# Coal Geology

# Coal Geology

## Second Edition

Larry Thomas
Dargo Associates Ltd

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# Preface

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. 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 USA 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.

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

Finally I would like to thank my wife Sue for her support, forbearance and assistance with the manuscript.

Larry Thomas
Dargo Associates Ltd

# 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, Datamine International and Palladian Publications, as well as the staff at John Wiley & Sons Ltd.

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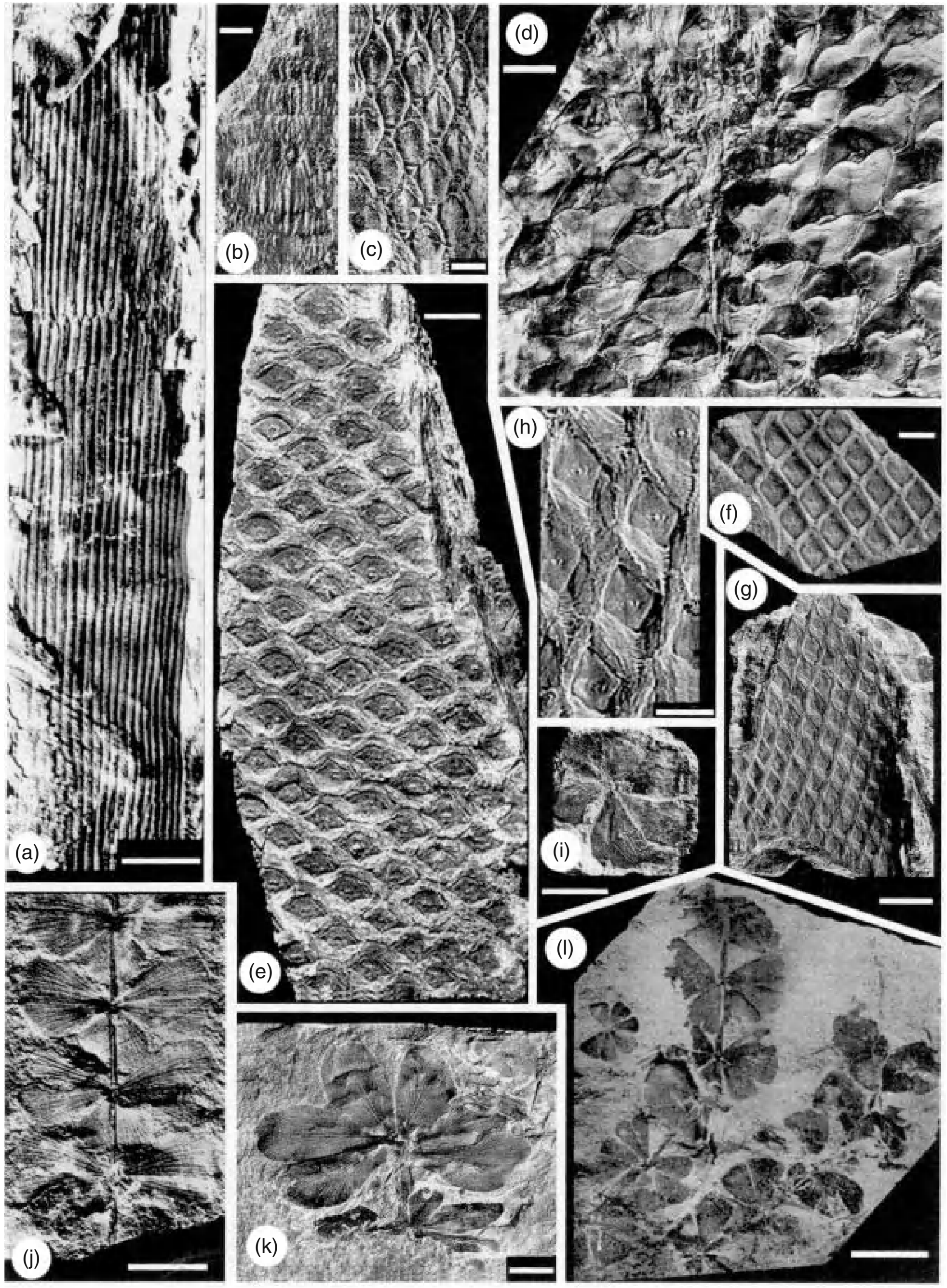


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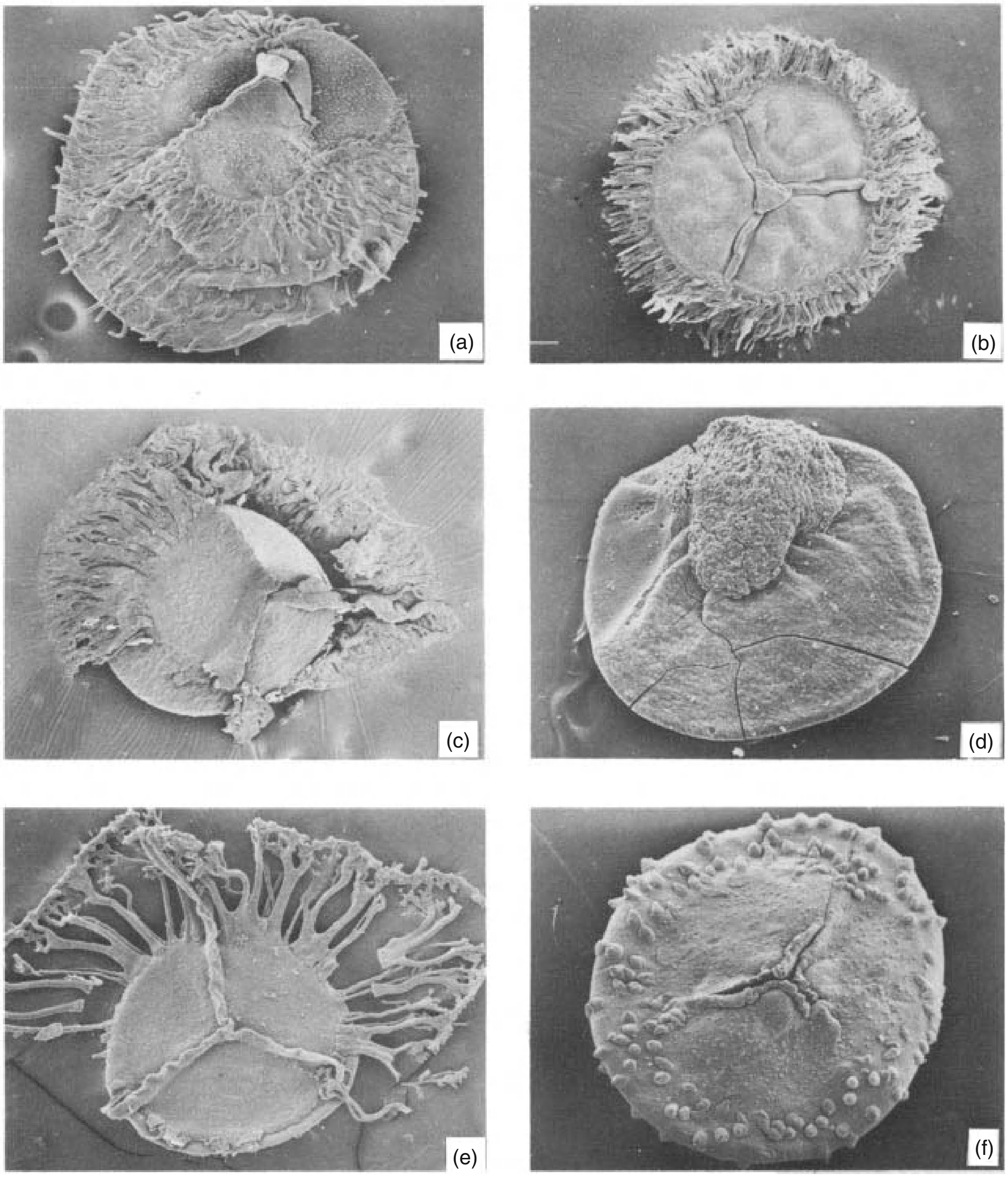


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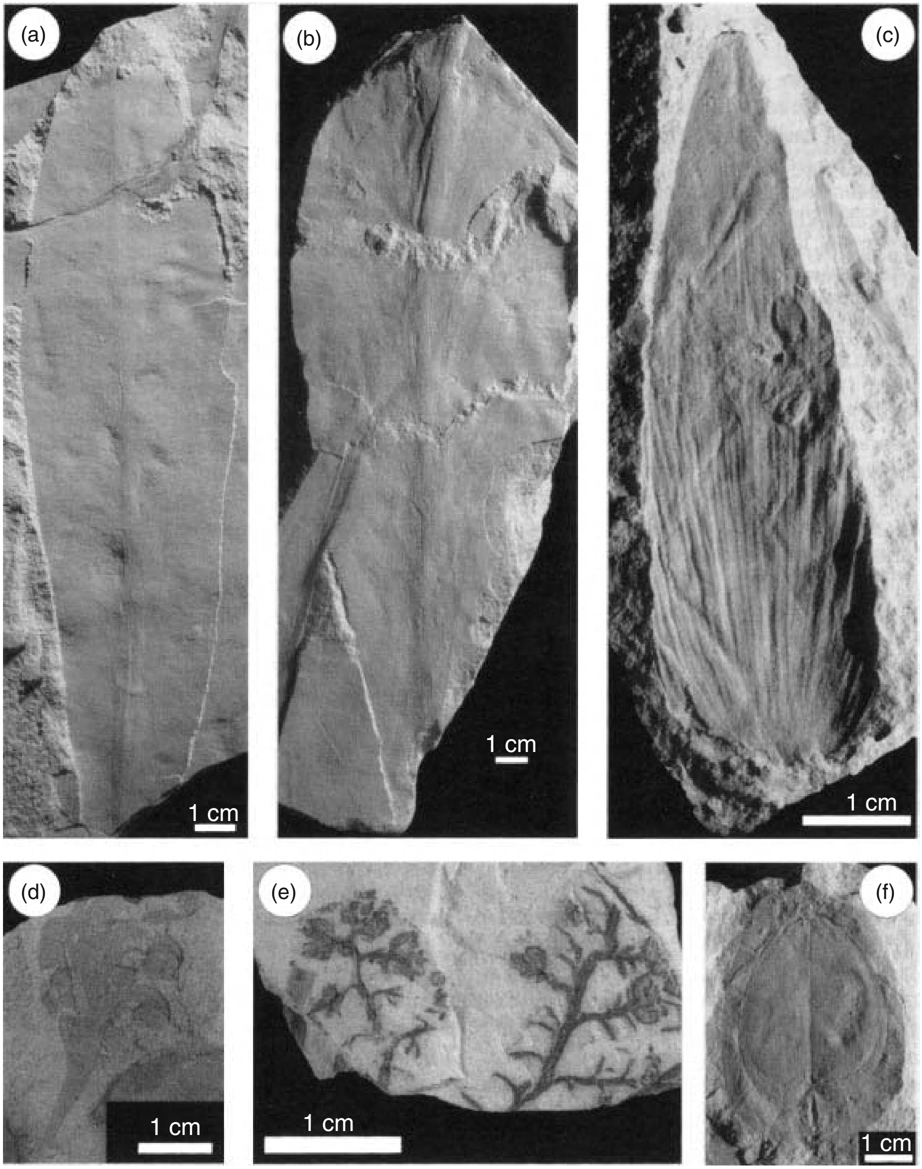


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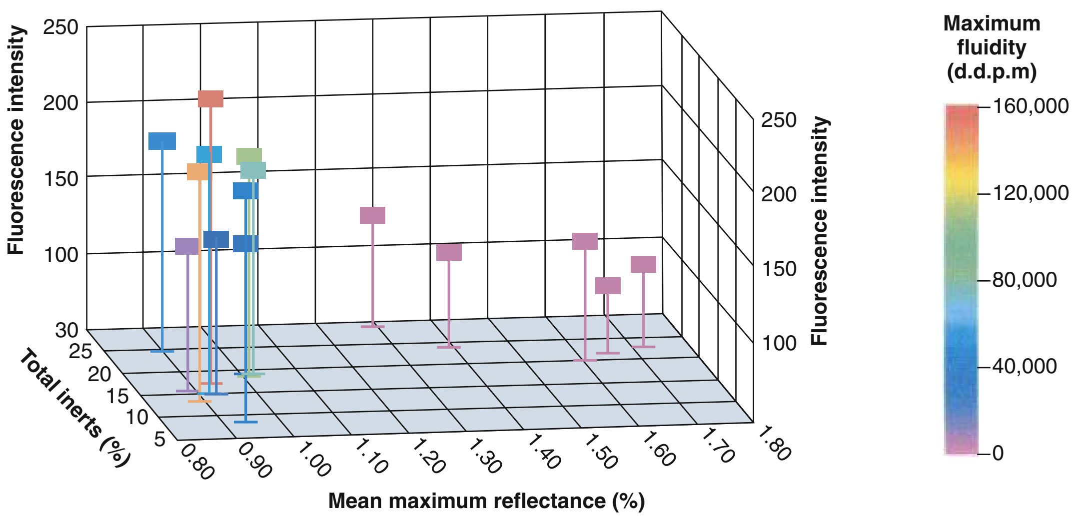


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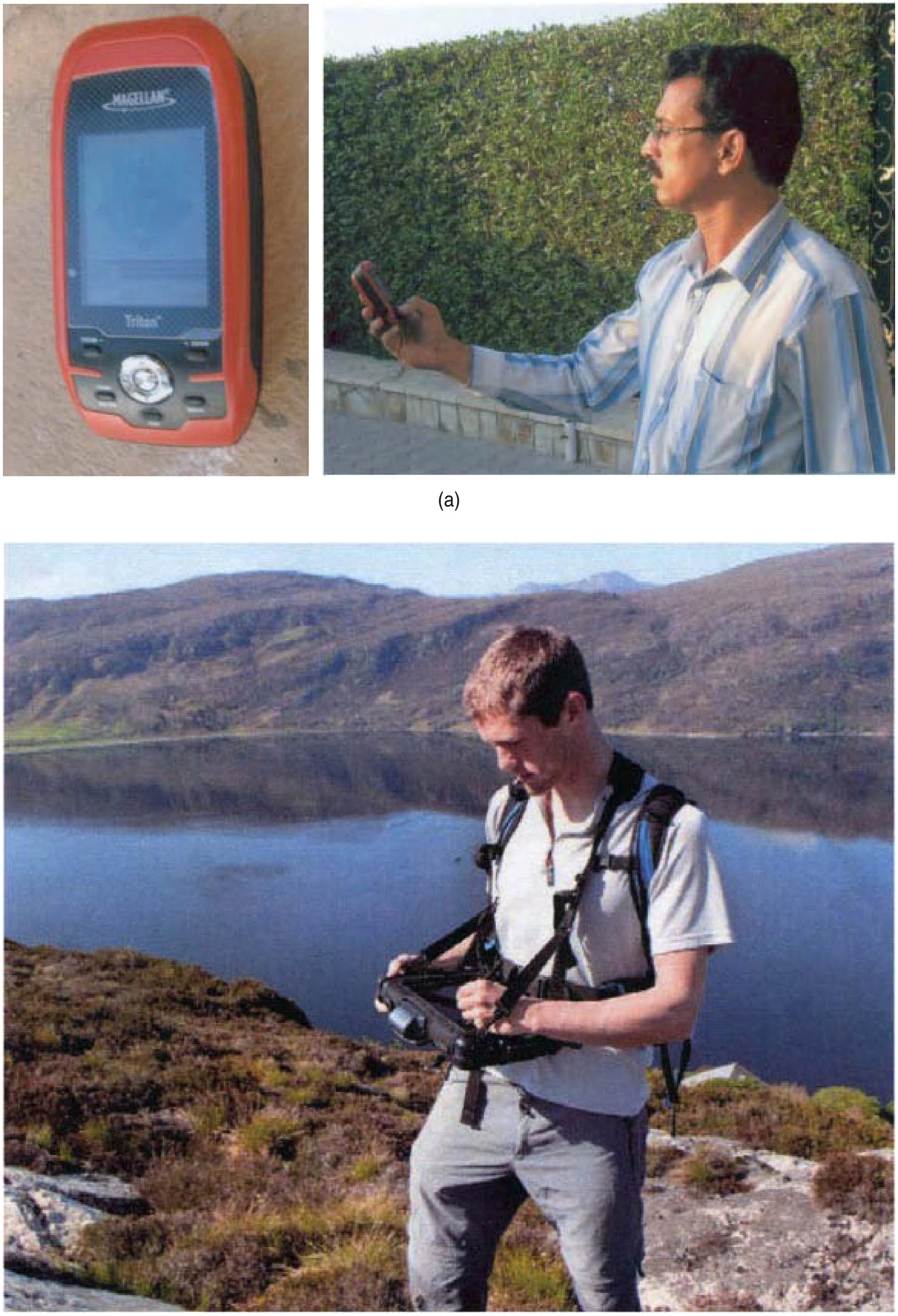


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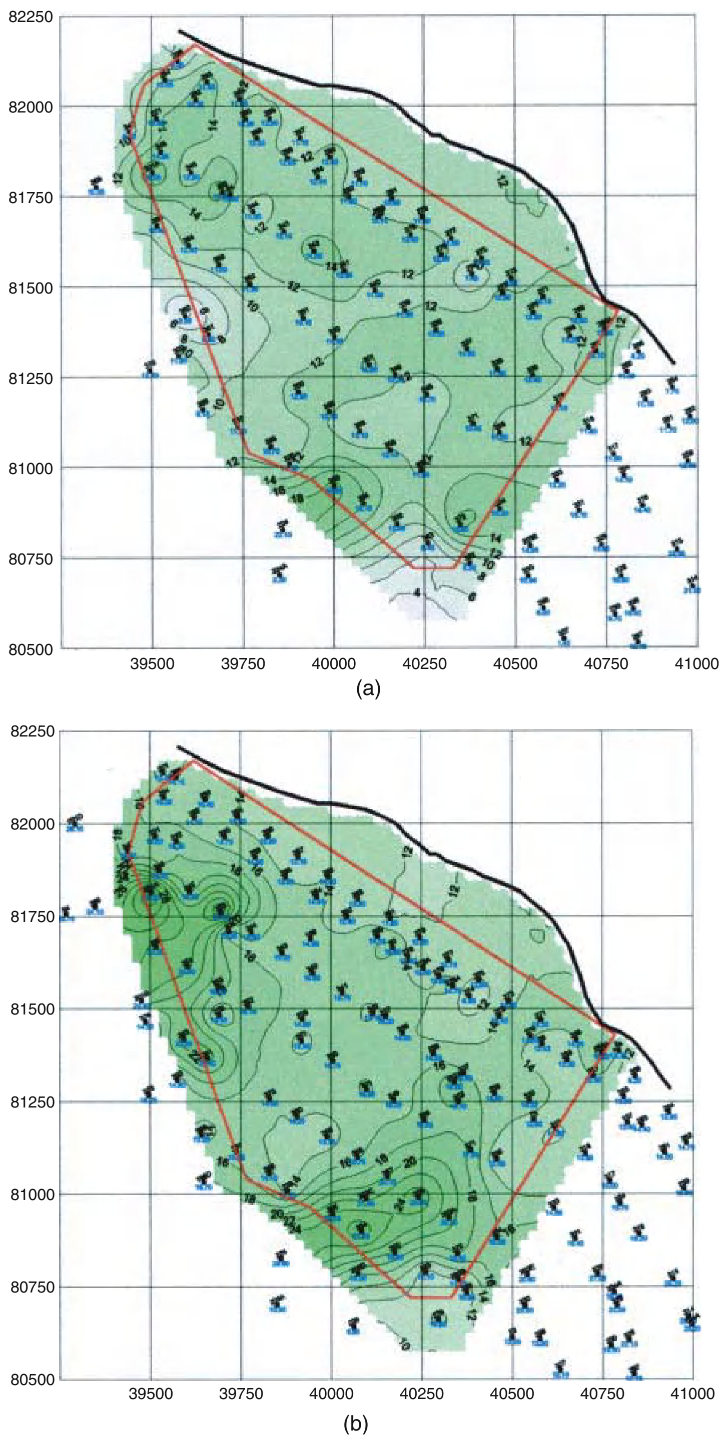


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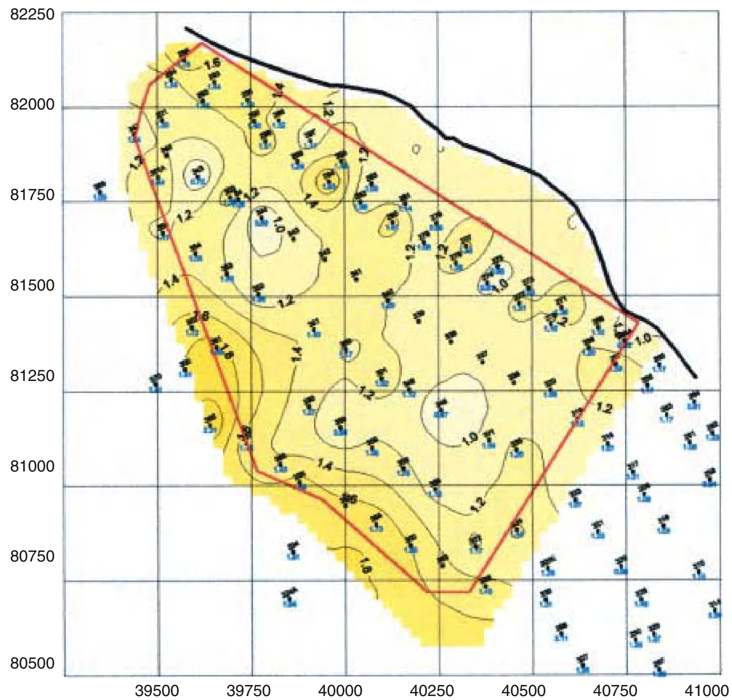


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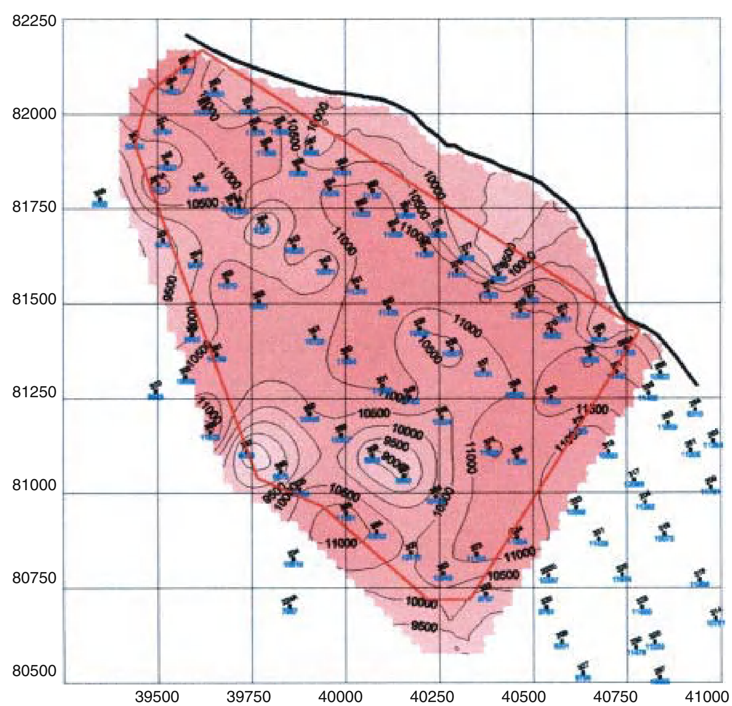


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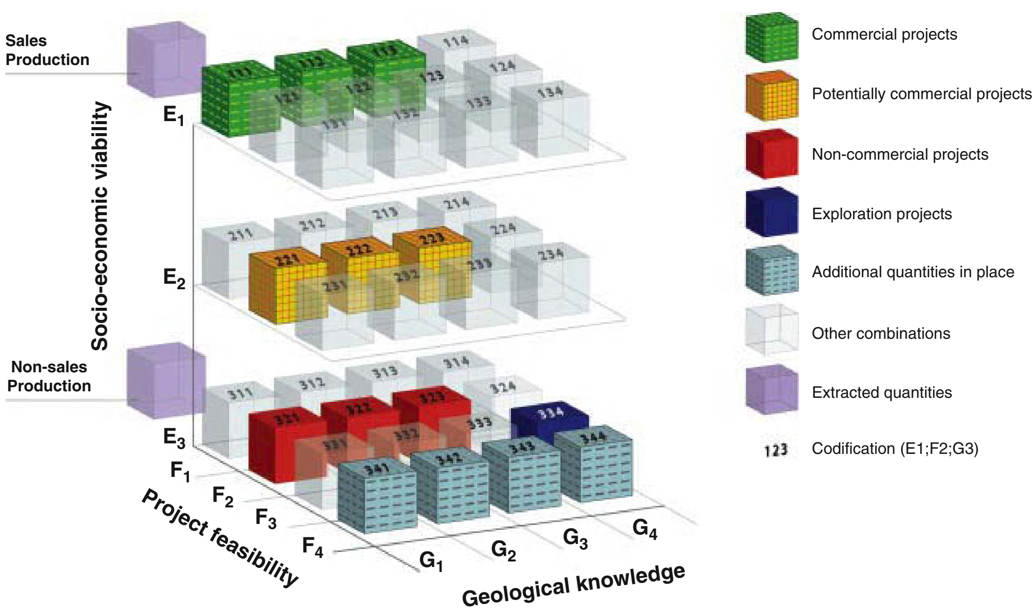


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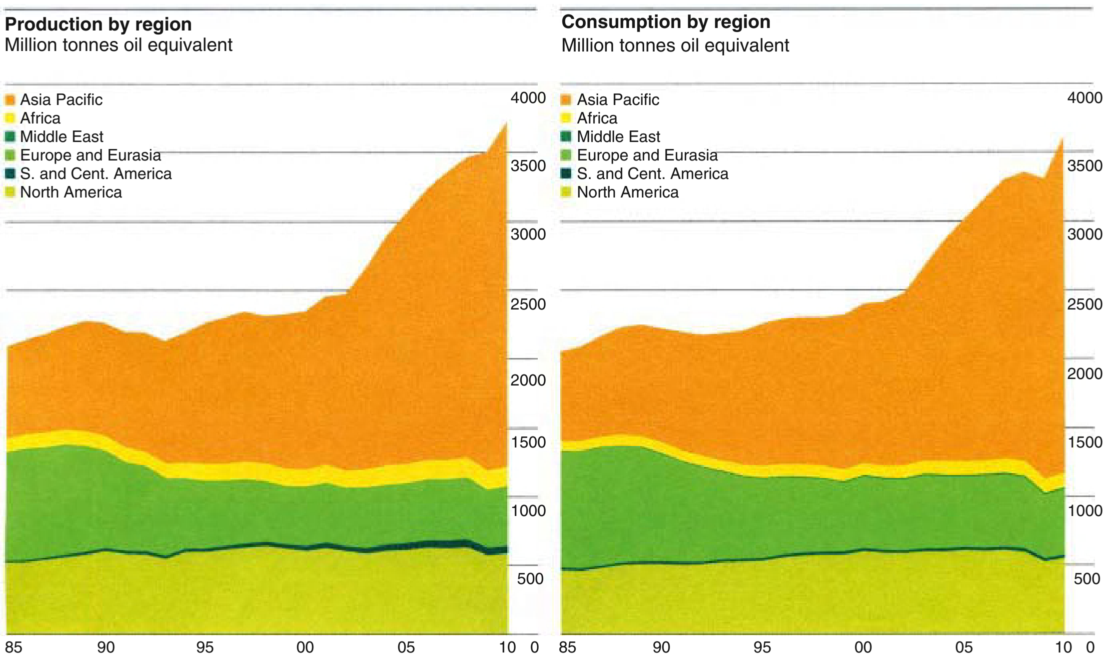


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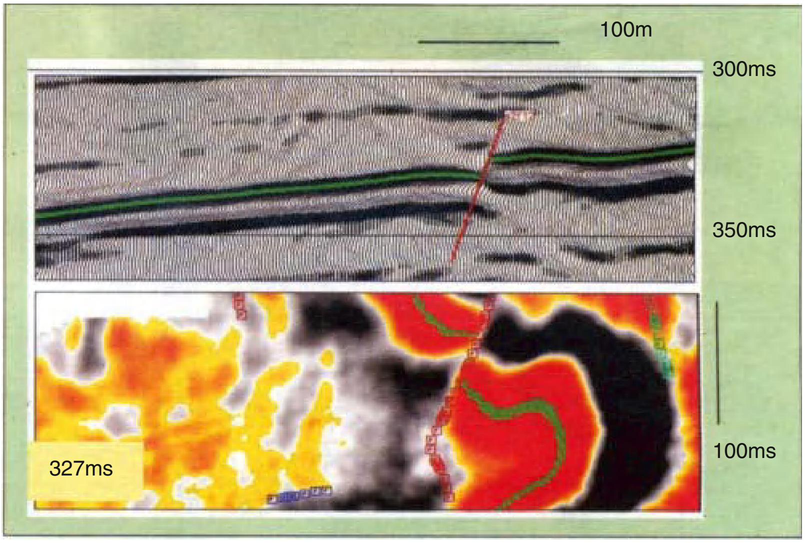


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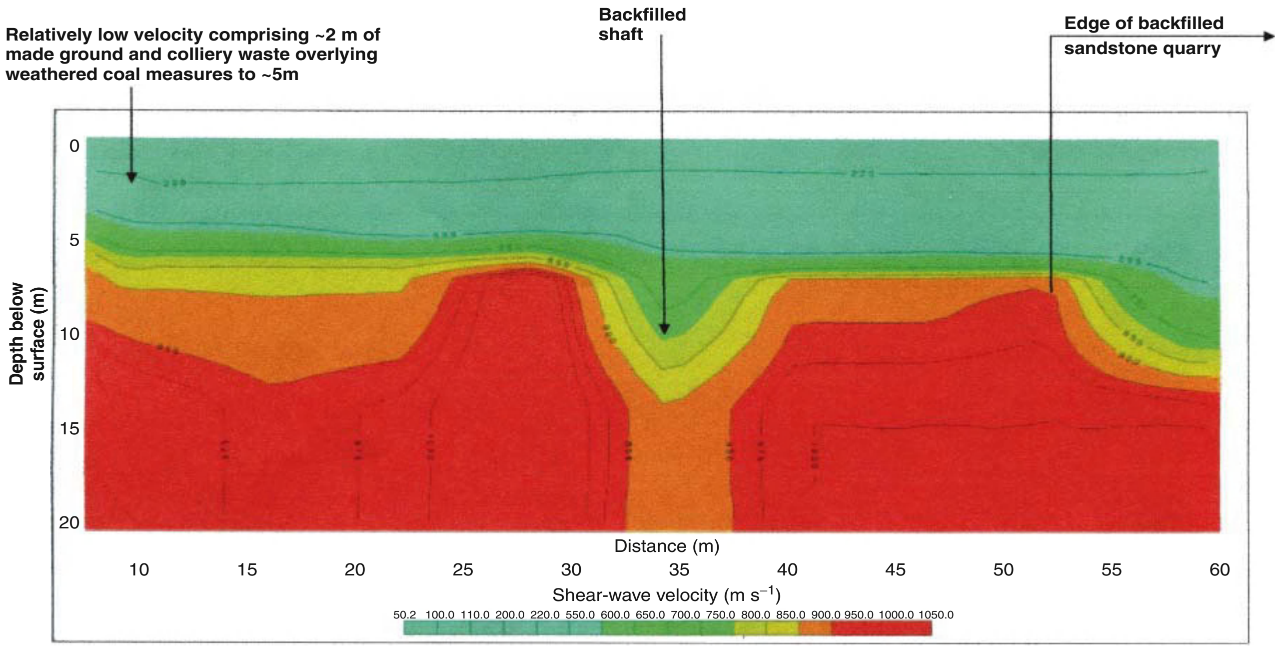


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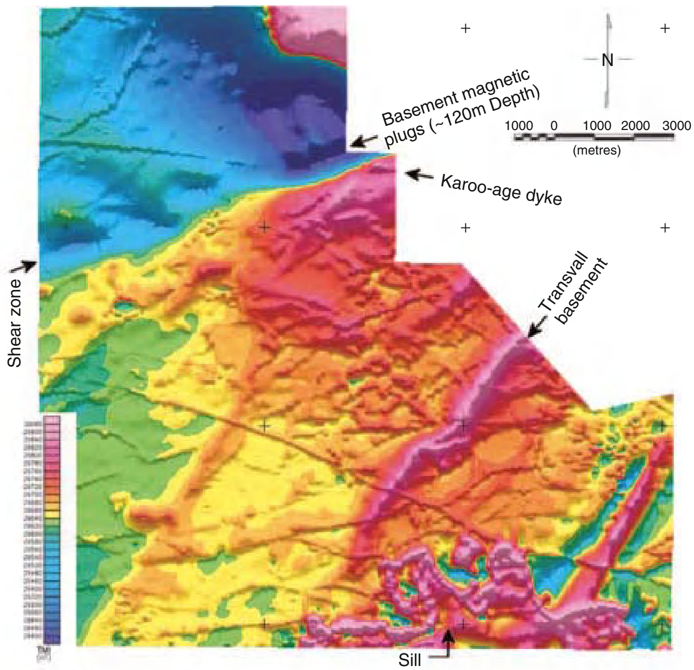


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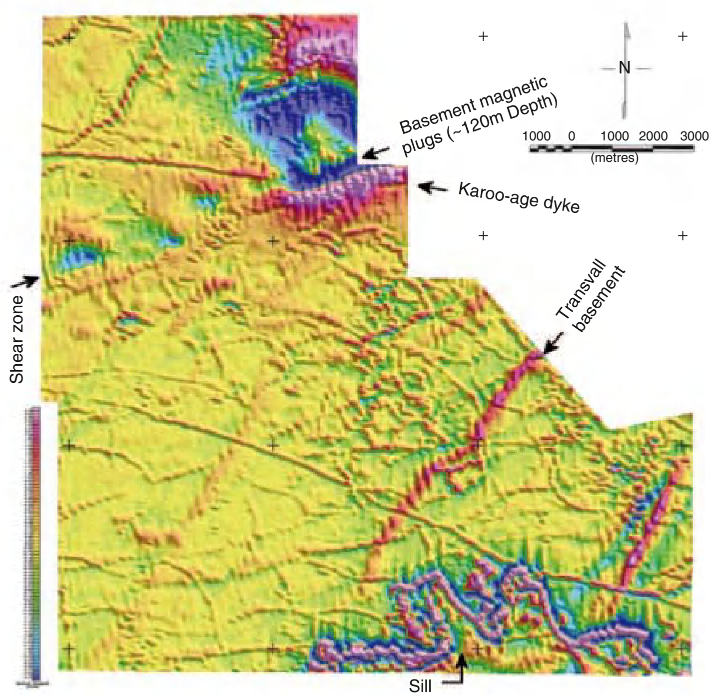


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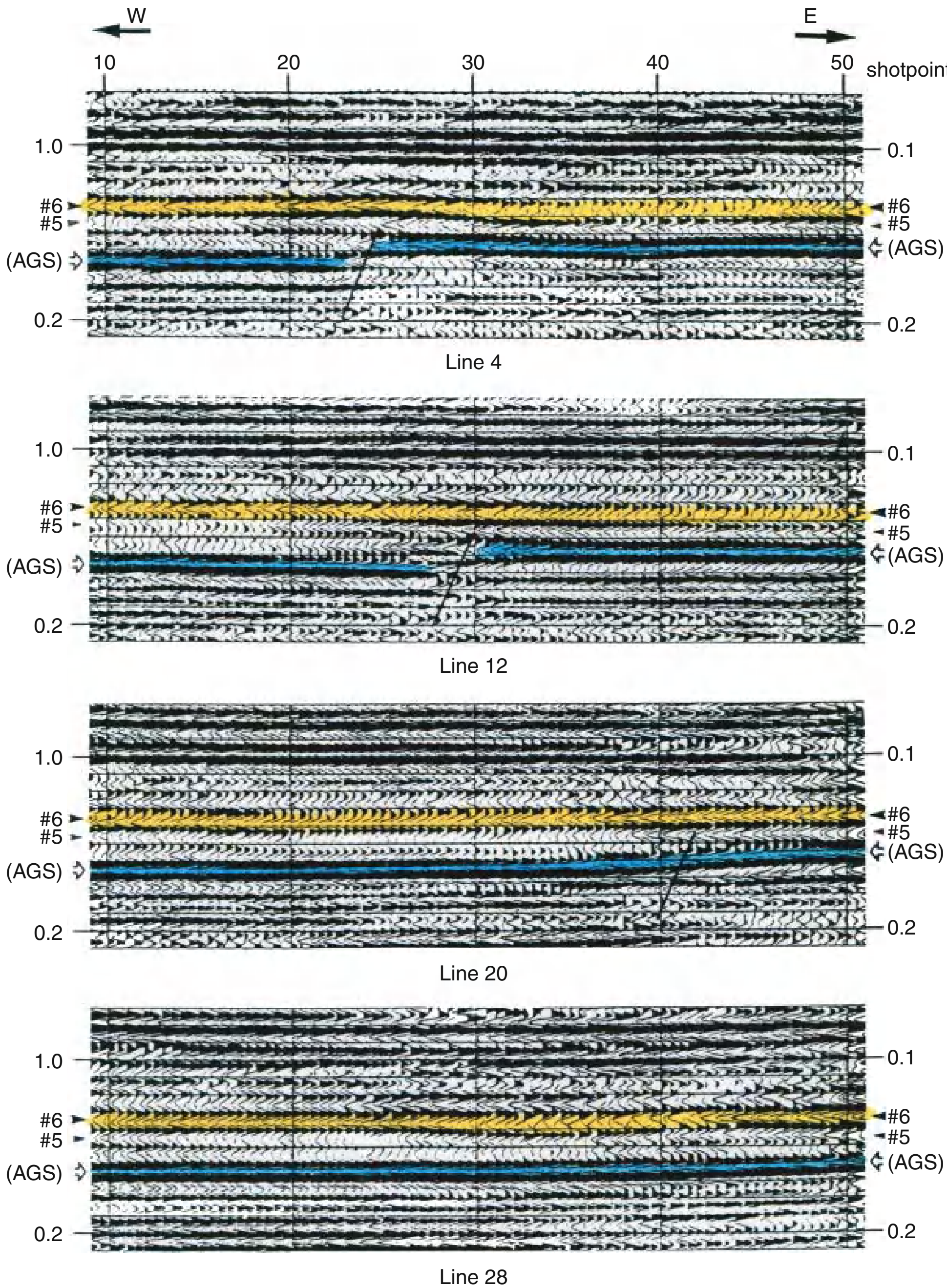


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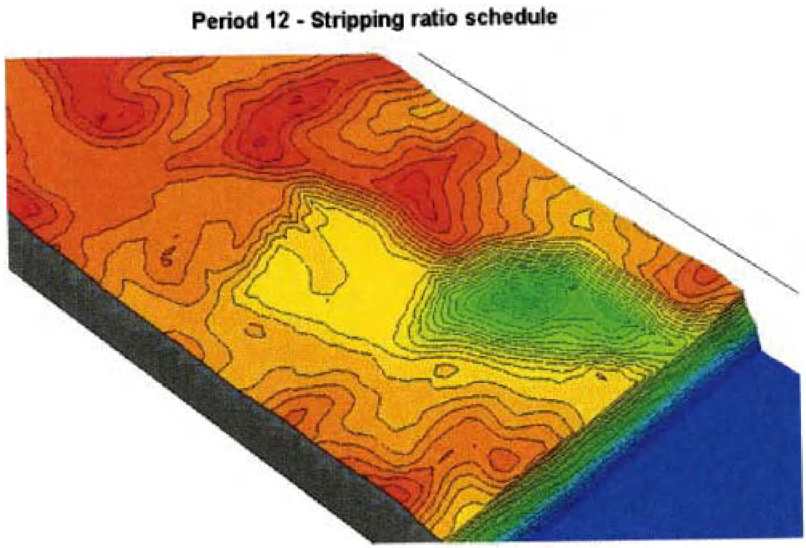


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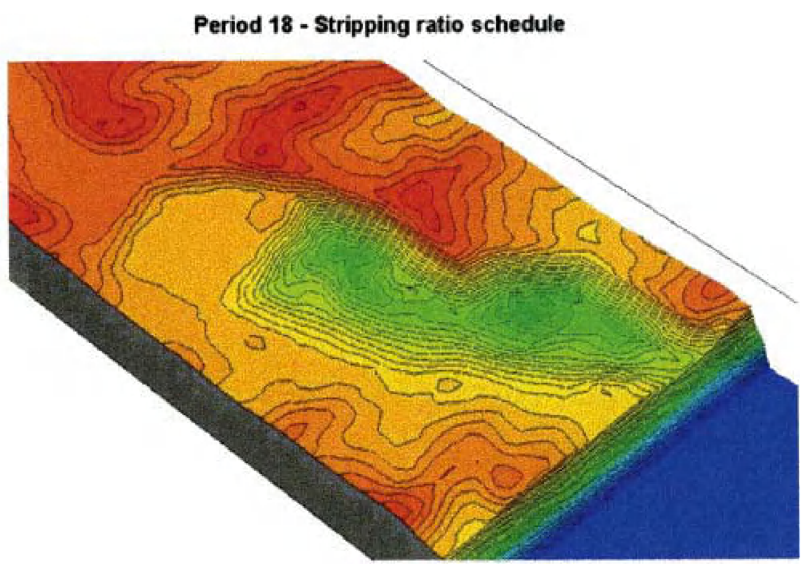


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# 1 Preview

## 1.1 Scope

The object of this book is to provide both geologists and those associated with the coal industry, as well as teachers of courses on coal, 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 covers only 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 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 to the techniques required to evaluate coal occurrences.

The episodes of coal development in the geological column (e.g. Carboniferous, Cretaceous, Paleogene and Neogene Periods – note that the Paleogene and Neogene Periods are sometimes referred to collectively as Tertiary) are given together with the principal coal occurrences worldwide. It is accepted that this is not totally exhaustive and that coal does occur in small areas not indicated in the figures or tables.

Current estimates of global resources and reserves of coal together with coal production figures are listed, and although these obviously become dated, they do serve to indicate where the major deposits and mining activity is currently concentrated.

In relation to the extraction of coal, understanding of the geophysical and hydrogeological properties of coals is an integral part of any coal-mine 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 gas from the coal and coal mines *in situ*, or by liquefying the coal as a direct fuel source, or by underground gasification of coal *in situ*. These technologies together are particularly significant in areas where conventional coal mining 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 is 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 re-occurrence of such phenomena. An outline is given of the types of environmental concerns that exist where coal is utilized, together with the current position on the improvements in technology in

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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, industrialized 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 thus 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. Currently coal accounts for 20% of all GHG emissions.

The world consumption of fossil fuels, and thus emissions of $CO_2$, will continue to increase, and fossil fuels still meet around 90% of primary energy requirements. The objectives 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 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 (Knapp, 2001). 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. It still remains to be seen whether such ambitions can be translated into a binding international agreement.

It remains a fact that many economies still depend on coal for a significant portion of their energy needs. Coal currently accounts for 29% of the world's consumption of primary energy, and, importantly, coal provides fuel for the generation of around 42% of the total of the world's electricity. In 2010, traded black coal amounted to 938 Mt of which 676 Mt was steam coal and 262 Mt was coking coal.

Coal reserves are currently estimated to be around 860 billion tonnes, 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 Upper Palaeozoic to recent.

Coals are the result of the accumulation of vegetable debris in a specialized 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 geographically separate areas 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 which 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 recognized 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 past 35 years, interest has grown rapidly in the study of sedimentological processes, particularly those characteristic of fluviatile 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, 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 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) suggest 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’. This concept has been modified to a model that relates lateral and vertical

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sequential changes to depositional settings that have been recognized in modern fluvial, deltaic and coastal barrier systems. The traditional model is based on the work carried out in the United States by Horne (1979), Horne *et al*. (1978, 1979), Ferm (1979), Ferm *et al*. (1979), Ferm and Staub (1984), Staub and Cohen (1979) in a series of studies in the 1970s. The sequences or lithofacies are characterized by the sedimentary features listed in Table 2.1. Other workers include Thornton (1979) and Jones and Hutton (1984) on coal sequences in Australia, and Guion, Fulton and Jones (1995) in United Kingdom.

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

| Recognition characteristics | | Fluvial and upper delta plain* | Transitional lower lower delta plain* | Lower delta plain* | Back-barrier* | Barrier* |
|---|---|---|---|---|---|---|
| I Coarsening upwards | A Shale and siltstone sequences | 2–3 | 2 | 1 | 2–1 | 3–2 |
| | >15.24 m (>50 ft) | 4 | 3–4 | 2–1 | 2–1 | 3–2 |
| | 1.524–7.62 m (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 |
| | >15.24 m (>50 ft) | 4 | 4 | 2–1 | 3 | 2–1 |
| | 1.524–7.62 m (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 |
| | Clay and silt | 3 | 2–3 | 1–2 | 2 | 3–2 |
| | Organic debris | 3 | 2–3 | 1–2 | 2–3 | 3 |
| | B Active channel sandstone fill | 1 | 2 | 2–3 | 2–3 | 2 |
| | Fine grained | 2 | 2 | 2–3 | 2–3 | 2 |
| | Medium and coarse grained | 1 | 2–3 | 3 | 3 | 2–3 |
| | Pebble lags | 1 | 1 | 2 | 2–3 | 3–2 |
| | Coal spars | 1 | 1 | 2 | 2–3 | 3–2 |
| III Contacts | Abrupt (scour) | 1 | 1 | 2 | 2 | 2–1 |
| | Gradational | 2–3 | 2 | 2–1 | 2 | 2 |
| IV Bedding | Cross beds | 1 | 1 | 1 | 1–2 | 1–2 |
| | Ripples | 2 | 2–1 | 1 | 1 | 1 |
| | Ripple drift | 2–1 | 2 | 2–3 | 3–2 | 3–2 |
| | Trough cross-beds | 1 | 1–2 | 2–1 | 2 | 2–1 |
| | Graded beds | 3 | 3 | 2–1 | 3–2 | 3–2 |
| | Point-bar accretion | 1 | 2 | 3–4 | 3–4 | 3–4 |
| | Irregular bedding | 1 | 2 | 3–2 | 3–2 | 3–2 |
| V Levee deposits | Irregularly interbedded sandstones and shales, rooted | 1 | 1–2 | 3–2 | 3 | 4 |
| VI Mineralogy of sandstones | Lithic greywacke | 1 | 1 | 1–2 | 3 | 3 |
| | Orthoquartzite | 4 | 4 | 4–3 | 1–2 | 1 |
| VII Fossils | Marine | 4 | 3–2 | 2–1 | 1–2 | 1–2 |
| | Brackish | 3 | 2 | 2 | 2–3 | 2–3 |
| | Fresh | 2–3 | 3–2 | 3–4 | 4 | 4 |
| | Burrow | 3 | 2 | 1 | 1 | 1 |

*1, abundant; 2, common; 3, rare; 4, not present.
Source: from Horne *et al*. (1979).

More recent studies have compared such established depositional models with modern coastal plain sedimentation, for example in equatorial Southeast Asia, and have concentrated in particular on modern tropical peat deposits: Cecil *et al.* (1993), Clymo (1987), Gastaldo, Allen and Huc (1993), McCabe and Parrish (1992). Studies by Hobday (1987), Diessel *et al.* (1992), Lawrence (1992), Jerzykiewicz (1992), Dreesen *et al.* (1995), Cohen and Spackman (1972, 1980), Flint, Aitken and Hampson (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.

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

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

### 2.2.2 The traditional model

#### 2.2.2.1 Coastal barrier and back barrier facies

The coastal end of the depositional model is characterized by clean barrier sandstones, which, in a seaward direction become finer grained and intercalate with red and green calcareous shales and carbonate rocks, the latter containing marine faunas. Landwards they grade 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 are therefore more quartzose than those sandstones in surrounding environments with the same source area.

They 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, 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 characterized 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 are characterized 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 American Carboniferous rocks, crevasse-splay deposits can be 10+ m in thickness and 30 m to 8 km wide.

In many cases, a transitional lower delta plain sequence is characteristic, featuring alternations of channel, interdistributary bay and crevasse-splay deposits: a depositional reconstruction is shown in Figure 2.3a, and generalized vertical sequence in Figure 2.3b.

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 mouth bar deposits of distributary channels, 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 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.

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

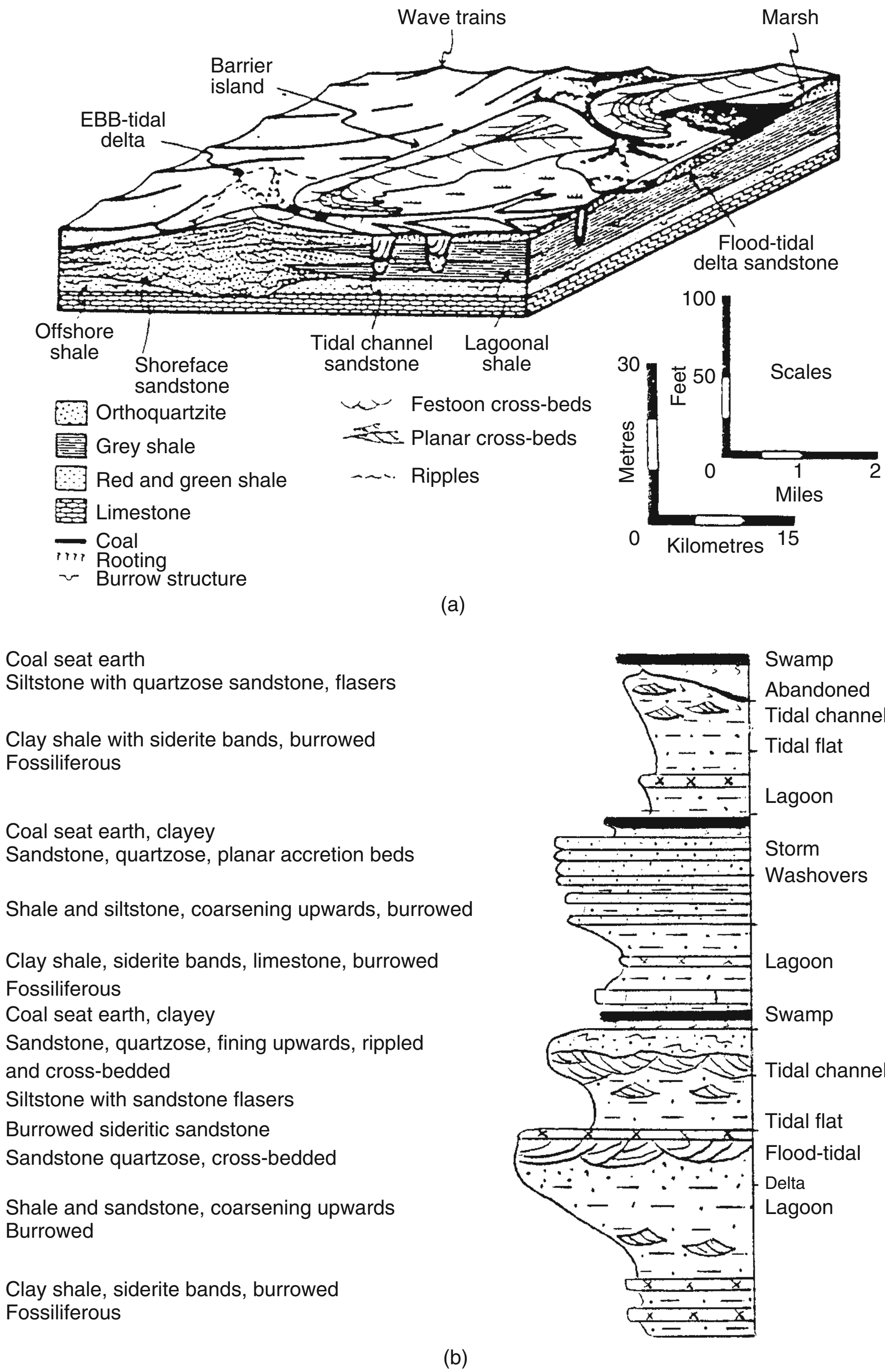



**Figure 2.1** (a) Barrier and back-barrier environments including tidal channels and flood-tidal deltas, based on exposures in Kentucky, United States. (From Horne *et al.*, 1979.) (b) Generalized vertical section through back-barrier deposits in the Carboniferous of eastern Kentucky, United States.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

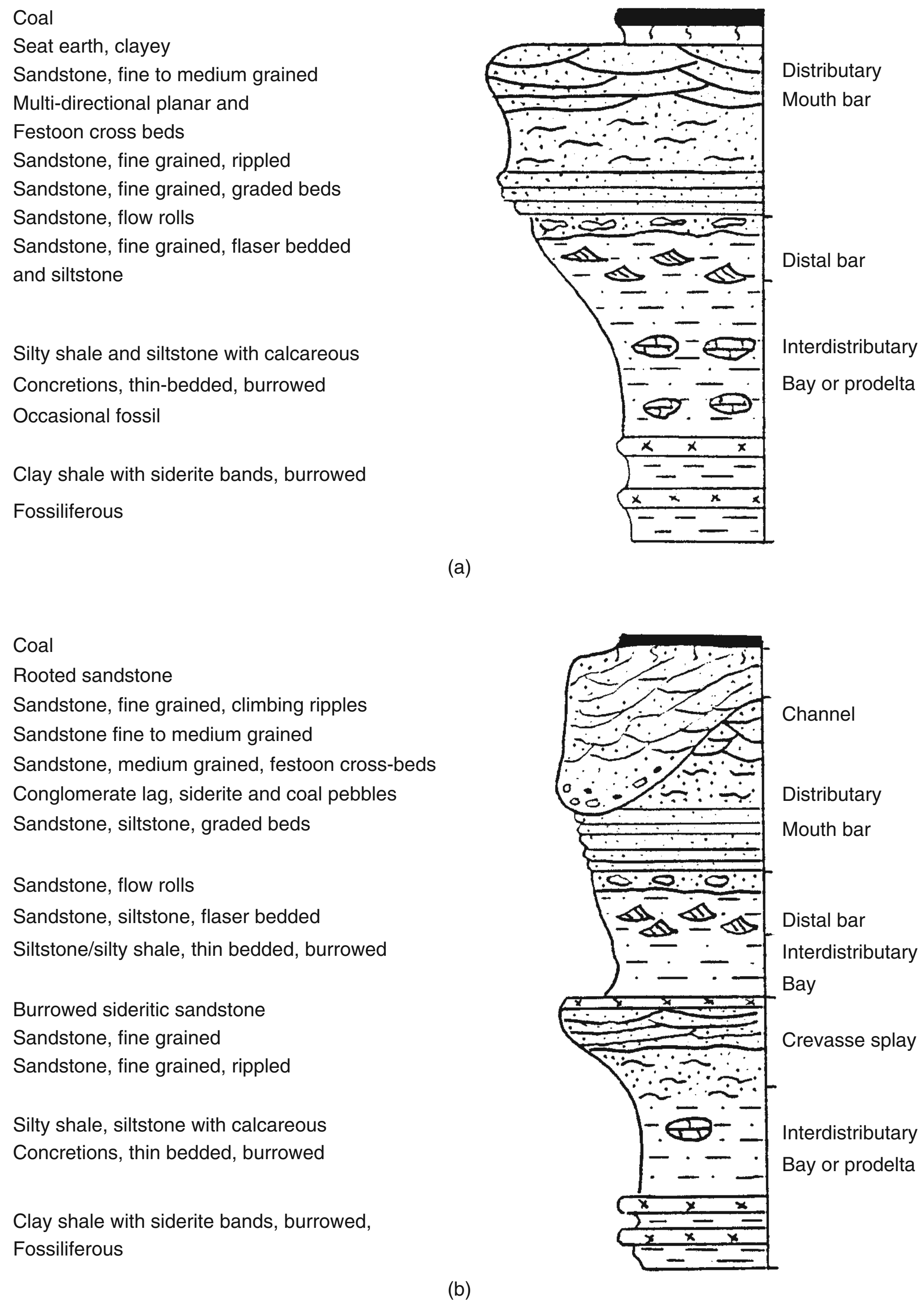



**Figure 2.2** Generalized vertical sequences through lower delta plain deposits in eastern Kentucky, United States. (a) Typical coarsening-upward sequence. (b) Same sequence interrupted by crevasse-splay deposits. (From Horne *et al.*, 1979.)

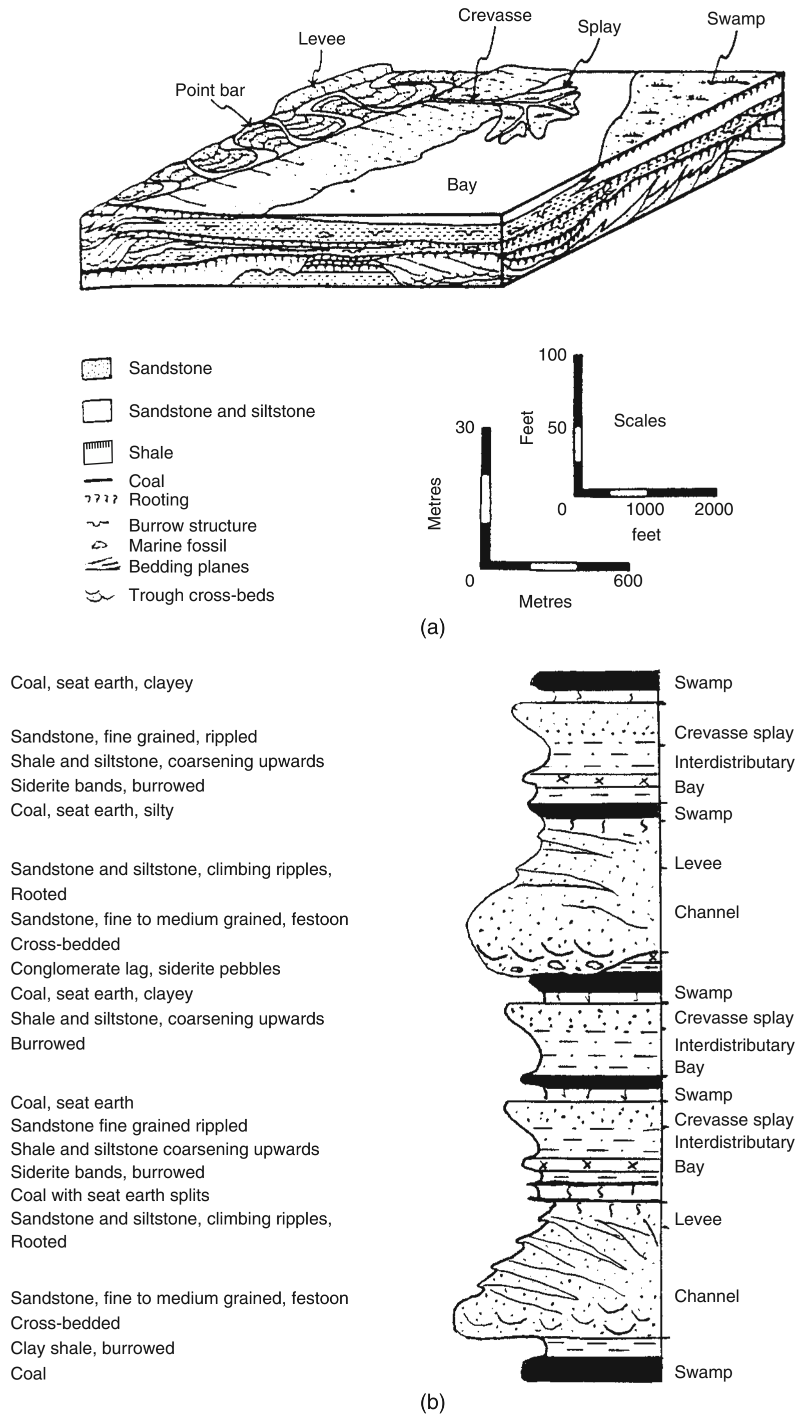



**Figure 2.3** (a) Reconstruction of transitional lower delta plain environments in Kentucky, United States. (From Horne *et al*., 1979.) (b) Generalized vertical sequence through transitional lower delta plain deposits of eastern Kentucky and southern West Virginia, United States. (From Horne *et al*.,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

abandoned channels, coals are generally relatively thin and widespread. Such coals are oriented parallel to the distributary patterns.

#### *2.2.2.3 Upper delta and alluvial plain facies*

In contrast to the thick fine-grained sequences of the lower delta plain facies, 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 in the lower part that include coal clasts. The sandstones are characterized 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. 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, United States.

Between the upper and lower delta plains, a transition zone exhibits characteristics of both sequences. This zone consists of a widespread platform on which peat mires are formed. This platform is cut by numerous channels and the sequence is 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.

### *2.2.3 Modern peat analogues*

The principal characteristics of a coal are its thickness, lateral continuity, rank, maceral 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 million hectares (WEC, 1998). This includes the tropical peats (>1 m thick) of South-east Asia which cover almost 200,000 $km^2$.

During the last 15 years, numerous studies have attempted to understand more fully how peat producing wetlands or mires are developed and maintained, and in particular how 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, but 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 2.5, 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.

**Mire** is now accepted as a general term for peat-forming ecosystems of all types.

**Bog** is generally confined to ombrotrophic peat-forming ecosystems.

**Bog forest** consists of ombrotrophic forested vegetation, usually an upper storey of coniferous trees and a ground layer of *Sphagnum* moss.

**Marsh** is an imprecise term used to denote wetlands characterized by floating vegetation of different kinds including reeds and sedges, but controlled by rheotrophic hydrology.

**Fen** is a rheotrophic ecosystem in which the dry season water table may be below the surface of the peat.

**Swamps** 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.

**Floating swamps** 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.

**Swamp forest** is a specific type of swamp in which trees are an important constituent, for example 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.

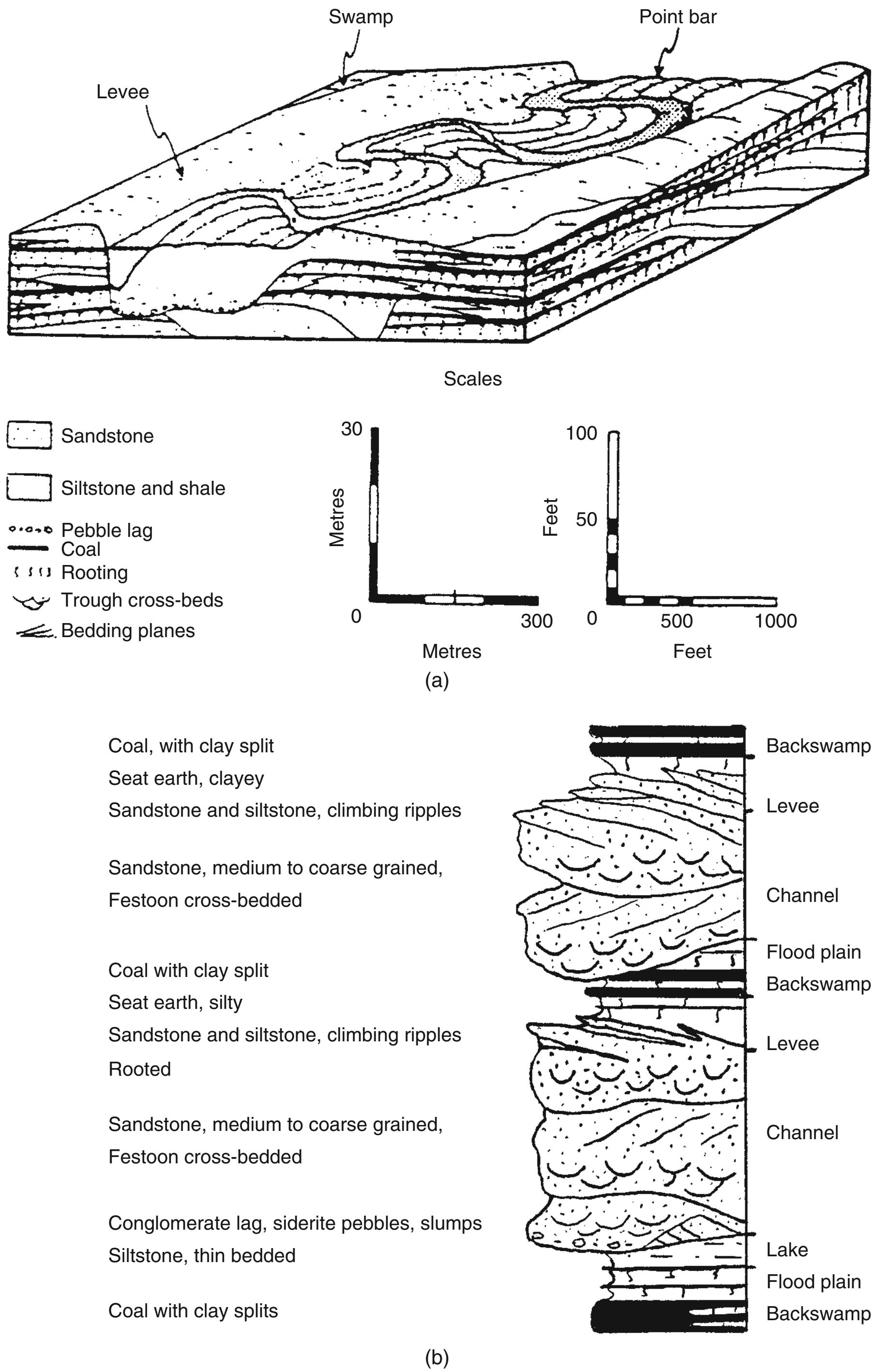



**Figure 2.4** (a) Reconstruction of upper delta plain–fluvial environments in Kentucky, United States. (From Horne *et al.*, 1979.) (b) Generalized vertical sequence through upper delta plain–fluvial deposits of eastern Kentucky and southern West Virginia, United States. (From Horne *et al.*, 1979.)

**Table 2.2** Classification of mires.

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

Source: adapted from Diessel (1992).

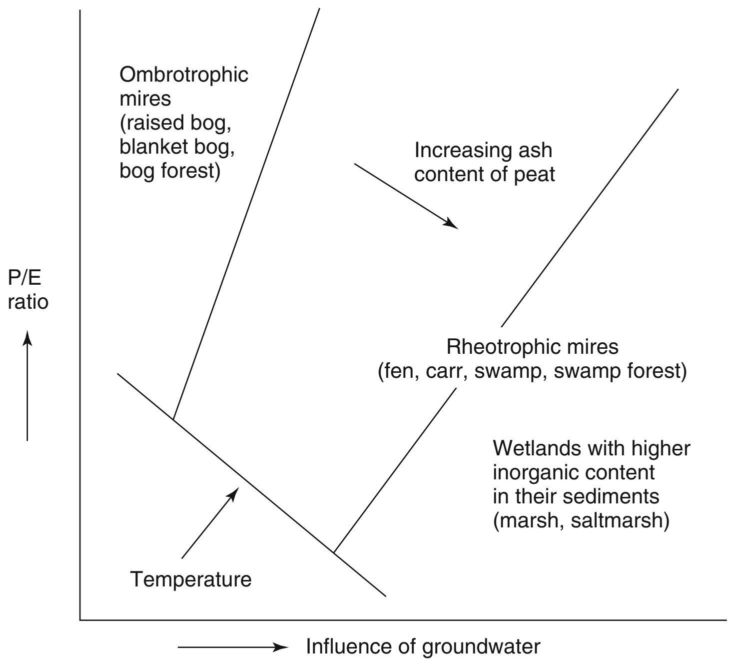



**Figure 2.5** Proposed relationship between mires in terms of the relative influence of rainwater and groundwater in their hydrological input. (Moore, 1987.)

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

$$\text{inflow} + \text{precipitation} = \text{outflow} + \text{evapotranspiration} + \text{retention}.$$

The conditions necessary for peat accumulation are therefore a balance between plant production and organic decay. Both are a function of climate, plant production and organic decay, and 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, 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:

1. terrestrialization, which is the replacement of a body of water (pond, lake, lagoon, interdistributary bay) by a mire;
2. paludification, which is the replacement of dry land by a mire, for example 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 2.6.

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 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

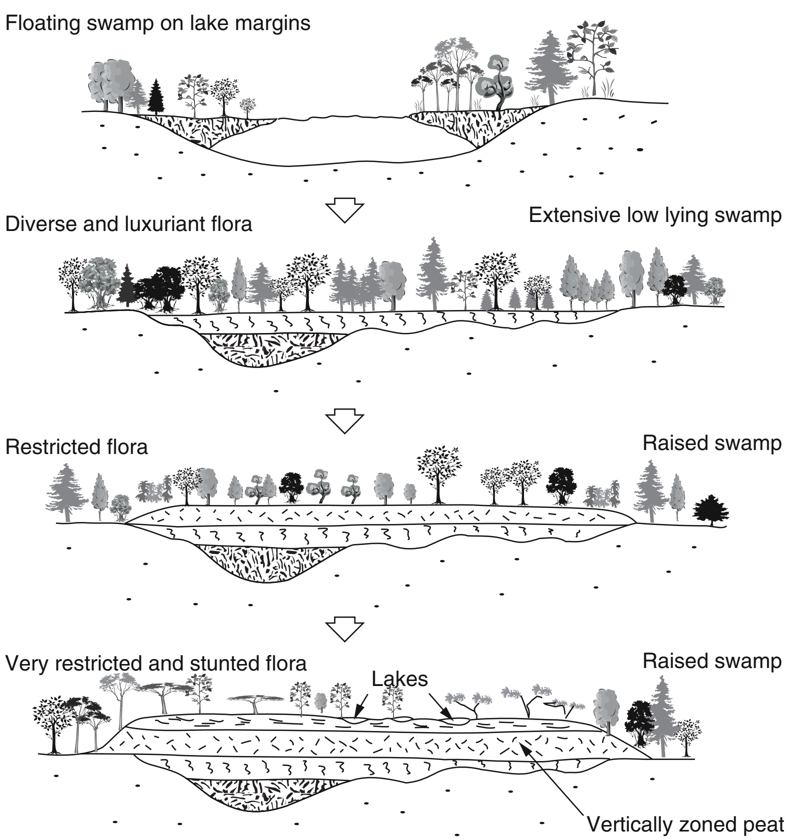



**Figure 2.6** Evolutionary sequence of swamp types showing the development of a raised swamp with distinct peat zonations. (From McCabe (1984), by permission of Blackwell Scientific Publications.)

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, for example 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 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, that is 5–10 kyr 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-scale 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 were formed under cool to temperate conditions, whereas the European Paleogene–Neogene coal formations began in tropical conditions in the Eocene, changing to temperate conditions in the Miocene.

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

1. fibrous or woody peat which shows the original plant structures only slightly altered by decay and may include branches, trunks and roots of trees;
2. pseudo-fibrous peat, comprised of soft plastic material;
3. 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;
4. 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 show 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 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 sediment 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 2.7).

The majority of coals are developed from plants that have formed peat close to where they grew. Such coals are underlain by seat earths 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, for example large rafts of peat or trees drifting on lakes or estuaries. Allochthonous coals do not have an underlying rootlet bed, but 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

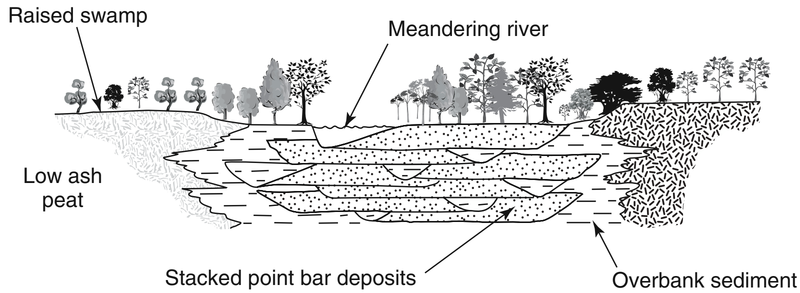



**Figure 2.7** Theoretical model of fluvial architecture in areas of raised swamps. The elevated swamp restricts overbank flooding and prevents avulsion, leading to the development of stacked channel sandstones. (From McCabe (1984), by permission of Blackwell Scientific Publications.)

sediments, and are thick and widespread (B.P.J. Williams, personal Communication, 2001).

### 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 are therefore 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. Land plants first appeared in the Early Devonian, and are significant in becoming life forms on land rather than in water, although most began in swampy environments. The carbonaceous shales of Emsian age (Early Devonian) 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 which 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, and had a poorly developed root system, of which *Stigmaria* is an example. Other associated forms had root systems for which details are poorly known. The lycopsids reproduced using the heterosporous technique, that is the ability to produce both megaspores and microspores, for example *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 cordaiteans from the Carboniferous of the United States. These are an extinct group of gymnosperm trees and shrubs characterized by large strap leaves and woody stems and were seed bearing; their nearest living relatives are the modern conifers. Plants in *Cordiates*-dominated peats probably grew in coastal mires in climate zones with seasons of low rainfall. Some authors interpret such *Cordiates*-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. obtusa* 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, China. Four floral assemblages were established, each reflecting the impact of climate changes, the so-called 'icehouse–greenhouse' climatic changes (Gastaldo, Dimichele and Pfefferkorn, 1996). Within these assemblages, some plant types are present throughout the period, for example, species of *Lepidodendron, Stigmaria, Sphenophyllum, Calamites* and *Cordiates* (Figure 2.8) as well as forms of *Pecopteris*

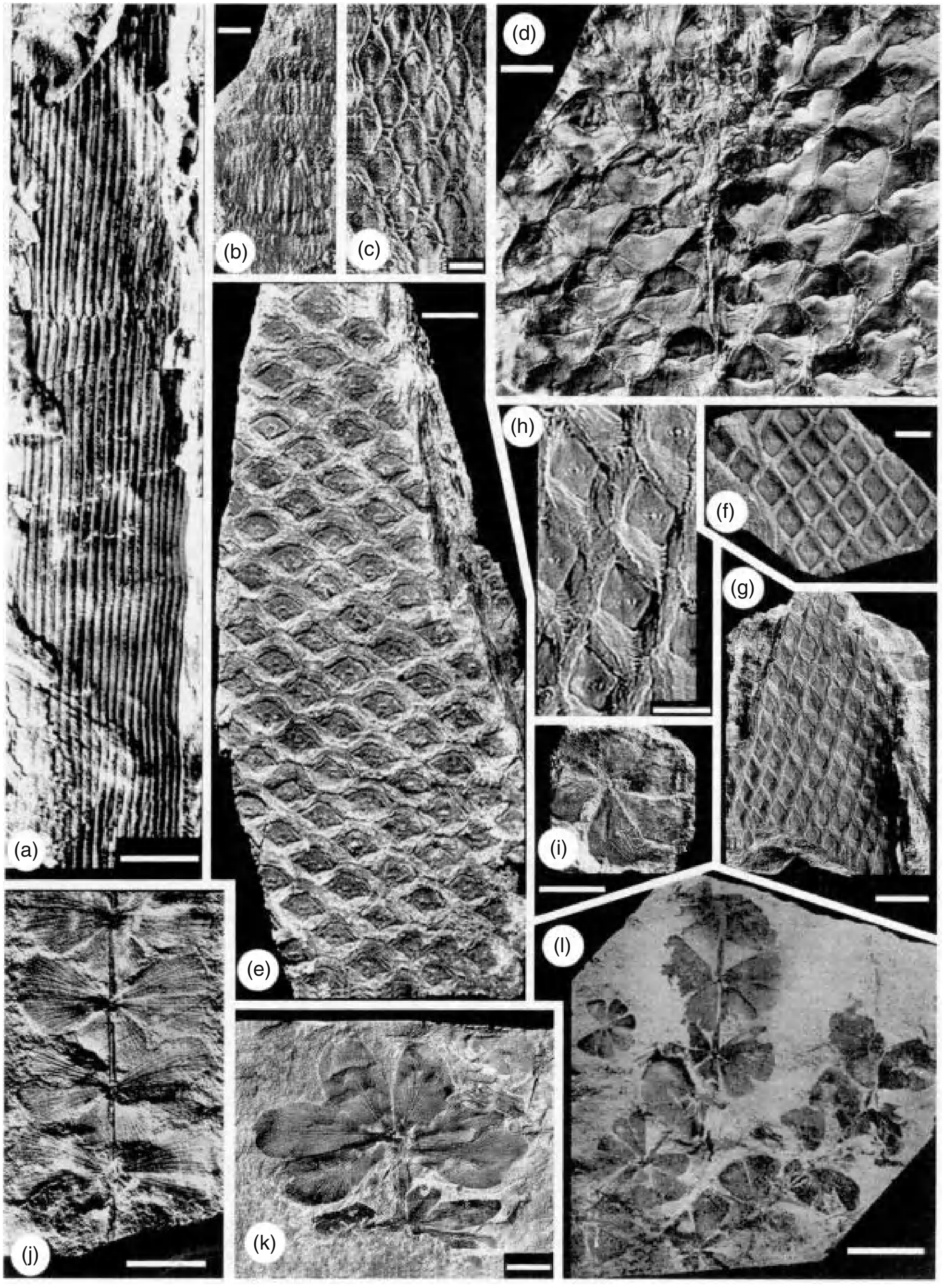



**Figure 2.8** Late Palaeozoic macrofossil assemblages, Weibei Coalfield, China. (a) *Calamites cistii* Brongniart. Upper Shihhotse Formation. (b) *Calamites cf. Schutzeiformis* Longmans. Upper Shihhotse Formation. (c) *Lepidodendron tienii* (Lee). Taiyuan Formation. (d) *Lepidodendron oculus-felis* Abb. Lower Shihhotse Formation. (e) *Cathaysiodendron acutangulum* (Halle). Upper Shihhotse Formation. (f) *Cathaysiodendron nanpiaoense* Lee. Taiyuan Formation. (g and 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. (Wang (2010). Permission of Elsevier Publications). This figure is reproduced in the Plates section.

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.

Bartram (1987) studied the distribution of megaspores (Figure 2.9) and the relationship to coal petrology using the Low Barnsley Seam (Westphalian B) from Yorkshire, United Kingdom. Six megaspore phases were recognized within the Barnsley Seam that suggested a floral progression with changing environment (Figure 2.10). However,

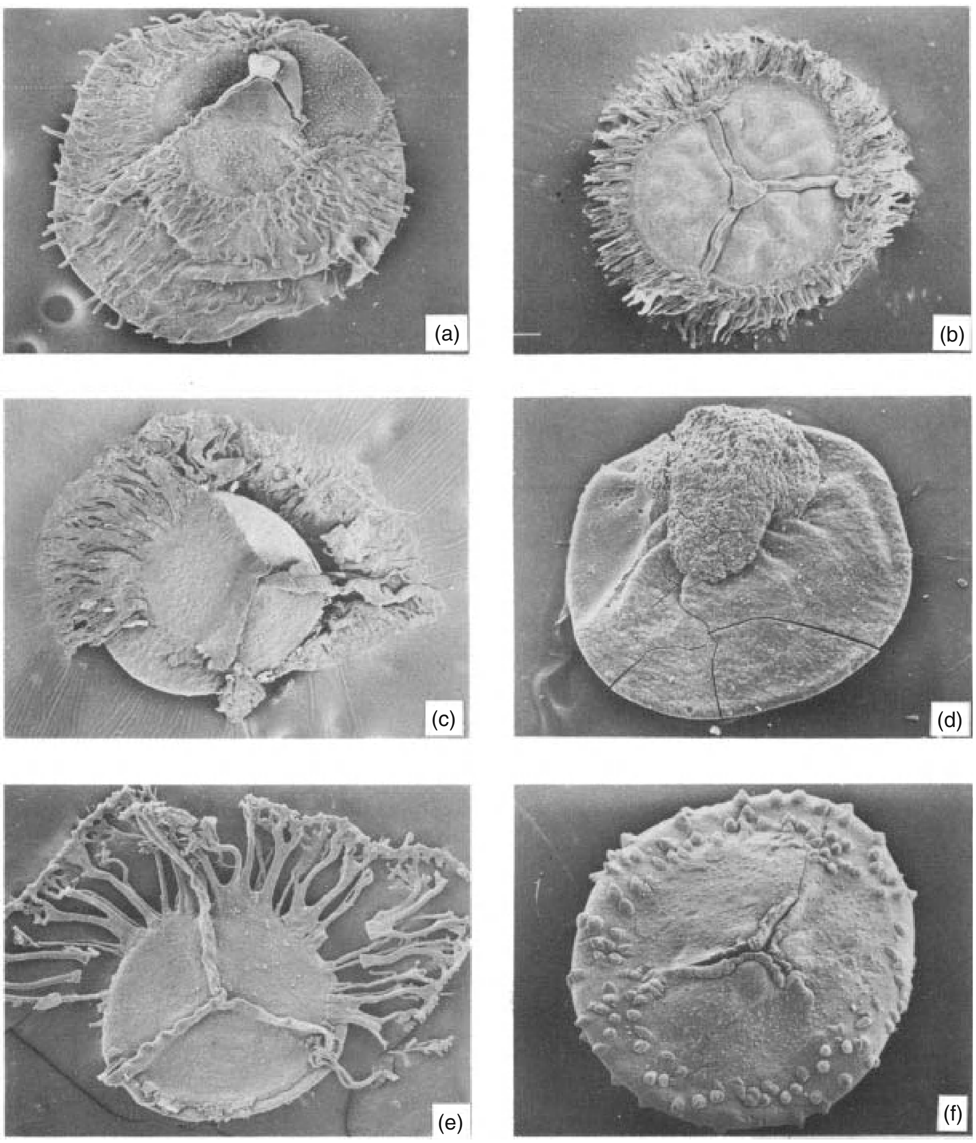



**Figure 2.9** Selected megaspores from the Low Barnsley Seam. (a) *Lagenicula subpilosa* (Ibrahim) Potonie & Kremp ×50, (b) *Setosisporites hirsutus* (Loose) Ibrahim ×50, (c) *Zonalesporites brasserti* (Stach & Zerndt) Potonie & Kremp ×25, (d) *Cystosporites varius* (Wicher) Dijkstra ×50, (e) *Zonalesporites rotates* (Bartlett) Spinner ×50, (f) *Tuberculatisporites mamillarius* (Bartlett) Potonie & Kremp ×25. (Bartram 1987) Permission of the Geological Society. This figure is reproduced in the Plates section.

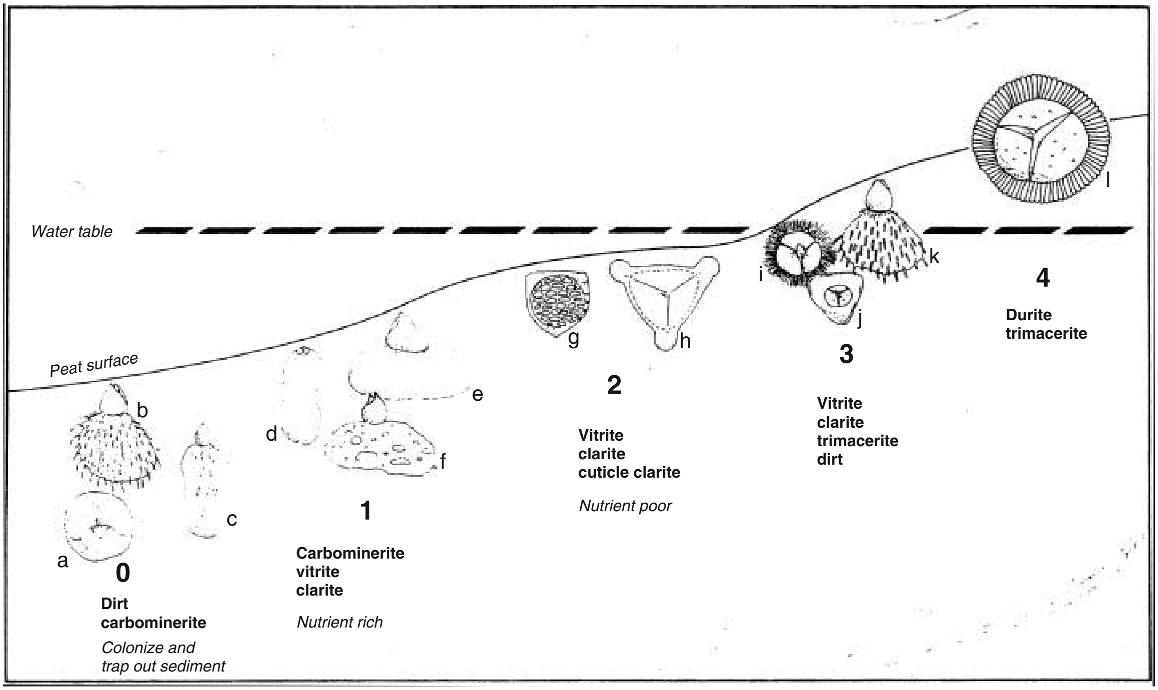



**Figure 2.10** Schematic diagram of idealised uninterrupted sequence of megaspore assemblages through a coal seam (From Bartram 1987). Permission of Geological Society.

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 (see 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 Paleogene–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, for example the Permian coals of the Sydney Basin, Australia (Diessel, 1992). The Permian 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 characterized 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, for example *Brasilodendron*, pecopterids and sphenopterid ferns and pollen-producing glossopterids (Figure 2.11) together with an increase in sphenophytes (leaf bearing) plants. Cordiateans 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 indicative of a hot humid climate such as tropical rain forest.

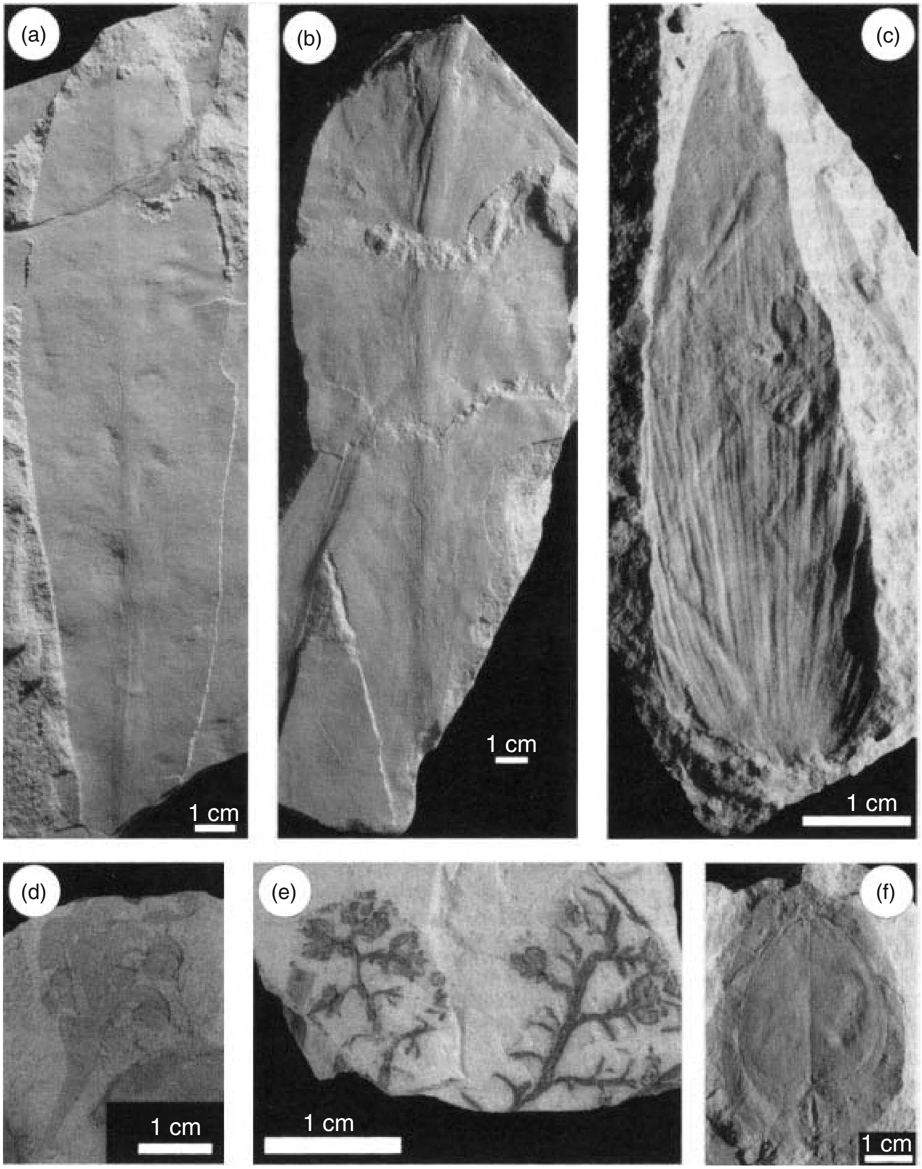



**Figure 2.11** Early Permian macrofossil assemblages, Parana Basin, Brazil. (a) *Glossopteris occidentalis*. (b) *Gangamopteris obovata* var.*Major*. (c) *Kawizophyllum* sp. (d) *Arberia minasica*. (e) *Coricladus quiterensis*. (f) *Samaropsis gigas*. (Iannuzzi (2010). Permission of Elsevier Publications). This figure is reproduced in the Plates section.

