites. Determination of drop shatter test
GOST 32812-2014	Coals. Determination of oxidation
GOST 50177.2-1992	Methods for the petrographic analysis of bituminous coal and anthracite: part 2. Method of preparing coal samples
GOST 50177.3-1992	Methods for the petrographic analysis of bituminous coal and anthracite: part 3. Method of determining maceral group composition
GOST 50904-1996	Oxidized pit coal and anthracites of Kuznetsky and Gorlovsky basins. Classification
GOST 50921-2005	Coal and coke >20 mm size. Method of strength determination after reaction with carbon dioxide
GOST 51586-2000	Brown coal, hard coal, and anthracites of Kuznetsky and Gorlovsky basins for power supply purposes – specifications
GOST 51588-2000	Brown coal, hard coal, and anthracites of Kuznetsky and Gorlovsky basins for technological purposes – specifications
GOST 55663-2013	Petrographic analysis of coal. Pt 2. Preparation of coal samples
GOST 55955-2014	Standard practice for determination of gas content of coal

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## Appendix B

### Tables of True and Apparent Dip, Slope Angles, Gradients, and Percentage Slope

Table of true and apparent dip.

	Angle between strike and direction of section															
	80°	75°	70°	65°	60°	55°	50°	45°	40°	35°	30°	25°	20°	15°	10°	5°
TRUE DIP	APPARENT DIP															
10	10	10	9	9	9	8	8	7	6	6	5	4	3	3	2	1
15	15	14	14	14	13	12	12	10	10	9	8	6	5	4	3	1
20	20	19	19	18	18	17	16	14	13	12	10	9	7	5	4	2
25	25	24	24	23	22	21	20	18	17	15	13	11	9	7	5	2
30	30	29	28	28	27	25	24	22	20	18	16	14	11	9	6	3
35	35	34	33	32	31	30	28	26	24	22	19	16	13	10	7	4
40	40	39	38	37	36	35	33	31	28	26	23	20	16	12	8	4
45	45	44	43	42	41	39	37	35	33	30	27	23	19	15	10	5
50	50	49	48	47	46	44	42	40	37	34	31	27	22	17	12	6
55	55	54	53	52	51	49	48	45	43	39	36	31	26	20	14	7
60	60	59	58	58	56	55	53	51	48	45	41	36	30	24	17	9
65	65	64	64	63	62	60	59	57	54	51	46	42	36	29	20	11
70	70	69	69	69	68	67	65	63	60	58	54	49	43	35	25	13
75	75	74	74	74	73	72	71	69	67	65	62	58	52	44	33	18
80	80	80	79	79	78	78	77	76	75	73	71	67	63	56	45	26
85	85	85	85	84	84	84	83	83	82	81	80	78	76	71	63	45

Values for true dip, etc. not stated in the table may be calculated using the following equation:

$$\tan(\text{apparent dip}) = \tan[\text{true dip} \times \sin(\text{angle between strike and direction of section})].$$

Table of dips of strata and of land surfaces, expressed in angles, gradients, and percentage slope.<sup>a</sup>

Angle of slope (°)	Gradient	Slope (%)
1	1 : 57	1.7
2	1 : 29	3.5
3	1 : 19	5.2
4	1 : 14	7.0
5	1 : 11.4	8.7
6	1 : 9.5	10.5
7	1 : 8.1	12.3
8	1 : 7.1	14.1
9	1 : 6.3	15.8
10	1 : 5.7	17.6
11	1 : 5.1	19.4
12	1 : 4.7	21.3
13	1 : 4.3	23.1
14	1 : 4.0	24.9
15	1 : 3.7	26.8
16	1 : 3.5	28.7
17	1 : 3.3	30.6
18	1 : 3.1	32.5
19	1 : 2.9	34.4
20	1 : 2.7	36.4
25	1 : 2.1	46.5
30	1 : 1.7	57.7
35	1 : 1.4	70.0
40	1 : 1.2	83.9
45	1 : 1.0	100.0
50	1 : 0.8	119.2
55	1 : 0.7	142.8
60	1 : 0.6	173.2
65	1 : 0.5	214.5
70	1 : 0.4	274.7
75	1 : 0.3	373.2
80	1 : 0.2	567.1
85	1 : 0.1	1143.0
90	1 : 0	

a) Only those values most commonly encountered are included in this table.

## Appendix C

### Calorific Values Expressed in Different Units

$\text{MJ kg}^{-1}$	$\text{Btu lb}^{-1}$ $(\text{MJ kg}^{-1}) \times$ 429.923	$\text{kcal kg}^{-1}$ $(\text{MJ kg}^{-1}) \times$ 239.006	$\text{lb steam/lb}$ $\text{coal}$ $(\text{MJ kg}^{-1}) \times$ 0.442 763	$\text{MJ kg}^{-1}$	$\text{Btu lb}^{-1}$ $(\text{MJ kg}^{-1}) \times$ 429.923	$\text{kcal kg}^{-1}$ $(\text{MJ kg}^{-1}) \times$ 239.006	$\text{lb steam/lb}$ $\text{coal}$ $(\text{MJ kg}^{-1}) \times$ 0.442 763
4.5	1 935	1 076	1.99	7.0	3 009	1 673	3.10
4.6	1 978	1 099	2.04	7.1	3 052	1 697	3.14
4.7	2 021	1 123	2.08	7.2	3 095	1 721	3.19
4.8	2 064	1 147	2.13	7.3	3 138	1 745	3.23
4.9	2 107	1 171	2.17	7.4	3 181	1 769	3.28
				7.5	3 224	1 793	3.32
5.0	2 150	1 195	2.21	7.6	3 267	1 816	3.36
5.1	2 193	1 219	2.26	7.7	3 310	1 840	3.41
5.2	2 236	1 243	2.30	7.8	3 353	1 864	3.45
5.3	2 279	1 267	2.35	7.9	3 396	1 888	3.50
5.4	2 322	1 291	2.39				
5.5	2 365	1 315	2.44	8.0	3 439	1 912	3.54
5.6	2 408	1 338	2.48	8.1	3 482	1 936	3.59
5.7	2 451	1 362	2.52	8.2	3 525	1 960	3.63
5.8	2 494	1 386	2.57	8.3	3 568	1 984	3.67
5.9	2 537	1 410	2.61	8.4	3 611	2 008	3.72
				8.5	3 654	2 032	3.76
6.0	2 580	1 434	2.66	8.6	3 697	2 055	3.81
6.1	2 623	1 458	2.70	8.7	3 740	2 079	3.85
6.2	2 666	1 482	2.75	8.8	3 783	2 103	3.90
6.3	2 709	1 506	2.79	8.9	3 826	2 127	3.94
6.4	2 752	1 530	2.83				
6.5	2 794	1 554	2.88	9.0	3 869	2 151	3.98
6.6	2 837	1 577	2.92	9.1	3 912	2 175	4.03
6.7	2 880	1 601	2.97	9.2	3 955	2 199	4.07
6.8	2 923	1 625	3.01	9.3	3 998	2 223	4.12
6.9	2 966	1 649	3.06	9.4	4 041	2 247	4.16

(Continued)



<b>MJ kg<sup>-1</sup></b>	<b>Btu lb<sup>-1</sup> (MJ kg<sup>-1</sup>) × 429.923</b>	<b>kcal kg<sup>-1</sup> (MJ kg<sup>-1</sup>) × 239.006</b>	<b>lb steam/lb coal (MJ kg<sup>-1</sup>) × 0.442 763</b>	<b>MJ kg<sup>-1</sup></b>	<b>Btu lb<sup>-1</sup> (MJ kg<sup>-1</sup>) × 429.923</b>	<b>kcal kg<sup>-1</sup> (MJ kg<sup>-1</sup>) × 239.006</b>	<b>lb steam/lb coal (MJ kg<sup>-1</sup>) × 0.442 763</b>
9.5	4 084	2 271	4.21	13.0	5 589	3 107	5.76
9.6	4 127	2 294	4.25	13.1	5 632	3 131	5.80
9.7	4 170	2 318	4.29	13.2	5 675	3 155	5.84
9.8	4 213	2 342	4.34	13.3	5 718	3 179	5.89
9.9	4 256	2 366	4.38	13.4	5 761	3 203	5.93
				13.5	5 804	3 227	5.98
10.0	4 299	2 390	4.43	13.6	5 847	3 250	6.02
10.1	4 342	2 414	4.47	13.7	5 890	3 274	6.07
10.2	4 385	2 438	4.52	13.8	5 933	3 298	6.11
10.3	4 428	2 462	4.56	13.9	5 976	3 322	6.15
10.4	4 471	2 486	4.60				
10.5	4 514	2 510	4.65	14.0	6 019	3 346	6.20
10.6	4 557	2 533	4.69	14.1	6 062	3 370	6.24
10.7	4 600	2 557	4.74	14.2	6 105	3 394	6.29
10.8	4 643	2 581	4.78	14.3	6 148	3 418	6.33
10.9	4 686	2 605	4.83	14.4	6 191	3 442	6.38
				14.5	6 234	3 466	6.42
11.0	4 729	2 629	4.87	14.6	6 277	3 489	6.46
11.1	4 772	2 653	4.91	14.7	6 320	3 513	6.51
11.2	4 815	2 677	4.96	14.8	6 363	3 537	6.55
11.3	4 858	2 701	5.00	14.9	6 406	3 561	6.60
11.4	4 901	2 725	5.05				
11.5	4 944	2 749	5.09	15.0	6 449	3 585	6.64
11.6	4 987	2 772	5.14	15.1	6 492	3 609	6.69
11.7	5 030	2 796	5.18	15.2	6 535	3 633	6.73
11.8	5 073	2 820	5.22	15.3	6 578	3 657	6.77
11.9	5 116	2 844	5.27	15.4	6 621	3 681	6.82
				15.5	6 664	3 705	6.86
12.0	5 159	2 868	5.31	15.6	6 707	3 728	6.91
12.1	5 202	2 892	5.36	15.7	6 750	3 752	6.95
12.2	5 245	2 916	5.40	15.8	6 793	3 776	7.00
12.3	5 288	2 940	5.45	15.9	6 836	3 800	7.04
12.4	5 331	2 964	5.49				
12.5	5 374	2 988	5.53	16.0	6 879	3 824	7.08
12.6	5 417	3 011	5.58	16.1	6 922	3 848	7.13
12.7	5 460	3 035	5.62	16.2	6 965	3 872	7.17
12.8	5 503	3 059	5.67	16.3	7 008	3 896	7.22
12.9	5 546	3 083	5.71	16.4	7 051	3 920	7.26

$\text{MJ kg}^{-1}$	$\text{Btu lb}^{-1}$ $(\text{MJ kg}^{-1}) \times$ <b>429.923</b>	$\text{kcal kg}^{-1}$ $(\text{MJ kg}^{-1}) \times$ <b>239.006</b>	<b>lb steam/lb</b> <b>coal</b> $(\text{MJ kg}^{-1}) \times$ <b>0.442 763</b>	$\text{MJ kg}^{-1}$	$\text{Btu lb}^{-1}$ $(\text{MJ kg}^{-1}) \times$ <b>429.923</b>	$\text{kcal kg}^{-1}$ $(\text{MJ kg}^{-1}) \times$ <b>239.006</b>	<b>lb steam/lb</b> <b>coal</b> $(\text{MJ kg}^{-1}) \times$ <b>0.442 763</b>
16.5	7094	3944	7.31	20.0	8598	4780	8.86
16.6	7137	3967	7.35	20.1	8641	4804	8.90
16.7	7180	3991	7.39	20.2	8684	4828	8.94
16.8	7223	4015	7.44	20.3	8727	4852	8.99
16.9	7266	4039	7.48	20.4	8770	4876	9.03
				20.5	8813	4900	9.08
17.0	7309	4063	7.53	20.6	8856	4924	9.12
17.1	7352	4087	7.57	20.7	8899	4947	9.17
17.2	7395	4111	7.62	20.8	8942	4971	9.21
17.3	7438	4135	7.66	20.9	8985	4995	9.25
17.4	7481	4159	7.70				
17.5	7524	4183	7.75	21.0	9028	5019	9.30
17.6	7567	4207	7.79	21.1	9071	5043	9.34
17.7	7610	4230	7.84	21.2	9114	5067	9.39
17.8	7653	4254	7.88	21.3	9157	5091	9.43
17.9	7696	4278	7.93	21.4	9200	5115	9.48
				21.5	9243	5139	9.52
18.0	7739	4302	7.97	21.6	9286	5163	9.56
18.1	7782	4326	8.01	21.7	9329	5186	9.61
18.2	7825	4350	8.06	21.8	9372	5210	9.65
18.3	7868	4374	8.10	21.9	9415	5234	9.70
18.4	7911	4398	8.15				
18.5	7954	4422	8.19	22.0	9458	5258	9.74
18.6	7997	4446	8.24	22.1	9501	5282	9.79
18.7	8040	4469	8.28	22.2	9544	5306	9.83
18.8	8083	4493	8.32	22.3	9587	5330	9.87
18.9	8126	4517	8.37	22.4	9630	5354	9.92
				22.5	9673	5378	9.96
19.0	8169	4541	8.41	22.6	9716	5402	10.01
19.1	8212	4565	8.46	22.7	9759	5425	10.05
19.2	8255	4589	8.50	22.8	9802	5449	10.09
19.3	8298	4613	8.55	22.9	9845	5473	10.14
19.4	8341	4637	8.59				
19.5	8383	4661	8.63	23.0	9888	5497	10.18
19.6	8426	4685	8.68	23.1	9931	5521	10.23
19.7	8469	4708	8.72	23.2	9974	5545	10.27
19.8	8512	4732	8.77	23.3	10017	5569	10.32
19.9	8555	4756	8.81	23.4	10060	5593	10.36