Gymnosperms, that is 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 characterized 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 Paleogene–Neogene Periods, gymnosperms and angiosperms dominate. Duigan (1965) carried out a review of Paleogene–Neogene brown coal flora from the Yallourn area of Victoria, Australia. In all, five gymnosperms and eleven angiosperms were identified.

Patton (1958) concluded that the forests that 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).

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.

### *2.2.3.2 Case studies*

A number of studies of peat accumulation in tropical Southeast Asia (Cecil *et al.*, 1993; Gastaldo, Allen and Huc, 1993; Neuzil *et al.*, 1993; Ruppert *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 United States. 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 rain forest environment. Gastaldo (2010) has compared the Rajang and Mahakam deltas in Borneo and has 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, Allen and Huc (1993) and Gastaldo (2010) observed no autochonous 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, with 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, recognized by the composition and distribution of mineral matter in coal.

Grady, Eble and Neuzil (1993) have 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 with maceral profiles from Carboniferous coals in the United States, and could be used to interpret the changing conditions in the original peat mire. Styan 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), 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. Figure 2.12 (Diessel, 1992), illustrates the concept of sequence stratigraphy with its terminology. Parasequences are selected representing depositional types within differing systems tracts. Vail (1987) has identified four kinds of system tract and Diessel (1992) has summarized this concept.

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 when combined with climate, determine lithofacies types over geologic time.

When a eustatic drop in sea level exceeds the rate of basin subsidence, it causes 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 basin-floor and slope fans 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, comprising immature or reworked clastic sediments.

In the opposite sense, a relative rise in sea level due to a eustatic rise combined with basin subsidence produces a **transgressive** systems tract. 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, which represents the upward termination of the transgressive systems tract. The periods of eustatic change may be short and numerous producing a sequence of delta

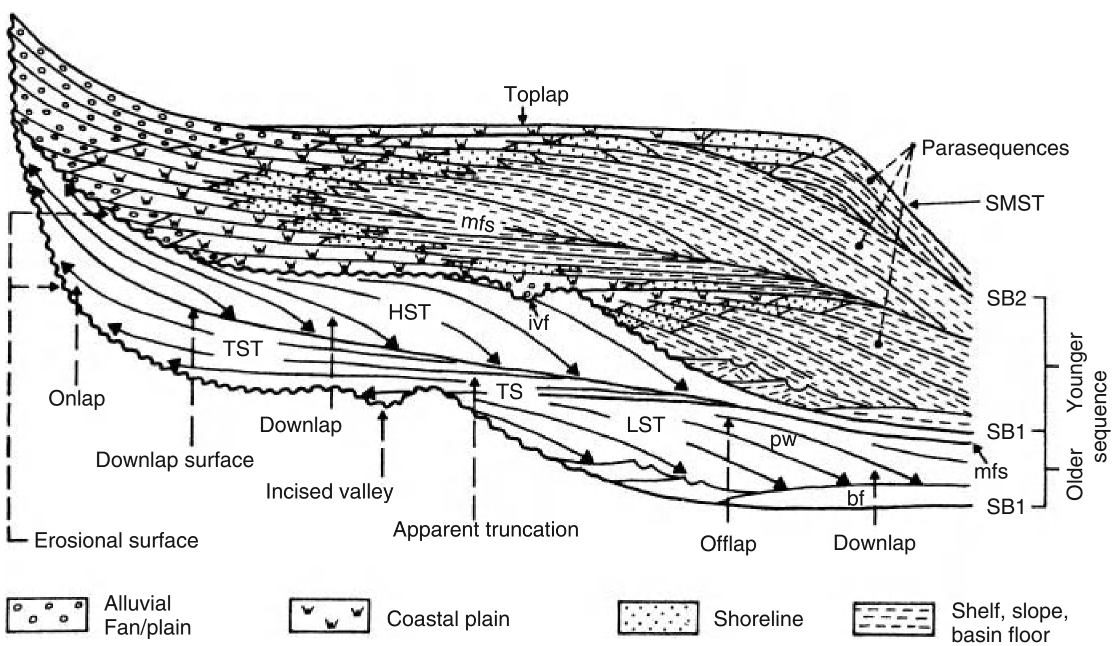



**Figure 2.12** Cartoon illustrating the concept of sequence stratigraphy and its terminology (After Vail 1987 and van Wagoner et al 1987). With permission from Springer-Science & Business Media. In: Diessel (1992), Coal-bearing Depositional Systems, (fig. 8.1). SBI, type 1 boundaries, TST transgressive 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.

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 **highstand** systems tract. It is during this phase that coal formation will be maximized and sedimentation reflecting a regressive sequence will be established.

These depositional settings of transgression and regression control coal formation, and in particular, the types of coal and their composition, and 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, its lateral development and whether it is synchronous or diachronous.

Figure 2.13 (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 transgressive systems tract (TST) and highstand systems tract (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 trangressive systems tract, the presence of seawater on or within the peat accumulation can produce high sulfur levels which 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, Aitken and Hampson (1995) applied sequence stratigraphy to Westphalian coal measure sequences of the United Kingdom. Their conclusions were that the development of regionally extensive mires required a slow rising water table, characteristic of a transgressive systems tract (TST). Coals of economic significance occur

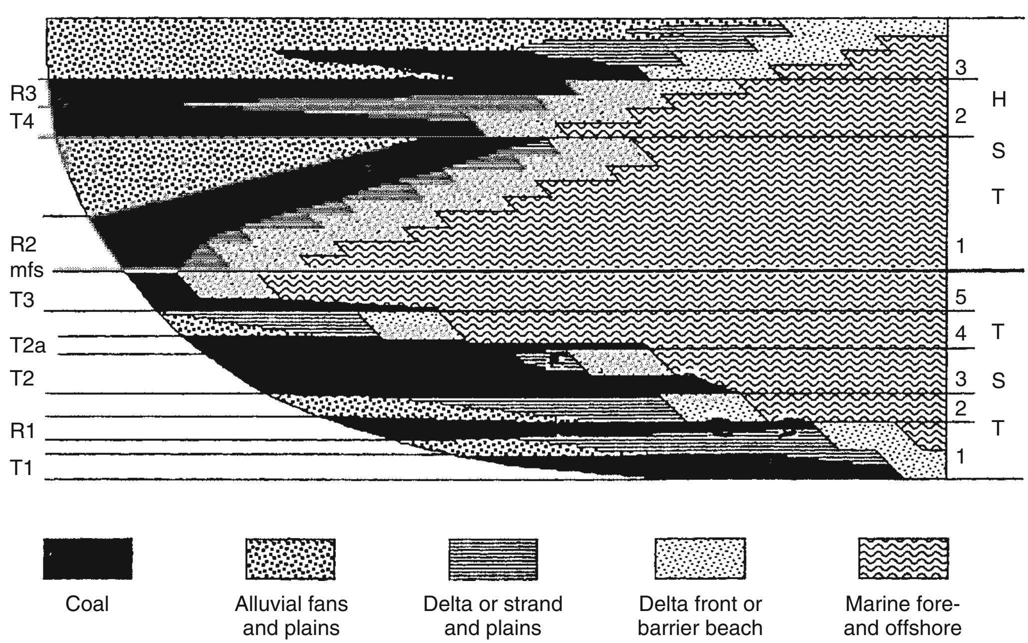



**Figure 2.13** Diagrammatic model of a number of sequence stratigraphic settings of coal seams. TST, transgressive systems tract; HST, highstand systems tract; $R_1$–$R_4$, marine regression; $T_1$–$T_4$, marine transgression. With permission from Springer-Science & Business Media. In Diessel (1992), Coal-bearing Depositional Systems, (fig.8.28).

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 is 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 recognizable and widespread marker horizons are rare. In part this stems from the very localized patterns of deposition within many coal-forming environments. However, some distinctive deposits may be present, 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 (e.g. 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 2.14 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, due 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 2.15, where borehole sections have been

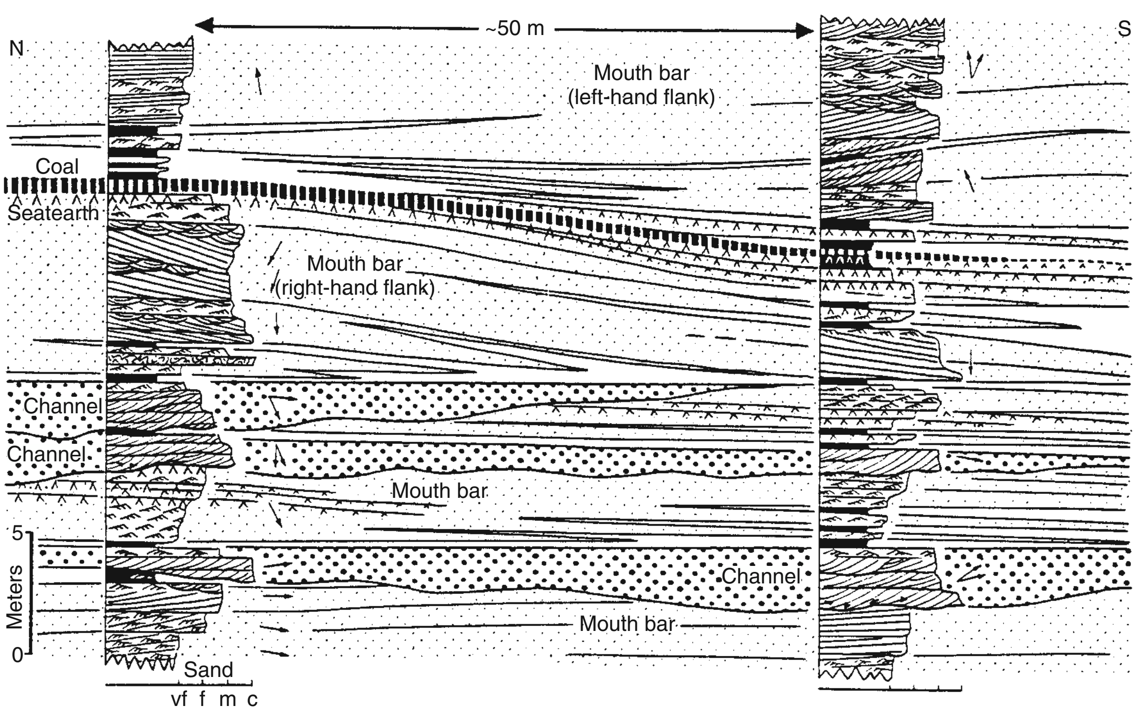



**Figure 2.14** Cross-section showing portions of superimposed mouth-bar lobes with associated seatearths and coal (Drønbreen locality, central Spitsbergen). The lobes prograde to the east (away from the viewer). (From Nemec, 1992.)

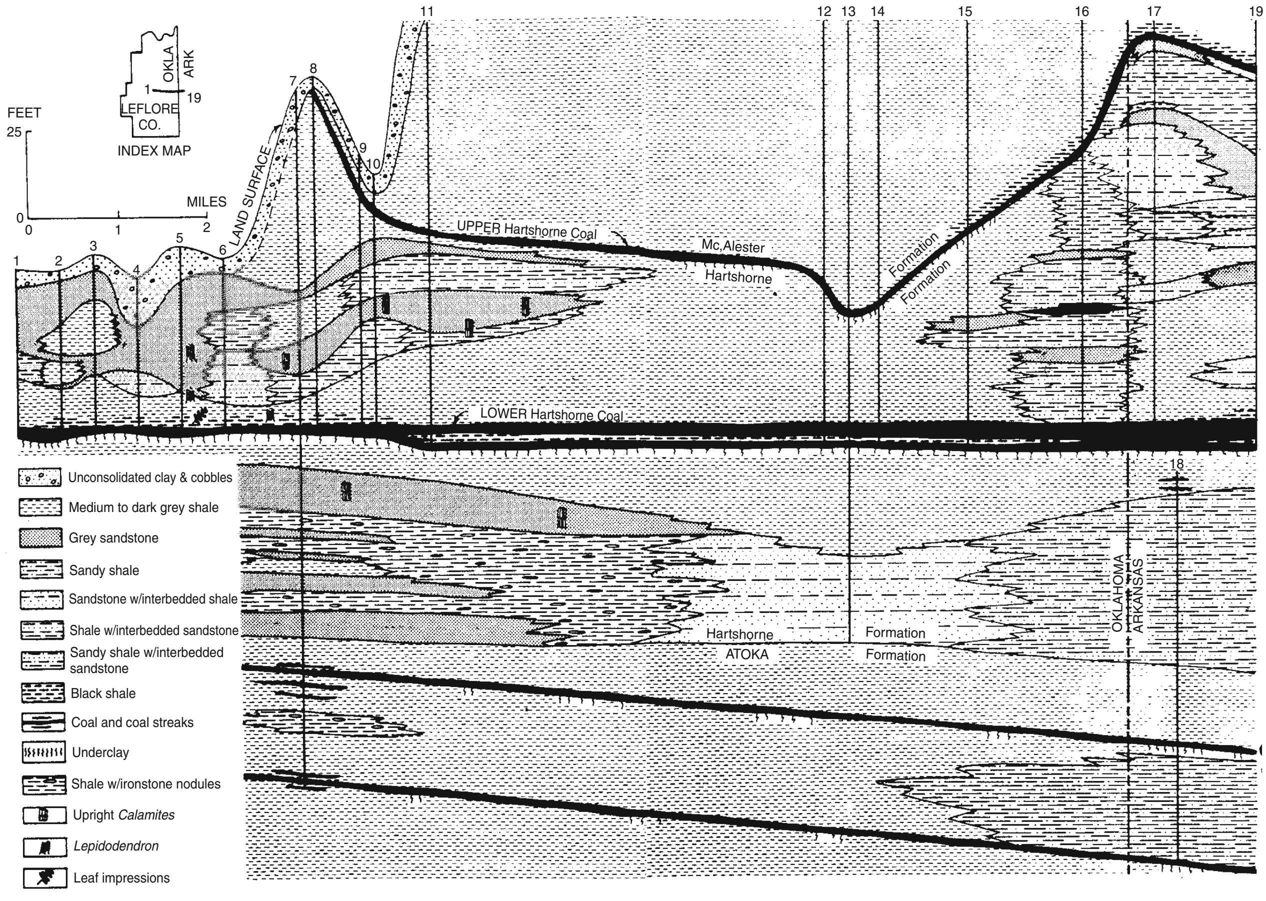



**Figure 2.15** Cross-section showing stratigraphic relations of coals and sandstones in the Hartshorne and Atoka Formations, Le Flore County, Oklahoma, and Sebastian County, Arkansas, United States. (From Rieke and Kirr, 1984.)

used to determine the facies character of coal-bearing strata in the Arkoma basin, United States (Rieke and Kirr, 1984). Similar lithotype correlations are shown in Figure 2.16 from the United States and Figure 2.17 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 2.18. Details of the variety of geophysical logs used in coal sequence correlation are described in Chapter 7.

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 2.19, 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, that is at points 2a, 2b, 2c and 2d. If Area 2 is considered for development, exploratory drill sites would be located at sites 2a to 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, interburden and overburden nature and thickness, together with the identification of igneous intrusions in the coal-bearing sequence. From borehole and surface data, all these features can be quantified and portrayed in plan or map form.

Facies maps are usually compiled for the area of immediate interest, that is the mine lease area, but plans covering larger areas can be produced which give a useful regional picture of coal development. Such a large-scale study is shown in Figure 2.20, 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 $km^2$. 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 maximize the recovery of the thicker areas of coal while avoiding the areas of ‘want’, that is those areas depleted in coal.

Figure 2.21 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 2.22 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 ‘leafs’ 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 divide further.

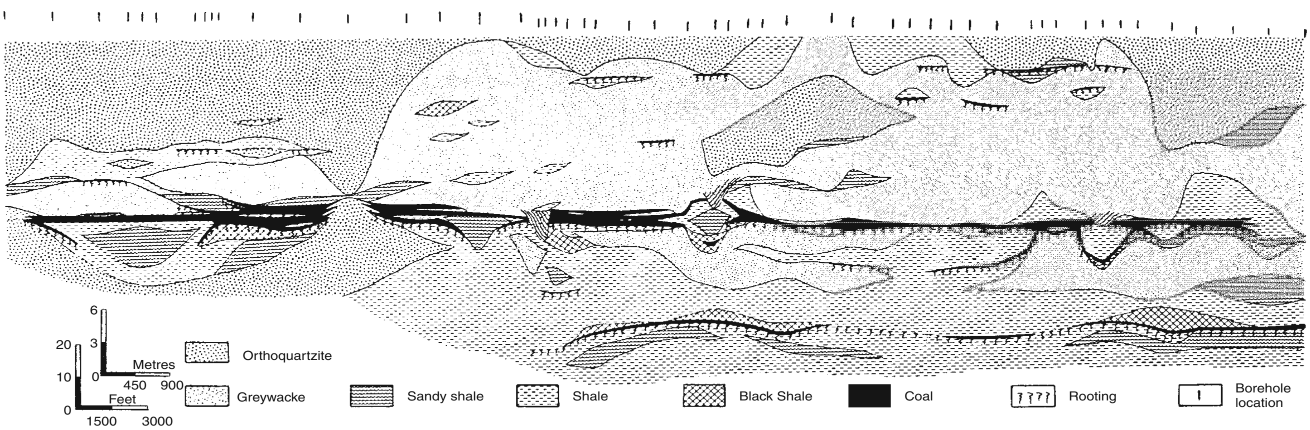



**Figure 2.16** Cross-section showing correlation of lithofacies and associated coals above and below the Beckley Seam, West Virginia, United States, based on borehole data. (From Ferm *et al.*, 1979).

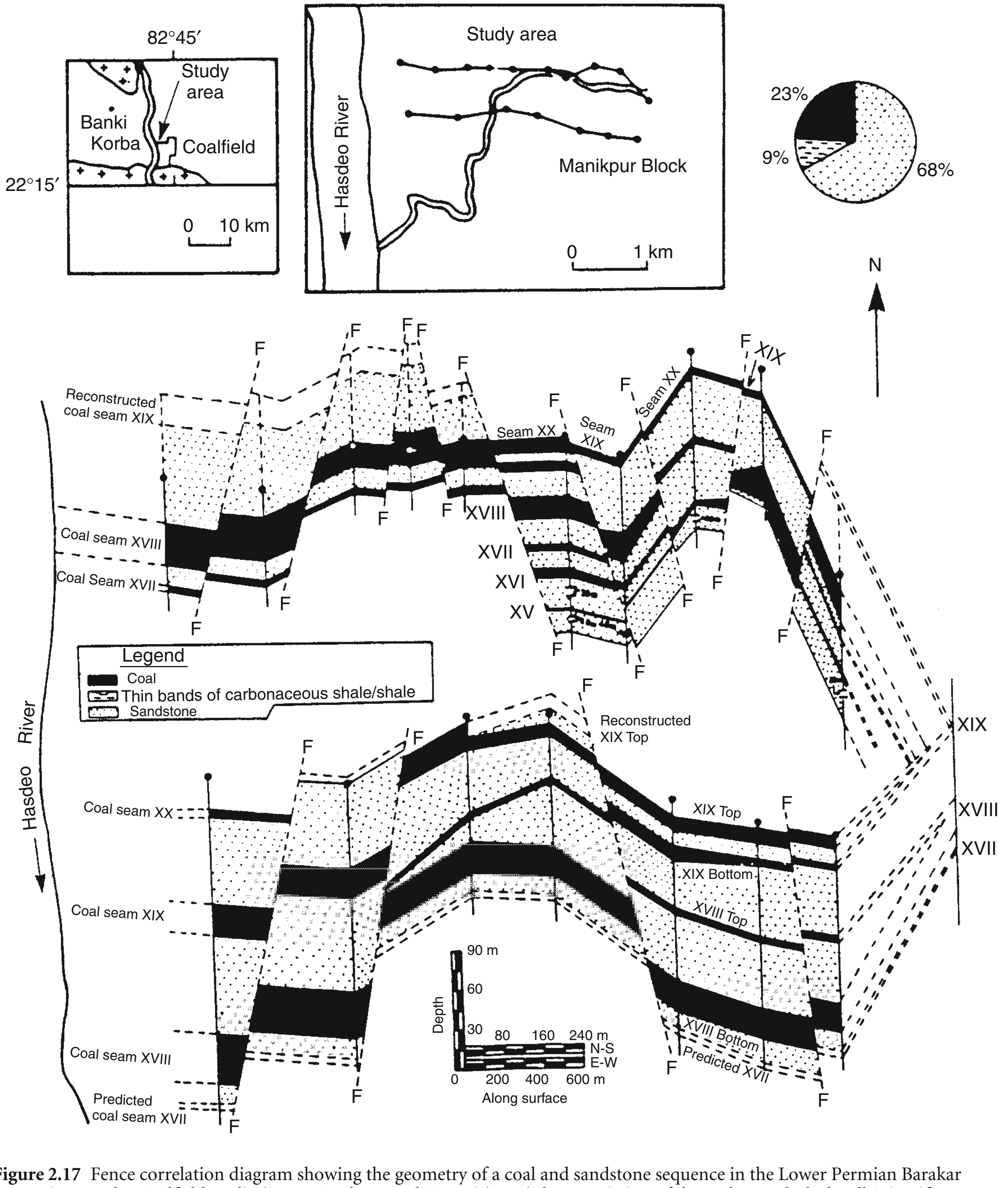



**Figure 2.17** Fence correlation diagram showing the geometry of a coal and sandstone sequence in the Lower Permian Barakar Formation, Korba Coalfield, India (From Casshyap and Tewari (1984), by permission of the author and Blackwell Scientific Publications.)

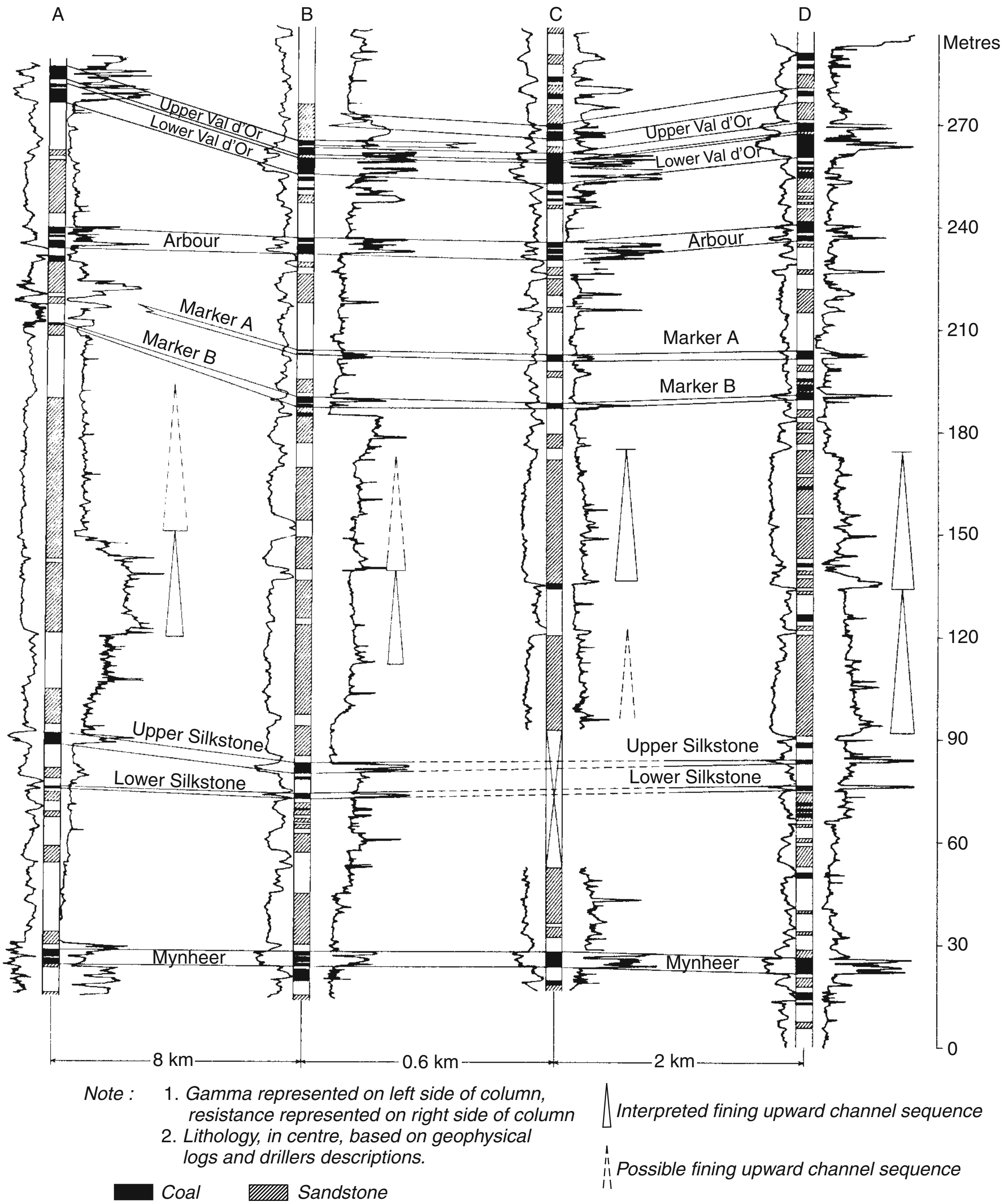



**Figure 2.18** Correlation based on geophysical logs, Coalspur Beds, Upper Cretaceous/Paleogene, Alberta, Canada. (From Jerzykiewicz and McLean (1980), Geological Survey of Canada, Department of Energy, Mines and Resources paper 79-12. Reproduced by permission of the Minister of Supply and Services, Canada.)

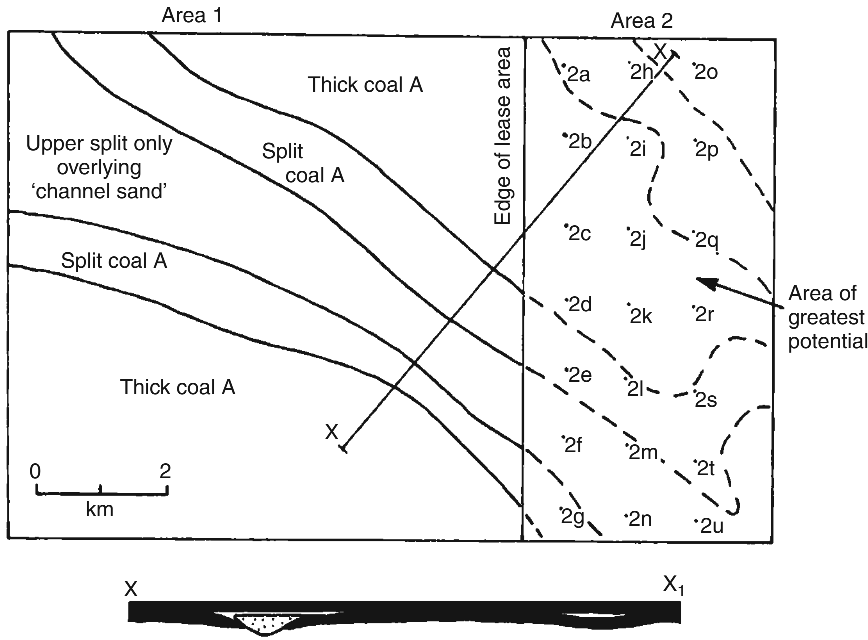



**Figure 2.19** Lithofacies map illustrating how such mapping can be extended to an adjoining area. To locate an additional area of thick coal and an area of coal split.

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 2.23 here illustrates differential splitting of a coal seam across a mine working; such variations are significant to the economics of coal mining, more particularly in underground operations.

Other types of seam splitting are known as 'S' or 'Z' splits (Figure 2.24). This feature is characterized 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, which are well documented in the United Kingdom (Fielding, 1984) and Australia, 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 interbasin areas. Coal seam splits are also formed by the influence of growth faulting (as described in section 2.3.1.2 and illustrated in Figure 2.31).

Splits are of considerable significance because 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 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, such 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 2.25). 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

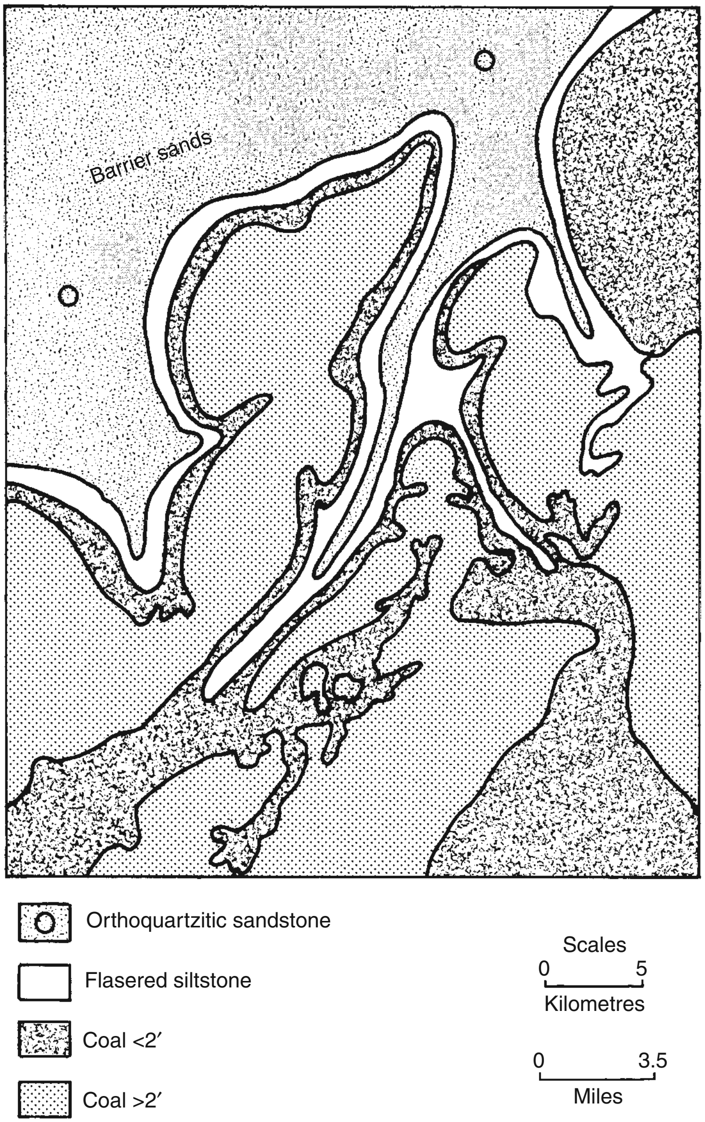



**Figure 2.20** Mapped lithotypes compiled from 1000 boreholes over an area of 1000 km$^2$, illustrating the regional depositional setting of the Beckley Seam, West Virginia, United States. (From Horne *et al.*, 1979.)

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 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 localization of channelling along the flanks of such flexures, producing elongate sandstone bodies which can influence mine planning operations.

### *2.2.6.3 Floor rolls*

These are the opposite phenomenon to washouts, and are characterized 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

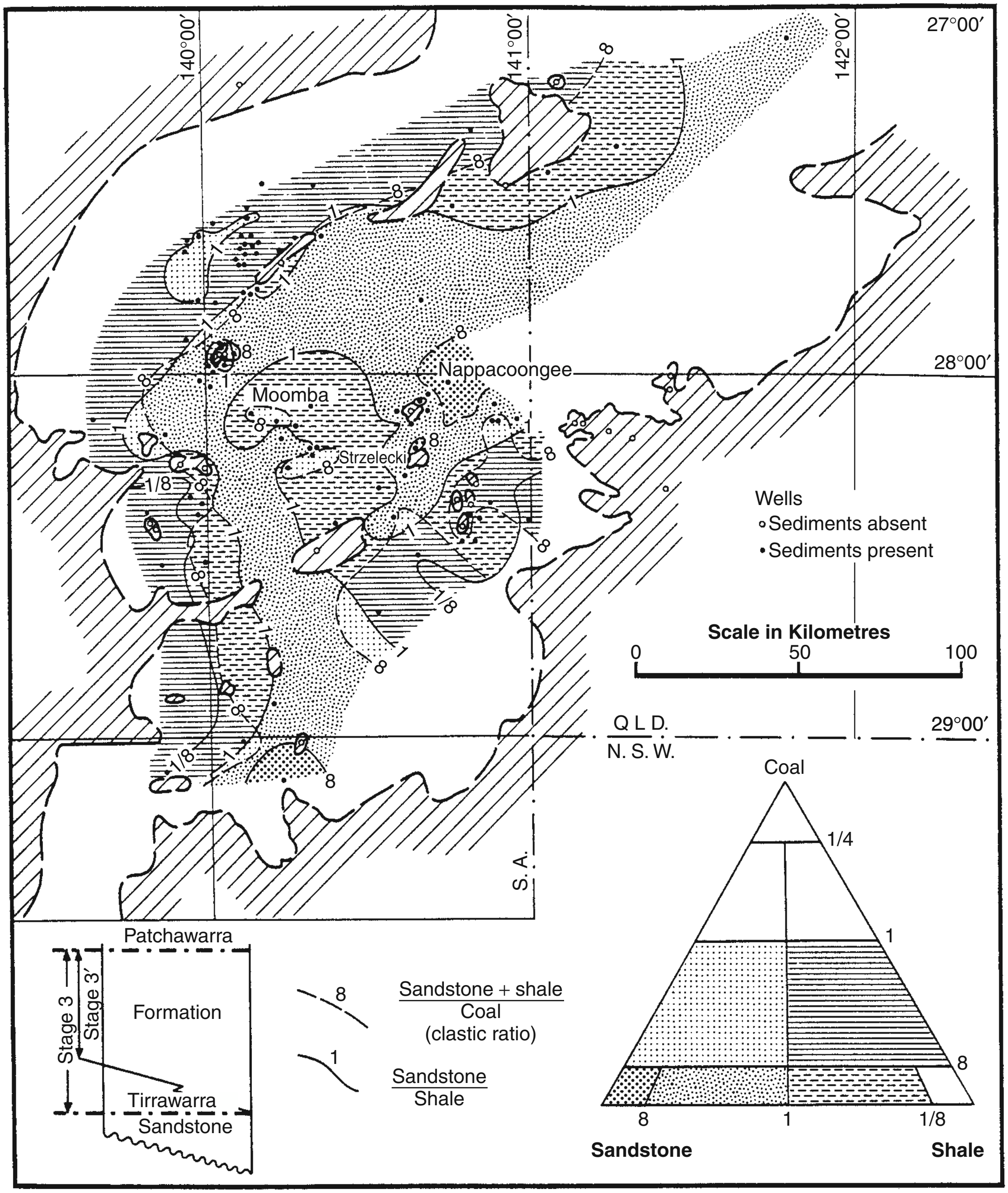



**Figure 2.21** Lithofacies map of part of the Patchawarra Formation (Permian) South Australia, showing sandstone/shale and sandstone + shale/coal ratios. (From Thornton, 1979.)

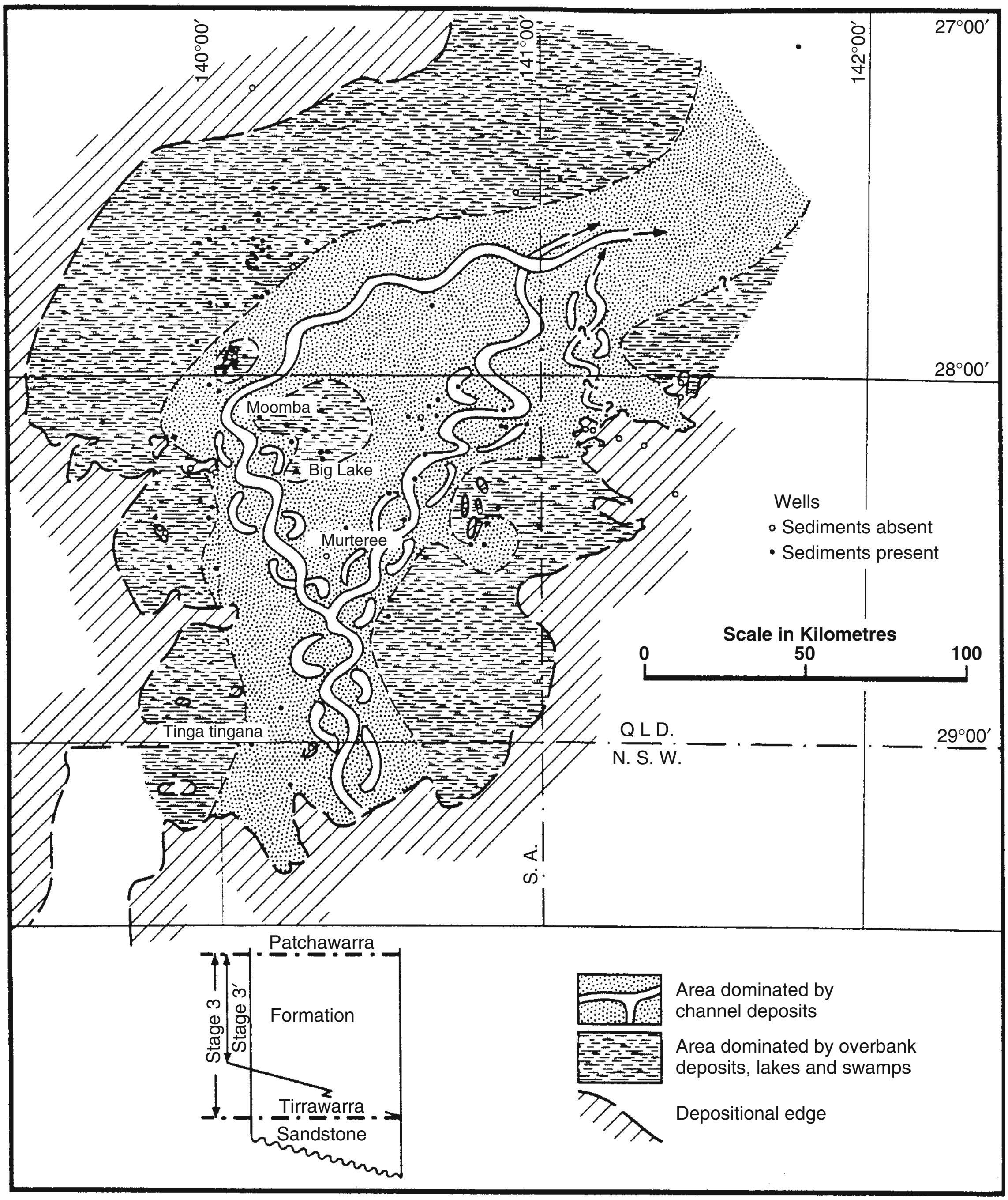



**Figure 2.22** Palaeogeographical reconstruction of the same interval as shown in Figure 2.21, indicating that the area of high clastics to coal ratio represents an area dominated by fluvial channel deposits. (From Thornton, 1979.)


**Figure 2.23** Development of a coal seam splitting in the Beckley Seam across a mine working. (From Ferm *et al.*, 1979.)

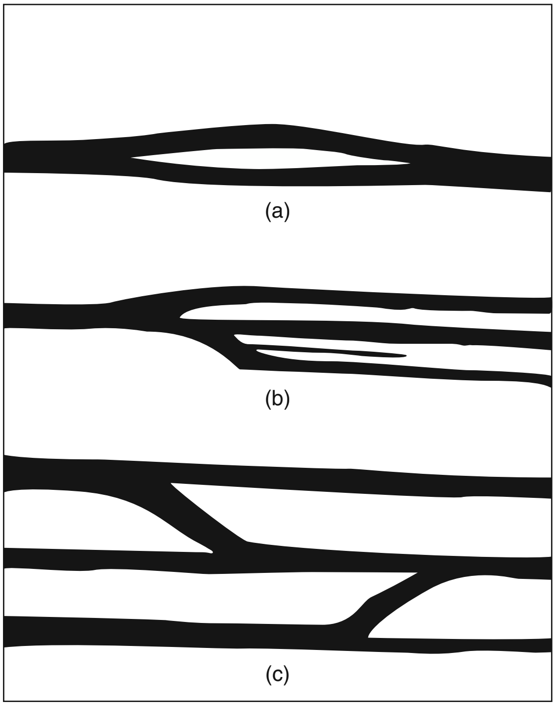



**Figure 2.24** Common types of coal seam split. (a) Simple splitting; (b) multiple splitting; and (c) 'Z' or 'S' shaped splitting.

differential compaction of peat around clastic deposits in the lower part of the seam as the upper part of the seam accumulates.

### 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 2.26 shows two contour maps of the northeastern Fuxin Basin, P. R. China (Wu, Li and Cheng, 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 2.21).

The importance of these is self evident because, depending on the economics of the mine site, coal that is 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, that is makes a poor floor and roof to the seam. Conversely there can be problems 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 2.27 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 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 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, that is 10:1–15:1. In addition if the

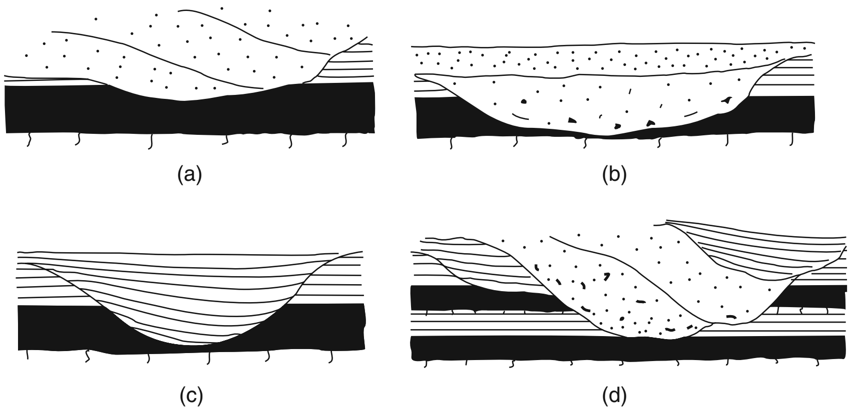


Figure 2.25 Channelling in coal seams. (a) Sand-filled channel producing a sandstone roof to the coal seam. (b) Sand and coal detritus filled channel with coal seam eroded. (c) Mudstone filled channel with coal seam eroded. (d) Multiple channel sequence with sandstone and mudstone fills – the channel has removed the upper leaf of the coal seam.

lithotypes include hard indurated sandstone that will require blasting, then this is an added cost which 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, this 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 summarized 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 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.

## 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,

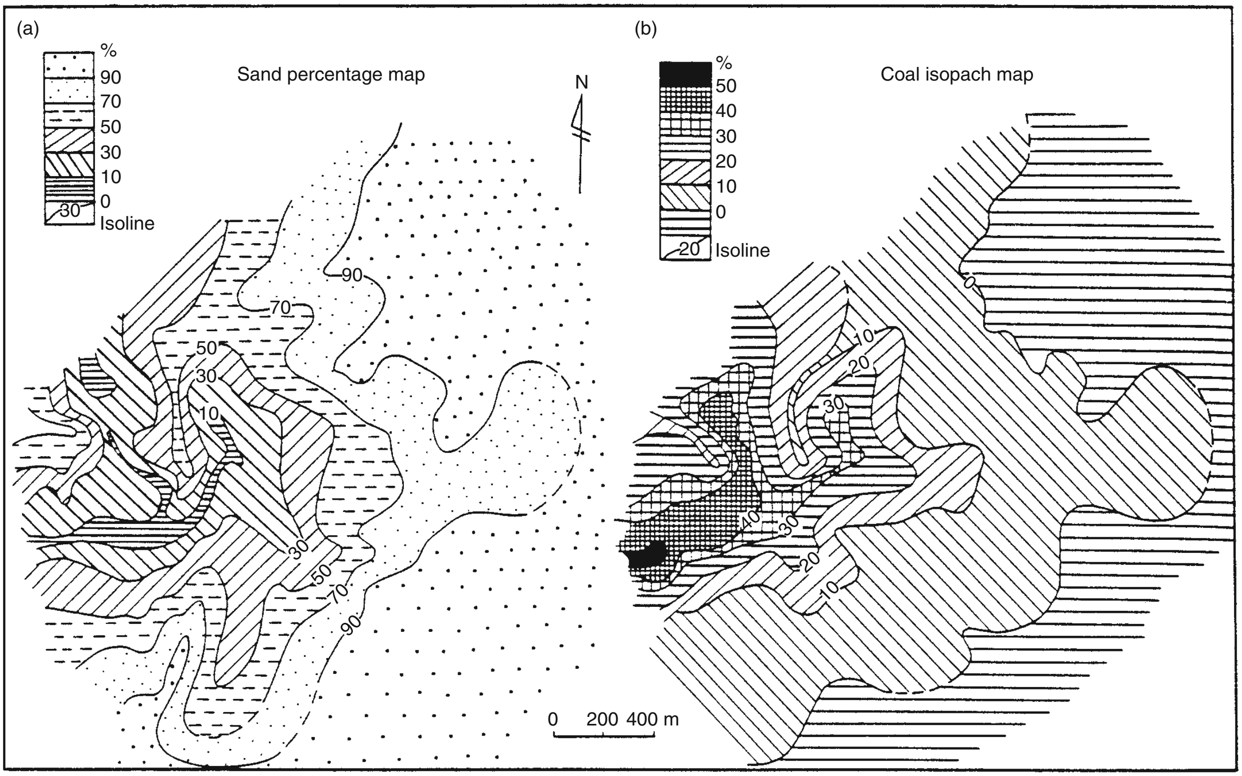



**Figure 2.26** Isopach maps of the northeastern Fuxin Basin, PRC showing (a) percentage of sandstone and (B) coal isopachs for the middle part of the Haizhou Formation (Lower Cretaceous), which demonstrate the relationship of increased sandstone thickness with decreasing coal thickness. (From Wu, Li and Cheng, 1992.)

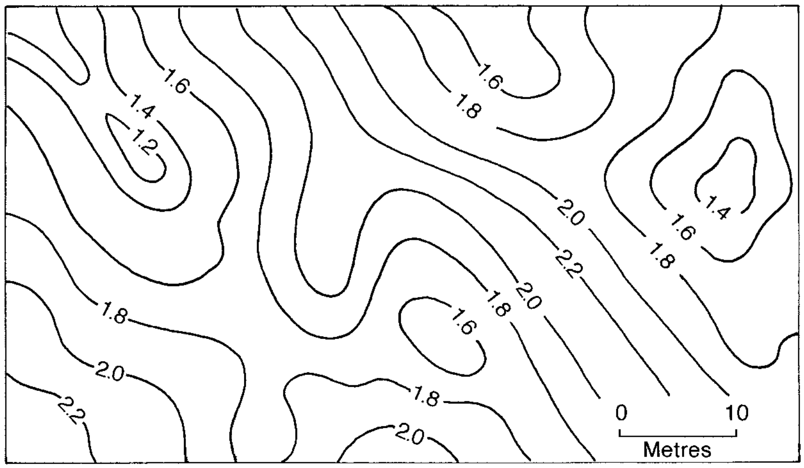



**Figure 2.27** Hypothetical example of a coal seam thickness isopach map (thickness values are in metres).

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 when 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 liquifaction effects, with the latter being characterized 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 2.28).

Instability within environments of deposition, whether induced by fault activity or simply by overloading of accumulated sediment, can produce movement of sediments in the form of gravity flows; Figure 2.29 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 2.27 shows a coal that has become intermixed with the surrounding sediment and is 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, United States, 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, United Kingdom, growth faults have again influenced sedimentation, here, in addition to active basement elements, faults are developed that owe their origin to

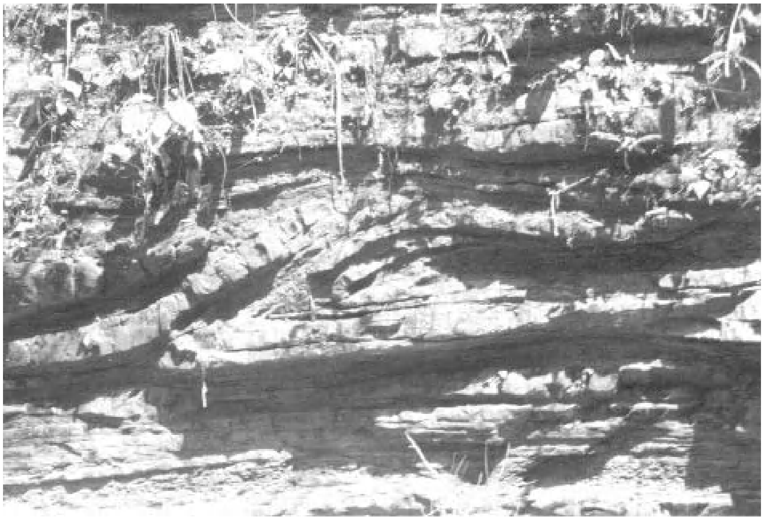


**Figure 2.28** Deformed bedding in Paleogene–Neogene coal-bearing sediments, East Kalimantan, Indonesia: hammer length is 37 cm. (Photograph by LPT.)

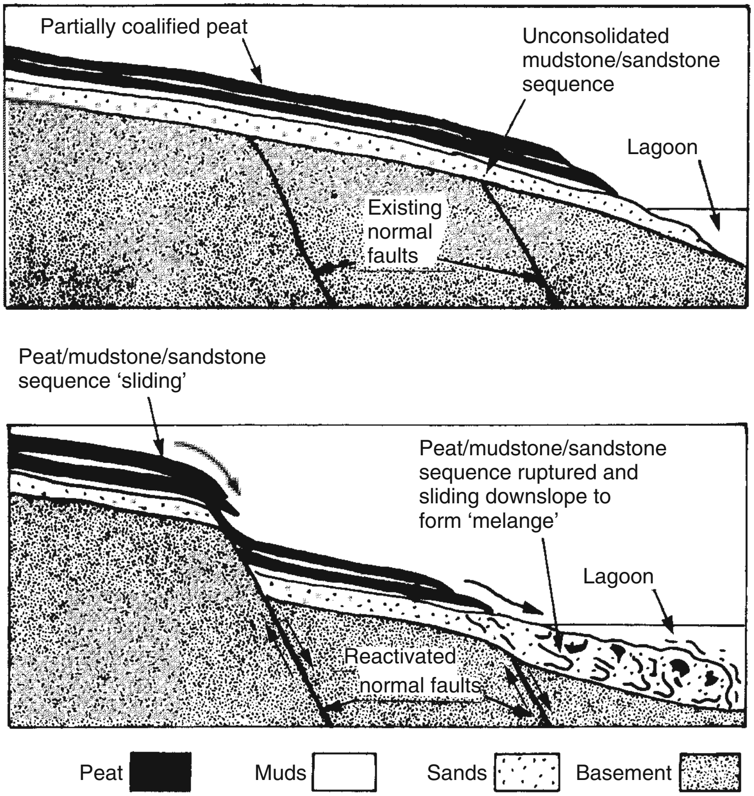



**Figure 2.29** Normal fault reactivation causing instability in a partially coalified peat sequence, with downslope slumping to produce a 'melange' of coal and intermixed sediment.

gravity sliding within the sedimentary pile (Elliott and 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 flatten 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.

(a) (b)

**Figure 2.30** Cores exhibiting "melange" or mixing of lithotypes due to gravity sliding in Paleogene-Neogene coal-bearing sediments, East Kalimantan, Indonesia. (a) left core: mixing of sandstone & siltstone with subordinate coal centre core: coal mixed with mudstone & siltstone right core: coal & mudstone mixing (b) All cores and stone/siltstone/coal mixing. (Photographs by LPT.)

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 resumed at the original level of the first peat. Figure 2.31 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, which 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 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 over-steepened fold axes. 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 these are 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 Bełchatów opencast coal mine in Poland, a large diapir has intruded into the coal-bearing sequence, dividing the coal reserve into two distinct areas (Figure 2.32).

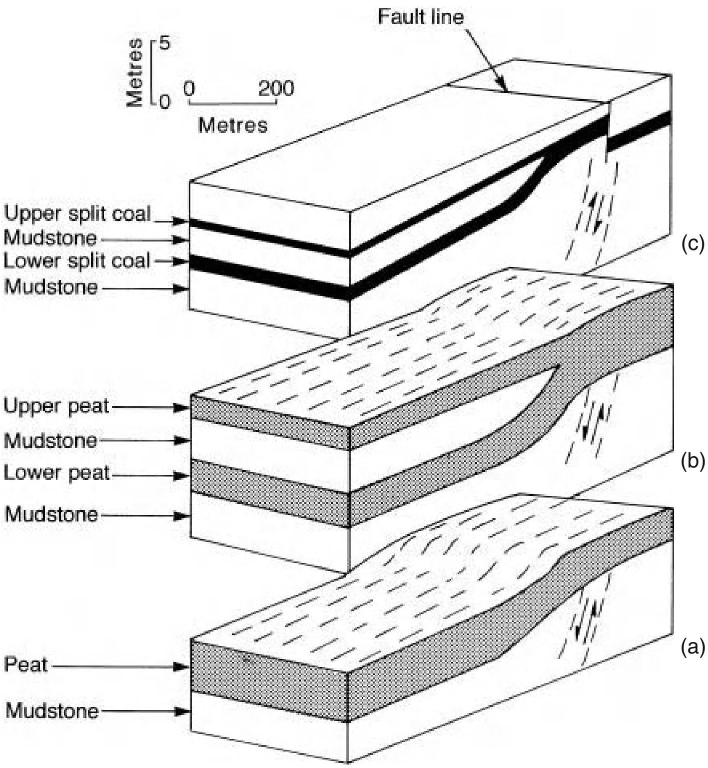



**Figure 2.31** Seam splitting caused by differential movement of faults during peat deposition. (a) Fault downthrow results in downwarping of the peat. (b) Downwarp filled in with mudstone, peat development resumed at original level. (c) Fault throw sense reversed, uplifting split coal and downthrowing unsplit section of the coal seam. (From Broadhurst and Simpson, 1983.)

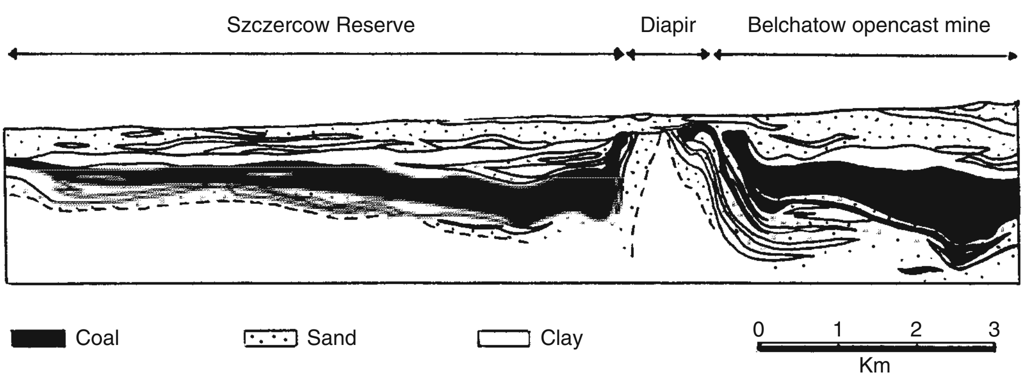



**Figure 2.32** Section across Belchatow opencast mine showing effect of a diapiric intrusion into the thick coal seam. The diapir in this instance is a salt dome and effectively divides the coal reserves into two distinct areas.

In the Kutei Basin in East Kalimantan, Indonesia, the established structural pattern continually evolved throughout the Paleogene–Neogene Periods. 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, and the resultant sequence is characterized in its upper part by upper delta plain and alluvial plain sedimentation with numerous coals. This structural pattern is shown later in Figure 2.46b.

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 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 thrusted and folded coalfields usually containing high rank coals.

These post-depositional structural elements can be simply summmarized as faults, joints (cleat), 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/cleat in coal*

Coal, and in particular all ranks of black coal, is noted for the development of its jointing, more commonly referred to as cleat. 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 2.33). Through-going cleats formed first and are referred to as face cleats, cleats that end at intersections with through-going cleats formed later and are called butt cleats. These fracture sets and partings along bedding planes impart a blocky character to coal. Figure 2.34a shows a well developed orthogonal cleat pattern in a Carboniferous bituminous coal from the Midlands, United Kingdom, and Figure 2.34b shows a strong cleat pattern in an anthracite of Triassic age from the Republic of Vietnam. Figure 2.34c shows cleat development together with conchoidal fracture in a Paleogene–Neogene brown coal from Republic of Serbia. Figure 2.34d 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

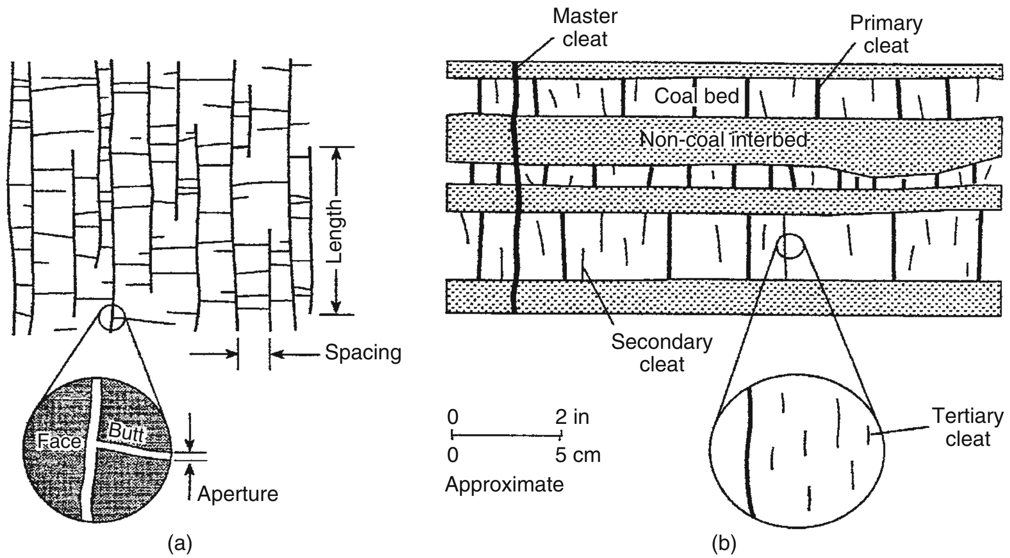



**Figure 2.33** Schematic Illustration of Cleat Geometries (a) Cleat-trace patterns in plan view. (b) Cleat hierarchies in cross-section view. These conventions are used for cleat: length is dimension parallel to cleat surface. And parallel to bedding; height is parallel to cleat surface and perpendicular to bedding; aperture is dimension perpendicular to fracture surface. Spacing between two cleats is a distance between them at right angles to cleat surface (Laubach 1998). (Reproduced with permission, Elsevier Publications.)

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 medium volatile bituminous coal, increasing to anthracite coals. Law (1993) found cleat spacing ranged from 22 cm in lignites (Ro (vitrinite reflectance) values of 0.25–0.38%) to 0.2 cm in anthracites (Ro 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

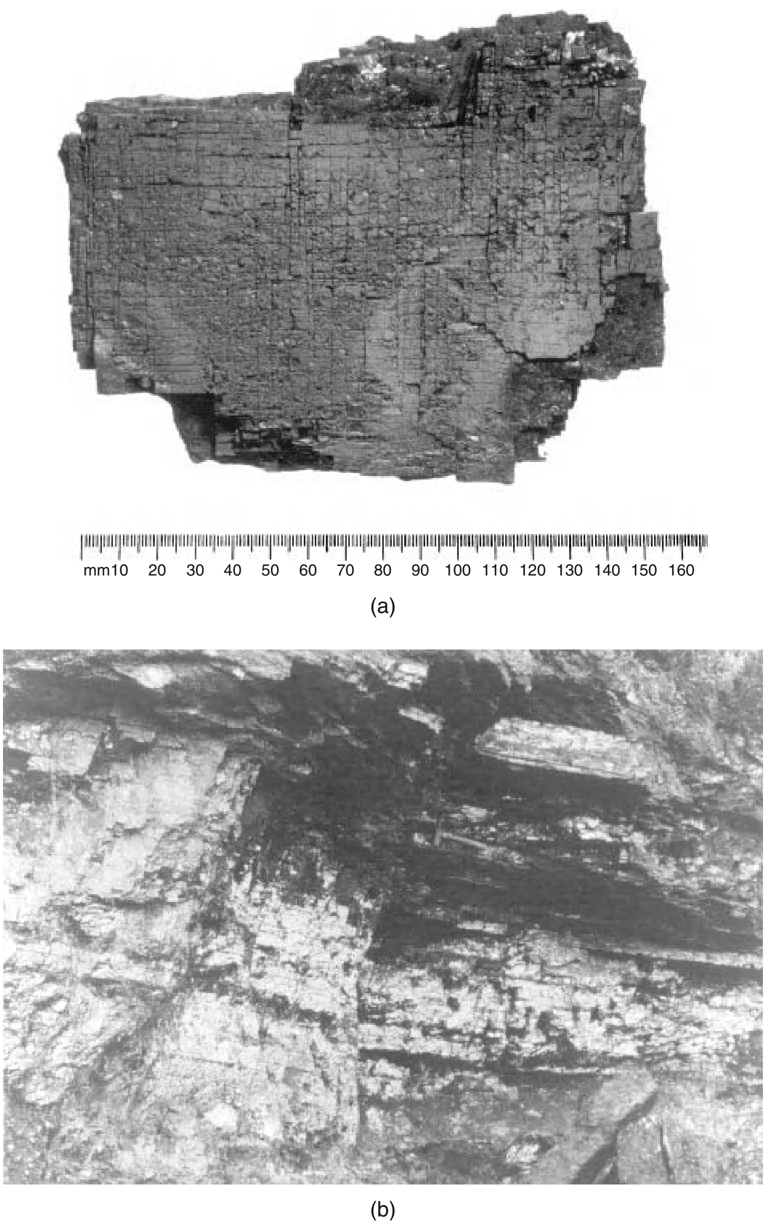



**Figure 2.34** (a) Orthogonal cleat pattern in Meltonfleet Coal, Upper Carboniferous, Yorkshire, United Kingdom: hammer length is 37 cm. (Reproduced by permission: IPR/25-6C British Geological Survey. © NERC.All rights reserved.) (b) Well developed joint and cleat pattern in Triassic anthracite, Republic of Vietnam. (Photograph by LPT.)

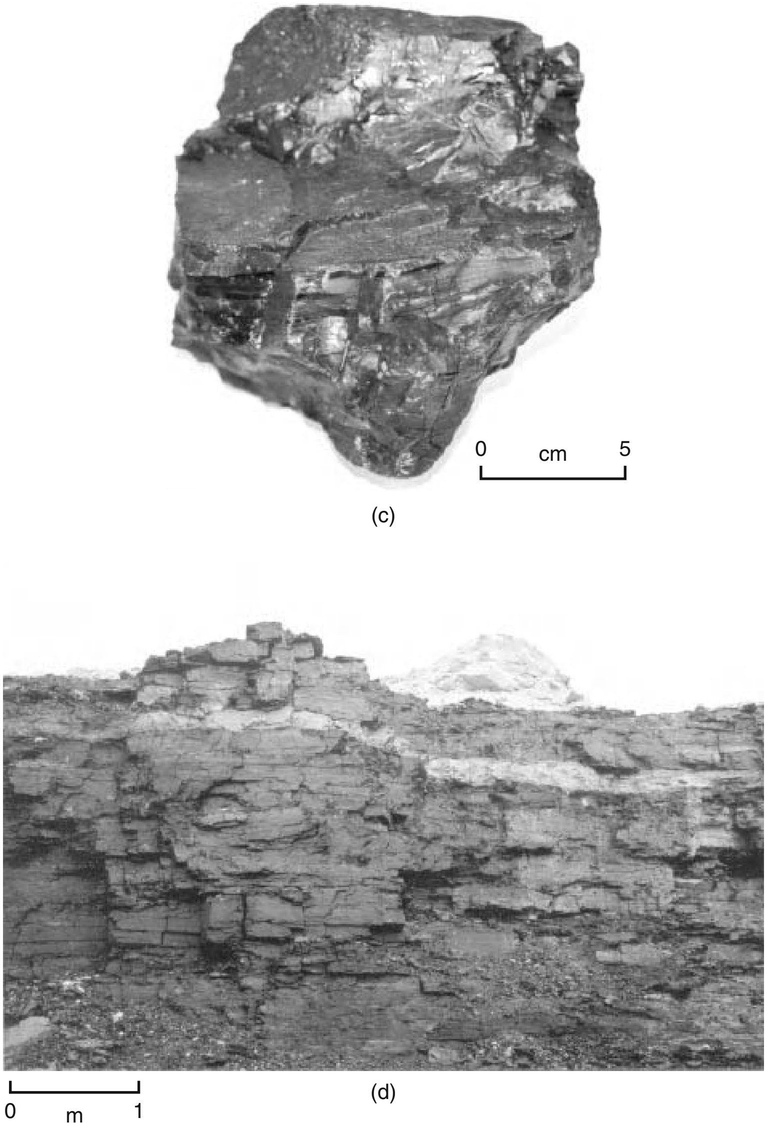



**Figure 2.34** (c) Cleat development and conchoidal fracture in Paleogene–Neogene brown coal, Republic of Serbia. (Photograph by courtesy of Dargo Associates Ltd.) (d) Well developed joint pattern in Permian (Gondwana) coal, central India. (Photograph by courtesy of Dargo Associates Ltd)

regular variations in cleat spacing with rank in the range lignite to high volatile bituminous coal might not occur. Another parameter that has been studied with regard to cleat spacing is coal type and ash content. Dawson and Esterle (2010) have 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 do dull coal lithotypes (Durain) (Stach, 1982). Coals with low ash content have smaller cleat spacings than do coals with high ash content. 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; principal fault types are described briefly in the following paragraphs.

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

This form of faulting is common, movements can be in the order of a few metres to hundreds of metres. Figure 2.35 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 2.36 shows a local highwall termination due to faulting in an opencast mine in Bosnie-Hertzegovina, and Figure 2.37 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 monoclinal flexure, particularly in overlying softer strata. Such faulting also produces drag along the fault plane, 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 United Kingdom shown in Figure 2.38.

Large-scale normal faults are produced by tensional forces pulling apart, or spreading, the crustal layer; where these faults run parallel, with the 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.

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 2.39. Lag faults are common in the coalfield of South Wales, United Kingdom.

**Reverse faults** are produced by horizontal stress with little vertical compression, which 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 $<45°$) are more significant. A typical reverse fault structure is seen in Figure 2.40, 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

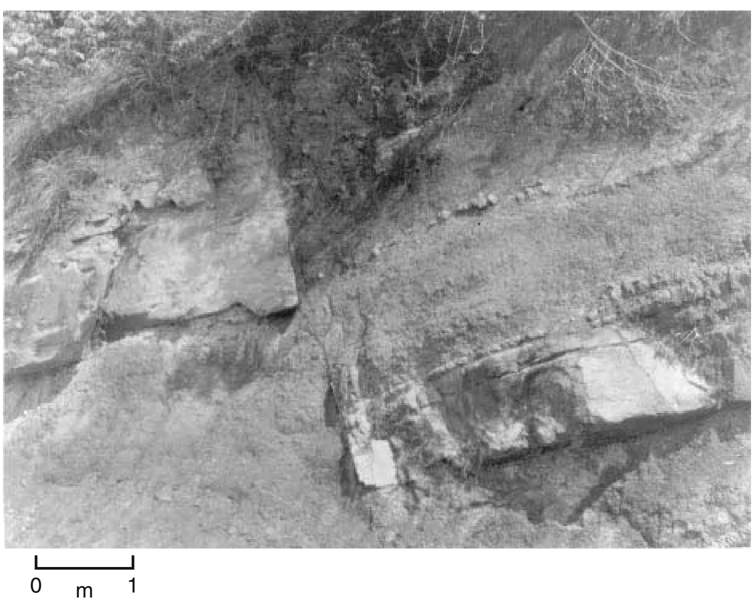



**Figure 2.35** Normal fault with downthrow of 2 m to the right. Paleogene–Neogene coal-bearing sediments, Sumatra, Indonesia. (Photograph by LPT.)

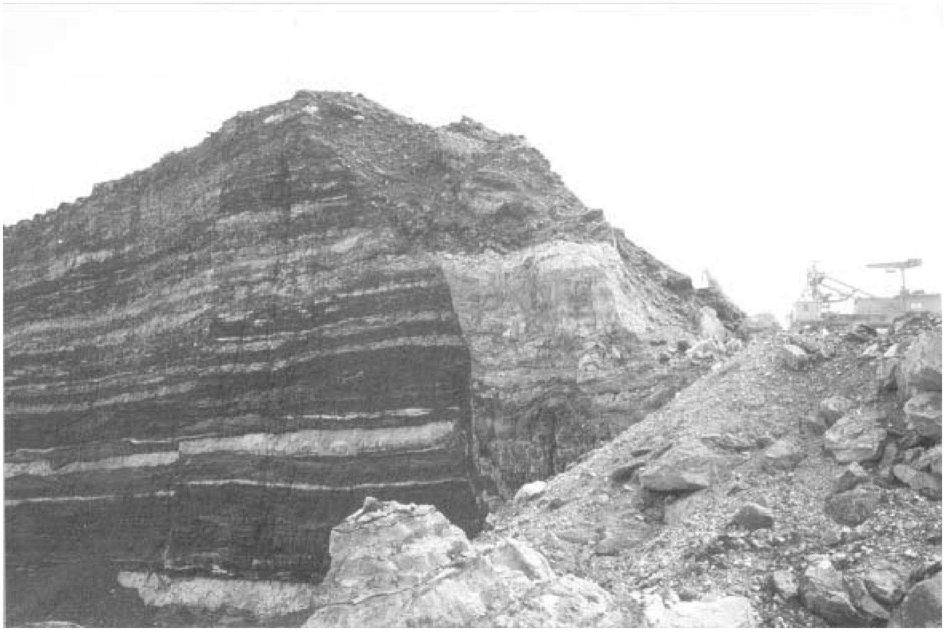


**Figure 2.36** Highwall termination due to faulting in Paleogene–Neogene brown coal opencast mine in Bosnia–Herzegovina: height of face is 10 m. (Photograph by courtesy of Dargo Associates Ltd.)

**Figure 2.37** Normal fault downthrowing overburden (light colour) against a coal seam. In an opencast mine in central India: height of face is 10 m. (Photograph by courtesy of Dargo Associates Ltd.)

may prefer to follow the bedding plane rather than to cut across them.

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, the seatearths often acting as a lubricant. One detrimental effect is the contamination of the coal seam with surrounding country rock, thereby reducing its quality and, in some cases, its minability.

In highly tectonized coal deposits, a great number of coal seam contacts have undergone some movement and shearing, in some cases the whole seam will have

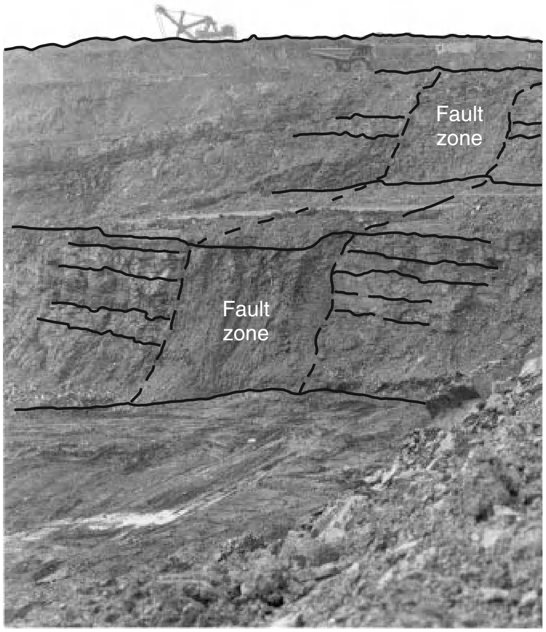



**Figure 2.38** Large fault zone exposed in high wall in an opencast mine, South Wales, United Kingdom. (Photograph by LPT.)

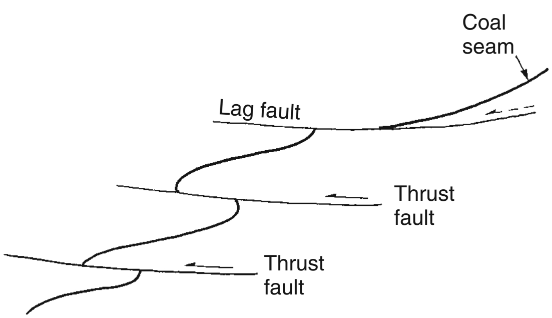



**Figure 2.39** Lag fault produced by retardation of the upper part of the sequence during the forward movement of the lower sequence by thrust faulting. (From Sherborn Hills (1975). *Elements of Structural Geology*, 2nd edn, with kind permission of Kluwer Academic Publishers.)

been compressed and moved, this may be displayed in coals as arcuate **shear planes** throughout, as portrayed in Figure 2.41.

The development of **thrust zones** in coal sequences can be illustrated in Figure 2.42, where 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 thrusting**. Such events are particularly common in coalfields that have suffered crustal shortening, as is the case in South Wales, United Kingdom (Gayer *et al.*, 1991), and in the Appalachians, United States. Thrusting is also accentuated where coal and mudstone sequences are sandwiched between thick sequences of coarse clastic rocks, the upper and lower portions of the sequence reacting to compressive forces quite differently to the incompetent coals and mudstones.

**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, and although important, 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, which 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 in mine workings in particular, 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). 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 of structural intensity. On a larger scale, the Coal Bedding Code is based on the principle that with increasing frequency of tectonic fracturing, the bedding planes in coal seams become progressively obscured. Frodsham and Gayer (1999) describe five categories of bedding plane obscurity and large-scale fracture intensity

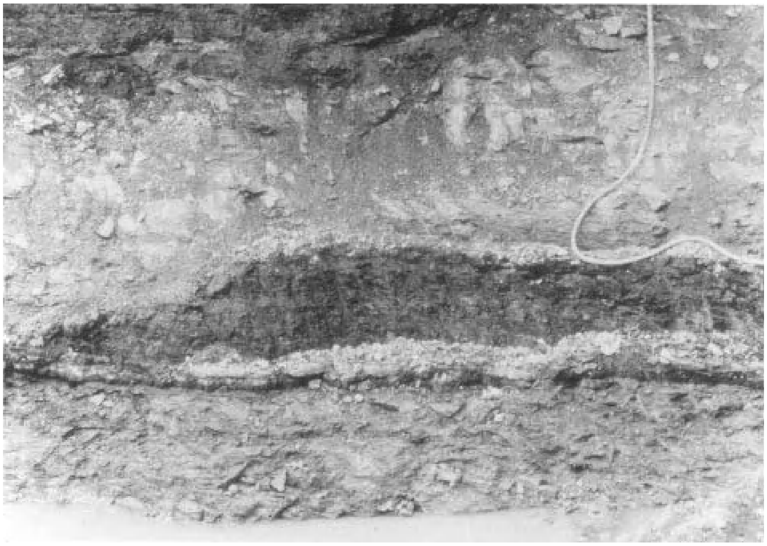


**Figure 2.40** Coal seam dislocated by reverse fault, throw 1.5 m, United Kingdom opencast mine. (Photograph by M. C. Coultas.)

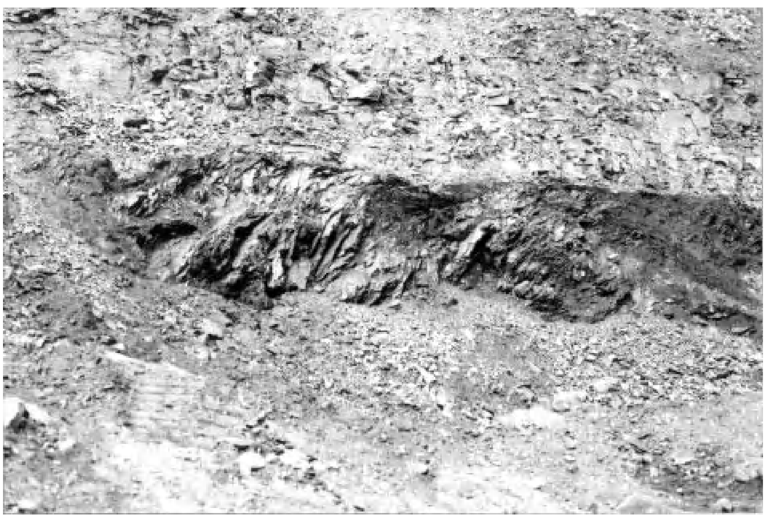


**Figure 2.41** Highly sheared anthracite coal seam (seam thickness 1.2 m) in opencast mine, South Wales, United Kingdom. (Photograph by LPT.)

within coal seams from South Wales, United Kingdom (Figure 2.43). 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 have 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, for example as shown in Figure 2.44. 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

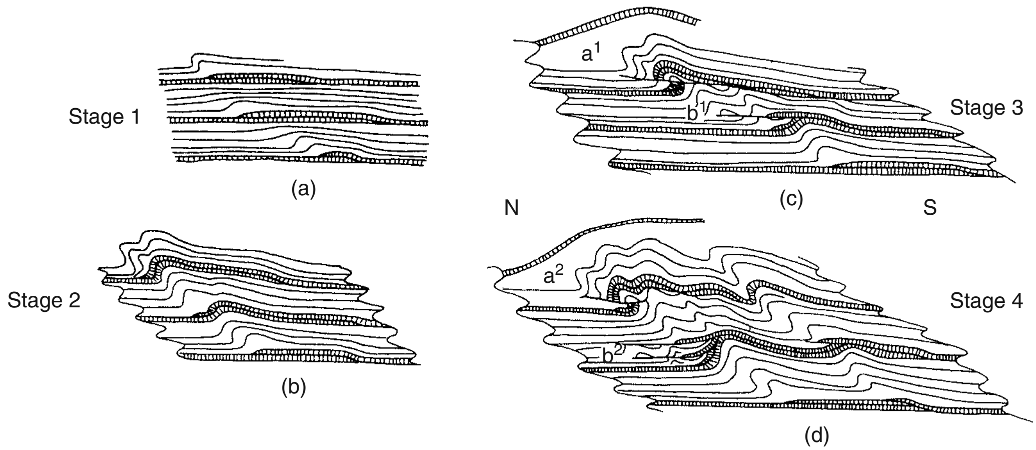



**Figure 2.42** Model for four stages in progressive easy-slip thrusting. (a) Thrusts develop simultaneously as flats along the floors of overpressured coal seams, cutting up to the roofs of the seams along short ramps; propagation folds grow at the thrust tips. (b) Thrusts continue to propagate with 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. (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 fractures; 1–2 polished | Strong, bright and hard but easily split along fractures |
| Normal | 4 | Mainly non-tectonic, several tectonic 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).

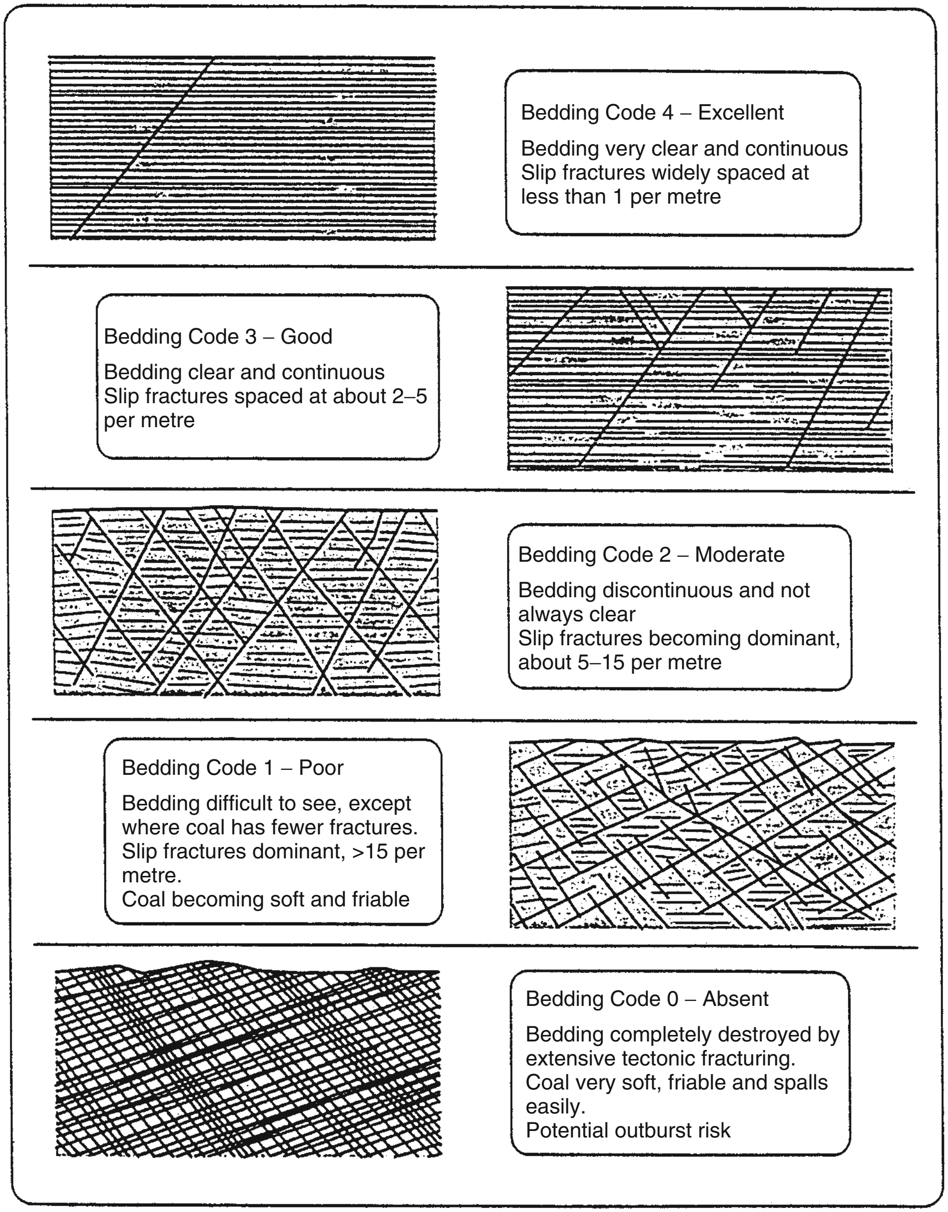



**Figure 2.43** The Coal Bedding Code, showing five categories of bedding plane obscurity and large-scale fracture intensity within coal seams in South Wales, United Kingdom. (From Frodsham and Gayer, 1999.)

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 evidence 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.

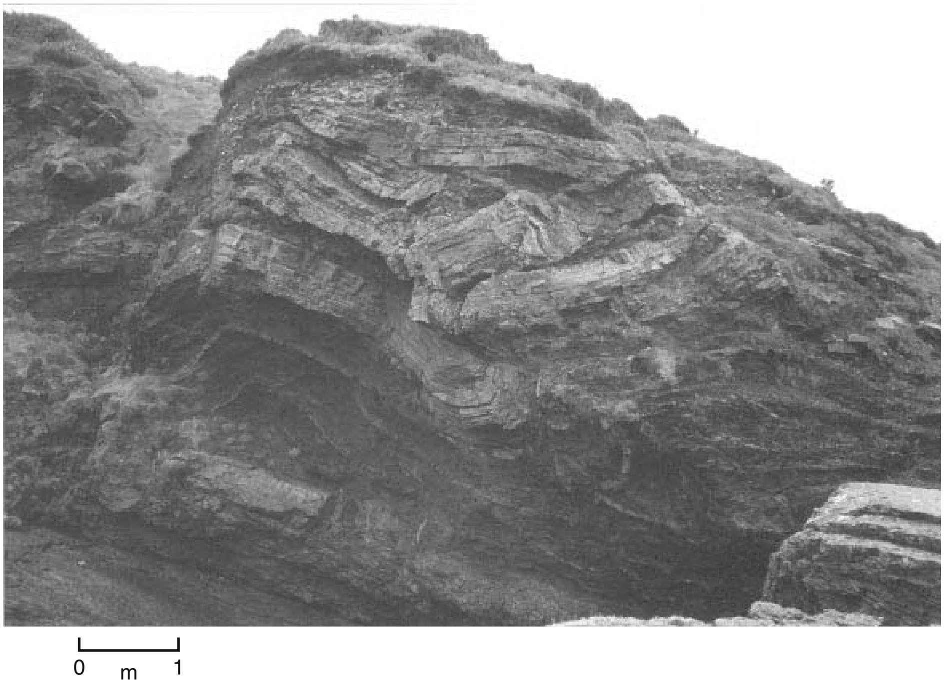


**Figure 2.44** Intensely folded Carboniferous coal-bearing sediments, Little Haven, Pembrokeshire, United Kingdom. (Photograph by courtesy of Dargo Associates Ltd.)

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, prevent further extraction. Therefore it is important to be sure that all readings taken reflect the true nature of the structure in the area of investigation.

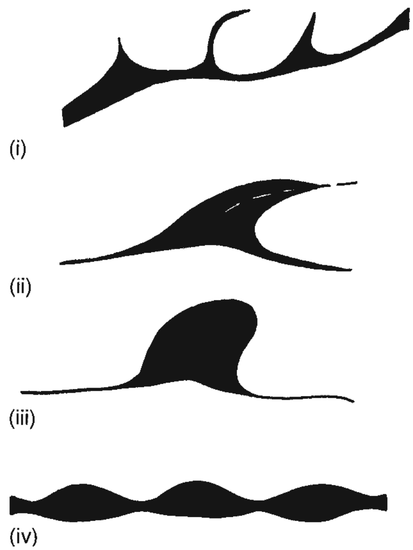
Compression of coal seams during folding can produce tight anticlinal folds with thrusting along the nose of the fold, these have been termed queue anticlines. Coal seams can be pinched out along the fold limbs and appear to have flowed into the axial areas of the anticlines. Where this occurs from two directions approximately normal to one another, coals can be concentrated in 'pepperpot' type structures. Such features are usually found only in highly tectonized coalfields, and examples of such intense deformation are illustrated in Figure 2.45a, and 2.45a coal seam squeezed in this way is shown in Figure 2.45b, here the coal has been compressed in and around the overlying sandstone. In many instances such structural complication will render a coal seam unminable 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 2.46a, where a pattern of zig-zag folds characterizes the Wurm Coal Basin, Germany, and Figure 2.46b, which shows a series of asymmetrical folds with associated thrusting from the Kutei Basin, Indonesia. 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.

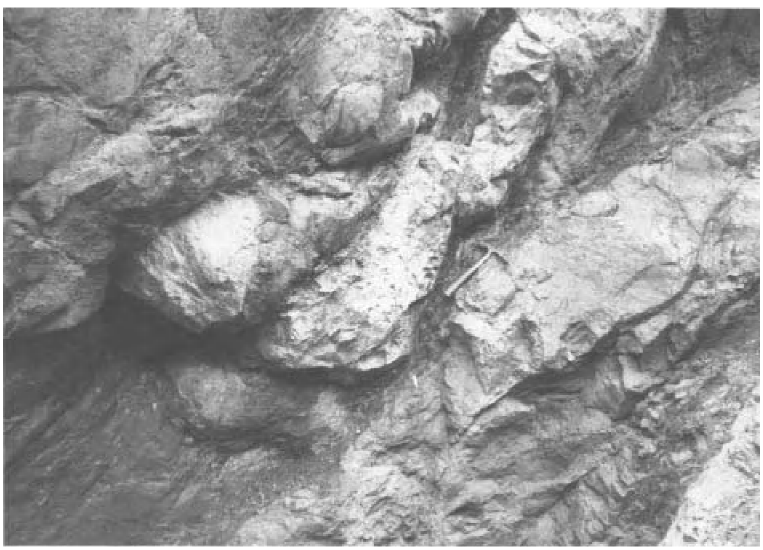
### *2.3.2.4 Igneous associations*

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 a marked loss




(a)



(b)

**Figure 2.45** (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. (Photograph by LPT.)

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, and good examples are found in Indonesia and the Philippines where Paleogene–Neogene subbituminous 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. For example in the Republic of Korea acidic dykes and sills are intruded into the coals; Figure 2.47 shows acidic igneous material intruding a coal seam in an underground working.

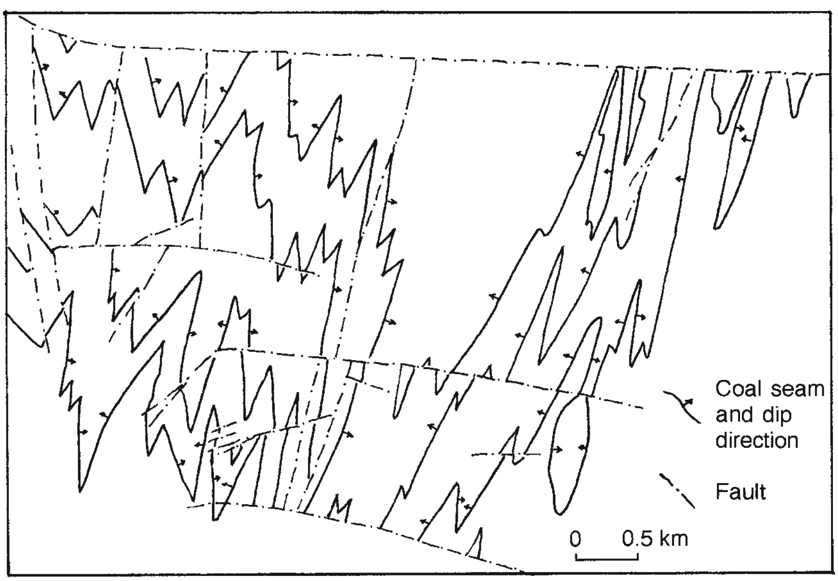
In areas where igneous intrusions are prevalent in mine workings, plans showing the distribution and size of igneous bodies are required in order to determine areas of volatile loss where the coal has been baked, and because of the hardness of the igneous material, tunnelling has to be planned with the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

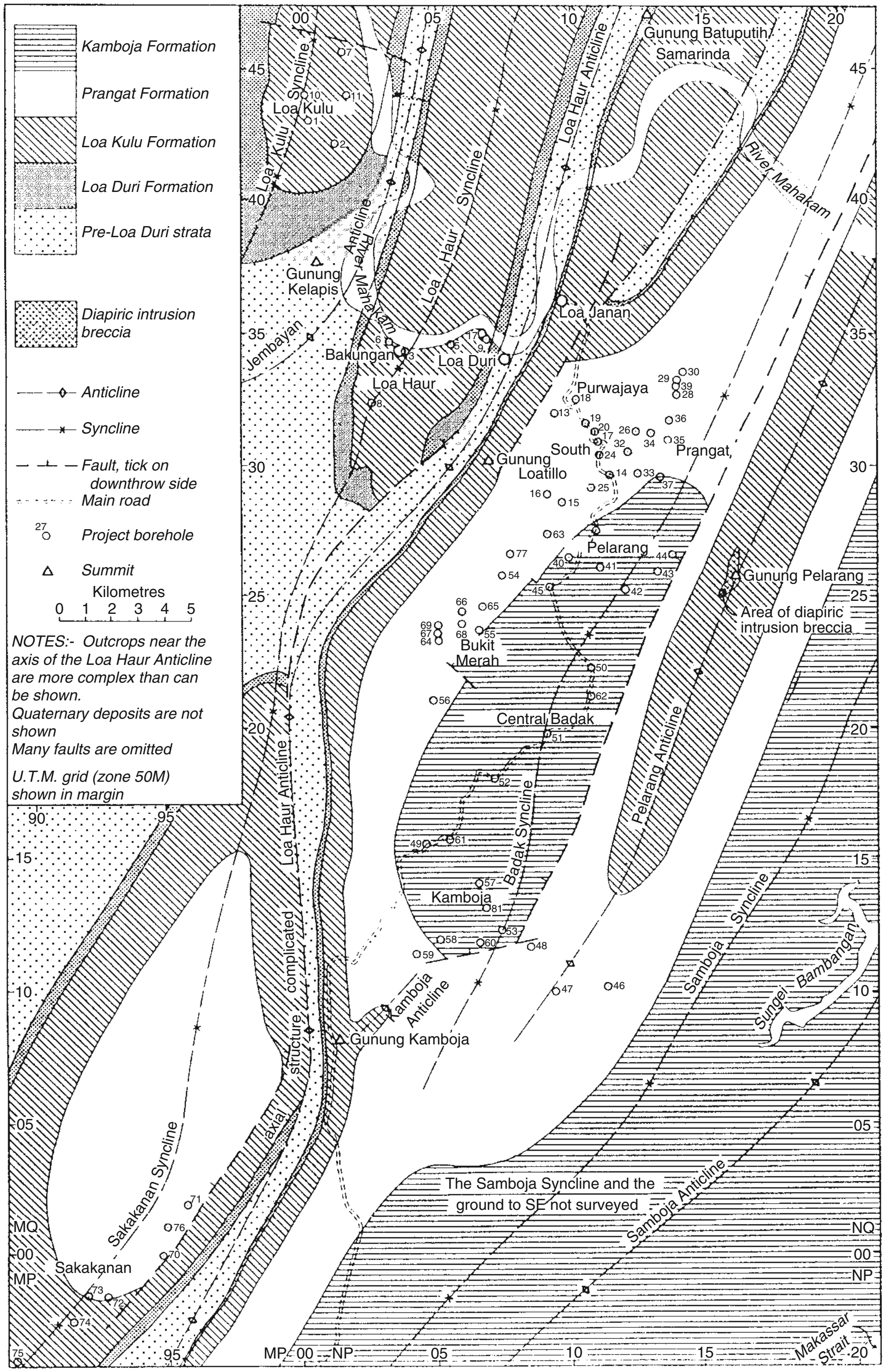
A common feature of coal-bearing sequences is the formation of ironstone, either as bands or as nodules. They usually consist of siderite ($FeCO_3$) and can be extremely hard. Where ironstone nucleation and development takes place either in, or in close proximity to a coal seam, this can deform the coal, cause mining difficulties, and, because of the difficulty in separating coal and ironstone when mining, will have an effect on the quality of the run-of-mine product.

Iron sulfide (FeS) 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




(a)

**Figure 2.46** Outcrop patterns in folded coalfields. (a) Zigzag folding of coal seams and associated faulting, Wurm Coal Basin, Germany. (From Stutzer and Noe (1940) by permission of the University of Chicago Press.)




(b)

**Figure 2.46** (b) Asymmetrical folding, broad synclines and sharp anticlines associated with thrusting, East Kalimantan, Indonesia. (From Land and Jones (1987), reproduced by permission of the author.)

**Figure 2.47** Jurassic anthracite (dark colour) intruded by granitic dykes and sills (light colour). Chungnam Coalfield, Republic of Korea. (Photograph by LPT.)

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 also when crushed is an industrial respiratory health hazard.

# 3 Age and Occurrence of Coal

## 3.1 Introduction

Although land plants first developed in the Lower Palaeozoic Era, and coal deposits of Devonian age are the earliest known, it was not until the Upper Palaeozoic Era, particularly the Carboniferous and Permian Periods, that sufficient plant cover was established and preserved to produce significant coal accumulations. Throughout the geological column, i.e. from the Carboniferous Period to the Quaternary Period, coal deposits have been formed. Figure 3.1 shows the generalized distribution of world coal deposits in terms of geological age and area (modified from Walker, 2000). Within this geological age range there have been three major episodes of coal accumulation.

The first took place during the Late Carboniferous to Early Permian periods. Coals formed at this time now form the bulk of the black coal reserves of the world, and 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 United States, 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, Southeast Asia, Australasia and Antarctica.

The second episode occurred during the Jurassic–Cretaceous period, and coals of this age are present in Canada, the United States, China and the CIS.

The third major episode occurred during the Paleogene–Neogene Periods. Coals formed during this period range from lignite to anthracite. Paleogene–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 characterized by thick seams and have often undergone minimal structural change. Paleogene–Neogene coals are also found worldwide, and are the focus of current exploration and production as the traditional Carboniferous coalfields become depleted or geologically too difficult to mine.

## 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 respond to 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 early Triassic time, followed by further rifting which 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 United States and 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, characterized 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 Paleogene–Neogene Periods (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

*Coal Geology*, Second Edition. Larry Thomas.

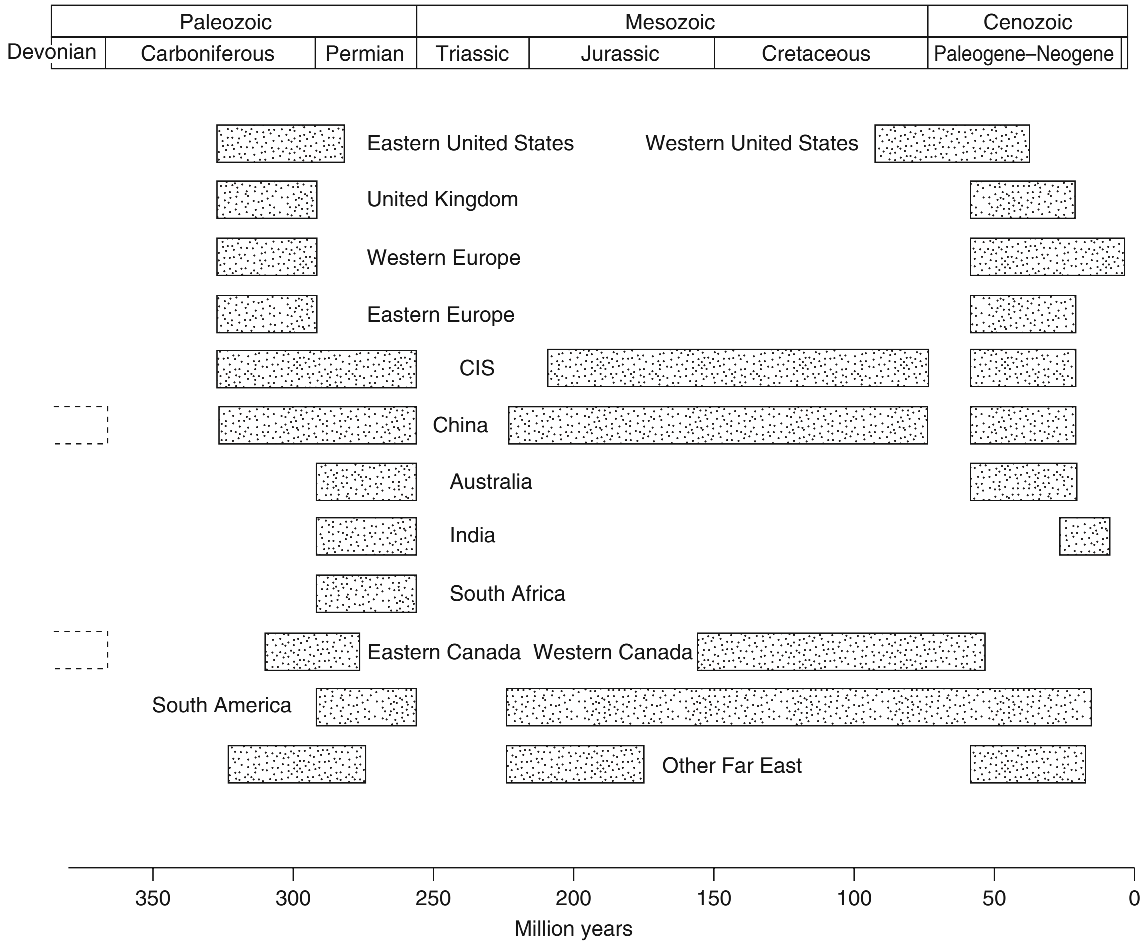



**Figure 3.1** Geological age distribution of the world's black coal and lignite deposits: CIS, Commonwealth of Independent States. (Modified from Walker, 2000.)

proportion of maceral types present in the coals (see section 2.2.3.2). 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 Paleogene–Neogene coals are, for the most part, lignites and are found worldwide, although in some areas they have undergone severe temperature and pressure changes which has produced higher rank coal, ranging from subbituminous to high volatile bituminous in areas such as Indonesia, Colombia and Venezuela.

## 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 characterized 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.

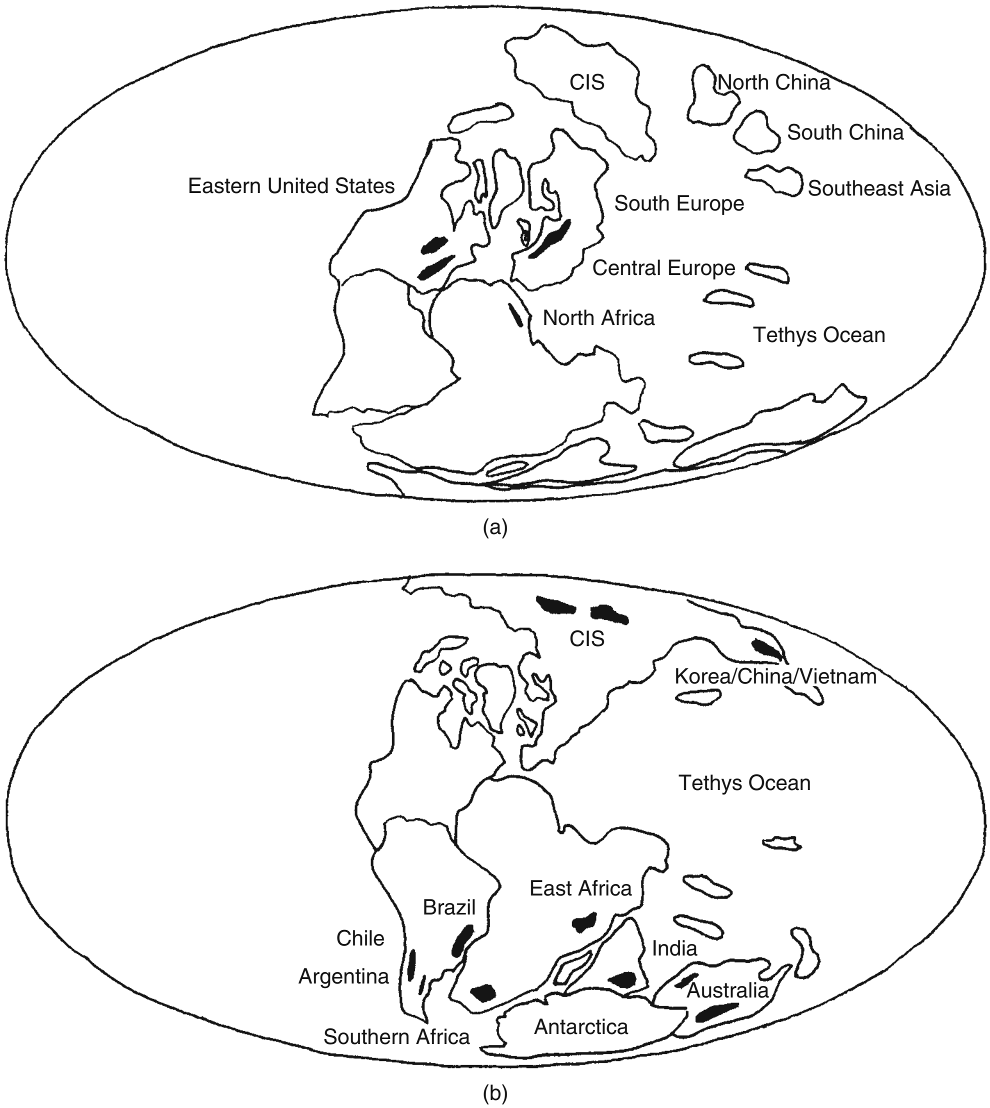



**Figure 3.2** (a) Palaeogeographical reconstruction of Late Carboniferous times showing principal areas of coal deposition. (b) Palaeogeographical reconstruction of Permian–Triassic times showing principal areas of coal deposition: CIS, Commonwealth of Independent States.

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 United States and also of China, 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 United Kingdom the long history in studying the Carboniferous (Westphalian) has enabled the chronostratigraphy

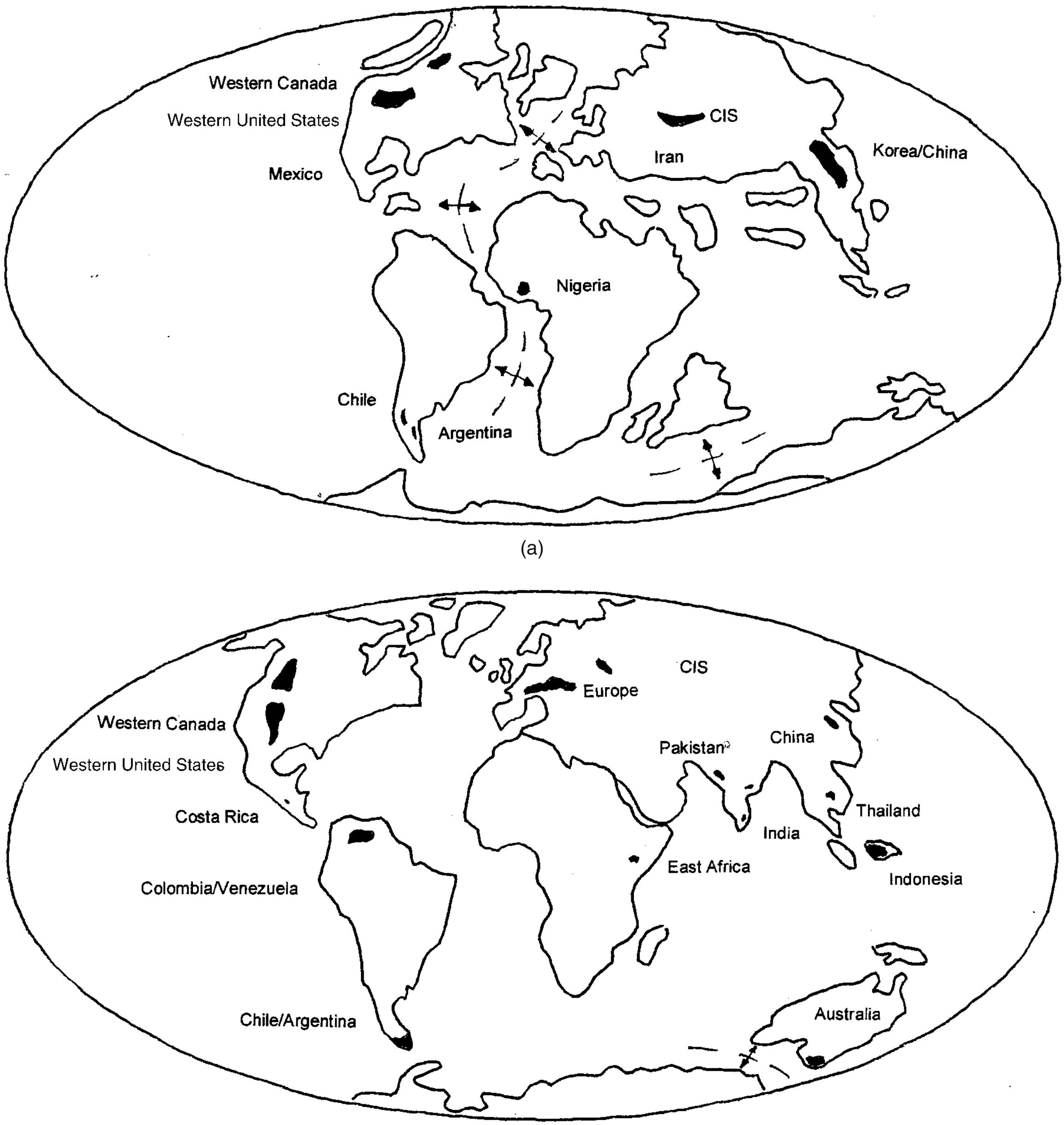



**Figure 3.3** (a) Paleogeographical reconstruction of Jurassic–Cretaceous times showing principal areas of coal deposition. (b) Palaeogeographical reconstruction of Paleogene–Neogene showing principal areas of coal deposition: CIS, Commonwealth of Independent States.

and biostratigraphy to be classified as shown in Table 3.1.

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

**Table 3.1** Detailed chronostratigraphy and biostratigraphy of the Carboniferous (Westphalian A–C) coal-bearing sequence of the United Kingdom.

| Chronostratigraphy | | | | Biostratigraphy | | |
|---|---|---|---|---|---|---|
| Series | Stage | Chronozone | Marine band | Nonmarine bivalves | Miospores | Megaflora |
| Westphalian | Bolsovian | Phillipsi | | *Anthraconauta phillipsi* Zone | *Torispora securis* (X) | *Paripteris linguaefolia* |
| | | | Cambriense | | | |
| | | Upper Similis-Pulchra | Shafton | | | |
| | | | Edmondia | | *Vestispora magna* (IX) | |
| | | | Carway Fawr | *Anthraconaia adamsi-hindi* | | |
| | | | Aegiranum | | | |
| | Duckmantian | Lower Similis-Pulchra | Sutton | *Anthracosia atra* | | |
| | | | Haughton | | | |
| | | | Clown | | | *Lonchopteris rugosa* |
| | | | Maltby | | | |
| | | Modiolaris | | *Anthracosia caledonica* | *Dictyotriletes bireticulatus* (VIII) | |
| | | | Lowton Estheria | | | |
| | | | | *Anthracosia phrygiana* | | |
| | | | | *Anthracosia ovum* | | |
| | | | Vanderbeckei | | | |
| | Langsettian | | | *Anthracosia regularis* | *Schulzospora rara* (VII) | *Lyginopteris hoeninghausii* |
| | | | | *Carbonicola crista-galli* | | |
| | | Communis | Low Estheria | | *Radiizonates aligerens* (VI) | |
| | | | | *Carbonicola pseudorobusta* | | |
| | | | | *Carbonicola bipennis* | | |
| | | | Kilburn | | | |
| | | | | *Carbonicola torus* | | |
| | | | Burton Joyce | | | |
| | | | Langley | | | |
| | | Lenisulcata | | *Carbonicola proxima* | *Triquitrites sinani Cirratriradites saturni* (SS) | |
| | | | Amaliae | | | |
| | | | | *Carbonicola extenuata* | | |
| | | | Meadow Farm | | | |
| | | | Parkhouse | | | |
| | | | Listeri | | | |
| | | | Honley | | | |
| | | | Springwood | | | |
| | | | | *Carbonicola fallax-protea* | | |
| | | | Holbrook | | | |

Source: Lake, 1999; 'IPR/23-10C British Geological Survey. © NERC. All rights reserved'.

coal deposits. This may be supported by geophysics and detailed sedimentological studies. In addition, this can be augmented by petrographical 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 three-dimensional picture of the coal deposit.

In the United Kingdom, detailed studies over the past 150 yr 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 stratigraphical 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 United States, a large amount of geological investigation has taken 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 and Glass, 1958; Potter and Simon, 1961; Potter, 1962, 1963). 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 coal mining economics by geological processes.

## 3.4 Age and geographical distribution of coal

A brief summary is given of the geographic distribution of the known coal deposits of the world. It is designed as a guide to the location of the principal coalfields throughout the world. The detailed stratigraphical ages of the deposits are not given, usually only the geological period in which they were formed. The distribution of coal deposits throughout the world are shown in Figures 3.7–3.15, and are dealt with in nine geographical regions. Details are based on Walker (2000) in his review of the major coal producing countries of the world, and from Saus and Schiffer (1999) in their review of lignite in Europe.

### 3.4.1 United States

The coal deposits of the United States 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 United States. 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 coal 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 southwest, with a width of 400 km in the north tapering to 25 km in the south. The Pennsylvanian sequence is around 1000 m thick and the basin is subdivided into: a northern region, comprising southwest Pennsylvania, eastern Ohio, northern Virginia, and northeastern 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 coal seams and the southern area has up to 26 economic coal seams. Seams are usually between 0.5 m and 3.6 m in thickness, with varying degrees of structural intensity. Deep mines characterize the older workings, but 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 largest reserves of coking coal in the United States 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, southwest Indiana and western Kentucky. Pennsylvanian sediments reach 1200 m in thickness with around 25 coals of economic interest. In Illinois, coal rank increases from high volatile bituminous C coal in the central and northern areas, to high

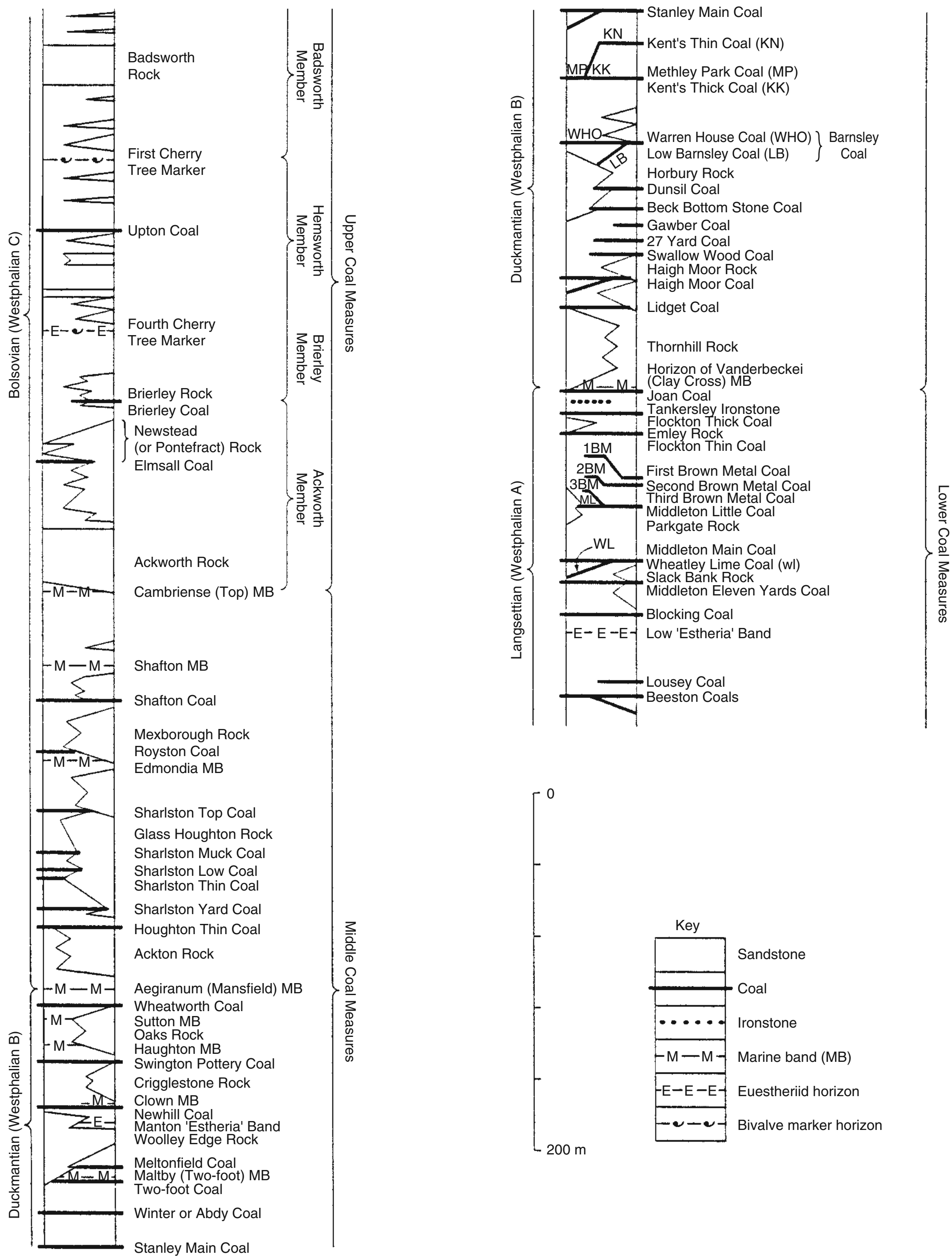



**Figure 3.4** Detailed stratigraphy of the Westphalian succession for the Wakefield area of Yorkshire, United Kingdom. (Lake, 1999; 'IPR/23-10C. British Geological Survey, © NERC. All rights reserved'.)

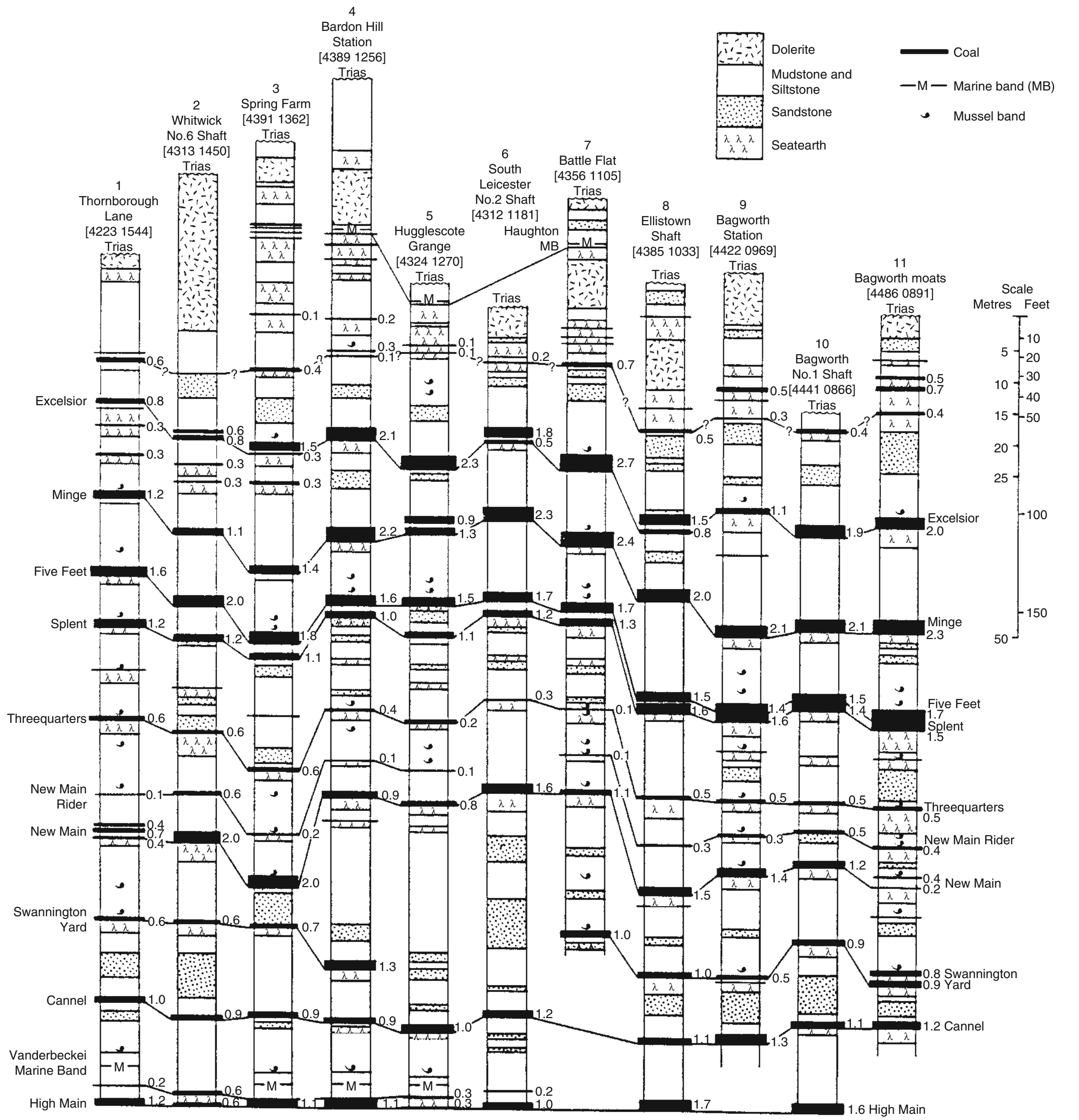



**Figure 3.5** Stratigraphical correlation based on faunal horizons and coals. (Worssam and Old, 1988; 'IPR/23-10C. British Geological Survey. © NERC. All rights reserved'.)

volatile bituminous B and A in western Kentucky. Coals generally have a high sulfur content of 3–7%, although low sulfur coals are present. Coal seams range from 0.5 m to 2.5 m in thickness, and two seams, the Springfield and Herrin Coals, are >1.5 m thick and cover thousands of square kilometres. These seams have accounted for more than 90% of production from this region. The Western Interior Basin includes deposits in Iowa, Missouri, Kansas, Oklahoma, Arkansas and Texas, and is characterized 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.

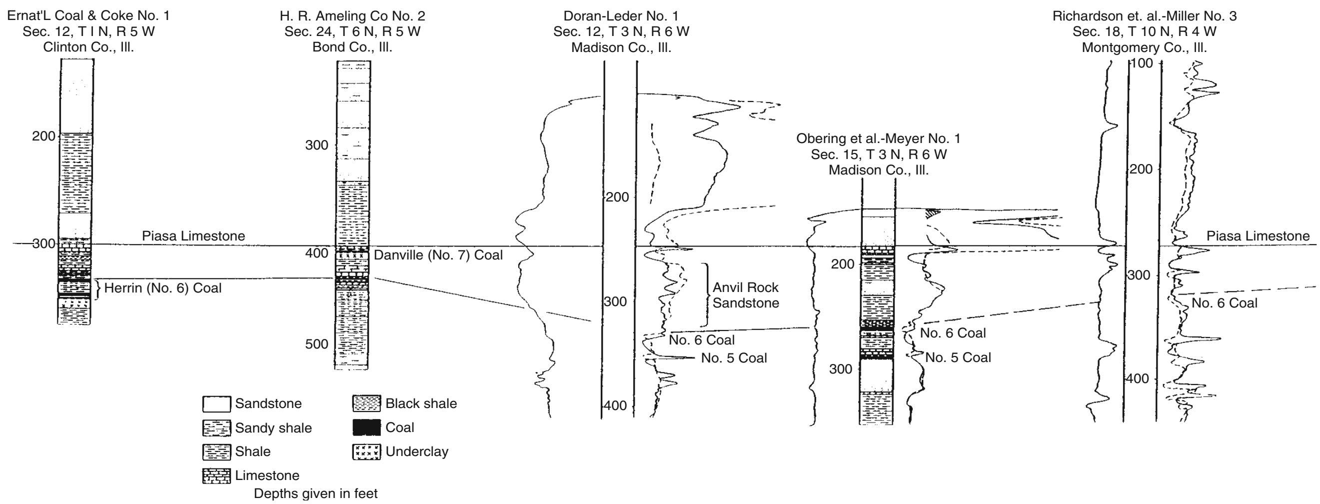



**Figure 3.6** Use of lithological and electric logs to show the stratigraphical relationships between the development of the Anvil Rock Sandstone and its effect on the Herrin No.6 Coal. (Potter and Simon, 1961.)

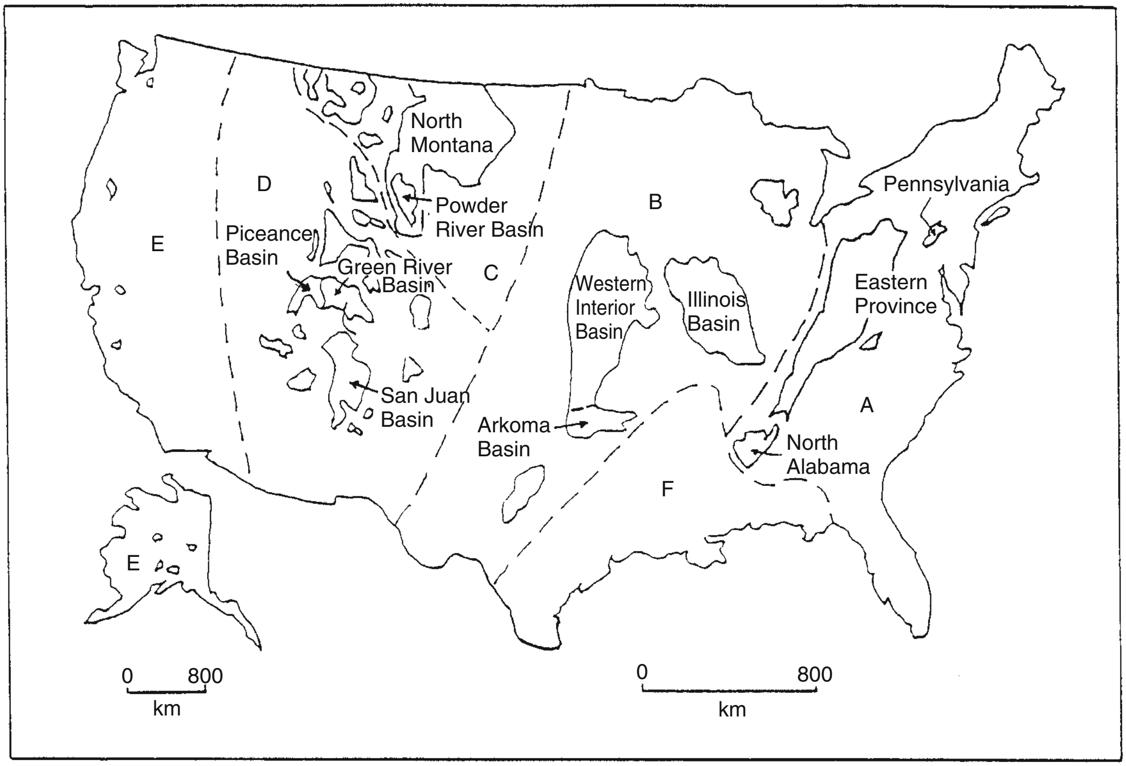



**Figure 3.7** Coal deposits of the United States. (A) Eastern Province; (B) Interior Province; (C) Northern Great Plains Province; (D) Rocky Mountains Province; (E) Pacific Coast Province; (F) Gulf Coastal Plain Province.

The Northern Great Plains Province contains coals of Cretaceous–Neogene age. The chief coal producing area is the Powder River Basin, which lies in northern Wyoming and southeastern Montana. Coals of Cretaceous age are present, but the major coal development occurred in the Paleogene. 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, which are bituminous in rank.

The Rocky Mountain Province contains coal preserved in a series of intermontane basins. In northwest Colorado and southwest 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 are Upper Cretaceous to Paleogene 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 Paleogene age. In southwestern Colorado and northeastern New Mexico, Late Cretaceous coals are present in the San Juan Basin. Most Cretaceous coal is bituminous and subbituminous, and the Paleogene coals are subbituminous and lignite. The seams are thick, in the range of 3.0 m to 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 Paleogene–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 tectonized 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 Paleogene–Neogene age. The lignite is produced from large opencast pits, where seams range from 1.0 m to 7.5 m in thickness. Production has greatly increased in the past 20 yr and is primarily for electricity generation.

### 3.4.2 Canada

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 Paleogene–Neogene and Mesozoic (Late Jurassic to Early Cretaceous) lignites and subbituminous coals, whereas those occurring in the mountains are Mesozoic high volatile to low volatile bituminous coals. The Late Jurassic to Early Cretaceous sediments are up to 2700 m in thickness in the mountainous region of southwest Alberta and southeast British Columbia and 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 northeastern British Columbia, ranging from 2 to 13 m in thickness. In the central Alberta Plains region, coals of Upper Cretaceous–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 are 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 supplies coal for generating electricity from a single 0.5 m seam. The Sydney Coalfield is now restricted to mining offshore and again 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

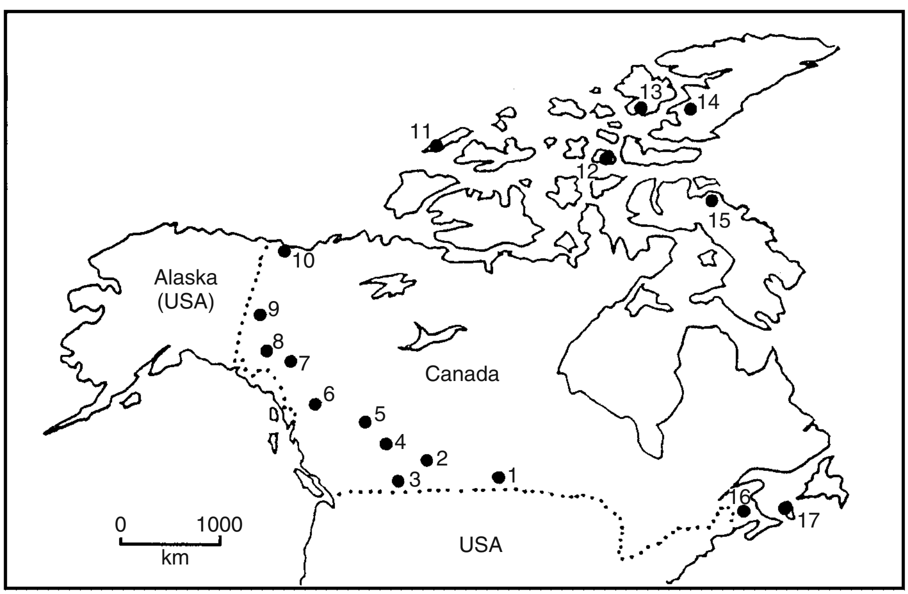



**Figure 3.8** Coal deposits of Canada. (1) South Saskatchewan; (2) Central and Eastern Alberta; (3) Southeast British Columbia; (4) West Central Alberta; (5) Northeast Coal Block; (6) Northwest British Columbia; (7) Watson Lake; (8) Whitehorse; (9) Dawson; (10) Mackenzie Bay; (11) Prince Patrick Island; (12) Cornwallis Island; (13) Axel Heiberg Island; (14) Ellesmere Island; (15) Baffin Island; (16) Minto Coalfield; (17) Sydney Coalfield.

coals, often highly tectonized, and Paleogene–Neogene subbituminous coals and lignites. The Mesozoic coals are principally from the Yukon Territory, and Paleogene–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 (Paleogene–Neogene) age are developed in a series of basins that stretch from the United Kingdom in the west to Turkey in the east. The full range of black and brown coals are present, and all of the most accessible

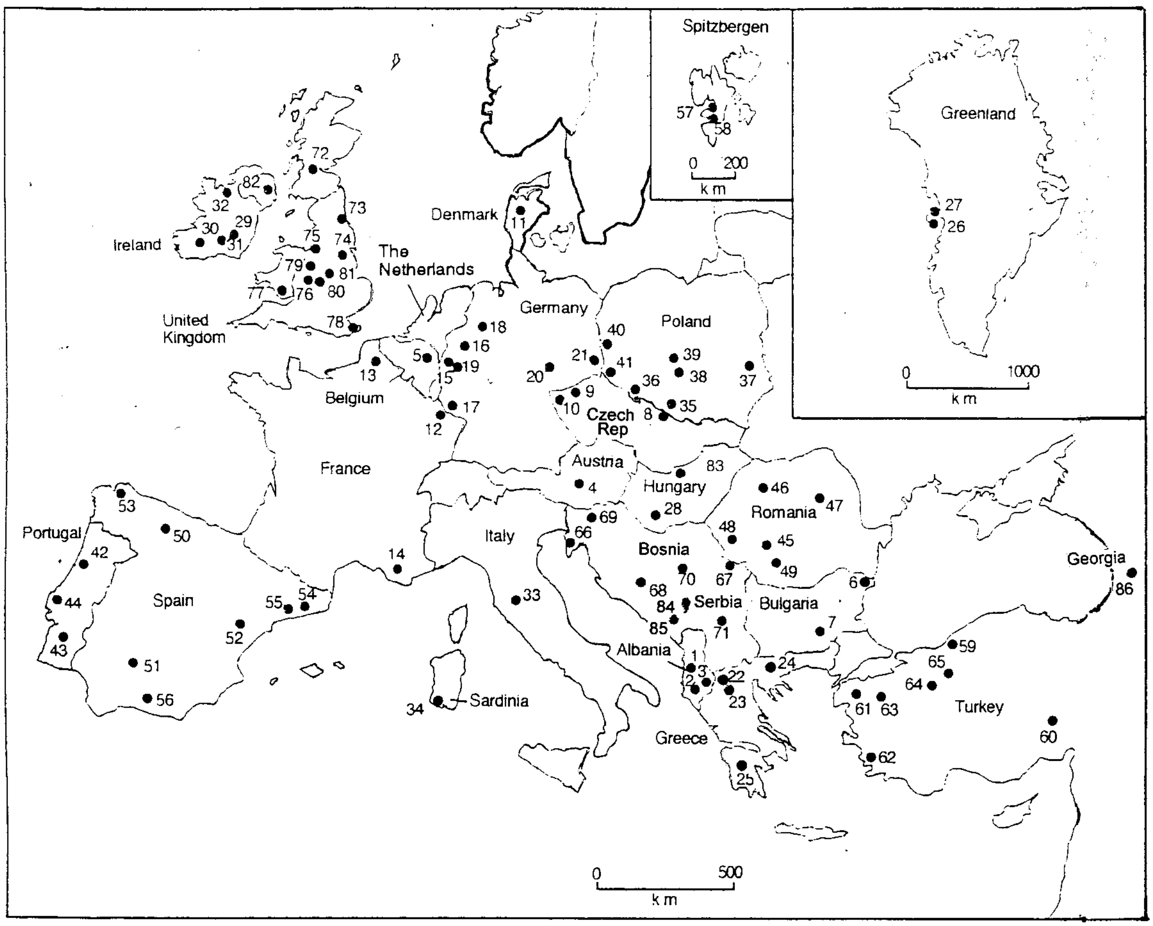



**Figure 3.9** Coal deposits of Europe. (1) Tirane; (2) Tepelene; (3) Korce; (4) West Styria; (5) Kempen; (6) Dobrudza; (7) Maritsa; (8) Ostrava-Karvina; (9) North Bohemia; (10) Sokolov; (11) Herning; (12) Lorraine; (13) Nord et Pas de Calais; (14) Provence; (15) Aachen; (16) Ruhr; (17) Saar; (18) Lower 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) Bełchatów; (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) Northeast 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.

deposits have been worked extensively over the past 150 yr. 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, Tepelene and Korce.

#### 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. Paleogene–Neogene lignite and subbituminous coal is present in the West Styria area. Mining is on a small scale only.

#### 3.4.3.3 Belgium

The Carboniferous coalfield of Tempen in the north of the country produces all of Belgium's coal. 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 are 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

Paleogene–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 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 of around 1%. Current lignite production is used for power generation.

#### 3.4.3.5 Bulgaria

The most extensive coal deposits in Bulgaria are lignites of Paleogene–Neogene age. Most of the mining areas are situated in the southwest and central parts of the country. The principal producers are the opencast mines at Chukurova, Pernik, Beli Breg and Maritsa East, with underground 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 35% of Bulgarian power generation. High rank bituminous coals are known from the Dobruja area on the east coast, 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 northeastern border. It is of Upper Carboniferous age and represents 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 coal seams 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.

Paleogene–Neogene lignites are located in the northwest of the Czech Republic, in North Bohemia (Most), western Bohemia (Sokolov) and in southern Moravia (Hodonin), and has been mined in all these areas. Seams are up to 30 m in thickness, with low sulfur contents and little structural disturbance. Other lignite deposits are present in the south of the country but are not currently being exploited.

#### 3.4.3.7 Denmark

Paleogene–Neogene lignites are found at Herning, but 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 tectonized, 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 southwestern France, all are structurally complex and mining has ceased in these areas. The coal mining 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 southeast 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 m 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 m to 2.0 m in thickness. This coal is not suitable for coking purposes.

The brown coals of Germany are of Paleogene–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, which has high volatile, 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.

### 3.4.3.10 Georgia

The Tkibuli Coalfield in the northwest of Georgia is reputed to have large reserves of Jurassic black coal, this is currently mined underground on a modest scale.

### 3.4.3.11 Greece

Lignite deposits occur widely across Greece. The principal deposits are Neogene to Quaternary in age. The principal deposits are situated in the north of Greece at Florina, Amyndaeon, Ptolemais and Elassona. To the south occurs the Megalopolis deposit and to the northeast, 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 60 m in thickness, and in Amyndaeon and Megalopolis, lignite seams are up to 30 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. The most significant are the subbituminous 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 coal mining is presently carried out in Holland.

### 3.4.3.14 Hungary

Black coal is mined in the southwest 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 ($>5$ m). The coal is weakly caking bituminous with high ash and sulfur contents. The area is a well-established coalfield, with mining in difficult conditions.

Brown coal deposits of Mesozoic and Paleogene–Neogene age are found in a northeast–southwest belt of country in the north of Hungary. Lignite is mined at Oroszlany, Tatabanya, Dorog and Visonta-Bukkabrany. The coals are mined by underground and opencast methods, with ash and sulfur contents of 20% and 1% respectively. All is utilized 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 northwest. The coal is anthracite with variable sulfur contents in the south, and medium volatile bituminous coal in the northwest. 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. Paleogene–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

The Kosovo Basin contains a lignite seam up to 100 m in thickness, and has potential to supply the electricity industry.

#### 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 coal mining industry. Black coal in Poland is centred around three coalfields, Upper Silesia, Lower Silesia and Lublin. These areas represent large basins 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 while the upper part has 60 coal seams, the thicker seams reaching 6–7 m. The basin is highly tectonized 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 tectonized. 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 Bogdanka underground mine is currently the only working mine in the Lublin Coalfield.

Paleogene–Neogene lignite basins are present in central and southwestern Poland, many of which are fault bounded. Production is centred on four areas, namely, Adamov, Bełchatów, 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 second largest European producer of lignite.

#### 3.4.3.20 Portugal

Anthracite of Carboniferous age is present in the northwest of the country, but is not worked extensively. 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 Paleogene–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 southwest Romania, and in the Jiu Valley. Here, bituminous coal of coking quality is mined for industrial use.

#### 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 Paleogene–Neogene brown coals that are not affected structurally. The latter are more extensive than the older deposits. The black coal deposits occur in the southeast 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.

Paleogene–Neogene lignites are 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 basin contains 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 south of Spain, southwest 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 northwest 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 northeast, 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 Paleogene–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 m in thickness.

### *3.4.3.25 Turkey*

Turkey has considerable reserves of Carboniferous black coal and Paleogene–Neogene brown coal. The principal black coal deposit is the Zonguldak Coalfield on the northern coast, where numerous seams ranging from 0.7 m to 10 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 subsidized.

Paleogene–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 northwest and west of the country respectively. The Oligocene is characterized 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 southwest 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.

### *3.4.3.26 United Kingdom*

In the United Kingdom, a number of coal-bearing basins are 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, Northeast England, the Yorkshire–Nottinghamshire region, the Lancashire–North Wales region, the East and West Midlands, South Wales and Kent. Much of the underground mining in the United Kingdom has now disappeared and been replaced by opencast operations.

The Yorkshire–Nottinghamshire region is the most important coal producing area in the United Kingdom, supplying coal for electricity generation and coking coal to industry. The region is less structurally affected than other areas, and has a long coal mining 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. Due to extensive working and difficult mining conditions mining in the region has now ceased. In northeast England, bituminous coals suitable for coking purposes are produced in opencast operations. The East and West Midlands contain four coal mining 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, for use in power generation. In Scotland, high volatile bituminous coals with low sulfur are opencast mined. This area has been mined extensively in the past. South Wales, once the principal coalfield in the United Kingdom, still produces bituminous coal and anthracite with low sulfur contents, from seams usually between 1.0 m and 3.5 m. Underground mining conditions are difficult and expensive and currently only two small underground mines are in operation. The bulk of production comes from opencast mining in the coal outcrop areas. Kent is a small coalfield in the southeast of the United Kingdom that contains high quality bituminous and anthracite coal, but is no longer a producer.

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

### 3.4.4 Africa

The principal coal occurrences are shown in Figure 3.10. The occurrences of black coal in Africa are: (i) those deposits of Carboniferous age found on the northern coast, in Morocco in the west and Egypt in the east; and (ii), more importantly, the widespread Karroo deposits of Late Carboniferous–Permian age, which are found throughout central and southern Africa (Haughton, 1969). The Karoo sequences were deposited on the Gondwana supercontinent which split apart in the Mesozoic Period, hence the similarities of African Gondwana coals with those of India and South America. Brown coals of Paleogene–Neogene age are present, but in Africa it is the black coals that are of prime interest.

#### 3.4.4.1 Angola

Lignites of Paleogene–Neogene age have been identified in Angola: (i) in the east around the headwaters of the Lungue-Bungo river, where seams of lignite up to 2.5 m have been recorded, and (ii) in the west around Luanda, where lignites are present in the Paleogene–Neogene coastal sediments. None of these deposits have been worked.

#### 3.4.4.2 Botswana

Botswana has large reserves of black coal of Karroo 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 m, 4.5 m and 2.0 m in thickness are present. At Mmamabula, seam thicknesses average 2.8 m, 5.4 m 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 Cameroun

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

#### 3.4.4.4 Egypt

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

#### 3.4.4.5 Ethiopia

Paleogene–Neogene brown coals are known from many localities on the Ethiopian Plateau, beds are up to 15 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 Karroo sediments on the western side of the island, where they overlie the Precambrian basement. At the southern end of the Karroo outcrop, five coal-bearing areas have been identified. The northernmost area is the Imaloto Coalfield, which contains seams averaging 1 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 m and 7 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 Paleogene–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 number of separate basins containing coal-bearing Karroo sediments 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.

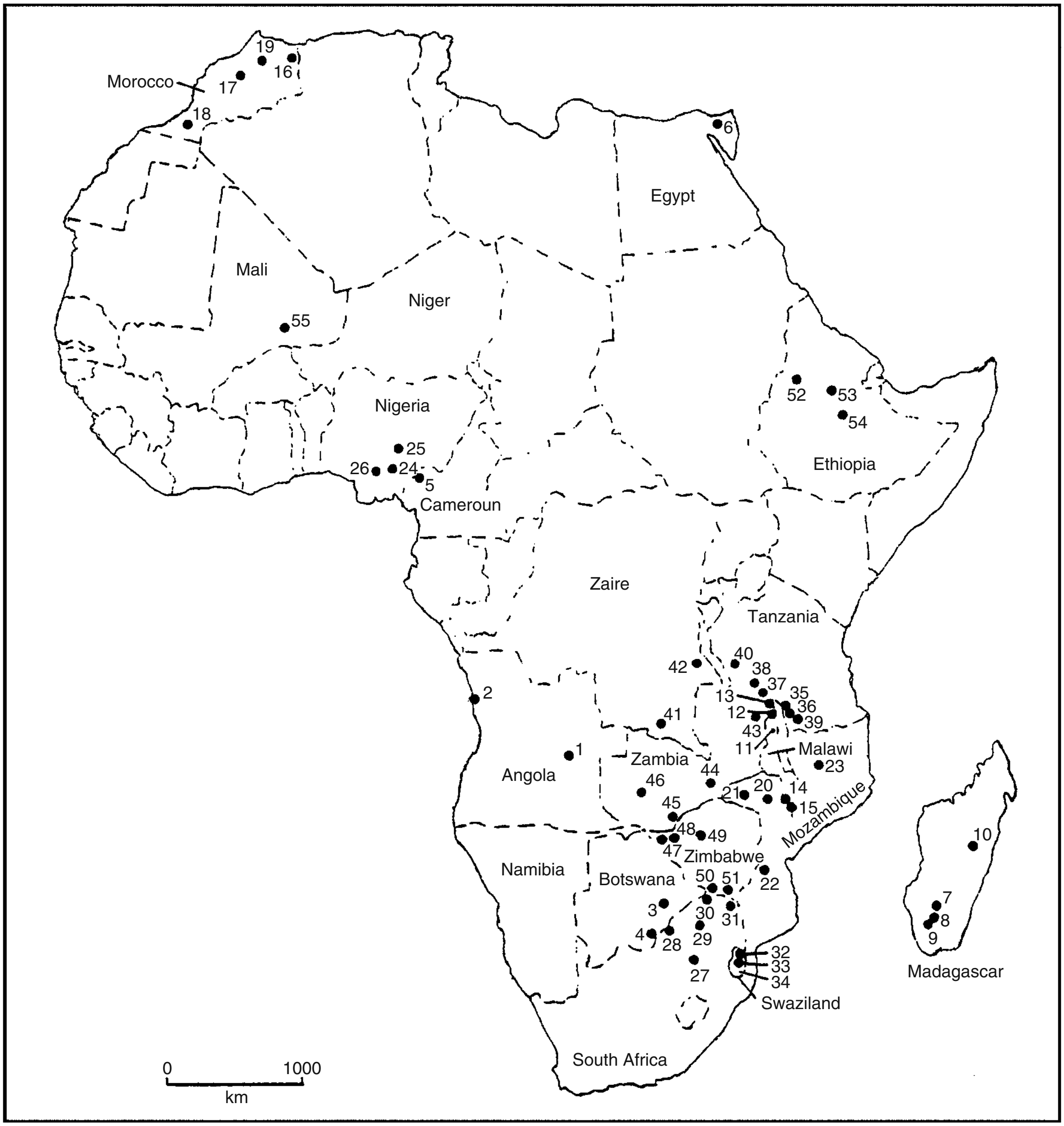



**Figure 3.10** Coal deposits of Africa. (1) Lungue-Bungo; (2) Luanda; (3) Moropule; (4) Mmamabula; (5) Bamenda; (6) Al Maghara; (7) Imaloto-Vohibory; (8) Sakoa; (9) Sakamena; (10) Antanifotsy; (11) Livingstonia; (12) North Rukuru; (13) Ngana; (14) Lengwe; (15) Mwabvi; (16) Jerada; (17) Ezzhiliga; (18) Tindouf-Draa; (19) Meknes-Fez; (20) Moatize; (21) Mmambansavu; (22) Chiomo; (23) Itule; (24) Enugu-Ezimo; (25) Orukpa-Okaba- Ogboyoga; (26) Asaba; (27) Karroo 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) Bubye; (52) Chelga; (53) Wuchalle; (54) Dobre-Brehan; (55) Bourem.

The Livingstonia Coalfield contains seams 1 m to 2 m in thickness, which are mined to supply fuel to local industry. At Ngana, one seam is up to 15 m in thickness, but seams usually average around 1 m, and 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 Paleogene–Neogene brown coals are recorded from the Mali–Niger Basin in the southeastern past of the country, around Bourem. Seams are thought to reach 2 m in thickness, and have moisture values of 24% and ash values of 21%.

#### 3.4.4.9 Morocco

Carboniferous black coals are found in the northeast 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

Karroo sediments are preserved in a series of basins in the Precambrian basement. The coal-bearing Karroo sediments crop out in 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 m to 4 m in thickness, structurally the coalfield is heavily faulted. Coal is low volatile bituminous, with high ash and low sulfur contents. The coal deposits at Mmambansavu, Chiomo and Itule are as yet little known.

#### 3.4.4.11 Namibia

The eastern half of Namibia is covered by post-Karroo sediments of the Kalahari Group. It is possible that Karroo 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 Upper Cretaceous–Neogene age. Little information is known but these coals may be worked locally.

#### 3.4.4.13 Nigeria

Coal-bearing sediments of Cretaceous and Paleogene–Neogene age overlie Precambrian basement in the southeastern 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 m to 3 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.

Paleogene–Neogene lignites are found in the Asaba region close to the River Niger, seams are 3 m to 7 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 Karroo System contains coal-bearing sediments of Carboniferous–Permian (Gondwana) age. The main Karroo 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 Karroo sequence was deposited directly onto basement and the coal seams are shallow and almost horizontal. They have been affected by numerous igneous intrusions, which have produced a great variation in rank, often very localized. The western area consists 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 northeast. Other coalfield basins in the northeast 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, but these are not worked at this time.

#### 3.4.4.15 Swaziland

Coal-bearing Karroo 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 Karroo sediments are preserved in depressions in the Precambrian basement, all are located in the southwest of the country. The Ruhuhu Coalfields have been known for a century but have never been fully developed. Of these, the Ketewaka-Mchuchma and Ngaka Coalfields are the most important. In these coalfields, coals occur in two zones, the lower containing the better coals, seams can be as thick as 7 m but this is exceptional. Coals are high to low volatile bituminous with high ash and low sulfur contents. Other coalfields with similar characteristics are those of Songwe-Kiwira, Galula, Njuga and Ufipa.

#### 3.4.4.17 Zaire

Small separate basins of coal-bearing Karroo sediments occur in the southeast of the country at Luena and Lukuga. Seams are up to 2 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

Karroo sediments are preserved in depressions in the Precambrian basement. A number of such basins are present in the east and southeast 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, and some coal has coking properties. The Maamba area in the southeast has seams 2 m to 3 m in thickness, and is high volatile bituminous with high ash content. The Maamba area produces most of Zambia's coal. At Kahare, coals are preserved beneath younger sediments and coal quality is similar to other Zambian coals, but this area has not yet been fully investigated.

#### 3.4.4.19 Zimbabwe

The Karroo sequence in Zimbabwe is preserved in the Zambezi Basin in the northwest, and the Limpopo Basin in the southeast. The northwest 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 m in thickness, and an upper steam coal up to 8 m, all generally with low sulfur contents. In the southern coalfields of Bubye 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 region of Afghanistan. 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 little or no coking properties. Coal is mined at Herat in the northwest 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 Paleogene–Neogene sediments in northwestern 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 northwest, the concealed coal

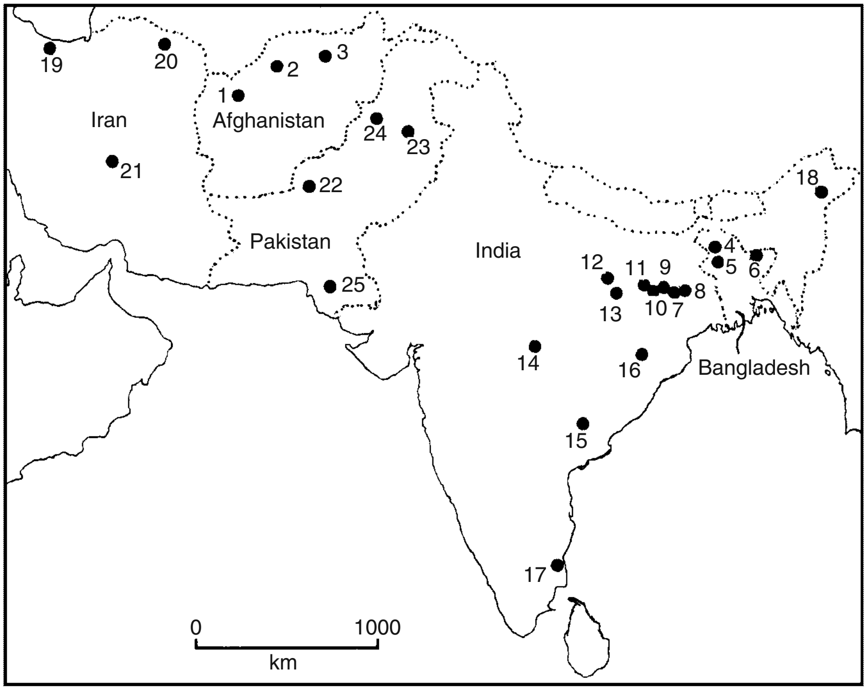



**Figure 3.11** Coal deposits of the Indian Subcontinent. (1) Heart; (2) Sari-i-Pul; (3) Dara-i-Suf; (4) Barapukuria-Khalaspir; (5) Jamalganj; (6) Sylhet; (7) Raniganj; (8) Jharia; (9) Bokaro; (10) Ramgarh; (11) Karanpura; (12) Singrauli; (13) Bisrampur; (14) Pench-Kanhan-Tawa; (15) Godavari; (16) Talchir;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

basins of Barapukuria, Khalaspir and Jamalganj contain numerous seams ranging in thickness from less than 1 m to 20 m and 30 m. The coals are medium to high volatile bituminous, with high ash and low sulfur contents and are now being exploited. In the east of Bangladesh, lower rank subbituminous and lignite coal is located at Sylhet.

#### *3.4.5.3 India*

In India coal resources are of Palaeozoic (Gondwana) and Cenozoic (Paleogene–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 northeastern and central eastern parts of peninsular India. Paleogene–Neogene brown coals are present in the northeastern and northwestern 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, Bisrampur, Pench-Kanhan and Tawa Valley in Madhya Pradesh, Kamptee, Bandar and Wardha Valley in Maharastra 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 m to 30 m, with an exceptionally thick seam of 134 m discovered in the Singrauli Coalfield. The coalfields have been faulted but otherwise are not highly tectonized. 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 Paleogene–Neogene coals are highly disturbed tectonically and are located in the mountainous regions of northeast India. In the Makum Coalfield in Assam, seams are lens-shaped, in places reaching thicknesses of 33 m. Coals are subbituminous to high volatile bituminous with high sulfur contents. In southern India, in the state of Tamil Nadu, Paleogene–Neogene lignites are found in Neyveli, the thickest seam being up to 20 m in thickness. Here the lignite is low volatile, with low ash and sulfur content. Lignite is now also being mined in the northwestern states of Gujarat and Rajasthan. All these areas will increase in importance as the demand for electricity generation increases.

#### *3.4.5.4 Iran*

The black coal deposits of Iran are Mesozoic (Jurassic) in age, with some lignites of Paleogene–Neogene age. The Jurassic coals are bituminous with high ash and sulfur contents, and have coking properties. All are strongly tectonized with seam thicknesses ranging from 1 m to 4 m. The coal supplies local needs and the metallurgical industry. Principal coalfields are located at Elburz and Khorasan in the north and at Kerman in central Iran. Lignites are found in northwest Iran but are not worked.

#### *3.4.5.5 Pakistan*

All the principal coalfields in Pakistan are of Paleogene–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 m in thickness, and the coal is subbituminous, non-coking with high sulfur content. In eastern Sindh province, the Thar Coalfield covers an area of 9000 $km^2$. 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 (Paleogene–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 Paleogene–Neogene ages. Of these the Paleogene–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 m thick and highly tectonized. 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 m thick and are medium volatile bituminous at Neuquen.

Paleogene–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 m to 2 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 m in thickness. They are subbituminous to bituminous with no coking properties. All the Paleogene–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 Paleogene–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 Paleogene–Neogene lignites are found in the Tarija Basin, where seams are thin, under 1 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 Piaui and Southern Brazil. Only the Southern Brazil region is currently considered prospective. The Amazon region contains lignites of Paleogene–Neogene age, seams are thin (less than 1.5 m)

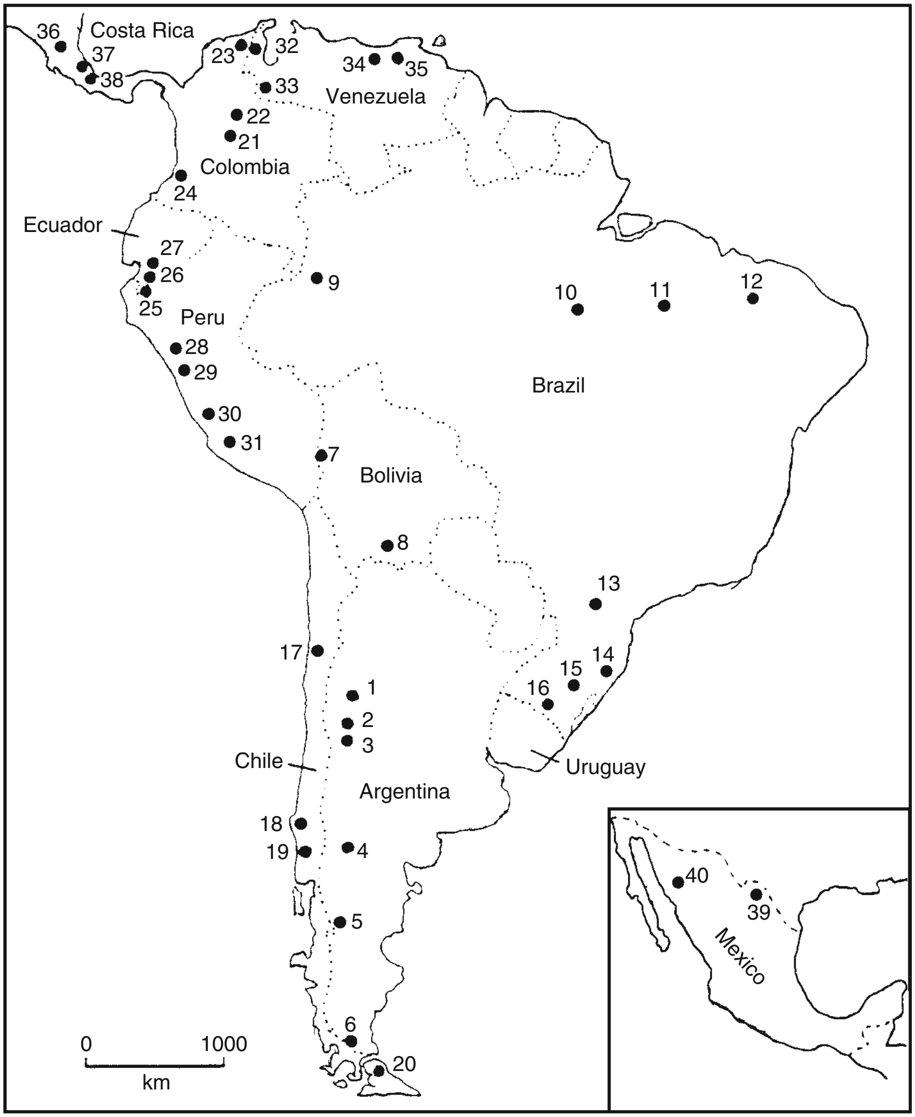



**Figure 3.12** Coal deposits of South America. (1) La Rioja; (2) San Juan; (3) Mendoza; (4) Neuquen; (5) Pico Quemado; (6) Rio Turbio; (7) Copacabana Peninsula; (8) Tarija Basin; (9) Amazon; (10) Rio Fresco; (11) Tocantins-Araguaia; (12) Western Piaui; (13) Parana; (14) Santa Catarina; (15) Rio Grande 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.

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 Piaui 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. Rio Grande do Sul Coalfield contains numerous seams up to 3 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 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 South Brazilian coal is mined as a thermal coal product.

Some lignites are present in the Sao Paolo 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 Concepcion and the coalfield extends offshore to a distance of 7 km. Dips are steep and faulting common. Seams are of Paleogene–Neogene age and average 1.0 to 1.5 m in thickness, and are high volatile bituminous coals with low ash and variable sulfur contents, and have poor coking properties. Coals of the Valdivia region are concealed beneath younger sediments. Seams reach 3 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–Neogene sediments are preserved. Coal seams up to 7 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 Paleogene–Neogene age are found in numerous localities in the northern half of Colombia. All have been highly tectonized 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. Paleogene–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 m in thickness, locally reaching 26 m. Because of the high quality of these coals, El Cerrejon is now a significant exporter of steam coal.

#### *3.4.6.6 Costa Rica*

Costa Rica contains deposits of Paleogene–Neogene coals and lignites. Individual seams are up to 1 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 southeast coast, and at Venado in the north of Costa Rica. Areally extensive peat deposits (up to 2 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 Paleogene–Neogene sequences of the Amazon Basin, the Pacific coast and intermontane basins in the Andes. Only the latter are considered to be of significance. The Malacatus Basin contains seams of up to 4 m in thickness, disrupted by faulting. To the north the Loja Basin contains seams of up to 2 m in thickness and the Canar-Azuay Basin has seams of up to 5 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 tectonized and are structurally complex. The principal coalfield is at Coahuila close to the border with Texas, United States, where shallow gently dipping seams reach 2 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 northwest of Mexico at Sonora where anthracites averaging 1 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 tectonized 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 northeast of Uruguay contains Carboniferous–Permian (Gondwana) sediments, which represent the southern extension of the South 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 Paleogene–Neogene in age and occur in a series of basins across the country north of the River Orinoco. 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 northwest of the country. Between 25 and 30 seams with thicknesses of between 0.5 m and 15 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 m to 10 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 coal 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 (CIS; former Soviet Union)

The CIS is the third largest coal producer in the world. It has vast reserves of all ranks of coal stretching across the whole of the country (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 Karaganda and Ekibastuz Basins are the principal coal producers. The Karaganda Basin contains a thick sequence of Carboniferous sediments in which 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 of similar quality to those in the Karaganda Basin.

#### 3.4.7.2 Russia

Russia has very large coal reserves of which those in eastern Siberia and the Russian Far East remain largely unexploited. Current black coal production is centred on the Kuznetsk and Pechora Basins and the Russian part of the Donetsk Basin. The Kuznetsk Basin is structurally complex, and the Carboniferous to Jurassic sequence is 7000–8000 m thick. Around 90% of the coal seams (ca. 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. The Pechora Basin also contains Permian coal-bearing sediments that

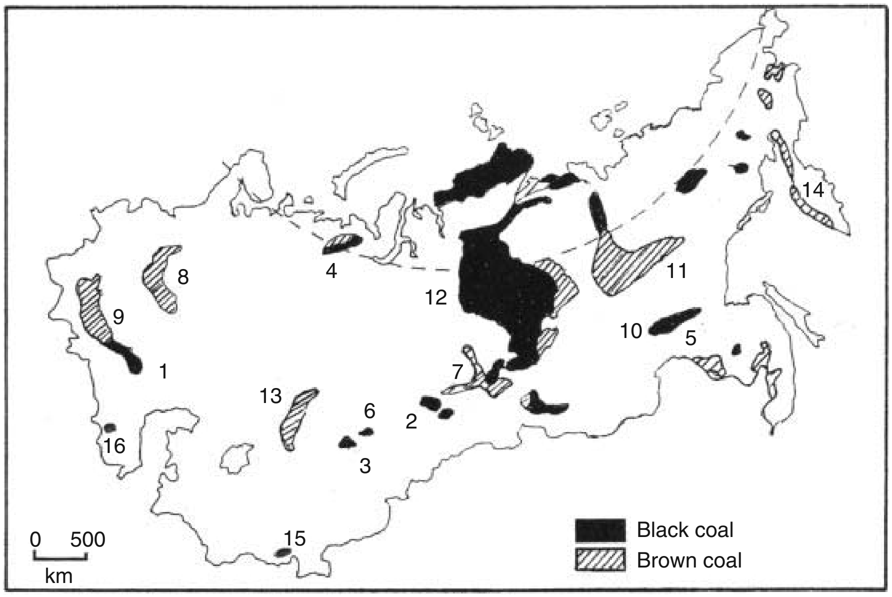



**Figure 3.13** Coal Deposits of the Commonwealth of Independent States (formerly USSR). (1) Donetsk Basin; (2) Kuznetsk Basin; (3) Karaganda Basin; (4) Pechora Basin; (5) South Yakutsk Basin; (6) Ekibastuz Basin; (7) Kansk-Achinsk Basin; (8) Moscow Basin; (9) Dnepr Basin; (10) Neryungri; (11) Lena Basin; (12) Tunguska Basin; (13) Turgay; (14) West Kamchatka Area; (15) Tajikistan; (16) Georgia.

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 so that little mining now takes place. In Eastern Siberia, the Kansk-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 is beginning to exploit the bituminous coals of East Zeddi by open pit methods, and the anthracite coal at Nazar-Aylok 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 0.5–2.5 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.

### *3.4.8 Far East*

The Far East region contains 13 countries with known coal deposits (Figure 3.14). By far the largest of these is the Peoples Republic of China, 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 Paleogene–Neogene in age, and occur in the northeast of the country close to the capital Bandar Seri Bagawan, and also in the headwaters of the Belait River in the southwest of Brunei. Coal seams are 0.5 m to 5 m in thickness and are high volatile bituminous with

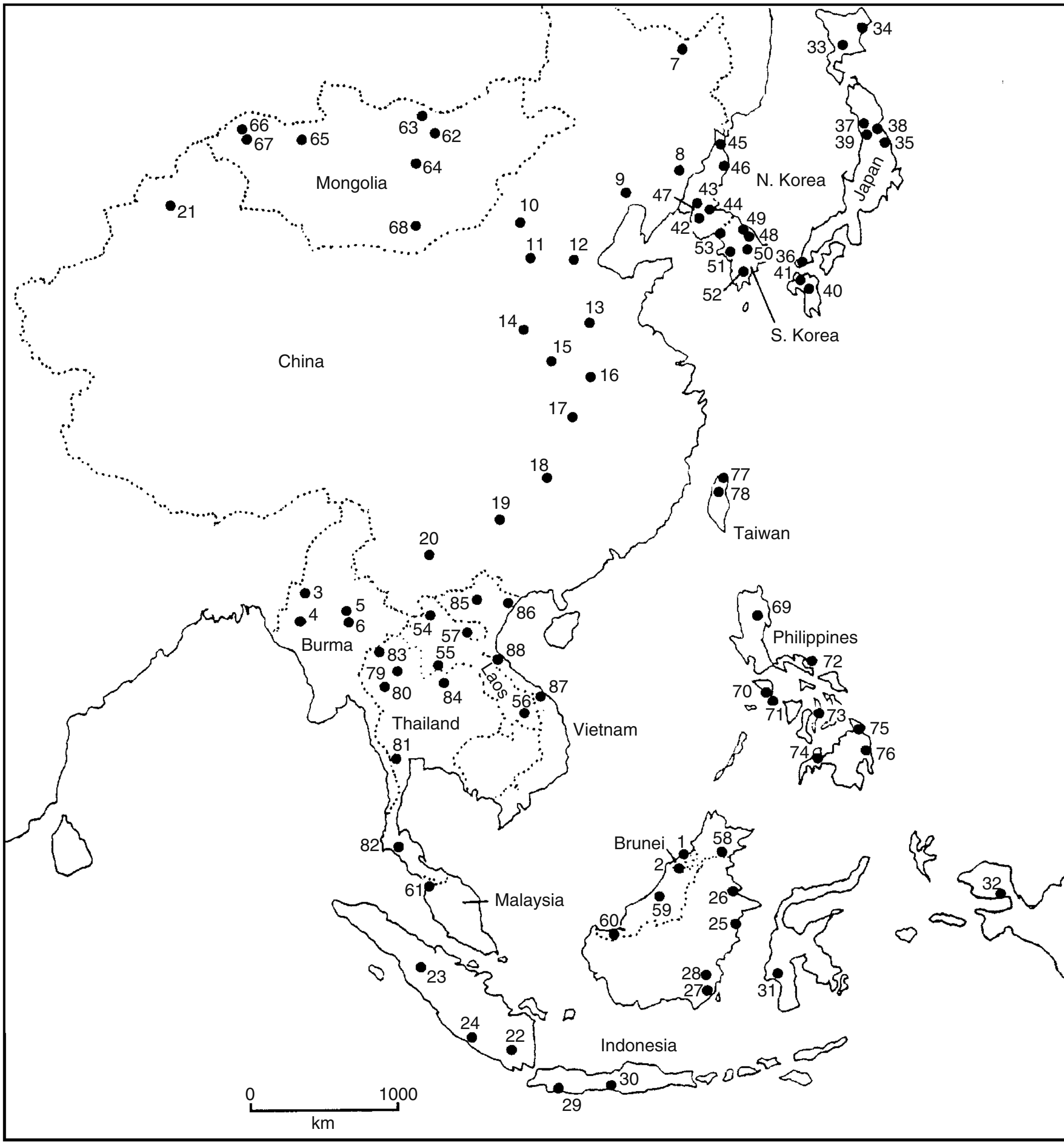



**Figure 3.14** Coal deposits of the Far East. (1) Bandar Seri Begawan; (2) Belait Basin; (3) Kalewa; (4) Pakkoku; (5) Panlaung; (6) Henzada; (7) Heilung-kiang; (8) Kirin; (9) Liaoning; (10) Shen Fu-Dong Shen; (11) Shanxi; (12) Hopeh; (13) Shantung; (14) Shaanxi; (15) Honan; (16) Anhwei; (17) Hupeh; (18) Hunan; (19) Kweichow; (20) Yunnan- Guizhou; (21) Xinjiang-Uygur; (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) Chikuho; (42) Pyongyang; (43) North Pyongyang; (44) Kowon-Muchon; (45) Kyongsang; (46) Kilchu-Myongchon; (47) Anju; (48) Samcheog; (49) Janseong-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) Mogoyn Gol; (66) Achit Nuur; (67) Khartabagat; (68) Tavan Tolgoi; (69) Cagayan; (70) Mindoro; (71) Semirara; (72) Catanduanes; (73) Cebu; (74) Zamboanga; (75) Gigaquit; (6) 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.

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

### 3.4.8.2 Burma

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

### 3.4.8.3 Peoples Republic of China

The Peoples Republic of China (PRC) is the world's largest coal producer. Three-quarters of all proven recoverable coal reserves occur in the northern half of PRC, and 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 yr 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 Paleogene–Neogene age. Carboniferous and Permian coals are found throughout eastern PRC, whereas Triassic coals are located in southeast PRC. Coals of Jurassic and early cretaceous age are located in Inner Mongolia and northeastern 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 is mined to supply power stations both domestically and for export. Mining is principally by underground methods with some small opencast operations.

Shaanxi Province contains five major coalfields, of these, the southern Huang Ling and Tongchuan coal mining 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%. 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 coal mining area in south Shanxi Province produces both steam and coking coal for home and export markets.

In Henan Province, the Hebi Coalfield situated in the north of the province is the largest producer, and together with the Gaocheng coal mines in the south 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 PRC. Seams range in thickness from 1 m to 10 m.

In northwest China, the Xinjiang-Uygur region contains large reserves of coal, up to 13 coal seams with a total thickness of 175 m is reported. These have yet to be exploited. In the northeast of 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 northeast 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 tectonized 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 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.

### 3.4.8.4 Indonesia

Indonesian coal deposits are Paleogene–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, three coalfield areas are currently exploited. At Bukit Asam at the southeastern end of the island, seams up to 12 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 is by both opencast and underground methods, and the coal is used for electricity generation and cement manufacture. In the Bengkulu region on the southwest coast of Sumatra, small occurrences of mostly subbituminous coals with low sulfur content are worked on a small scale.

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 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 northeastern 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 have not yet been developed. Large deposits of recent peat are present in West Kalimantan but these have not been developed commercially.

#### 3.4.8.5 Japan

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

#### 3.4.8.6 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 northwest, and Kowon-Muchon in the east. All have been highly tectonized, consequently seam thicknesses are variable due to intense folding, however thicknesses of 5 m and 15 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 northeast of the country. The Kyongsang and Kilchu-Myongchon Coalfields contain lignites, and the Tumangang in the extreme northeast contains subbituminous coal, which is used for electricity generation. The Anju Coalfield is located north of Pyongyang and is a large deposit of subbituminous coal, which is being developed as an opencast operation.

#### 3.4.8.7 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 are underground, but due to difficult mining conditions and unsuitable coal quality, little or no mining is currently in operation. The principal Palaeozoic coalfields are Samcheog, Jeongseon, Kangnung, Danyang and Mungyeong. These coalfields are highly tectonized and intensely folded, seam thicknesses vary considerably due to the squeezing of the coals, 1 m to 2 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.

Paleogene–Neogene deposits containing thin seams of lignite are found in small areas bordering the southeast coast of South Korea.

### *3.4.8.8 Laos*

Palaeozoic, Mesozoic and Cenozoic coals are present in Laos. The Palaeozoic deposits are chiefly anthracite with a high ash content. Three principal occurrences are 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 m to 6 m are present, but 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 m to 10 m. The coals are high volatile bituminous with low ash and low sulfur contents. Cenozoic brown coals are present in several Paleogene–Neogene basins located in the east of the country, chiefly at Muongphan, with other occurrences at Khang Phanieng, Hua Xieng and Bam O. These Paleogene–Neogene basins are highly faulted and contain subbituminous coals and lignite. At Muongphan, lignite seams are 1 m to 6 m in thickness and have high volatile and ash contents. All Laotian coal produced is used for local needs.

### *3.4.8.9 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 Paleogene–Neogene age. Those in Sabah are subbituminous with some coking properties, but often with high sulfur contents; these have been mined at Silimpopon 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.

### *3.4.8.10 Mongolia*

Coal deposits in Mongolia are concentrated in the north of the country. Highly tectonized 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 Baganur Coalfield, where seam thicknesses can be up to 25 m. In the same region occur the coalfields of Sharin Gol and Nalayh, and in the west of the country are the coalfields of Achit Nuur and Khartarbagat. 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 China will enable coal production to increase rapidly.

### *3.4.8.11 Philippines*

Throughout the Philippines archipelago are situated a series of Paleogene–Neogene basins containing coal-bearing sediments. The coals are predominantly of sub-bituminous rank although variations in rank do occur related to local structure and contemporaneous and recent igneous activity. Small-scale underground mining characterizes the bulk of the coal exploration in the Philippines, however, those deposits at Cagayan, Semirara and Zamboanga could be further developed.

The northern island of Luzon contains the Cagayan Basin, this area is only partially explored, but is known to contain seams up to 2 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 opencast mining operations. Such coals would be suitable for local electricity generation. The island of Mindoro has coal deposits in the south, where 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 m and 12 m in thickness. The coals are subbituminous with low ash and sulfur. Catanduanes Island contains lenticular seams up to 5 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 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.

#### *3.4.8.12 Taiwan*

Coals in Taiwan are Paleogene–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 tectonized, 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 m in thickness and because of the high level of tectonic disturbance there are only underground operations working at increasingly deeper levels. This may result in the cessation of mining in these areas.