(Continued)

$\text{MJ kg}^{-1}$	$\text{Btu lb}^{-1}$ $(\text{MJ kg}^{-1}) \times 429.923$	$\text{kcal kg}^{-1}$ $(\text{MJ kg}^{-1}) \times 239.006$	$\text{lb steam/lb coal}$ $(\text{MJ kg}^{-1}) \times 0.442\ 763$	$\text{MJ kg}^{-1}$	$\text{Btu lb}^{-1}$ $(\text{MJ kg}^{-1}) \times 429.923$	$\text{kcal kg}^{-1}$ $(\text{MJ kg}^{-1}) \times 239.006$	$\text{lb steam/lb coal}$ $(\text{MJ kg}^{-1}) \times 0.442\ 763$
23.5	10 103	5 617	10.40	27.0	11 608	6 453	11.95
23.6	10 146	5 641	10.45	27.1	11 651	6 477	12.00
23.7	10 189	5 664	10.49	27.2	11 694	6 501	12.04
23.8	10 232	5 688	10.54	27.3	11 737	6 525	12.09
23.9	10 275	5 712	10.58	27.4	11 780	6 549	12.13
				27.5	11 823	6 573	12.18
24.0	10 318	5 736	10.63	27.6	11 866	6 597	12.22
24.1	10 361	5 760	10.67	27.7	11 909	6 620	12.26
24.2	10 404	5 784	10.71	27.8	11 952	6 644	12.31
24.3	10 447	5 808	10.76	27.9	11 995	6 668	12.35
24.4	10 490	5 832	10.80				
24.5	10 533	5 856	10.85	28.0	12 038	6 692	12.40
24.6	10 576	5 880	10.89	28.1	12 081	6 716	12.44
24.7	10 619	5 903	10.94	28.2	12 124	6 740	12.49
24.8	10 662	5 927	10.98	28.3	12 167	6 764	12.53
24.9	10 705	5 951	11.02	28.4	12 210	6 788	12.57
				28.5	12 253	6 812	12.62
25.0	10 748	5 975	11.07	28.6	12 296	6 836	12.66
25.1	10 791	5 999	11.11	28.7	12 339	6 859	12.71
25.2	10 834	6 023	11.16	28.8	12 382	6 883	12.75
25.3	10 877	6 047	11.20	28.9	12 425	6 907	12.80
25.4	10 920	6 071	11.25				
25.5	10 963	6 095	11.29	29.0	12 468	6 931	12.84
25.6	11 006	6 119	11.33	29.1	12 511	6 955	12.88
25.7	11 049	6 142	11.38	29.2	12 554	6 979	12.93
25.8	11 092	6 166	11.42	29.3	12 597	7 003	12.97
25.9	11 135	6 190	11.47	29.4	12 640	7 027	13.02
				29.5	12 683	7 051	13.06
26.0	11 178	6 214	11.51	29.6	12 726	7 075	13.11
26.1	11 221	6 238	11.56	29.7	12 769	7 098	13.15
26.2	11 264	6 262	11.60	29.8	12 812	7 122	13.19
26.3	11 307	6 286	11.64	29.9	12 855	7 146	13.24
26.4	11 350	6 310	11.69				
26.5	11 393	6 334	11.73	30.0	12 898	7 170	13.28
26.6	11 436	6 358	11.78	30.1	12 941	7 194	13.33
26.7	11 479	6 381	11.82	30.2	12 984	7 218	13.37
26.8	11 522	6 405	11.87	30.3	13 027	7 242	13.42
26.9	11 565	6 429	11.91	30.4	13 070	7 266	13.46

$\text{MJ kg}^{-1}$	$\text{Btu lb}^{-1}$ $(\text{MJ kg}^{-1}) \times$ 429.923	$\text{kcal kg}^{-1}$ $(\text{MJ kg}^{-1}) \times$ 239.006	$\text{lb steam/lb}$ $\text{coal}$ $(\text{MJ kg}^{-1}) \times$ 0.442 763	$\text{MJ kg}^{-1}$	$\text{Btu lb}^{-1}$ $(\text{MJ kg}^{-1}) \times$ 429.923	$\text{kcal kg}^{-1}$ $(\text{MJ kg}^{-1}) \times$ 239.006	$\text{lb steam/lb}$ $\text{coal}$ $(\text{MJ kg}^{-1}) \times$ 0.442 763
30.5	13 113	7 290	13.50	34.0	14 617	8 126	15.05
30.6	13 156	7 314	13.55	34.1	14 660	8 150	15.10
30.7	13 199	7 337	13.59	34.2	14 703	8 174	15.14
30.8	13 242	7 361	13.64	34.3	14 746	8 198	15.19
30.9	13 285	7 385	13.68	34.4	14 789	8 222	15.23
				34.5	14 832	8 246	15.28
31.0	13 328	7 409	13.73	34.6	14 875	8 270	15.32
31.1	13 371	7 433	13.77	34.7	14 918	8 294	15.36
31.2	13 414	7 457	13.81	34.8	14 961	8 317	15.41
31.3	13 457	7 481	13.86	34.9	15 004	8 341	15.45
31.4	13 500	7 505	13.90				
31.5	13 543	7 529	13.95	35.0	15 047	8 365	15.50
31.6	13 586	7 553	13.99	35.1	15 090	8 389	15.54
31.7	13 629	7 576	14.04	35.2	15 133	8 413	15.59
31.8	13 672	7 600	14.08	35.3	15 176	8 437	15.63
31.9	13 715	7 624	14.12	35.4	15 219	8 461	15.67
				35.5	15 262	8 485	15.72
32.0	13 758	7 648	14.17	35.6	15 305	8 509	15.76
32.1	13 801	7 672	14.21	35.7	15 348	8 533	15.81
32.2	13 844	7 696	14.26	35.8	15 391	8 556	15.85
32.3	13 887	7 720	14.30	35.9	15 434	8 580	15.90
32.4	13 930	7 744	14.35				
32.5	13 972	7 768	14.39	36.0	15 477	8 604	15.94
32.6	14 015	7 792	14.43	36.1	15 520	8 628	15.98
32.7	14 058	7 815	14.48	36.2	15 563	8 652	16.03
32.8	14 101	7 839	14.52	36.3	15 606	8 676	16.07
32.9	14 144	7 863	14.57	36.4	15 649	8 700	16.12
				36.5	15 692	8 724	16.16
33.0	14 187	7 887	14.61	36.6	15 735	8 748	16.21
33.1	14 230	7 911	14.66	36.7	15 778	8 772	16.25
33.2	14 273	7 935	14.70	36.8	15 821	8 795	16.29
33.3	14 316	7 959	14.74	36.9	15 864	8 819	16.34
33.4	14 359	7 983	14.79				
33.5	14 402	8 007	14.83	37.0	15 907	8 843	16.38
33.6	14 445	8 031	14.88	37.1	15 950	8 867	16.43
33.7	14 488	8 055	14.92	37.2	15 993	8 891	16.47
33.8	14 531	8 078	14.97	37.3	16 036	8 915	16.52
33.9	14 574	8 102	15.01	37.4	16 079	8 939	16.56

## Appendix D

### Coal Statistics

1 million tonnes coal equivalent

= 1 million tonnes of coal at  $28.0 \text{ MJ kg}^{-1}$

=  $6692 \text{ kcal kg}^{-1}$  gross calorific value

1 million tonnes oil equivalent

$\approx 1.5$  million tonnes of coal

$\approx 3.0$  million tonnes of lignite

1 t of coal at  $25.1 \text{ MJ kg}^{-1}$  or  $6000 \text{ kcal kg}^{-1}$   
will produce approximately  $2400 \text{ kW h}^{-1}$  of  
electricity

A 1000 MW power station requires  $3 \times 10^6$  t  
of coal at  $25.1 \text{ MJ kg}^{-1}$  per annum

1 t of coal at  $25.1 \text{ MJ kg}^{-1}$  or  $6000 \text{ kcal kg}^{-1}$   
will produce approximately 7.5–9.0 t of cement

1 t of coal at 28% volatile matter, after coking,  
will produce approximately 1.5 t of iron



## Appendix E

### Methane Units Converter

To convert from:	To:	Multiply by:
Cubic foot	Cubic metre	0.028 32
Pound	Kilogram	0.453 6
Short ton	Tonne	0.907 2
Btu	Joule	1055
Cubic foot methane	Pound methane	0.042 46
	Btu	1014.6
	kW h <sup>-1</sup>	0.297 4
	Tonne CO <sub>2</sub> equivalent	0.000 404
	Tonne C equivalent	0.000 11
	Gram	19.26
Pound methane	Btu	23 896
	kW h <sup>-1</sup>	7
	Tonne CO <sub>2</sub> equivalent	0.009 53
	Tonne C equivalent	0.002 6
Btu	kW h <sup>-1</sup>	0.000 293
Methane GWP	CO <sub>2</sub> GWP	21
CO <sub>2</sub>	C equivalent	0.272 73
Methane	C equivalent	5.727 3

GWP, global warming potential.

Source: Adapted from US Environmental Protection Agency Coalbed Methane Outreach Program.

## Glossary

**Abandoned-mine methane** See *Coalmine methane*

**Air-dried basis** The data are expressed as percentages of the air-dried coal; this includes the air-dried moisture but not the surface moisture of the coal

**Allochthonous** Redeposited sedimentary material originating from distant sources

**Allogenic** Externally generated

**Anastomose** Rivers comprising two or more channels separated by floodplain deposits

**Angiosperms** Plants with covered seeds

**Anthracite** The highest rank coal, characterised by low volatile matter (<10%) and high carbon content. Semi-anthracite is coal midway between low-volatile bituminous and anthracite

**Anthropogenic** Human activity or influence

**Ash** The inorganic residue remaining after the combustion of coal. It is less than the mineral matter content because of the chemical changes occurring during combustion, i.e. the loss of water of hydration, loss of carbon dioxide, and loss of sulfurous gases from sulfides

**Assigned reserves** Coal that can be mined on the basis of current mining practices and techniques through the use of mines currently in existence or under construction

**As-received basis** The data are expressed as percentages of the coal including the total moisture content, i.e. including both the surface and the air-dried moisture content of the coal

**As-received moisture** The total moisture of a coal sample when delivered to the laboratory

**Autochthonous** Indigenous material formed in situ

**Autogenic** Internally generated

**Billion** 1 000 000 000

**Bituminous coal** Bituminous coal lies between subbituminous coal and semi-anthracite in terms of rank. Usually divided into three subgroups: low volatile, medium volatile, and high volatile

**Blackdamp** A coalmine gas that is non-explosive, consisting mainly of carbon dioxide and nitrogen

**Bouguer anomaly** The anomalous gravity value obtained after latitude correction, elevation correction (both free-air and Bouguer corrections), and terrain corrections have been applied to observed gravity data

**Bouguer correction** A correction to gravity data because of the attraction of the rock between the station and the datum elevation, usually mean sea level

**Brown coal** See *Lignite*

**Bucketwheel excavator (BWE)** Large earth-moving machines using a boom with a rotating wheel on which are a series of buckets with a cutting edge for excavating soft overburden and brown coal or lignite

**Bulk sample** Large coal sample, 5–25 t, for test work to establish coal's performance under actual conditions of usage

- C & F** Cost and freight, where cargo is purchased at port of delivery
- Calorific value (CV)** Also known as specific energy (SE). Is the amount of heat per unit mass of coal when combusted. See *Gross calorific value* and *Net calorific value*
- Carbon dioxide (CO<sub>2</sub>)** In mines, a constituent of blackdamp. A gas emitted by the use of coal and other fossil fuels into the atmosphere
- Carbon monoxide (CO)** Originates from the incomplete oxidation of coal, especially after methane explosions or underground fires. The gas is combustible and poisonous
- Channel sample** A channel of uniform cross-section is cut into the coal seam; all the coal within the cut section is collected (for the whole seam or for a series of plies, i.e. divisions of the coal seam)
- CIF** Cost, insurance and freight, where cargo is purchased at port of delivery, including insurance
- Cleat** Jointing in coal appearing as regular patterns of cracks that may have originated during coalification
- Coal-bed methane (CBM)** Occurs as a free gas (methane CH<sub>4</sub>) within the ore space or fractures in coal, or as adsorbed molecules on the organic surface of coal
- Coal liquefaction** Use of a number of techniques to yield liquid hydrocarbons and solid products from coal
- Coalmine methane (CMM)** Methane released from coal by coalmining, and is captured/released by ventilation and/or mine drainage wells in active and abandoned coalmines
- Coal preparation** Physical and mechanical processes applied to coal to make it suitable for a particular use
- Coal price** Calculated using formulas, which include base price for coal, calorific value, transport cost, and any local costs
- Coal seam gas** Synonymous with *Coal-bed methane (CBM)*; it is a term used in Australia
- Coal truck** Smaller vehicle for transporting coal on private and public highways
- Coalification** The alteration of vegetation to form peat, succeeded by the transformation of peat through lignite, subbituminous, bituminous, semi-anthracite, to anthracite, and meta-anthracite coal
- Coking coal** A coal suitable for carbonisation in coke ovens. It must have good coking and caking properties, and rank should be high-to medium-volatile bituminous coal
- Core sample** Coal collected from borehole cores, usually unweathered
- Cuttings sample** Coal fragments collected from drilling medium, from boreholes; less accurate than core sampling
- Density** The weight per unit volume, e.g. grams per cubic centimetre, tonnes per cubic metre
- DES** Delivered ex-ship; cargo considered delivered after unloading at port
- Dragline** Large earth-moving machines with bucket capacities 5–30 m<sup>3</sup>, used for overburden removal
- Dry ash-free basis** The coal is considered to consist of volatile matter and fixed carbon on the basis of recalculation with moisture and ash removed
- Dry basis** The data are expressed as percentages of the coal after all the moisture has been removed
- Dry, mineral matter free basis** The data are expressed as percentages of the coal on the basis of recalculation with moisture and mineral matter removed
- Embryophytes** Land plants
- Emissions** Those gases and particulates vented by power stations and industrial users. In particular, those that have a detrimental effect on the environment
- Exploration** The examination of an area by means of surface geological mapping, geophysical techniques, the drilling of boreholes, and sampling of coals
- Extractable reserves** See *Recoverable reserves*