#### *3.4.8.13 Thailand*

In Thailand virtually all the known coal deposits are of Paleogene–Neogene age, together with some Mesozoic coals found in the northeast of Thailand at Na Duang. The Paleogene–Neogene sediments are preserved in a series of basins, of these, the Mae Moh basin in northwest Thailand is the most extensive. Other basins in close proximity are Mae Tip, Li, Mae Tun and Vaeng Haeng. Other Paleogene–Neogene coals are found east of Bangkok at Nong Ya Plong, and at Krabi in the extreme southwest. Seams in Thailand generally range from 2 m to 12 m, however at Mae Moh and Krabi, seams up to 30 m are worked. Most coals are relatively undisturbed structurally. These Paleogene–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 as 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 industry.

#### *3.4.8.14 Peoples 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 northeast 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 northeast 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 m to 8 m are worked. Coals are low volatile anthracites with low ash and sulfur contents. In central Vietnam, in the Nong Son area is a thick seam up to 20 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.

### *3.4.9 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.

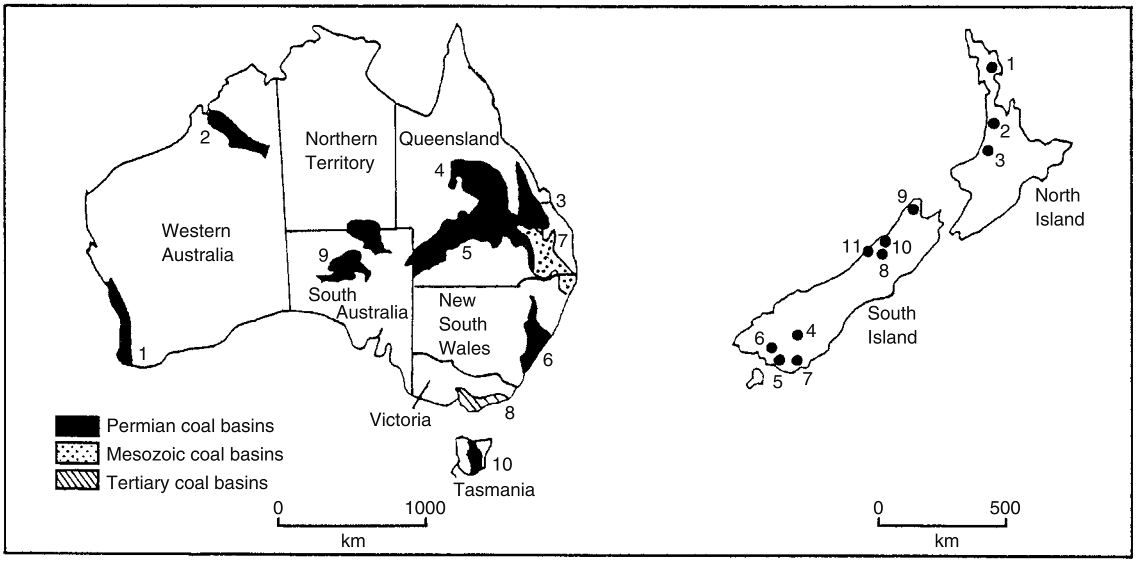



**Figure 3.15** Coal deposits of Australasia. Australia: (1) Collie Basin; (2) Fitzroy Basin; (3) Bowen Basin; (4) Galilee Basin; (5) Cooper Basin; (6) Sydney Basin; (7) Brisbane Basin; (8) Gippsland Basin; (9) Ackaringa Basin; (10) Tasmania. New Zealand: (1) Northland Coalfield; (2) Waikato Coalfields;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

The oldest coals are low ash and low sulfur seams that reach thicknesses of up to 30 m. The uppermost Permian contains four workable seams. Seam splitting is common, and in the west, igneous intrusions have locally affected the coals. 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, for example 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, and in the west, are severely affected by igneous intrusions, which although produce an increase in rank, have also destroyed large reserves of coal. Seam thicknesses reach 10 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. The principal mining districts are 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 m 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. Elsewhere in Western Australia, coals have been located but have yet to be developed.

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 developed extensively.

The Paleogene–Neogene coals of the Gippsland basin in Victoria are thick developments of lignite, and the principal seams reach enormous thicknesses of 300 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–Neogene in age. The coalfields are located in the western part of North Island, and in the northwestern and southeastern 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 m. They are subbituminous with low ash and low sulfur contents. To the north, the Northland area has a few seams up to 2 m thick and are of poorer quality. To the south of the Waikato Coal region lies the Taranaki Coal area where seams are usually less than 3 m in thickness, and are subbituminous with higher sulfur contents.

In South Island are located the Cretaceous–Neogene coalfields of Otago, Southland, Ohai and Kaitangata in the far south, and on the northwest coast, the Westland Coal region includes the Greymouth, Pike River, Charleston, Buller, Reefton and Collingwood Coalfields. The southern coalfields have seams up to 6 m and 10 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, Collingwood and Reefton. In the Buller and Greymouth Coalfields coals range from high volatile to low volatile bituminous.

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 southeast 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 by: (i) the nature of the make-up of the original organic and inorganic accumulation; and (ii) by the degree of diagenesis it has undergone.

The inherant constituents of any coal can be divided into 'macerals', the organic equivalent of minerals, and 'mineral matter', 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, 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 determines 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 and they typically have a banded appearance. **Sapropelic** coals are composed of a restricted variety of microscopic plant debris and 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 below. 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.

1. 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 come vitrinite-rich clarite.
2. 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.
3. Durain is grey to black with a dull lustre and fractures 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 can occur as extensive layers within a coal seam. Durain is composed of the microlithotypes durite and trimacerite.
4. Fusain is black, soft, friable and easily disintegrates into a black fibrous powder. Fusain occurs in coals as lenses, usually several millimetres thick, often concentraing 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 the above terms to describe coals in borehole cores and in exposures. The

*Coal Geology*, Second Edition. Larry Thomas.

**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: McCabe, 1984.

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

| Stopes (1919) | Schopf (1960); ASTM Standard (1978) | Australian Standard |
|---|---|---|
| Banded (humic) coals | | |
| Vitrain | Vitrain | Coal, bright |
| | | Coal, bright, dull bands |
| Clarain | Bright | Coal, dull and bright |
| | Moderately bright | Coal, mainly dull with numerous bright bands |
| Durain | Attrital mid-lustre coal | Coal, dull minor bright bands |
| | Moderately dull | |
| | Dull | |
| Fusain | Fusain | Coal, dull |
| Non-banded (sapropelic) coals | | |
| Cannel coal | Cannel coal | Coal, dull conchoidal (canneloid) |
| Boghead coal | Boghead coal | |
| Impure coals | Bone coal | Coal, stoney (or shaley) |
| | Mineralized coal | Coal, heat altered |
| | | Coal, weathered |

Source: Ward (1984) with permission of Blackwell Scientific Publications.

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 an 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 are summarized by McCabe (1984) in Table 4.1.

In the United States, 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.

**Table 4.3** Macroscopic description of lignites.

| Field observations | Laboratory observations | | | | Additional features |
|---|---|---|---|---|---|
| Structure | Texture | Colour | Gelification | Inclusions | |
| Pure coal (non-xylitic) | Unbanded coal | Pale yellow | Gelified groundmass | Resin bodies | Cracking to non-cracking |
| Pure coal (xylitic); fibrous/brittle; tree stumps, trunks. etc. | Moderately banded coal | Medium light yellow | Gelified tissues | Cuticles | Fracture even |
| 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 |
| Impure coal (xylitic) | Highly banded coal | Medium brown/dark brown/black | | | |

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

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 coatings and infillings. In the United States, a 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 above 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 eaxample 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 (xylitic) 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 gelification 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 as shown in Table 4.4 (Taylor *et al.*, 1998). The classification recognizes lithotype groups, lithotypes and lithotype varieties. The structure and constituents of the lithotype of soft brown coal can be recognized with the naked eye, and lithotypes can be distinguished by their degree of gelification and colour. The ICCP classification recognizes four coal types as described by Taylor *et al.* (1998).

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

| Lithotype group (constituent elements) | Lithotype (structure) | Lithotype variety (colour; gelification) |
|---|---|---|
| Matrix coal | Stratified coal | Brown (weakly gelified) Coal Black (gelified) Coal |
| | Unstratified coal | Yellow (ungelified) Coal Brown (weakly gelified) Coal Black (gelified) Coal |
| Xylite-rich coal | | |
| Charcoal-rich coal | | |
| Mineral-rich coal | | |

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

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 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 Paleogene–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 mineralized. Inclusions of charcoal or gelified 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 mineralization 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 gelification level, weathering character and physical 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.

#### *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, that is 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 recognized. 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** (**liptinite**) – spores, resins and cuticles;
3. **inertinite** – oxidized plant material.

The original classification of maceral groups is referred to as the Stopes–Heerlen System given in Table 4.6 Other detailed descriptions are well summarized by (McCabe, 1984) in Table 4.7. 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, it must contain 5% of accessory macerals. All microlithotypes may contain amounts of mineral matter, but if this reaches 20% then the microlithotype is referred to as a carbominerite (Taylor *et al.*, 1998).

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

| Lithotype | Abbreviation | Colour | Texture* | Gelification | Weathering pattern | Physical properties |
|---|---|---|---|---|---|---|
| Dark | Dk | Dark brown to medium brown | High (20–30%) wood content. Often small (<25 cm) fragments | Gelification, 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 gelification 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 | Gelification uncommon. Confined mainly to wood | Cracks shallow. Irregular pattern | Intermediate physical properties |
| Light | Lt | Light brown | Medium to low wood content | Gelification rare | Cracks generally fine. Random orientation | Generally soft and relatively light (low SG) |
| Pale | Pa | Pale brown to yellow brown | Wood present but uncommon | Gelification very rare | Few extensive cracks | Soft, crumbles readily, very low SG |

*Wood content includes all plant fragments clearly distinguishable from the groundmass. Physical properties: SG = specific gravity.

Source: Taylor *et al* (1998) modified after George (1975). Reproduced by permission of Gebruder Borntraeger

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

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

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

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

Smyth (1984) relates the microlithotype composition of Permian coals in eastern Australia to depositional environments as shown in Figure 4.3. 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, low in the Late Permian.

1. **Vitrite** comprises 95% of the vitrinite macerals telinite and collinite in bands at least 50$\mu$m thick (Figure 4.2). Vitrite occurs in coal seams as elongated lenses several millimetres thick. Vitrite originated in anaerobic conditions due to high groundwater table levels in the peat mire. Vitrite 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 Paleogene–Neogene coals are generally rich in vitrinite and comparatively rich in liptinite, usually having >20% inertinite.
2. **Liptite** 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.
3. **Inertite** microlithotypes contain >95% inertinite macerals, which include inertodetrite, 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. Inertodetrite, 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%. Inertodetrite is present in Northern Hemisphere Carboniferous coals and other coals as a minor constituent. The high level of inertinite in Gondwana coals has been attributed to the peat being oxidized to a high degree during formation (Plumstead, 1962). Taylor *et al.* (1998) consider the characteristic petrographic composition of many Gondwana coals to be attributable to a climate of wet cool summers and freezing winters, with the oxygen content of inertinite having been retained in its structure at an early stage as a result of drying or freeze drying. Similar material in warmer climates would have proceeded toward vitrinization.
4. **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 Paleogene–Neogene hard coals. Liptinite-rich clarites may owe their formation from algae, lipid-rich plants and animal plankton, and as such may grade into sapropelic coals.
5. **Vitrinertite** contains 95% vitrinite and inertite. It is rare in Carboniferous coals and common in inertinite-rich Gondwana coals.
6. **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) suggest 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.

**Table 4.7** Macerals and group macerals recognised in hard coals.

| Maceral Group | Maceral | Morphology | Origin |
|---|---|---|---|
| Vitrinite (huminite) | Telinite | Cellular structure | Cells walls of trunks, branches, roots, leaves |
| | Collinite | Structureless | Reprecipitation of Dissolved organic matter in a gel form |
| | Vitrodetrinite | Fragments of vitrinite | Very early degradation of plant and humic peat particles |
| | Sporinite | Fossil form | Mega-microspores |
| | Cutinite | Bands which may have appendages | Cuticles – the outer layer of leaves, shoots and thin stems |
| Exinite (liptinite) | Resinite | Cell filling layers or dispersed | Plant resins, waxes and other secretions |
| | Alginite | Fossil form | Algae |
| | Liptodetrinite | Fragments of Exinite | Degradation residues |
| | Fusinite | Empty or mineral filled cellular structure; cell structure usually well preserved | Oxidized plant material – mostly charcoal from burning of vegetation |
| | Semifusinite | Cellular structure | Partly oxidized plant Material |
| | Macrinite | Amorphous 'cement' | Oxidized gel material |
| Inertinite | Inertodetrinite | Small patches of fusinite, semi-fusinite or macrinite | Redeposited inertinites |
| | Micrinite | Granular, rounded Grains ~1 μm in diameter | Degradation of macerals during coalification |
| | Sclerotinite | Fossil form | Mainly fungal remains |

Source: McCabe (1984).

7. **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.

**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: McCabe (1984).

In low-rank coals, that is lignites and subbituminous coals, the **vitrinite** maceral group is referred to as **huminite**, and is regarded as equivalent to, and the precursor of, the vitrinite macerals found in higher rank coals. Bustin *et al*. (1983) classify huminite macerals as summarized in Table 4.9 and give details of their origin and their equivalents in the hard coals. The increase of coal rank leads to the homogenization 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) give a maceral classification of brown coals based on the Australian Standard 2856.2 (1998) (Table 4.10).

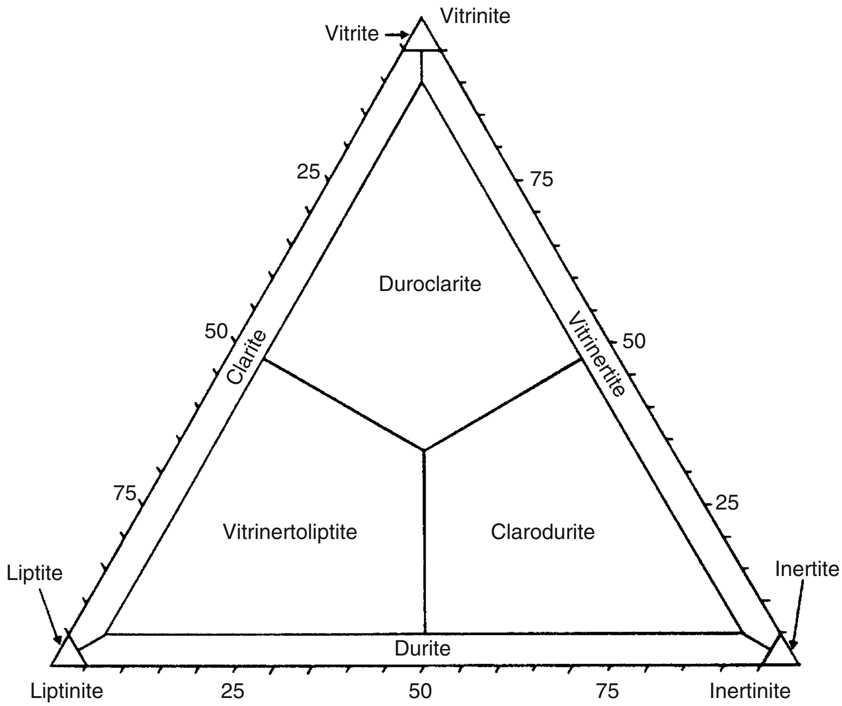



**Figure 4.1** Diagrammatic representation of microlithotype classification. (From Bustin *et al.* (1983), by permission of Geological Association of Canada.)

The relationship of types of plant debris to microlithotypes in lignite microfacies has been proposed by Kasinski (1989) and 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 4.4. 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.

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, 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.

### *4.1.3 Mineral content of coals*

The mineral content of coal is the non-combustible inorganic fraction, which 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.

**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

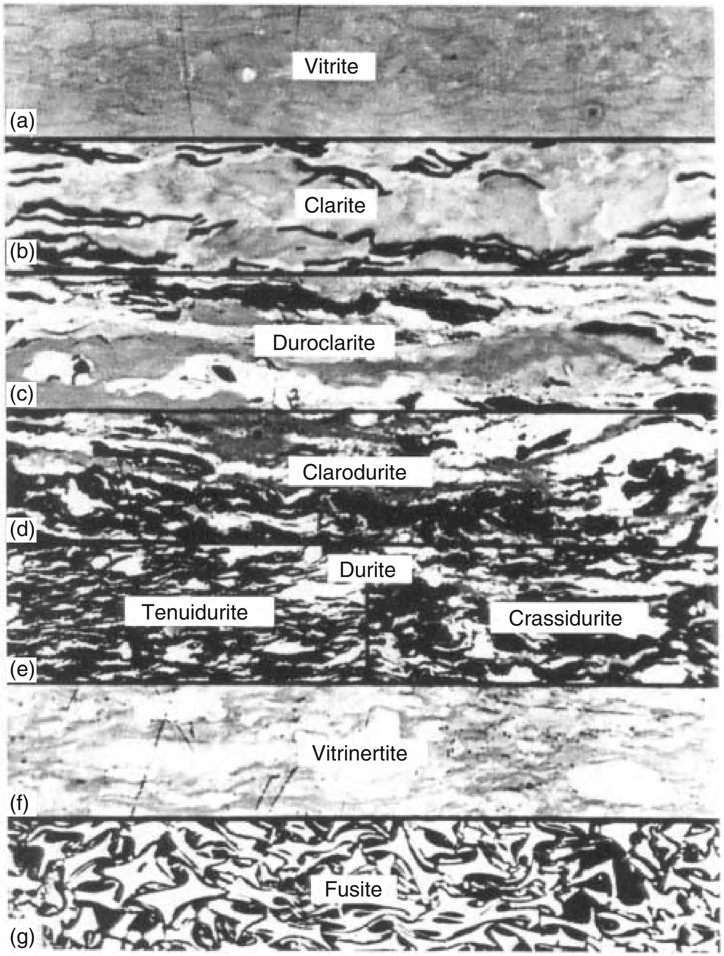



**Figure 4.2** Microlithotypes: (a) vitrite from a high-volatile Ruhr coal, polished surface, oil imm. ×300; (b) clarite from a Saar coal, polished surface, oil imm. ×300; (c) duroclarite from a high volatile ruhr coal, polished surface, oil imm. ×300; (d) clarodurite from a high volatile Ruhr coal, polished surface, oil imm. ×300; (e) durite from a high volatile Ruhr coal, showing both tenuidurite and crassidurite, polished surface, oil imm. ×300; (f) vitrinertite from a high volatile Ruhr coal, polished surface, oil imm. ×300; (g) fusite from a high volatile Ruhr coal, polished surface, oil imm. ×300. (From Taylor, 1998.)

the peat swamp may be incorporated into the organic layer by differential compaction within the swamp and by bioturbative action.

Windborne mineral matter is important, as this can be a significant contributer 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 mineralization 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.

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 can be a secondary product from aqueous solutions. Chemical conditions at the site of deposition also influence the

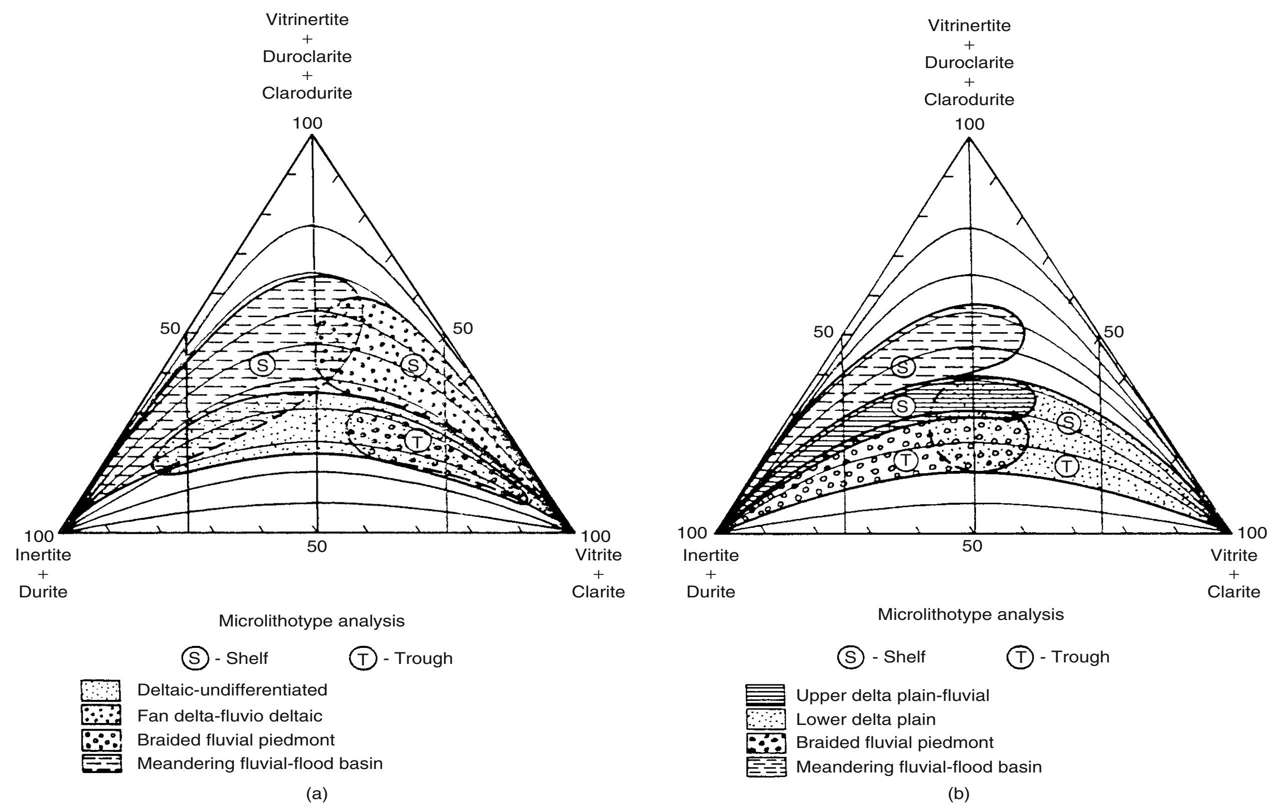



**Figure 4.3** Microlithotype analysis related to depositional environment, for Early and Late Permian coals, Australia (Smyth, 1984.)

**Table 4.9** Huminite macerals.

| Maceral | Origin | Petrological features | Equivalents 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 $<10\,\mu m$, product of degradation of huminite macerals | Desmocollinite |
| Densinite | Same as attrinite | Tighter packed than attrinite | |
| Gelinite | Derived from colloidal humic solutions which 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: Bustin *et al.* (1983), by permission of Geological Association of Canada.

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.

**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 (1998) and George and Mackay (1991).

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) 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 above forms of mineral content in coals can be identified macroscopically by the field geologist in outcrop and borehole core. There are other minerals

**Table 4.11** Characteristics of Tertiarty 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 (above 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, detrinite and colinite (10%); high sporinite content; significant addition of clay minerals; strong bituminization; traces of decay |
| Pollen-resin grains | | Relatively high frequency of pollen grains (60%); resin grains (20%); fungi spores (10%); relatively low wood | Distinct addition of sapropelite (10–20%) and tissue frequency (15%); higher inertinite content; low content telinite, detrinite, colinite (20% total); small distinct addition of clay minerals |
| Coniferae Wood-Tissue | Highest frequency of Taxodiacae–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 detrinite (20%) |
| Fungi-Sporesand wood Tissue | Highest frequency of Myricaceae – Cyrillaceae–*Alnus* association (30–40%); addition of Polypodiaceae spores | Relatively high frequency of resin grains (20%) and fungi spores (10%); relatively low pollen grain frequency (40%) | Highest colinite content (40%); relatively high telinite content (30%) and detrinite (20%); high content of gelinite; significant resinite addition |
| *Sequoia* Wood-Tissue | Highest frequency of *Sequoia–Pinus* association | High content of wood tissue with texture for *Sequoia* | High content of telinine and detrinite with the characteristic vein texture |

Source: based on Kasinski (1989).

that may be present in coal that affect its future potential use, but which cannot be seen in hand specimen and are 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, and in particular their resistance and response to water. It will also influence the composition of mine dust with a diameter of below 5$\mu$m, particularly in underground operations. Significant amounts of quartz in dust affects 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: the low ash fusion point causes deposition of ash and corrosion in the heating chamber and convection passes of the boiler. Figure 4.5 shows 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

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

| Group | Mineral | Occurrence* |
|---|---|---|
| Clay minerals | Illite–Sericite | Common–abundant |
| | Montmorillonite | Rare–common |
| | Kaolinite | Common–abundant |
| | Halloysite | Rare |
| Iron disulfides | Pyrite | Rare–common |
| | Marcasite | Rare–common |
| Carbonates | Siderite | Common–very common |
| | Ankerite | Common–very common |
| | Calcite | Common–very common |
| | Dolomite | Rare–common |
| | Aragonite | Rare |
| | Witherite | Rare |
| | Strontianite | Rare |
| Oxides | Haematite | Rare |
| | Quartz | Rare–common |
| | Magnetite | Very rare |
| | Rutile | Very rare |
| Hydroxides | Limonite | Rare–common |
| | Goethite | Rare |
| | Diaspore | Rare |
| Sulfides (other than iron) | Sphalerite | Rare |
| | Galena | Rare |
| | Millerite | Very rare |
| | Chalcopyrite | Very rare |
| | Pyrrhotite | Very rare |
| Phosphates | Apatite | Rare |
| | Phosphorite | Rare |
| | Goyazite | Rare |
| | Gorceixite | Rare |
| Sulfates | Barite | Rare |
| | Gypsum | Very rare |
| Silicates (other than clays) | 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 |
| Salts | Gypsum | Rare |
| | Bischofite | Very rare–common |
| | Sylvin(Sylvite) | Very rare–common |

(*continued overleaf*)

**Table 4.12** (*Continued*)

| Group | Mineral | Occurrence* |
|---|---|---|
| | Halite | Very rare–common |
| | Kieserite | Very rare–common |
| | Mirabilite | Very rare–rare |
| | Melanterite | Very rare |
| | Keramohalite | Very rare |

*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 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: Taylor *et al.* (1998).

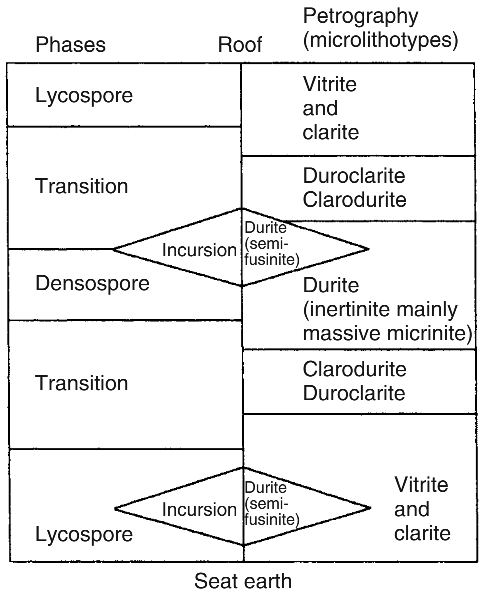



**Figure 4.4** Diagrammatic profile of a coal seam showing the sequence of miospore phases and petrographic types. (From Smith (1968) with permission.)

in coal of phosphorous minerals, usually in the form of phosphorite or apatite, causes slagging in certain boilers, and steel produced from such phosphorous-rich coals tends to be brittle.

Halide minerals such as chlorides, sulfates and nitrates are present in coal usually as infiltration products deposited from brines migrating through the sedimentary sequence. 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 summarized 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 are 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).

### *4.1.4 Coal petrography*

The microscopic study of coal has enabled a better understanding of its organic and mineral components and its industrial utilization. Classic petrographic studies in relation to geology were carried out by Thiessen (1920), Stach (1982) and Teichmuller and Teichmuller(1982), Teichmuller (1987,1989) on Carboniferous coals in Europe. Teichmuller (1987, 1989) compared coal petrography and genesis of coal and the process of coalification, whereas other studies dealt with the

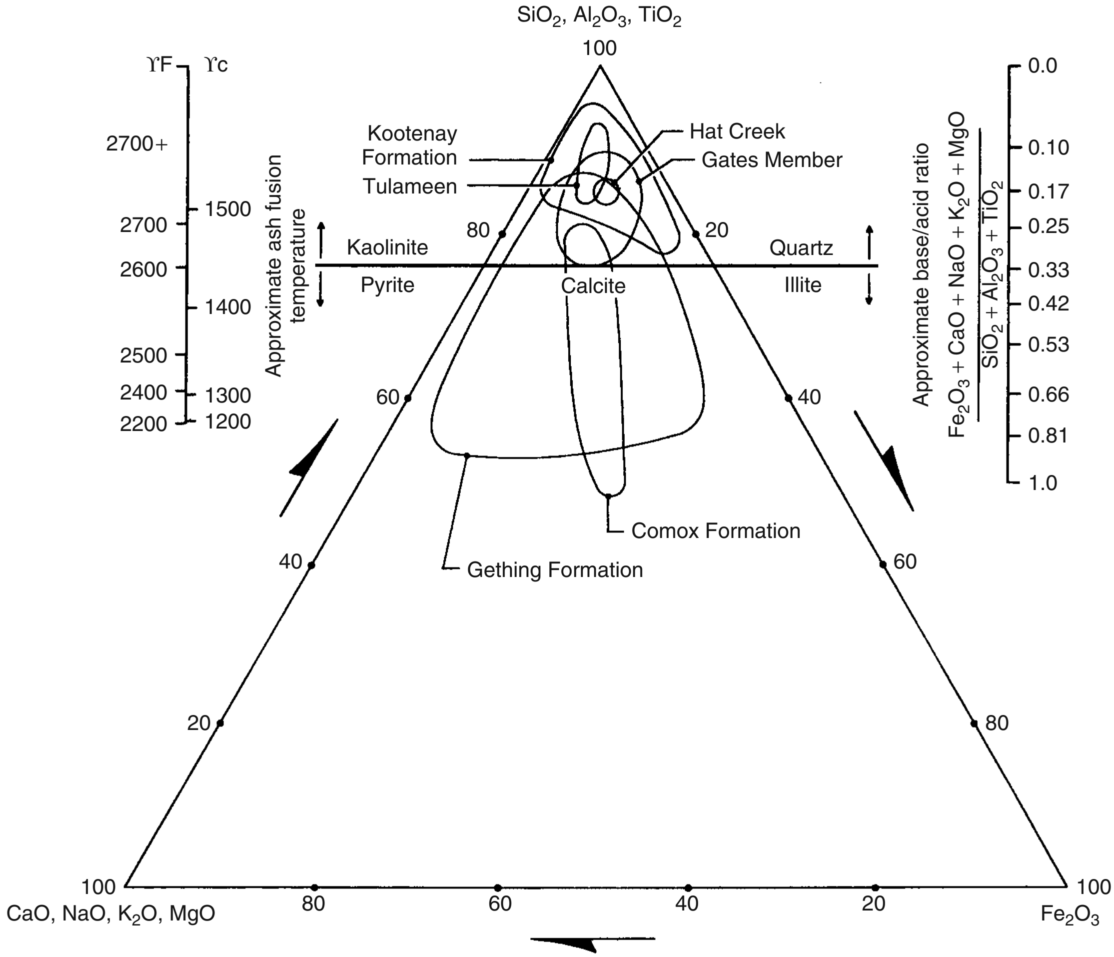



**Figure 4.5** Ternary diagram showing ash chemistry of some of western Canada's coals, together with approximate ash fusion temperatures, and base-acid ratios. Approximate boundary of pyrite–calcite–illite (marine influenced) coal ashes and kaolin–quartz (freshwater dominated) coal ashes is also shown. (From Pearson (1985) with permission.)

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 super continent 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 Paleogene–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 are low in inertinite. The majority of these coals are low rank and do not exhibit the fine banding that characterizes the older coals (Taylor *et al.*, 1998).

#### *4.1.4.1 Industrial utilization*

The principal uses of black coals on a worldwide 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, but even so coal still retains a 42% 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

**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 | 100–3000 | 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 | 8–1800 | 160 |
| Chromium (Cr) | 0.5–60 | 5–1000 | 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 | 200–3000 | 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 |
| Phosphorous (P) | 10–3000 | 35–5300 | 700 |
| Rubidium (Rb) | 2–50 | 20–1000 | 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 | 50–1000 | 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 | 1000–10000 | 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 | 60–2000 | 160 |

Source: Taylor *et al.* (1998).

Callcott (1990) and Taylor *et al.* (1998). Coals that are to be used for conventional coke production must have three essential properties.

1. They must be within a specific range in rank for the coking process to occur, that is bituminous coal.
2. They must possess a high proportion of fusible macerals (>40% vitrinite) to form a strong well-fused coke.
3. 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. 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 the formation of coke. However, a stronger coke is obtained if the vitrinite is reinforced by inertinite. The liptinite group are characterized by high H/C 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 semifusinite, do not fuse during carbonization due to their insufficient hydrogen content. These macerals are characterized by higher O/C 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 $R_o$ maximum) 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 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 reactives 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 reactives present. This ratio of inerts to reactives 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 peaks at the point at which the optimum 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 of SI, CBI to 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 calorific value of the coal that is of prime interest, that is the percentage of combustable matter against non-combustable matter (mineral matter and water). Liptinite with high H/C ratio has the highest calorific value followed by vitrinite and inertinite, however, vitrinite and inertinite increase in calorific value with increasing rank whereas liptinite declines.

## 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

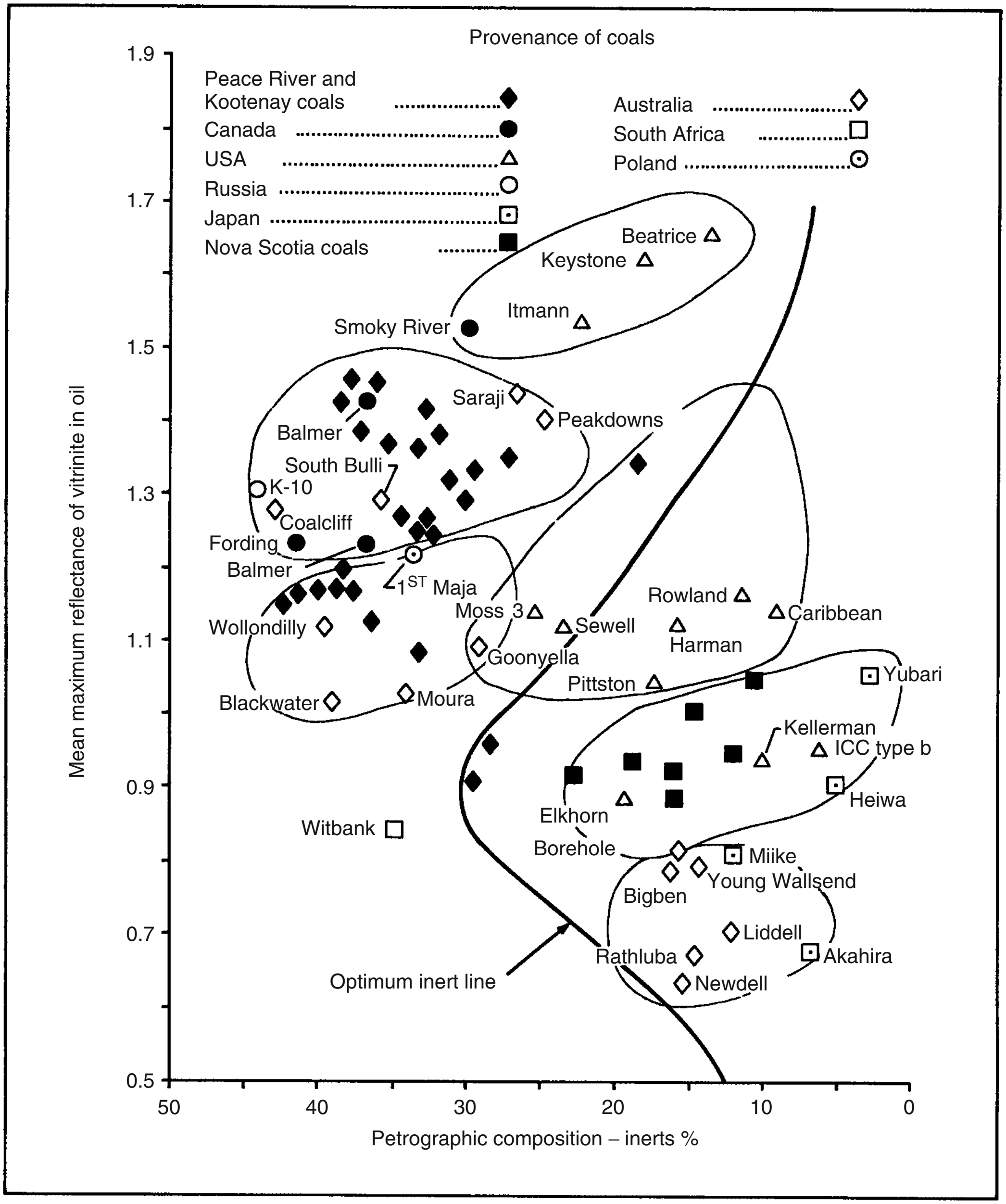



**Figure 4.6** Petrographic compositions of coking coals traded internationally. (From Pearson (1980) with permission.)

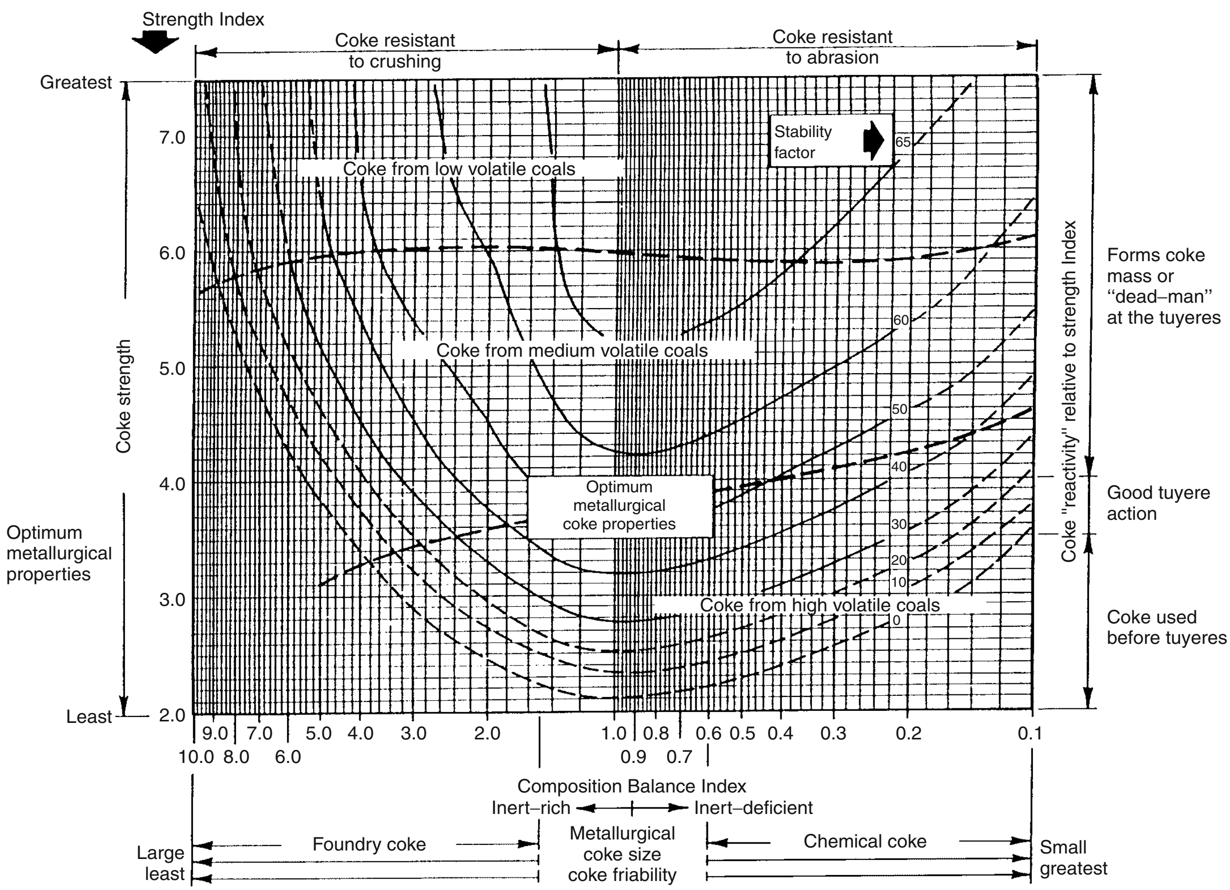



**Figure 4.7** Characteristics of coke attainable from bituminous coals. (From Zimmerman, 1979.)

**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 gelification, 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, | Decreased moisture contents and O/C, $CO_2$ and $H_2O$ 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 $R_o$, increased fluorescence, increased extract yields, decrease in density and sorbate accessibility, increased strength |
| 4 Debituminization | Uppermost high volatile A to low volatile bituminous | Cracking, expulsion of low molecular weight hydrocarbons, especially methane | 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 XRD peaks, increased sorbate accessibility, anisotrophy, strength ring condensation and cleat healing |

Source: Taylor *et al.* (1998), according to Levine (1993).

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), who describe the major stages of coalification from peat to meta-anthracite, which are summarized in Table 4.14. The table 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, that is 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 bacterialogical activity, which degrades the peat and may be assisted in this by the rate of burial, pH and levels of groundwater in the swamp. With increased burial, bacterialogical activity ceases, and is 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 calorific value 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, and coals can therefore 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, carbon dioxide and water,

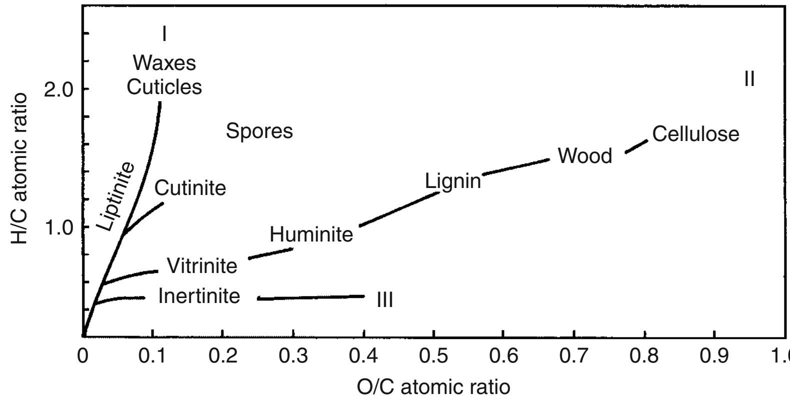



**Figure 4.8** Diagram showing the coalification tracks of liptinite, inertite and huminite–vitrinite. (From Bustin *et al.* (1983), based on van Krevelen (1961), by permission of Geological Association of Canada.)

but water is quickly lost and the methane to carbon dioxide ratio increases with rank. These changes in the physical and chemical properties of the coal are in reality the changes to the inherant coal constituents. During coalification the three maceral groups become enriched in carbon and each maceral group (i.e. exinite, inertinite and huminite (vitrinite)) follows 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.

In reflected light the reflectance progressively increases, whereas in transmitted light organic materials become opaque and plant structure becomes difficult to recognize. The optical properties of vitrinite have enabled it to be used as an indicator of rank. Teichmuller and Teichmuller (1982) describe the method used in detail as applied to the medium volatile bituminous to meta-anthracite and semigraphite range of coals, that is 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 have shown the utility of reflectance in low-rank lignitic coals, provided that care is taken in the selection of the component measured. Ward (1984) suggests rank classes in terms of vitrinite reflectance (Table 4.15), and Table 4.16 shows the changing pattern of coal composition with increasing coalification (Diessel, 1992). This increase in vitrinite reflectance according to increase in coal rank is shown in Figure 4.9(a) for New Zealand coals, which have high proportions of vitrinite

**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 (approximately) |
| Anthracite | >3.00 |

Source: Ward (1984) with permission of Blackwell Scientific Publications.

and most fall within a restricted band on a volatile matter/calorific value plot. The mean reflectance values given in Figure 4.9b are reported to be on the 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 (d.m.m.f.) for non-marine Canadian Cretaceous coal, Australian non-marine Gondwana coal and marine-influenced Pennsylvanian coal from the United States. In general, the non-marine Cretaceous coals possess lower volatile yields than the non-marine Gondwana and marine-influenced Pennsylvanian coals. This

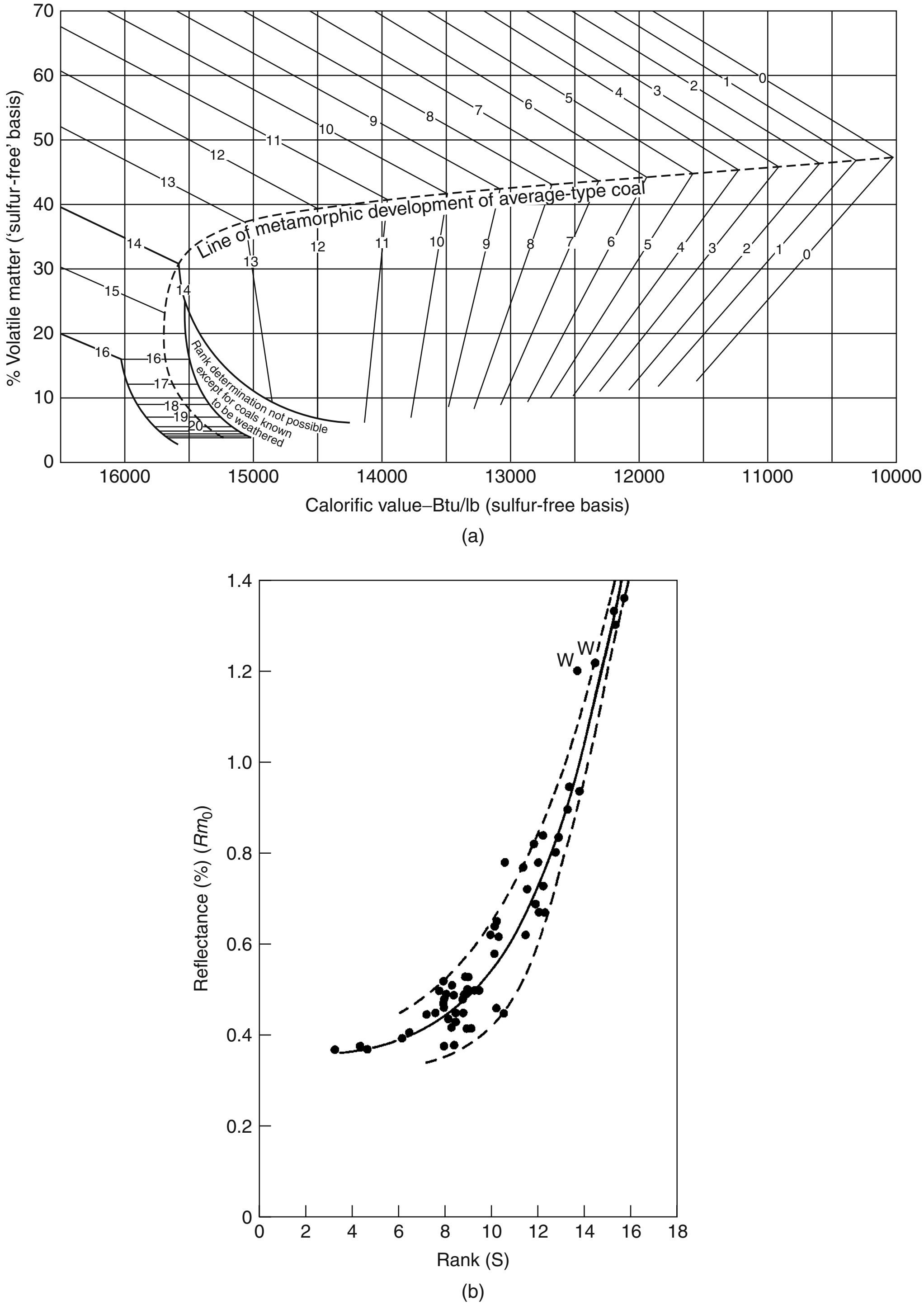
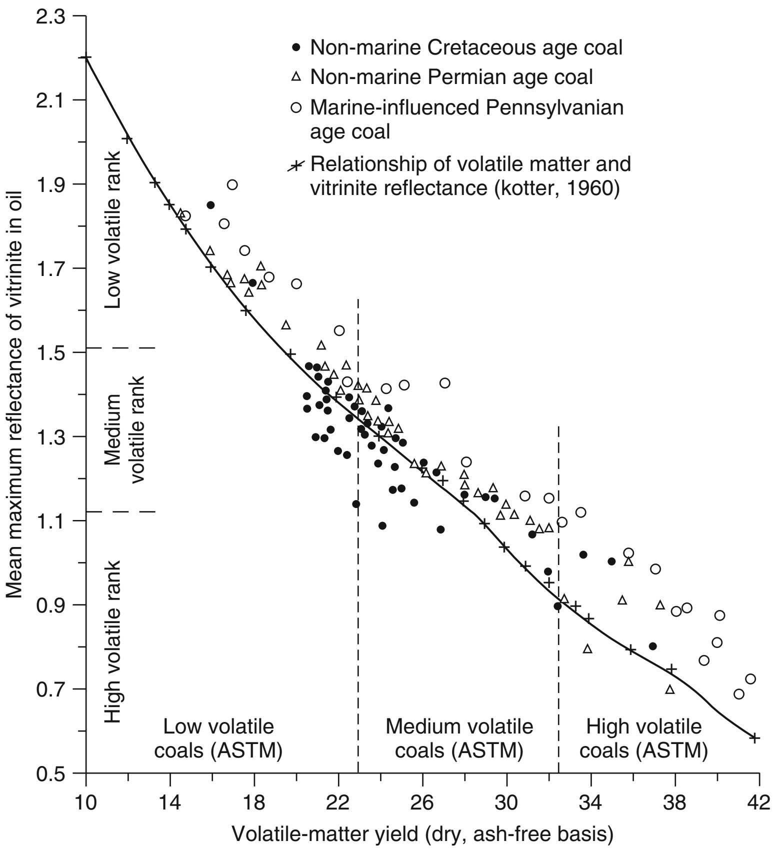



**Figure 4.9** (a) Rank scale of coal using axes of volatile matter and calorific value. (From Suggate (1959), with permission of the Royal Society of New Zealand.) (b) Reflectance/rank relationship (note: W = weathered coal). (From Suggate and Lowery (1982), with permission of the Royal Society of New Zealand.) (c) Relationship between $R_{o\ max}$ and volatile matter yield (d.m.m.f.) for non-marine Canadian Cretaceous coals, Permian Australian coals and marine-influenced Pennsylvanian coals from the United States. (From Pearson (1985) with permission.)




(c)

**Figure 4.9** (*Continued*)

**Table 4.16** Some rank parameters showing the changing pattern of coal composition with increasing coalification.

| Rank stages | % carbon (d.a.f.) | % volatile matter (d.a.f.) | Gross CV (MJ/kg) | *% 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 |

Source: Diessel *et al.* (1992).

variation is a consequence of compositional differences, the Cretaceous coals having a higher inertinite content (Pearson, 1985).

Fluorescence microscopy of the liptinite macerals and the colouration 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.

#### *4.2.2.1 Temperature*

Temperature changes can be achieved in the two following ways.

1. 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, methane and water, and the surrounding sediments will show evidence of contact metamorphism, for example, the local development of high rank coal in the Gondwana coals of South Africa and India, and in the Paleogene–Neogene coals of Sumatra, Indonesia.
2. 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 phenomena and Hilts Law states:

> '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 dependant on the geothermal gradient and the heat conductivity of the rocks. Where the geothermal gradient is high (70–80°C per km depth), bituminous rank can be attained at depths of 1500 m (Upper Rhine Graben, Germany), whereas in the same area, the same rank is reached at depths of 2600 m when the geothermal gradient is lower (40°C $km^{-1}$) (Stach, 1982). Similar basinal studies have shown variations in geothermal gradient in different parts of the basin (Teichmuller and Teichmuller (1982; Teichmuller, 1987). Studies of the Remus Basin in the Canadian Arctic show differing geothermal gradients of 55°C $km^{-1}$ in the eastern part, and 20°C $km^{-1}$. 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 anthracitization is due to the proximity of a magmatic heat source. The anthracite field has a present-day geothermal gradient of 25°C $km^{-1}$. Figure 4.10 illustrates the manner in which ASTM rank boundaries vary in depth from surface according to the geothermal gradient, as reflected by variations in the moisture and calorific value relationships.

#### *4.2.2.2 Time*

Usually coalification temperatures are lower than was once inferred from experimental coalification studies. Stach (1982) quotes temperatures of the order of 100–150°C 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', while 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 advance when additional heat is supplied, for example, from intrusive

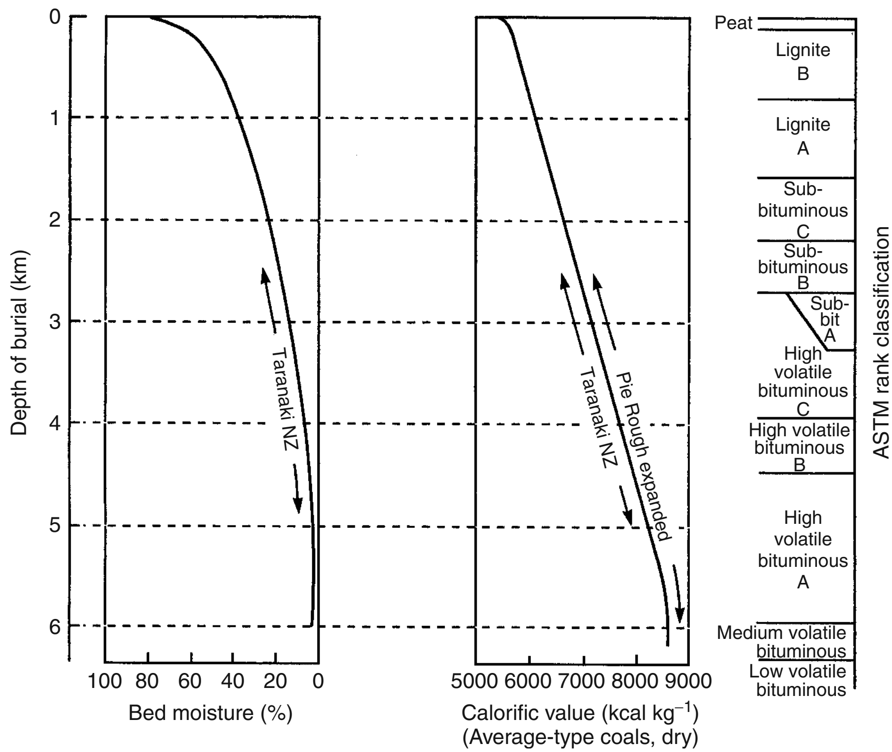



**Figure 4.10** Composite sequence providing an example of the relationship between depth, calorific value and ASTM rank. The average geothermal gradient for this sequence is estimated at 26–27° C km$^{-1}$. (From Suggate, 1982.)

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, this 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, then they can be blended with other selected coals to achieve the 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 organizations have defined standards of procedure that should be consulted (Appendix 1).

A knowledge of the most commonly determined properties of a coal is important, in particular those which are deleterious to the coal. Such coal analyses are essential in the evaluation of a coal deposit, that is 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, for example, the discovery that a coal has good coking properties when it was originally considered for a steam coal product.

An outline is given below 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.

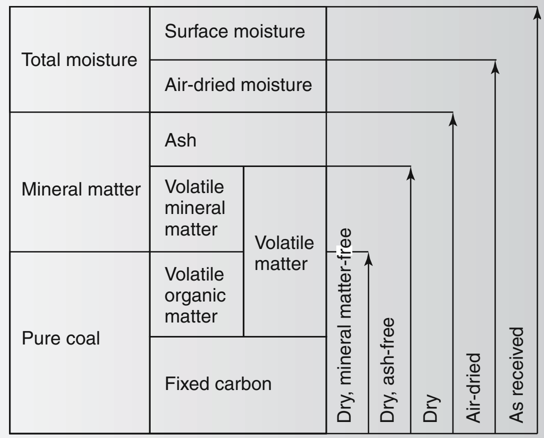
Coal analyses are often reported as a proximate or an ultimate analysis. Proximate analysis is a broad analysis that determines the amounts of moisture, volatile matter, fixed carbon and ash. This is the most fundamental of all coal analyses and is of great importance in the practical use of coal. The tests are highly dependent on the procedure used, and different results are obtained using different times and temperatures. It is therefore important 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, that is 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, and 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 the basis on which 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 that do not indicate on what basis they are presented. Indeed they are often listed together on different bases that are not stated.

**Table 4.17** Components of coal reporting to different bases.




Source: Ward (1984) with permission of Blackwell Scientific Publications.

Coal analyses may be reported as follows (see Table 4.17):

1. 'As received' basis (a.r.), 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.
2. 'Air-dried' basis (a.d.b.). 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.
3. 'Dry' basis (dry). The data are expressed as percentages of the coal after all the moisture has been removed.
4. 'Dry ash-free' basis (d.a.f.). The coal is considered to consist of volatile matter and fixed carbon on the basis of recalculation with moisture and ash removed. It should 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.
5. 'Dry, mineral matter-free' basis (d.m.m.f.). 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 formulae for the calculation of results to the above bases (reference BS ISO 1170–2008, Appendix 1). In addition, the following countries have developed equations to calculate the mineral matter content of their coals.

**Table 4.18** Formulae for calculation of results to different bases.

| Given result | Wanted result | | | | |
|---|---|---|---|---|---|
| | As sampled (as received) (as despatched) | Air dried (as fired) | Dry | Dry, ash-free | Dry, mineral-matter-free |
| As sampled (as received) (as despatched) (as fired) | – | $\frac{100 - Mad}{100 - Mar}$ | $\frac{100}{100 - Mar}$ | $\frac{100}{100 - (Mar + Aar)}$ | $\frac{100}{100 - (Mar + MMar)}$ |
| Air dried (as analysed) | $\frac{100 - Mar}{100 - Mad}$ | – | $\frac{100}{100 - Mad}$ | $\frac{100}{100 - (Mad + Aad)}$ | $\frac{100}{100 - (Mad + MMad)}$ |
| Dry | $\frac{100 - Mar}{100}$ | $\frac{100 - Mad}{100}$ | – | $\frac{100}{100 - Ad}$ | $\frac{100}{100 - MMd}$ |
| Dry, ash-free | $\frac{100 - (Mar + Aar)}{100}$ | $\frac{100 - (Mad + Aad)}{100}$ | $\frac{100 - Ad}{100}$ | – | $\frac{100 - Ad}{100 - MMd}$ |
| Dry, mineral-matter-free | $\frac{100 - (Mar + MMar)}{100}$ | $\frac{100 - (Mad + MMad)}{100}$ | $\frac{100 - MMd}{100}$ | $\frac{100 - MMd}{100 - Ad}$ | – |

M = moisture %; A = ash %; MM = mineral matter %; ar = as received basis; ad = air dried basis; d = dry basis.
Source: BS 1016-100 (1994). Reproduced with permission of BSI under Licence Number 2002 SK/0003.

1. North America: original Parr formula

$$MM = 1.08A + 0.55S;$$

modified Parr formula

$$MM = 1.13A + 0.47Spyr + Cl.$$

2. United Kingdom: BCURA formula

$$MM = 1.10A + 0.53S + 0.74CO_2 - 0.36;$$

KMC formula (revised by British Coal)

$$MM = 1.13A + 0.5Spyr + 0.8CO_2 - 2.8SAsh + 2.8SSulf + 0.3Cl.$$

3. Australia: Standards Association of Australia formula

$$MM = 1.1A.$$

In the above equations MM = mineral matter (%), A = ash (%), S = total sulfur (%), Spyr = pyritic sulfur (%), SSulf = sulfate sulfur (%), SAsh = sulfur in ash (%), Cl = chlorine % and $CO_2$ = carbon dioxide (%). All values are expressed on an air-dried basis.

### *4.3.1.2 Proximate analysis*

***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 at all 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 the following set of empirically determined definitions.

1. Surface moisture. This is adventitious moisture, not naturally occurring with the coal and which can be removed by low temperature air drying (ca. 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 (ca. 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 this generally used term, the following terms are being increasingly used, moisture holding capacity (MHC), capacity moisture or equilibrium moisture (EQ). It is not within the scope of this book to detail the analytical procedure required but suffice to say that they are lengthy and expensive.

These terms relate to the in-bed or *in situ* moisture of a coal. Numerically the MHC of a bituminous coal will be higher than air-dried moisture and lower than total moisture. Technically it is the MHC that increases with decreasing rank (Figure 4.11). 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 calorific value in steam coals and lowers the amount of carbon available in coking coals.

***Ash***

The ash of a coal is that 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 coal after losing the volatile components such as $CO_2$, $SO_2$ and $H_2O$, 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 calorific value. Recommended maximum ash contents for steam coals for use as pulverized fuel are around 20% (air-dried), but for some stoker-fired boilers, much lower values are desirable. In coking coals, a maximum of 10–20% (air-dried) is recommended, as higher ash contents reduce the efficiency in the blast furnace.

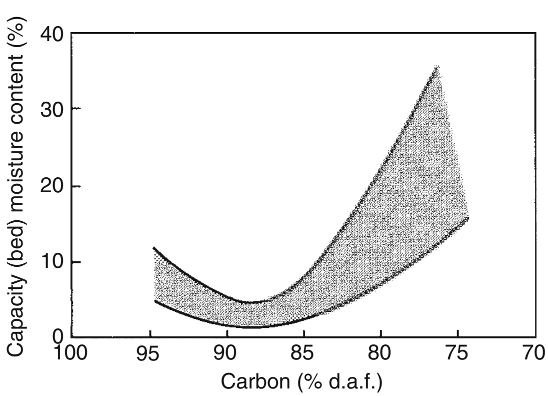



**Figure 4.11** Generalized variation of capacity (or air-dried) moisture contents with rank. (From Berkowitz, 1979.)

***Volatile matter***

Volatile matter represents that 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 is not usually necessary in commercial practice.

In pulverized fuel firing for electricity generation, most boilers are designed for a minimum volatile matter of 20–25% (d.a.f.). In stoker firing for electricity generation, the volatile matter limits recommended are 25–40% (d.a.f.). 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% (air-dried) but values of 16–36% can be used.

***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, that is 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.

***Carbon and hydrogen***

These are liberated as $CO_2$ and $H_2O$ when the coal is burned and are most easily determined together.

However, $CO_2$ may be liberated from any carbonate minerals present, and $H_2O$ 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.

***Nitrogen***

The nitrogen content of coal is significant particularly in relation to atmospheric pollution. Upon combustion of the coal, nitrogen helps to form oxides, which may be released as flue gases and thereby pollute the atmosphere, and consequently coals that are low in nitrogen are preferred by industry. Coals should not as a rule have nitrogen contents of more than 1.5–2.0% (d.a.f.) because of these $NO_x$ emissions.

***Sulfur***

As in the case of nitrogen, the sulfur content of coals presents problems with utilization 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.

1. Organic sulfur, present in the organic compounds of the coal.
2. Pyritic sulfur, present as sulfide minerals in the coal, principally iron pyrite.
3. 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% (air-dried); the maximum value will depend upon local emission regulations. In the cement industry, a total sulfur content of up to 2.0% (air-dried) is acceptable, but a maximum of 0.8% (air-dried) is required in coking coals, because higher values affect the quality of steel.

***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 oxidized, oxygen may be present in oxides, hydroxides and sulfate minerals, as well as oxidized 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*

***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 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.

***Carbon dioxide***

Carbon dioxide in coal occurs in the carbonate mineral matter fraction. The carbonates liberate $CO_2$ on combustion, and contribute to the total carbon content of the coal, however, this reaction reduces the amount of energy available from the coal.

***Chlorine***

The chlorine content of coal is low, usually occurring as the inorganic salts of sodium, potassium and calcium chloride. 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 chloride content of 0.2–0.3% (air-dried), and for coals used in the production of cement, a maximum of 0.1% (air-dried) is recommended.

***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).

***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 that make up the lithosphere. These are $SiO_2$, $Al_2O_3$, $TiO_2$, CaO, MgO, $K_2O$, $Na_2O$, $P_2O_5$, $Fe_2O_3$ and $SO_3$. 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 content should be a maximum of 3% in ash, as high alkalis cause high coke reactivity.

***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, whereas 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; 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, and 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, that is its calorific value 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 calorific value (CV) of a coal is the amount of heat per unit mass of coal when combusted. Calorific value is often referred to as specific energy (SE), particularly in Australia. The CV of a coal is expressed in two ways.

1. The gross calorific or higher heating value. This is the amount of heat liberated during testing in a laboratory, when a coal is combusted under standardized conditions at constant volume, so that all of the water in the products remains in the liquid form.
2. The net calorific or lower heating value. During actual combustion in furnaces, the gross calorific value is never achieved because some products, especially water, are lost with their associated latent heat of vapourization. The maximum achievable calorific value under these conditions is the net calorific value at constant pressure. This can be calculated and expressed in absolute joules, calories per gram, or Btu per pound. The simplified equations for these are as follows:

$$\text{net CV} = \text{gross CV} - 0.212\text{H} - 0.024\,M \quad \text{in MJ kg}^{-1};$$
$$\text{net CV} = \text{gross CV} - 50.7\text{H} - 5.83\,M \quad \text{in kcal kg}^{-1};$$
$$\text{net CV} = \text{gross CV} - 91.2\text{H} - 10.5\,M \quad \text{in Btu lb}^{-1};$$

where H = hydrogen (%) and $M$ = moisture (%).

As an approximate value, in bituminous coals, gross as received calorific value can be converted to net as received calorific value by subtracting the following values: 1.09 MJ $kg^{-1}$; 260 kcal $kg^{-1}$; 470 Btu $lb^{-1}$. It should be noted that in practice, the United States use Btu $lb^{-1}$, the United Kingdom has used Btu $lb^{-1}$, although the British coal industry uses gigajoule per tonne (this is not used elsewhere). South Africa and Australia use megajoule per kilogram, whereas kilocalories per kilogram is usually used in the rest of the world. A conversion chart is given in Appendix 3.

#### *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, that is how it will behave in a furnace or boiler. A laboratory prepared and moulded ash sample (either in the shape of a cone, cube or cylinder) is heated in a mildly reducing or oxidizing atmosphere, usually to about 1000–1600°C. Four critical temperature points are recognized:

1. initial deformation temperature (IT), the temperature at which the first rounding of the apex or corners of the sample occurs;

2. softening (sphere) temperature (ST), the temperature at which the moulded sample has fused down to a lump, the width of which equals its height;
3. hemisphere temperature (HT), the temperature at which the mould sample has fused down to a lump the height of which is half of its width;
4. 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 oxidizing atmosphere. The IT and FT temperatures 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°C is recommended as lower values lead to excessive clinker formation. In PF (pulverized fuel) combustion, in dry bottom boilers a minimum IT of 1200°C and in wet bottom boilers a maximum of 1300°C are recommended.

### 4.3.2.3 Caking tests

***Free swelling index***

The free swelling index (FSI) in BSI nomenclature (the crucible swelling number (CSN) in ISO nomenclature) is a measure of increase in the volume of coal when heated, without the restriction 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 (Figure 4.12). Coals with a low swelling index (0–2) are not suitable for coke manufacture. Coals with high swelling numbers (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.

***Roga index test***

The Roga index 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 Roga index values and free swelling index values. For example, the Roga index values 0–5 are equivalent to the free swelling index values 0–$1/2$, values 5–20 = 1–2, values 20–45 = $2\,1/2$–4 and values > 45 = > 4.

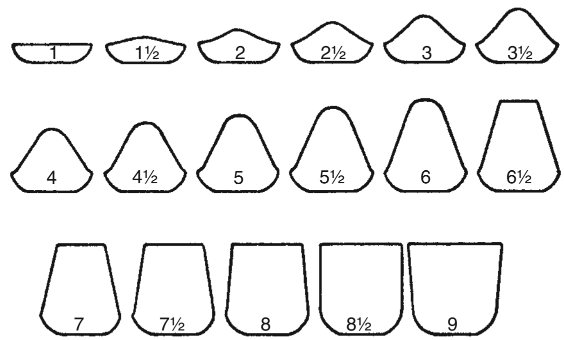



**Figure 4.12** Characteristic profiles of coke buttons for different values of the crucible swelling number (free swelling index). (From BS 1016-107.1 (1991). Reproduced with permission of BSI under licence number 2002SK/0003.)

### 4.3.2.4 Coking Tests

***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, 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 free swelling indexes as follows, Gray–King coke type A–B is equivalent to the free swelling index value 0–$1/2$, C-G2 = 1–4, F–G4 = $4\,1/2$–6, G3–G6 = $6\,1/2$–8 and G7 or above = $8\,1/2$–9.

***Fischer assay***

This test is most widely used for testing low-rank coals to low-temperature carbonization. The percentages of coke, tar and water driven off by the dry coal are determined, and gas is calculated by subtraction.

***Gieseler plastometer***

To form coke, coal passes from a solid form through a fluid or plastic state to become a fused porous solid. The

**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 | | |
| A | B | C | D | E | F | G | G1 | G2 | Gx |
| 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, however, that 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 or three 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. |

Source: BS 1016-107.2 1991. Reproduced with permission of BSI under licence Number 2002SK/0003.)

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 capable of measuring to only 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.

***Audibert–Arnu dilatometer***

Coals shrink during carbonization, 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 Ruhr dilatometers. Dimensional changes in a coal can be measured as functions 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.

***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 test is widely used in China, Russia and other Eastern European countries. The test was known as the Sapozhnikov Plastometric Test and first developed by Sapozhnikov and Bazilevich (1938).

***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

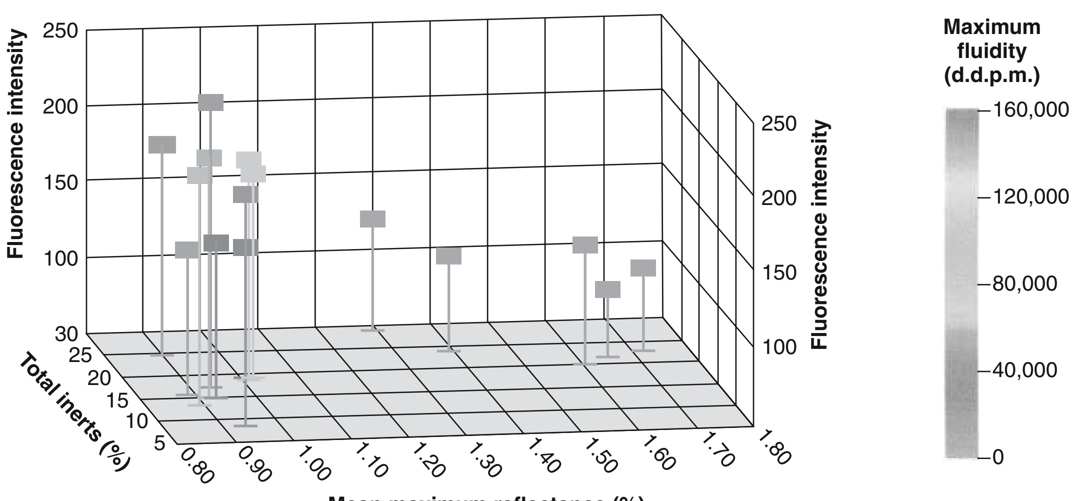



**Figure 4.13** Relationship between coal rank, coal type and fluorescence. (Pearson 2011). (Reproduced with permission from Pearson Coal Petrography Inc.) This figure is reproduced in colour in the Plates section.

the determination of several physical properties. These are the coal's density, hardness, grindability, abrasiveness, size distribution and float–sink tests.

*4.3.3.1 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.

*4.3.3.2 Hardness and grindability*

In modern commercial operations, coals are required to be crushed to a fine powder (pulverized) before being fed into a boiler. The relative ease with which a coal can be pulverized 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 pulverized 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 pulverized product. The HGI varies with coal rank as shown in Figure 4.14.

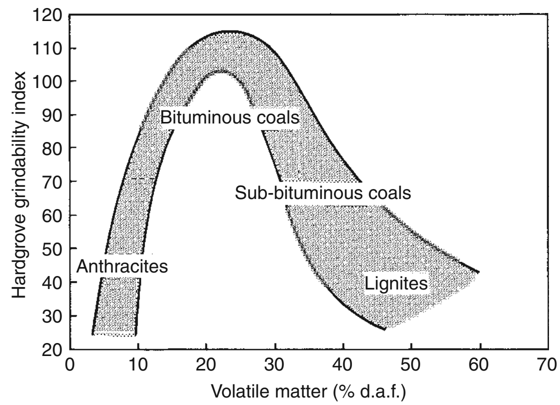



**Figure 4.14** Generalized variation of the Hardgrove grindability index with rank. (From Berkowitz, 1979.)

*4.3.3.3 Abrasion index*

Coarse mineral matter in coal, particularly quartz, can cause serious abrasion of machinery used to pulverize coal. Coal samples are tested in a mill equipped with four metal blades. The loss in mass of these blades determines the 'abrasion index', and is expressed as milligrams of metal per kilogram of coal used.

*4.3.3.4 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.

*4.3.3.5 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, and so 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 4.20. Using the results given in Table 4.20, 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 classic washability curves, i.e. 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 curve which plots column L against column J, and the elementary ash curve which plots column G against

**Table 4.20** Washability data.

| Relative density | | | Fractional | | | Cumulative floats | | | Cumulative sinks | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| A<br>Sink RD | B<br>Float RD | C<br>Midpoint RD | D<br>Weight % | E<br>Ash % | F<br>Ash points | G<br>Weight % | H<br>Ash points | I<br>Ash % | J<br>Weight % | K<br>Ash points | L<br>Ash % |
| | | $\frac{A+B}{2}$ | | | $\frac{D \times E}{100}$ | $\sum D \downarrow$ | $\sum F \downarrow$ | $\frac{H}{G} \times 100$ | $\sum D \uparrow$ | $\sum F \uparrow$ | $\frac{K}{J} \times 100$ |
| – | 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 S.C. Frankland.

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. Classic washability curves cannot be used to calculate yield or quality of middlings, and 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 4.18). The angle of lines drawn from point A to intersect the abscissa represent 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 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 classic 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 calorific value is lowered, and caking is eliminated. There is also a loss of floatability during washing of the coal.

**Table 4.21** Parameters used in Seyler's coal classification.

| Genus | Hydrogen (%) | Class (% carbon) | | | | | | |
|---|---|---|---|---|---|---|---|---|
| | | Anthracite | Carbonaceous | Bituminous | | | Lignitous | |
| | | (>93.3) | (93.3–91.2) | 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<br>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<br>True anthracite | Pseudo-anthracite (sub-carbonaceous) | Pseudo-anthracite (sub-meta-bituminous) | Pseudo-anthracite (sub-ortho-bituminous) | Pseudo-anthracite (sub-para-bituminous) | | |

Source: Ward (1984), with permission of Blackwell Scientific Publications.

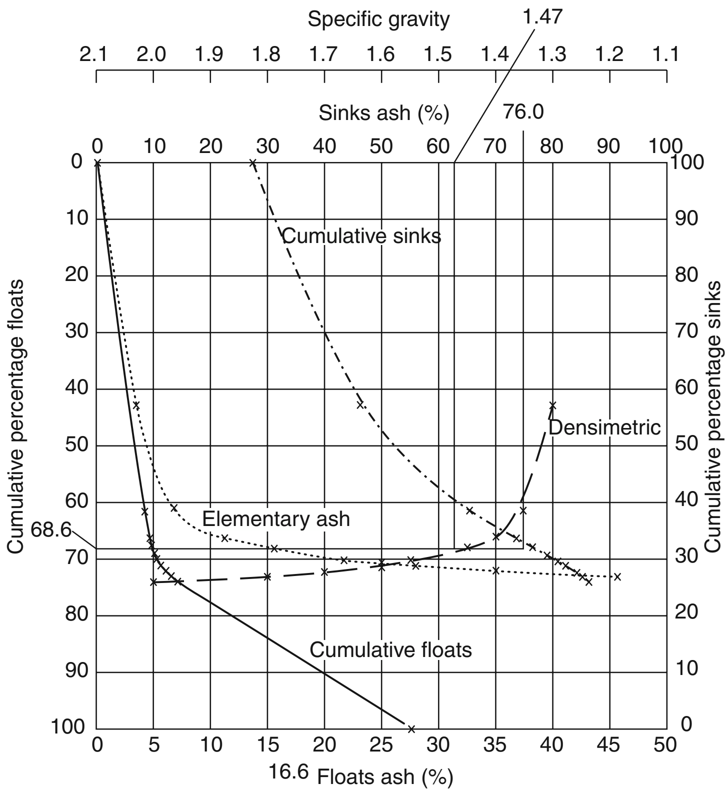



Figure 4.15 Washability curves based on data given in Table 4.18. (Reproduced by courtesy of S. C. Frankland.)

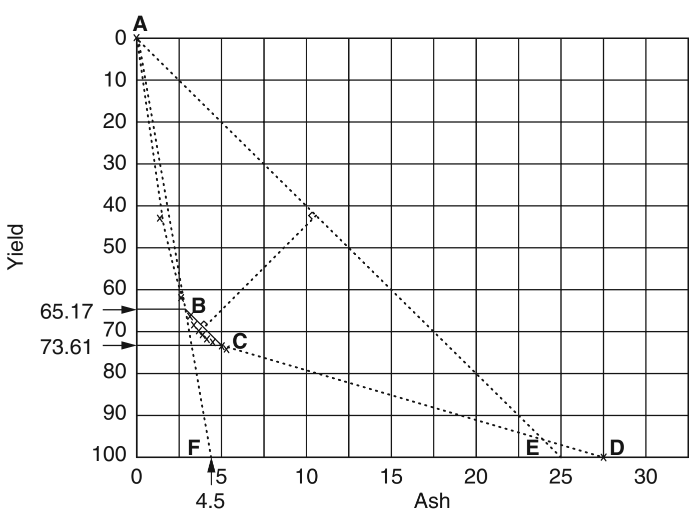



Figure 4.16 Middlings or M curve. (Reproduced by courtesy of S. C. Frankland.)

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 oxidized 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 oxidize 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 oxidized. The oxidized 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°C rise in temperature at least up to 100°C. It has also been demonstrated that low-rank coal produces heat when wetted, and that if dispersed pyrite is present, reactivity is increased tenfold.

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. The following factors contribute to the possibility of combustion: if the

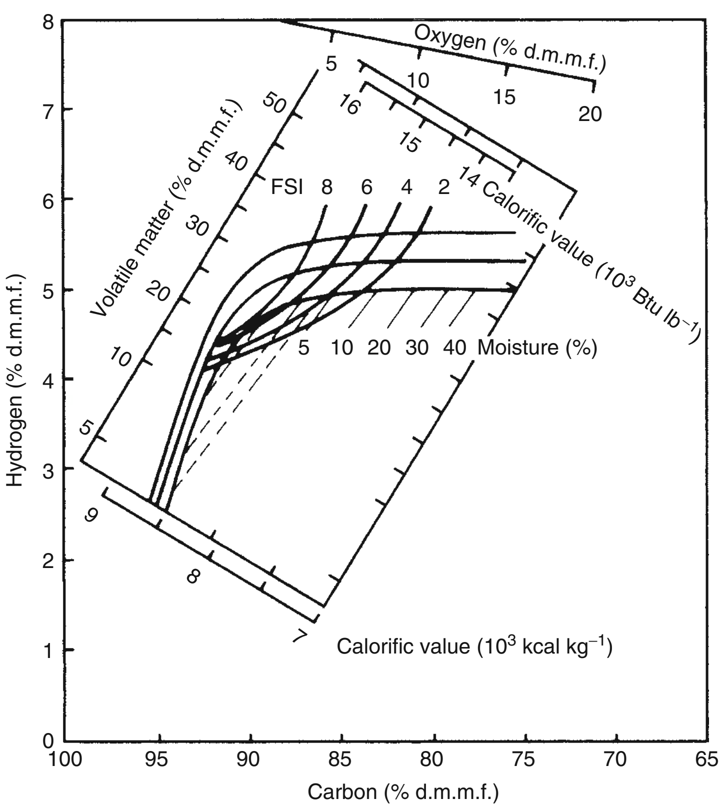



**Figure 4.17** Seyler's coal chart. This version shows relationships between elemental composition, volatile matter contents, moisture contents and caking properties. (From Berkowitz (1979), based on Seyler (1931, 1938).)

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 minimize 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 the coal's 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 range from the percentage of fixed carbon and volatile matter (on a dry mineral matter free basis), calorific value (on a moist mineral matter free basis), the caking properties of coal (FSI and Roga index) 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 (Figure 4.17). This classification is applicable, however, 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 4.21. The following principal commercial classifications of coal are in current use.

### *4.4.1 North America*

The ASTM (American Society for Testing and Materials) classification (D 388-99) is used on a worldwide basis (Appendix 1). This is based on two coal properties, the fixed carbon values and the calorific values (on d.m.m.f. basis). The higher rank coals are classified according to fixed carbon on the dry basis, the lower rank coals are classified according to gross calorific value on the moist basis. A correlation of the ranks property and volatile matter with the mean maximum reflectance group of macerals (D 2798-09) is used as supplemental information. Further classification is given for those coals with agglomerating or coking properties – see Table 4.22. 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 (1964) before privatization and is shown in Table 4.23. It uses a three-figure numeric code to classify bituminous and anthracite coals. The first two digits are based on the amount of volatile matter in the coal (on d.m.m.f. basis) and the third digit is based on the Gray–King assay value. Coals with less than 19.6% volatile matter (d.m.m.f.) 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 oxidized by weathering may be distinguished by adding the suffix W to the coal code.

### *4.4.3 Europe*

In Europe the Codification System for medium- and high-rank coals was published by the United Nations Economic Commission for Europe (UNECE, 1988) and approved by the International Organization for Standardization (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 calorific value <24.0 MJ kg$^{-}$ (5700 Kcal kg$^{-1}$) and a mean random reflectance ($\overline{R}_r$) < 0.6% (Figure 4.18).

The Codification System is applicable to both run-of-mine and washed coals. The coals are characterized by using a 14 digit code number based on eight property-related parameters providing information concerning rank, type and grade of a coal (Table 4.24). In addition a list of 'Supplementary Parameters' is used where appropriate, for example chlorine content and ash fusion temperature. The code is based on:

**Table 4.22** ASTM classification of coals by rank [*].

| Class/Group | Fixed carbon limits (dry, mineral-matter-free basis), % | | Volatile matter limits (dry, mineral-matter-free basis), % | | Gross calorific value limits (moist,[†] mineral-matter-free basis) | | | | Agglomerating character |
|---|---|---|---|---|---|---|---|---|---|
| | | | | | Btu $lb^{-1}$ | | Mj $kg^{-1}$[‡] | | |
| | Equal or greater than | Less than | Greater than | Equal or less than | Equal or greater than | Less than | Equal or greater than | Less than | |
| Anthracitic: | | | | | | | | | |
| Meta-anthracite | 98 | – | – | 2 | – | – | – | – | Non-agglomerating |
| Anthracite | 92 | 98 | 2 | 8 | – | – | – | – | |
| Semianthracite[§] | 86 | 92 | 8 | 14 | – | – | – | – | |
| Bituminous: | | | | | | | | | |
| Low volatile bituminous coal | 78 | 86 | 14 | 22 | – | – | – | – | |
| Medium volatile bituminous coal | 69 | 78 | 22 | 31 | – | – | – | – | Commonly agglomerating[**] |
| High volatile *A* bituminous coal | – | 69 | 31 | – | 14,000[¶] | – | 32.6 | – | |
| High volatile *B* bituminous coal | – | – | – | – | 13,000[¶] | 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 | Agglomerating |
| Subbituminous: | | | | | | | | | |
| 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 | |
| Lignitic: | | | | | | | | | |
| Lignite *A* | – | – | – | – | 6,300[††] | 8,300 | 14.7 | 19.3 | |
| Lignite *B* | – | – | – | – | – | 6,300 | – | 14.7 | |

*This classification does not apply to certain coals.
[†]Moist refers to coal containing its natural inherent moisture but not including visible water on the surface of the coal.
[‡]Megajoules per kilogram. To convert British thermal units per pound to megajoules per kilogram, multiply by 0.002326.
[§]If agglomerating, classify in low volatile group of the bituminous class.
[¶]Coals having 69% or more fixed carbon on the dry, mineral-matter-free basis shall be classified according to fixed carbon, regardless of gross calorific value.
**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[‡] bituminous group.
[††]Editorially corrected.
Soruce: reprinted with permission from ASTM D388–1999.

**Table 4.23** Coal classification system used by British Coal (revision of 1964).

| Coal rank code | | | Volatile matter (d.m.m.f.) (%) | Gray–King coke type* | General description |
|---|---|---|---|---|---|
| Main class(es) | Class | Subclass | | | |
| 100 | | | Under 9.1 | A | |
| | 101[†] | | Under 6.1 | A | Anthracites |
| | 102[†] | | 6.1–9.0 | | |
| 200 | | | 9.1–19.5 | A–G8 | Low Volatile steam coals |
| | 201 | | 9.1–13.5 | A–C | Dry steam coals |
| | | 201a | 9.1–11.5 | A–B | |
| | | 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 | Prime coking coals |
| | | 301a | 19.6–27.5 | G4 and over | |
| | | 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 | Medium volatile, weakly caking to non-caking coals |
| 400–900 | | | Over 32.0 | A–G9 and over | High volatile coals |
| 400 | | | Over 32.0 | G9 and over | High volatile, very strongly caking coals |
| | 401 | | 32.1–36.0 | G9 and over | |
| | 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 | | |
| 900 | | | Over 32.0 | A–B | High volatile, noncaking coals |
| | 901 | | 32.1–36.0 | A–B | |
| | 902 | | Over 36.0 | | |

*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.

[†]To divide anthracites into two classes, it is sometimes convenient to use a hydrogen content of 3.35% (d.m.m.f.) 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 recognize more than two classes.

*Notes:*

**1**. Coals that have been affected by igneous intrusions ('heat-altered' coals) occur mainly in classes 100, 200 and 300, and when recognized should be distinguished by adding the suffix H to the coal rank code, e.g. 102H, 201bH.

**2**. Coals that have been oxidized by weathering may occur in any class and when recognized should be distinguished by adding the suffix W to the coal rank code, e.g. 801W.

Source: reproduced by permission of the British Coal Corporation.