- Fines** Very small coal with a maximum size that is usually less than 4 mm
- Firedamp** A mixture of methane (CH<sub>4</sub>) content of 5–15% with fresh air and is highly explosive
- Flaser bedding** A primary sedimentary structure consisting of fine sand or silt lenticles that are commonly aligned or cross-bedded
- Float–sink tests** The separation of coal and mineral matter particles by immersion in a series of liquids of known relative density. The process is designed to reduce the ash level of the coal and so improve the product to be sold
- Flue gas desulfurisation (FGD)** A process designed to recover the acidic sulfur compounds from the flue gas prior to release to the atmosphere
- Fluidised-bed combustion (FBC)** The burning of fuel on a bed subjected to an upward gas flow that causes the bed to be suspended, resulting in the transference of heat at very high rates
- FOB** Freight on board; cargoes purchased when loaded at port of embarkation
- Gamma-ray log** A geophysical well log that records natural radioactivity
- Geological losses** Losses to be deducted from measured reserves due to geological constraints, e.g. faults, washouts, seam splitting
- Geophysical logging** Continuous measurements of the variation with depth of selected physical properties of rocks with geophysical measuring tools (sondes) located in boreholes
- Goaf** Mined out area, behind advancing longwall mining operations
- Gondwanaland** Southern segment of Pangaea
- GPS** Global Positioning System; use of satellite signals to enable accurate positioning on the ground
- Grab sample** Collection of coal sample from outcrop or stockpile
- Gravity survey** Measurement of local and regional variations in the Earth's gravitational field. Gravity data displayed as Bouguer anomaly values in milligals (mGal)
- Gross calorific value** The amount of heat liberated during the combustion of a coal in the laboratory under standardised conditions at constant volume, so that all of the water in the products remains in liquid form
- Groundwater** Water below the ground surface and below the water table
- Groundwater rebound** Rising water levels after cessation of long-term pumping
- Gymnosperms** Plants with naked seeds
- Higher heating value** See *Gross calorific value*
- Highwall** Face of opencast mine towards which the coal is mined; may be single face or a series of benches if a series of coals is mined
- Highwall mining** Remotely controlled mining method that extracts coal from the base of an exposed highwall in an opencast mine
- Humic** Coals formed from a diversified mixture of macroscopic plant debris; almost all economic coals are of this type
- In-seam seismic (ISS) survey** Use of generated channel waves propagating in the coal seam to detect discontinuities in advance of mining
- Indicated reserves** Includes all coal conforming to the thickness and depth limits defined in the reserve base; bounded by similar distance limits as for indicated resources
- Indicated resources** A coal resource with a higher confidence level than inferred resources, for which the points of observation plus interpretive data are sufficient to allow a realistic estimate of average coal thickness, areal extent, depth range and coal quality. Points of observation are normally less than 1 km apart

**Inferred reserves** Includes all coal conforming to the thickness and depth limits defined in the reserve base; bounded by similar distance limits as for inferred resources

**Inferred resources** A coal resource for which tonnage and quality can be estimated at a low level of confidence, but should provide sufficient understanding of the geology to estimate continuity of coal seams, range of coal thickness, and coal quality. Inferred coal resources may be estimated using points of observation up to 4 km apart

**In-situ reserves** The quantity of coal in the ground within geological and economic limits. This can include both mineable and unmineable reserves, for which the term resources may be used

**Kyoto Protocol** The Conference of the Parties (COP-3) held in Kyoto, Japan in 1997, at which a protocol outlining the emissions targets set for all the attending countries was drawn up

**Laurasia** Northern segment of Pangaea

**Lignite** A low-rank coal characterised by a high moisture content. A coal is considered a lignite if it contains >20% in-situ moisture. Lignite is generally referred to as brown coal

**Long ton** 2240 lb; deadweight tons are expressed as long tons

**Longwall mining** Mining of coal in elongated panels from a single face, either moving forwards (advance) or by working back towards the entry roadway (retreat)

**Macerals** The microscopically recognisable organic constituents of coal. A given maceral may differ significantly in composition and properties from one coal to another; this variation may depend on the rank of the coal

**Magnetic susceptibility** Measure of the degree to which a rock may be magnetised, which is related to the content of ferromagnetic minerals

**Magnetic survey** Measurements of local and regional variations of the Earth's magnetic field or its components (e.g. vertical field). Magnetic anomaly values are given in nanoteslas (nT). Differences of the Earth's geomagnetic field are attributed to rocks with variable magnetic susceptibilities

**Marketable reserves** Those tonnages of coal available for sale. If the coal is marketed raw, the marketable reserves will equal the raw coal tonnage. If the coal is beneficiated, the marketable reserves are calculated by applying the predicted yield to the run-of-mine or raw coal tonnage

**Measured reserves** Those resources for which the density and quality of points of measurement are not more than 0.5 km apart and are sufficient to allow a reliable estimate of the coal tonnage

**Measured resources** A coal resource where points of observation plus interpretative data are sufficient to allow a reliable estimate of average coal thickness, areal extent, depth range, and coal quality sufficient to generate detailed mine plans and determine mining and beneficiation costs plus the specification of a marketable product using data normally 0.5 km apart

**Megawatt hour (MWh)** A unit of electricity denoting the work of  $1 \times 10^6$  W acting for 1 h

**Metallurgical coal** See also *Coking coal*. Coal suitable for metallurgical use because of its coking qualities and chemical characteristics

**Methane (CH<sub>4</sub>)** A gas produced by the decomposition of organic material. Methane consists of carbon and hydrogen and when mixed with air it forms a highly combustible gas. Also known as 'firedamp'

**Middlings** The result of cleaning coal to produce two products: a prime product and a lower quality or 'middlings' product. The percentage yields of the two products are calculated by using an M-curve graph

**Milligal (mGal)** Unit of acceleration for gravity measurements. 1 Gal =  $1 \text{ cm s}^{-2}$



- Mineable reserves** The tonnages of in-situ coal contained in seams or sections of seams for which sufficient information is available to enable detailed or conceptual mine planning
- Mineral matter** The inorganic components of coal. This does not equate to the ash content; mineral matter includes other components, such as carbon dioxide, sulfur oxides, and water of hydration lost upon combustion of the coal
- Mire** General term for peat-forming ecosystems of all types
- Nanotesla (nT)** Unit of magnetic flux density.  $1 \text{ nT} = 1 \times 10^{-9} \text{ T}$
- Net calorific value** During combustion in furnaces, the maximum achievable calorific value is the net calorific value at constant pressure. It is calculated and expressed in absolute joules, calories per gram, or Btu per pound
- NO<sub>x</sub>** A group of gaseous pollutants, notably nitrous oxide (N<sub>2</sub>O), released in flue gases vented to the atmosphere
- Ombrogenous** Peats whose moisture content is dependent on rainfall
- Ombrotrophic** Peats fed by rainfall
- Opencast mining** Surface mining of coal seams that may be overlain by variable amounts of overburden
- Oxidation** State when coal is exposed to oxygen and the properties of coal begin to change; in particular, the lowering of calorific value
- Paludification** Replacement of dry land by a mire due to a rising groundwater table
- Pangaea** Supercontinent existing until Triassic Period
- Passive seismic method** Seismic investigations using listening capabilities only with no seismic energy produced by the investigator
- Peat** The first stage in the coalification process. The in-situ moisture is high, often >75%. Original plant structure is clearly visible
- Pennsylvanian** Upper part of Carboniferous Period; name used in USA
- Permeability** The ability of water to flow through an aquifer
- Piezometer** Boreholes sealed throughout their depth in such a way that they measure the head of groundwater at a particular depth in the horizon selected
- Pillar sample** Large blocks of undisturbed coal taken in underground workings to provide technical information on strength and quality of the coal
- Porosity** Property of a rock possessing pores or voids
- Possible reserves** See *Inferred reserves*
- Probable reserves** See *Indicated reserves*
- Radioactivity log** A well log of natural or induced radiation. Usually refers to a gamma-ray log, but sometimes also to a density log, neutron log, etc.
- Rank** Coals range in composition and properties according to the degree of coalification. Rank is used to indicate this level of alteration; the greater the alteration, the higher the rank. Lignites are low-rank coals, and anthracites are high rank
- Raw coal** Coal that has received no preparation, other than possibly screening
- Reconnaissance** Preliminary examination of a defined area to determine if coal is present; includes broad-based geological mapping and coal sampling
- Recoverable reserves** The reserves that are or can be extracted from a coal seam during mining. Recoverable reserves are obtained by deducting anticipated geological and mining losses from the in-situ reserves. (Also known as *Extractable reserves*)
- Reserves** The quantity of mineral that is calculated to lie within given boundaries. The reserves are described dependent on certain arbitrary limits in respect of thickness, depth, quality, and other geological and economic factors
- Resources** The amount of coal in place before exploitation
- Rheotrophic** Peat fed by water flow

**Rock mass rating** An indicator of the behaviour of rock mass surrounding an underground excavation using a number of geotechnical parameters

**Room-and-pillar mining** Coal extracted from square or rectangular rooms or entries in the coal seam, leaving coal between the rooms as pillars to support the roof

**RQD** Rock quality designation; a quantitative index based on core recovery by diamond drilling

**Run-of-mine coal** Coal produced by mining operations before preparation

**Saleable coal** The total amount of coal output after preparation of the run-of-mine coal. It equals the total run-of-mine tonnages minus any material discarded during preparation

**Saleable reserves** See *Marketable reserves*

**Sapropelic** Coals formed from a restricted variety of microscopic organic debris, which can include algae

**Seismic survey** A programme for mapping geological structures by creating seismic signals from an explosion or other seismic source and observing the arrival time of the reflected and refracted seismic waves

**Shovel** Electric and hydraulic types, for overburden and coal removal

**Silo** Large, cylindrical storage container for coal, usually at dispatch areas

**Size range** Indicates the largest and smallest sizes of particles in a coal sample or stream

**Slurry** Particles concentrating in a portion of the circulating water and water-borne to a treatment plant of any kind. Or, fine particles <1 mm in size recovered from a coal preparation process and containing a substantial proportion of inerts

**Short ton** 2000 lb

**Specific gravity** The ratio of the density of a substance to the density of a reference substance (also known as relative density)

**Specimen sample** Orientated sample of coal collected for studies in laboratory of optical fabric or structural features of coal

**Sphenophytes** Leaf-bearing plants

**Spontaneous combustion** The adsorption of oxygen onto the surface of coal to promote oxidation and produce heat. The propensity to spontaneous combustion is related to rank, moisture content, and size of coal

**Steam coal** Coal not suitable as metallurgical coal because of its non-coking characteristics; primarily used for the generation of electric power

**Stockpile** Coal storage, either covered or uncovered, from which coal is conveyed to the transportation system (road, rail, or sea) or directly for use

**Strip mining** Surface mining of coal seams at outcrop

**Stripping ratio (SR)** The ratio of the thickness of overburden to that of the total workable coal section

**Subbituminous coal** Lies in rank between lignite and bituminous coal. Typical in-situ moisture levels are 10–20%

**Sulfur** May be a component of the organic and/or mineral fractions of a coal. Forms sulfur dioxide during coal combustion, a serious pollutant. It is also undesirable in coking coals because it contaminates the hot metal

**Sulfur dioxide (SO<sub>2</sub>)** A principal gaseous pollutant released by flue gases vented to the atmosphere

**Terrestrialisation** Replacement of a body of water by a mire

**Thermal coal** See *Steam coal*

**Time-lapse seismic surveys** Repeating a seismic survey at exactly the same location to determine changes that have occurred with time, usually due to liquid or gas migration. Multiple 3D seismic surveys run at different times are sometimes called 4D seismic surveys, with the fourth dimension being time

**Tonne** 1000 kg or 2204.6 lb (or metric ton, especially in the USA)

**Tomography** Any of several techniques for making detailed X-rays of a predetermined plane section of a solid object while blurring out the images of other planes

**Topogenous** Peat whose moisture content is dependent on surface water

**Transmissivity** An aquifer's effectiveness to transmit water as calculated by Darcy's law

**Truck** Large-capacity vehicle usually employed for transporting overburden material from the mine to dump areas; size usually tailored to size of earth-moving equipment; not used on public highways

**Tuyere** Tube, nozzle or pipe through which air is blown into a furnace or forge

**Unassigned reserves** Coal that would require additional mine facilities for extraction. The extent to which unassigned reserves will actually be mined depends on future economic and environmental conditions

**Underground coal gasification** Coal gasification conducted in situ as an alternative method of capturing energy

**Vertical seismic profiling (VSP)**

Measurement of the response of a detector geophone at various depths in a borehole to a seismic source at the surface

**Volatile matter** Represents that component of the coal, except for moisture, that is liberated at high temperature in the absence of air

**Washability curves** The results of float-sink tests, plotted graphically as a series of curves. Used to calculate the amount of coal that can be obtained at a particular quality, the density required to effect such a separation, and the quality of the discard left behind

**Washed coal** Coal that has been beneficiated by passing through a coal preparation wash plant

**Water table** Upper limit of the saturated zone below ground surface

**Westphalian** Upper part of Carboniferous Period; name used in Europe

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## Index

### a

- Abandoned mine methane (AMM) 350, 353, 355
- Abandoned workings 386, 418
- Abrasion index 133
- Abrasiveness 133
- Absorber 409
- Absorption 410
- Accidents 415
- Acid corrosion 410
- Acid mine drainage (AMD) 387, 388, 390, 391, 395
- Acid mine water 387, 389
- Acidic 19, 388
- Acoustic impedance 246
- Acoustic scanning tools 271, 279, 280, 323
- Acoustic signal 246
- Acoustic transducer 279
- Active coalmine methane 350
- Adit 312, 313
- Adsorption 343, 346
- Aerial photographs 181, 182, 183
- Aeromagnetic survey 260–262
- Afghanistan 84
- Africa 68
- Air borne imagery 181
- Air dried basis 122
- Air pollution, *see* Atmospheric pollution
- Air quality 419
- Albania 75
- Algae 101
- Algeria 418
- Alginite 98, 102
- Alkaline mine water 387
- Allochthonous 19, 25
- Allogenic 24
- Alluvial plain 11, 43
- Aluminium 388
- Amelioration 53, 77, 83, 87, 94
- American Petroleum Institute (API) Units 271
- American Society for Testing and Materials (ASTM) 99, 116, 136, 139, 178, 180
- Ammonia 345
- Ammonia scrubbing 409
- Ammoniacal nitrogen 388
- Ammonium chloride 388
- Analysis 122
  - proximate 123
  - ultimate 125
- Anastomosed 11, 14
- Angiosperm 23, 58, 377
- Angola 80
- Angularity 198
- Ankerite 107
- Anoxic 17
- Antarctica 57, 58
- Anthracite, *see* Coal
- Anthropogenic 419
- Anticline, *see* Folds
- Antimony 407
- Anvil Rock Sandstone 65
- Apatite 110
- Appalachians 6, 68
- Aquicludes 25, 290
- Aquifers 290, 291, 296, 298, 299, 303, 329, 387, 418
  - confined 290, 291, 298, 307
  - unconfined 290, 291, 307
- Aquitards 290
- Arbitration 432
- Area-of-influence technique 228
- Argentina 85
- Arsenic 407
- Artesian pressure 299
- Artificial roadways 326
- As received basis 122
- As sampled basis 124
- Ash 18, 19, 39, 76, 79, 85, 86, 97, 110, 124, 176, 405, 408
- Ash analysis 126
- Ash free basis 124
- Ash fusion temperatures 110, 113, 128, 404
- Asthma 416
- Atmospheric pollution 411
- Attrinite 106
- Audibert–Arnu dilatometer 129
- Australia 6, 73, 95, 105, 156, 208, 241, 253, 258, 282, 318–320, 326, 327, 329, 334, 337, 339, 344, 353, 361, 365, 368, 374, 377, 380, 381, 389, 418, 419, 420
- Ackaringa 95
- Bowen–Surat Basin 95, 362

- Australia (*contd.*)
- Brisbane 95
  - Burratorang Valley 95
  - Cardiff 95
  - Clarence–Moreton Basin 362
  - Collie Coalfield 95
  - Cooper Basin 95, 282, 381
  - Dalrymple Bay 427, 429
  - Fitzroy Basin 95
  - Galilee Basin 95, 362
  - Gippsland Basin, 95 381
  - Gloucester Basin 362
  - Gunnedah Basin 362
  - Hunter Valley 95, 389
  - Latrobe Valley 96, 328, 338
  - Muja 95
  - New South Wales 95, 361
  - Queensland 95, 348, 361, 374
  - South Australia 95
  - Southern District 95
  - Sydney Basin 95, 362
  - Tasmania 95
  - Victoria 95
  - Western Australia 95
  - Western District 95
- Australian Standards (AS) 99, 123, 145, 146, 177, 179
- Austria 75
- Authigenic minerals 107
- Autochthonous 19, 25
- Autogenic 24
- Automatic coal loading 398
- Automatic weighing 427
- Autothermal reactor 379
- Average Structural Index (ASI) 50
- b**
- Backfill 374
- Bacteria 376, 390, 391
- Baked sediments 262
- Bali Action Plan 421
- Balkan Endemic Nephropathy 416
- Bangladesh 84, 259, 260
- Barite 111
- Barium 388
- Barnsley Seam 21, 22
- Barometric pressure 355
- Barrel reclaimers 425
- Barrier 7, 8, 24, 42
- Barrier sand bars 8
- Basalt 249, 250
- Base-acid ratio 110
- Base flow 289
- Bases calculation 122
- Basic oxygen furnace 413, 414
- Basin subsidence 25, 40
- Batters 232
- Baum jig 424
- BCURA formula 123
- Beckley seam 35
- Bedding Code 50, 51
- Belgium 75
- Bench 329, 336
- Beneficiation 83, 423
- Bentonite 390
- Biostratigraphy 61, 62
- Bioturbation 8
- Bitumen 118, 397
- Bituminisation 116
- Bituminite 102
- Bituminous, *see* Coal
- Bivalve 61
- Blackdamp 344
- Blade bit 185
- Blanket bog, *see* Bog
- Blast furnace 125
- Blasting 316, 327
- Block model 329, 330
- Bog 15
- Bog forest 15, 16
- Boghead coal, *see* Coal
- Boilers 128, 404
- dry bottom 128
  - stoker 128
  - wet bottom 128
- Bolivia 86
- Bolts/bolting 265
- Bord and pillar, *see* Room & pillar
- Boreholes 25, 29, 184, 188, 189, 194, 227, 286, 295, 297, 320, 323
- cored 38, 188, 189, 296, 352
  - cross measure 352
  - deviated 368
  - directional 368
  - horizontal 352
  - logging 243
  - open 184, 185, 193, 194, 296
  - part cored 189, 190
  - pilot 190
- Boron 111
- Bosnia-Herzegovina 45, 75, 394
- Gacko 75, 335, 337
- Botswana 80
- Bottom ash 408
- Bouguer Gravity Anomaly 258, 259
- Box cut 299
- Brazil 86, 208, 258, 363, 375
- Breakouts 281, 323
- Brines, *see* Saline waters
- Briquettes 415
- Britain, *see* United Kingdom
- British Coal 140
- British Standards (BS ISO) 123, 128, 130, 195
- British Thermal Units (BTU) 127, 128, 147, 380
- Brunei 89
- Bucketwheel excavator (BWE) 329, 331, 334–336
- Bulgaria 75
- Bulldozer 333
- Bulk carriers 429
- Burma, *see* Myanmar
- Burnt coal 260, 277
- Burnt zone 262
- Burrow, *see* Bioturbation
- c**
- Cadastral maps 171
- Cadmium 407
- Caking properties 128

- Caking tests 128  
 Calcite 98, 107, 110  
 Calcium 388  
 Calcium carbonate 409  
 Calcium chloride 126  
 Calcium hydroxide 409  
 Calcium oxide 409  
 Calcium sulphate 388  
 Calliper log, *see* Geophysical logs  
 Calorific value 113, 115, 127, 128, 203, 275, 365, 404, 406, 423  
     gross calorific value 127, 433  
     net calorific value 127, 433  
 Cameroun 81  
 Campsites 172  
 Canada 57, 66, 208, 211, 299, 357, 360, 375, 419, 420, 428  
     Alberta 74, 357, 360  
     British Columbia 74, 360  
     Frazer River 25  
     Minto Coalfield 74  
     National Instrument (NI) 211  
     New Brunswick 74  
     Northwest Territories 74  
     Nova Scotia 74  
     Saskatchewan 74  
     Sydney Coalfield 74  
     Yukon Territory 74  
 Capital, *see* Costs  
 Carbagillite 110  
 Carbon (C) 125, 346, 375, 377  
 Carbon Capture & Storage (CCS) 286, 416, 418  
 Carbon dioxide (CO<sub>2</sub>) 117, 126, 343, 344, 345, 386, 410, 411, 414, 416–419  
 Carbon dioxide sequestration 257, 365, 418  
 Carbon monoxide (CO) 343, 344  
 Carbonaceous mudstone/shale 18, 20  
 Carbonate minerals 44, 55  
 Carboniferous 6, 9, 11, 20, 21, 24, 44, 46, 57, 59, 61, 75–81, 85, 86, 102, 103, 113, 176, 292, 361  
 Cargo 433  
 Cash flow 330  
 Casing, *see* Boreholes  
 Catagenesis 345  
 Catchment area 289  
 Cathaysian 20, 22  
 Caving 277  
 Cavities 398  
 Cavity pressure 368  
 Cellulose 19  
 Cement industry 409, 414  
 Cenozoic 75, 84, 88–91, 95  
 Central and South America 70  
 Chain conveyor 314  
 Channel-fill deposits 8, 39  
 Channel sample, *see* Sample  
 Channel waves 264  
 Charcoal 20  
 Chile 86  
 China (Peoples Republic of) 57, 61, 92, 113, 222, 241, 252, 268, 313, 326–328, 332, 336, 338, 341, 353, 360, 365, 368, 374, 375, 379, 399, 403, 413–415, 417, 419, 422, 425, 427–429  
     Anhui 92  
     Datong 92  
     Erenhot 360  
     Feicheng 374  
     Guizhou 92  
     Hailar 360  
     Henan 92  
     InnerMongolia 375  
     Junggar 361  
     Liaoning 92  
     OrdosBasin 360, 361  
     Qinhuangdao 425  
     QinshuiBasin 360  
     Shaanxi 92  
     Shandong 92, 374  
     Shenmu–Denfeng 92  
     Tianshan–Qilan Mountains 382  
     Turpan Depression 382  
     Xinjiang–Uygur 92  
     Yunnan 92  
 Chinese coal resources/reserves classification 224  
 Chlorides 110, 388  
 Chlorine 111, 126, 407, 408  
 Chlorite 109  
 Chromium 407  
 Chronostratigraphy 20, 61, 62  
 CIS (formerly USSR) 57, 71, 88, 332  
     Angren 373  
     Donbass (Donetsk) 89, 354, 361  
     Kamanskaya 373  
     Kansk–Achinsk 89  
     Kazakhstan 88, 208, 354, 361, 375  
         Ekibastuz 88, 361  
         Karaganda 361  
         Samarsky 361  
         Zavialor 361  
     Kuzbass (Kuznetsk) 89, 354, 361  
     Lisichansk 373  
     Moscow 89  
     Neryungri 89  
     Pechora 89, 354, 361  
     Russian Federation 88, 208, 217, 220, 354, 365, 368, 420  
     Shatsky 373  
     South Yakutsk 89  
     Taldinskaya 361  
     Tula 373  
     Ukraine 354, 361, 420  
     Yushno–Abinsk 373  
 Clarain 98  
 Clarite 103, 104, 105, 113  
 Clarodurite 104, 105  
 Classification, *see* Coal classification  
 Clastics to coal ratio 36–37