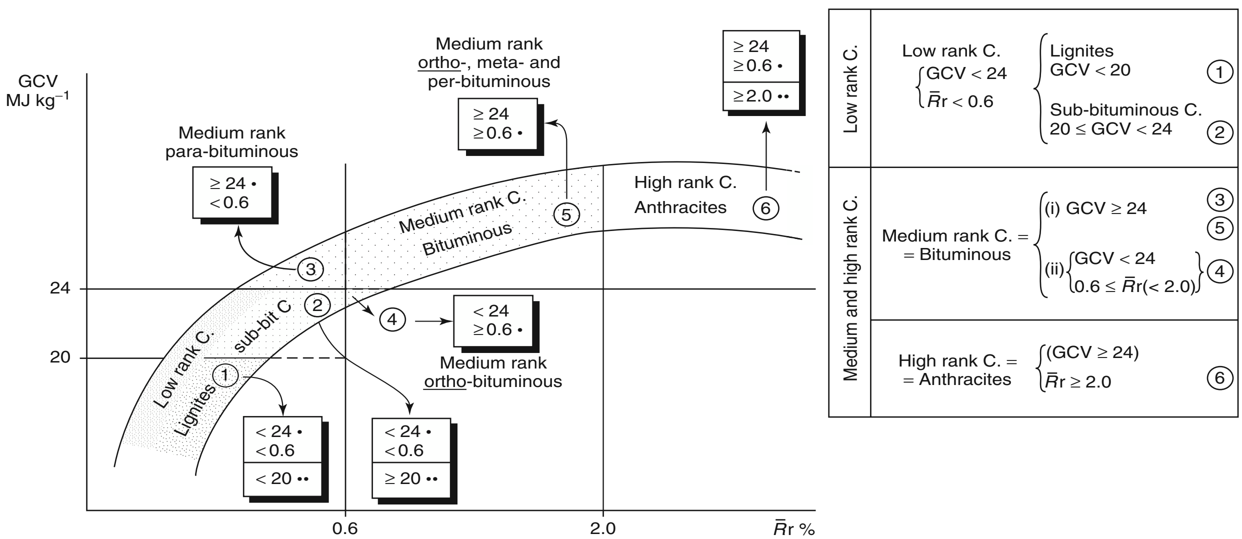



**Figure 4.18** The UNECE main coal classification categories, defined by gross calorific value (MJ $kg^{-1}$ m, a.f.) and vitrinite reflectance % in oil ($R_{ro}$%). (Reproduced with permission of the United Nations Economic Commission for Europe.)

**Table 4.24** Codification system used by UNECE for medium and high rank coals (1988).

| | Random reflectance of vitrinite % | | Characte-ristics of reflec-togram | Maceral group composition index % by vol. (mmF) 4 = Inertinite; 5 = Liptinite | | | | Swelling crucible number | | Volatile matter mass % (d.a.f.) | | Ash mass % (db) | | Total sulfur mass % (db) | | Gross calorific value MJ kg$^{-1}$ (d.a.f.) | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Digit | 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 | 0–½ | 48 | ≥48 | 00 | 0–<1 | 00 | 0.0–<0.1 | 21 | <22 |
| | 03 | 0.30–0.39 | 1 | 1 | 10–<20 | 1 | 0–<5 | 1 | 1–1½ | 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 | 2–2½ | 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 | 3–3½ | 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 | 4–4½ | 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 | 5–5½ | 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 | 6–6½ | 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 | 7–7½ | 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 | 8–8½ | 32 | 32–<34 | 08 | 8–<9 | 08 | 0.8–<0.9 | 29 | 29–<30 |
| | 11 | 1.10–1.19 | | 9 | ≥90 | 9 | ≥40 | 9 | 9–9½ | 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 | ≥39 |
| | 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 | | | | | | | | | | | | | | | |

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1. mean random reflectance ($\overline{R}_r$) as two digits – codes 2–50 cover $\overline{R}_r$ 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 indicates the upper limit of a 5% range of the liptinite content;
4. the sixth digit indicates the crucible swelling number in terms of the lower limit of two ½ step numbers;
5. the seventh and eighth digits indicate the lower limit of a 2% range of volatile matter down to 10% (d.a.f.) and a 1% range when volatile matter <10%;
6. the ninth and tenth digits indicate the lower limit of 1% range of the ash (d.b.);
7. the eleventh and twelfth digits indicate the lower limit of a 0.10% range of the total sulfur content (d.b.) multiplied by 10;
8. the thirteenth and fourteenth digits indicate the lower limit of 1 MJ kg$^{-1}$ range of the gross calorific value (d.a.f.).

All of the testing is in accordance with ISO Standards as listed in Appendix 1. An example of the Codification System when applied to an Australian coal is:

| | | Digit number |
|---|---|---|
| $\overline{R}_r$ | 1.25 | 12 |
| Reflectogram | $s = 0.14$, no gap | 1 |
| Maceral composition: | | |
| inertinite | 51 | |
| liptinite | 3 | 51 |
| FSI | 6½ | 6 |
| Volatile matter (% d.a.f.) | 24.3 | 24 |
| Ash content (% d.b.) | 5.53 | 05 |
| Total sulfur content (% d.b.) | 0.42 | 04 |
| Gross CV (MJ kg$^{-1}$ d.a.f.) | 35.9 | 35 |
| Code number | 12 1 51 6 24 05 04 35 | |

**Table 4.25** Australian classification of hard coal.

| 1st digit (coal class) | | | 2nd digit (coal group) | | 3rd digit (coal subgroup) | | 4th digit (ash number) | |
|---|---|---|---|---|---|---|---|---|
| Value | Volatile matter (d.m.m.f.) (%) | Gross calorific value (d.a.f.) (MJ kg$^{-1}$) | Value | Crucible swelling number | Value | Gray–King coke type | Value | Ash (dry basis %) |
| 1 | 10.0 | | 0 | 0–½ | 0 | A | (0) | 4.0 |
| 2 | 10.1–14.0 | | 1 | 1–2 | 1 | B–D | (1) | 4.1–8.0 |
| 3 | 14.1–20.0 | | 2 | 2½–4 | 2 | E–G | (2) | 8.1–12.0 |
| 4A | 20.1–24.0 | | 3 | 4½–6 | 3 | G1–G4 | (3) | 12.1–16.0 |
| 4B | 24.1–28.0 | | 4 | 6½–9 | 4 | G5–G8 | (4) | 16.1–20.0 |
| 5 | 28.1–33.0 | | | | 5 | G9– | (5) | 20.1–24.0 |
| 6 | 33–41* | >33.82 | | | | | | |
| 7 | 33–44* | 32.02–33.82 | | | | | | |
| 8 | 35–50* | 28.43–32.02 | | | | | | |
| 9 | 42–50* | 27.08–28.42 | | | | | | |

Source: Ward (1984), based on Australian Standard 2096 (1987).
*Values for information only.

**Table 4.26** Classification of low-rank coals.

| Coal type | Gross CV MJ $kg^{-1}$ (m, a.f.) | Moisture (% a.r.) | $R_{ro}$% |
|---|---|---|---|
| Low rank C (Ortholignites) | 15.0 | <75 | – |
| Low rank B (Meta-lignites) | 15–20 | – | – |
| Low rank A (Subbituminous) | 20–24 | – | <0.6 |

Source: UNECE (2000), reproduced with permission of UNECE.

#### 4.4.4 Australia

The Australian Standard Coal Classification for hard coals again assigns a multidigit number to determine coal type. The first digit represents volatile matter for coals with less than 33% volatile matter (d.m.m.f.) and gross calorific value (d.a.f.) 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 (dry basis) of the coal, see Table 4.25.

#### 4.4.5 South africa

In South Africa, coals are divided for commercial purposes into three broad classes on the basis of volatile matter (d.a.f.). These are South African anthracite, semi-anthracite and steam coal. Coals of each class are graded on the basis of calorific value (a.d.), ash (a.d.) 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 United Nations Economic Commission for Europe decided to abandon the term 'brown' coal, and devised a new Classification and Codification System for low-rank coals (UNECE, 2000) – see Tables 4.26 and 4.27.

**Table 4.27** Codification system used by UNECE for low rank coals (2000).

| Digit 1, 2 Code no. | Gross calorific value MJ $kg^{-1}$ (d.a.f.) | 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 | 15.00–15.98 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: UNECE (2000), reproduced with permission of UNECE.

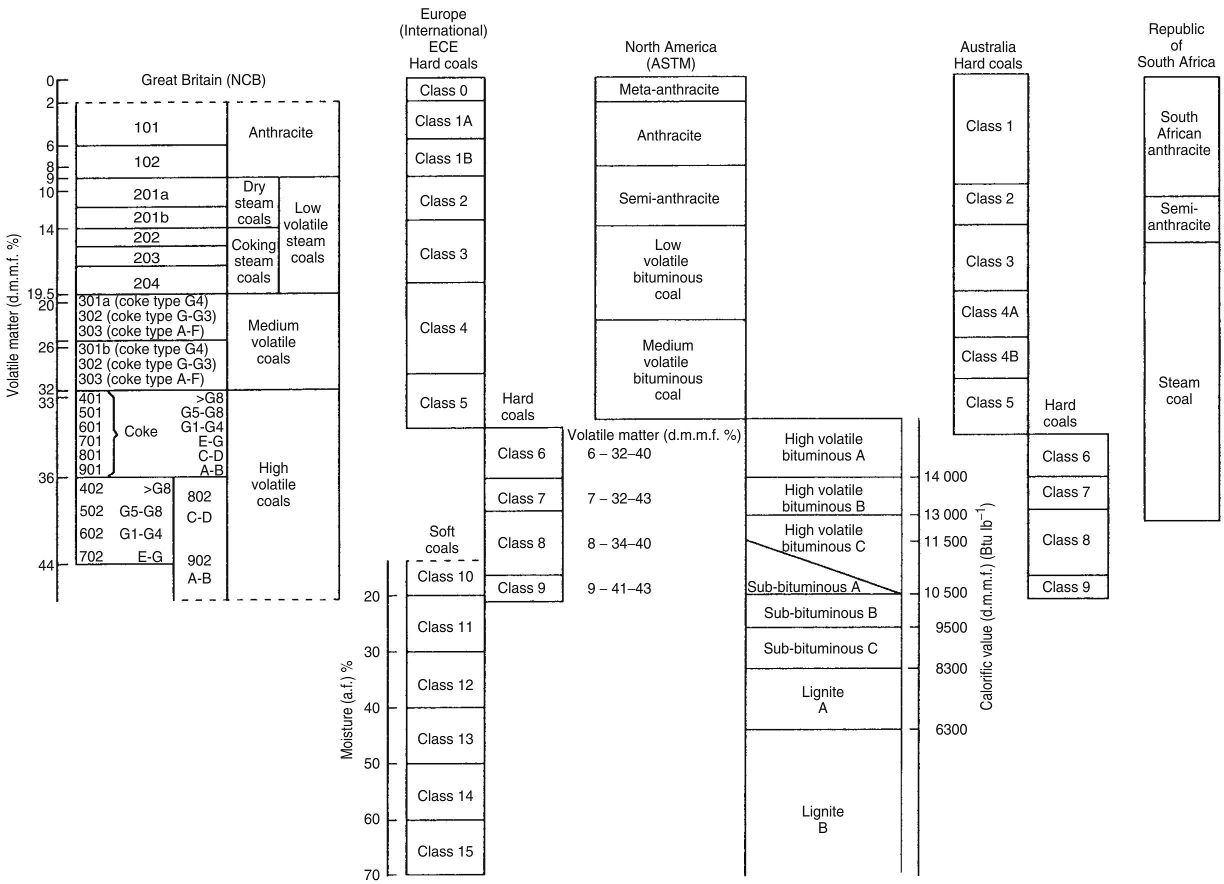



**Figure 4.19** Interrelationships of coal classification systems used in various countries. (Unpublished data, reproduced by permission of BP Coal Ltd.)

**Table 4.28** Classification of Russian coals.

| Letter code | Name of mark | Volatile matter (% d.a.f.) | Vitrinite reflectance (Ro %) | Plastic layer (*y*) (mm) | Moisture (% a.r.) | Gross calorific value (Mj kg$^{-1}$ wet ash-free basis) |
|---|---|---|---|---|---|---|
| B | Brown coal | 10–>48 | <0.6 | – | 1B>40<br>2B 30–40<br>3B < 30 | <24 |
| 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 | – | | |

Source: Russian Coal (2010).

As stated above, the threshold between low-rank and medium- and high-rank coals is defined as: low-rank coals are those coals with a gross calorific value less than 24.0 MJ kg$^{-1}$ (moist, a.f. or m, a.f.) and a vitrinite reflectance in oil ($R_{ro}$%) less than 0.6%. The gross calorific value calculation uses total moisture according to ISO 1015–1975, and low-rank coals are classified as shown in Figure 4.18.

As for medium- and high-rank coals, the UNECE have devised a codification system for low-rank coals based on an eight digit number.

1. Digits one and two, the gross calorific value in MJ kg$^{-1}$ (m, a.f.).
2. Digits three and four, the total moisture (%) recalculated to as received basis (a.r.).
3. Digits five and six, the ash content (%) recalculated to dry basis (d.b.).
4. Digits seven and eight, the total sulfur content (%) recalculated to dry basis (d.b.).

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 is carried out. These supplementary parameters consist of the numerous chemical and physical tests outlined above and which are also used for the medium- and high-rank coals. The inter-relationships 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) 25,543-88 is based upon the degree of metamorphism, which is defined according to the mean value of vitrinite reflectance index ($R_o$).

**Table 4.29** Approximate coal rank relationships between GOST (Russian) and ASTM (USA) coal classifications.

| FSU, GOST 25543-88 | | | USA, ASTM 380-98a | |
|---|---|---|---|---|
| Brown Coals | Brown (B) | 1B | ligB | Lignite |
| | | | ligA | |
| | | 2B | subC | Subbituminous |
| | | 3B | subB | |
| Hard Coals | Long-Flame (D) | | sub A | |
| | | | hvCb | Bituminous |
| | Long-Flame-Gas (DG), Gas (G), Gas-Fat (GZh), Gas-Fat-Mearge (GZhO), and part Fat (Zh) | | hvBb | |
| | | | 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 | |
| | Anthracites (A) | A1 | | Anthracitic |
| | | A2 | | |
| | | A3 | ma | |

Source: US Geological Survey Open File Report (2011).

Another principal parameter used is volatile matter on the dry ash-free basis. The Russian system uses the term mark of coal rather than rank, although these are not considered entirely synonymous. In addition, calorific value is used for the classification of coals by types, that is brown coal, hard coal and anthracite. Other parameters are used to characterize caking properties, namely the thickness of the plastic layer ($y$), free swelling index (SI) and Roga index (RI). Coals are divided into seventeen marks, each mark is encoded with the latin equivalent letter of the Russian letter which represents the full name of the coal mark (Table 4.28).

These coal marks have been approximately compared with the ASTM D388-99 standard coal classification (Table 4.22) and is shown in Table 4.29. Only the Russian mark SS within the GOST standard 25,543-88 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 China

The People's Republic of China uses a not dissimilar system to the Russian. The Chinese system is based on volatile matter, and 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 subclasses respectively.

**Table 4.30** Classification of Chinese coals.

| Symbol | Coal type (abbreviated) | Volatile matter (% d.a.f.) | Caking index (GRL) | Plasic layer (*y*) (mm) | Dilatometer (*b*) | Moisture (%) | Gross calorific value (Mj kg$^{-1}$ m.a.f.) |
|---|---|---|---|---|---|---|---|
| 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 | >50–65 | >25 | (<150) | | |
| | | >10–20 | >65 | | | | |
| 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 | >50–65 | <25 | <220 | | |
| | | 37 | >35–65 | | | | |
| 1/2ZN | 1/2Middle Sticky coal | >20–37 | >30–50 | | | | |
| RN | Weak-sticky Coal | >20–37 | >5–30 | | | | |
| BN | Unsticky coal | >20–37 | <5 | | | | |
| CY | Long flame coal | >37 | <5–35 | | | | |
| HM | Brown coal | (1) >37 | | | | <30 | 24 |
| | | (2) >37 | | | | >30–50 | |

Source: China Shen Zhou Mining and Resources (2007)

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 4.30 (Shen Zhou Mining and Resources 2007).

# 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 that are acquired for testing and analysis in order to assess the nature and composition of the parent body. They are 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 greenfield 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.

Sampling may have to be undertaken in 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 mineralization, and coals in close contact with major faults and igneous intrusions.

## 5.2 *In situ* sampling

Several types of *in situ* samples can be taken, dependent upon the analysis required.

### 5.2.1 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 can easily lead to a bias in selection, for example the bright coal sections attract attention. However, grab samples can be used to determine vitrinite reflectance measurements, as an indicator of coal rank.

### 5.2.2 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), it may be possible to see only sections of the roof and coal immediately below, or the floor and coal immediately above. To obtain a full seam section under these circumstances, two or more overlapping 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, and 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.

*Coal Geology*, Second Edition. Larry Thomas.

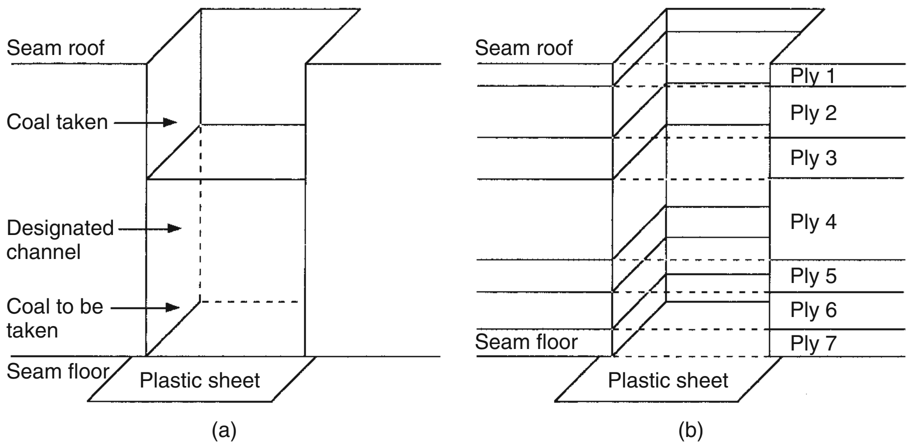



**Figure 5.1** Channel sampling procedure. (a) Whole-seam channel sampling and (b) coal seam ply channel sampling.

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 $m^{-1}$ of coal thickness. Such channel samples will provide a composite quality analysis for the seam, that is an analysis of all the coal and mineral matter present in the seam as a whole. Although 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, which 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.

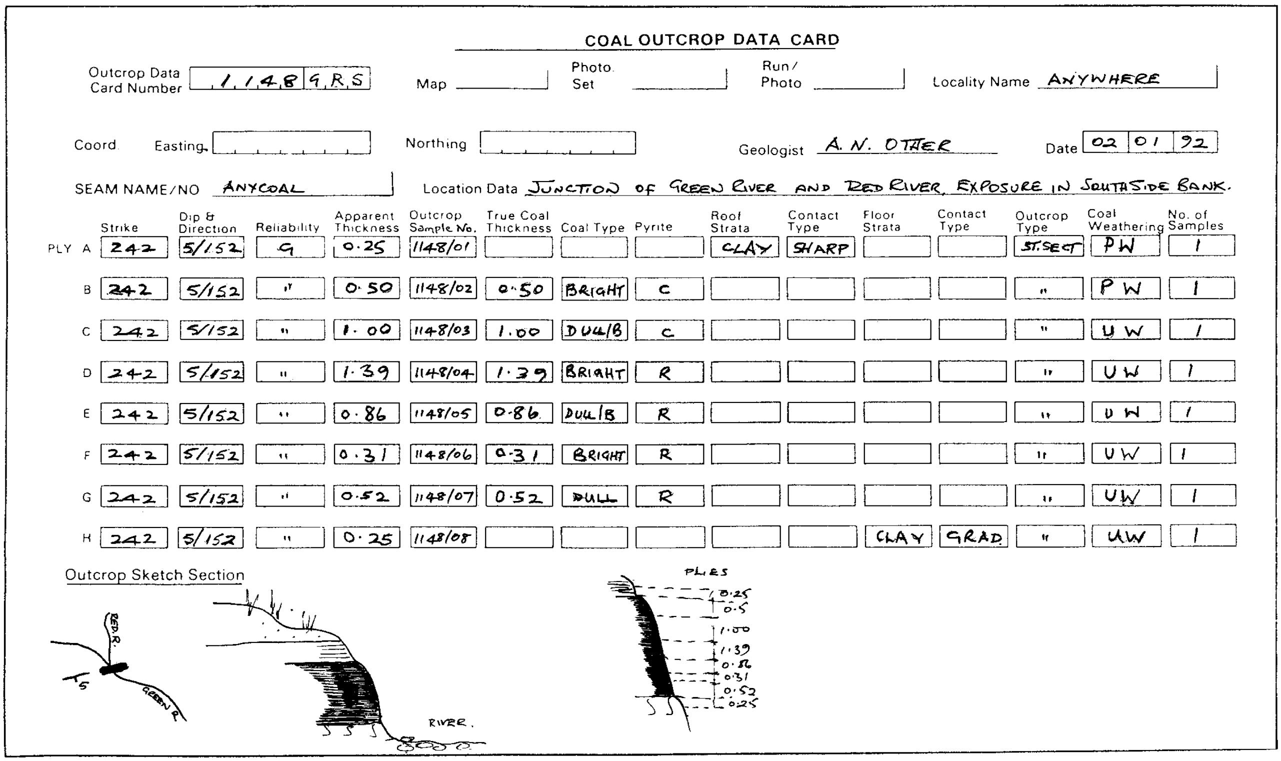
Coal seams are rarely homogeneous throughout their thickness, most are divisable into distinct lithological sections. Plies are lithological subdivisions of the seam, each of which has a uniform character. When the lithology changes, such as at a clay parting in the seam, a separate ply is designated. 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 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 the 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.

1. Record card number.
2. Map or aerial photograph number on which locality is located.
3. Location of sample point, grid reference or reference number.
4. Description of the locality, stream section, working face, etc., including dip, strike, coal seam roof and floor contacts.
5. Extent of weathering, fracturing, mineralization, etc.
6. Lithological description of each ply interval.
7. Thickness of each ply interval.
8. Designated sample number of each ply interval.

Space can also be allocated on the record card for analytical details, that is 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



COAL OUTCROP DATA CARD

Outcrop Data Card Number 1,1,4,8 | 9,R,S    Map ______    Photo Set ______    Run/Photo ______    Locality Name ANYWHERE

Coord. Easting ______    Northing ______    Geologist A. N. OTHER    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 | 5/152 | G | 0.25 | 1148/01 | | | | CLAY | SHARP | | | ST.SECT | PW | 1 |
| B | 242 | 5/152 | " | 0.50 | 1148/02 | 0.50 | BRIGHT | C | | | | | " | PW | 1 |
| C | 242 | 5/152 | " | 1.00 | 1148/03 | 1.00 | DULL/B | C | | | | | " | UW | 1 |
| D | 242 | 5/152 | " | 1.39 | 1148/04 | 1.39 | BRIGHT | R | | | | | " | UW | 1 |
| E | 242 | 5/152 | " | 0.86 | 1148/05 | 0.86 | DULL/B | R | | | | | " | UW | 1 |
| F | 242 | 5/152 | " | 0.31 | 1148/06 | 0.31 | BRIGHT | R | | | | | " | UW | 1 |
| G | 242 | 5/152 | " | 0.52 | 1148/07 | 0.52 | DULL | R | | | | | " | UW | 1 |
| H | 242 | 5/152 | " | 0.25 | 1148/08 | | | | | | CLAY | GRAD | " | UW | 1 |

Outcrop Sketch Section



**Figure 5.2** Coal outcrop data card. (Reproduced by permission of BP Coal Ltd.)

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, so it is essential therefore that care must be taken to ensure that the sample numbers do not get lost or obliterated during transit to 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 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 (ca. 0.25 m wide) cut from the fresh coal from which ply samples are collected for analysis.

### 5.2.3 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 undertaken only in special circumstances, such as when mining becomes difficult or new roadways or faces are planned.

### 5.2.4 Core samples

Core sampling is an integral part of coal exploration and mine development. It has the advantage of producing non-weathered coal including the coal seam floor and roof, and unlike channel samples, core samples preserve the lithological sequence within the coal seam.

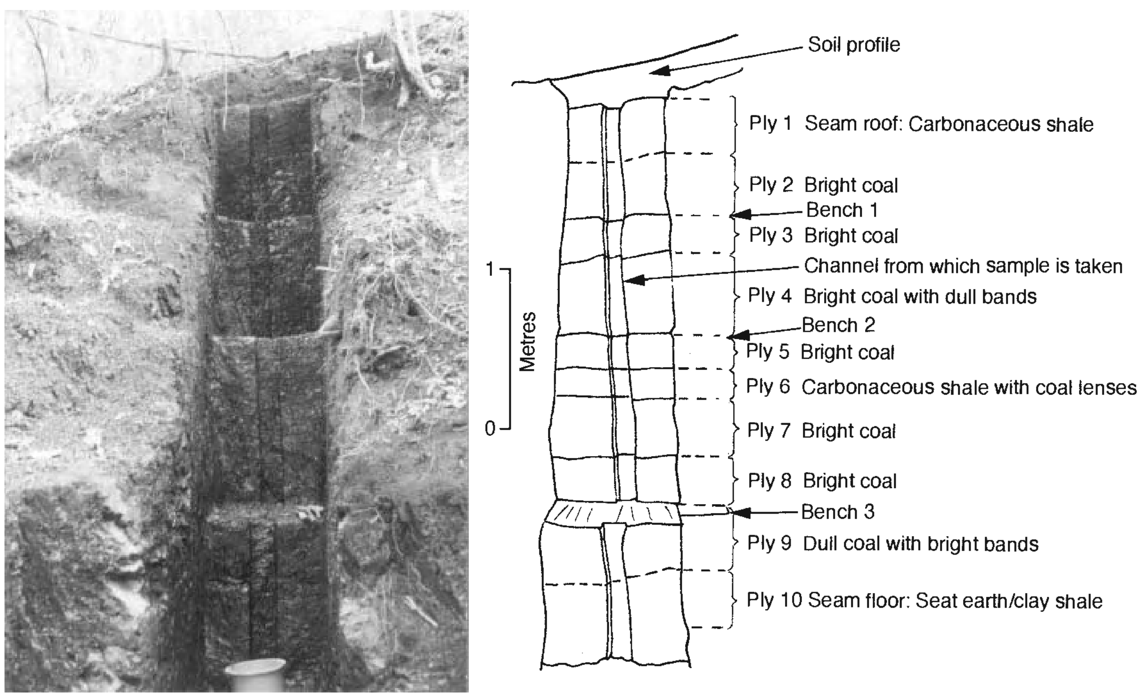



**Figure 5.3** Surface coal ply channel sample taken in shallow dipping seam. Central narrow channel taken for ply sample analysis, including coal seam roof and floor. (Photograph by LPT.)

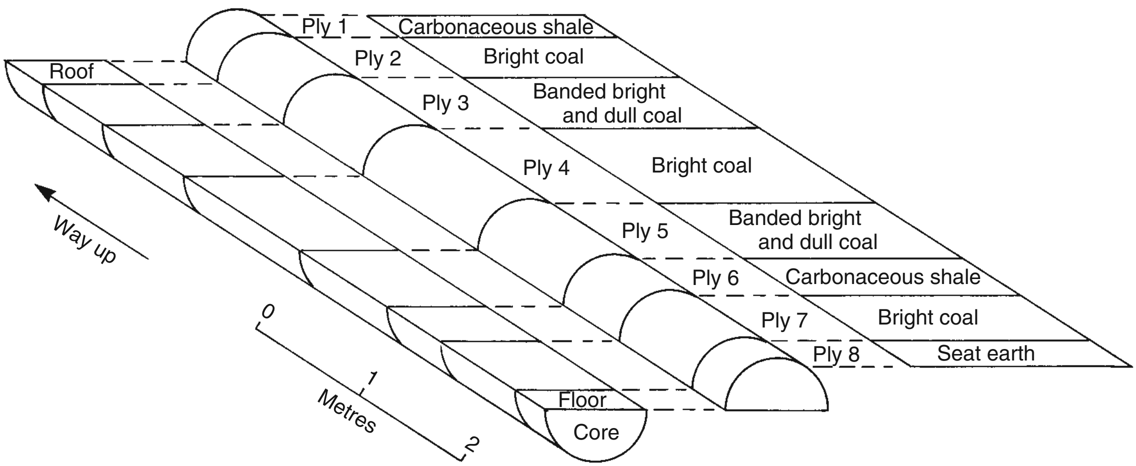



**Figure 5.4** Ply sampling of borehole core; run of samples to include all the coal seam and roof and floor. Core may be split in this fashion and one half kept for future examination and analysis.

First, the borehole core has to be cleaned if drilling fluids have been used, and then lithologically logged. Following this, the lithological 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 later in Figure 6.20).

Large diameter cores may be split lengthways with a bolster chisel and then one of the halves ply sampled, the other being retained for future analysis.

### 5.2.5 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. Contamination from strata above the coal also may be included, and a close study of the geolog will determine whether this is so.

### 5.2.6 Specimen samples

Orientated specimens of coal may be collected so that their precise orientation can be re-created in the laboratory. The dip and strike of the coal is marked on the specimen before removal. This method is commonly used for studies of the optical fabric of the coal, or of the structural features in the coal.

### 5.2.7 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, which is designed to indicate the coal's likely performance under actual conditions of usage.

Steam coal samples are taken for small combustion tests in a pulverized fuel (pf) rig, to simulate conditions in a PF boiler. Pulverized 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 coal samples are taken to carry out moving wall oven tests, that is to determine how much the coal swells when it 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 United States, low-volatile coking coals (volatile matter (VM) = 20–25%, SI = 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.

### *5.2.8 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 because the exposure of the coal to air will allow oxidation to take place during storage and this 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 pressurized cylinder containing nitrogen in a plastic sample bag, then add the coal sample, 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 lighter than air so inevitably some is 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 *ex 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:

$$5.24 \times \text{mean particle size} = x\,\text{kg}$$

(where mean particle size is top size × bottom size).

5.24 is an empirically determined number quoted in BS1017-1 and Australian Standard 4264 (Appendix 1).

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 1–2 yr before analysis.

## 5.3 *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 seam roof and/or floor contamination 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). Increments are taken using three methods.

1. **Systematic sampling**, where increments are spaced evenly in time or in position over the unit.
2. **Random sampling**, where increments are spaced at random but a prerequisite number are taken.
3. **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.

1. 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, where moving streams of coal (conveyors) are sampled by:
   (a) **falling stream samplers**, which 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;
   (b) **cross-belt samplers**, which move across the belt pushing a section of coal to the side while the belt runs;
   (c) the **stop belt method**, whereby the conveyor is stopped and all coal occurring within a selected interval, usually a couple of metres, is collected (Figure 5.5).

   Laurila and Corriveau (1995) state 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 BSISO 13909 or ASTM D2234 (Appendix 1).
2. 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 tipplers.
3. 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.
4. 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 onboard during unloading, once the bottom of the barge is partially uncovered.
5. When 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 can be speeded up if automated auger units are employed. It is important that all levels in the stockpile are sampled.

**Figure 5.5** The collection of a stop belt sample from a main conveyor. (From Mazzone 1998.)

**Table 5.1** Minimum number of increments required for gross samples of a single coal consignment up to 1000 t.

| 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 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: Osborne (1988) '*Coal Preparation Technology*' published by Graham and Trotman, with kind permission of Kluwer Academic Publishers.

Table 5.1 indicates the minimum number of increments required for gross samples of single load consignment up to 1000 t, from all of the above.

## 5.4 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 specialized purpose and have set restricted specifications for the coal. The coal producer, that is the mining company, will have assessed the potential market before developing any coal deposit, that is 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

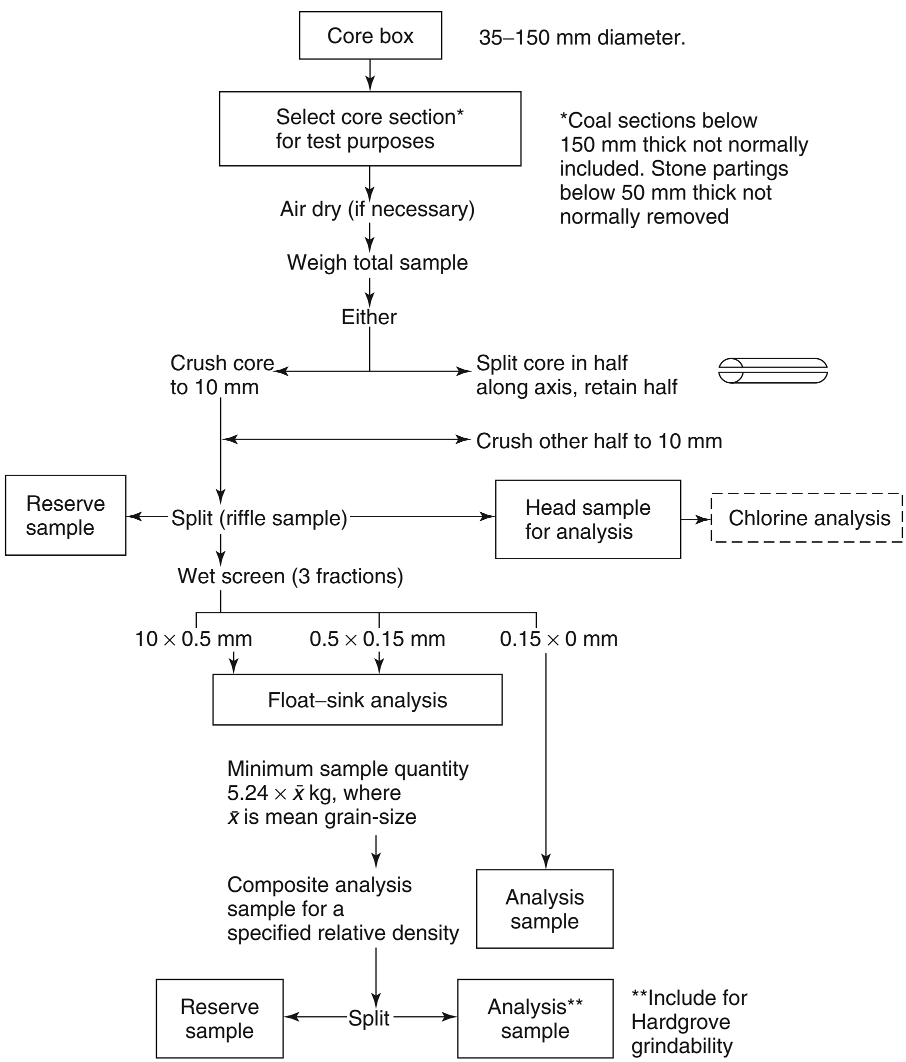



**Figure 5.6** Sample preparation diagram for drill core samples from a steam (thermal) coal deposit. (From Osborne, 1988.)

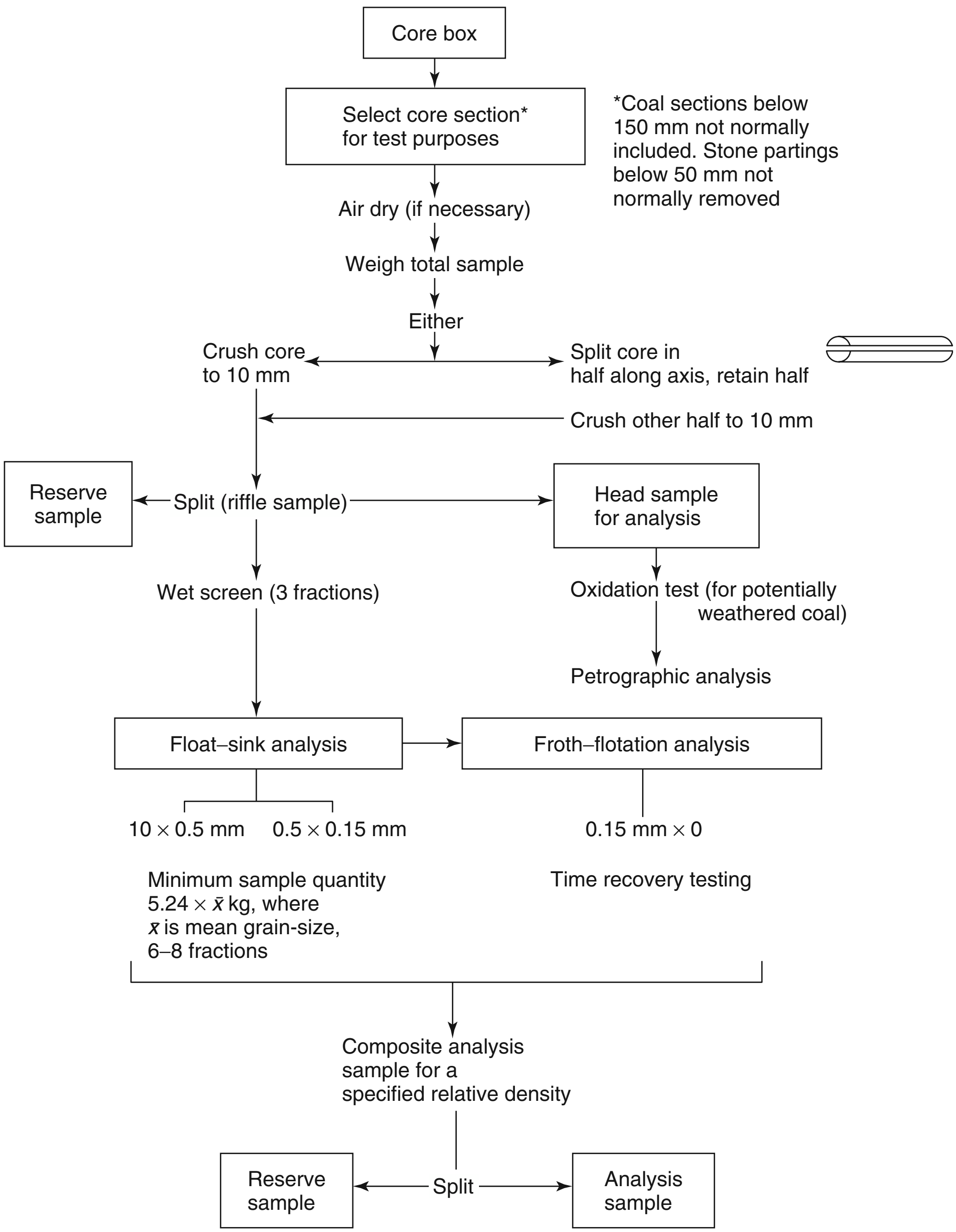



**Figure 5.7** Sample preparation diagram for drill-core samples from a coking (metallurgical) coal deposit. (From Osborne, 1988.)

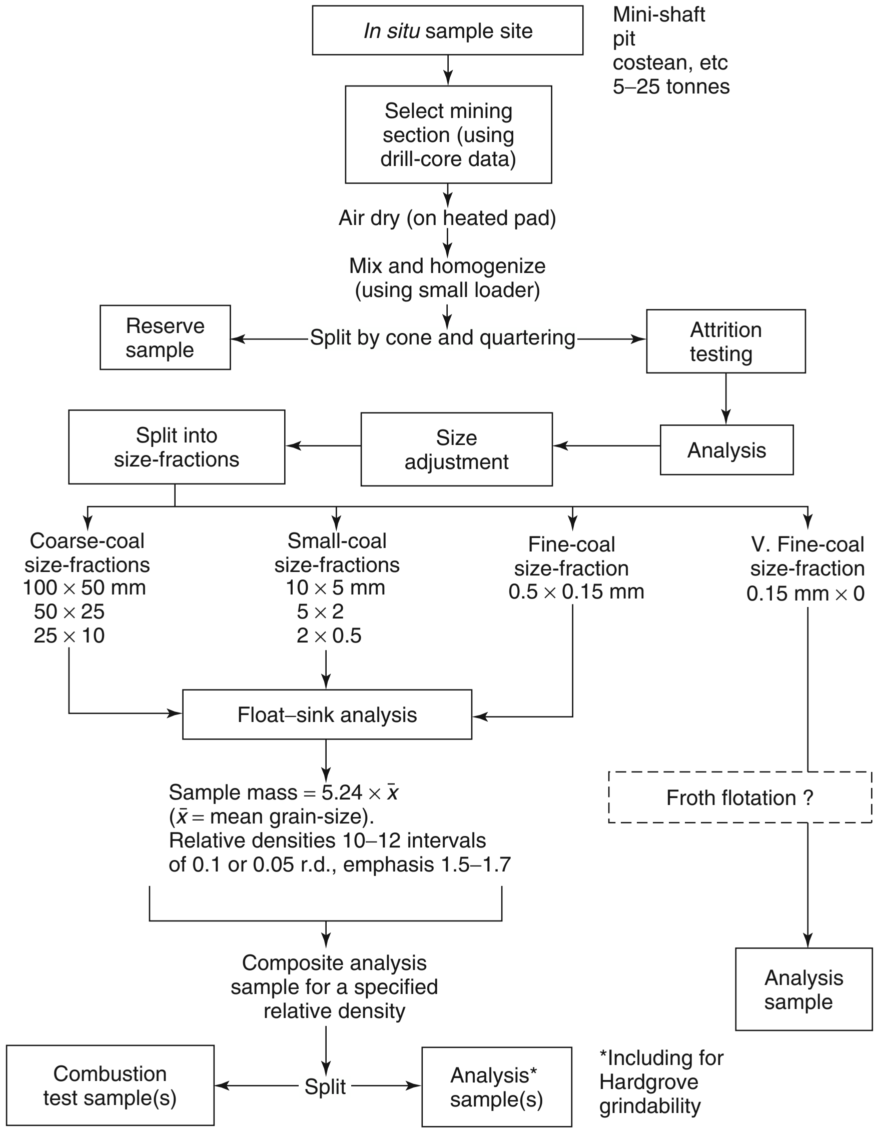



**Figure 5.8** Sample preparation diagram for bulk sample(s) from a steam (thermal) coal deposit. (From Osborne, 1988.)

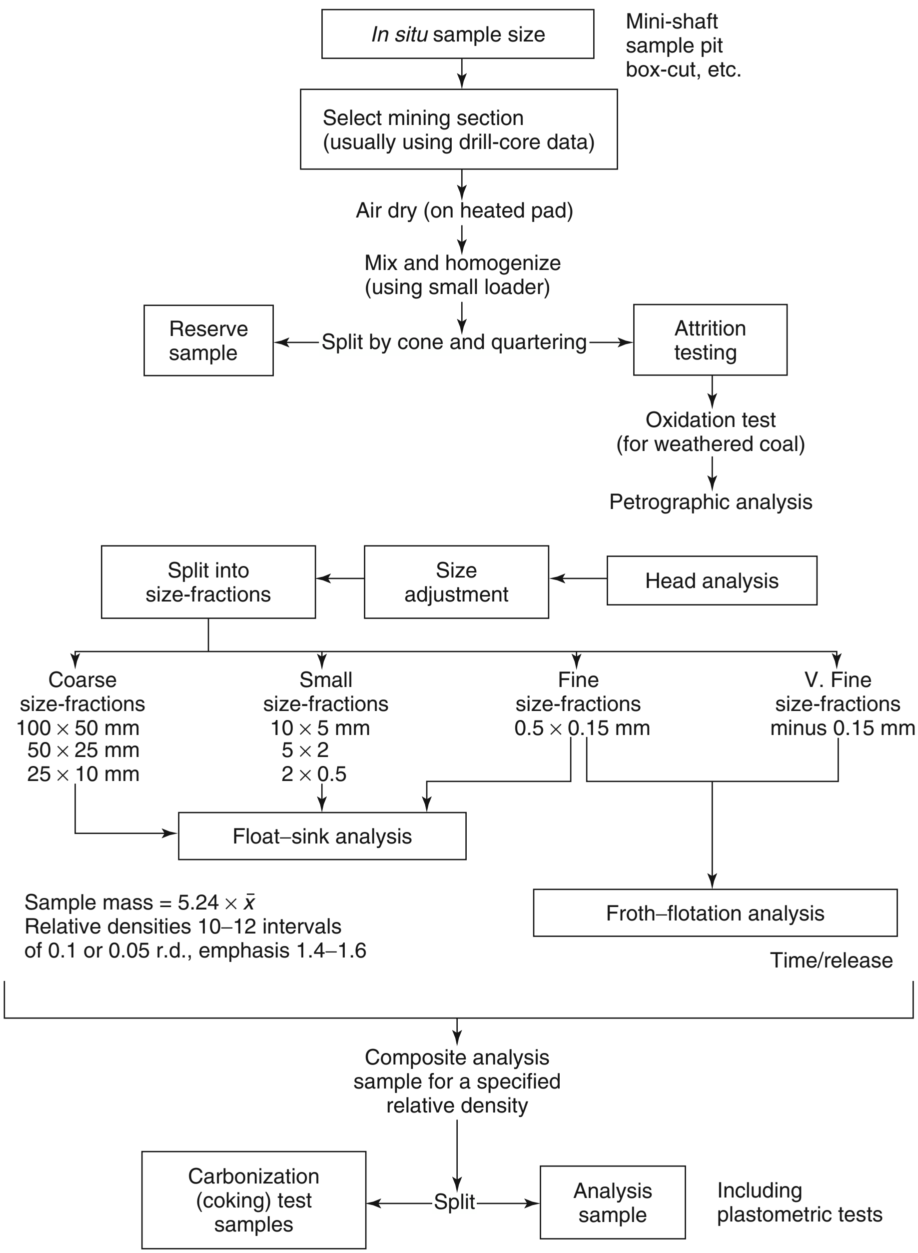



**Figure 5.9** Sample preparation diagram for bulk sample(s) from a coking (metallurgical) coal deposit. (From Osborne, 1988.)

procedures outlined in sections 5.2.2, 5.2.4, and 5.2.6, and sent to the laboratory where they are weighed, crushed and split for analysis.

### 5.4.1 Outcrop/core samples

The procedures for weighing, crushing and splitting outcrop/core samples are shown in Figure 5.6 for steam (thermal) coals and Figure 5.7 for coking (metallurgical) coals. However, there is no universal set procedure and differences do occur.

In Australia, a standard similar to that shown in Figures 5.6 and 5.7 is set, with the difference that 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. 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 used only if the coal has coking properties.

In South Africa, the coal is generally crushed to only 25 mm, and the coal is then analysed for proximate analysis, total sulfur and calorific value. Float–sink tests are done but no simulated product is made. In Australia and South Africa, the fine fraction (0.5 mm or 0.1 mm) is screened out before analysis, dependant upon expectations for coal preparation. In the United Kingdom, a similar procedure is used except that for outcrop/core samples no float–sink analysis is carried out, only proximate analysis, total sulfur and calorific value.

In the United States there are no defined crushed parameters, and proximate analysis, total sulfur and calorific value are determined. Float–sink analysis is done, with the results reported to zero, which often means that the fines have to be screened out. Because in the United States 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.

### 5.4.2 Bulk samples

The procedure for sample preparation of bulk samples of both steam and coking coals is shown in Figure 5.8 for steam (thermal) coals and Figure 5.9 for coking (metallurgical) coals. The coals are analysed as for outcrop/core samples with the additional tests for combustion and coking properties.

### 5.4.3 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 calorific value, 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 in large stockpiles augured samples may be taken, it is difficult to obtain a truly representative sample.

A number of types of on-line analysers 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 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 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

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 which 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, which 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 past 20 yr, the use of microelectronics-based technology has resulted in significant changes in how coal exploration is carried out, in particular, 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 widely used and forms the geological database on which computerized studies are developed.

## 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 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) have 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, and those of 1:2,000 and 1:5,000 are most commonly used. If the area has not been surveyed previously, aerial photographs will be used for map compilation (see section 6.2.2).

*Coal Geology*, Second Edition. Larry Thomas.

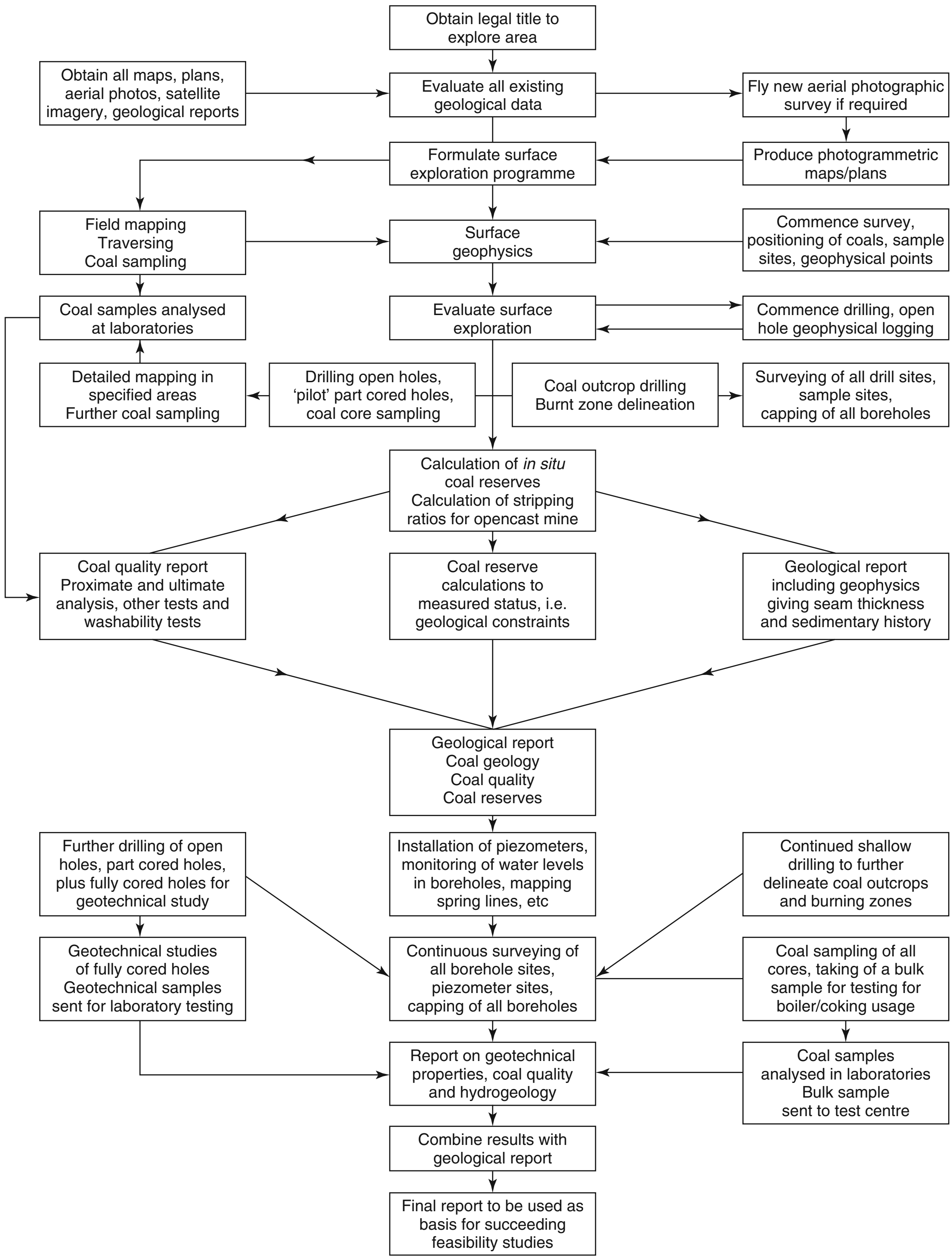



**Figure 6.1** Flow diagram to show principal activities requiring a geological input during the exploration phase of a coal mine development project.

Modern advances in remote sensing, satellite imaging and global positioning systems (GPS), together with enormous improvements in communications, have changed the 'pioneering' aspect of field coal exploration. However, fieldwork will be necessary at all stages of coal mine 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 in low relief arid or scrub country, outcrop may be almost non-existent but unrestricted access by 4-wheel-drive vehicles and trail bikes is possible.

Safety during fieldwork is of prime importance, and suitable clothing should be worn dependent on the climatic region, for example 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 is 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 ones own health and hygiene is 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, for example the daily foot inspection is important as the feet take the brunt of fieldwork particularly in mountainous and jungle areas. Carelessness about ones health can result in discomfort and an inability to carry out fieldwork, and in extreme cases can give rise to the necessity of leaving 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 practice 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, and all notebooks, field record sheets and maps should be clearly labelled and kept dry and 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 programmes provide the ability to construct maps in the field for any desired purpose, for example 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

**Figure 6.2** Field equipment used by coal geologists.

as faults and fold axes and igneous intrusions, all of which if present influence future mining conditions. If for the area of interest, there is already geological information available, then further fieldwork 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 (30 m 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, and when lithological associations have been recognized they should be linked up wherever possible between traverses. Extrapolation across country from one traverse to another should take into account 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. Calculation of true and apparent dip, slope angles, gradients and percentage slope values are shown in Appendix 2.

Significant coal-bearing outcrops and the surrounding strata should be mapped in detail, and stratigraphic sections should be measured where exposure permits.

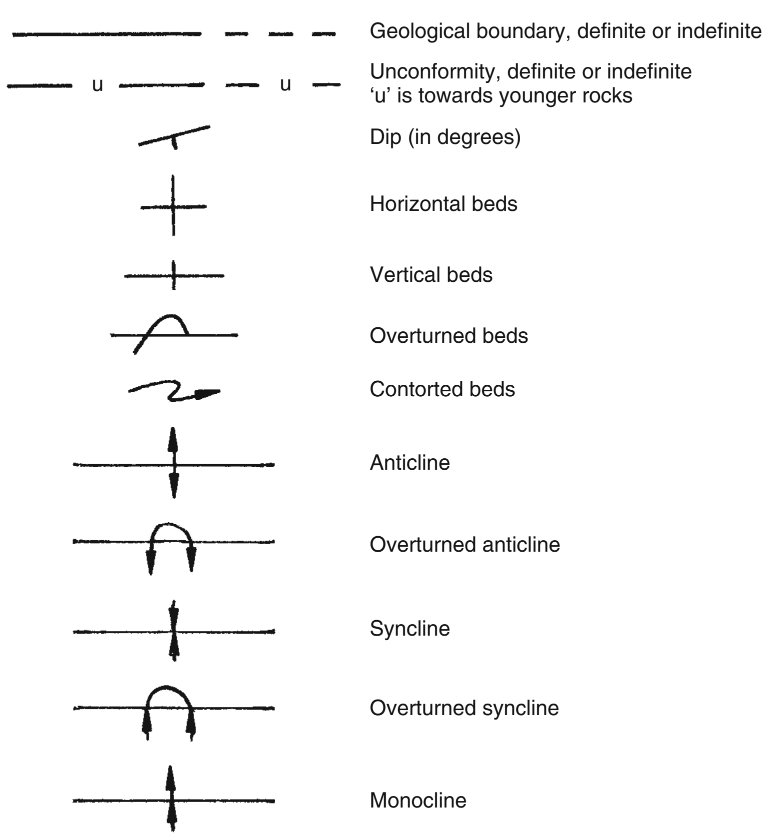



**Figure 6.3** Symbols for geological maps.

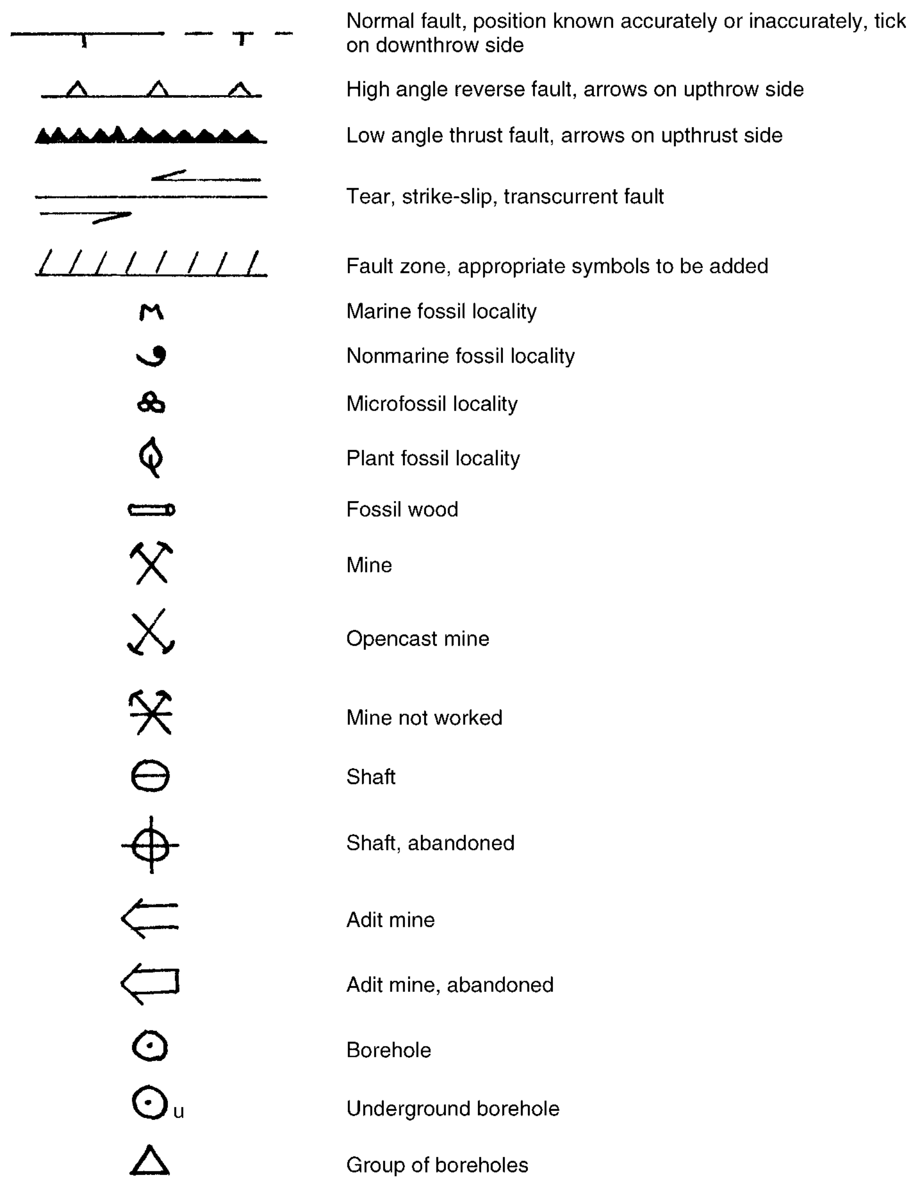



**Figure 6.3** (*Continued*)

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, as 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

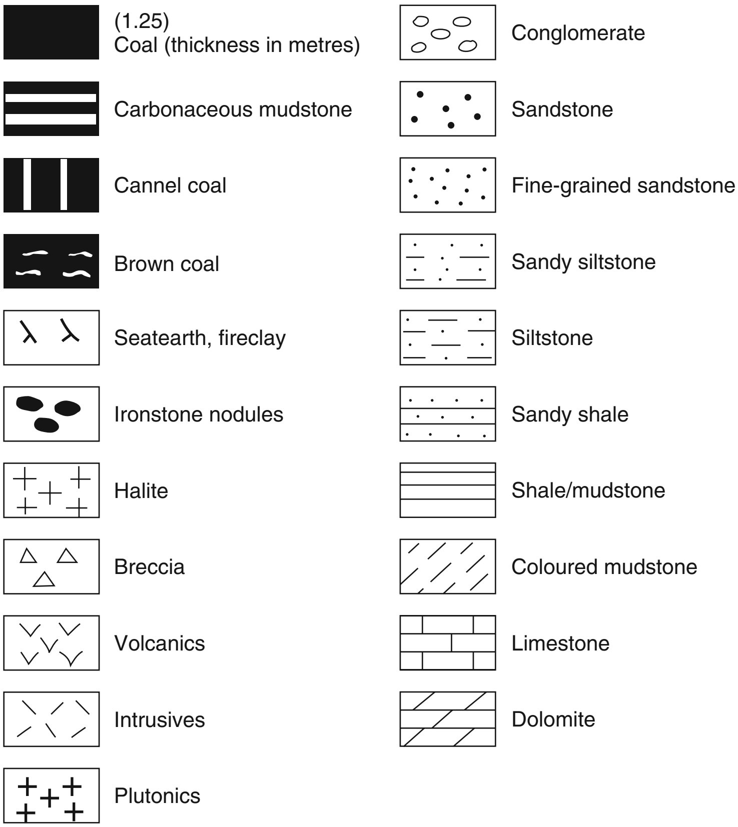



**Figure 6.4** Graphic portrayal of principal lithotypes in coal-bearing sequences.

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 has been compiled to produce the final geological map.

It is essential to be able to recognize all the lithological types associated with coal and coal-bearing sequences, and the ability to do this undoubtedly improves with experience. In the United States, handbooks have been produced illustrating in colour all the lithologies found in coal-bearing sequences in the eastern United States (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 except for the occasional local lithological term. The descriptions of the varieties of sandstone, siltstone, shales, mudstones and coals and their intimate interrelationships 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. Figure 6.7 shows examples of bright coals: a banded bituminous coal showing pyrite mineralization on the cleat face from the Carboniferous of the United Kingdom; a bright non-banded high volatile coal of Tertiary age from Indonesia; a bright high rank anthracite from the Jurassic, Republic of Korea; and a banded, high ash Gondwana coal from India.

In Australia, the Australian Standard AS2916-1986 (SAA, 1986) 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, weathered and heat-altered coals. The system used in South Africa closely follows the Australian Standard, and this is also the case for countries

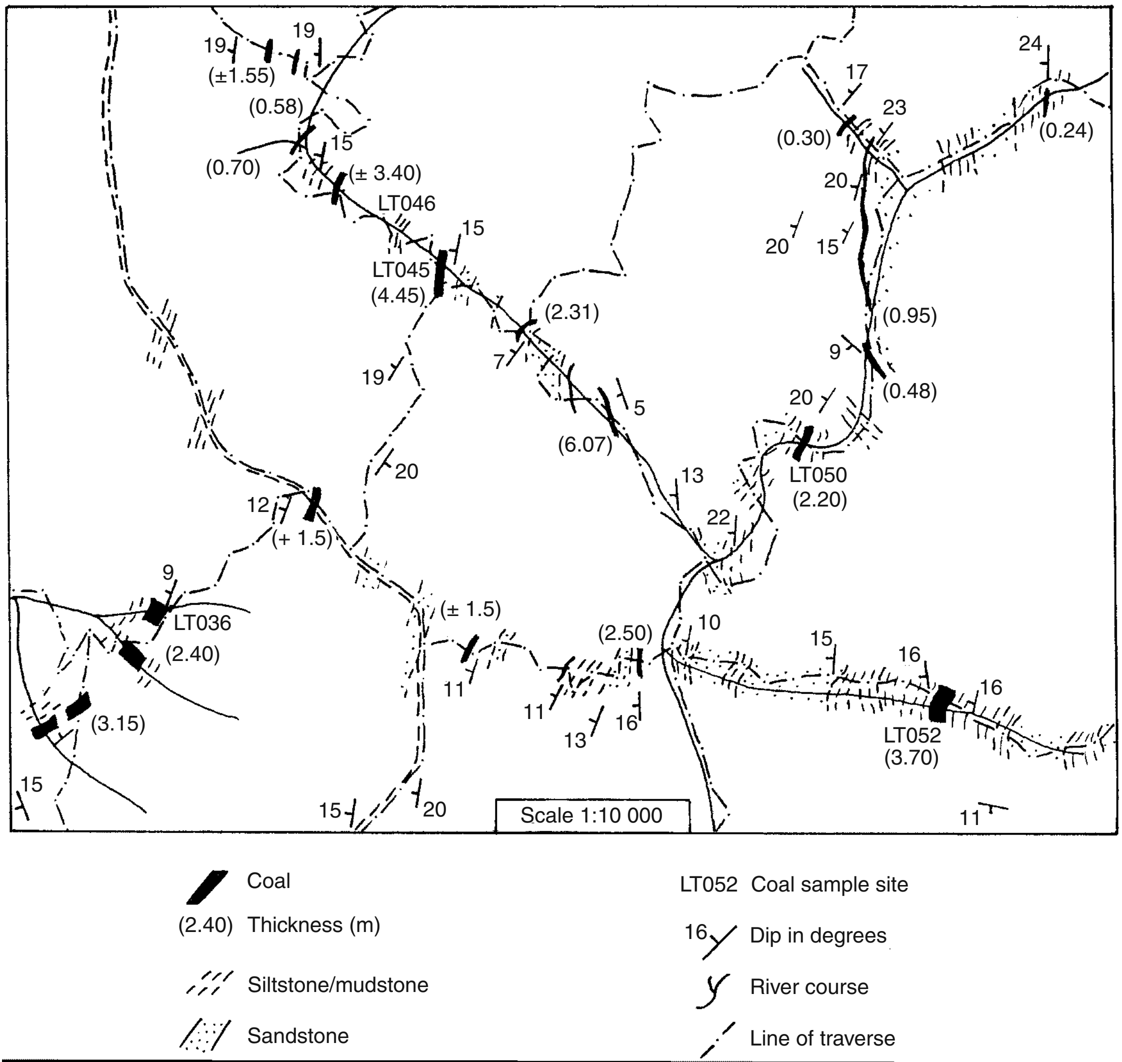



**Figure 6.5** Typical traverse survey showing coal outcrops, sample sites and structure.

influenced by these areas, for example Indonesia. In the United Kingdom, 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 United States, there is no standardized system for the graphic representation of coals, however, an example produced for the ASTM 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, liquifaction structures;
8. degree of weathering – from fresh to completely weathered;
9. roof/floor and/or seam structure – flat, rolling, discontinuities of bedding, splitting;
10. mineralization.

### 6.2.2 Global positioning system (GPS)

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 revolutionized field exploration, especially in those more remote areas where green-field exploration takes place.

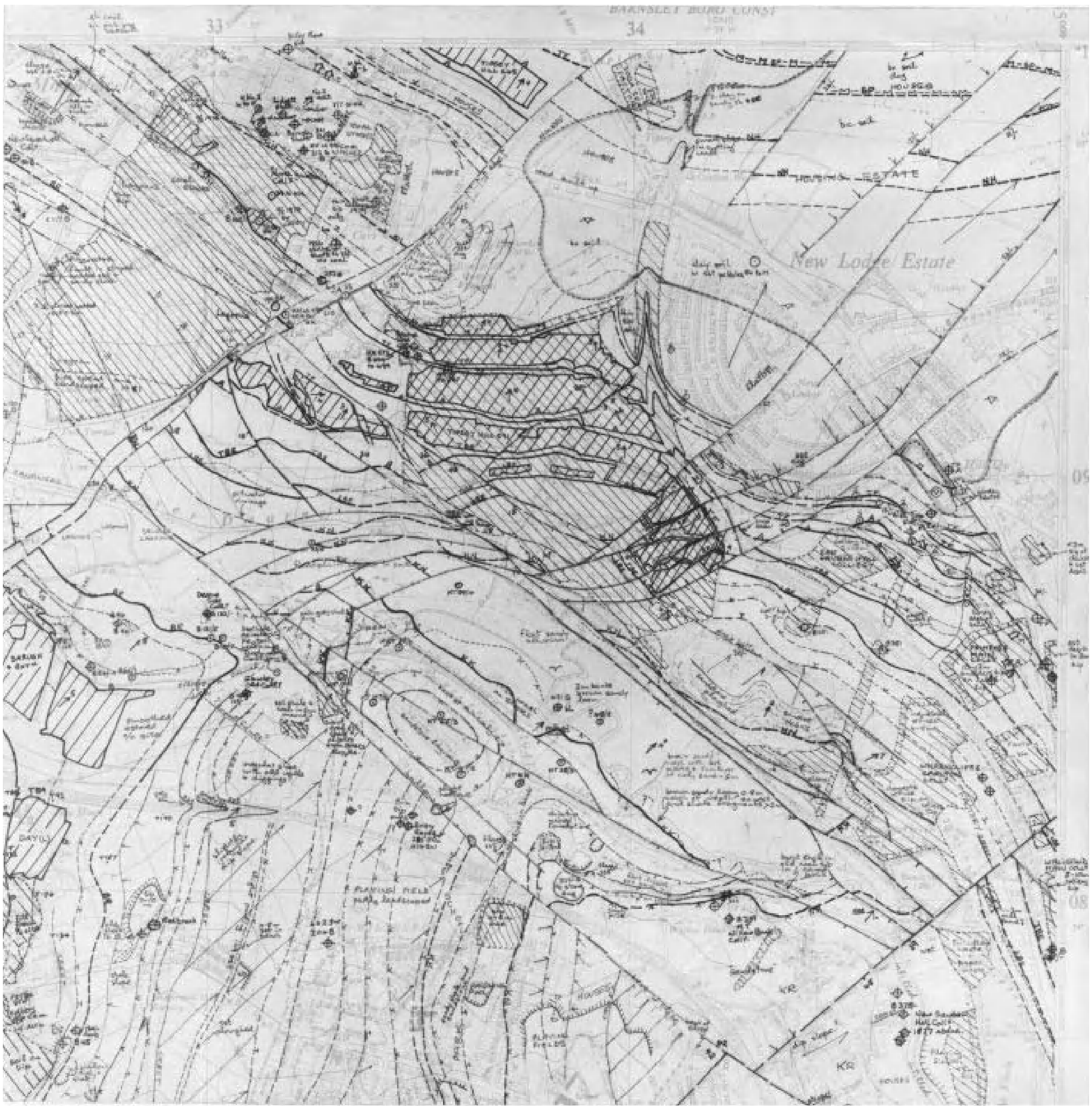
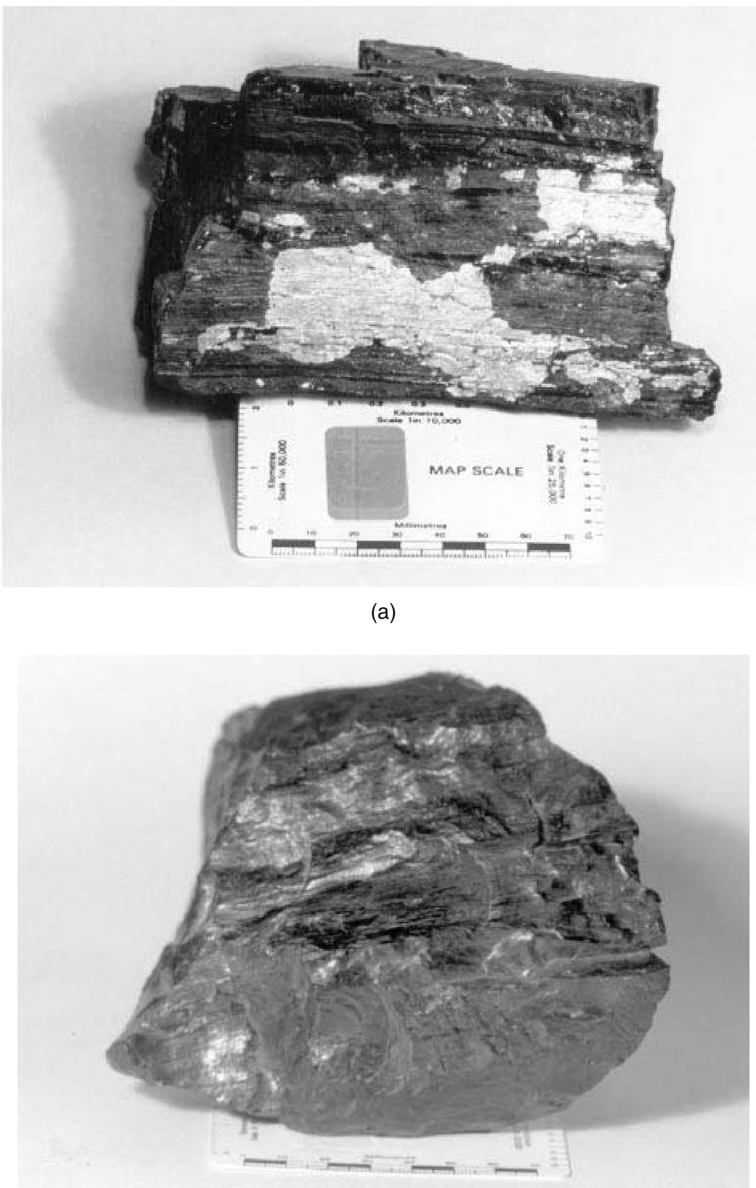



**Figure 6.6** Field map of a UK Coalfield area, showing geology and past and present coal mining activity; scale 1:10,000. 1PR/25-6C British Geologial Survey.© NERC. All rights reserved.

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 (0.00001°) in latitude and longitude (Milsom and Eriksen, 2011). A typical handheld GPS receiver is shown in Figure 6.11a and these are produced by a number of manufacturers.

### 6.2.3 Portable personal computers (PC)

Handheld 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 are loaded onto a PC workstation for final map preparation. Various models cover a range of software capabilities.




(a)

(b)

**Figure 6.7** Varieties in black (hard) coal. (a) Banded bituminous coal from Northumberland, United Kingdom, with pyrite mineralization on the cleat face. (b) High volatile, low ash bituminous coal, non-banded, from East Kalimantan, Indonesia.


(c)

(d)

**Figure 6.7** (c) Bright, high rank anthracite, highly tectonized, from Samcheog Coalfield, Republic of Korea. (d) Banded, high ash bituminous coal from Talcher Coalfield, India. (Photographs by M. C. Coultas and LPT.)

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 be the standardization of data recording by different geologists, the combination with GPS means location and orientation accuracy is achieved, 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 visualization systems. Figure 6.11b shows a tablet PC in use in the field.

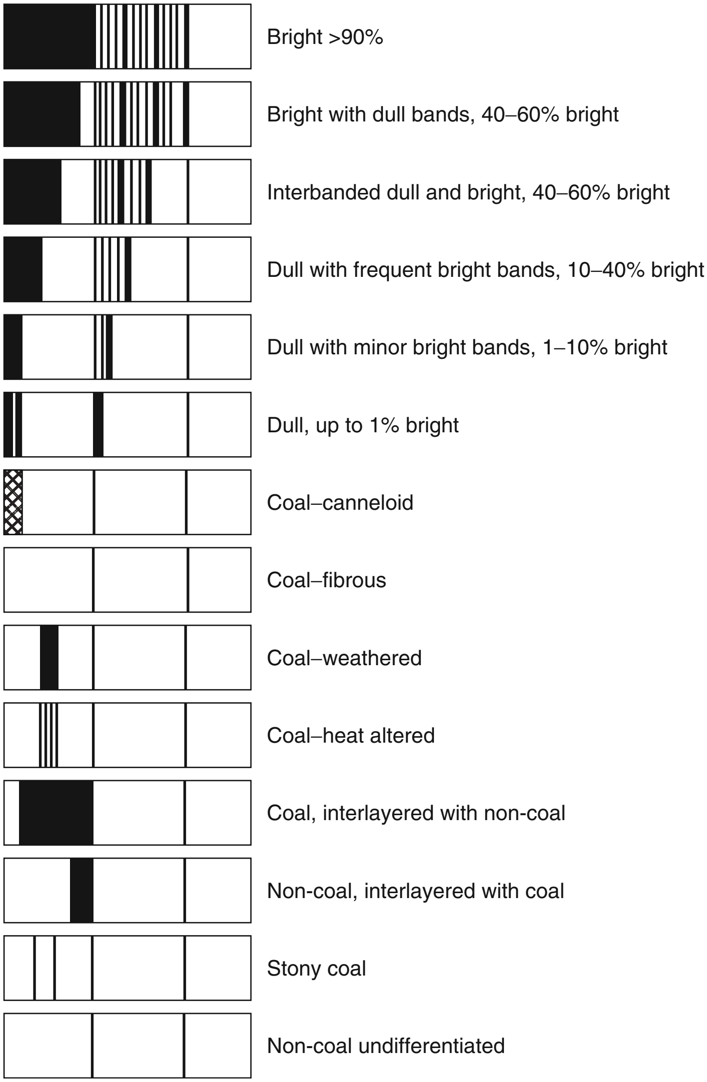



**Figure 6.8** Graphic representation of coal seams. (Based on Australian Standard 2916-1986 (SAA, 1986).)

### *6.2.4 Remote sensing*

#### *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. (EOSTAT), and from the French SPOT series of spacecraft. Both have worldwide coverage from which data can be obtained in computer-compatible tape (CCT) and photographic form for ground scenes, each 34,225 $km^2$ 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 spectrum are most useful for interpreting geological features. Landsat interpretation is usually used in conjunction with geological and geophysical maps, if available. As

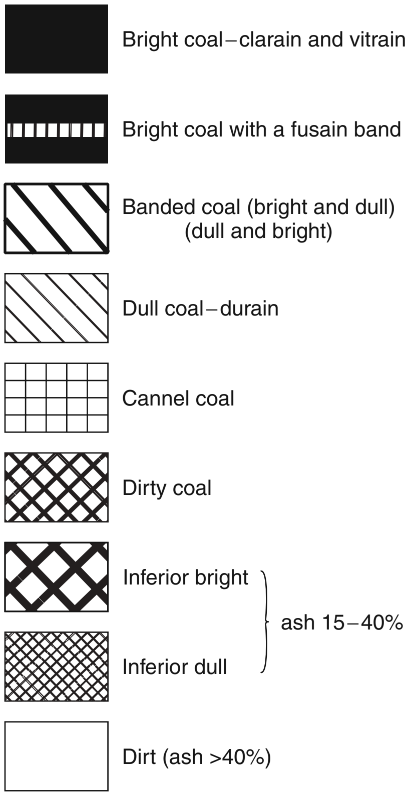



**Figure 6.9** Graphic representation of coal seams as used in United Kingdom.

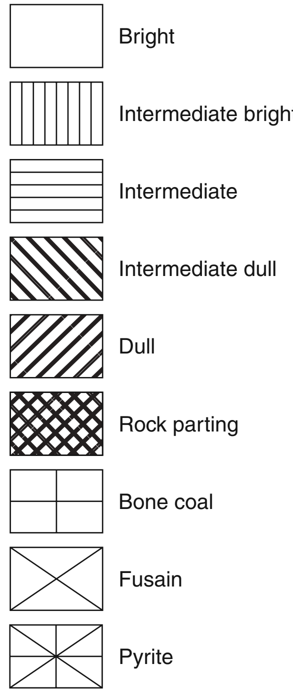



**Figure 6.10** Example of a graphic representation of coal seams as used in the United States. (From Goscinski and Robinson (1978) *ASTM Technical Publication 661*.)

well as highlighting geological structures, variations in rocks, soils and vegetation is 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 recognized.

The SPOT spacecraft collect data for ground scenes of 3600 $km^2$ 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 (SMIRR) and shuttle imaging radar (SIR) have geological applications. Satelliteborne radar has the great advantage of penetrating the persistent cloud cover and forest cover that characterize large areas, particularly those in the tropics.

### *6.2.4.2 Airborne imagery*

Aerial photographs have been available for the past 80 yr, 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 (three dimensional) 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 photo-interpretation 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. On plans the symbols used to portray such geological features are shown in Figure 6.12.

When no, or inadequate aerial photograph 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 later can be accurately surveyed and used as permanent reference


(a)

(b)

**Figure 6.11** (a) Handheld GPS receiver, Magellan Triton 400. (Photograph by LPT. Reproduced by permission of Dargo Associates Ltd.) (b) Rugged tablet PC for capturing digital geological field data. (British Geological Survey. © NERC. All rights reserved.) This figure is reproduced in colour in the Plates section.

points. The scale of the photographs should be in 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 can still reflect closely the underlying geology. When combined with fieldwork, subtle lithological differences become apparent. Such photographic interpretation

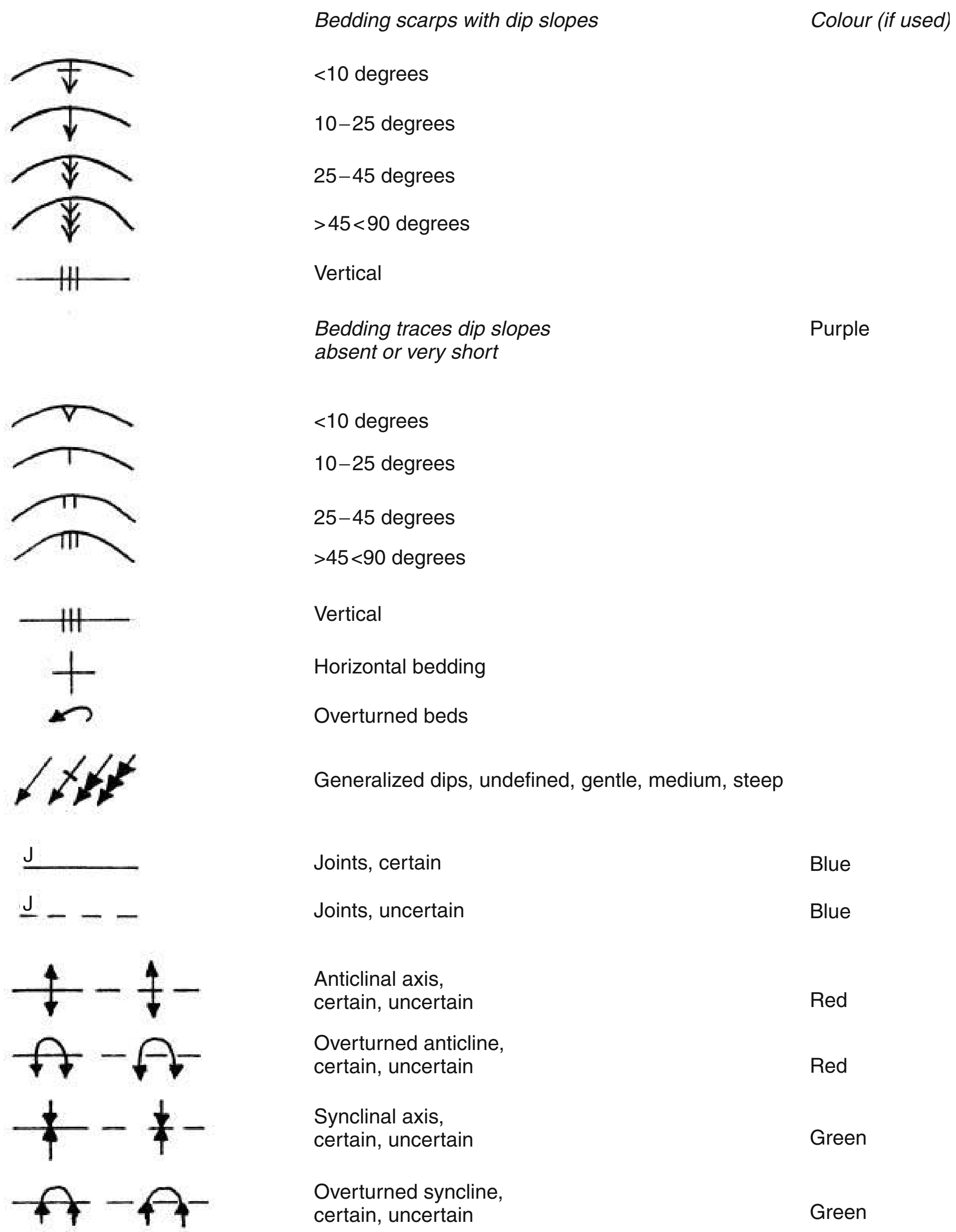
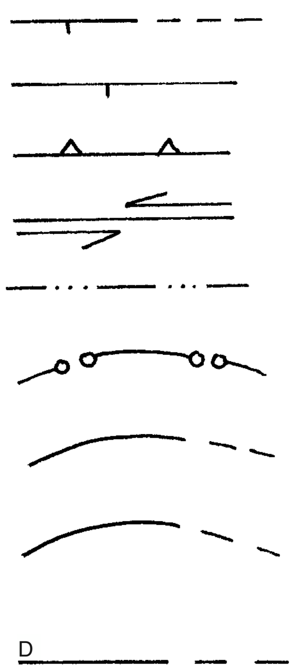



**Figure 6.12** Photogeological symbols for use on aerial photographs and photogrammetric plans (from various sources.)



| | |
|---|---|
| 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*)

is essential when used in reconnaissance and detailed exploration fieldwork, in identifying traverses and pinpointing physical features on those traverses, for example river junctions and road intersections, coal outcrops and for locating preliminary drilling sites.

The drawbacks with aerial photographs are that in areas of high rainfall, cloud-free conditions are infrequent, which prohibits 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 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 features. The SLAR can be used to compensate for the lack of aerial photograph cover, as it can highlight structural features not seen at ground level, accurately locate structural elements in the field and provide a regional structural framework that can be used for planning field traverses and exploration drilling grids. The SLAR has been used extensively in the United States 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 openhole 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.

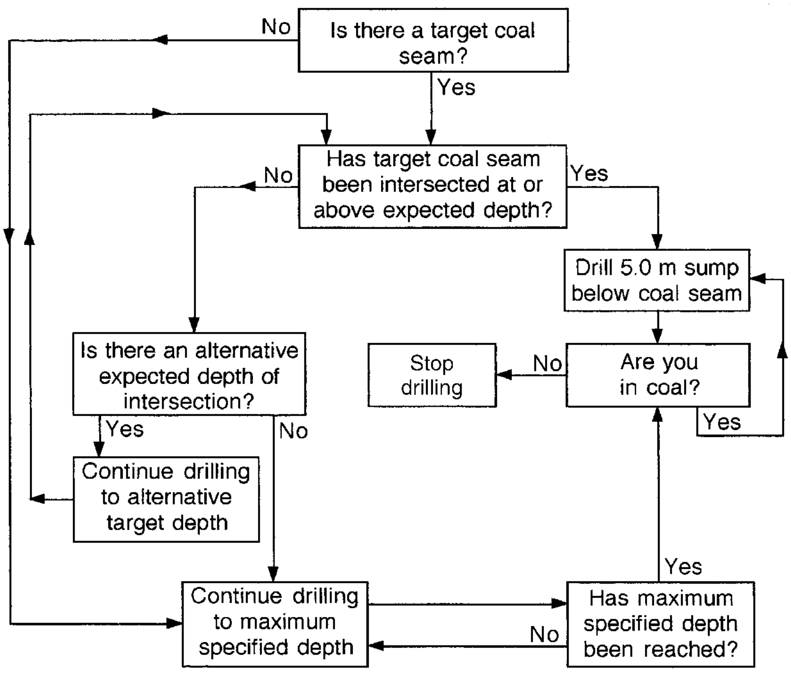



**Figure 6.13** Drilling procedure to correctly complete a borehole containing one or more targeted coal seams. (Reproduced by courtesy of M. C. Coultas.)

If the geologist is supervising junior or less experienced geological staff who may 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) openhole 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 stabilize hole sidewalls is now replacing bentonite mud. The use of foam has the added advantage of allowing boreholes to be used later 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 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 resharpened 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

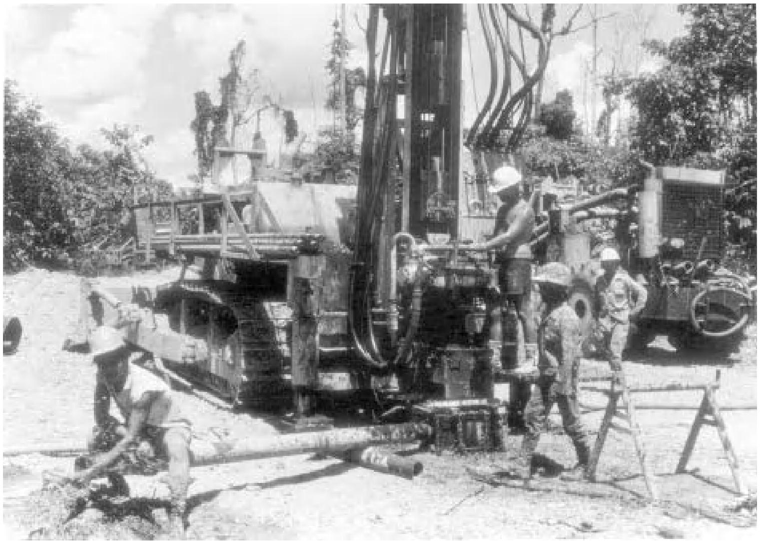


**Figure 6.14** Dando dual air–mud flush rig mounted on a bulldozer for use in difficult tropical terrain. (Photograph by M. C. Coultas.)