- Clay minerals 106, 109, 110
- Clean coal technology 422
- Cleat 44, 45, 46, 55, 176
- Coal
  - age 57–60, 64
  - anthracite 44, 45, 67, 68, 76, 83, 86–89, 91–94, 97, 116, 120, 177, 178, 235, 236, 237, 277, 313, 345, 361, 388, 404
  - attrital 98
  - banded 99
  - bituminous 18, 44, 45, 59, 67, 69, 73, 74, 76–96, 97, 113, 115, 116, 120, 176, 178, 235–237, 277, 343, 345, 348, 358, 361, 375, 378, 380, 396
  - black 75, 76, 80, 81, 92, 97, 237, 293, 328, 336, 337, 341
  - boghead 101, 118
  - boghead-cannel 98
  - bone 99, 178
  - bright 176, 177
  - brown 23, 25, 75, 76, 78, 80, 81, 89, 91, 95, 97, 113, 237, 283, 301, 328, 336, 337, 341, 344, 380, 415
  - cannel 98, 101, 118
  - cannel-boghead 98
  - charcoal-rich coal 100
  - coking 69, 74, 78, 80, 83, 84, 89, 93, 95, 96, 156, 164, 166, 404
  - description 178
  - dirty 177, 179
  - dull 176, 177
  - genesis 5
  - geographic distribution 64–96
  - hard 97
  - humic 97
  - impure 98
  - lignite 44, 45, 59, 64, 73, 74, 76–82, 84–86, 88, 90–96, 97, 99, 105, 120, 192, 235–237, 248, 250, 277, 301, 302, 336, 341, 345, 348, 404, 415, 416
  - matrix coal 99
  - meta-anthracite 117
  - mineral-rich coal 100
  - per-hydrous 136
  - physical properties 131
  - rank, *see* Rank
  - raw 424
  - saleable 424
  - sapropelic 97, 118
  - semi-graphite 117
  - steam (thermal) 69, 95, 156, 163, 165, 404, 433
  - subbituminous 59, 73, 74, 77, 81, 82, 85–87, 89, 90, 91, 93–96, 97, 99, 235–237, 277, 345, 359, 408
  - subhydrous 136
  - type 97
  - xylite-rich coal 100
- Coal analysis 162
- Coal Bedding Code 50
- Coal blocks, *see* Reserves
- Coal bursts 257, 326, 327
- Coal classification 136, 147
  - Australia 141, 145, 146
  - China (PRC) 149, 150
  - Europe 137
  - North America 136, 139
  - Russia 148, 149
  - South Africa 146
  - United Kingdom 136, 138, 140
  - United Nations 142, 143, 144, 146
- Coal codification system 137
- Coal contracts 431
  - indexed 432
  - spot purchase 431
  - term 431
- Coal cutting machine 314
- Coal description 178
- Coal exploration 169
- Coal export routes 430
- Coal exporting countries 431
- Coal face 314
- Coal fires 396
- Coal floor 152, 154, 178, 190, 192, 270, 313
- Coal gasification, *see* Underground coal gasification
- Coal handling 429
- Coal importing countries 431
- Coal industry 385
- Coal interburden/overburden ratio 37
- Coal liquefaction 1, 378, 379, 380
- Coal marketing 1, 430
- Coal markets 430
- Coal mine development 386
- Coal mine methane (CMM) 350, 351, 353, 354
- Coal outbursts, *see* Coal bursts
- Coal outcrop data card 153
- Coal oxidation, *see* Oxidation
- Coal panel, *see* Panel
- Coal pillars, *see* Pillar
- Coal plough 314
- Coal preparation 385, 423, 424
- Coal price 433
- Coal price index 433, 434
  - formulas 433
- Coal production 311, 329, 339
- Coal purchase 433
- Coal quality 29, 38, 203, 329, 377, 423
- Coal recovery 226
- Coal reserves, *see* Reserves
- Coal resource/reserve
  - classification, *see* Reserves; Resources
- Coal roof 152, 154, 178, 190, 192, 270, 313
- Coal sampling, *see* Sampling
- Coal seam gas, *see* CBM
- Coal splitting 29, 31, 35, 38, 42, 43
- Coal stockpiles, *see* Stockpiles
- Coal suppliers 432
- Coal thickness 29, 35
- Coal trade 2

- Coal transport, *see* Transport
- Coal unloaders 429  
   continuous 429  
   grab 429
- Coal use 401
- Coal weighing 427
- Coalbed methane (CBM) 1,  
   344–365, 418, 422  
   biogenic 345  
   thermogenic 345
- Coalbed methane drainage  
   351–355
- Coalbed methane generation  
   345
- Coalbed methane production  
   349, 355, 358–363
- Coalbed methane projects  
   356
- Coalbed methane resources  
   356, 358, 359
- Coalbed methane retention  
   346
- Coalification 19, 113, 116,  
   117, 118, 120, 121, 345,  
   346, 348, 375  
   biochemical 116, 117, 376
- Coarsening upwards  
   sequences 8
- Coastal barrier 6
- Coastal plain 6
- Cobalt 379
- Coke 403, 413, 414
- Coke production 413
- Coking properties 129
- Collinite 102
- Colombia 59, 87, 241, 328,  
   332, 429  
   El Cerrejon 337
- Colour 196
- Combined Heat & Power  
   Plant (CHP) 413, 414
- Combustible 97, 343
- Combustion 124, 343, 407  
   chamber 405
- Compaction 120, 390
- Compaction ratio 18
- Compensation 303
- Competent Person 211, 212,  
   225
- Composition Balance Index  
   (CBI) 115
- Compressibility 293
- Compression 45
- Computer 1, 179, 180,  
   192–206, 232  
   applications 201  
   contour plans 203–205  
   modelling 201, 324  
   models 201  
   block 201, 203  
   cross-section 203  
   graphics 324  
   gridded 201, 203
- Computer programs 227,  
   231
- Concave slope 392
- Concrete 409
- Condenser 410
- Conductivity 245
- Cone of depression 291, 292,  
   297, 301, 303, 306
- Conglomerate 10, 12, 13,  
   175, 198
- Constant Head Test 296
- Contamination 387
- Continuous miners 316, 318
- Contour maps 329
- Controlled retracting injection  
   point (CRIP) 367, 369
- Convex slope 392
- Conveyors 334, 336, 341, 425
- Copenhagen Accord 2, 421
- Cordaites 20
- Core 188–192
- Core barrel 188
- Core box 189, 191
- Core logging 190–192
- Core logging shed 192
- Core logging sheet 192, 193
- Core losses 191
- Core recovery 198
- Core sizes 188
- Cored boreholes, *see*  
   Boreholes
- Corpocollinite 102
- Corpohumite 106
- Correlation 28
- Corrosion 389
- Cost and Freight (C&S) 433
- Cost, Insurance & Freight  
   (CIF) 433, 434
- Costa Rica 87
- Costs 331, 333, 334, 340
- Cratonic basin 61
- Cretaceous 21, 23, 57, 60, 73,  
   74, 79, 81, 82, 87, 89,  
   92, 96, 118, 348, 376,  
   381
- Crevasse-splay 8, 10, 18, 31
- CRIRSCO 208, 209, 211–213,  
   220–225
- Critical width 399
- Cross belt sampler 157
- Cross borehole seismic, *see*  
   Seismic
- Crucible swelling number  
   (CSN), *see* Free  
   swelling index
- Crystalline basement 258
- Cumulative floats curve 133,  
   134
- Cumulative sinks curve 133,  
   134
- Cuticles 102
- Cutinite 102, 103
- Cycle 6
- Cyclothem 6
- Czech Republic 75, 257, 326,  
   354
- d**
- Darcy's Law 290, 293
- Data collection 169
- Debituminisation 116
- Deforestation 391
- Deformation 41, 325
- Degassing 354–355
- Dehydration 116
- Delivered Ex-Ship (DES) 433
- Delta/deltaic 5, 6, 8, 18
- Demurrage 432
- Denmark 76
- Dense medium 424
- Densimetric curve 134, 135
- Densinite 106
- Density 132, 244, 246
- Densospore 109



- Depositional models 5, 6  
 Depression 399  
 Depressurisation 293  
 Depth 235, 347  
 Desmocollinite 102  
 Desorbed 356  
 Detrital minerals 106  
 Devonian 11, 19, 57, 74  
 Dewatering 299–309  
 Diachronous 27  
 Diagenesis 44, 97  
 Diapir 43, 44  
 Diapiric intrusion 43, 44  
 Diesel 391  
 Diffraction 253  
 Digital recording unit 265  
 Dilution 211  
 Dioxins 405, 407  
 Dips 50, 153–155, 178, 199, 277–278  
 Dipmeter logs, *see* Geophysical logs  
 Directionally drilled linking 367  
 Discard 134  
 Discharge 289, 428  
 Discharge hydrograph 289  
 Discharge point 298  
 Discontinuities 199, 292, 325, 329  
 Dispatch systems 425  
 Distillate oil 379  
 Distributary mouth bar deposits 8  
 Diversion 392  
 Doha Amendment 421  
 Dolerite 86, 260  
 Domed peat 20, 25  
 Domestic use 415  
 Double tube core barrel 188  
 Dozer, *see* Bulldozer  
 Dragline 329, 330, 331, 335  
 Drainage 302, 327, 394  
 Drawdown 291, 301  
 Drill casing 183  
 Drill holes, *see* Boreholes  
 Drill sites 28  
 Drilling 183–190, 327  
 Drilling bits 184, 185, 188  
 Drilling fluid 184, 185  
 Drilling programme 183, 184, 329  
 Drilling rigs 184–190  
   air flush 184, 185, 289  
   mud flush 184, 185, 188, 289  
   portable 189, 190  
   rotary 184, 188  
   wireline 188  
 Drivages 314, 320  
 Drum shearer 314, 315, 316  
 Dry ash free basis 122  
 Dry basis 122  
 Dry bottom boiler, *see* Boilers  
 Dry mineral matter free basis 123  
 Dump trucks 332, 333, 334  
 Durain 45, 98  
 Durban Platform 421  
 Durite 104, 105  
 Duroclarite 104, 105  
 Dust 397, 416  
 Dust suppression 398  
 Dyke 260–262, 313  
 Dynamite 248, 250
- e**
- Economic feasibility 207  
 Economic limits 329  
 Economics 315  
 Ecosystem 15  
 Ecuador 87  
 Egypt 382  
 Electric arc furnace 413, 414  
 Electric logs, *see* Geophysical logs  
 Electrical conductivity 245  
 Electrical resistivity 243–245, 262  
 Electricity generation 77, 79, 82, 83, 85, 93, 95, 125  
 Electromagnetic survey 263  
 Electrostatic precipitators 411  
 Elementary ash-curve 134  
 Embryophytes 19  
 Emissions 2, 404–407, 412, 419, 420  
 Emissions levels 407, 419  
 Emissions standards 380, 406, 419, 420  
 Emsian 18  
 Encapsulites 131  
 End Wall 306  
 Engineering characteristics 279  
 Environment 289, 365, 385  
 Environmental constraints 422  
 Environment of deposition 5  
 Environmental Impact & Social Assessment (EISA) 387  
 Environmental legislation 398  
 Environmental planning 386  
 Environmental regulations 418, 419  
 Eocene 18  
 Ephemeral streams 289  
 Erosion 371  
 Escalation 432  
 Ethane (C<sub>2</sub>H<sub>6</sub>) 343, 344  
 Ethiopia 85  
 Europe 57, 58, 67, 75, 208, 212, 311, 318, 319, 334, 365, 375, 414, 417, 418, 428  
 European Union (EU) 374, 419, 420, 434  
 Eustacy 6  
 Eustatic 25, 26  
 Eutrophic 16  
 Evaporation 289, 295  
 Evapotranspiration 16  
 Excavation 396  
 Excavator, *see* Shovel  
 Exhaust gases 405  
 Exinite 98, 102, 117  
 Exothermic 366, 396  
 Exploration 169, 170, 171, 184, 329, 386  
   results 209  
   target 209  
 Explosive 248, 336  
 Exporting countries 430  
 Exporting ports 430

- Exporting routes 430  
 Extensometers 325  
 Exudatinite 102
- f**
- Facies 8, 11, 27  
 Facies correlation 27  
 Facies maps 29  
 Fair Treatment System 433  
 Falling head test 296  
 Falling stage 26, 27  
 Falling stream sampler 157, 158  
 Falling weight 256  
 Far East 72, 89  
 Faults 40, 42, 43, 44, 45, 46, 47, 48, 250–252, 313  
   drag 46  
   lag 44, 48  
   thrust 47  
 Fauna 175  
 Feasibility Study 225  
 Fen 15, 16  
 ‘Fence’ diagram 28  
 Ferric hydroxide 388  
 Ferric oxide 388  
 Ferric sulphate 388  
 Ferrous sulphate 388  
 Ferruginous waters 388  
 Field equipment 172  
 Field traverses 173, 175, 176  
 Fieldwork 171  
 Fining upwards sequences 8  
 Fireclays, *see* Seatearth  
 ‘Firedamp’ 343  
 Fischer Assay 129  
 Fischer–Tropsch synthesis 378, 379  
 Fixed carbon 125  
 Flame structures 40  
 Flash pyrolysis 379  
 Float sink tests 133–135  
 Floating swamp 15  
 Floodplain 18, 22  
 Flooding 18, 31, 313  
 Floodtide delta deposits 8, 9  
 Floor, *see* Coal  
 Floor heave 299, 301, 303  
 Floor rolls 29, 34
- Flora 20–24  
 Flow diagram 170, 184, 206  
 Flow nets 329  
 Flow rates 309  
 Flue gas 407, 414  
 Flue gas desulphurisation (FGD) 405, 409, 410  
 Fluidised bed combustion (FBC) 411  
   bubbling (BFBC) 411, 412  
   circulating (CFBC) 411, 412  
   pressurised 411  
 Fluorescence microscopy 118, 131  
 Fluorinite 102  
 Fluvial 6, 11, 13, 19, 31, 381  
 Fluvialite 5, 6  
 Fly ash 405, 408, 410  
 Focused electric log, *see* Geophysical logs  
 Folds 42, 44, 50  
   zig zag 53  
 Foraminifera 175  
 Force majeure 432  
 Foredeep basin 61  
 Fossil content 175, 192  
 Fouling 126  
 Fracture logging 194, 195, 199  
 Fracture spacing index 194, 198  
 Fracture inducement 357  
 Fracture orientation 320  
 Fracture pressure 320  
 Framework Convention on Climate Change (FCCC) 420  
 France 344, 354, 362  
 Free on Board (FOB) 433  
 Free swelling index (FSI) 128  
 Freezing method 312  
 Freight charges 433  
 Froth flotation 424  
 Fusain 98  
 Fusinite 98, 102, 103, 113  
 Fusite 104, 105
- g**
- G value test 131  
 Gallium 408  
 Gas 343, 356, 371  
 Gas drainage, *see* Drainage  
 Gas emissions 352, 354  
 Gas seepage 401  
 Gas to liquids (GTL) 380  
 Gasification 1, 367, 374  
 Gasoline 379  
 Gassy 314, 315, 354  
 Gaussian copula 231  
 Gelification 99, 100  
 Gelinite 106  
 Geochemical 383  
 Gelocollinite 102  
 Gelovitrinite 106  
 Geological assurance 207, 214, 221  
 Geological data/database 271  
 Geological losses 207, 212, 232  
 Geological maps 171, 173  
 Geological survey 171  
 Geological symbols 173–174  
 Geophone 248, 253  
 Geophysical logging unit 270  
 Geophysical Logs (Geologs) 28, 39, 243, 269  
   calibration 271  
   calliper 271, 276, 280, 281, 286  
   density 271, 273, 282, 283, 285–287  
   dipmeter 271, 277–279  
   electrical 277  
   electromagnetic 243  
   gamma ray 271–273, 282, 284–287  
   gamma spectrometry 276  
   neutron 271, 274–276  
   radiometric 243  
   resistivity 271, 273, 277, 287  
   sonic 271, 278–280, 282, 284, 287  
   temperature 280

- Geophysics 243, 320  
 Georgia 76, 77  
 Geostatistics 229  
 Geotechnical 36, 194  
 Geotechnical logging sheet 200  
 Geothermal gradient 120  
 Germanium 408  
 Germany 76, 338, 341, 354, 362, 379, 418  
 Gieseler plastometer 129, 131  
 Glacial deposits 334  
 Glacio-eustatic 27  
 Global energy 403  
 Global positioning systems (GPS) 171, 179, 180, 201, 246  
 Glossopteris 22  
 Goaf 320, 328, 352  
 Gob gas 351, 353  
 Gondwana 18, 19, 22, 44, 46, 57, 58, 80, 84–86, 88, 95, 102, 111, 118, 135, 156, 177, 258, 259, 378, 382  
 Graben 46  
 Grader 333  
 Grainsize 198  
 Graphic display 175, 179, 180  
 Graphitisation 116  
 Gravimeter 257  
 Gravity 257–259  
 Gravity sliding 41  
 Gravity survey 257, 269  
 Gray–King coke type 129, 130  
 Greece 77, 380  
 Greenfields exploration 169  
 Greenhouse gases (GHG) 2, 365, 386, 416, 417, 420, 422  
 Greenland 77  
 Grindability 404  
 Ground magnetic 243  
 Ground penetrating radar 262, 269  
 Groundwater 15, 16, 17, 20, 21, 27, 111, 289–307, 329, 356, 367, 387, 389, 415  
     contour maps 298  
     flow 292, 298  
     network 297  
     rebound 307–309  
 Groundwater contamination 374  
 Groundwater table, *see* Water table  
 ‘Gun’ energy source 248  
 Gymnosperm 20, 23  
 Gypsum (CaSO<sub>4</sub>·2H<sub>2</sub>O) 111, 409
- h**
- Hardgrove grindability index (HGI) 132  
 Hardness 132  
 H/C atomic ratio 377  
 Head 291  
 Health and Hygiene 344, 415, 416  
 Heat rate 406  
 Heavy metals 419  
 Helical scan 280  
 Helium 294  
 Helsinki Protocol 419  
 Herrin No. 6 seam 65, 73  
 Heteroatoms 378  
 Heterosporous 20  
 High and dry placement method 390  
 High resolution seismic, *see* Seismic  
 Highstand systems tract 26, 27, 28  
 Highwall 46, 47, 268  
 Highwall depth 233  
 Highwall mining 338  
     auger 338, 339  
     continuous 338, 339  
 Higher heating value, *see* Calorific value  
 Hilt’s law 120  
 Hole to surface Seismic, *see* Seismic  
 Holland (The Netherlands) 77  
 Homosporous 20  
 Humification 15, 17  
 Huminite 102, 104, 105, 113, 117  
 Hungary 77, 420  
 Hydration 135  
 Hydraulic conductivity 290, 293  
 Hydraulic gradient 290–291  
 Hydraulic pressure 290  
 Hydraulic supports 314, 315  
 Hydrocarbons 346, 375–377  
 Hydrochloric acid 409  
 Hydrodynamic pressure 290, 303  
 Hydrofluorocarbons 419  
 Hydrofracking 367  
 Hydrofracturing 323  
 Hydrogen 19, 113, 117, 125, 275, 375–379, 406  
 Hydrogen chloride 409  
 Hydrogen donor 379  
 Hydrogen index 276  
 Hydrogenation 378  
 Hydrogeological barrier 298  
 Hydrogeological characteristics 289  
 Hydrogeological cycle 289  
 Hydrogeological data 295  
 Hydrogeological model 295  
 Hydrogeological properties 289  
 Hydrogeological relationships 295  
 Hydrogeological studies 299  
 Hydrogeology 15, 289–309, 313  
 Hydrology 289  
 Hydrolysis 135  
 Hydrophilic 424  
 Hydrophobic 424  
 Hydrostatic Pressure 290
- i**
- Iceland 420  
 Igneous 44

- Igneous intrusions 40, 53,  
     74, 78, 90, 151, 270  
 Igniter-burner 367  
 Ignition wells 367  
 Illite 109  
 Impermeable 290, 303  
 Importing countries 430, 431  
 Impulse source 246  
 Increments 159–160  
 India 58, 84, 222, 224, 258,  
     259, 262, 268, 303, 328,  
     332, 338, 362, 375, 379,  
     395, 403, 413, 415, 426,  
     429  
     Jharia 84, 362, 396  
     Orissa 84  
     Raniganj 84, 362  
     Singrauli 362  
     Talcher 84  
 Indian subcontinent 69  
 Indonesia 24, 43, 59,  
     262–263, 328, 332, 363,  
     375, 379, 381, 382, 397,  
     426, 427, 429  
     Java 90  
     Kalimantan 90, 262–263,  
     363, 397, 398  
     Berau 90, 429  
     Kutei Basin 52, 363,  
     381, 382  
     Mahakam Delta 25, 382  
     Makassar Strait 43  
     Senakin–Tanah Grogot  
     90  
     Tanjung 90  
     Tanjung Bara 426  
     Sulawesi 90  
     Sumatra 90, 363  
     Bengkulu 90  
     Bukit Asam 90  
     Ombilin 90  
     West Irian 90  
     Bintuni 90  
     West Kalimantan 91  
 Industrial Emissions Directive  
     420  
 Inertinite 59, 98, 102, 105,  
     106, 113, 114, 115, 117,  
     118, 135, 378, 382  
 Inertite 102  
 Inertodetrinite 102, 103  
 Inerts 114  
 Infiltration 306  
 Inflow 299–302  
 Infrared radiation 407  
 Injection wells 367, 368, 369,  
     374  
 In-seam roadways 320  
 In-seam seismic (ISS), *see*  
     Seismic  
 Interactive graphics 324, 330  
 Interburden 29, 336, 389  
 Interdistributary bay 10, 12  
 Intergranular flow 292  
 Integrated gasification  
     combined cycle (IGCC)  
     plants 417  
 Intermittent streams 289  
 International Committee for  
     Coal and Organic  
     Petrology (ICCP) 99  
 International Finance  
     Corporation (IFC)  
     420  
 International Liquids from  
     Coal (LFC) 380  
 International Organisation for  
     Standardisation (ISO)  
     137  
 International traded coal  
     403  
 Interpretive data 209  
 Investment capital 314  
 Iran 84  
 Ireland 77, 212  
 Iron 106  
 Iron & Steel production 403,  
     404, 413  
 Iron pyrite, *see* Pyrite  
 Ironstone 53  
 Isopach maps 35, 38  
 Isotopes, *see* Radioactive  
 Italy 77, 419
- j**
- Japan 91, 379, 413, 420  
 Japanese Benchmark Price  
     433
- Jig, *see* Baum jig  
 Joints 44, 199, 270, 346  
 JORC Code 208, 209, 211,  
     212, 222, 223  
 Jurassic 57, 58, 60, 76,  
     85, 91, 94, 361, 377,  
     382
- k**
- Kalahari Group 82  
 Kaolinite 109, 110, 111  
 Karroo 80, 81, 260  
 Kellog reactor 379  
 Kerogens 376, 377  
 Kerosene 391  
 Kings–Maries–Crossley  
     Formula (KMC) 123  
 Kladno coalfield 250  
 Korea (Dem. Peoples Rep)  
     90, 241  
 Korea (Rep) 94  
 Kosovo 77, 78  
 Kriging 77, 78  
 Kyoto Protocol 2, 417, 420,  
     422
- l**
- Lacustrine 19  
 Lag deposits 18  
 Lagoons 8  
 Lakes/lacustrine 17, 375,  
     376, 381  
 Landfill 395  
 Land restoration 328  
 Landsat imagery 180, 181  
 Landscape 387  
 Landslides 313  
 Langmuir isotherm 246  
 Laos 91  
 Laterite 258  
 Laurasia 58, 59  
 Lawrence Livermore  
     Laboratory 367  
 Lead 407, 408  
 Legal tenure 225  
 Lepidodendron 20–21  
 Lidar 181  
 Lignin 19  
 Lignite, *see* Coal

- Lime 409  
 hydrated 409  
 quick 409
- Limestone 6, 9, 28, 175, 244,  
 252, 293
- Limestone/gypsum wet  
 scrubbing process  
 409
- Liptite 102
- Liptinite 102, 103, 106, 113,  
 115, 118, 375, 376,  
 382
- Liptodetrinite 103
- Liquefaction 40
- Liquid fuel 375
- Liquid solvent extraction  
 (LSE) 380
- Liquidated damages 432
- Lithofacies 26, 31, 34,  
 36
- Lithofacies maps 29, 31
- Lithological logging 284
- Lithology 6, 192, 193,  
 277–278
- Lithosphere 126
- Lithostratigraphy 20
- Lithotype 6, 28, 31, 32, 35,  
 45, 99
- Loading 425–427
- Locality 153
- Locomotives 344
- Long Range Trans-Boundary  
 Air Pollution (LRTAP)  
 419
- Long spaced density log, *see*  
 Geologs
- Longwall mining, *see* Mining
- Losses, *see* Geological
- Low wall depth 233
- Lower delta plain 7, 8, 10,  
 11, 12
- Lower heating value, *see*  
 Calorific value
- Lowstand 26, 27
- Lung cancer 416
- Lurgi-gasifier 379
- Lycophytes 22
- Lycopsida 20
- Lycospore 109
- m**
- ‘M’ curve 134
- Macerals 23, 25, 58, 97, 376
- Macrinite 102, 103
- Macrostructural 41
- Magnesium 388
- Magnetic anomaly 260, 262
- Magnetic profile 263
- Magnetic properties 245
- Magnetic survey 259, 260
- Magnetic susceptibility 244,  
 245
- Magnetite 245, 262
- Malagasy Republic 81
- Malawi 81
- Malaysia 24, 91
- Mali 81
- Mangrove 15, 20
- Manholes 343
- Marcasite (FeS<sub>2</sub>) 388
- Marine fauna 6, 28
- Marine regression 26–28
- Marine transgression 26–28
- Marker beds 175
- Market share 430
- Marketing 423, 430
- Marsh 15, 16
- Mass strength, *see* Rock  
 strength
- Material strength, *see* Rock  
 strength
- Mechanical strength 131,  
 132
- Megaspores 20, 21, 22
- Melange 42
- Mercury 408
- Mercury porosimetry 294
- Mesotrophic 16
- Mesozoic 74, 75, 78, 86, 87,  
 90, 91, 92, 94, 95, 113
- Metallurgical industry 127
- Metamorphic 117, 244, 245,  
 345
- Methane (CH<sub>4</sub>) 117, 294,  
 343–364, 375, 388, 399,  
 419  
 thermogenic 345
- Methane drainage 351
- Methane monitor 344
- Methane synthesis reaction  
 366
- Mexico 87, 355
- Micrinite 102, 103
- Microelectronics 169
- Microgravity 243
- Microlithotypes 102, 103,  
 105, 109
- Micropores 356
- Micro-resistivity 277
- Microspores 20, 21
- Microstructural 40
- Middlings 134, 135
- Migration interval 255
- Mine batters 335
- Mine boundaries 313
- Mine design 328, 329
- Mine development 314
- Mine drainage 390
- Mine layout 320, 324, 326
- Mine operation 312
- Mine planning 329
- Mine plans 324
- Mine transport 334–337
- Mine waste 388
- Minerals 109–110
- Mineral matter 25, 97
- Mineral precipitation 53, 55
- Mineralisation 189
- Mineralogy 193
- Mineralotrophic 16
- Mining 311–341  
 auger 338  
 continuous 316–318  
 conventional 316  
 highwall 338  
 auger 338, 339  
 continuous 338, 339  
 longwall 265, 266, 314,  
 316, 317, 320, 326, 340  
 advance 314, 315  
 retreat 314, 315, 320  
 multi-seam 233, 320  
 opencast, *see* Opencast  
 room and pillar 313, 314,  
 316, 318, 320, 322, 324,  
 340  
 shortwall 315  
 strip 335