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 groundwater influx, when small compressors 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 liklihood 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) and Edeco (Figure 6.15A) dual air- and fluid-flush rigs and the Mayhew 1000 (Figure 6.15B) a fluid-flush rig.

### *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; diamond drilling, however, 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. Recovery of the rock core is facilitated by a non-rotary second metal tube within the core barrel. The core passes into this second tube and is protected from damage. This is called a double tube core barrel. Even better core recoveries can be obtained by a triple tube core barrel, whereby an additional smooth metal tube, split longitudinally, is placed inside the non-rotating inner segment of the double tube core barrel.

When the core is to be removed, the split inner tube is withdrawn with the core still inside, it is then be laid out horizontally and the upper half of the tube is removed to expose the core for examination, following which it is transferred to a segmented core box for logging at a later stage. Removal of the core barrel can be time consuming, but this can be alleviated by 'wire-line' drilling, which allows the central part of the core barrel to be drawn up the centre of the hollow drill rods on a steel cable, with the rods themselves only being 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

(a)

(b)

**Figure 6.15** (a) Edeco rig operating in the United Kingdom; (b) Mayhew 1000 truck mounted rig, operating in tropical conditions, Indonesia. (Photographs by M. C. Coultas.)

**Table 6.1** Core sizes for wireline, conventional and air-flush drilling.

| Core barrel type | Type and hole/ core diameters (mm) | Model | | | | | |
|---|---|---|---|---|---|---|---|
| Wireline | Q Series (Boart Longyear) | AQ | BQ | NQ | HQ | PQ | |
| | Hole | 48.0 | 60.0 | 75.8 | 96.0 | 122.6 | |
| | Core | 27.0 | 36.5 | 47.6 | 63.5 | 85.0 | |
| | Diamand-Boart | ADBG.AQ | BDBG.BQ | NDBG.NQ | HDBG.HQ | PDBG.PQ | SDB.G |
| | Hole | 47.6 | 59.6 | 75.3 | 95.6 | 122.2 | 145.3 |
| | Core | 27.0 | 36.4 | 47.6 | 63.5 | 85.0 | 108.2 |
| Conventional | Double tube swivel type | HWF | PWF | SWF | WWF | ZWF | |
| | Hole | 98.8 | 120.0 | 145.4 | 174.5 | 199.6 | |
| | Core | 76.2 | 92.1 | 112.8 | 139.8 | 165.2 | |
| Air-flush | | 412F | HWAF | | | | |
| | Hole | 105.2 | 99.4 | | | | |
| | Core | 75.0 | 70.9 | | | | |

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 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. Casing may be metal or PVC, and 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, which 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 one metre compartments with lids, usually of wood, as shown in Figure 6.16, 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 core in boxes the wrong way up but 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 and the coal seam lithotypes are recorded in detail together with any partings or splits. In addition the degree of weathering, mineralization 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 cored only for the coals and the coal roof and floor, with the depths at which cores are taken being predetermined by the previous exploratory openhole drilling. These part-cored boreholes are sited as 'pilot' holes next to completed openholes in order to accurately predict the depths and thicknesses of the coal seams. Figure 6.17 shows a typical exploration borehole grid with openholes 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 can be dismantled into man-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 (lawn mower 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, in order to obtain greater penetration, the downward pressure may be increased by adding

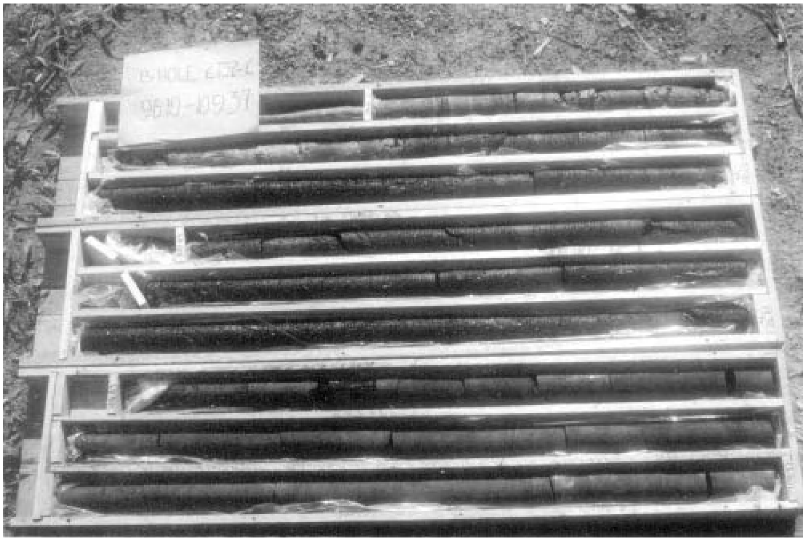


**Figure 6.16** Borehole core laid out in wooden boxes with depth markers, awaiting examination by the geologist. (Photograph by LPT.)

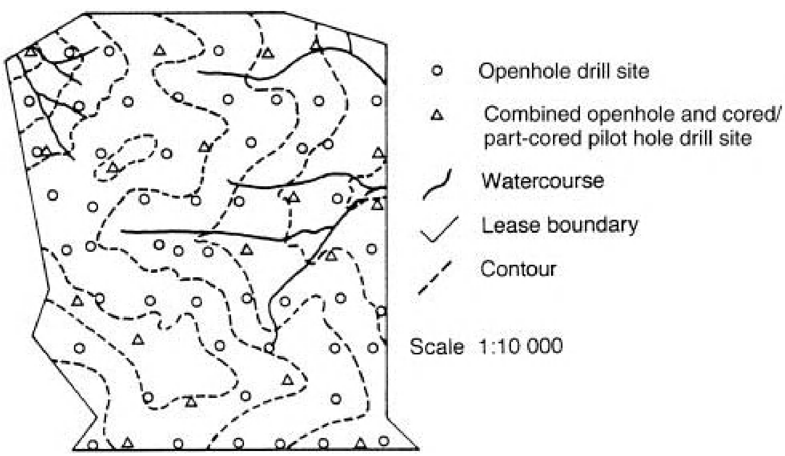



**Figure 6.17** Exploration drilling grid showing distribution and position of openholes and part-cored pilot holes.

increasing numbers of personnel to sit/stand on the top of the rig as shown in Figure 6.18.

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, the Minuteman and the Voyager 1000, 2000 and 2400 Series 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. 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

**Figure 6.18** Portable drilling rig using manpower to exert downward pressure on the drill bit. In use in East Kalimantan, Indonesia. (Photograph by LPT.)

actual depth penetrated, this includes both solid core 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 6.19 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, and 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, and great care

**Figure 6.19** Coal geologist logging borehole core in a core shed on-site. (Photograph by LPT.)

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 was 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 (AZCL; 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, and 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 6.16, and any special feature of the coal or its contact with the beds above and below should be photographed. Figure 6.20 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. 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 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, while the other is left until the core has dried out sufficiently 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 portable computer. Figure 6.21 shows a core logging sheet giving depths, lithological description, rock strength, weathering, bedding character and sample numbers. As a general guide, the following features should be recorded.

1. Lithology: dominant lithotype, colour and shade, grain size and sorting, distinctive mineralogy and cementation, associated lithotypes and relative proportions.
2. 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.
3. Mechanical characters: degree of weathering, degree and types of fracturing, orientation and frequency, strength characteristics, mineralization.

### 6.3.4.2 Openhole logging

The logging of open boreholes involves the identification of rock chippings collected for every metre drilled. The

**Figure 6.20** 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. (Photograph by M. C. Coultas.)

chippings are washed, laid out on polyethylene sheets on benches, and then examined by the geologist, as shown in Figure 6.22. 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.

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 his/her 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, but 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 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

| Project code | Type | Number | Suffix | Geophy. | Co-ord. Datum | Easting | Northing | Elevation | Total Depth | Depth to Base of Weathering | Depth to Water Table | Core Size (mm) | Incl. | Azim. | Units |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| CCL | | 51 | | | | | | 24.45 | 115.82 | | | 63.5 | | | |

| Commenced | Completed | Geo. | Drill Contr. | Driller | Geoph. Contr. | Plugs | Casing Depth | Status | Other Records |
|---|---|---|---|---|---|---|---|---|---|
| 12/11/87 | 15/11/87 | JGC | | | | | | | |

HOLE No. CCL 51
PAGE 2 OF 3
ENCODED BY [illegible]

| Depth to Base of Unit | Recovered Thickness | Rock type | Shade | Colour | Lith. Adj. | Grain Size | Tex. | Weath | Mec. St. | Minor Rock Rel. | % | Bedding Sed. Str. | Base | Dip | Cond. | Disc. Spac. Dip | Qual. Adj. A Adj. Qual. | Formation | Seam | Split | Occur. | Sample Number | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | CT | | DEBDB | | | | S | R2 | IB | | | | | | | | | | | | | |
| 60.98 | 0.61 | LG | | DB | | | | S | R2 | | | XLIL | DF | 10 | | | | LNC | | | | CCL51US2 | |
| 62.01 | 1.03 | LC | | DEB | | | | | S5 | | 71 | IL | | 10 | | | | LNC | | | | CCL51US3 | |
| | | CT | | DB | | | | S | R2 | TM | | | | | | | | | | | | | |
| 65.40 | 3.39 | CL | | LGMG | ST | | | | S5 | | 87 | | | | | | | LNC | | | | | |
| | | IS | | MGBF | SR | FM | | | R4 | TN | | | | | | | | | | | | | |
| | | | | IRONSTONES AT: 62.84 - 62.92, 63.11 - 63.2, 63.50 - 63.57, 63.98 - 64.06, 64.84 - 64.96m | | | | | | | | | | | | | | | | | | | |
| 65.55 | 0.15 | CL | | MG | SF | | | | S5 | | | | | | | | | LNC | | | | CCL51LL4 | |
| 65.70 | 0.15 | CL | | MG | ST | | | | S5 | | | | IN | | | | | LNC | | | | CCL51LL5 | ROOF LOWER LIGNITE 65.70M |
| 66.75 | 1.05 | LG | | DBBK | | | | S | R2 | | | XLIL | | 10 | | | SCYLM | LNC | LLG | | C | CCL51LL6 | |
| 67.70 | 0.95 | LG | | DBRB | | | | S | R2 | | | XL | | | | J3V | CWLLN | LNC | LLG | | C | CCL51LL7 | |
| 68.72 | 1.02 | LG | | DBRB | | | | S | R2 | | 68 | | | 5 | | F3L | | LNC | LLG | | C | CCL51LL8 | |
| | | CT | | DBBK | | | | S | R2 | TU | | | | | | | | | | | | | |
| 69.80 | 1.08 | LG | | DBRB | | | | S | R2 | | 81 | XL | | 5 | | F3L | | LNC | LLG | | C | CCL51LL9 | |
| | | LG | | DBBK | WD | | | S | R3 | XN | | | | | | | | | | | | | |
| 70.90 | 0.90 | LG | | DBRB | | | | S | R2 | | 69 | XL | | 5 | | F4L | | LNC | LLG | | C | CCL51LL10 | |
| | | CT | | DBMB | | | | S | R2 | BU | | | | | | | | | | | | | |
| 73.19 | 2.29 | LG | | DBRB | | | | S | R2 | | 91 | XLIL | | 5 | | F3W | | LNC | LLG | | C | CCL51LL11 | |
| | | CT | | DEBMB | | | | S | R2 | XN | | | | | | | | | | | | | |
| 74.52 | 1.33 | LG | | DBRB | | | | S | R2 | | 88 | XL | | 5 | | F4W | | LNC | LLG | | C | CCL51LL12 | |
| | | CT | | DB | | | | S | R2 | MU | | | | | | | | | | | | | |
| 76.00 | 1.48 | LG | | DBRB | | | | S | R2 | | 89 | XL | | | | F3W | | LNC | LLG | | B | CCL51LL13 | |
| | | LG | | DOB | FC | | O | S | R1 | TN | | | | | | | | | | | | | |
| 82.00 | 6.00 | LG | | DBRB | | | | S | R2 | | 95 | MS | | | | F3W | | LNC | LLG | | B | CCL51LL14 | |
| | | LG | | DOB | FC | | O | S | R1 | XN | | | | | | | | | | | | | |
| 88.00 | 6.00 | LG | | DBRB | | | | S | R2 | | 94 | XL | | 5 | | F3W | | LNC | LLG | | B | CCL51LL15 | |
| | | LG | | DOB | FC | | O | S | R1 | XN | | | | | | | | | | | | | |
| 94.00 | 6.00 | LG | | DBRB | | | | S | R2 | | 89 | XL | | 5 | | F3W | | LNC | LLG | | B | CCL51LL16 | LL2/LL1 BOUNDARY RE-ASSESSED AT 91.88M |
| | | LG | | DOB | FC | | O | S | R1 | TN | | | | | | | | | | | | | SAMPLE 16 THEREFORE INCLUDES SPLITS A AND B |
| 95.18 | 1.18 | LG | | DB | | | | S | R2 | | 70 | XL | | | | F3L | | LNC | LLG | | A | CCL51LL17 | BOUNDARY IS TAKEN TO NEAREST SAMPLE i.e. 94.00 m. |

END OF RECORD

**Figure 6.21** 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.

**Figure 6.22** Coal geologist logging borehole chip samples in an on-site core shed. (Photograph by M. C. Coultas.)

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**, for example moderately weak
**weathering**, for example fresh
**texture and structure**, for example thinly cross-laminated
**colour**, for example light grey
**grain size**, for example fine to medium
**name**, for example sandstone
**other properties**, for example slightly silty with mudstone laminae.

Definitions of the various levels of strength, weathering, texture, structure and grain size vary. Those given below are the definitions given in British Standard 5930. 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) and the Geological Society Engineering Group Working Party (1970).

### *6.4.1 Strength*

There are several kinds of rock strength, which can be determined accurately only 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. 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 groundwater, 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.

**Table 6.2** Terms used to assess material strength in the field.

| Material | Strength | Characteristic of material |
|---|---|---|
| Rock | 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 ca. 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 |
| Cohesive soil and clay | C4 Stiff | Can be indented by thumb nail, 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 |
| Non-cohesive soils | 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 discolouration and maximum strength |
| W5 Slightly weathered | Discolouration along major discontinuity surfaces, may be some discolouration 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, discolouration penetrates deeply and the original fabric is present only 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 |

### 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, and uniformity between geologists is difficult to achieve. There are several published rock colour charts, for example the *Rock Color Chart* published by the Geological Society of America, and these can be used together with supplementary terms such as light, dark, mottled, etc., and secondary descriptors such as reddish, greenish. Colour is important to note because it can be an aid to correlation and, during the

**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 |

course of an investigation, it may become apparent that a distinctive coloured horizon is of particular significance.

### 6.4.5 Grain size

Grain size is important in the description of rock and soil material, and its omission can be justified only when describing mudstones, claystones, shales and siltstones in hand specimen. 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, for example angular, subangular, subrounded, and rounded, and of form, for example equidimensional, flat, elongated, flat and elongated, and irregular.

### 6.4.6 Total core recovery (TCR)

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 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 (SCR)

This is the total length of pieces of core recovered which have a full diameter, expressed as a percentage of the full core run. Like total core recovery, this can be less or more than 100%.

### 6.4.8 Rock quality designation (RQD)

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). The calculation of RQD is 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 core lengths terminated only by natural fractures being considered:

$$\mathrm{RQD}(\%) = \frac{100 \times \text{length of core in pieces} > 100\,\text{mm}}{\text{length of borehole}}$$

As with total core and solid core recovery, percentages can be more than 100%, but RQD cannot be greater than solid core recovery. For core with one large fracture along the entire length, 0% should be recorded. The RQD is sometimes expressed for lithological units rather than core runs. Rock quality designation 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 (FI)

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 index for subunits should be given. An upper limit to the fracture spacing index should be defined, above which the index is not calculated but is recorded as being greater than the defined limit, for example >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.

### 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, for example B = bedding, J = joint, F =

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-0(4) | Broken rock |

fault, S = shear, Fr = fracture, FrZ = fracture zone, SZ = shear zone and FZ = fault zone.

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, undulose or curvi-planar, and if seen, the surface can be described as smooth, rough, polished or slickensided (Figure 6.23). The aperture is usually described according to its width: that is, closed = <0.1 mm, tight = 0.1–1.0 mm, open = 1–5 mm, and wide = >5 mm (West, 1991). The infill is a description of any material filling the aperture, usually materials such as clay, calcite and broken rock.

An example of a combined geotechnical logging sheet is shown in Figure 6.24.

### 6.4.11 Rock mass rating (RMR)

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, Lien and Lunde (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 $Q$), where by

$$Q = \frac{RQD \times J_r \times J_w}{J_n \times J_a \times SRF}$$

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, Lion and Lunde, 1974). Although RMR values are usually applied to civil engineering excavations, an understanding of the various parameters used is important when planning and implementing excavations in underground coal mines.

## 6.5 Computer applications

There are now a large number of software packages available that provide integrated geological data management, analysis and visualization. Geological data collected in the field in the form of direct observation, sample points, borehole data, survey and location data, can all be inputted into the selected software programme to produce point data plans, contour plans, survey plans, three-dimensional surface diagrams and volumetrics, as well as producing cross-sections and vertical logs.

The use of global positioning systems (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 inputted into the geological database or encoded directly into a personal computer or tablet (Figure 6.11b) carried in the field or on site. These 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

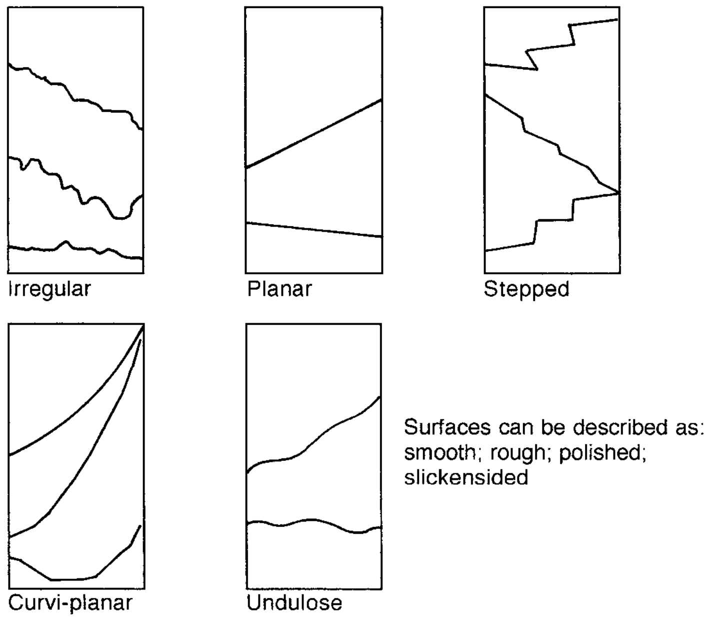



**Figure 6.23** Types of discontinuities: including bedding and joints that have no displacement, and faults and shears that have a measurable or unknown displacement. (Reproduced by permission of Dargo Associates Ltd.)

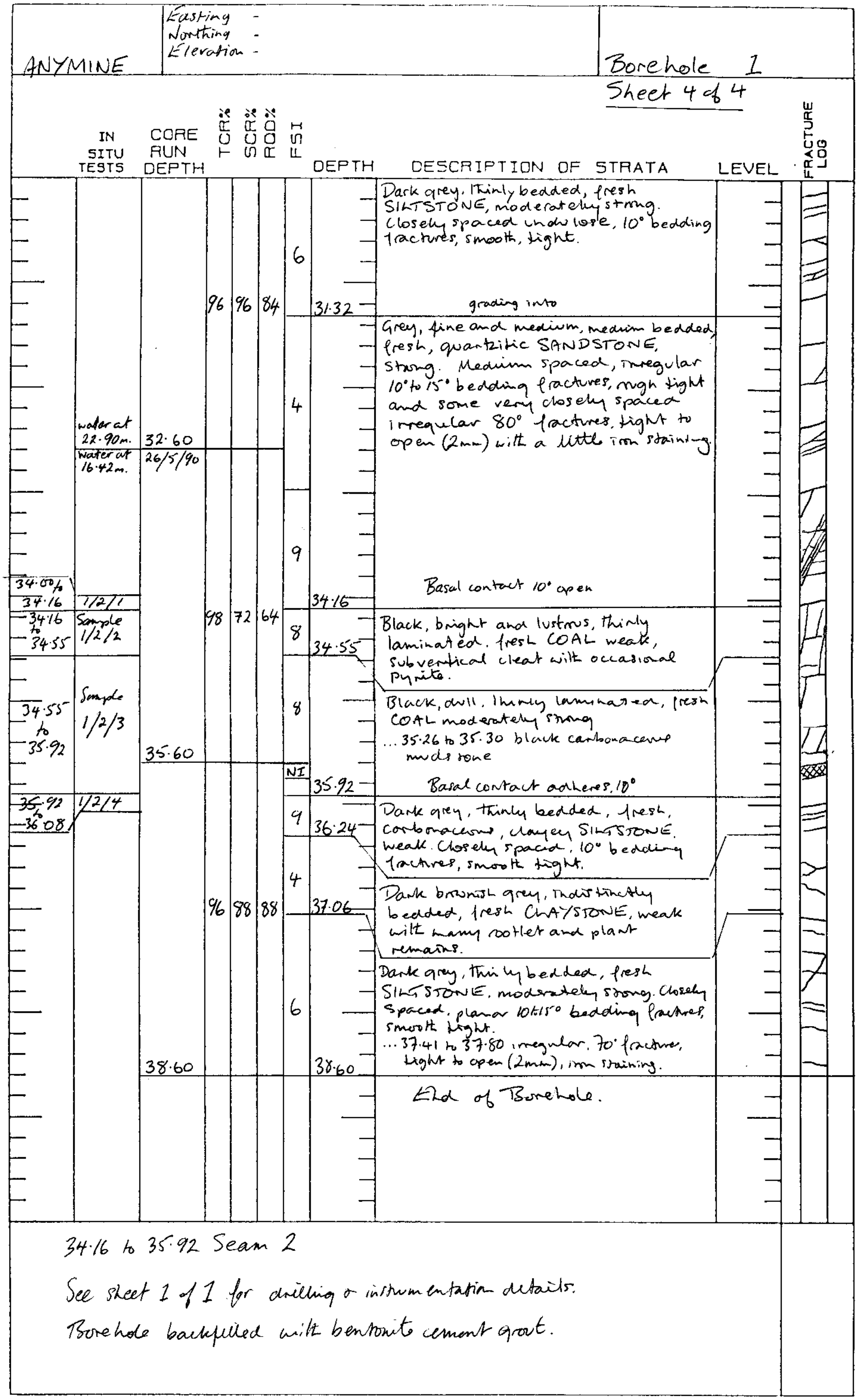
in the input and that the right type of process is selected to produce correct results.

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, that is a node, the data points closest to the node can be used. However, this can result in the selection of data from only one set of data points, for example a line of boreholes or a sample traverse. To reduce the bias, many programmes 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 pre-set 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 6.25 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 6.26a shows coal seam thickness of a single seam dipping to the southwest with the line of outcrop running northwest–southeast. The thickening of the coal shown by the bunching of the contours in the west and southwest is due to the coal seam splitting and including increasing thicknesses of interburden. This can be contrasted with Figure 6.26b, which shows seam thickness excluding any partings or interburden. Errors tend to show up as 'bulls eyes' on the plan. Where this occurs, the data for that location needs 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


ANYMINE
Easting -
Northing -
Elevation -
Borehole 1
Sheet 4 of 4

| IN SITU TESTS | CORE RUN DEPTH | TCR% | SCR% | RQD% | FSI | DEPTH | DESCRIPTION OF STRATA | LEVEL | FRACTURE LOG |
|---|---|---|---|---|---|---|---|---|---|
| | | | | | 6 | | Dark grey, thinly bedded, fresh SILTSTONE, moderately strong. Closely spaced undulose, 10° bedding fractures, smooth, tight. | | |
| | | 96 | 96 | 84 | | 31.32 | grading into | | |
| water at 22.90m. | 32.60 | | | | 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 16.42m. | 26/5/90 | | | | 9 | | | | |
| 34.00 to 34.16 | 1/2/1 | | | | | 34.16 | Basal contact 10° open | | |
| 34.16 to 34.55 | Sample 1/2/2 | 98 | 72 | 64 | 8 | 34.55 | Black, bright and lustrous, thinly laminated, fresh COAL weak, subvertical cleat with occasional pyrite. | | |
| 34.55 to 35.92 | Sample 1/2/3 | | | | 8 | | Black, dull, thinly laminated, fresh COAL moderately strong ... 35.26 to 35.30 black carbonaceous mudstone | | |
| | 35.60 | | | | NI | 35.92 | Basal contact adheres, 10° | | |
| 35.92 to 36.08 | 1/2/4 | | | | 9 | 36.24 | Dark grey, thinly bedded, fresh, carbonaceous, clayey SILTSTONE, weak. Closely spaced, 10° bedding fractures, smooth tight. | | |
| | | 96 | 98 | 88 | 4 | 37.06 | Dark brownish grey, indistinctly bedded, fresh CLAYSTONE, weak with many rootlet and plant remains. | | |
| | 38.60 | | | | 6 | 38.60 | 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. | | |
| | | | | | | | End of Borehole. | | |

34.16 to 35.92 Seam 2
See sheet 1 of 1 for drilling & instrumentation details.
Borehole backfilled with bentonite cement grout.

**Figure 6.24** Example of a geotechnical logging sheet. (Reproduced by courtesy of M. C. Coultas.)

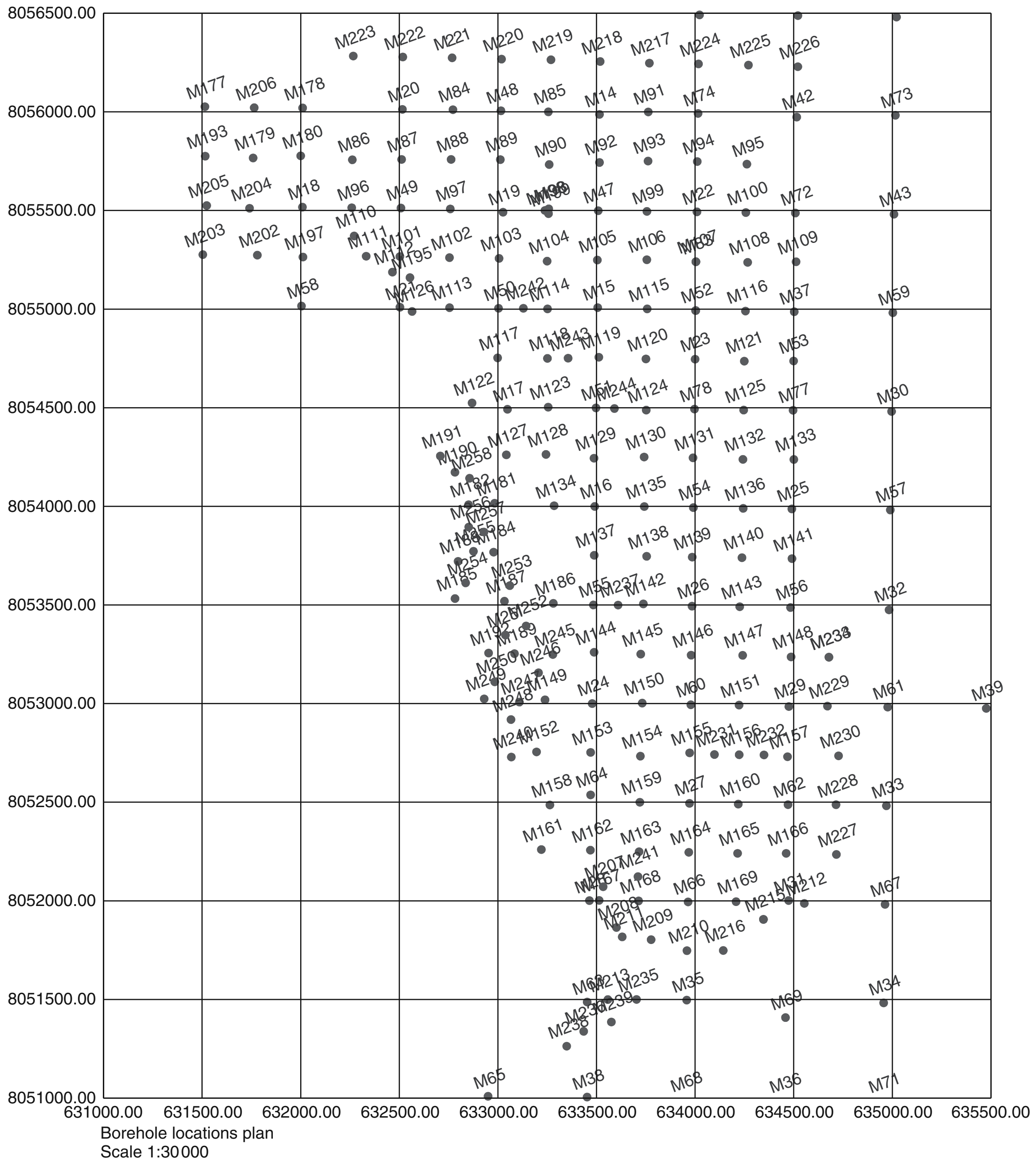



**Figure 6.25** Computer generated borehole location plan. (Courtesy of Dargo Associates Ltd.)

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 to determine the economic potential of the coal seam(s). Figure 6.27a shows a contour plan of total sulfur for the same coal seam as shown in Figure 6.26. The higher sulfur content values are concentrated along the southwestern edge of the plan, coinciding with the

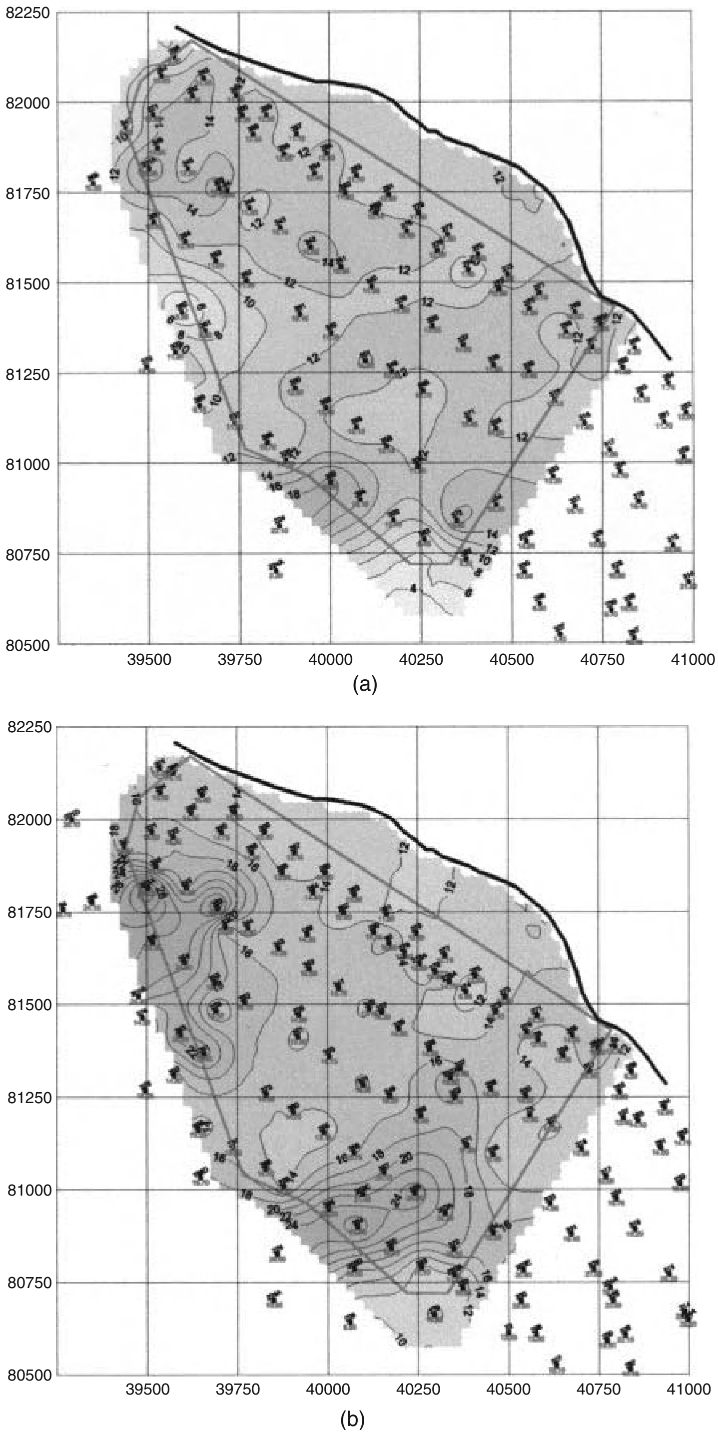



**Figure 6.26** (a) Coal thickness contour map with borehole locations. (b) Full seam thickness (including partings) contour map with borehole locations. (Reproduced by permission of Dargo Associates Ltd.) This figure is reproduced in colour in the Plates section.

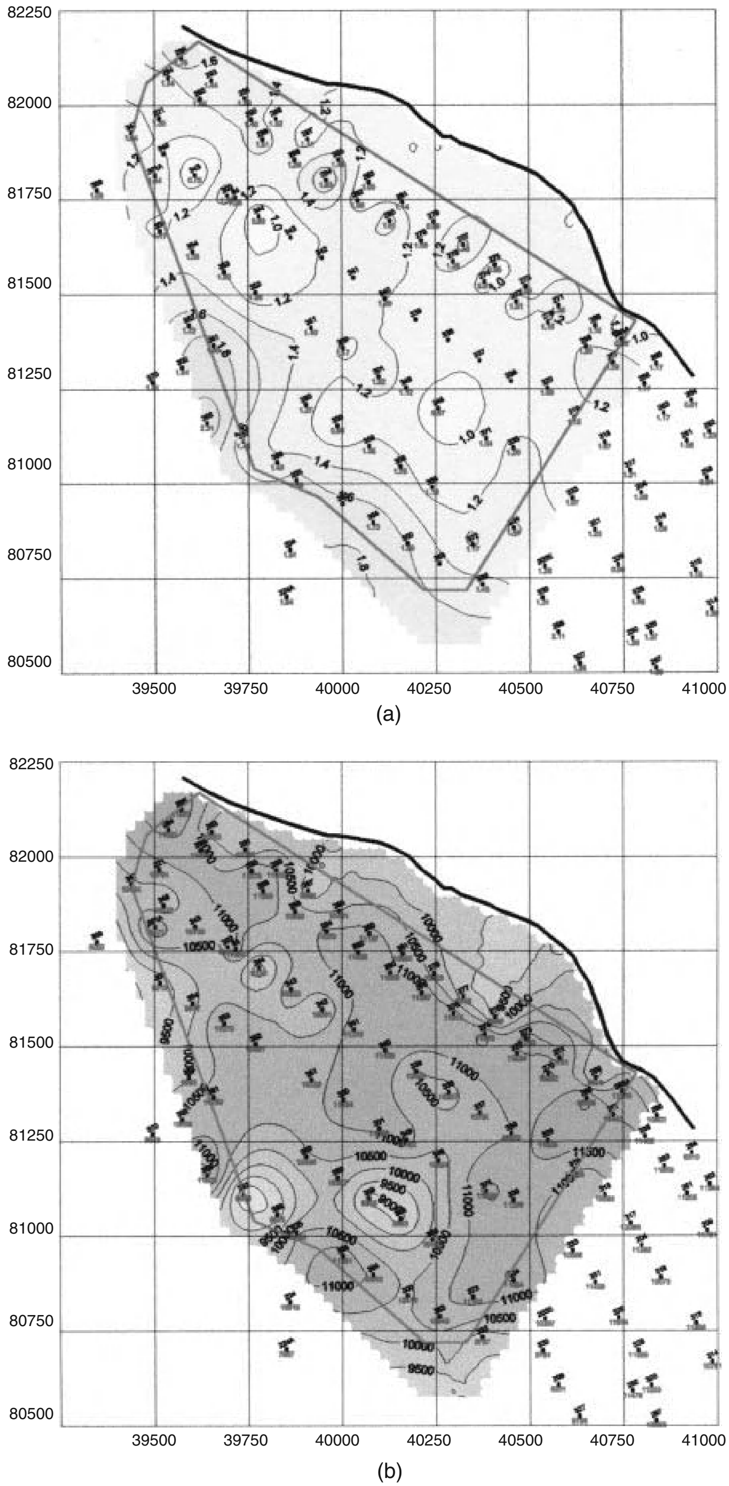



**Figure 6.27** (a) Total sulfur content contour map with borehole locations. (b) Net calorific value contour map with borehole locations. (Reproduced by permission of Dargo Associates Ltd.) This figure is reproduced in colour in the Plates section.

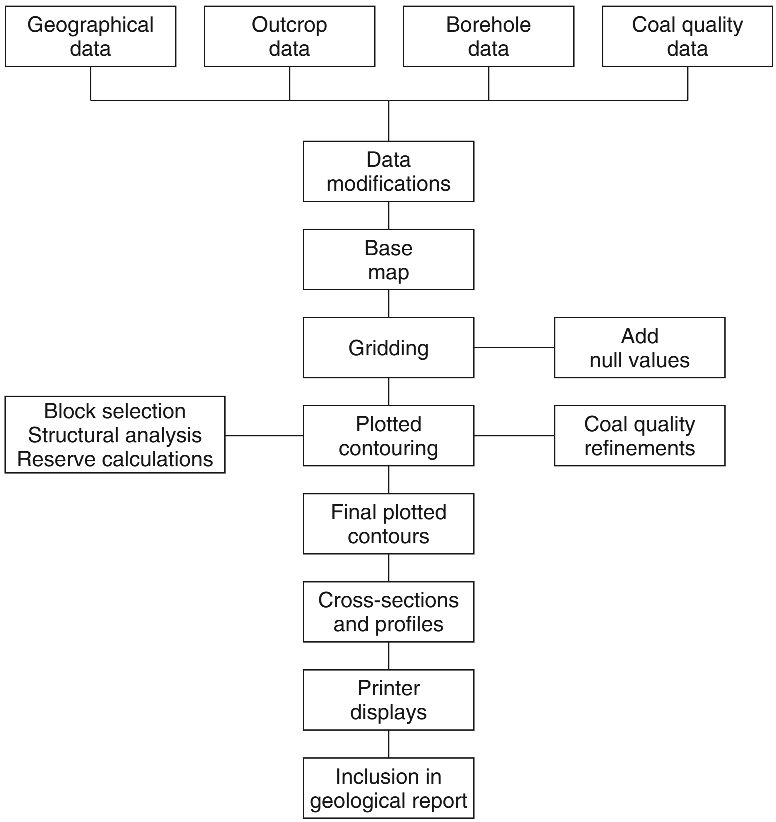



**Figure 6.28** Simplified flow diagram of a geological contouring program. (Reproduced by permission of Dargo Associates Ltd.)

increase in interburden within the seam. Figure 6.27b shows the variation in net CV for the same seam, and the lower net CV values again 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 programme 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 accumulated automatically so that the results for each coal seam are rapidly available. More complex volumetric programmes can be used, 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 illustrates the stages in the processing of geological field data to produce contour plans, profiles and reserve calculations (Figure 6.28).

# 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, namely according to:

1. their degree of geologic assurance;
2. their degree of economic feasibility.

There is a third subdivision, which 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, 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 geologic 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 concern is more with capacity and deliverability, whereas resource analysis is valid for longer term assessment, that is for 10 yr hence and longer.

## 7.2 Classification of coal resources and reserves

A number of countries and organizations 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, which 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.

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, with the current version dating from 2004. At the same time as the development of the 'JORC Code' a committee of the Council of Mining and Metallurgical Institutions from Australia, Canada, Chile, South Africa,

*Coal Geology*, Second Edition. Larry Thomas.

United States and United Kingdom – the Combined Reserves International Reporting Standards Committee (CRIRSCO) – reached a provisional agreement for the use of standard definitions for Mineral Resources and Ore Reserves. 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). These standards, codes and guidelines are now published and adopted by the relevant professional bodies in the above countries and other countries in Europe. The use of such standards are 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 Southeast 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.

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. Details of the principal resource and reserve classifications currently in use are given below.

Documentation detailing exploration results, coal resources and coal reserves estimates from which a Public Report is produced, must be prepared by, or under the direction of, and signed by a Competent Person or Persons. A Competent Person must be a professional member of an approved Institution 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 can be another technical expert.

### *7.2.1 Australia*

The JORC Code had been adopted by the Australasian Institute of Mining and Metallurgy (The AusIMM) and the Australian Institute of Geosciences (AIG) and has been incorporated in the listing rules of the Australian and New Zealand stock exchanges. Although other variations of resource and 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.

The main principles governing the application of the JORC Code (2004) are, clarity of report, inclusion of all relevant data pertaining to the judgement of the status of mineral resources and ore reserves, and competency; that is, carried out by suitably qualified and experienced persons approved by a recognized 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.

1. **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 and openhole cuttings, and downhole geophysical logs. Points of observation for coal quality are obtained from exposure and/or borehole core sampling, with the latter having an acceptable level of recovery (95% linear recovery).
2. **Interpretive Data** are observations supporting the existence of coal and include results from mapping and geophysical surveys.
3. **Exploration Results** are reports of coal occurrences that, due to insufficient data, cannot be given a resource value or quality parameters.

The relationship between mineral resources and mineral reserves is shown in Figure 7.1. The JORC Code (2004) gives the following classification.

1. *Coal resources*:
   (a) A **Resource** is an occurrence of coal in such form, quality and quantity that there are reasonable prospects for eventual economic extraction.
   (b) **Inferred Resource** is that part of a coal resource for which tonnage and quality 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 coal resources may be estimated using points of observation up to 4 km apart.
   (c) **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. Indicated resources may be estimated using data obtained from points of observation normally less than 1 km apart. Trends in coal thickness and quality should not be extrapolated more than half the distance between points of observation.

(d) **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 is to 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. Measured coal resources may be estimated using data obtained from points of observation normally less than 500 m apart. This distance may be extended if the competent person considers that any variation to the estimate would be unlikely to significantly affect potential economic viability. Coal resource estimates are not precise calculations, as they are dependent upon the interpretation of limited information on the extent, continuity and thickness of coal seams and their quality.

**2.** *Coal reserves*:

(a) A **Coal Reserve** is the economically mineable part of a measured and/or indicated resource. It includes diluting materials and allowances for mining losses.

(b) **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. Factors such as mining method, marketing, legal, environmental, social and governmental have been assessed to demonstrate that at the time of reporting, coal mining can be justified.

(c) **Proved Coal Reserve** is the economically mineable part of a measured coal resource. Allowances are made for losses in mining plus the factors given for probable coal reserves. Proved coal reserves have been assessed to demonstrate that at the time of reporting, coal mining can be justified. Proved coal reserve is the highest confidence category of coal reserve estimates.

(d) **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.

### *7.2.2 Canada*

The Canadian Institute for Mining, Metallurgy and Petroleum (CIM) approved the CIM Standards on Mineral Resources and Reserves – Definitions and Guidelines in 2000. This was updated in 2010 to reflect the more detailed guidance available, and maintain consistency with current regulations.

The mineral resource and reserve categories laid down are similar to those in the CRIRSCO Template and JORC Code (2004) 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 (2004) (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. For coal mining and other mineral properties it is the National Instrument (NI) 43–101. Guidelines are also set out in the Geological Survey of Canada Paper 88–21, 'A standardised coal resource/reserve reporting system for Canada', (Hughes, Klatzel-Mudry and Nikols, 1989), but these must be in accordance with the CIM definition categories for public reporting. 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 NI 43–101.

All of the 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 contrast, CIM guidelines consider that a comprehensive study of the viability of the coal project must have been performed, which includes mining method, pit configuration and coal preparation (if required) and demonstrates that economic extraction is justified.

### *7.2.3 Europe (including United Kingdom)*

The Pan European Reserves and Resources Reporting Committee (PERC) Code (2008) sets out the minimum standards, recommendations and guidelines for public reporting of exploration results, mineral resources and mineral reserves in the United Kingdom, Ireland and

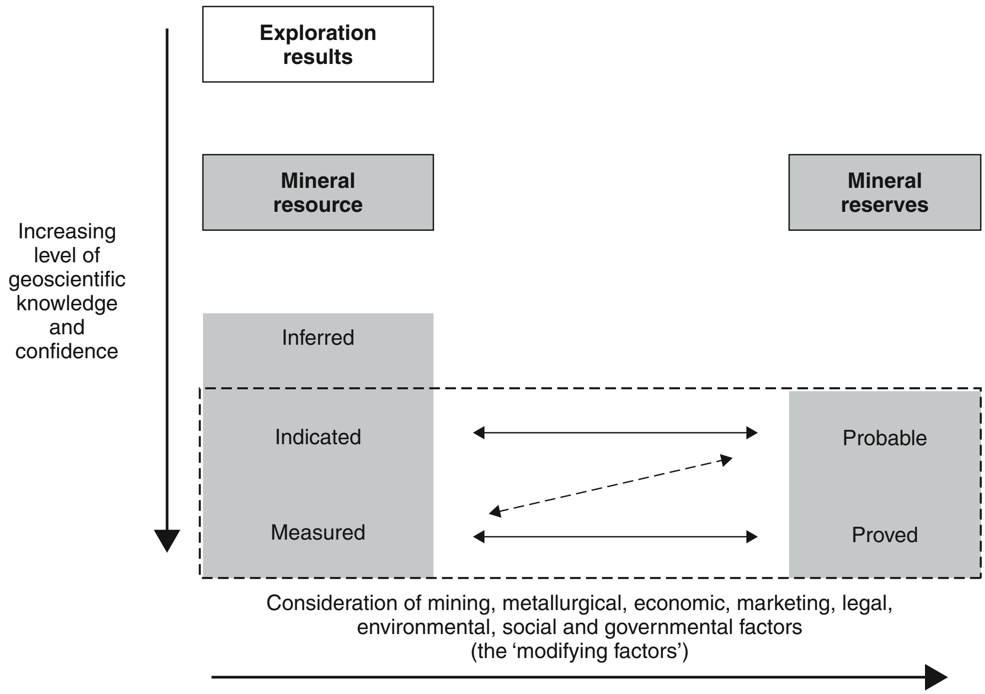



**Figure 7.1** Relationship between mineral resources and mineral reserves. (Reproduced with permission of PERC and CRIRSCO.)

Europe. 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. The PERC Code draws from and is consistent with the CRIRSCO Template and national codes from which it is derived. The definitions in the PERC Code are either identical to or not materially different from those international definitions, and the relationship of coal resources and coal reserves again conform, as shown in Figure 7.1.

As in the case of the JORC Code, coal resources are subdivided in order of increasing geological confidence, that is inferred, indicated and measured categories. Specific distances between points of observation are not given, but in the case of indicated coal resources, 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.

Prior to the PERC Code, countries such as the United Kingdom and Germany used classifications developed by their publicly owned coal industries, chiefly for the underground mining of black coal (Cook and Harris, 1998). These industries have now disappeared and private coal mining companies needing to finance mining operations are working to CRIRSCO Codes requirements.

### 7.2.4 South Africa

In South Africa, the South African Code for Reporting of Mineral Resources and Mineral Reserves, the SAMREC

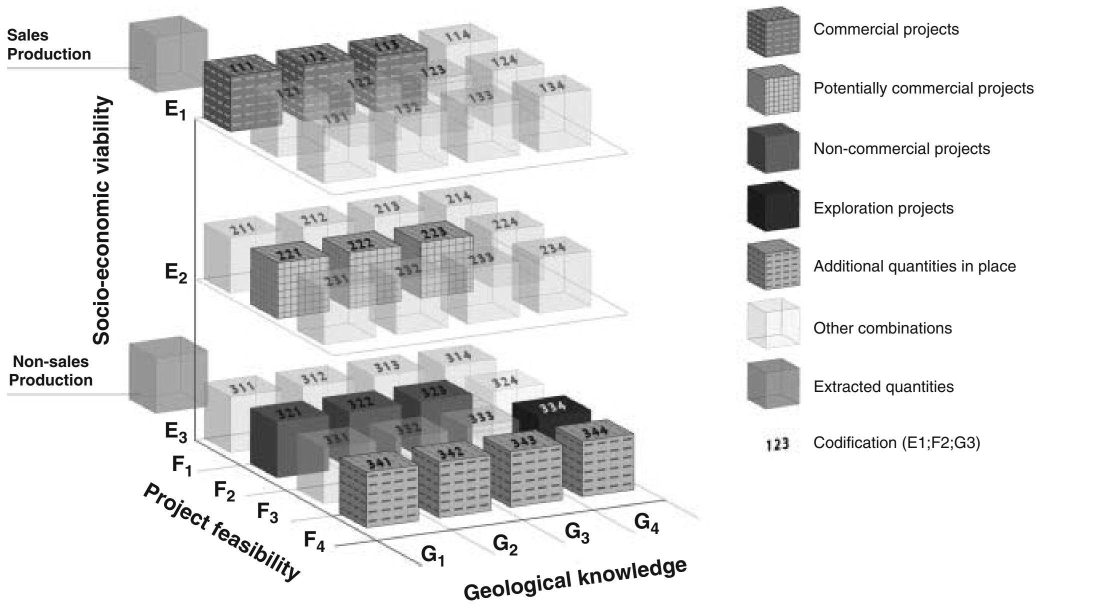



**Figure 7.2** UNFC-2009 resource and reserve categories and examples of classes (UNFC, 2009). This figure is reproduced in colour in the Plates section.

Code (2009), is applicable to all minerals for which public reporting of exploration results, mineral resources and mineral reserves is required. This is required by the Johannesburg Stock Exchange. Again the SAMREC Code is virtually identical to the JORC and CIM Codes.

### *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 United Nations framework Classification for Fossil Energy and Mineral Resources which would have worldwide application. This has resulted in the development of the United Nations Framework Classification for Fossil Energy and Mineral Reserves and Resources (UNFC-2009). This 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.

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 three-dimensional 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 into categories:

- E1 – extraction and sale of product has been confirmed to be economically viable;
- E2 – extraction and sale of product is expected to become economically viable in the foreseeable future;
- E3 – extraction and sale of product is not expected to become economically viable in the foreseeable future or evaluation is at a 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 into categories:

F1 – feasibility of extraction by a defined development project or mining operation has been confirmed;
F2 – feasibility of extraction by a defined development project or mining operation is subject to further evaluation;
F3 – 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, which in this case is 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 categories:

G1 – quantities and qualities associated with a known deposit that can be estimated with a high level of confidence.
G2 – quantities and qualities associated with a known deposit that can be estimated with a moderate level of confidence.
G3 – quantities and qualities associated with a known deposit that can be estimated with a low level of confidence.
G4 – estimated quantities and qualities associated with a potential deposit, based primarily on indirect evidence.

Unlike JORC (2004), UNFC-2009 appears to give no precise definition of the levels of concentration of data points for each G category.

In addition to the E, F and G categories, subcategories are represented by a numerical code, 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, and are summarized in Table 7.1.

### *7.2.6 United States*

In 1999, in association with the other members of CRIRSCO, the Society for Mining, Metallurgy and Exploration (SME) produced the Guide for Reporting Exploration Information, Mineral Resources and Mineral Reserves. The definitions given are again in broad agreement with the 'JORC Code', however, there are differences between the content of the SME Guide and the US Securities and Exchange Commission (SEC), who have not accepted the internationally recognized concepts of reporting mineral resources and the signing off of technical reports by a competent person.

Prior to the international rationalization of the reporting of mineral resources and reserves, a coal resource classification system was published by the US Geological Survey (Wood *et al*., 1983). The system is based on the concept by which coal is classified into **Resource, Reserve Base** and **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 for 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 heirarchy of coal resources and reserves categories outlined by the US Geological Survey is given in Figure 7.3, and the application of the reliability categories based on distance from points of measurement, that is coal outcrops and boreholes, by the US Geological Survey is demonstrated in Figure 7.4.

1. **Original Resources** represent 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.
2. **Remaining Resources** include all coal after coal produced and coal lost in mining is deducted.

3. **Identified Resources** are those resources for which 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.
4. **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.
5. **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.
6. **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 to be used. The reserve base includes coal categories based on

**Table 7.1** UNFC-2009 resource/reserve 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[a] | 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][b] | 3.2 | 3 | 4 |
| | | Additional quantities in place | | 3.3 | 4 | 4 |

[a] Development pending projects may satisfy the requirements for E1

[b] Generic sub classes have not been defined here, but it is noted that in petroleum the terms prospect, lead and play are commonly adopted.

Source: UNFC (2009).

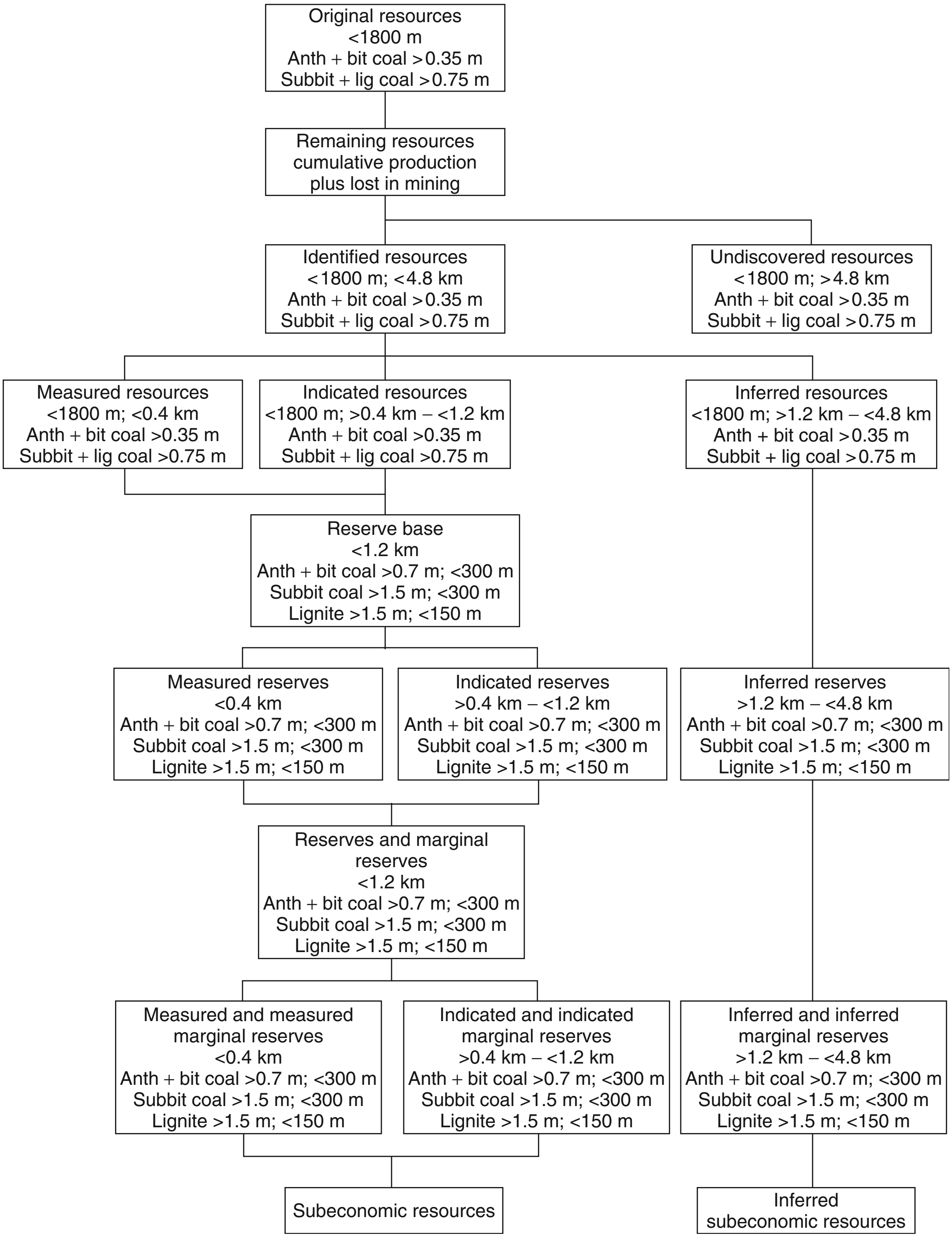



**Figure 7.3** Criteria for distinguishing coal resource categories, adapted from US Geological Survey hierarchy of coal resources (Wood *et al.*, 1983): anth = anthracite; bit = bituminous; subbit = subbituminous; lig = lignite.

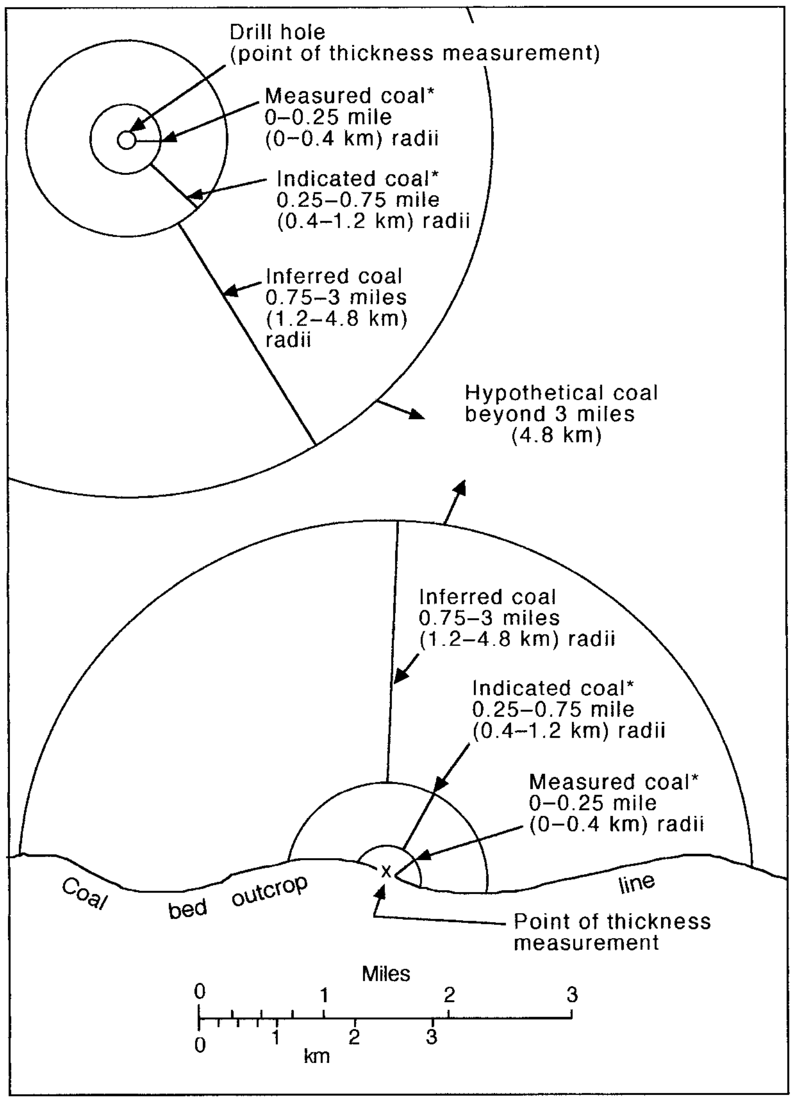



**Figure 7.4** Diagram showing reliability categories based solely on distance from points of measurement. *Measured and Indicated coal can be summed to demonstrated coal. (From Wood *et al.*, 1983.)

the same distance parameters given for coal resources but further defining the coal thickness and depth criteria: that is anthracite and bituminous coal to be 0.7 m or more and subbituminous coal to be 1.5 m or more thick, and to occur at depths not more than 300 m; lignite to be 1.5 m or more thick at depths not greater than 150 m.

1. **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.
2. **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.
3. **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.

4. **Marginal Reserves** are those reserves that border on being economic, that is 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.
5. **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*

The Russian system for reporting estimates of coal resources and reserves has a different objective from the CRIRSCO Template. The Russian system comprises 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 contrasts with the CRIRSCO 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. The Russian standard of public reporting of exploration results, resources and reserves of solid minerals including coal, has not been included in the range of internationally accepted standards and codes. Dixon (2010) has produced guidelines on the alignment of Russian minerals reporting standards and the CRIRSCO Reporting Template.

As in the case of other resource reporting systems, the Russian Federal Government Agency State Commission on Mineral Reserves (FGU GKZ) requires an independent audit on resource estimates to be carried out by recognized GKZ competent persons. These are normally members of the Society of Experts of Russia for Mineral Resource Usage (OERN), which is a member organization of the European Federation of Geologists (EFG).

The Russian classification system and its relationship to the various levels of exploration and investigation is shown in Table 7.2. According to the level of geological knowledge, the Russian system identifies four levels of resource. These are in order of decreasing geological knowledge, A, B, $C_1$ and $C_2$, which correspond to coal resource categories in the CRIRSCO Reporting Template. The CRIRSCO measured and indicated resources are equivalent in terms of definition to Russian resource categories A, B, $C_1$ and $C_2$ based only on the level of geological knowledge. The Russian system also has additional categories of 'prognostic resources' designated $P_1$, $P_2$ and $P_3$. These include resources less well known than $C_2$ 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 1st (simplest) to 4th (extremely complex). Category A resources are only in 1st level of geological complexity, category B resources are in 1st and 2nd levels of geological complexity, category $C_1$ can include 1st, 2nd and 3rd levels of geological complexity and where areas have been studied in detail, can include the 4th level of geological complexity. Category $C_2$ 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, that is a pre-feasibility study, and a higher level of detail required for a TEO of permanent conditions, that is a full feasibility study. These two 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 that are economically exploitable, and 'off balanced' resources that 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 Reporting Template (Dixon, 2010). Figure 7.5 summarizes the relationship between the Russian classification and the CRIRSCO Reporting Template (Dixon, 2010).

It should be noted that the countries of Eastern Europe that have previously used the Russian Classification are now reviewing their classifications and some are recommending the adoption of the UNFC-2009 Code, for example Serbia (Ilic *et al*., 2009). The conversion of coal resources and reserves estimated according to the Russian system as equivalents in the CRIRSCO Reporting Template, for example the JORC Code, again requires the signature of a Competent Person.

### *7.2.8 Peoples Republic of China*

The traditional Chinese resource classification was developed from the former USSR system, and is now phased out. This classification used categories A to F based

**Table 7.2** Russian classification system related to stage of study Dixon (2010) Reproduced with permission of GKZ and CRIRSCO.

| Exploration stages | Final documentation of stage completion | Resource/reserves, categories | | Prognostic resources, categories | | |
|---|---|---|---|---|---|---|
| | | A, B, C1 | C2 | P1 | P2 | P3 |
| Detailed exploration work | **TEO of 'permanent conditions'**<br>**Russian reserves estimation report** | **'Fully explored' deposit**<br>1. Balance (economic)<br>2. Off-balance (potentially economic) | 1. Balance (economic)<br>2. Off-balance (potentially economic) | Not considered at this stage | | |
| Deposit delineation (Estimation) work | **TEO 'provisional conditions'**<br>**Russian reserves estimation report** | **'Estimated' deposit**<br>1. Balance (economic)<br>2. Off-balance (potentially economic) | 1. Balance (economic)<br>2. Off-balance (potentially economic) | Not considered at this stage | | |
| Prospecting work | **Preliminary estimation**<br>(TES (scoping study) on geologically justified concepts on the sizes and characteristics of known mineralization) | 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, mineralization fields, etc.** | |

↑ Increasing level of technical and economic knowledge

← Increasing level of geological knowledge and confidence

Source: Dixon (2010).

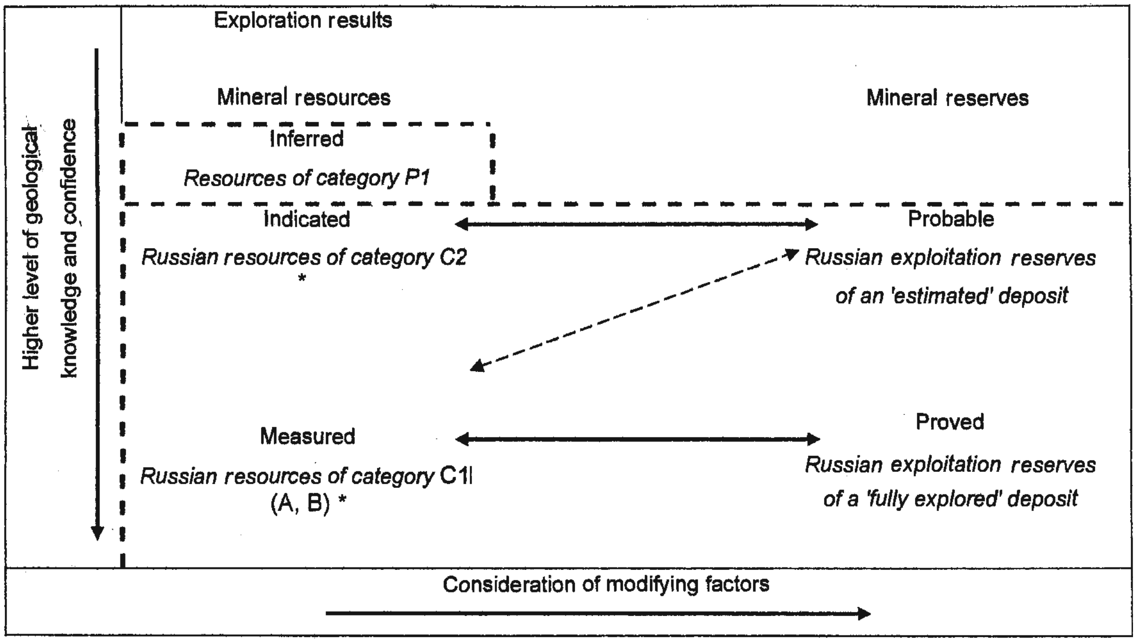



**Figure 7.5** A partial mapping of the Russian and CRIRSCO classifications Dixon (2010) Reproduced with permission of GKZ and CRIRSCO.

on decreasing levels of geological confidence. This was replaced with the 1999 Chinese Mineral Resource/Reserve Classification, which was based in part on the 1999 United Nations Framework Classification of Fossil Energy and Mineral Resources (UNFC), which included the incorporation of CRIRSCO definitions (in 1998). This classification was reviewed in 2007/2008 as the current Chinese Classification System is 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 classifies resources and 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 contains 16 categories, which are referenced by a three digit number in the same EFG notation as outlined in the UNFC Classification (see section 7.2.5). Figure 7.6 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 has, 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 has also been a category of '**Undiscovered** resources', which has no equivalent in the CRIRSCO Codes.

The requirement of a competent person to sign off on resource and reserve reporting is not applicable in the Peoples Republic of China. However, where foreign investors are involved, a competent person must provide a comparison between resource estimates that are non-compliant with CRIRSCO Codes such as the JORC Code. Shen Zhou Mining and Resources (2007), and Bucci (2011) have produced comparison tables for the Chinese classification and the JORC Code (2004) (see Table 7.3). The Chinese classification system is currently under review to simplify its numerous categories and bring it into line with the CRIRSCO and JORC Codes (Stoker, 2009).

### *7.2.9 India*

India coal resources are reported on the basis of the Indian Standard Procedure 1956. The Geological Survey of India locates coal-bearing areas and has assessed and classified coal into categories, inferred, indicated and proven. The Central Mine Planning and Design Institute then converts such 'indicated' reserves into the 'proven'

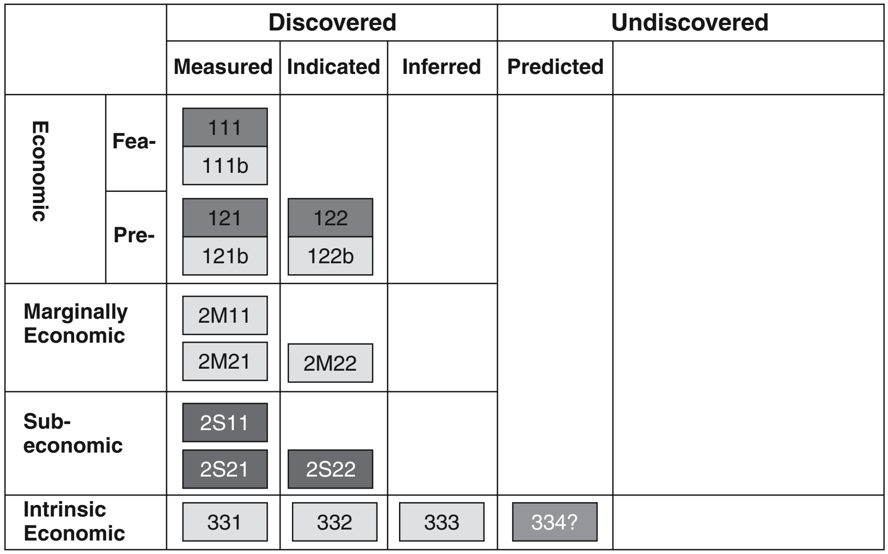



**Figure 7.6** Chinese mineral resources and reserves classification (Stoker, 2009).

**Table 7.3** Comparison of Chinese resource classification and JORC (2004).

| Old classification | | A & B | | C | | | D | E & F |
|---|---|---|---|---|---|---|---|---|
| New classification | | | | | | | | |
| "E" economic evaluation (100) | Designed mining loss accounted | Recoverable reserve (111) | Probable recoverable reserve (121) | | Probable recoverable reserve (122) | | | |
| | Designed mining loss not accounted (b) | Basic reserve (111 b) | Basic reserve (121 b) | | Basic reserve (122 b) | | | |
| Marginal economic (2M00) | | Basic reserve (2M11) | Basic reserve (2M21) | | Basic reserve (2M22) | | | |
| 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) |
| JORC | | - | | | | | Unclassified or Exploration Potential | |
| | | | | | Inferred | | | |
| | | | Proved/probable reserve OR Indicated resource | | | | | |
| | | Proved/probable reserve OR measured resource | | | | | | |

Source: Bucci (2011) and Shen Zhou Mining and Resources (2007).

category through detailed exploration. This is a geological resource classification without the assessment of coal quality or mineability of the coal. It has included coal resources that are unavailable for exploitation or areas sterilised by fire, etc.

Chand and Sarkar (2006) stated that India is 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 has stated that the total coal inventory is not so important, what is important is how technically feasible it is to mine the coal. The TERI suggests that India should adopt the UNFC-2009 Code for resource and reserve classification (Kulkarni, 2011). This will give a more realistic assessment of India's coal resources and reserves related to economic and technical viability.

## 7.3 Reporting of resources and reserves

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, for example openhole, 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. Details on each coal seam within the lease area.
2. On a depth basis, in regular depth increments if sufficient information is available.
3. 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, for example 1.0 m.
4. 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 utilization, should be given, for example total sulfur and calorific value. Subdivisions of the resources may be made for areas of oxidized coal and heat-affected coal.

A summary of all the factors relevant in determining the categories of coal resources and reserves assessment is shown in Figure 7.7. 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 and 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 and reserves, seam depth contours, seam isopachs, quality contours for each seam and all areas not to be mined.

The geological information required for resource and reserve assessments are 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 and 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 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 and reserve categories are 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. 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.

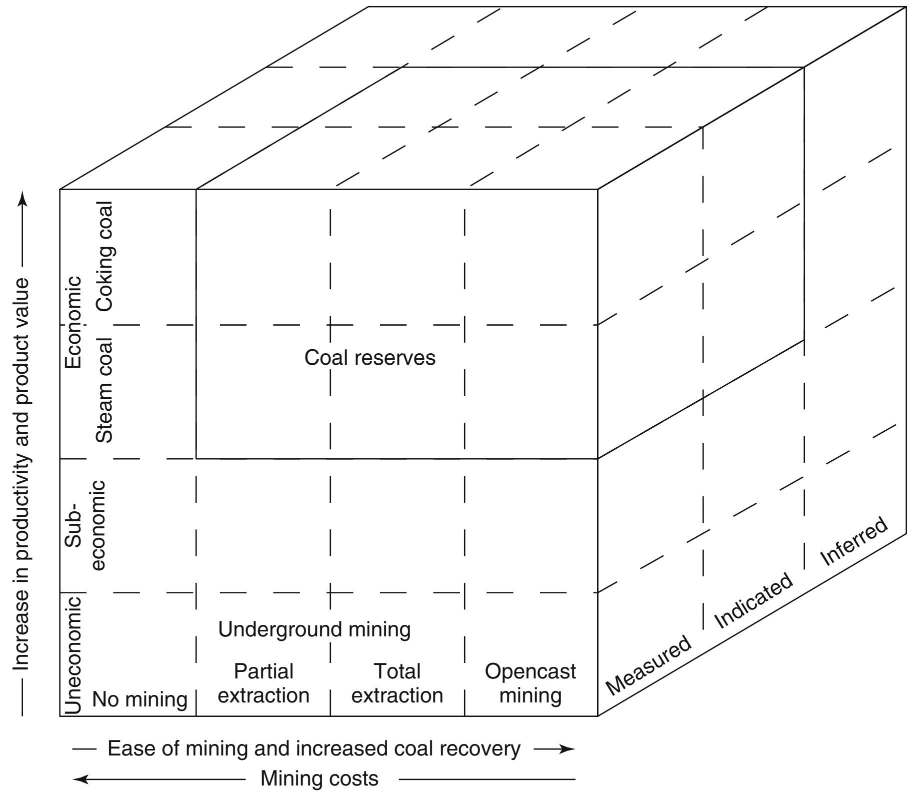



**Figure 7.7** Variable factors in coal reserves assessment. (Adapted from Ward, 1984.)

### *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/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 due diligence on the project. Therefore it is important for the coal geologist to be able to undertake 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 (for each defined resource/reserve block):

$$\text{coal thickness (m)} \times \textit{area} \times \text{RD} = \text{total metric tonnes},$$

where RD stands for relative density. Coal thickness is determined at each point of measurement.

The area of each resource/reserve block is measured on the map or plan by either of the following ways.

1. 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 7.8 shows such a planimeter being used in this fashion.
2. The co-ordinates of each block area are entered into a computer program specified for the purpose, and an areal calculation is obtained.

The relative density is normally taken from a total seam section, that is, 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 can be adopted dependent upon the known average ash content of the seam, this would be


**Figure 7.8** Digital planimeter being used to calculate coal reserve areas within a working mine. (Photograph by M.C. Coultas.)

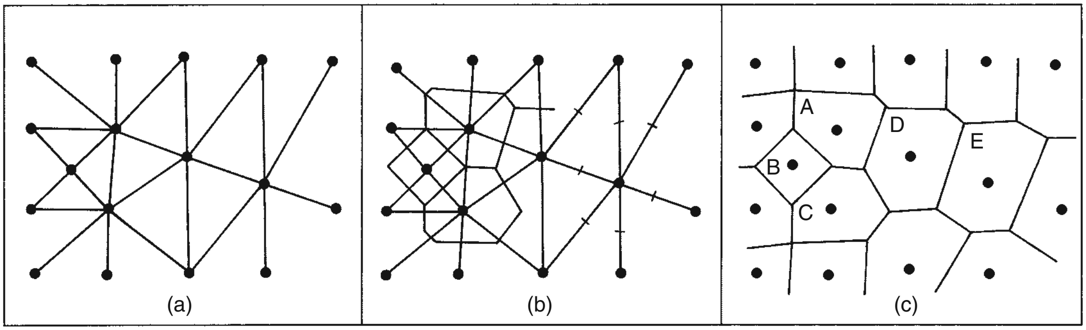
in the order of 1.3–1.4 in coals with reasonably low ash contents. This means that an area of 1 km$^2$ ($1 \times 10^6$ m$^2$) underlain by a coal with a thickness of 1.0 m and with a density of 1.4, will contain $1.4 \times 10^6$ t of coal. Resource tonnages are usually quoted in million tonnes (Mt) and are usually rounded off to the nearest 100,000, 10,000 or 1000 t 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 mid points between the point of measurement and those surrounding it, resulting in the original point of measurement forming the centre of the polygon. Figure 7.9 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 reserves, mistakes can occur, for example, in the construction of blocks of influence for a triangular grid, Figure 7.10(a) shows the correct selection of areas of interest, whereas Figure 7.10(b) illustrates 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 reserves; also widely spaced drill holes leave uncertainty as to the continuity of coal seams. Polygons based on widely spaced data points gives no indication of the accuracy of the results, and it is possible that the actual and calculated reserves may differ by significant amounts.

A practical development of the polygon method is to outline reserve 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 reserves may lie within the boundary zone. The calculated reserves therefore will be unreliable because of their dependence on the subjective selection of the boundary zone.

One method of improving the reliability of the boundary width for a deposit is to estimate 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




Calculation of reserves for polygons A – E:

| | Area ($m^2$) | 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 7.9** Polygon method for calculation of *in situ* coal resources and 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, for example 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).

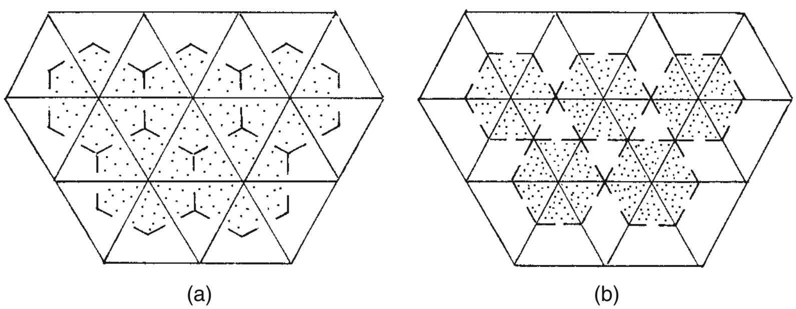



**Figure 7.10** Construction of polygons (blocks) of influence for an equiangular triangular grid. (a) Correct construction of elementary blocks. (b) Incorrect construction of elementary blocks. (From Tasker, 1985.)

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, this method is not applicable.

In an area that has been drilled extensively, detailed contouring of the coal seams can enable reserve estimates to be made as a further definition of the estimated reserve figure; this method is also useful to highlight areas of geological hazard such as washouts or faulting.

The use of computers now allows the input of geological data into a geologically significant computer model of the deposit. In this way coal deposits can be subdivided into major block units, each equating to the proposed working district of a coal mine. Within these units, smaller

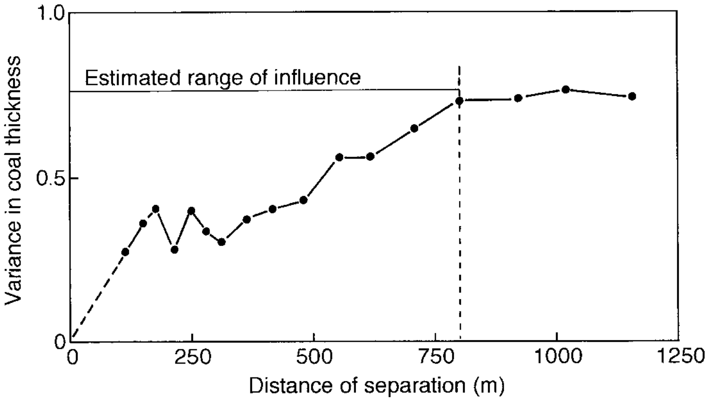



**Figure 7.11** Variogram of coal thickness. (From Ventner, 1976.)

structurally delineated blocks provide the basic geological data, which are transformed into computer data. Within each of these smaller units the coal seams provide the basic data for the whole assessment,, that is depth, thickness, dip and size of the area. For example, the minimum coal thickness to be evaluated could be taken as 0.30 m, while 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).

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 is digitized 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 digitizing 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.

In Germany, where this method has been developed, 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 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, but such conditions are only likely to occur in fairly well-defined coal deposits.

### *7.3.3.2 Opencast coal mining*

***Stripping ratio calculation***

In opencast mining operations, overburden to coal ratios are often quoted on a volumetric basis, that is bank (*in situ*) cubic metres (BCM) of overburden per tonne of coal *in situ*. The calculation of the stripping ratio (SR) is:

$$\text{stripping ratio} = \frac{\text{overburden cubic metres (BCM)}}{\text{coal cubic metres} \times \text{coal RD}},$$

where RD is Relative density. For coal deposits where the relative density of the coal is essentially constant, the stripping ratio is expressed simply as 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 stripping ratios are most conveniently calculated by computer and these data can be plotted later as stripping ratio contour plans.

In the United Kingdom overburden and stripping ratios are calculated somewhat differently, and can be

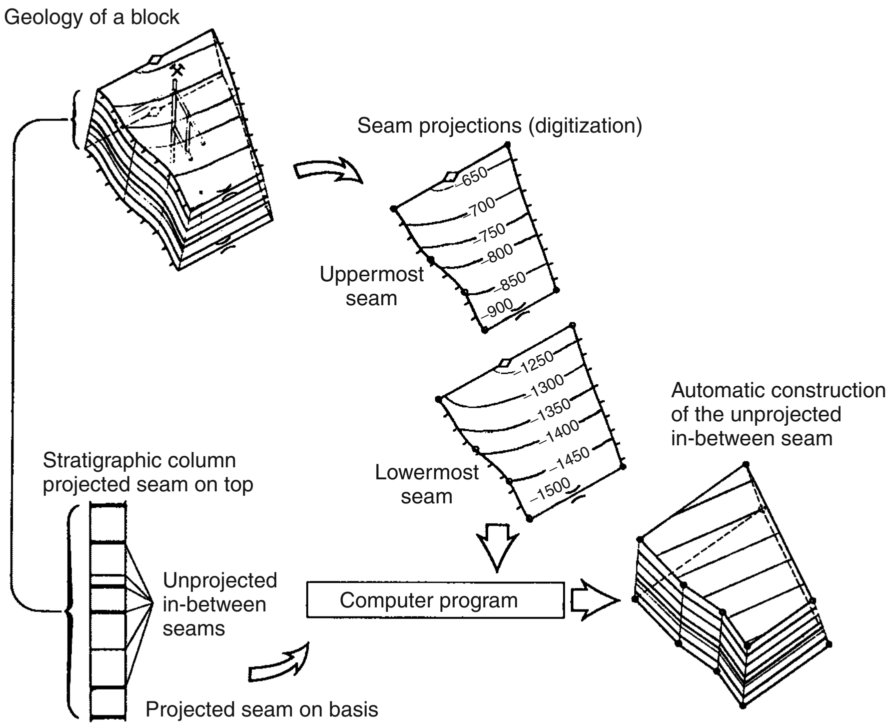



**Figure 7.12** Schematic diagram of the calculation of sectional areas using the block model for coal resource/reserve calculations. (From Juch *et al.*, 1983.)

considered as either:

overburden ratio (without batters)
= *in situ* vertical ratio

or

actual stripping ratio (including batters)
= working ratio.

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:

overburden (*in situ* vertical) ratio = *in situ* overburden thickness (including coal thickness) − *in situ* coal thickness × *in situ* coal thickness.

The overburden ratio can also be calculated on a volumetric basis:

overburden (*in situ* vertical) ratio = *in situ* overburden volume (including coal volume) − *in situ* coal volume × *in situ* coal volume

For mining a multiple coal seam sequence, the interburden, that is 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 stripping ratio. The stripping ratio selected for any coal deposit is based on the economics of removing and rehandling overburden, coal quality and geotechnical limitations.

### *Depth of planned opencast mining*

In the United Kingdom a useful equation used for the rapid estimation of the highwall depth is

highwall depth = 2[(overburden ratio × *in situ* thickness) + *in situ* thickness] − low wall depth

and for estimating the average depth of the coal excavation area

coal excavation area average depth = highwall depth + low wall depth × 2.

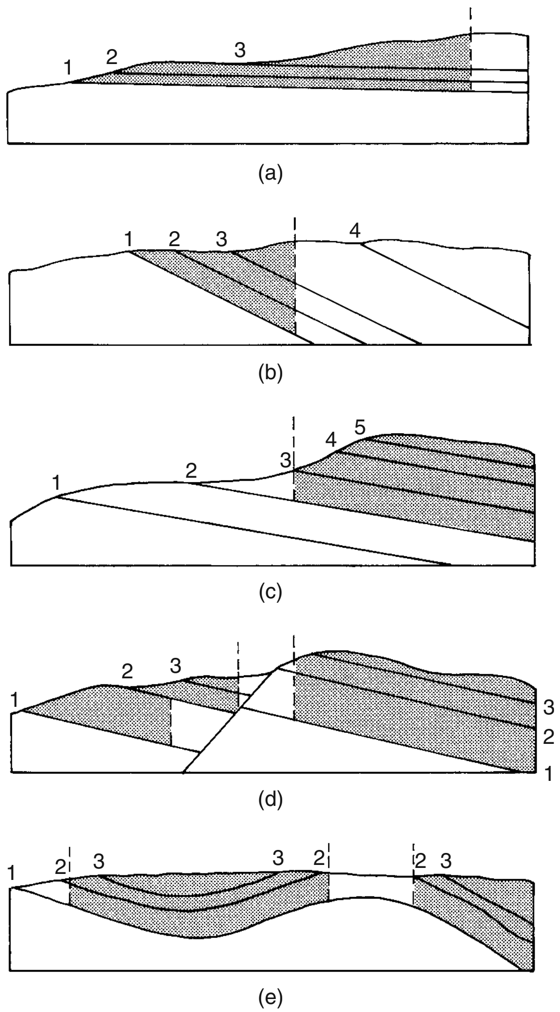


**Figure 7.13** Effects of geological structure and topography on stripping ratios. For this example, the SR cut-off value is taken at 10:1 BCM; only shaded area(s) will be considered for mining. Coal seams are numbered in ascending order: (a) effects of shallow dips; (b) effects of steep dips; (c) effects of topography; (d) effects of faulting; (e) effects of folding.

### *7.3.3.3 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 stripping ratio. 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 is too small, then the whole block may be discounted. If the bord and pillar (= 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 coal mines. 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, in opencast operations a 10% geological loss can be expected, whereas in underground operations, geological losses of 25–50% may occur.

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.

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.

### *7.3.3.4 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 or stripping ratios 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, for example beneath motorways;
2. stress fields in the coal seams, which may require the reorientation 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.

#### 7.3.3.5 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, for example opencast pit walls that 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, for example 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:

1. physical (coal quantity and quality);
2. rate dependent relationships (quantity of coal A per quantity of coal B);
3. 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 coal, that is for anthracite and bituminous coals, and for brown coal, that is subbituminous coal and lignite, are given in Table 7.4. These figures are taken from the World Energy Council (WEC) Survey of Energy Resources (WEC, 2010) and from the BP Statistical Review of World Energy (BP, 2011). These proven reserves are those that can be regarded with reasonable certainty to be recoverable from known deposits under existing economic and operating conditions, and does not include those large resources of coal that are currently being, or will in the future, fully evaluated.

The table gives an overall total of 860,938 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 (404,762 Mt) and brown coal (456,176 Mt) resources. Those countries with $1 \times 10^6$ t 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 7.4). The distribution of proven reserves shows little change over the past 20 yr, the chief increases being in the Asia Pacific region.

### 7.4.2 World coal production

#### 7.4.2.1 Coal production statistics

Production figures for black and brown coals are shown in Table 7.5 for the six world regions for the years 2007–2010. Total world coal production based on 2010 figures, is 3731.4 Mt oil equivalent, based on the BP Statistical Energy Review 2011. 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 Peoples Republic of China is the largest producer of black coal and Germany produces the greatest tonnage of brown coal.

In 2010, world hard coal production increased by 6.8% compared with 1.8% in 2009. Growth in hard coal production from non-OECD countries reached 8.4%.