- truck and shovel 331  
 underground, *see*  
 Underground mining  
 up-dip 299
- Mining equipment 330  
 Mining hazards 313  
 Mining method 311  
 Mining panel 314, 315  
 Mining systems 314  
 MINI-SOSIE 248, 253, 264  
 Miocene, *see*  
 Palaeogene–Neocene
- Miospore 105, 108  
 Mire 15–19, 23, 25, 31, 44, 58  
 Mobile loader 316  
 Modifying factors 209  
 Moisture 122, 123, 124  
 adventitious 124  
 as delivered 124  
 air dried 124  
 equilibrium 124  
 inbed 124  
 inherent 125  
 in situ 124  
 as received 124  
 surface 124  
 total 124
- Moisture content 123, 396  
 Moisture holding capacity  
 124
- Mongolia 91, 208  
 Monitoring 344, 368  
 Montenegro 78, 336  
 Morgantown Energy Centre  
 273
- Morocco 81  
 Moving wall oven test 156  
 Mozambique 82  
 Mudstone 8–12, 18, 28, 33,  
 34, 39, 40, 43, 47, 48,  
 64, 175, 176, 195, 198,  
 244, 253, 271, 292, 293,  
 318  
 marine 28, 61, 271
- Myanmar (Burma) 92
- n**
- NAEN 219, 221, 223  
 Namibia 82
- Neutrons 275  
 Neutron log, *see* Geophysical  
 logs
- Netherlands, *see* Holland  
 New Zealand 96, 262, 264,  
 344, 376, 377, 382, 420
- Nickel 407, 408  
 Niger 82  
 Nigeria 82  
 Nitrates 110  
 Nitrogen (N<sub>2</sub>) 19, 113, 125,  
 156, 343, 345, 346, 378  
 Nitrogen oxides (NO<sub>x</sub>)  
 405–408, 410–412  
 Nitrous oxide (N<sub>2</sub>O) 419  
 North America, *see* Canada;  
 United States of  
 America
- Norway 382, 420  
 Nuclear Well Logging  
 Instruments 245
- o**
- Observation well 291, 292,  
 295, 297, 298
- O/C atomic ratio 117  
 Oil 375, 376, 377, 393  
 Oil shales 101  
 Old mine workings 253  
 Oligotrophic 16  
 Ombrogenous peat 15, 16,  
 18, 24, 25  
 Ombromorphic peat 16  
 Ombrotrophic peat 20  
 On-line analysers 163  
 Opencast 33, 36, 73, 76, 78,  
 90, 152, 232, 233, 235,  
 256, 268, 293, 296, 298,  
 299, 303–307, 311, 312,  
 328, 331–336, 389, 394,  
 395, 396
- Openhole drilling 184, 193  
 Openhole logging 193, 194  
 Open pit, *see* Opencast
- Organisation for Economic  
 Co-operation &  
 Development (OECD)  
 411
- Organoclastic 97
- Outcrop mapping 172  
 Overburden 29, 232, 328,  
 330, 334, 335, 389,  
 395  
 Overburden ratio 232, 233,  
 235  
 Overcoring 323  
 Overhead ropeways 392  
 Overhead shields 314, 315  
 Oxidants 419  
 Oxidation 135, 153, 396,  
 397  
 Oxidation reaction 365  
 Oxygen 117, 126, 344, 378  
 Oxygenation 390  
 Ozone 407
- p**
- Packer test 296  
 Pakistan 84, 389  
 Tar 389
- Palaeogene–Neogene 18, 21,  
 23, 43, 44, 46, 47, 53,  
 57–60, 73–75, 77–82,  
 84–96, 107, 111, 176,  
 248, 292, 301, 348, 350,  
 376, 377, 382
- Palaeobotanical 183  
 Palaeogeographic 31, 59, 60  
 Palaeostrike 43  
 Palaeozoic 5, 20, 21, 75, 78,  
 85, 90, 91  
 Paludification 17  
 Panel 314, 315, 324  
 Pangaea 58  
 Paralic 22  
 Parasequences 25  
 Paris Agreement 414, 421,  
 422  
 Parr formula 114, 123  
 Particle size distribution, *see*  
 Size distribution
- Particulates 405, 406, 408,  
 410, 414
- Partings 18, 31, 75  
 Passive seismic 257  
 Patchawarra Formation  
 36–37  
 Payloader 427

- Peat 6, 11, 15–25, 32, 42, 44,  
91, 99, 100, 116, 117,  
207, 293  
amorphous 18  
pseudo-fibrous 18  
fibrous 18
- Peatification 116
- Pectin 19
- Penalties 432
- Pennsylvanian 6, 8, 20, 24,  
61, 69
- Pepper pot structure 52
- PERC 208, 212, 223
- Percolation 390
- Perfluorocarbons 419
- Permeability 44, 290, 292,  
296, 298, 309, 346, 348,  
367
- Permeable 290
- Permian 19, 21, 22, 23, 57,  
59, 80, 92, 105, 113,  
361, 381
- Persistent Organic Pollutants  
Regulations 419
- Peru 87
- Petrographic 25
- Petroleum 375, 381
- pH value 409
- Philippines 93, 382
- Phosphorite 110
- Phosphorus 110, 113, 126
- Photogeological symbols  
181–183
- Photogrammetric maps 171
- Phreatic zone 290
- Piezometer 291, 295–298,  
302
- Piezometric surface 290, 307
- Pillar 314, 316, 318, 320, 326,  
339, 396, 399, 401  
configuration 324  
extraction 316
- Pit optimisation programmes  
311
- Planimeter 227
- Plant efficiency 410
- Plastic Layer Test 129
- Plastic membrane 390
- Plate tectonics 57, 318
- Pleistocene 258
- Pliocene, *see*  
Palaeogene–Neogene
- Ply 152
- Pneumoconiosis 416
- Point bar deposits 19
- Point load test 195, 196
- Points of measurement/  
observation 209, 212,  
226, 227, 229
- Poland 78, 251, 326, 354,  
361, 365, 379, 389, 420  
Belchatow 44, 328, 338  
Lower Silesia 78, 361  
Lublin 78, 251, 361  
Upper Silesia 78, 251, 257,  
361, 389
- Political instability 385
- Pollutants 387, 405, 413
- Pollution 390
- Polychlorinated  
dibenzofurans (PCDF)  
407
- Polychlorinated  
dibenzo-para-dioxins  
(PCDD) 407
- Polygon 228, 229, 232
- Pore 294, 346
- Pore distribution 347
- Porosimetry 294
- Porosity 44, 275, 287, 293,  
294, 346
- Porous medium 290
- Port charges 432
- Portugal 78
- Potassium 245
- Potentiometric surface 290,  
291, 301
- Power generation, *see*  
Electricity
- Power Purchase Agreement  
(PPA) 432
- Power station 408, 426
- Powered supports, *see*  
Hydraulic supports
- Precambrian 82
- Precipitation 16, 18, 391
- Prefeasibility study 225
- Pressure 59, 120
- Pressurisation field 297
- Pridolian 20
- Probing distance 269
- Prodelta 8
- Production, *see* Coal;  
Methane; Water
- Production boreholes, *see*  
Wells
- Propane (C<sub>3</sub>H<sub>8</sub>) 346
- Proximate analysis 123
- Pteridophytes 22
- Pteridosperms 22
- Pteropsida 20
- Pulse radar 264, 269
- Pulverised fuel (pf)  
boiler 405, 410
- Pumice 244
- Pumping 303
- Pumping test 297
- Pumping well 291, 292, 299,  
303
- Pycnometer 132
- Pyrite 25, 55, 98, 107, 135,  
176
- Pyrolysis 367, 375, 377,  
378–380
- Pyrrhotite 245
- q**
- Quartz 55, 106, 110
- Quaternary 57
- Quebec 20
- Quenching 396
- Queue anticline 52
- r**
- Radiation 271  
natural 271
- Radioactive 264  
isotopes 377, 389  
surveys 264
- Radioactivity 121, 245, 246
- Radiometric 243, 245–246,  
*see also* Geophysical  
logs
- Radium 389
- Radon 344, 416
- Rail, *see* Transport
- Rainfall 295

- Rain gauge 295  
 Rain water 15  
 Raised bog, *see* Bog  
 Raised swamp, *see* Swamp  
 Rank 5, 11, 15, 44, 53, 67, 73, 78, 85, 97, 111, 115–119, 126, 131, 132, 151, 275, 313, 346, 347, 359, 362, 363, 376, 396, 425  
 Rank gradient 120  
 Reactive 115  
 Rebound, *see* Groundwater  
 Recent 23, 258  
 Recharge 290  
 Reclamation 392  
 Reconnaissance 171  
 Recrystallisation 244  
 Redox potential 15  
 Reduction reaction 366  
 Reflectance, *see* Vitrinite reflectance  
 Reflection coefficient 245  
 Reflection seismic, *see* Seismic  
 Refraction seismic, *see* Seismic  
 Regression 27  
 Rehabilitation 341  
 Relative density 132  
 Remote sensing 180  
 Reserve base 219  
 Reserves 1, 203, 211, 328  
   assessment 222, 226  
   calculations 227, 228  
   extractable 235  
   indicated 219  
   inferred 219  
   marginal 224  
   marketable 211, 213  
   measured 219  
   mineable in situ 233, 235  
   probable 211, 212  
   proved 211, 212  
   proven 211, 212  
 Reserve economics 235  
 Reservoir 356  
 Reservoir enhancement 357  
 Reservoir pressure depletion 356  
 Resin 102  
 Resinite 102, 103  
 Resistivity, *see* Electrical resistivity  
 Resources 1, 169, 203, 209, 211  
   analysis 207  
   assessment 226  
   calculation of 227, 228, 229  
   categories 207  
   identified 216  
   indicated 210, 212, 219, 222, 229  
   inferred 209, 212, 219, 222, 229  
   measured 210, 212, 219, 229  
   original 216  
   proved recoverable 171  
   remaining 216  
   subeconomic 219  
 Resource/reserve block 231  
 Resource/reserve classification 208–225  
 Resource/reserve maps 226  
 Resource/reserve reporting 225, 226, 235  
 Resource/reserve tonnages 227  
 Restoration 394  
 Retrofit 410  
 Revegetation 390, 396  
 Reverse combustion linking 367  
 Rhaetic 86  
 Rheotrophic 15, 16, 20  
 Rising head test 296  
 Rivers 428–429  
   Barito 429  
   Berau 429  
   Danube 428  
   Mahakam 429  
   Mississippi 428  
   Missouri 428  
   Rhine 428  
 Roadway 314, 325, 326  
 Rock bolt, *see* Roof bolt  
 Rockbursts 257  
 Rock colour charts 197  
 Rock-Eval analysis 377  
 Rock mass rating (RMR) 201  
 Rock mechanics 318  
 Rock quality designation (RQD) 194, 198  
 Rock strength 195, 197, 286  
 Roga index test 129  
 Roller bits 185  
 Romania 78, 380  
 Room and pillar, *see* Mining  
 Roof, *see* Coal  
 Roof bolt 316, 318, 326  
   column resin 318  
   mechanical point anchor 318  
   strain gauge 326  
 Roof bolt machine 320  
 Roof collapse 398  
 Roof conditions 320  
 Roof support 318  
 Rootlet bed, *see* Seatearth  
 Rosary structure 52  
 Rotary drum shearer, *see* Drum shearer  
 Rotational slides 392  
 Ruhr dilatometers 129  
 Run of mine 151, 213  
 Run off 290
- S**
- Sacrificial roadways 320  
 Saddle Reefs 52  
 Safety 171, 415  
 Safety lamp 344  
 Sales, *see* Marketing  
 Saline waters 388, 418  
 Salt dome 44  
 Sample bags 153, 155, 156  
 Sample locations 153  
 Sample logging sheets 155  
 Sample preparation 162–166  
 Sample size 156  
 Sample storage 156  
 Sampling 151–166  
   *ex situ* 151, 157–162  
   hand 157  
   random 157  
   ships 160

- Sampling (*contd.*)  
   stockpile 160, 162  
   stop belt 160, 162  
   stratified random 157  
   systematic 157  
   wagons 160  
 in situ 151–156, 162  
   bulk 156, 162  
   channel 151–154  
   channel ply 152, 153, 154  
   core 154, 155, 162, 163  
   cuttings 155  
   grab 151  
   gross 160, 161  
   pillar 154  
   specimen 155  
 SAMREC 213  
 SAMVAL 203  
 Sandstone 6–13, 30, 33, 34, 38–40, 43, 52, 61, 63, 175, 176, 178, 193, 195, 198, 244, 253, 271, 277, 284, 292, 293, 315, 318, 343  
 SASOL 375, 378  
 Saturated Bed Thickness 291  
 Saturation Zone 290  
 Satellite imagery 171, 180  
 Schmidt hammer 195, 196  
 Sclerotinite 102  
 Scoping study 225  
 Screens 424  
 Sea water washing 409  
 Sealing walls 303, 306  
 Seam, *see* Coal  
 Seatearth 6, 19, 29, 293  
 Second Sulphur Protocol 419  
 Sedimentary basins 260  
 Sedimentary dykes 40  
 Sedimentation 5, 26, 28  
 Sedimentation pond 392  
 Seismic 25, 246–257  
   cross borehole 253–255  
   high resolution 250, 251  
   hole to surface 253, 254  
   in-seam (ISS) 243, 248, 264–266, 268  
   micro-seismic 268  
   passive 257  
   reflection 246, 248, 257  
   reflection coefficients 245  
   refraction 256  
   sections 249, 267  
   shallow 243, 249  
   surveys 246–248, 252, 253, 257, 265, 266, 418  
   velocity 244, 246, 254, 257, 258  
   vertical seismic profile (VSP) 253, 254  
 Seismic data 265  
 Selective catalytic reduction (SCR) 410  
 Selenium (Se) 407, 408  
 Semifusinite 102–103  
 Sequence stratigraphy 25, 27, 28  
 Serbia 78, 301–304, 380  
 Seyler's classification 136, 138  
 Seyler's coal chart 137  
 Shaft 312, 399  
   downcast 312  
   sinking 312, 313  
   upcast 312  
   vertical 312  
 Shear 47  
 Shear zones 306  
 Shearer, *see* Drum shearer  
 Ship 397, 429  
   cape size 429  
   geared vessels 429  
   Handymax 429  
   Handysize 429  
   Panamax 429  
 Ship loader 425  
 Shipment 427  
 Short spaced density logs, *see* Geophysical logs  
 Shot point 248  
 Shovel 331–333  
   backhoe 331–333  
   electric 331–333, 335–336  
   front shovel 331, 332  
   hydraulic 331–333, 335–336  
   Shuttle cars 316  
 Side looking airborne radar (SLAR) 181, 183  
 Side wall 344  
 Siderite 8, 53, 107  
 Sigillaria 20  
 Silicosis 416  
 Sills 52, 260, 261  
 Silo 425  
 Siltstone 8–13, 34, 64, 175, 176, 178, 198, 271, 292, 293, 318  
 Silurian 11, 19  
 Size 158, 161, 163, 164, 396  
 Size distribution 133  
 Slagging 110, 410  
 Slickensides 199  
 Slope 394  
   stability 394  
 Slovakia 375  
 Slovenia 375  
 Slumping 41, 178  
 Smectite 109  
 Smothering 396  
 SNOX 410  
 Sodium 388  
 Sodium bicarbonate 410  
 Sodium chloride 245  
 Sodium sulphate 409  
 Sofia Protocol 419  
 Soil erosion 391  
 Solar radiation 136  
 Solvent 379  
 Solid core recovery (SCR) 194  
 Sonde 270  
 Sorting 192  
 Sound proofing 415  
 Sound waves 278  
 South Africa 146, 177, 208, 213, 241, 258, 260, 314, 318, 320, 332, 337, 339, 375, 406  
   Richards Bay 427  
 South America, *see* Central and South America  
 Spain 374, 380, 417  
 Specific energy, *see* Calorific value

- Specific retention 293  
 Spermatophyta 22  
 Sphenophytes 22  
 Sphenopsida 20  
 Sphagnum bog, *see* Bog  
 Spitzbergen 79  
 Splitting, *see* Coal splitting  
 Spoil 392, 394  
   dumping 392  
   heaps, *see* Tip  
 Spontaneous combustion  
   135, 328, 396, 425  
 Spores 20, 22, 23, 175  
 Sporinite 98, 103  
 SPOT imagery 181  
 Spot purchases, *see* Coal  
   contracts  
 Spring 290  
 Stability 299  
 Stackers 425  
 Steam char reaction 366  
 Steam turbines 405  
 Steel production 110, 113,  
   403, 413  
 Stockpile 397, 405, 408, 425  
 Stockyard 425  
 Stopping 316  
 Storage coefficient 291  
 Storm washover sands 8  
 Stratigraphic 6, 25, 64, 65  
 Stratigraphy 25, 61–63  
 Streams 289  
 Strength index (SI) 115  
 Stress 318, 322, 348  
   fields 318, 319, 320  
   horizontal 318–320, 322,  
   326  
   measurements 319  
   vertical 318–320, 326  
 Stress relief 320  
 Strike 175, 178  
 Strike-slip faults 49  
 Strip mining, *see* Opencast  
 Stripping ratio 232, 234, 329,  
   330, 332, 338  
 Strontium 388  
 Structure 40, 196  
 Structure contours 232  
 Subbituminous, *see* Coal  
 Suberinite 103  
 Subsidence 398–403  
 Sulfates 110, 135  
 Sulfide 389  
 Sulfur 19, 25, 38, 39, 42, 55,  
   59, 68, 76, 79, 81, 85,  
   86, 89, 90, 93, 107, 113,  
   125, 126, 167, 407  
   dioxide (SO<sub>2</sub>) 405, 406,  
   408–411, 423  
   hexafluoride 419  
   inorganic 55, 126  
   organic 126  
   pyritic 126  
   trioxide 410  
 Sulfuric acid (H<sub>2</sub>SO<sub>4</sub>) 388,  
   410  
 Superficial deposits 248  
 Support 314, 315, 327  
 Surface Geophysical Methods  
   246  
 Surface tension 293  
 Surface water 289, 290  
 Swamp 15, 16, 17, 19, 22, 25  
 Swamp forest 15, 16  
 Swaziland 83  
 Swelling index, *see* Free  
   swelling index  
 Symbols 173–175, 177, 179,  
   180, 182, 183  
 Synchronous 27  
 Syncline 43  
 Syndepositional 40  
 Systems tract 25, 27
- t**
- Tadpole plots 278  
 Tailings 395  
 Taiwan 94  
 Tajikistan 89  
 Tanzania 83  
 Tar 379  
 Target coal seams 184  
 Taxodiaceous 23  
 Telinite 102, 103, 106  
 Telocollinite 102, 106  
 Temperature 15, 17, 18, 59,  
   118, 120, 135, 327, 343,  
   346, 377, 420  
 Temperature gradient 327  
 Terrestrialisation 17  
 Tertiary, *see* Palaeogene–  
   Neogene  
 Tethys 59  
 Textinite 106  
 Texture 196  
 Thailand 94  
 Thallium 407  
 Theodolite 173  
 Thermal maturity 348  
 Thickness, *see* Coal thickness  
 Thorium 121  
 Thrust, *see* Faults  
 Tidal channel deposits 8, 9  
 Tidal flat 9  
 Tip 392, 394  
   failure 392  
   reclamation 395  
 Tippler 428  
 Titano-magnetite 245  
 Tonnage requirements 432,  
   433  
 Tonstein 28, 44  
 Topogenous peatlands 15, 16  
 Topographic map 171  
 Topographic surface 232  
 Topography 175, 312, 392  
 Total core recovery (TCR)  
   190, 191, 198  
 Total dissolved solids (TDS)  
   387  
 Toxic/toxicity 407  
 Trace elements 38, 112, 113,  
   127  
 Trafficability 398  
 Transgressive systems tracts  
   26–28  
 Transmissivity 290  
 Transport 415, 423  
   barge 415, 428  
   conveyors 415, 425  
   rail 415, 426, 427  
   road 415, 426, 427  
   sea 415  
 Triassic 23, 44, 58, 59, 79, 85,  
   91, 418  
 Trimacerite 104  
 Triple tube core barrel 188



- Truck 329, 332, 334, 341, 394, 426  
 fleet management 334, 336, 337
- Tunnels 198
- Turkey 79
- Two-way travel time (TWT) 246–248, 250
- Tuyere 115
- U**
- Ukraine 89
- Ulminite 106
- Ultimate analysis 125
- Unconfined, *see* Aquifer
- Underground coal gasification (UCG) 365–375, 387, 422  
 UCG pilot test 373  
 UCG production 373  
 UCG site characteristics 372  
 UCG technology 366
- Underground fires 328
- Underground geophysical methods 264
- Underground mines/mining 152, 232, 293, 298, 301–303, 311–313, 324, 327, 328, 396, 415
- Underground roadways 312
- United Kingdom (UK) 6, 61, 63, 79, 80, 176, 212, 232, 241, 248, 253, 257, 264, 269, 307, 313, 319, 323, 326, 339, 348, 355, 362, 364, 365, 379, 380, 387, 389, 392, 394, 395, 399, 408, 428  
 East & West Midlands 80  
 Kent 80, 348  
 Lancashire–North Wales 8  
 Scotland 80  
 South Wales 41, 47–49, 80, 120, 313, 348, 362, 391, 392, 393  
 Yorkshire–Nottinghamshire 61, 63, 80, 362, 413
- United Nations (UN) 213
- United Nations Economic Commission for Europe (UNECE) 208, 213, 214, 216, 419
- United Nations Framework Classification of Fossil Energy and Mineral Resources (UNFC) 213–216, 222–225
- United Nations Framework Convention on Climate Change (UNFCCC) 419, 421
- United States of America (USA) 57, 58, 61, 66, 67, 156, 176, 177, 183, 208, 241, 248, 262, 263, 269, 311, 313, 318, 319, 320, 326, 327, 331, 332, 338, 344, 346, 355, 357–359, 365, 368, 379, 380, 388, 390, 391, 394, 403, 404, 407, 408, 413, 414, 417, 418, 420–422, 427–429
- Alabama 69, 358, 359
- Appalachian Basin 68, 359
- Arkansas 73
- Arkoma Basin 73
- California 74
- Colorado 73, 355, 358, 359, 404
- Green River Basin 73, 328
- Illinois 69, 395, 404
- Indiana 69, 395, 404
- Iowa 73
- Kansas 73, 359
- Kentucky 69, 338, 391, 395, 404
- Missouri 73
- Montana 73, 358
- New Mexico 358, 359
- Ohio 69
- Oklahoma 73, 359
- Pennsylvania 68, 359, 388, 391
- Piceance Basin 348, 349, 358, 359
- Powder River Basin 73, 335, 359, 380
- San Juan Basin 346, 348, 357, 358, 359
- Tennessee 69
- Texas 73, 359
- Utah 355, 359
- Virginia 69, 358, 359
- Washington 74
- West Virginia 69, 338, 359, 374
- Wyoming 358, 359, 374, 404
- United States Environment Protection Agency (EPA) 387
- United States Geological Survey 216, 217
- Upper delta plain 7, 13, 43
- Upwards coarsening 8
- Uranium 121, 389
- Uruguay 88
- Uzbekistan 89
- V**
- Vadose zone 290
- VALMIN 208
- Vanadium 111
- Variance 230
- Variogram 229, 230
- Vascularplants 376
- Vegetation 392
- Velocity 387
- Venezuela 59, 88, 328, 332, 337, 382, 427
- Ventilation 312, 327, 344, 351, 353
- Ventilation air methane (VAM) 351, 353
- Vibration white finger 416
- Vibroseis 253
- Vietnam 44, 93, 313, 375
- Vitrain 45, 97
- Vitrinertite 104, 105
- Vitrinertoliptite 104
- Vitrinite 20, 59, 102, 113, 115
- Vitrinite reflectance 113, 117, 118, 131, 151
- Vitrite 102, 105, 113

- Vitrodetrinite 103  
 Voids 290  
 Volatile matter 39, 125  
 Volatile organic compound (VOC) 419  
 Volcanics 28, 107, 175  
 Volumetric 329  
 Vulcanism 38, 43
- W**
- Wagon 427, 428  
 bottom discharge 428  
 tippler 428  
 top discharge 428  
 ‘Want’ 31  
 Washability curves 133, 134, 135  
 Washability data 133  
 Washouts 29, 34, 248, 250, 251, 270  
 Waste incineration 407  
 Water conductivity 245  
 Water gas shift reaction 366  
 Water quality 405  
 Water supply 303, 312  
 Water table 20, 291, 292, 295, 302, 303, 308, 309, 399
- Water treatment 307  
 Water velocity 295  
 Watercourses 387  
 Waterproof screen 302  
 Watershed 289  
 Wave, *see* Seismic  
 Wax 377  
 Weathering 131, 135, 151–153, 196, 197, 248  
 Weir 295  
 Wells 290, 299, 302, 303, 354, 355, 356, 363, 374  
 gob 351  
 production 298, 351, 367, 374  
 vertical 299, 351, 374  
 Wellman–Lord Process 410  
 Westphalian 14, 61  
 Wetland 15  
 Wheel loaders 331–333, 337  
 Wildlife conservation 395  
 Wireline drilling, *see* Drilling  
 Wirtgen machine 337  
 Work force 314  
 Workable coal section 232  
 Worked out coal 253
- Workings 299  
 Working area 314  
 Working depth 329  
 Working ratio 232  
 World Bank 420  
 World coal consumption 239, 240, 339  
 World coal production 237, 238, 339, 340, 403  
 World coal reserves 235–237  
 World coal reserves/production ratio 242  
 World CBM resources 358  
 Wyodak coal 251
- X**
- X-ray fluorescence 167  
 Xylitic 100
- Y**
- Yield 134, 135, 235, 379
- Z**
- Zaire 83  
 Zambia 83  
 Zimbabwe 83, 363  
 Zinc 111, 407