**Table 7.4** World Coal Reserves (WEC 2010) & BP Statistical Review of World Energy (2011).

| Region | Country/region | Bituminous + anthracite | Subbituminous + lignite | Total | R/P ratio |
|---|---|---|---|---|---|
| North America | United States | 108501 | 128794 | 237295 | 241 |
| | Canada | 3474 | 3108 | 6582 | 97 |
| | Mexico | 860 | 351 | 1211 | 130 |
| | Total | 112835 | 132253 | 245088 | 231 |
| South and Central America | Brazil | – | 4559 | 4559 | * |
| | Colombia | 6366 | 380 | 6746 | 91 |
| | Venezuela | 479 | – | 479 | 120 |
| | Other | 45 | 679 | 724 | * |
| | Total | 6890 | 5618 | 12508 | 148 |
| Europe and Eurasia | Bulgaria | 2 | 2364 | 2366 | 82 |
| | Czech Republic | 192 | 908 | 1100 | 22 |
| | Germany | 99 | 40600 | 40699 | 223 |
| | Greece | – | 3020 | 3020 | 44 |
| | Hungary | 13 | 1647 | 1660 | 183 |
| | Kazakhstan | 21500 | 12100 | 33600 | 303 |
| | Poland | 4338 | 1371 | 5709 | 43 |
| | Romania | 10 | 281 | 291 | 9 |
| | Russian Federation | 49088 | 107922 | 157010 | 495 |
| | Spain | 200 | 330 | 530 | 73 |
| | Turkey | 529 | 1814 | 2343 | 27 |
| | Ukraine | 15351 | 18522 | 33873 | 462 |
| | United Kingdom | 228 | – | 228 | 13 |
| | Other | 1440 | 20735 | 22175 | 317 |
| | Total | 92990 | 211614 | 304604 | 257 |
| Africa and Middle East | South Africa | 30156 | – | 30156 | 119 |
| | Zimbabwe | 502 | – | 502 | 301 |
| | Other Africa | 860 | 174 | 1034 | * |
| | Middle East | 1203 | – | 1203 | * |
| | Total | 32721 | 174 | 32895 | 127 |
| Asia Pacific | Australia | 37100 | 39300 | 76400 | 180 |
| | China | 62200 | 52300 | 114500 | 35 |
| | India | 56100 | 4500 | 60600 | 106 |
| | Indonesia | 1520 | 4009 | 5529 | 18 |
| | Japan | 340 | 10 | 350 | 382 |
| | New Zealand | 33 | 538 | 571 | 107 |
| | North Korea | 300 | 300 | 600 | 16 |
| | Pakistan | – | 2070 | 2070 | * |
| | South Korea | – | 126 | 126 | 60 |
| | Thailand | – | 1239 | 1239 | 69 |
| | Vietnam | 150 | – | 150 | 3 |
| | Other | 1582 | 2125 | 3707 | 114 |
| | Total | 159326 | 106517 | 265843 | 57 |
| World | Total | 404762 | 456176 | 860938 | 118 |

*More than 500 years.
Source: based on WEC (2010) and BP (2011).

**Table 7.5** World Coal Production (WEC 2010) & BP Statistical Review of World Energy (2011).

| Region | Country | 2007 | 2008 | 2009 | 2010 |
|---|---|---|---|---|---|
| North America | United States | 587.7 | 596.7 | 540.9 | 552.2 |
| | Canada | 36.0 | 35.6 | 32.5 | 34.9 |
| | Mexico | 6.0 | 5.5 | 5.1 | 4.5 |
| | Total | 629.7 | 637.8 | 578.5 | 591.6 |
| South and Central America | Brazil | 2.3 | 2.5 | 1.9 | 2.1 |
| | Colombia | 45.4 | 47.8 | 47.3 | 48.3 |
| | Venezuela | 5.6 | 4.5 | 2.7 | 2.9 |
| | Other | 0.3 | 0.4 | 0.5 | 0.5 |
| | Total | 53.6 | 55.2 | 52.4 | 53.8 |
| Europe and Eurasia | Bulgaria | 4.7 | 4.8 | 4.6 | 4.8 |
| | Czech Republic | 23.3 | 21.1 | 19.5 | 19.4 |
| | France | 0.2 | 0.1 | $<0.05$ | $<0.05$ |
| | Germany | 51.5 | 47.7 | 44.4 | 43.7 |
| | Greece | 8.6 | 8.3 | 8.4 | 8.8 |
| | Hungary | 2.0 | 1.9 | 1.9 | 1.9 |
| | Kazakhstan | 50.0 | 56.8 | 51.5 | 56.2 |
| | Poland | 62.3 | 60.5 | 56.4 | 55.5 |
| | Romania | 6.7 | 6.7 | 6.4 | 5.8 |
| | Russian Federation | 148.0 | 153.4 | 142.1 | 148.8 |
| | Spain | 6.0 | 3.7 | 3.5 | 3.3 |
| | Turkey | 15.8 | 17.2 | 17.4 | 17.4 |
| | Ukraine | 39.9 | 41.3 | 38.4 | 38.1 |
| | United Kingdom | 10.3 | 11.0 | 10.9 | 11.0 |
| | Other | 16.7 | 17.3 | 16.9 | 16.1 |
| | Total | 446.1 | 452.0 | 422.1 | 430.9 |
| Middle East | Total | 1.0 | 1.0 | 1.0 | 1.0 |
| Africa | South Africa | 139.6 | 142.4 | 141.2 | 143.0 |
| | Zimbabwe | 1.3 | 1.0 | 1.1 | 1.1 |
| | Other | 0.9 | 0.9 | 0.8 | 0.8 |
| | Total | 141.8 | 144.2 | 143.1 | 144.9 |
| Asia Pacific | Australia | 217.2 | 220.7 | 228.8 | 235.4 |
| | China | 1501.1 | 1557.1 | 1652.1 | 1800.4 |
| | India | 181.0 | 195.6 | 210.8 | 216.1 |
| | Indonesia | 133.4 | 147.8 | 157.6 | 188.1 |
| | Japan | 0.8 | 0.7 | 0.7 | 0.5 |
| | New Zealand | 3.0 | 3.0 | 2.8 | 3.3 |
| | Pakistan | 1.6 | 1.8 | 1.6 | 1.5 |
| | South Korea | 1.3 | 1.2 | 1.1 | 0.9 |
| | Thailand | 5.1 | 5.0 | 5.0 | 5.0 |
| | Vietnam | 22.4 | 23.0 | 25.2 | 24.7 |
| | Other | 23.3 | 24.3 | 29.2 | 33.4 |
| | Total | 2090.2 | 2180.1 | 2314.8 | 2509.4 |
| World | Total | 3362.4 | 3470.3 | 3511.8 | 3731.4 |

Source: based on WEC (2010) and BP (2011).

**Table 7.6** World Coal Consumption (WEC 2010) & BP Statistical Review of World Energy (2011).

| Region | Country | 2007 | 2008 | 2009 | 2010 |
|---|---|---|---|---|---|
| North America | United States | 573.3 | 564.1 | 496.2 | 524.6 |
| | Canada | 32.3 | 28.9 | 23.3 | 23.4 |
| | Mexico | 9.1 | 6.9 | 8.6 | 8.4 |
| | Total | 614.7 | 599.9 | 528.1 | 556.3 |
| | Argentina | 0.4 | 1.1 | 1.2 | 1.2 |
| South and Central America | Brazil | 13.4 | 13.5 | 11.7 | 12.4 |
| | Chile | 3.8 | 4.1 | 3.7 | 3.7 |
| | Colombia | 2.4 | 2.8 | 3.7 | 3.8 |
| | Ecuador | – | – | – | – |
| | Peru | 0.5 | 0.5 | 0.5 | 0.5 |
| | Trinidad and Tobago | – | – | – | – |
| | Venezuela | <0.05 | <0.05 | <0.05 | <0.05 |
| | Other | 2.1 | 2.2 | 2.0 | 2.1 |
| | Total | 22.6 | 24.2 | 22.9 | 23.8 |
| Europe and Eurasia | Austria | 2.9 | 2.7 | 2.2 | 2.0 |
| | Azerbaijan | <0.05 | <0.05 | <0.05 | <0.05 |
| | Belarus | <0.05 | <0.05 | <0.05 | <0.05 |
| | Belgium and Luxembourg | 5.5 | 4.8 | 4.6 | 4.9 |
| | Bulgaria | 7.8 | 7.5 | 6.3 | 6.6 |
| | Czech Republic | 19.3 | 17.4 | 16.2 | 16.0 |
| | Denmark | 4.7 | 4.1 | 4.0 | 3.8 |
| | Finland | 4.6 | 3.4 | 3.7 | 4.6 |
| | France | 12.3 | 11.9 | 9.9 | 12.1 |
| | Germany | 85.7 | 80.1 | 71.7 | 76.5 |
| | Greece | 8.5 | 8.1 | 8.1 | 8.5 |
| | Hungary | 2.9 | 2.8 | 2.5 | 2.6 |
| | Republic of Ireland | 1.5 | 1.4 | 1.3 | 1.4 |
| | Italy | 17.2 | 16.7 | 13.1 | 13.7 |
| | Kazakhstan | 30.8 | 34.0 | 31.7 | 36.1 |
| | Lithuania | 0.2 | 0.2 | 0.1 | 0.2 |
| | Netherlands | 9.0 | 8.5 | 7.9 | 7.9 |
| | Norway | 0.4 | 0.5 | 0.3 | 0.5 |
| | Poland | 57.9 | 56.0 | 51.9 | 54.0 |
| | Portugal | 3.3 | 3.2 | 3.3 | 3.4 |
| | Romania | 7.4 | 7.4 | 6.6 | 6.2 |
| | Russian Federation | 93.5 | 100.4 | 91.9 | 93.8 |
| | Slovakia | 3.8 | 3.2 | 3.5 | 2.7 |
| | Spain | 20.2 | 15.6 | 10.5 | 8.3 |
| | Sweden | 2.2 | 2.0 | 1.6 | 2.0 |
| | Switzerland | 0.1 | 0.1 | 0.1 | 0.1 |
| | Turkey | 31.0 | 31.3 | 32.0 | 34.4 |
| | Ukraine | 39.7 | 40.3 | 35.0 | 36.4 |
| | United Kingdom | 38.4 | 35.6 | 29.6 | 31.2 |
| | Uzbekistan | 1.4 | 1.4 | 1.4 | 1.3 |
| | Other | 16.0 | 16.8 | 15.2 | 15.7 |
| | Total | 528.3 | 517.8 | 466.4 | 486.8 |
| Middle East | Iran | 1.3 | 0.9 | 1.1 | 1.1 |
| | Israel | 8.0 | 7.9 | 7.7 | 7.7 |
| | Total | 9.3 | 8.7 | 8.8 | 8.8 |
| Africa | Algeria | 0.7 | 0.6 | 0.2 | 0.3 |
| | Egypt | 1.2 | 1.2 | 0.6 | 0.7 |

**Table 7.6** (*Continued*)

| Region | Country | 2007 | 2008 | 2009 | 2010 |
|---|---|---|---|---|---|
| | South Africa | 85.1 | 84.7 | 87.7 | 88.7 |
| | Other | 6.0 | 6.2 | 5.5 | 5.7 |
| | Total | 93.1 | 92.7 | 94.1 | 95.3 |
| Asia and Pacific | Australia | 54.2 | 51.8 | 51.7 | 43.4 |
| | Bangladesh | 0.4 | 0.6 | 0.4 | 0.5 |
| | China | 1438.4 | 1479.3 | 1556.8 | 1713.5 |
| | China Hong Kong SAR | 7.5 | 7.0 | 7.6 | 6.3 |
| | India | 210.3 | 230.4 | 250.6 | 277.6 |
| | Indonesia | 37.8 | 30.1 | 34.6 | 39.4 |
| | Japan | 125.3 | 128.7 | 108.8 | 123.7 |
| | Malaysia | 7.1 | 5.0 | 4.0 | 3.4 |
| | New Zealand | 1.6 | 2.0 | 1.6 | 1.0 |
| | Pakistan | 5.1 | 5.3 | 4.7 | 4.6 |
| | Philippines | 5.9 | 7.0 | 6.7 | 7.7 |
| | South Korea | 59.7 | 66.1 | 68.6 | 76.0 |
| | Taiwan | 41.8 | 40.2 | 38.7 | 40.3 |
| | Thailand | 14.1 | 15.3 | 14.5 | 14.8 |
| | Vietnam | 10.1 | 10.0 | 14.0 | 13.7 |
| | Other | 18.0 | 19.7 | 22.1 | 18.9 |
| | Total | 2037.5 | 2098.4 | 2185.3 | 2384.7 |
| World | Total | 3305.6 | 3341.7 | 3305.6 | 3555.8 |

Source: based on WEC (2010) and BP (2011).

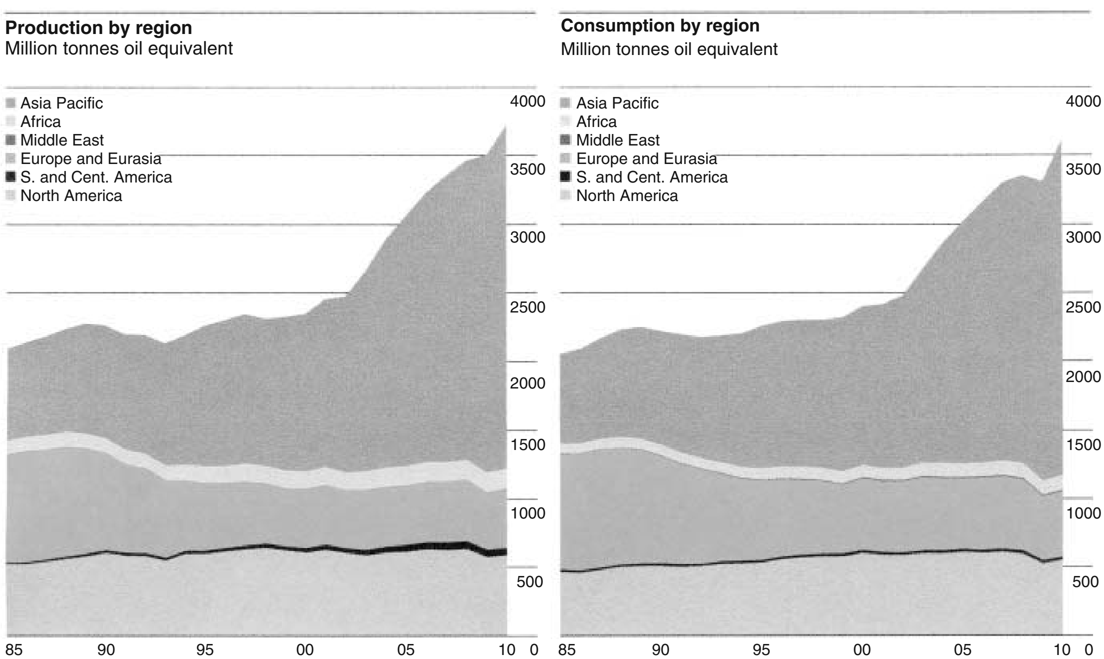



**Figure 7.14** Coal production and consumption from 1985 to 2010. Reproduced from BP Statistical Review of World Energy (2011). This figure is reproduced in colour in the Plates section.

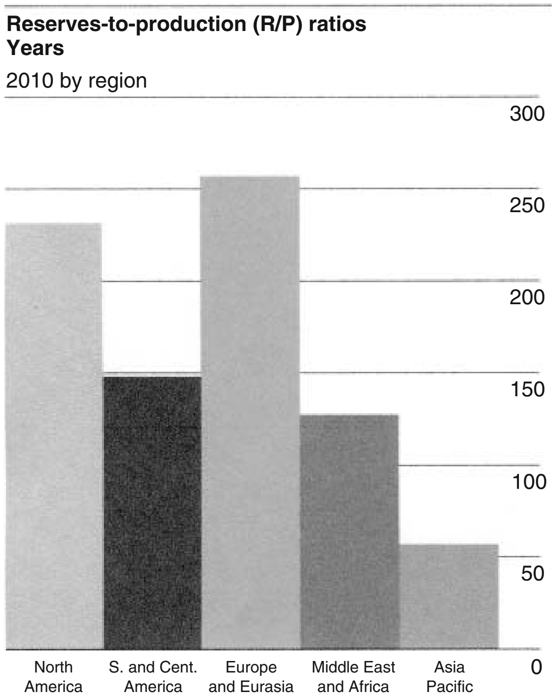



**Figure 7.15** Fossil fuel/RP ratios at end 2010. Reproduced from BP Statistical Review of World Energy (2011).

Brown coal production in the OECD countries has declined while non-OECD coal production has risen (WCA, 2011). Over the past 20 yr, the development of the coal industry in countries such as Indonesia and Colombia has increased production in their regions. Indonesia has accounted for over 75% of global incremental growth in recent years. The political and economic reorganization in Eastern Europe will influence future production figures in Europe. New coal-producing countries such as Mongolia and Pakistan will also have a strong regional influence on the distribution and use of coal in their regions.

#### *7.4.2.2 Regional production and consumption*

Table 7.6 shows the principal areas of coal consumption, some 3555.8 Mt oil equivalent in total. This can be compared with Table 7.5 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 1985–2010. Those countries in which coal production is greater than consumption and where the coal quality is internationally marketable, are coal exporters, for example United states, Australia and South Africa. Smaller producers such as Colombia export a large percentage of their total production, whereas others such as the People's Republic of China 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, and this production is intended for the export market.

#### *7.4.2.3 Reserves/production ratio (R/P)*

The reserves/production 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 reserves of coal in 2010 were sufficient to meet 118 yr of global production, by far the largest R/P ratio of any fossil fuel. Figure 7.15 illustrates R/P ratios by region.

# 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 particular 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. Smaller scale, more detailed examination of the coal deposits utilizes shallow seismic, ground magnetic, electrical resistivity and microgravity coupled with the geophysical logging of all boreholes, which in turn involves the use of density, electrical, electromagnetic and radiometric techniques.

In established coal mining areas, the combination of geophysical logging with high-resolution seismic and ground magnetic surveys contribute significantly in the delineation of economic mining areas, both for opencast and underground operations. In underground operations, the use of in-seam seismic is now used as a tactical tool in planning the orientation of mining areas, in particular 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 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 in others covering more specific aspects of coal geophysics exploration, such as in-seam seismic (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). 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

*Coal Geology*, Second Edition. Larry Thomas.

**Table 8.1** Table of physical properties of coals and associated sedimentary and igneous rocks.

| Lithology | Density (wet) (g cm$^{-3}$) | | Seismic velocity | Magnetic susceptibility (×10 SI units) | | Electrical resistivity (Ωm) |
|---|---|---|---|---|---|---|
| | Range | Average | (km s$^{-1}$) | Range | Average | Range |
| Sandstone | 1.61–2.76 | 2.35 | 3.6 | 0–20 | 0.4 | 1–6.4 × $10^8$ |
| Shale | 1.77–3.20 | 2.40 | 2.8 | 0.01–15 | 0.6 | 20–2 × $10^3$ |
| Limestone | 1.93–2.90 | 2.55 | 5.5 | 0–3 | 0.3 | 50–1 × $10^7$ |
| Lignite | 1.10–1.25 | 1.19 | | | | 9–200 |
| Bituminous Coal | 1.20–1.80 | 1.32 | 1.8–2.8 | – | 0.02 | 0.6–1 × $10^5$ |
| Anthracite | 1.34–1.80 | 1.50 | 1.8–2.8 | – | 0.02 | 1 × $10^{-3}$–2 × $10^5$ |
| Acid Igneous rock | 2.30–3.11 | 2.61 | 4.0–5.5 | 0–80 | 8.0 | 4.5 × $10^3$ (wet granite)<br>1.3 × $10^6$ (dry granite) |
| Basic Igneous rock | 2.09–3.17 | 2.79 | 4.0–7.0 | 0.5–100 | 25.0 | 20–5 × $10^7$ (dolerite) |
| Metamorphic rock | 2.40–3.10 | 2.74 | 5.0–7.0 | 0–70 | 4.2 | 20–1 × $10^4$ (schist) |

Source: based on Telford, Geldart and Sheriff *et al.* (1990).

underlying basement or as intrusives into the coal-bearing strata.

Low and high rank coals (1.1–1.8 gm cm$^{-3}$) are less dense than the surrounding sediments (1.6–2.9 gm cm$^{-3}$), which in turn are less dense than igneous and metamorphic rocks (2.1–3.1 gm cm$^{-3}$). In sedimentary rocks, the wide range of density is due to variations in porosity, 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 gm cm$^{-3}$. Density also increases with the degree of metamorphism, as recrystallization reduces pore space to form a denser rock as well as converting some minerals to more dense 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 1.8–2.8 km s$^{-1}$, and mudrocks such as shales have similar values. Sandstones have a higher value, which increases with increasing quartz content, and dense limestones together with igneous and metamorphic rocks have much higher velocities of 4.0–7.0 km s$^{-1}$.

### *8.2.3 Seismic reflection coefficients*

The seismic reflection coefficient determines whether an interface gives a reflection and depends upon the density as well as 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 measurement 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, titano-magnetite or pyrrhotite. It is worth noting that the sulfide minerals such as pyrite, which is a common mineral in coals and associated sediments, has 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 rationalized 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 (Ω-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, which 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; 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, electromagnetic, gravity and magnetic surveys together with reflection and refraction seismic surveys are used, usually as 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 GPS 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 past 30 yr, and it is now applied to coal mining with increases accuracy and confidence. Background information and full discussion of equipment and techniques used in gathering and processing seismic reflection data are not given in this text but are referred to in the reference section.

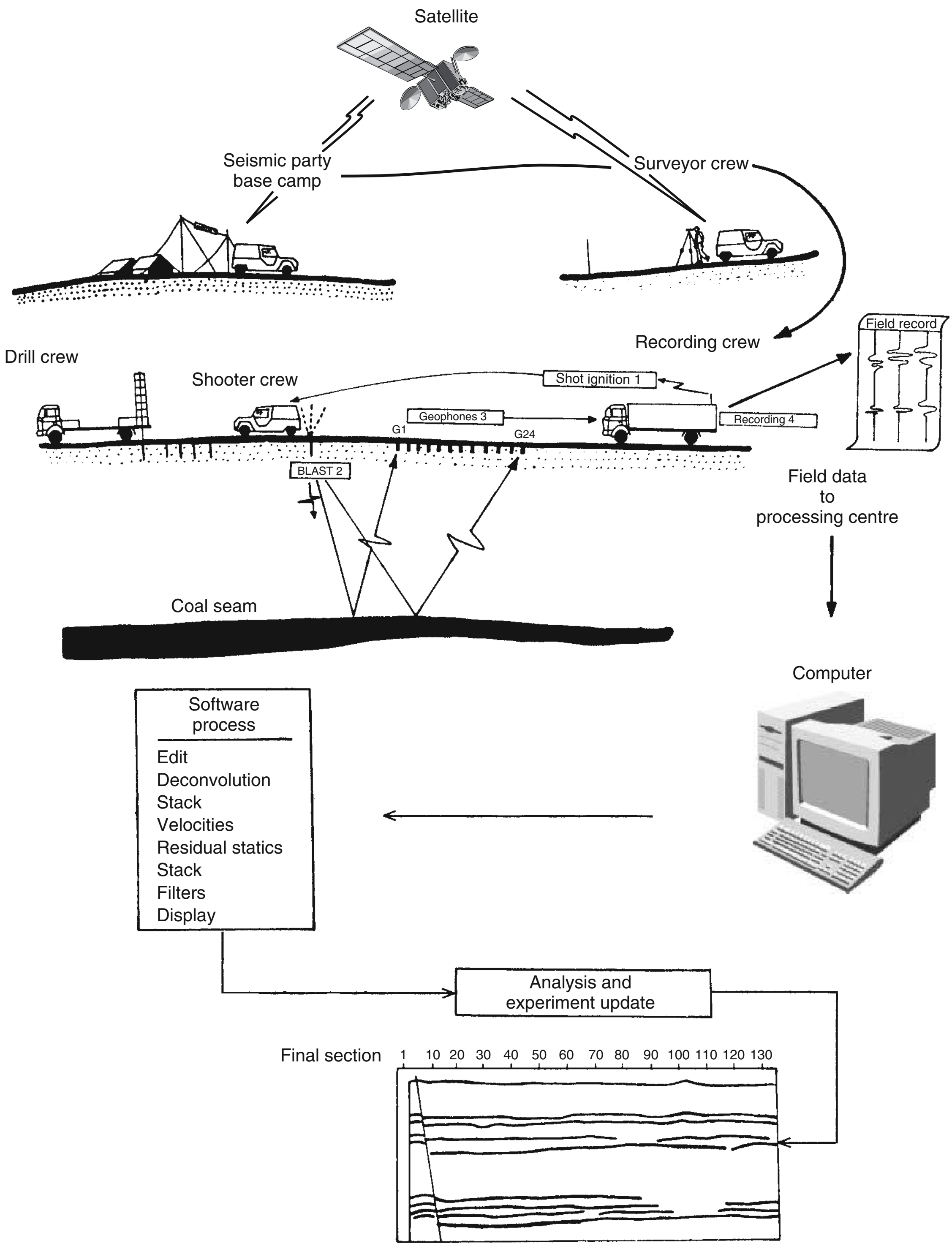



**Figure 8.1** Seismic reflection survey (a) Field data acquisition; (b) seismic data processing. (Adapted from Peace, 1979.)

### *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 km $s^{-1}$ in hard, dense limestone and ranges between 1.8 and 2.8 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 change in velocities and density at a boundary is 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 other sediments may be between 35% and 50%. This produceslarge 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 seismic wave has been generated, that is 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 km $s^{-1}$.

The advance in the identification of smaller and smaller disturbances in coal seams is due mainly 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 carried out successfully in the United Kingdom, where surface seismic exploration is combined with in-seam seismic methods (see section 8.4.1); the latter being able to map smaller faults that are below the resolution of surface seismic (Carpenter and Robson, 2000). The ability to identify faults, folds, washouts, seam splits and thickness changes by theuseof seismic reflection techniques is an effective method for pinpointing potential geological hazards, which can then be built into mine planning and design.

The sources commonly used in seismic reflection surveys for coal are either the detonation of an explosive charge such as dynamite (but this produces environmental problems in populated areas), or the 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, that is 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 is a lot

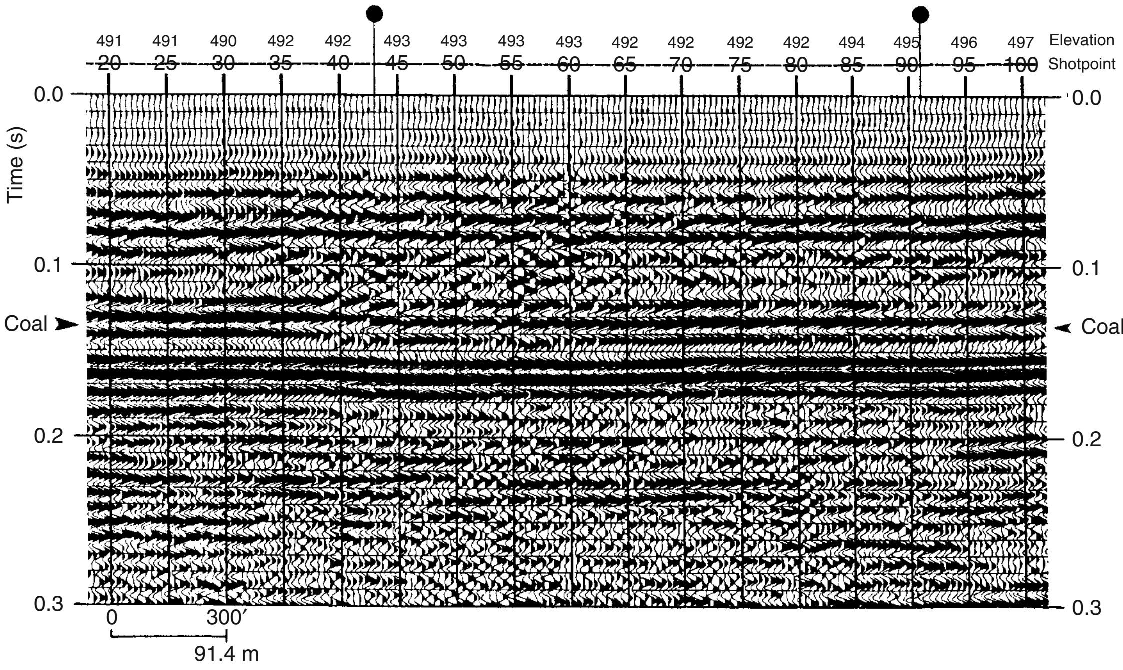



**Figure 8.2** Seismic section showing a robust and continuous coal seam reflection. This indicates uniform coal seam thickness with no detectable geological disturbance. (From Gochioco, 1991.)

more difficult. Figure 8.2 illustrates a seismic reflection profile across a coalfield in the U.S.A. The section shows the coal seam reflection to be robust and continuous from SP20 to SP100, indicating uniform coal seam thickness across the entire section.

Figure 8.3a shows a seismic reflection profile across a lignite deposit in Northern Ireland, United Kingdom, 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 Paleogene–Neogene clays, which have a very low attenuation for seismic energy and suggests an expectation of good high-frequency transmission. This is also dependant 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 Paleogene–Neogene 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 impedence 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 s and 0.1 s (TWT) on the southeast margin mark the top and bottom of the lignite. The weathered–fresh basalt interface is seen on the southeast margin at about 0.13 s (TWT) and can be seen clearly to shallow towards the northwest. In addition two faults with downthrows to the southeast can be detected. The irregularly shaped body at X on 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 has been carried out in the Kladno coalfield in Bohemia. Oplustil, Pesek and Skopec (1997) describe 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, including a large fault with a downthrow of tens of metres with no equivalent in any mine. There is also some suggestion of faults being the result of synsedimentary movements in the deposit.

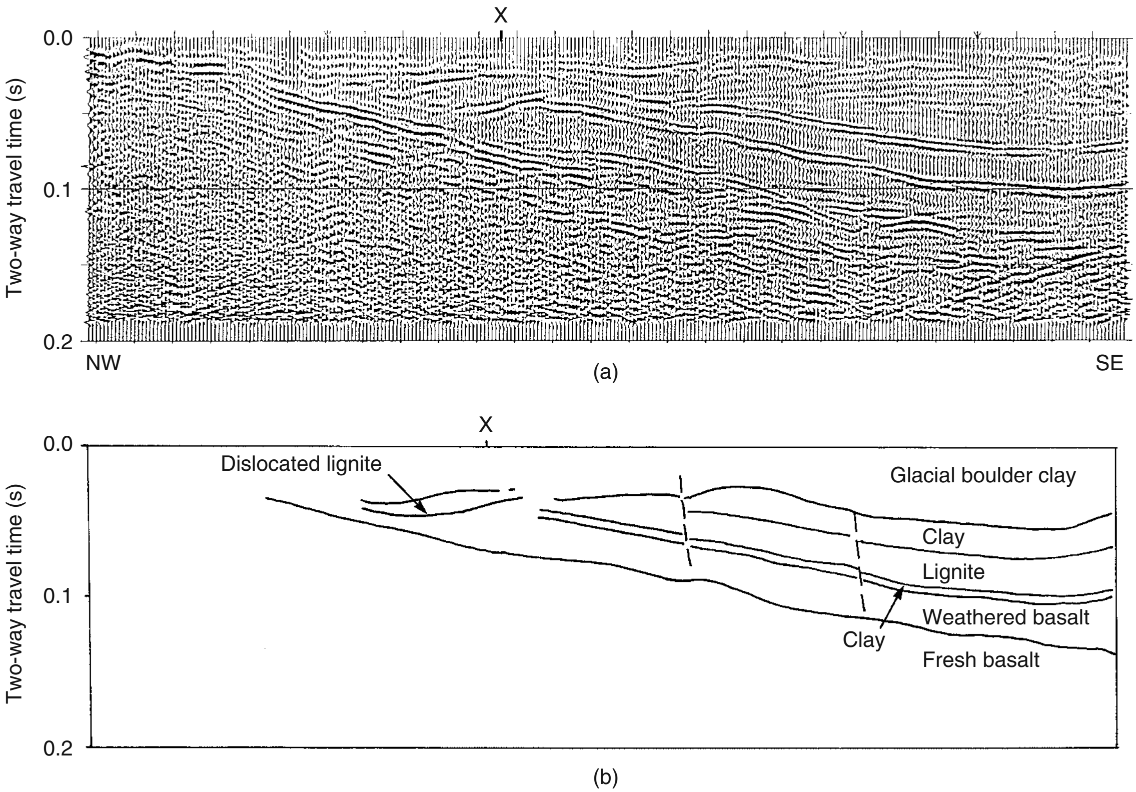



**Figure 8.3** (a) Shallow seismic reflection survey, Northern Ireland, United Kingdom. (b) Interpretation of seismic section in (a), clearly showing interbedded lignite and underlying basalt. (Unpublished figure produced by permission of Antrim Coal Company Ltd.)

In Wyoming, United States, 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 sea; 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 the washout are quite distinct, from which 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 have been 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 has revealed the three-dimensional model of this coal basin. Use of high-resolution seismic have been applied to the exploration and development of coal deposits in the Upper Silesian and Lublin Coal Basins in Poland. High-resolution seismic surveys have been carried out in various geological and mining conditions to determine commercial coal

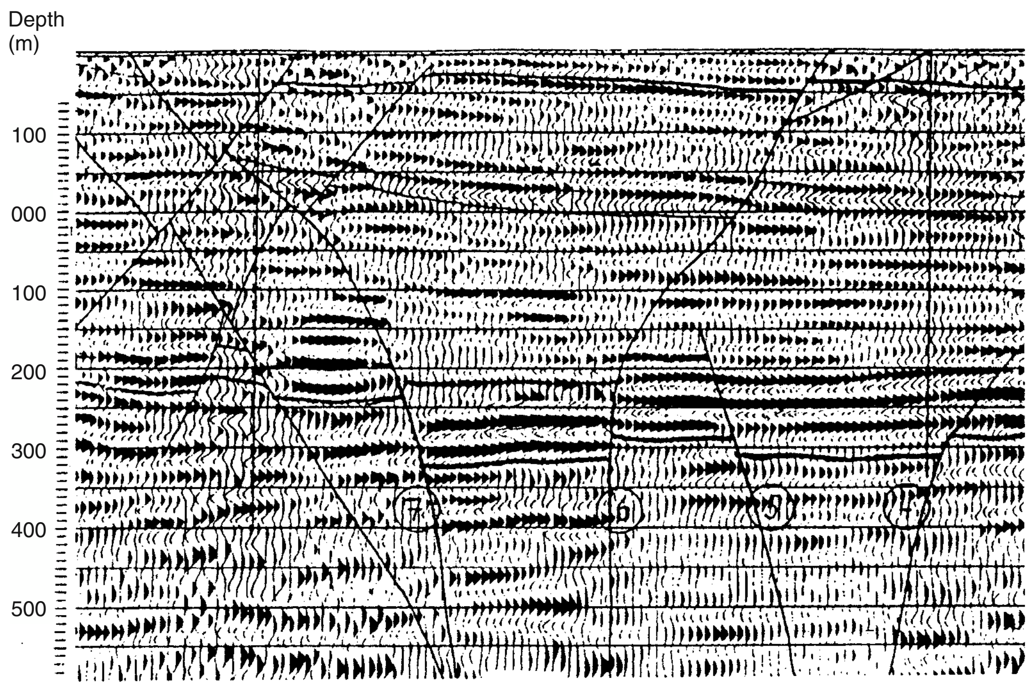



**Figure 8.4** Depth seismic profile with interpreted faults. (From Opulstil, Pesek and Skopec, 1997.)

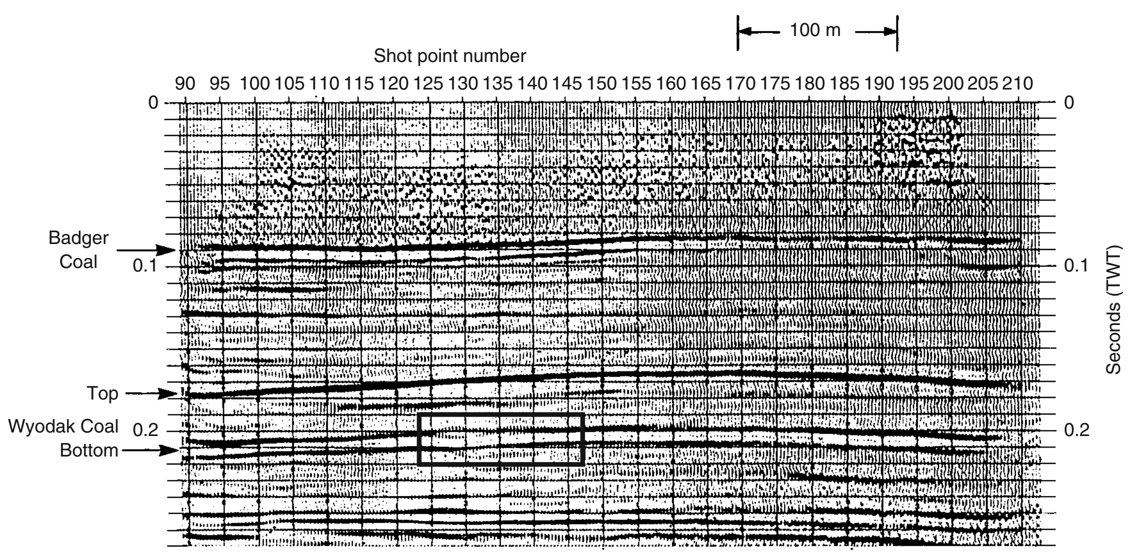



**Figure 8.5** Seismic profile showing anticline structure of Wyodak Coal and the good reflection of Badger Coal. Seismic anomaly (boxed area) is interpreted as a channel cut into the base of the Wyodak Coal, Wyoming, United States. (From Greaves, 1985.)

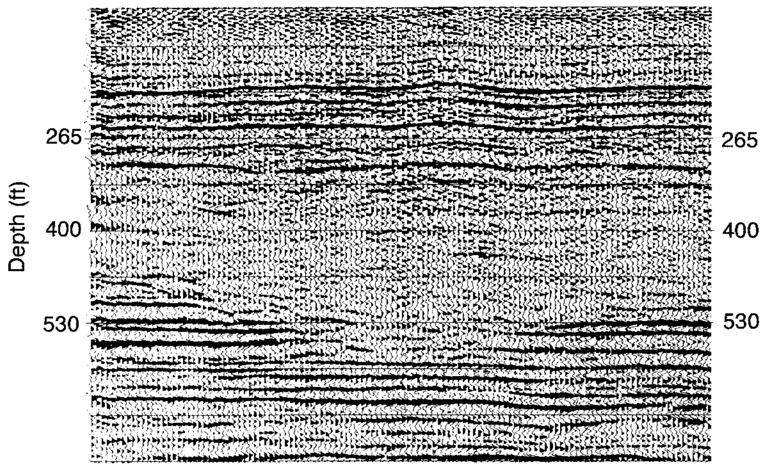



**Figure 8.6** Seismic reflection profile showing a buried channel at a depth of 152.4 m (500 ft), Texas Gulf Coast, United States. (From Peace, 1979.)

seams, locate non-coal areas and faults, and solve some problems associated with mining conditions. These surveys have enabled improved designs of new mines and/or expansion of existing ones to provide more productive and safer coal mines (Pietsch and Slusarczyk, 1992).

For the past two decades all new coal mines in China have been required to be preceded by a seismic survey before approval of the mine plan, and commencement of coal production (Pu and Xizun, 2005). An early example of the success of seismic surveys is in the Chensilou coal mine where the geological survey detected only a couple of faults but a subsequent seismic survey revealed 24 faults and the workface and coal production plan were completely altered, with an estimated saving of US$ 20 million. Initially these surveys were all two-dimensional, but more recently they were replaced by three-dimensional seismic surveys, which formed 95% of all seismic surveys in 2003, when a total of 350 surveys had been completed. These three-dimensional seismic surveys allow display not only of colour seismic sections but also colour time and horizontal slices (Figure 8.7), thereby helping to detect the areas without coal, old

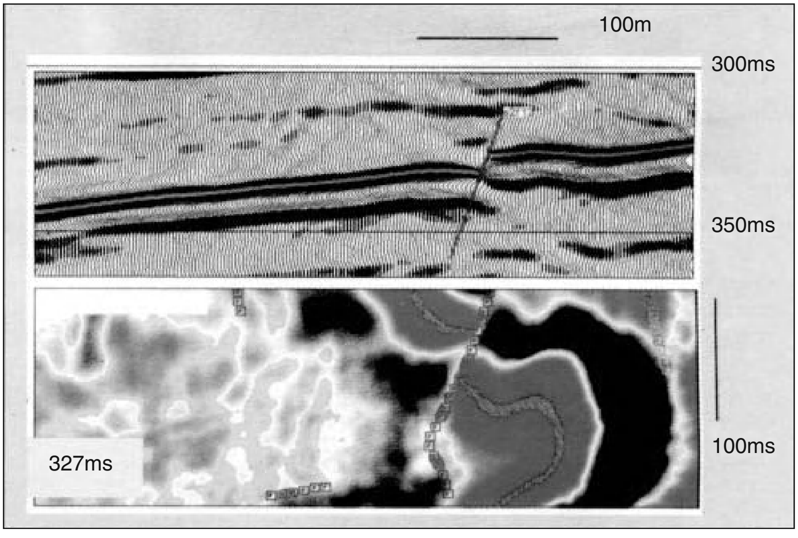



**Figure 8.7** 3D seismic survey in China, a fault is seen in section (top) and time slice (bottom) (Pu & Xisun 2005). First Break reproduced with permission. This figure is reproduced in colour in the Plates section.

coal workings and especially coal seam faults. The fault detection success quoted by Pu and Xizun (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 coal mine planning and production.

Quan-Ming, Yang and Xing-Ping (2008) have had considerable success from three-dimensional seismic survey results in identifying many collapsed breccias before mining in certain northern Chinese coalfields, 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) have developed new data processing techniques for three-dimensional seismic surveys in areas of complex coal geology. In particular the use of pre-stack Kirchoff time migration has, in comparison with previous post-stack time migration, produced better images of coal seam reflections. These recent modern three-dimensional seismic surveys in Chinese coalfield studies have helped to significantly increase Chinese coal production.

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 wackers can put 800 to 1500 impulses into the ground for each record. Although the mini-SOSIE has been widely used, particularly with success in Australia, difficulties have been encountered in using the technique in dissected hilly terrain combined with deep weathering profiles. A number of vibrating sources have now been developed to optimize 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 United Kingdom, seismic reflection techniques have been developed by using an energy source situated at depth in a borehole 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 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 m, 54 m, 58 m, 70 m and 128 m all give a good reflection, but the thin seam at 80 m is hardly seen at all. Other reflections at 100 m and 110 m may be weaker reflections from differing lithotypes in the sequence.

This technique has been used to correlate between boreholes. Figure 8.9a and b 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 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.

Cross-hole seismic reflection surveys are possible in certain opencast exploration sites that have closely spaced boreholes (Goulty *et al.*, 1990; Kragh, Goulty and Findlay, 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 vertical seismic profiling, but 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 United Kingdom 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 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, and/or 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

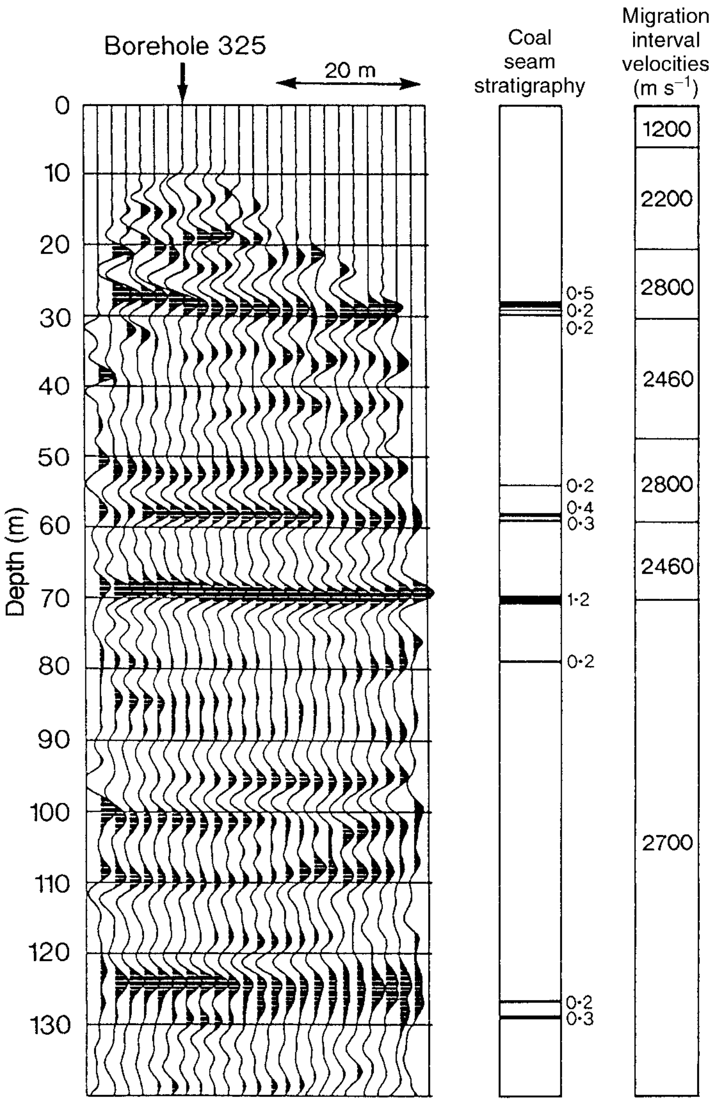


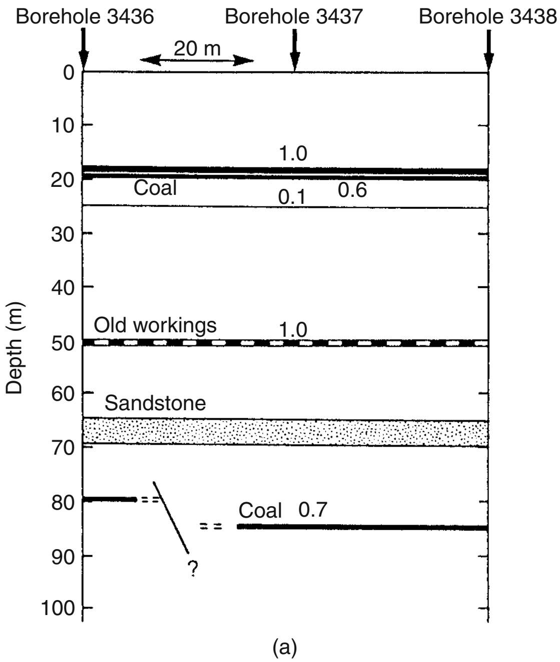

**Figure 8.8** Seismic depth section obtained from the hole-to-surface survey in boreholes at a United Kingdom opencast site, with the coal seam stratigraphy and velocity profile used for migration. (From Kragh, Goulty and Findlay, 1991.)

considerably over a short distance. Figure 8.10 shows part of a seismic section in which the original processing is compared with subsequent 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, which was not detected on the original section (Carpenter and Robson, 2000).

In the Bowen Basin, Australia, three-dimensional seismic exploration has been effectively integrated into standard work practices. Typically, accurate delineation of structure is the primary objective for the three-dimensional seismic survey. In addition, stratigraphic interpretation of the three-dimensional seismic data has contributed to predicting roof and floor conditions that impact mining operations (Peters, 2005).

### *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 a falling weight for shallow studies. Refracted waves differ from reflected waves in that the



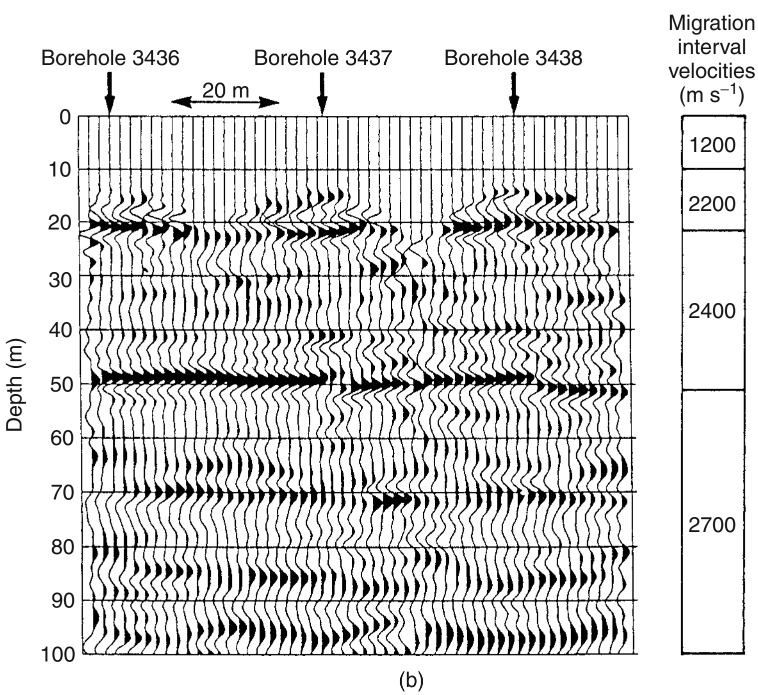




**Figure 8.9** (a) Coal seam stratigraphy for three boreholes at an opencast site, United Kingdom. (b) Combined seismic section from the three hole-to-surface surveys conducted in these boreholes with the velocity profile used for migration. The section is true scale, zero phase, normal polarity, and an equal energy trace normalization has been applied. (From Kragh, Goulty and Findlay, 1991.)

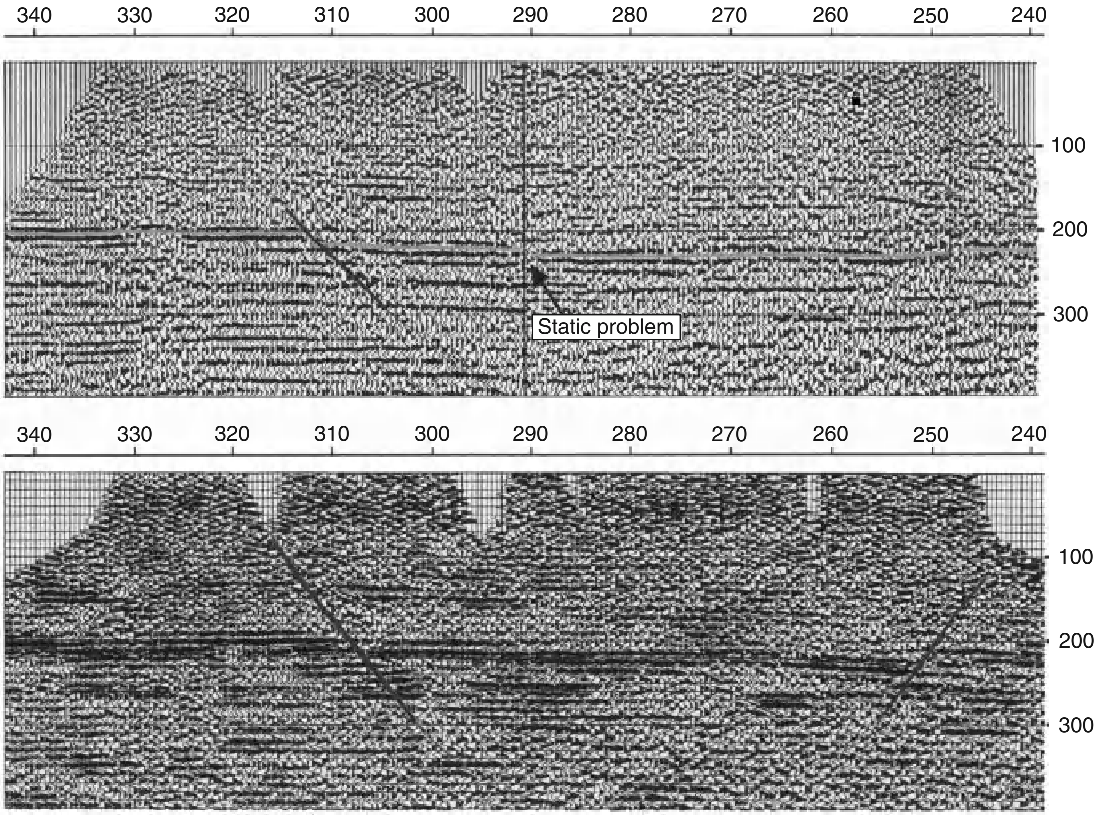



**Figure 8.10** Reprocessed seismic data line showing stronger reflections and structure hitherto undetected. (From Carpenter and Robson, 2000.)

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 (Goulty 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 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 rock-bursts or seismic activity induced by mining equipment and/or blasting. Kalab (1997) and Holub (1997) describe such seismic activity in the Ostrava-Karvina coalfield in the Czech Republic, and Redmayne, Richards and Wild (1998) describe similar seismic effects in the Midlothian coalfield, United Kingdom. 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) notes that over 50,000 tremors were recorded in this basin in the 30 years to 2004 and separates 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 coal mine safety in United States coal mines.

In addition to three-dimensional seismic reflection surveys, the use of time-lapse seismic surveys referred to as four-dimensional seismic, is increasing in order to determine underground mining subsidence (Goulty, 2008) and gas migration, and to identify geological conditions pertaining to hydrocarbon reservoirs and the suitability for $CO_2$ 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) describes the development of a new portable seismic instrument with this capability, stating that studies have been carried out over mines in the United Kingdom and describing the demonstration of the detection and identification of surface signals specifically from hammer pounding (to simulate trapped miners) at around 300 m below surface in a working coal mine in West Virginia, United States.

Passive refraction microtremor (ReMi) surveys utilize standard field seismic refraction recording equipment to record ambient background noise produced by microtremors caused by natural and anthropogenic activities. The production of two-dimensional shear-wave velocity sections have allowed the identification of reworked ground to a depth of 5 m below ground level, deeper strata at depths below 10 m, 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 with spacing that 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.

Areas with an anomalously high Bouguer gravity values (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 coalfields, that is 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 $km^2$. 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. 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 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 southeast 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 $km^2$ in the eastern trough, containing a coal seam 10.0 m in thickness, which is now being mined.

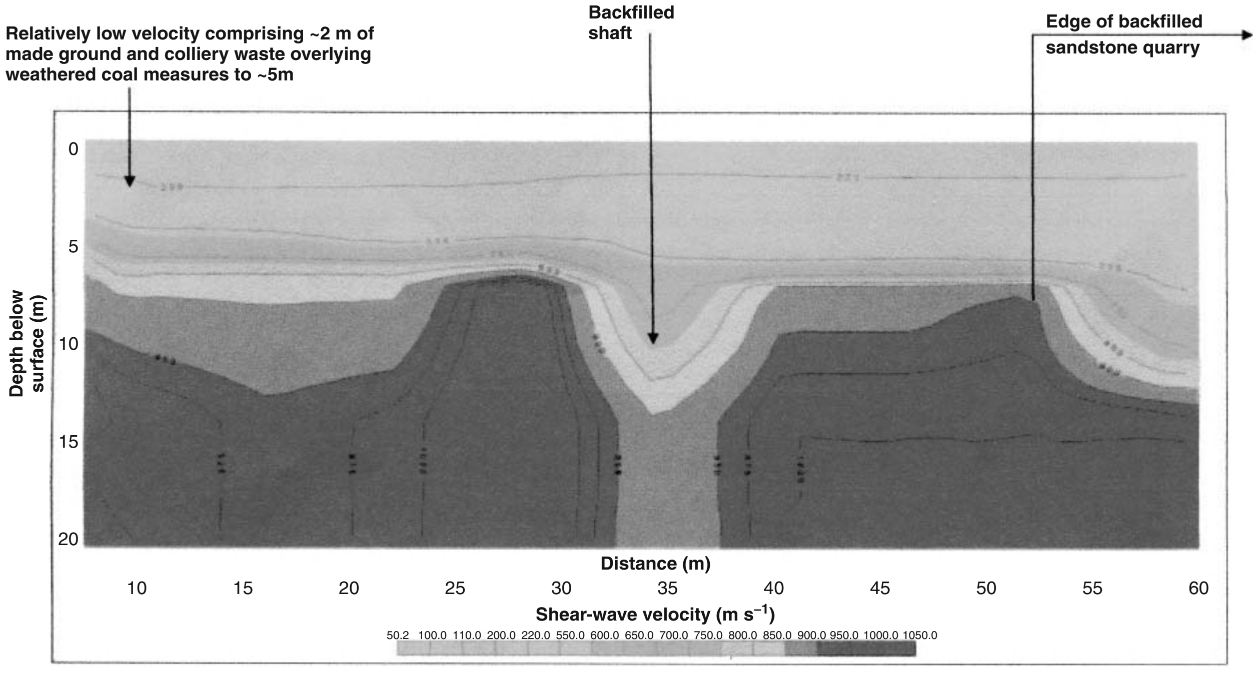



**Figure 8.11** A 2D pseudo-section of contoured shear-wave velocity data (Raines *et al.* 2011). Reproduced with permission of Geological Society of London. This figure is reproduced in colour in the Plates section.

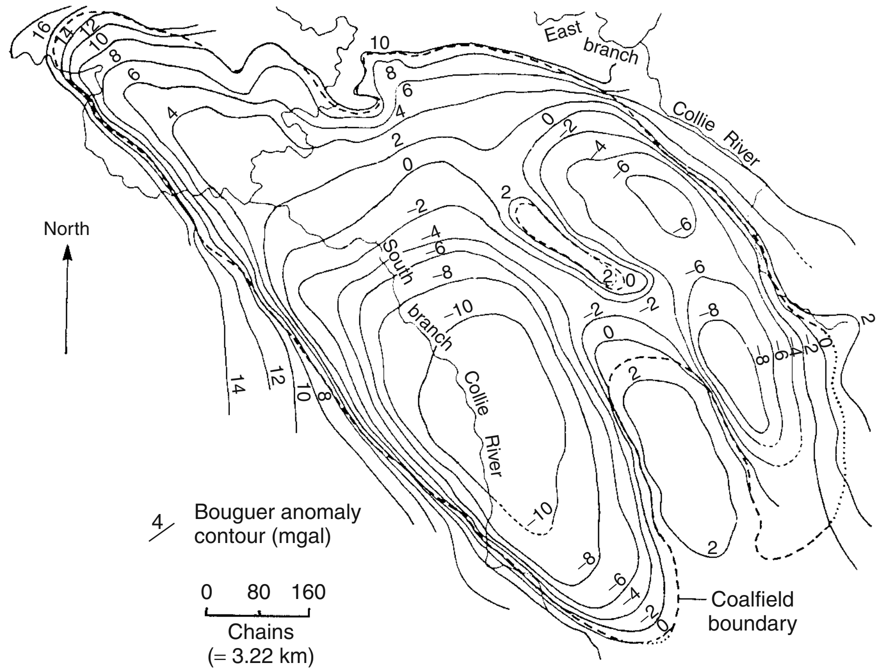



**Figure 8.12** Bouguer anomaly map of the Collie coalfield, Western Australia. (From Parasnis, 1986.)

In India and Bangladesh, similar coalfield basins have been identified. In northwest 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 Paleogene–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 in aeroplanes (Studinger, Bell and Frearson, 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 percent. 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 northwest 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 structure 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

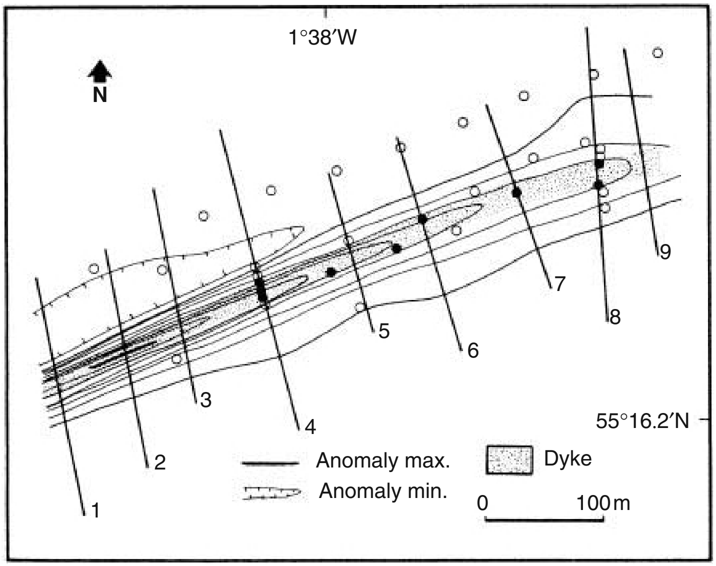
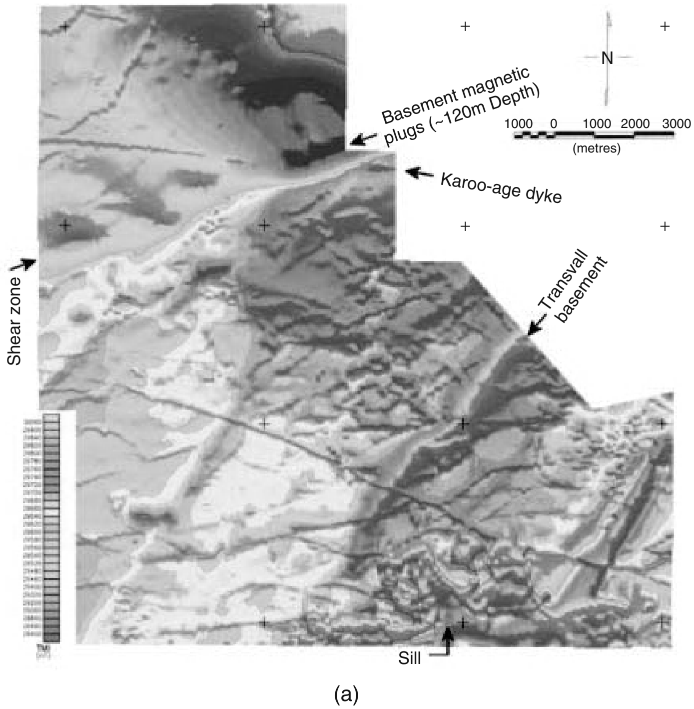



**Figure 8.13** Magnetic anomaly map for part of the Causey Park dyke, United Kingdom. Contours are at 100 nT intervals. Boreholes that encountered dolerite at rockhead are shown as solid circles. (From Goulty *et al.*, 1984.)

metres. An example is shown in Figure 8.13 of a lenticular shaped magnetic anomaly from Causey Park, northeast England, that has the configuration of a dolerite dyke as (Goulty *et al.*, 1984). The anomaly contours are expressed in 100 nanoTeslas (nT) intervals; a nanoTesla is 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 devolatilization 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 Karroo 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 modem 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 the Eastern Transvaal. Figure 8.14 shows a high-resolution magnetic survey over an Eastern Transvaal Coalfield, highlighting Karroo-age dykes and sills and also 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 minimize surface magnetic effects.

In the delineation of burnt zones in coal seams, the magnetic susceptibility of unbaked sedimentary rocks is quite low, but 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, because 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, Hatherton and Mumme, 1985). By carrying out a series of traverses perpendicular to the strike of the coal, a zone



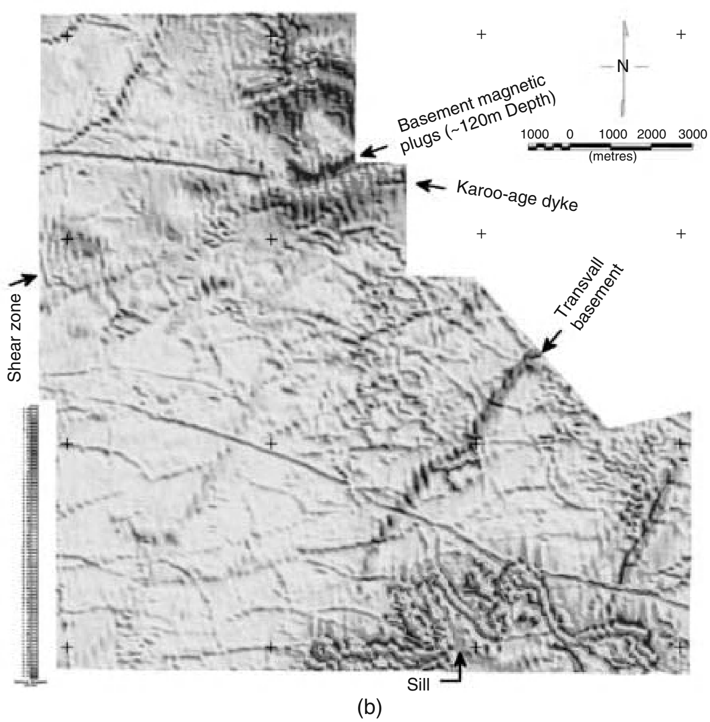




**Figure 8.14** High-resolution aeromagnetic survey over an Eastern Transvaal Coalfield, South Africa. (From Campbell, 2005.). Reproduced by permission of The Geological Society of South Africa. This figure is reproduced in colour in the Plates section.

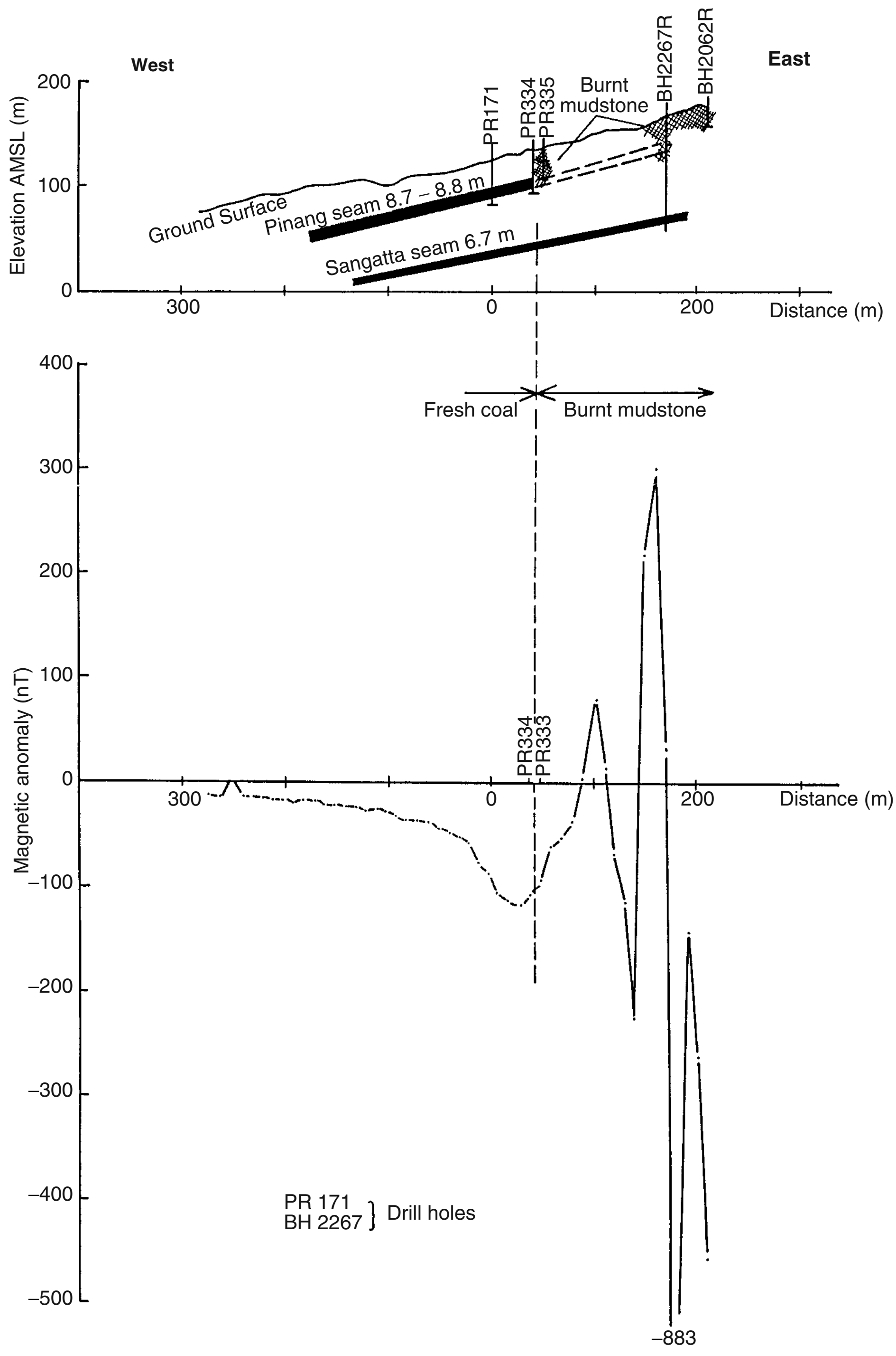



**Figure 8.15** Magnetic profile over burnt coal and geological cross-section, East Kalimantan, Indonesia.

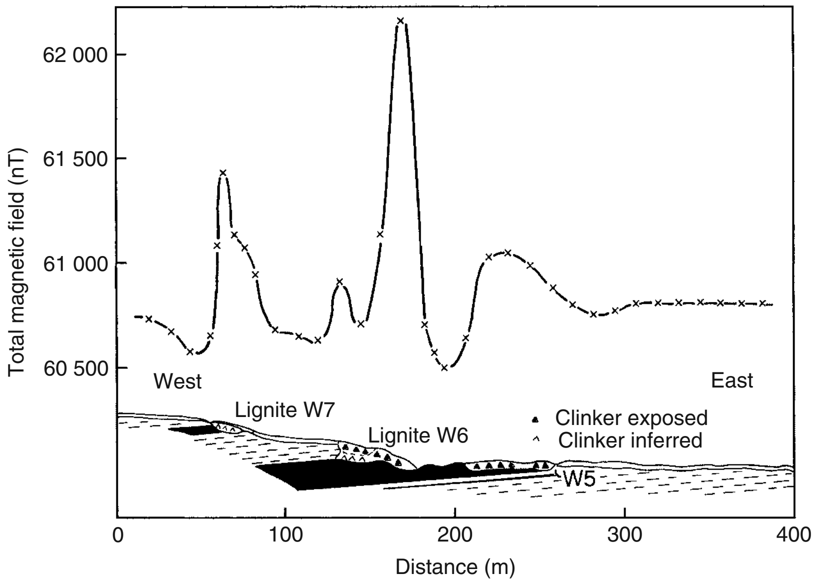



**Figure 8.16** Total magnetic field profile and geological cross-section, southern end of New Vale Coal County, Eastern Southland, New Zealand. (From Lindqvist, Hatherton and Mumme, 1985.)

of burnt coal can be determined accurately 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 geologic boundaries, fractures and cavities.

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, United States, where resistivity has detected the splitting of a coal seam into two, separated by a sand body. Electrical resistivity has been used in the United Kingdom to locate old concealed mine shafts and other mining cavities. Multielectrode resistivity measurements have been used in the United States to map shallow coal mine 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 (GPR) methods*

Ground-penetrating radar, also known as ground-probing radar (GPR), methods have proliferated over the past decade as more advanced instruments and three-dimensional surveys have been employed. The new GPR systems have been able to detect features at depths of tens of metres deeper than were previously possible. Franke and Utsi (2009) have identified a coal seam at greater than 30 m depth, which was detected as along with voids in an East Asian coal strata.

#### *8.3.4.3 Electromagnetic (EM) surveys*

In the United States helicopterborne and ground EM surveys have been useful in studies associated with coal bed methane in the Powder River Basin in Wyoming (Lipinski *et al.*, 2008) and using thermal infrared imagery and hehicopter EM conductivity to examine the abandoned underground coal mining workings in Pennsylvania (Love *et al.*, 2005).

### *8.3.5 Radioactive methods*

In coalfields, radioactive surveys can be used to trace marine shales 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, in-seam seismic and pulse radar methods have been used underground, and gravity measurements can be made in coal mine galleries to detect faults and voids in underground coal mines, provided appropriate bulk density values are obtained and the gallery corrections are applied (Casten and Gram, 1989)

### *8.4.1 In-seam seismic surveys (ISS)*

These surveys involve the use of channel waves propagating in the coal seam to detect discontinuities in advance of mining. In-seam seismic survey uses 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 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 Air Gun techniques may be used. In the United Kingdom, a pneumatic piston impactor has been tested for this purpose in mines where the use of roof and rib bolting as roadway support acts as anchor points for the geophones. Additional tests have used surface seismic sources to indirectly produce channel wave 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 in-seam seismic (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) describes a three-dimensional seismic survey conducted in advance of coal mine development in the Illinois Basin, to better define a geological structure with the potential to adversely affect longwall mining conditions. The three-dimensional seismic data indicated an abrupt change in coal-seam elevation referred to as a roll. This was seen to trend south then southeast in the coal reserve area. Figure 8.18 shows four in-line seismic sections 240 ft apart, which indicate that the steep western flank of the roll decreases as the roll trends to the southeast (Gochioco, 2004).

The ability to detect small-scale faulting by ISS is dependent on the particular attenuation characteristics of the coal seam concerned. In-seam seismic survey can detect faults that are below the resolution of surface seismic, and can improve the positioning of faults interpreted from the surface seismic, both of which have economic significance; faults that disrupt the coal seam can be detected within 200–300 m. Enhancement using 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 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 8.19c). On this evidence, the face was stopped short to avoid damage to working equipment.

### 8.4.2 Underground gravity surveys

Underground gravity surveys have been made in coal mine galleries to detect faults and voids in underground coal mines, with the provision that appropriate bulk density values are obtained and the gallery corrections, that is free air correction, Bouguer and terrain corrections are applied (Casten and Gram, 1989)

### 8.4.3 Ground-penetrating radar (GPR) techniques

Ground-penetrating radar, 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 continous coal cutting machines, and for safety reasons. Pulse radar methods have been employed in underground workings in the United Kingdom and United States to determine coal thickness. In the United States, 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

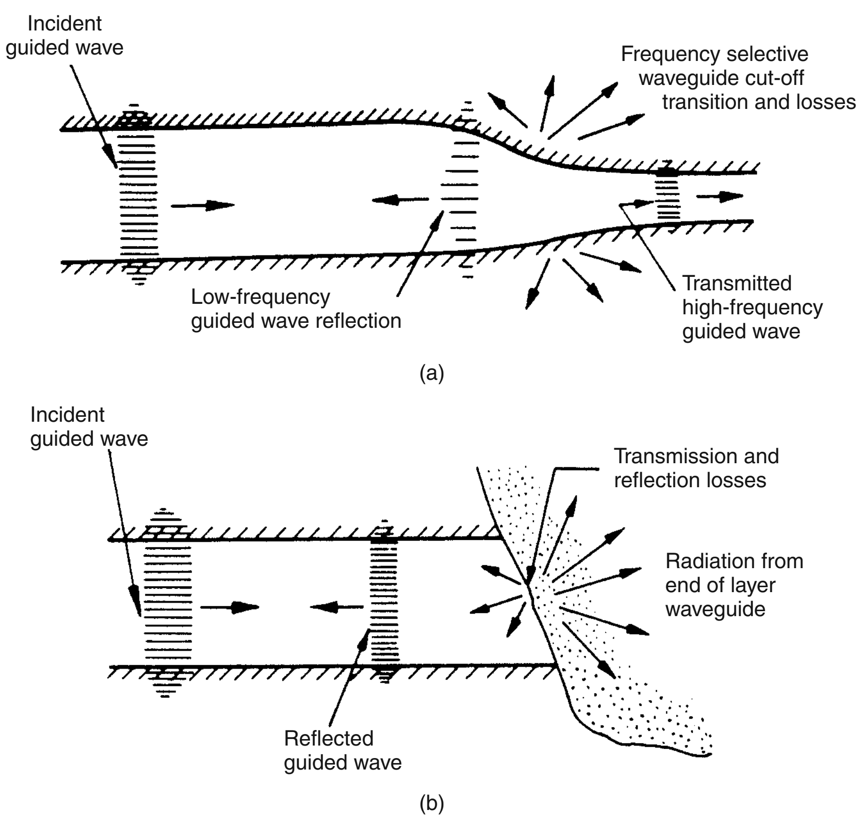



**Figure 8.17** Conceptual behaviour of channel waves on encountering (a) a coal seam pinchout; (b) a channel sand cutout; (c) a fault with a throw less than the seam thickness; (d) a fault with a throw greater than the seam thickness. From Jackson (1981), reproduced by permission of IEA Coal Research – The Clean Coal Centre.

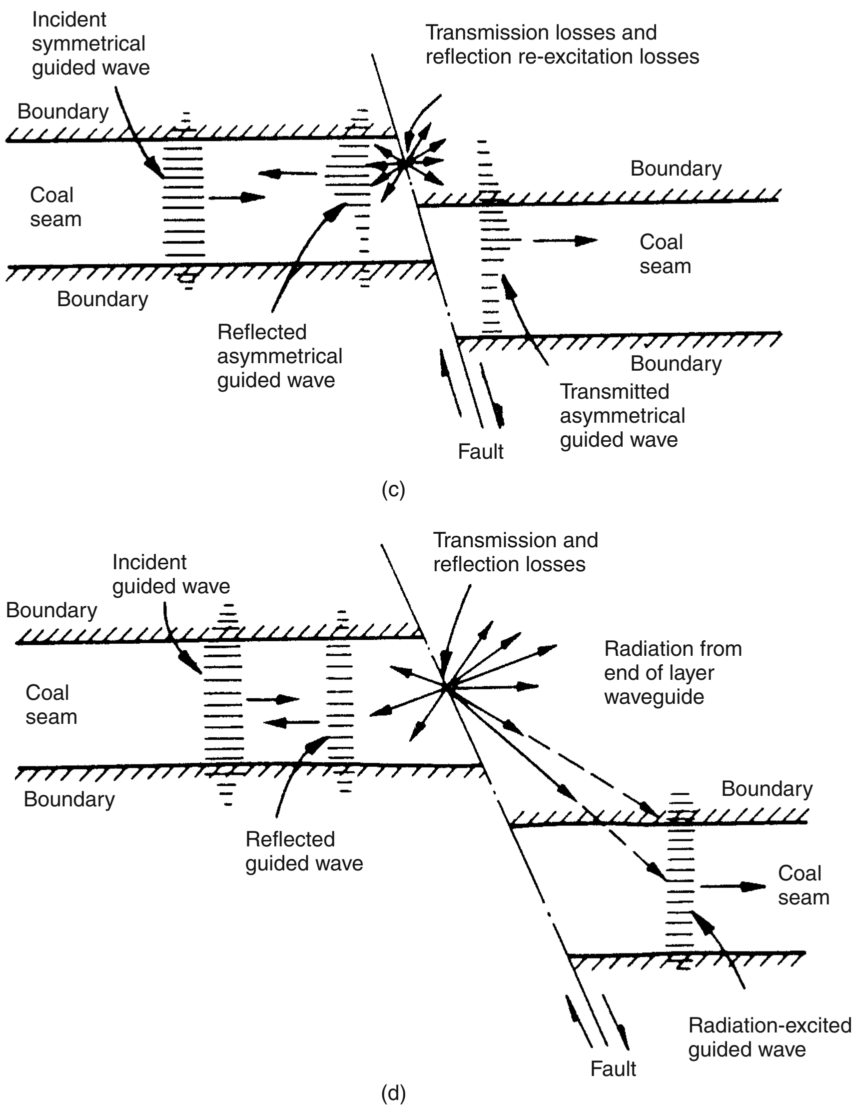



**Figure 8.17** *(continued)*

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) are the measurement of the variation with depth of particular physical properties of surrounding rocks with 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.

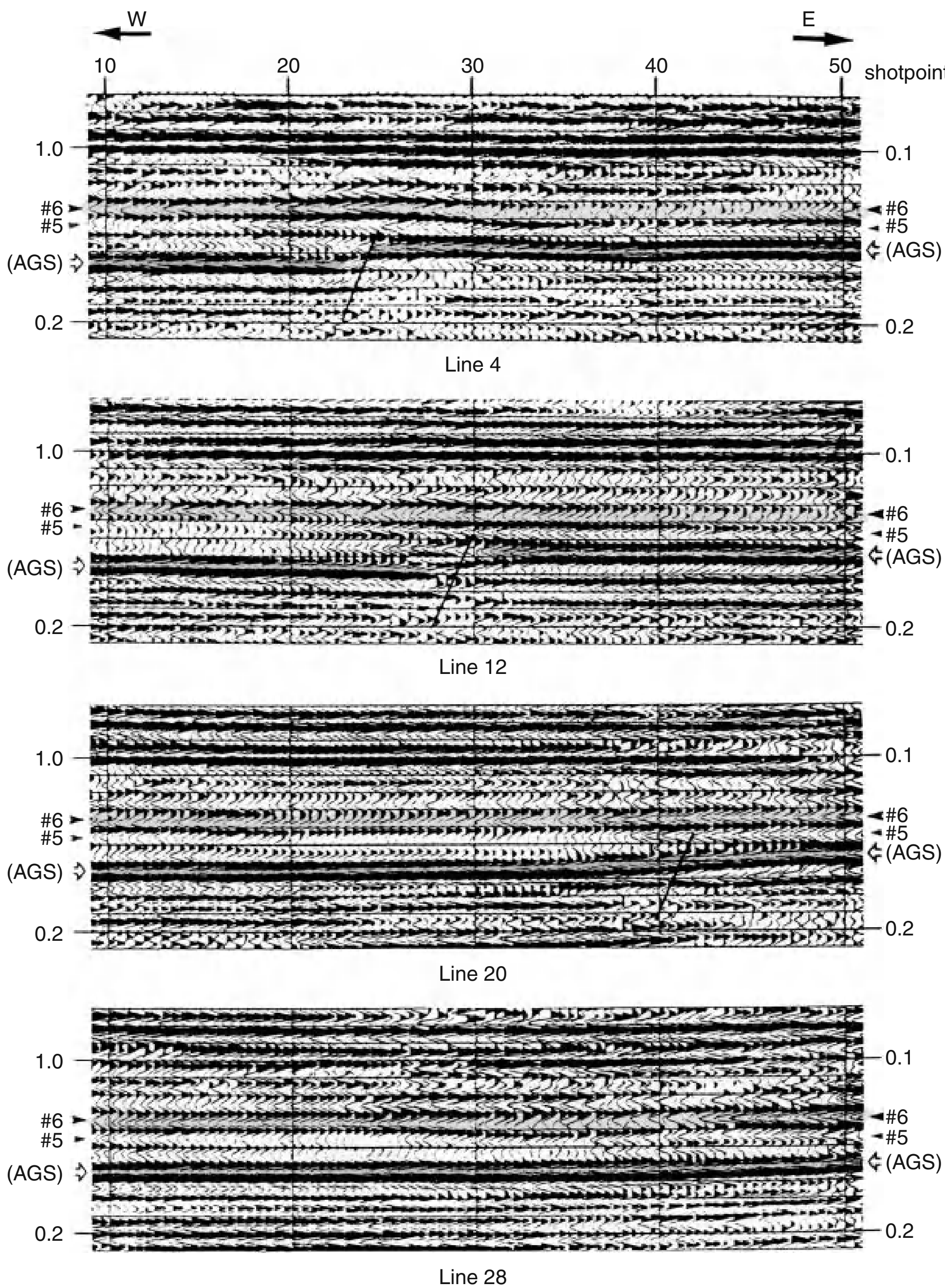



**Figure 8.18** Four line seismic sections 240 ft apart, showing a roll feature trending to the southeast direction beneath the 3D survey area. (Gochioco 2004) with authors and Society of Exploration Geophysicists permission. This figure is reproduced in colour in the Plates section.

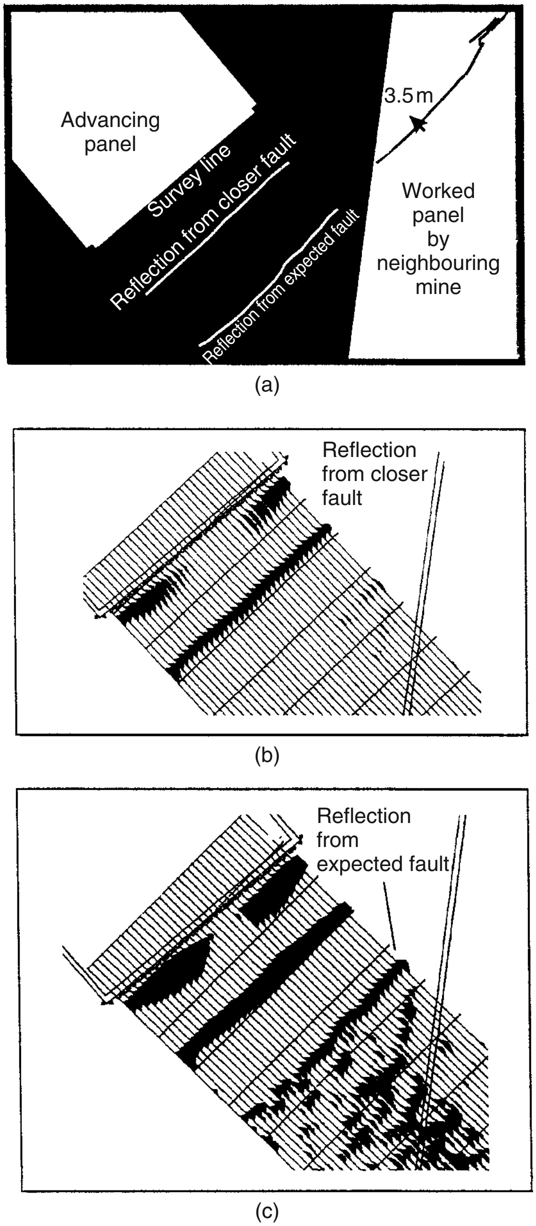



**Figure 8.19** In-seam seismic reflection survey used to detect faulting in advance of longwall mining. by permission IMC Geophysics Ltd.

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, with the latter 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 characterize 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 openholes 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;

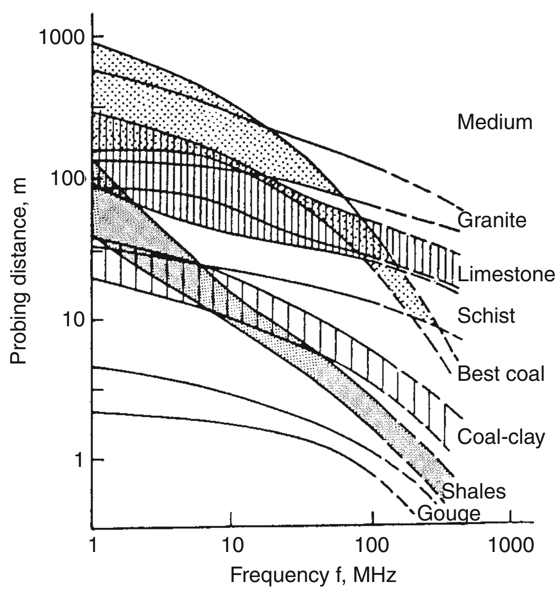



**Figure 8.20** Probing distances as a function for different geological lithologies (from Cook 1975 reproduced in Reynolds 2011) by permission of the author and Society of Exploration Geophysicists.

**Figure 8.21** Mobile geophysical logging unit. (Reproduced with permission of Robertson Geologging Ltd.) This figure is reproduced in colour in the Plates section.

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 resources and 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 standardized log units or recognized 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 isotope potassium-40 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 included clay material 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, which is also the case for very low radiation levels in clean coal. Figure 8.22 shows the relationship of the gamma ray 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 are often calibrated according to the American Petroleum Institute (API) Standards pit in the United States 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 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

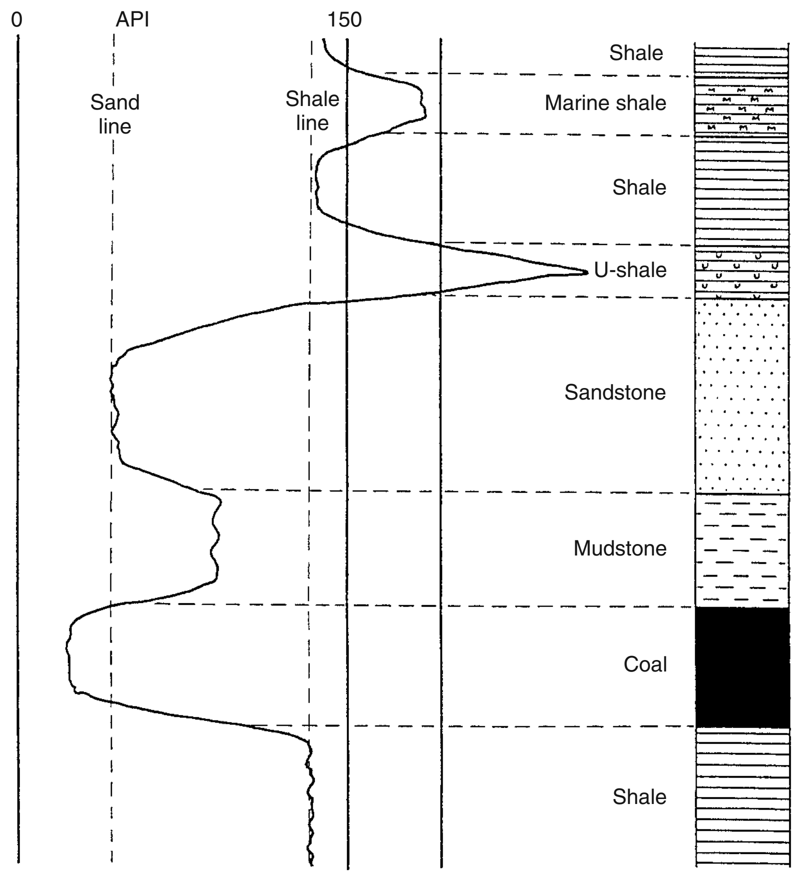



**Figure 8.22** Coal-bearing sequence lithotypes and gamma ray log response. (Reproduced with permission of Reeves Oilfield Services Ltd.)

casing tends only to attenuate the radiation received, with the shape of the log being 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 shows the relationship of the gamma ray log to lithology, where the lignite has low radioactivity but a sand containing a high percentage of detrital orthoclase feldspar rich in potassium has 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 back scattered, 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 affected equally 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 (caliper), 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 caliper 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 relationships between 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 g $cm^{-2}$ 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 characterizing 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 minimized 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

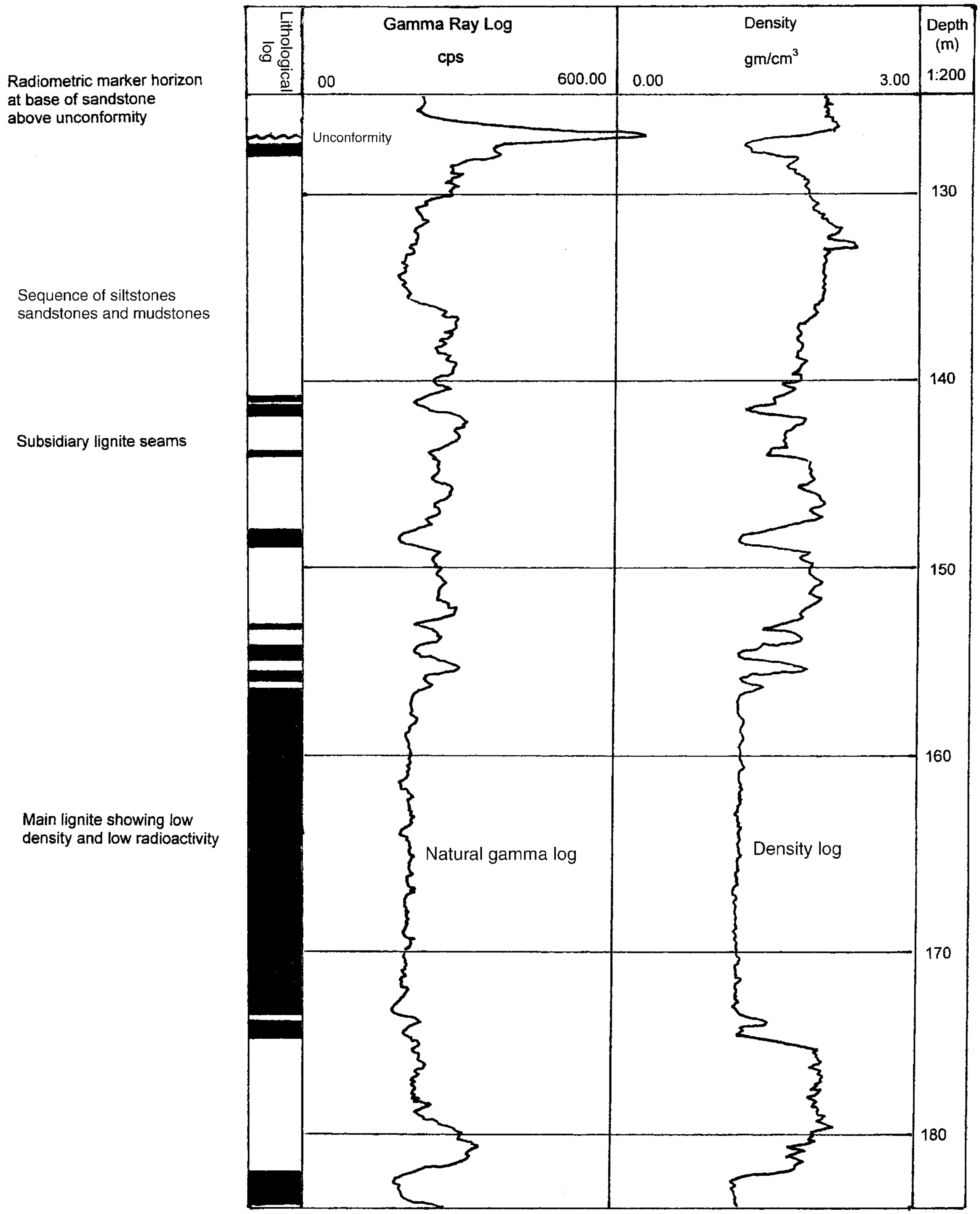



**Figure 8.23** Section showing gamma, density and resistivity logs over a principal lignite seam, Thar Coalfield, Pakistan. (Reproduced by permission of Oracle Coalfields plc.)

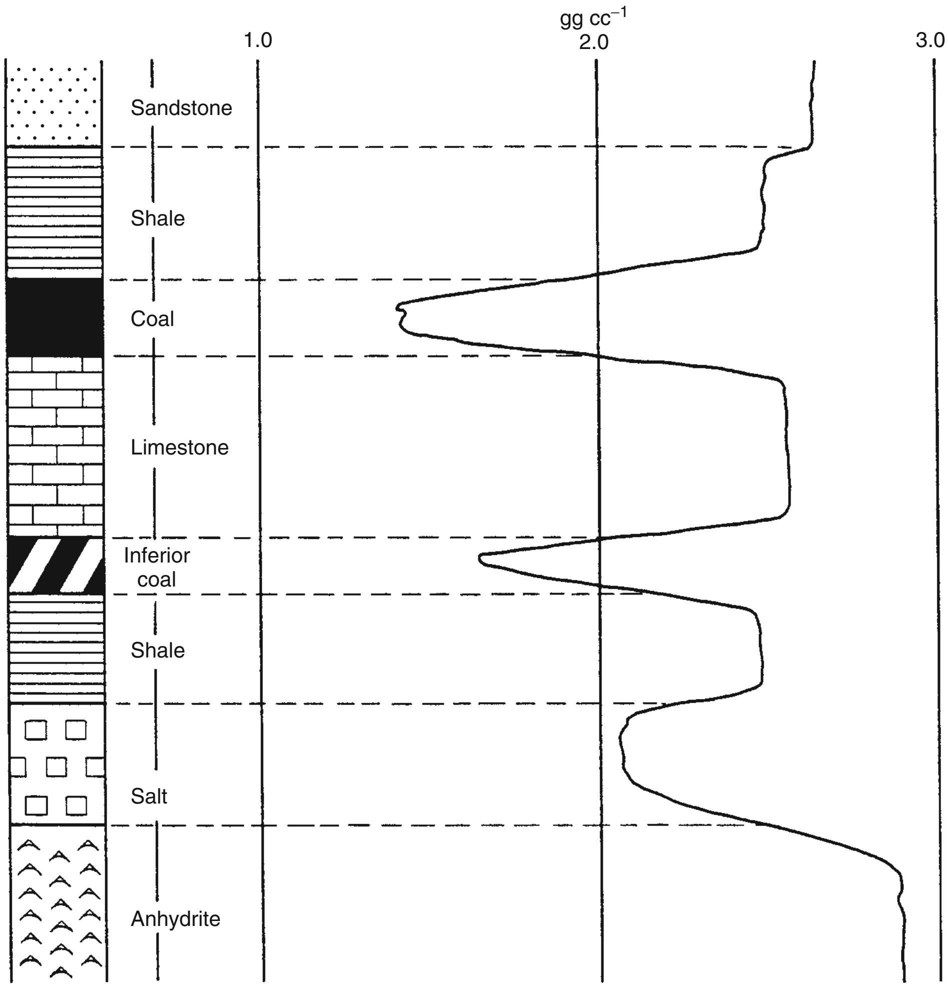



**Figure 8.24** Long spaced density log response to coal. (Reproduced with permission of Reeves Oilfield Services Ltd.)

that provides a continuous 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, which on an ash-free basis can be considered as a coal rank parameter. Where moisture is relatively constant, the neutron log

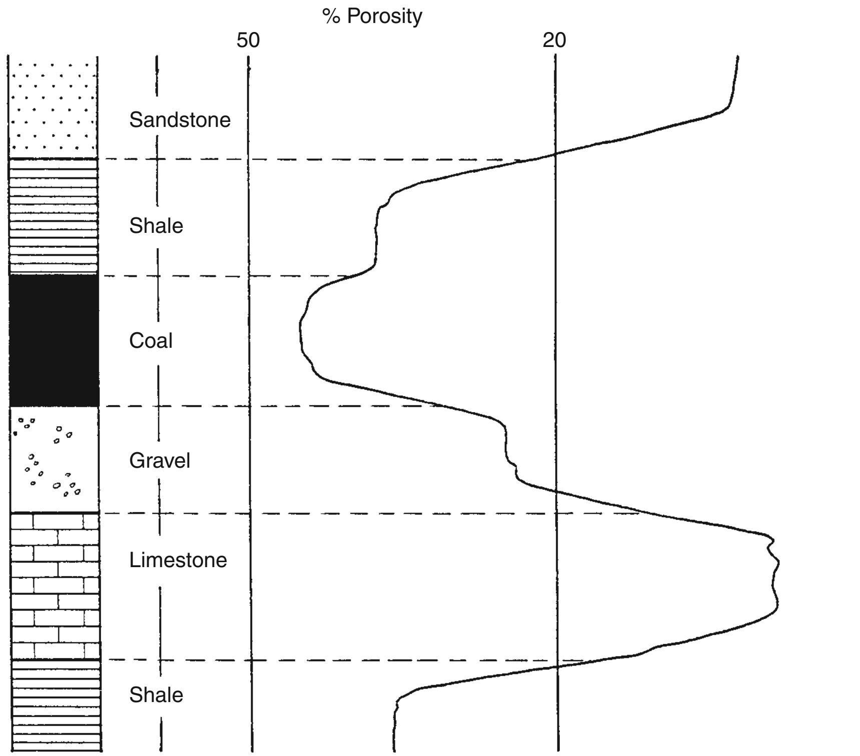



**Figure 8.25** Response of neutron log over a coal seam. (Reproduced with permission of Reeves Oilfield Services Ltd.)

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 cannot be used for such accurate interpretation of thickness. The same approach as used with density logs must be taken to arrive at average formation log values for use in quantitative work. From experience, bed boundary location can be interpreted approximately 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 synthesize rock quality indices, which can be related directly 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) describe 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 Fe, Si, Al, Mn, Ti, Zn, K and Ca. 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.

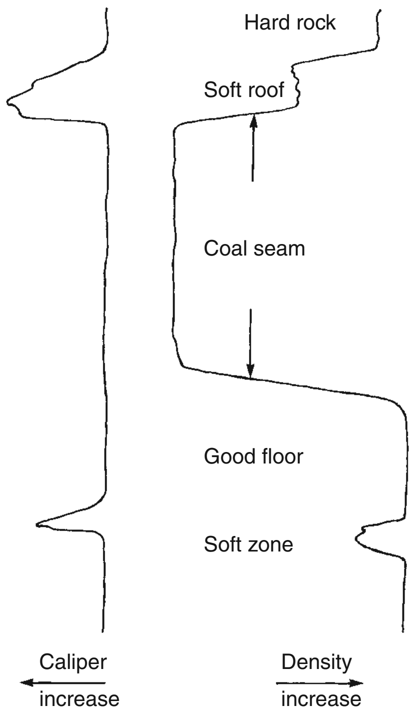



**Figure 8.26** Use of the combination of caliper and density logs to determine coal seam roof and floor characteristics. (Reproduced with permission of Reeves Oilfield Services Ltd.)

### 8.5.2 Caliper log

The caliper log measures the borehole diameter, and its main use is for correcting long and short spaced density readings. The caliper 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 caliper 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 caliper is travelling.

The caliper log in association with the density log can be used to indicate rock strength. Figure 8.26 shows the response of caliper and density logs across a coal seam that has a good sound floor, but has 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.

### 8.5.3 Electric logs

In the examination of coal-bearing sequences electric logs may be used to support radioactive logs, but are rarely

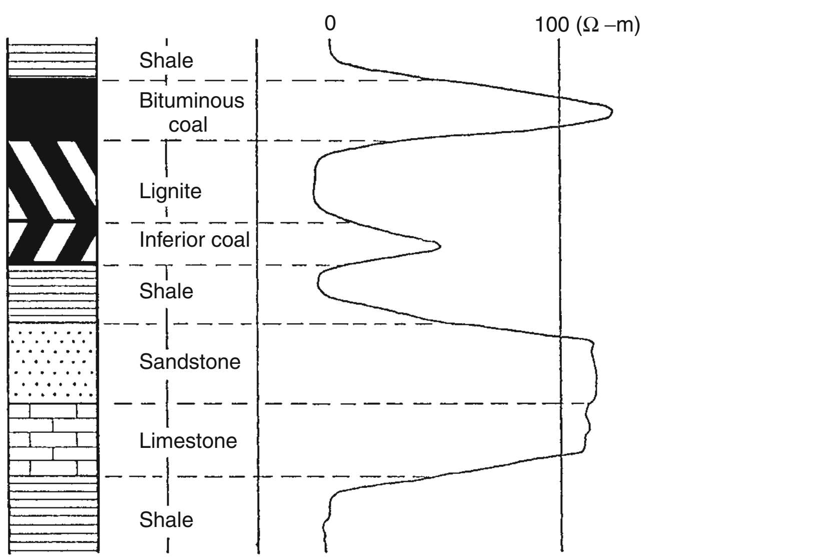



**Figure 8.27** Response of resistivity log to coal-bearing lithotypes. (Reproduced with permission of Reeves Oilfield Services Ltd.)

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 superceded 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 is oxidized 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, which 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 caliper arm holds the dipmeter pads containing the micro-resistivity electrodes against the borehole wall. The upper part contains the magetometers 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 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 tadpole plots for a shallow dipping coal seam in the United Kingdom.

### 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 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 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

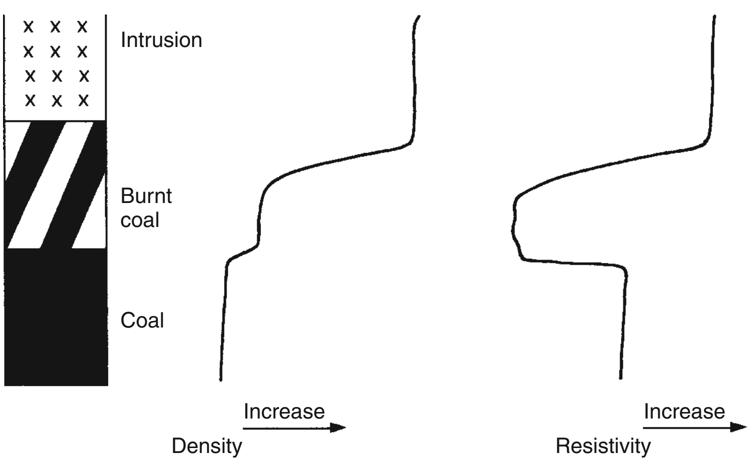



**Figure 8.28** Response of density and resistivity logs over a burnt coal zone. (Reproduced with permission of Reeves Oilfield Services Ltd.)

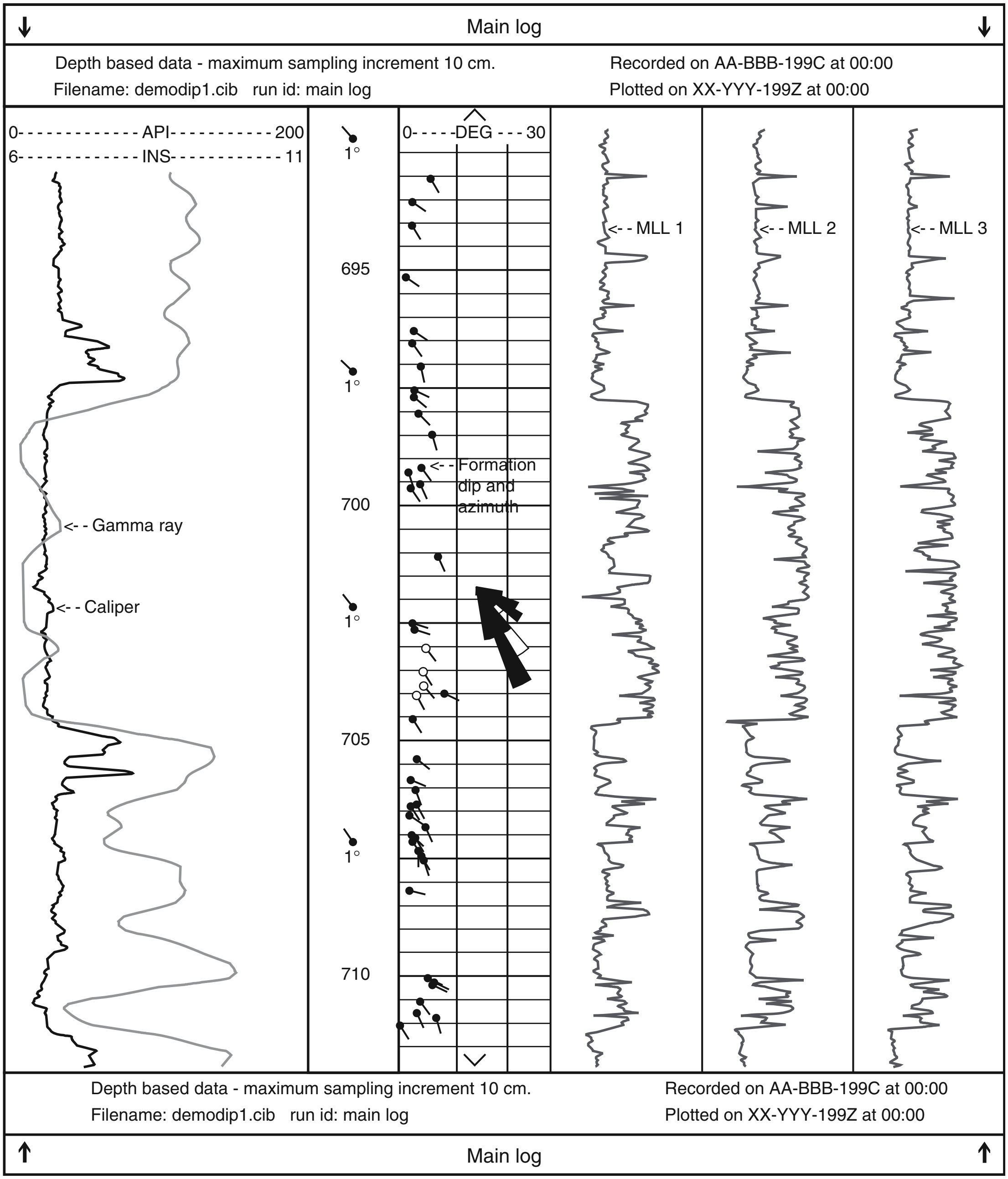



**Figure 8.29** Dipmeter tadpole plot at the target seam depth, Fillongly Hall borehole, United Kingdom. (From Firth (1999), reproduced with permission of Reeves Oilfield Services Ltd.)

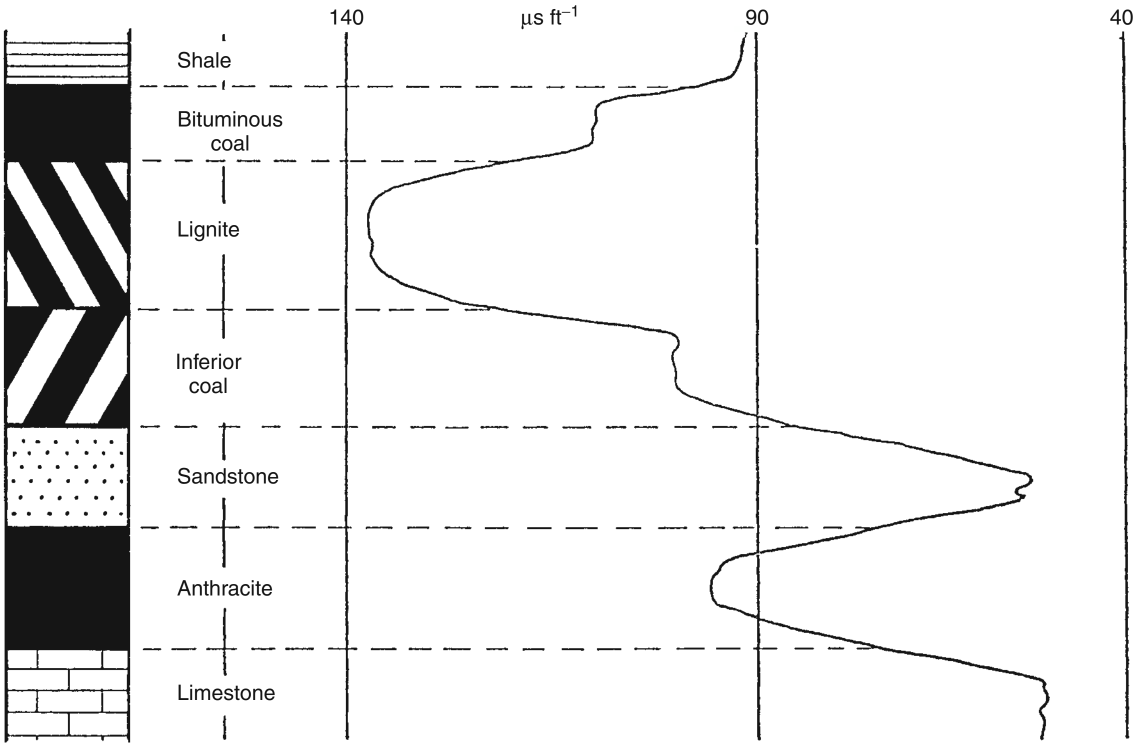



**Figure 8.30** Response of sonic log to coal-bearing lithotypes. (Reproduced with permission of Reeves Oilfield Services Ltd.)

with amplitude and travcl time characteristics that are measured 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, which 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 centralized in the borehole. The tool is equipped with a pair of optical in-line centralizers, each centralizer comprising three articulating polished steel arms. The arms expand and contract as the borehole changes shape, thereby maintaining centralization, 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, which facilitates depth correlation to core data and other openhole 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° caliper 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 maximizing roof and wall stability (see section 10.2.2.3).

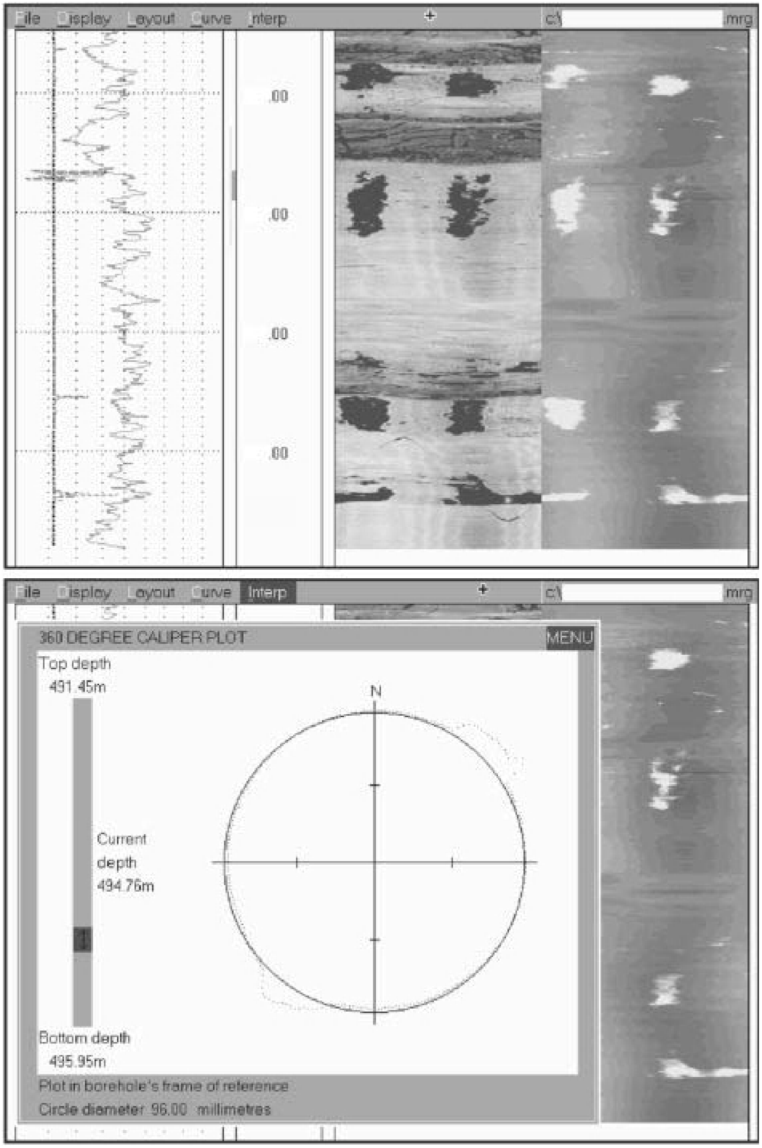



**Figure 8.31** Breakout as it appears on amplitude (left) and transit time (right) images (upper picture), and its portrayal on a 360° caliper plot (lower picture). (Reproduced by permission of Reeves Oilfield Services Ltd.)

### *8.5.7 Temperature log*

The standard bottom hole temperature is important in planning underground mining operations, and in particular, ventilation systems. Changes in the temperature log can also 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

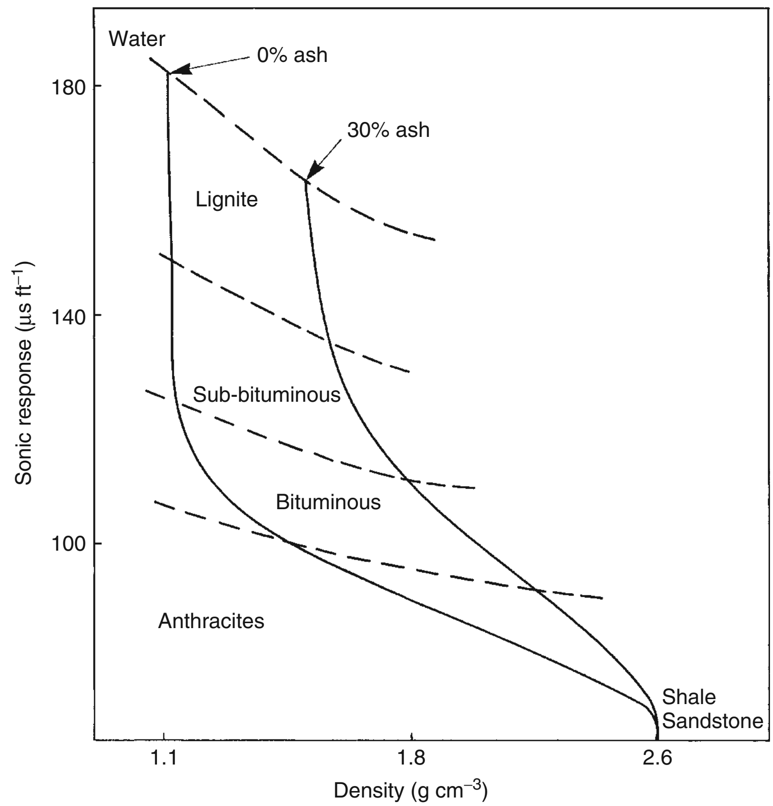
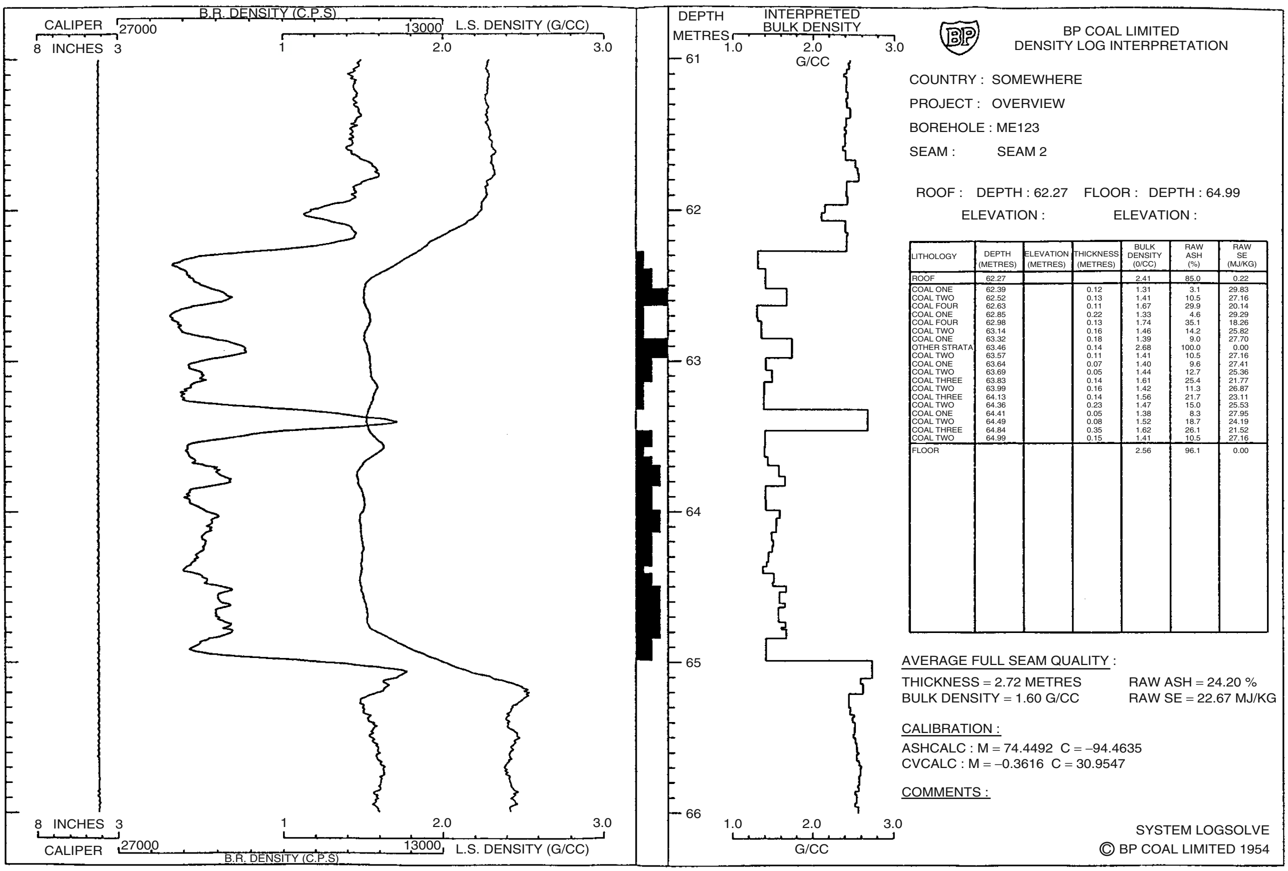



**Figure 8.32** Interpretation of coal rank from sonic and density logs. (Reproduced with permission of Reeves Oilfield Services Ltd.)

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 calorific value calculations together with density and caliper 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 that 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 caliper, the density end-point is determined by the desired matrix type, that is 2.65 g cm$^{-3}$ for a sandstone matrix. Figure 8.35 shows the cross-plot of depth-matched gamma ray and density data, where sandstones and shales are indicated and coal can be identified due to its unique combination of low gamma ray and low density. Coal is plotted where densities of less than 1.9 g cm$^{-3}$ are found, and caliper 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




| LITHOLOGY | DEPTH (METRES) | ELEVATION (METRES) | THICKNESS (METRES) | BULK DENSITY (0/CC) | RAW ASH (%) | RAW SE (MJ/KG) |
|---|---|---|---|---|---|---|
| ROOF | 62.27 | | | 2.41 | 85.0 | 0.22 |
| COAL ONE | 62.39 | | 0.12 | 1.31 | 3.1 | 29.83 |
| COAL TWO | 62.52 | | 0.13 | 1.41 | 10.5 | 27.16 |
| COAL FOUR | 62.63 | | 0.11 | 1.67 | 29.9 | 20.14 |
| COAL ONE | 62.85 | | 0.22 | 1.33 | 4.6 | 29.29 |
| COAL FOUR | 62.98 | | 0.13 | 1.74 | 35.1 | 18.26 |
| COAL TWO | 63.14 | | 0.16 | 1.46 | 14.2 | 25.82 |
| COAL ONE | 63.32 | | 0.18 | 1.39 | 9.0 | 27.70 |
| OTHER STRATA | 63.46 | | 0.14 | 2.68 | 100.0 | 0.00 |
| COAL TWO | 63.57 | | 0.11 | 1.41 | 10.5 | 27.16 |
| COAL ONE | 63.64 | | 0.07 | 1.40 | 9.6 | 27.41 |
| COAL TWO | 63.69 | | 0.05 | 1.44 | 12.7 | 25.36 |
| COAL THREE | 63.83 | | 0.14 | 1.61 | 25.4 | 21.77 |
| COAL TWO | 63.99 | | 0.16 | 1.42 | 11.3 | 26.87 |
| COAL THREE | 64.13 | | 0.14 | 1.56 | 21.7 | 23.11 |
| COAL TWO | 64.36 | | 0.23 | 1.47 | 15.0 | 25.53 |
| COAL ONE | 64.41 | | 0.05 | 1.38 | 8.3 | 27.95 |
| COAL TWO | 64.49 | | 0.08 | 1.52 | 18.7 | 24.19 |
| COAL THREE | 64.84 | | 0.35 | 1.62 | 26.1 | 21.52 |
| COAL TWO | 64.99 | | 0.15 | 1.41 | 10.5 | 27.16 |
| FLOOR | | | | 2.56 | 96.1 | 0.00 |

**Figure 8.33** Density log interpretation with interpreted coal seam bulk densities, raw ash content and raw calorific value calculations, together with density and caliper logs.

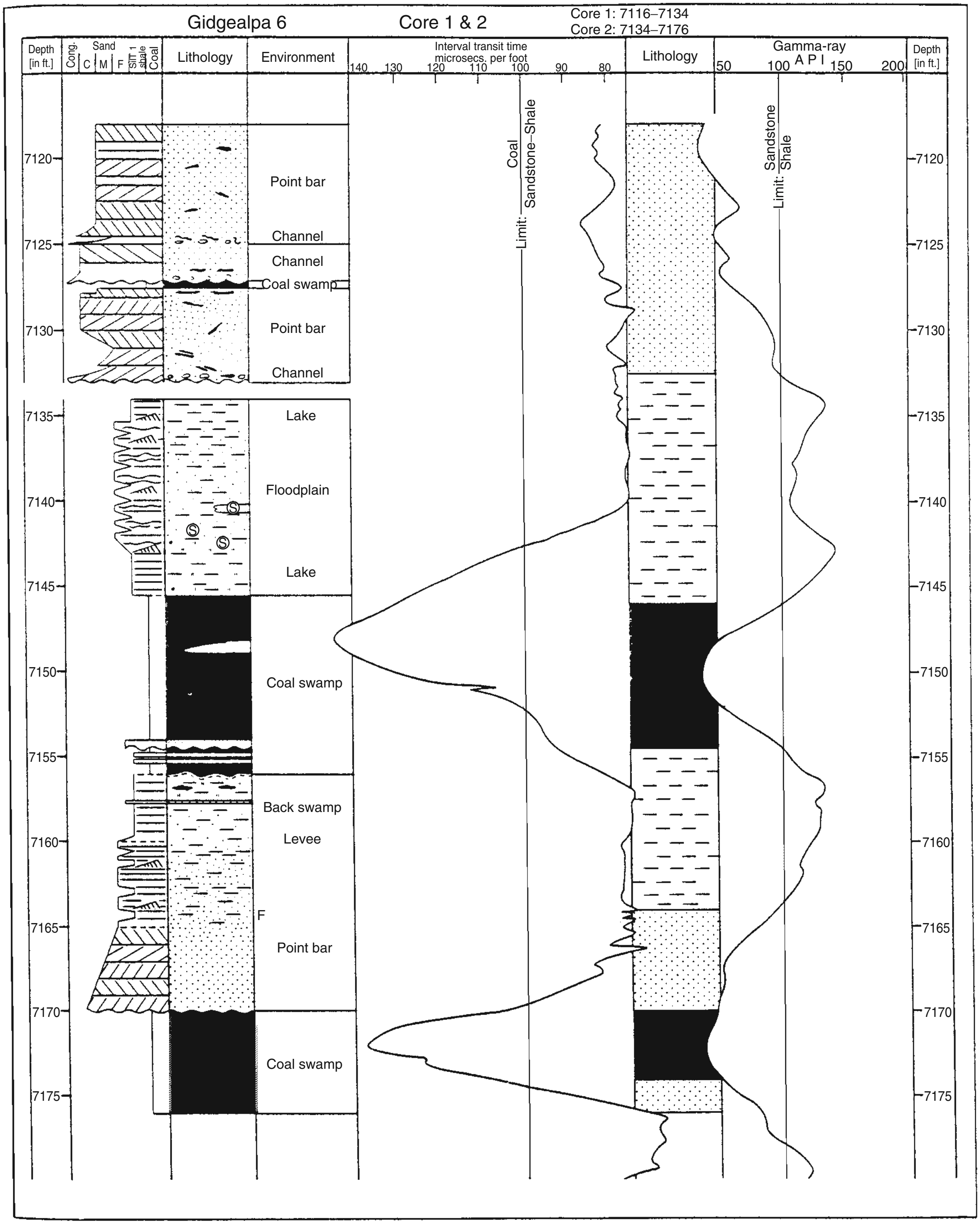



**Figure 8.34** Combination of lithological, sonic and gamma logs to identify coal seams and sandstones as potential hydrocarbon reservoirs. (From Thornton, 1979.)

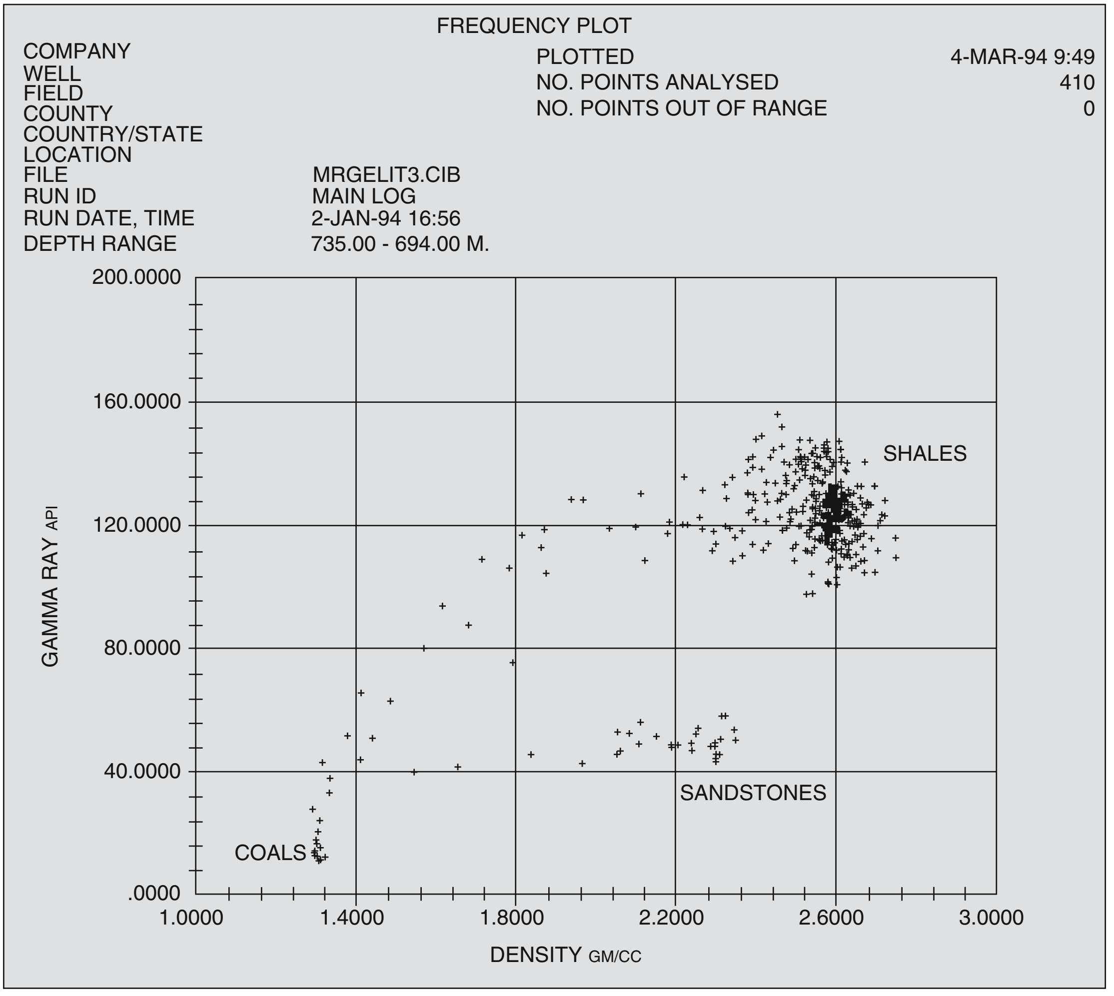



**Figure 8.35** Cross-plot of depth-matched gamma ray and density data. (Reproduced by permission of Reeves Oilfield Services Ltd.)

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 summarized 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 responses in a variety of lithologies is shown in Figure 8.37.

More sophisticated geophysical logging techniques can provide the following additional information: lithology

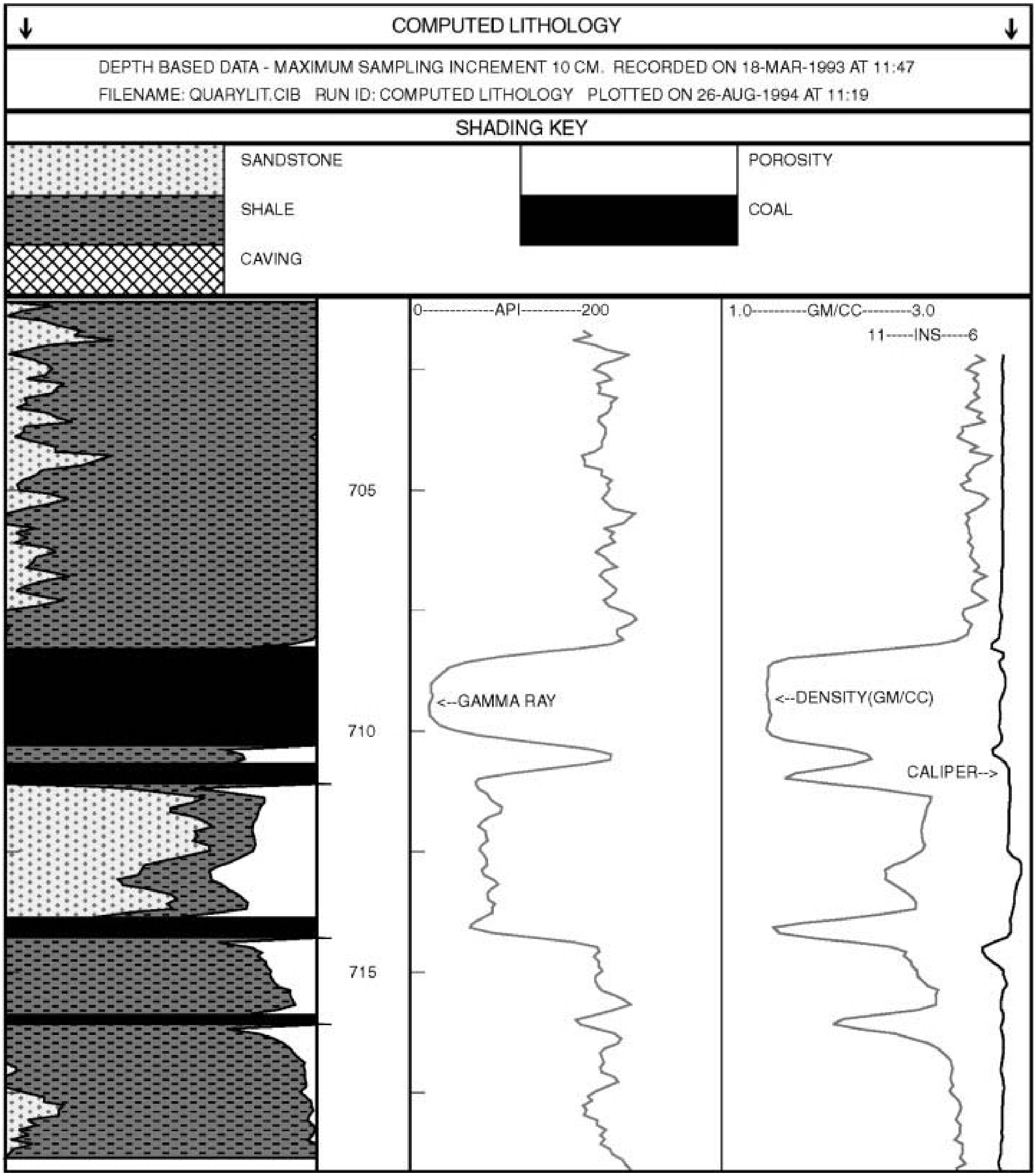



**Figure 8.36** Computed lithology analysis derived from data shown in Figure 8.34. (Reproduced by permission of Reeves Oilfield Services Ltd.)

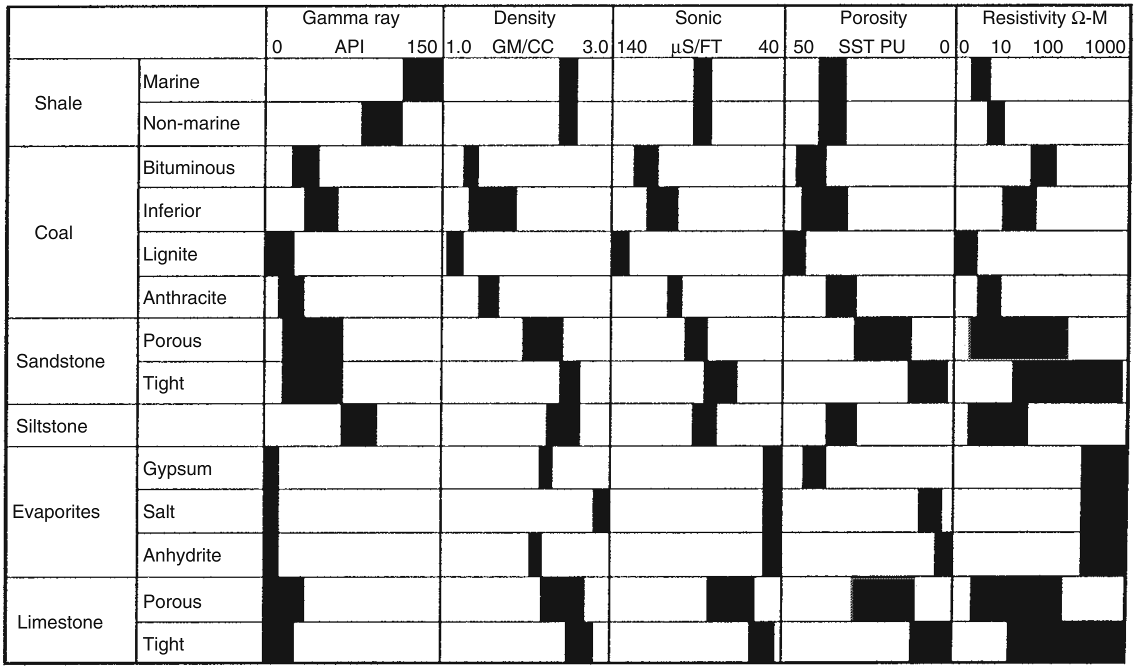



**Figure 8.37** Summary of log responses in various lithologies. (Reproduced with permission of Reeves Oilfield Services Ltd.)

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.

# 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, whereas 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 and groundwater will affect mine operations, for example, the National Coal Board (UK) publication '*Technical Management of Water in the Coal Mining Industry*' (1982). A summary of the basic properties of groundwater and those aspects of hydrogeology directly related to coal-bearing sequences and coal mining 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 vaporizes 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, that is 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 watercourses will need to be re-routed 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

*Coal Geology*, Second Edition. Larry Thomas.

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 at 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 are more commonly referred to as confining or impermeable horizons.

Groundwater usually flows under a hydraulic gradient, that is 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

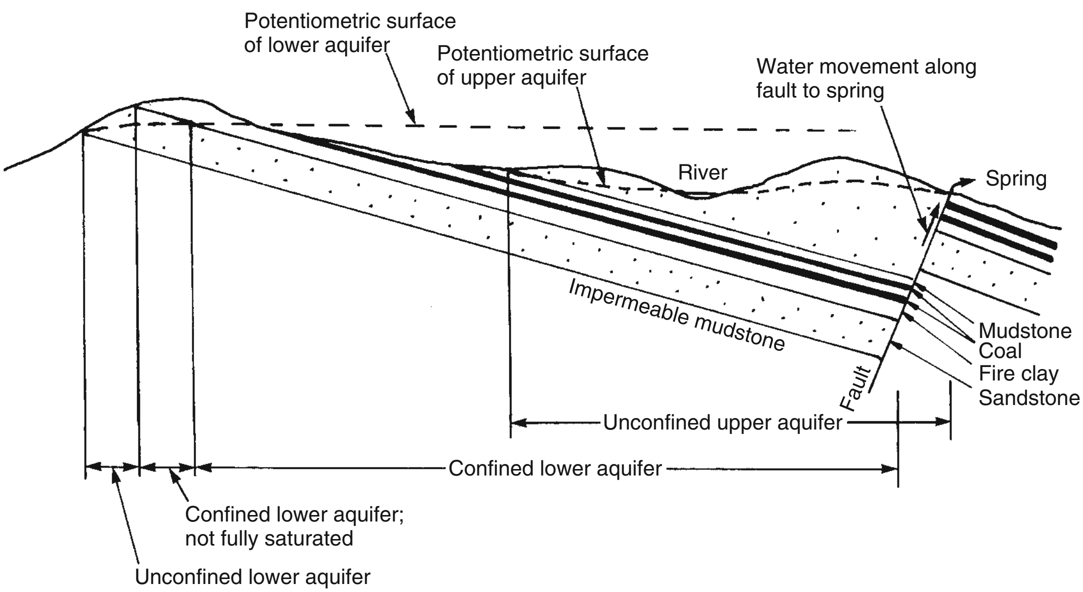



**Figure 9.1** Groundwater conditions in a coal-bearing sequence, showing an upper unconfined aquifer and a lower confined aquifer with an intervening coal–mudstone sequence.

a fluid will flow through a porous medium at a rate ($Q$) which is proportional to the product of the cross-sectional area through which flow can occur ($A$), the hydraulic gradient ($i$) and hydraulic conductivity ($K$), that is

$$Q = KiA.$$

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.

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 as well as 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 toward 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, and 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.

## 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, whereas 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 Paleogene–Neogene strata, 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, that is 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, which if open will hold and allow groundwater flow, that is 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.

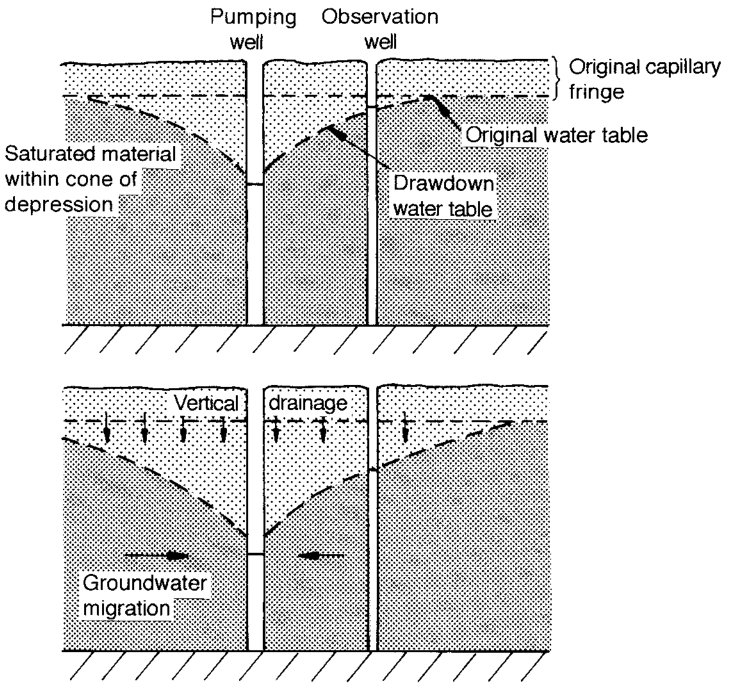



**Figure 9.2** Drawdown of the water table due to pumping in an unconfined aquifer. (From Price, 1996.)

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 because they have geotechnical significance as horizons of weakness when subjected to increased stress or sudden depressurization.

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, 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 due to the fact that coals are dominated by discontinuities, which 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 day$^{-1}$.

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 peat tends 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.

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, Nandi and Walker (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:

1. total pore volume $V_T$ accessible to helium as estimated from helium and mercury densities;
2. pore volume $V_1$ contained in pores >300 Å in diameter;
3. pore volume $V_2$ contained in pores 300–12 Å in diameter;
4. pore volume $V_3$ contained in pores <12 Å, $V_3 = V_T - (V_1 + V_2)$.

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:

1. porosity in coals with carbon content ($C$) <75% is predominantly due to macropores;
2. porosity in coals with carbon content ($C$) 85–91% is predominantly due to micropores;
3. porosity in coals with carbon content ($C$) 75–84% is associated with significant proportions of macro- meso- and microporosity.

**Table 9.1** Indicative porosities and hydraulic conductivities for unconsolidated and consolidated sediments that characterize coal-bearing sequences.

| Sediment type | Lithotype | Grain size (mm) | Porosity (%) | Hydraulic conductivity ($K$) (m/d) |
|---|---|---|---|---|
| Unconsolidated | 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 |
| Consolidated | 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 | $10^{-4}$–$10^{\dagger}$ |
| | Limestone | Variable | 0.1–30* | $10^{-5}$–$10^{\dagger}$ |

*Secondary porosity.
†Secondary permeability.
Source: Brassington (1988).

**Table 9.2** Gross open pore distributions in coals. See text for definitions of distributions.

| Sample | Rank* | $C$ (% d.a.f.) | $V_T$ (% $cm^3\,g^{-1}$) | $V_1$ ($cm^3\,g^{-1}$) | $V_2$ ($cm^3\,g^{-1}$) | $V_3$ ($cm^3\,g^{-1}$) | $V_3$ (%) | $V_2$ (%) | $V_1$ (%) |
|---|---|---|---|---|---|---|---|---|---|
| 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 | Nil | 27.0 |
| PSOC-135 | mv | 88.3 | 0.042 | 0.016 | 0.000 | 0.026 | 61.9 | Nil | 38.1 |
| PSOC-4 | hvA | 83.8 | 0.033 | 0.017 | 0.000 | 0.016 | 48.5 | Nil | 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 | Nil | 59.1 |
| PSOC-89 | Lignite | 63.3 | 0.073 | 0.064 | 0.000 | 0.009 | 12.3 | Nil | 87.7 |

*Bituminous coals: lv = low volatile, mv = medium volatile, hv = high volatile.
Source: Gan, Nandi and Walker (1972).

## 9.4 Collection and handling of hydrogeological data

During any mining operation it is important to minimize 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 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 are 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 flow of rivers and streams is calculated by measuring the water velocity and the river cross-sectional area, or by installing a weir, the former more suitable for rivers, 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, 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, with their siting 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 enable the monitoring boreholes to be sited favourably. Once the set of boreholes or wells has been constructed, hydraulic testing can be carried out. Controlled test pumping consists of a constant rate test followed by a recovery test, both of selected duration dependent on the geological character of the sequence being tested. Such testing is designed to determine the hydraulic properties of any aquifer present and the hydraulic interconnection and hydraulic gradients within these aquifers. Figure 9.3 shows a pumping test in operation in the Thar Coalfield, Pakistan.

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

**Figure 9.3** Pumping test in operation in the Thar Coalfield, Pakistan. (Photograph by Dargo Associates Ltd.)

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 also can 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 9.4 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 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.

**Figure 9.4** Piezometer group measuring water levels in two formations in an opencast working. (Photograph by LPT.)

Observation boreholes and piezometers are regularly placed to determine groundwater levels and the pressurization field around the proposed position of a high wall 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.

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 actual cross-section 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 utilize this 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

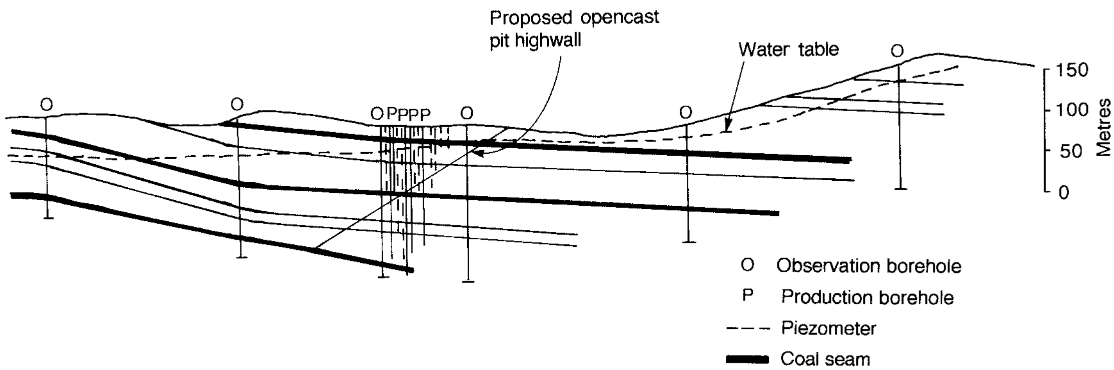



**Figure 9.5** Section across proposed opencast pit showing position of observation boreholes, production boreholes and piezometers, placed to determine groundwater levels and pressurization field around proposed position of highwall in opencast pit.

enter the mine. Some aquifers have a finite flow, 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, which can produce a large volume of water that 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 either 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,

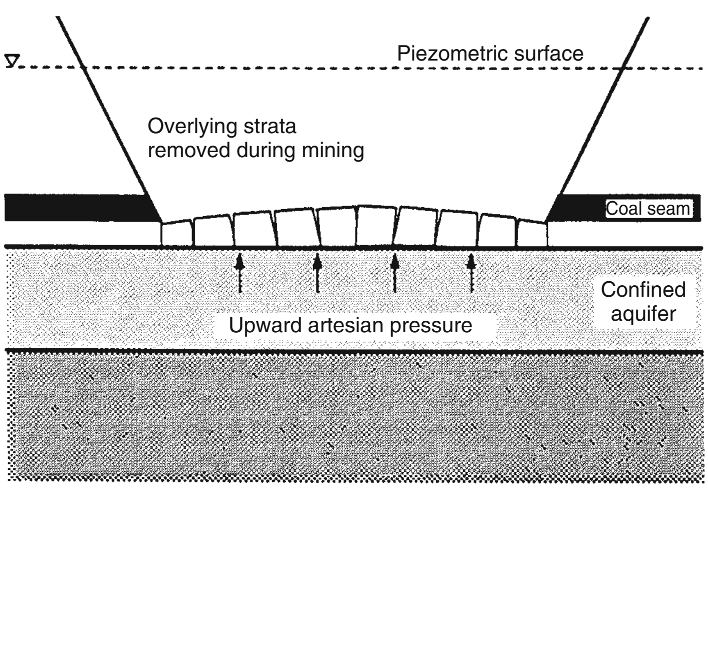



**Figure 9.6** Floor heave resulting from upward artesian pressure. (From Clarke (1995), with permission of IEA Coal Research.)

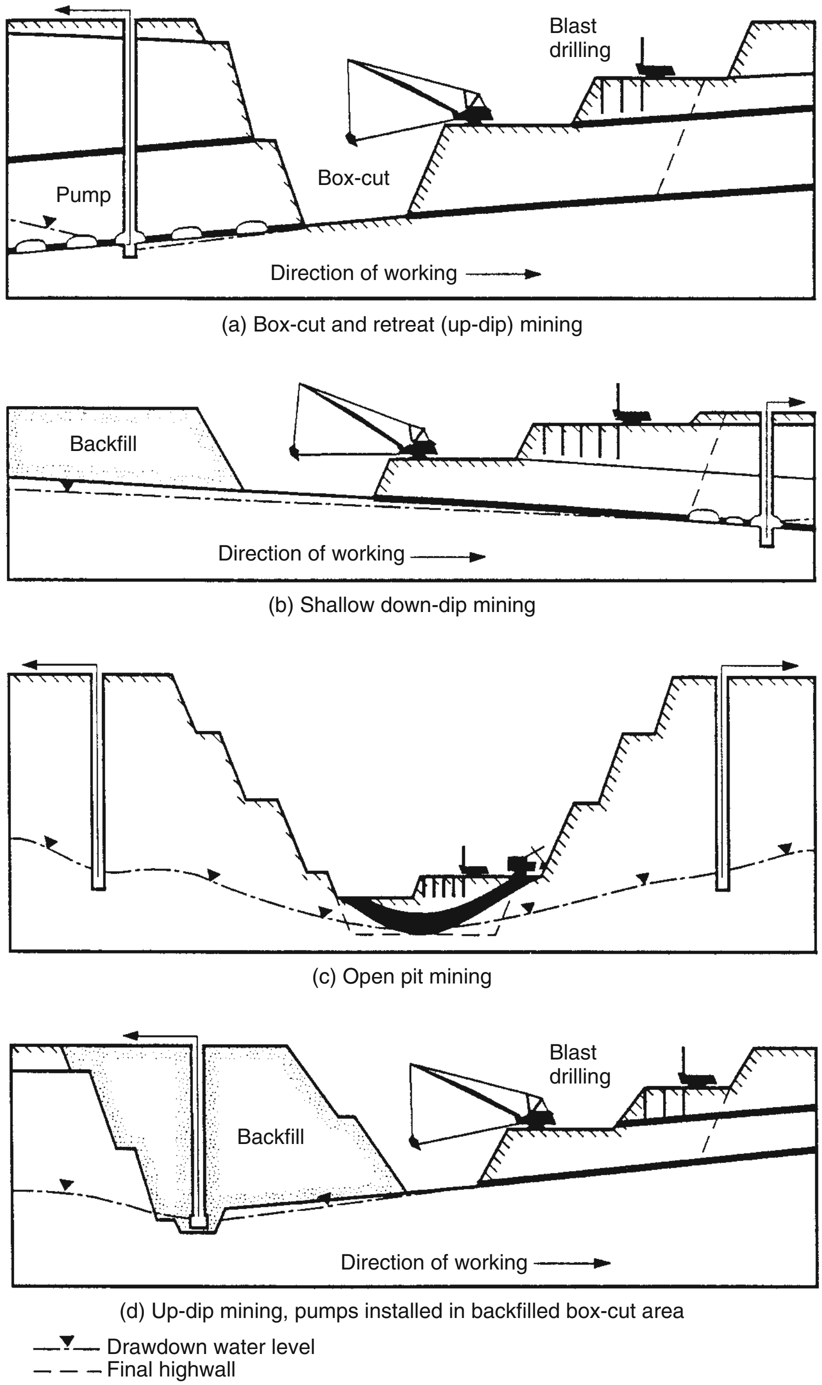



**Figure 9.7** Types of opencast mining in which advance dewatering methods are used. (From Clarke (1995), with permission of IEA Coal Research.)

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, which 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 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

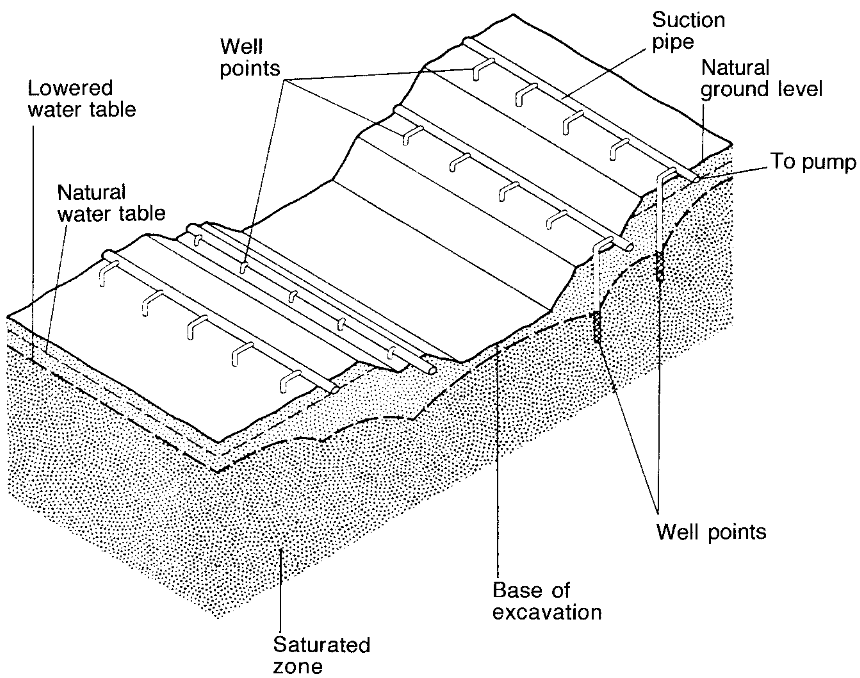



**Figure 9.8** Site dewatering. A two-stage dewatering scheme using well points to lower the water table below the base of an excavation. (From Price, 1996.)

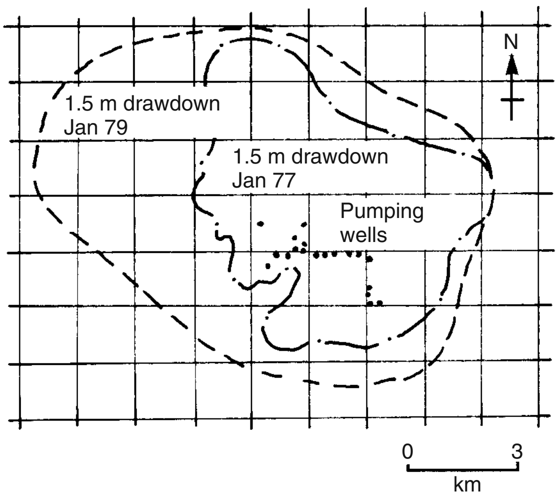



**Figure 9.9** Migration of the cone of depression due to pumping. (From Clifton, 1987.)

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, 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).

The opencast Drmno lignite mine in Serbia is located close to the confluence of the Danube and Mlava Rivers. The lignite-bearing sequence is characterized by Quaternary and Paleogene–Neogene sands and clays with lignite seams of varying thicknesses. The lignites exhibit numerous discontinuities which 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, Subaranovic and Polomic, 2008).

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, sited as the mine advances, a waterproof screen, and bench and drainage channel systems within the lignite workings. 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 post-lignite mining (Pavlovic, Subaranovic and Polomic, 2008). Figure 9.10 shows the final layout of the dewatering and water management measures required to safeguard the mine from water incursion (Pavlovic, Subaranovic and Polomic, 2008).

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 the piezometric surface. The brown coal mines of Germany produce over 100 Mt $yr^{-1}$ and operate at depths up to 400 m. At Hambach, the pit area is up to 20 $km^2$ and the depth is up to 400 m. From this mine alone, $350 \times 10^6$ $m^3$ of water have 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, which can affect buildings, underground conduits, and roads and railways in the area. Such changes can have strong influences on local populations, for example 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 9.11 shows the predicted effect of dewatering an area proposed for opencast mining in India. Figure 9.11a shows the existing water table, which slopes westwards towards a major water course. Figure 9.11b 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 2 yr of pumping, the likely effect is shown in Figure 9.11c, where a maximum lowering of the water table is 64 m. The lowering of the water table will affect an area 10–12 $km^2$ and will cause the local shallow

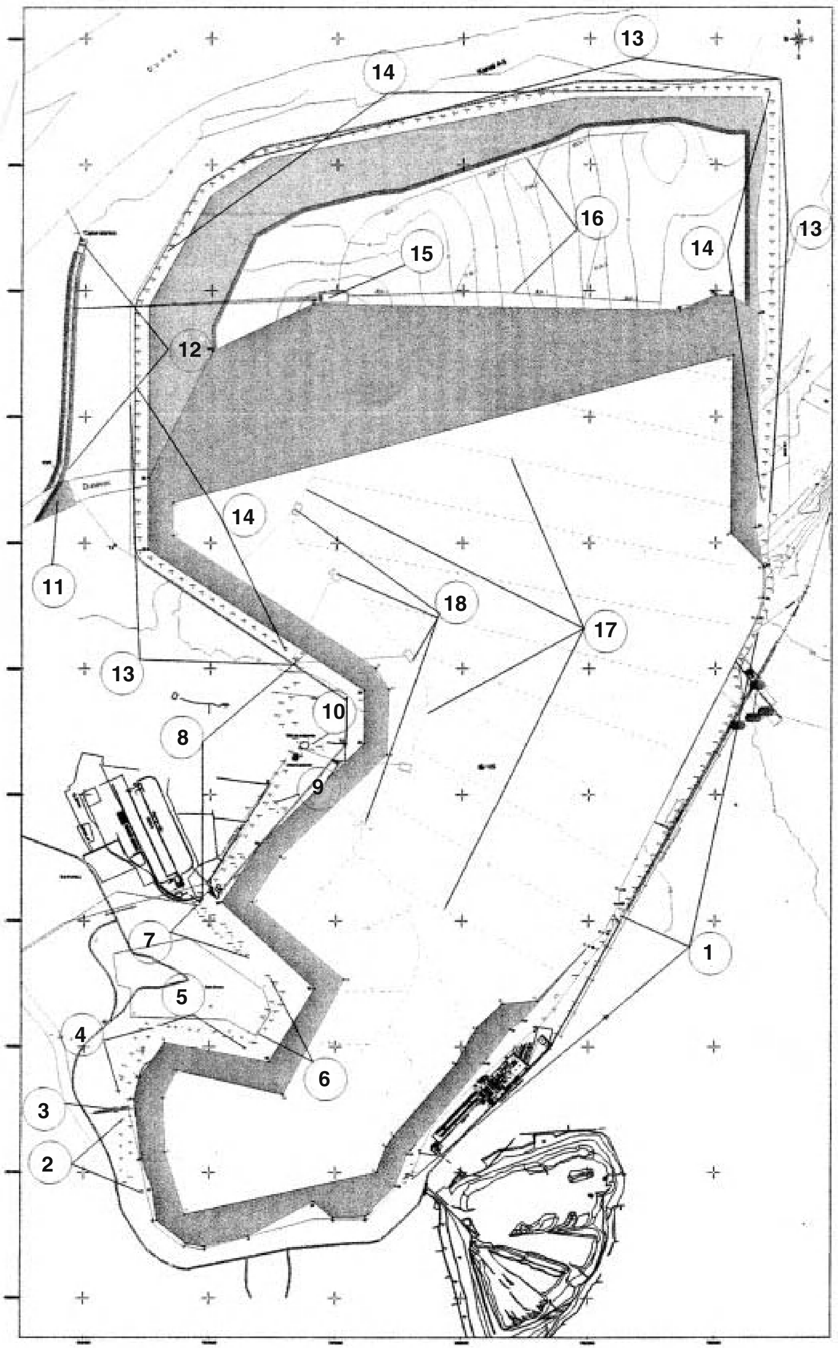



Figure 9.10 System for protection of the Open Cast mine Drmno against water at the end of mining (Pavlovic *et al* 2008) with permission.

| | |
|---|---|
| 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 9.10** (*Continued*)

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 depressurization wells, to reduce hydraulic pressure in an aquifer, perhaps to improve slope stability or prevent floor heaving. A more specialized technique is the use of sealing walls, this involves the construction of an impermeable barrier between a prolific source of groundwater and the pit. 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, permitting greater consolidation and thus subsidence, particularly in the floor of the opencast pit. Conversely, dewatering will be required to depressurize the pit floor, the high wall 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, which otherwise may be subject to failure, particularly when shear zones and other discontinuities have been identified.

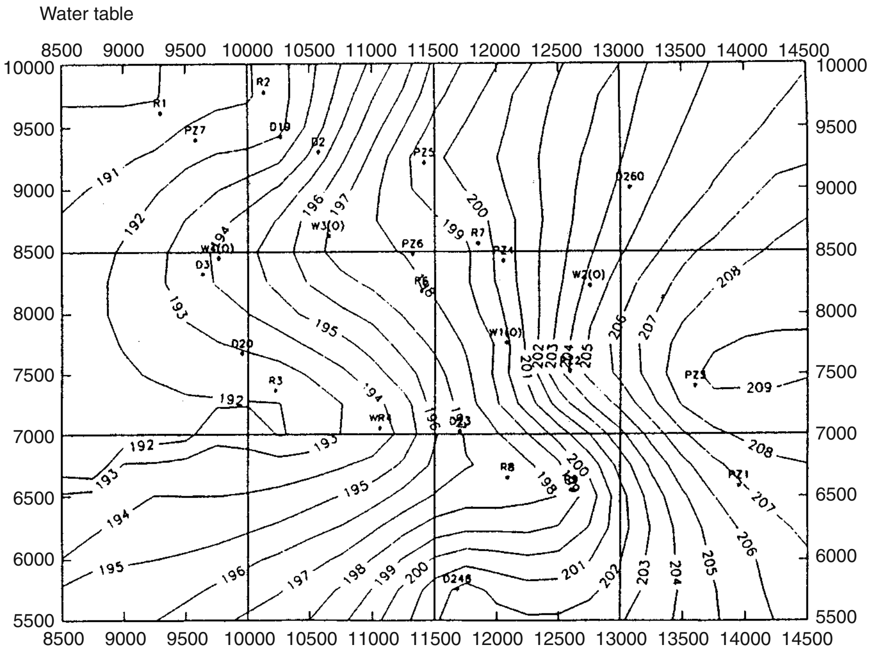
All of these measures are designed to control the groundwater conditions surrounding and within opencast coal mining operations, and are now a major factor in assessing the technical and financial viability of such mines.

### 9.5.2 Dewatering of underground mines

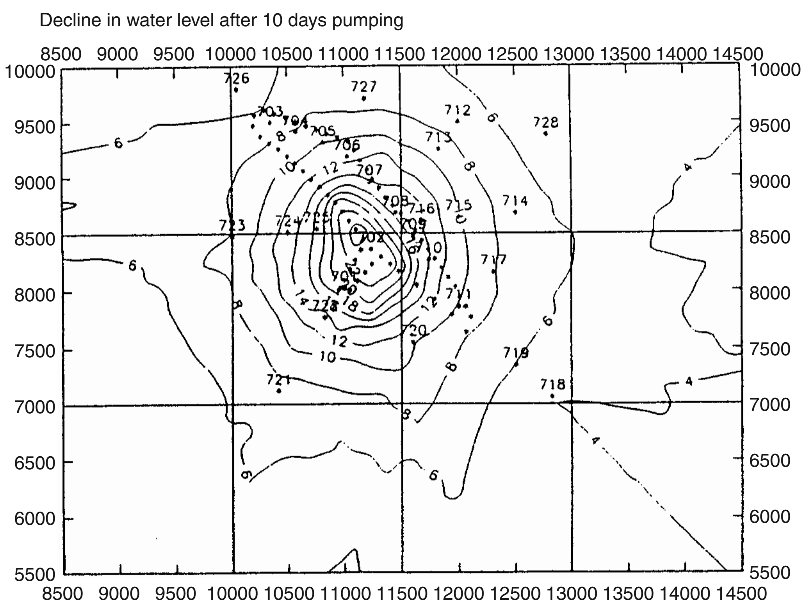
Numerous underground mining operations require the removal of water from the mine workings. Many mines, especially in old mining districts, are connected underground, so that pumping in one mine can control water levels in neighbouring mines. Water entering underground coal operations can be from several sources. From groundwater flow, from natural precipitation at the surface and from abstractions from rivers, wells, etc. 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, which may enter mines at greater depths; the latter effect occurring 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 co-ordinated. 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,

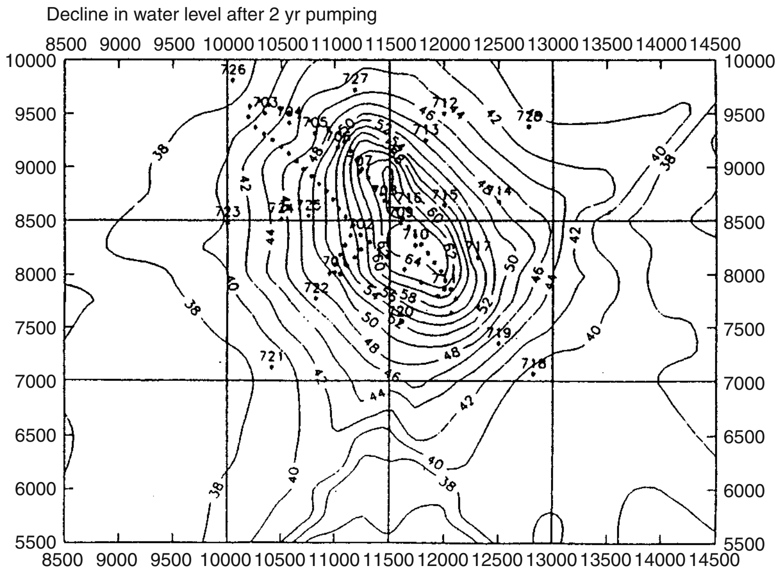



(a)




(b)

**Figure 9.11** Theoretical study to predict the decline in water level after selected periods of pumping, for a proposed opencast mine in India. Drawdown levels are predicted to increase to 64 m after 2 yr pumping.




(c)

**Figure 9.11** (*Continued*)

faults and interseam connections must be recorded. Effective use of all of these 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 United Kingdom, mine waters pumped from all active underground mines and adjacent abandoned workings can total up to $1 \times 10^6$ m$^3$ per day; including the Nottinghamshire Coalfield totalling $14 \times 10^6$ m$^3$ in 1991, and the Durham Coalfield totalling $70 \times 10^6$ m$^3$ in 1994. Similarly in Germany, the Ruhr Coalfield pumped $114 \times 10^6$ m$^3$ in 1994 (Clarke, 1995). It is an ongoing operation and expense to drain water from underground workings both to facilitate working conditions but also to minimize 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 utilized 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 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 oxidize that coal immediately in contact with it. This has the effect of reducing the coal quality (see section 4.3.4).

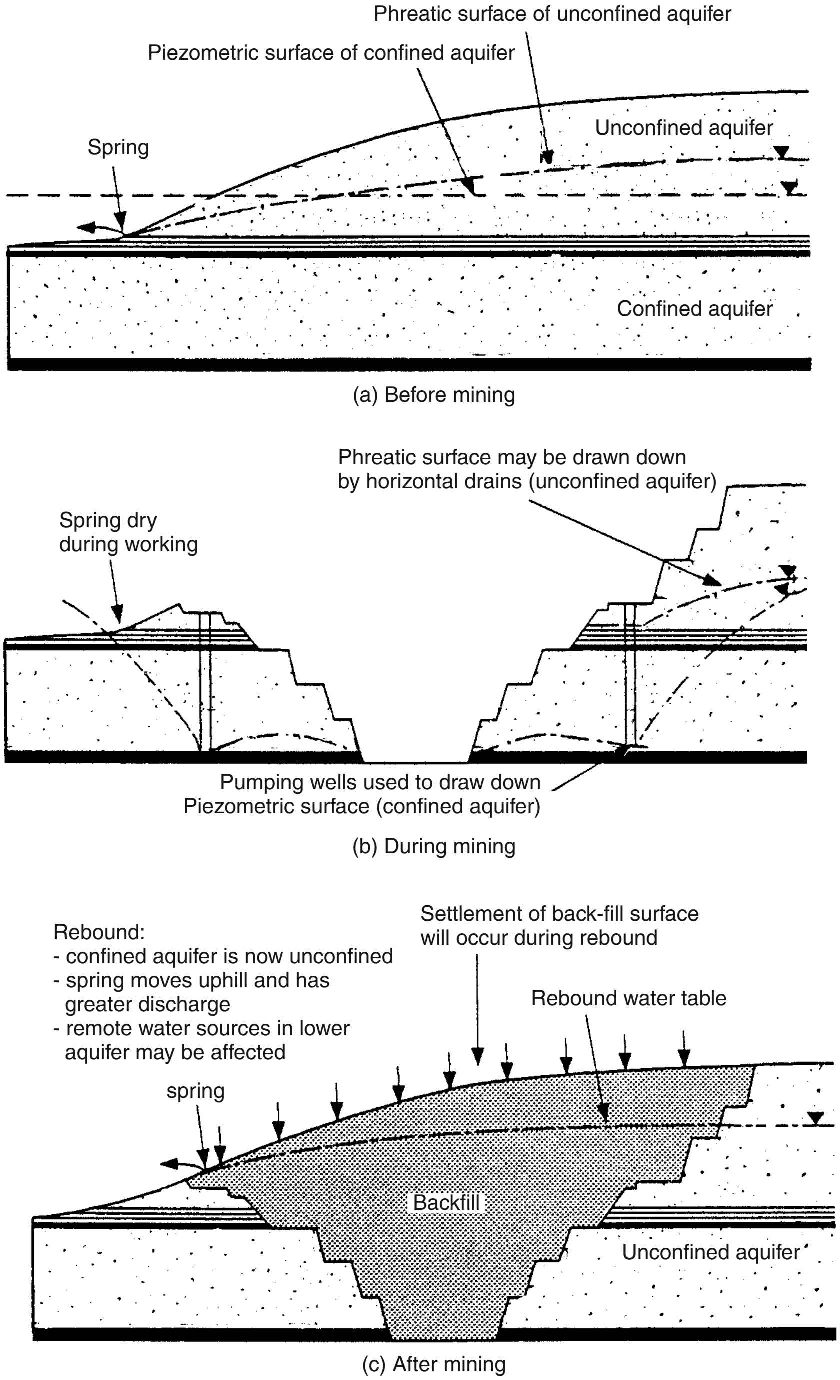



**Figure 9.12** Rebound effect on the water table after cessation of opencast mining, showing changes to the local aquifer system. (From Clarke (1995), with permission of IEA Coal Research.)

## 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 9.12 shows the affect of dewatering a mine and the resultant water table rebound effect once pumping has ceased. The original situation in Figure 9.12a 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 9.12b. However, after the mine has ceased working and been backfilled and pumping has been stopped, both aquifers would now be unconfined and a water table rebound would occur, as shown in Figure 9.12c. The time taken for this to occur will depend upon the rate of recharge to the system and also the nature and permeability of the backfill material.

Closure of deep mines in Yorkshire, United Kingdom, 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) have 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 are about to do so, and which 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. Coal mining takes place in the geological medium, consequently geological conditions have a profound influence throughout the life of a mining operation. In this context, coal mining 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, for example engineering geology and geotechnics, hydrogeology, coal analysis and detailed geophysics. The ultimate aim is to maintain the coal mine'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 mechanized operation and the selection of the mine layout to optimize 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 700 yr and in most other major coal producing countries for over 150 yr, 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.

Coal mining 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 utilize coals of all ranks and qualities in the industrial process.

## 10.2 Underground mining

To develop an underground coal mine, four main operations have to be carried out, namely: (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

*Coal Geology*, Second Edition. Larry Thomas.

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 2 yr or more; the deepest shafts in the United Kingdom (comparable to coal mines anywhere) are 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 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 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 United States and China, this makes development quicker and yields a coal tonnage at the same time, but is only possible when the coal is 2–3m thick and has a strong roof.

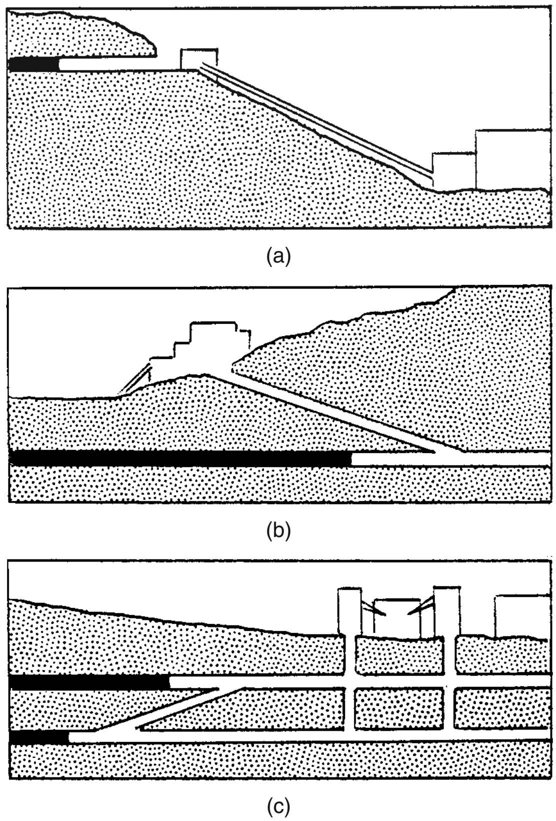



**Figure 10.1** Methods of entry for underground mines. (a) In-seam adit; (b) inclined shaft; (c) vertical shafts. (Adapted from Ward, 1984.)

### *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, for example 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 (United Kingdom), North Vietnam, southwest 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, and a roof formed by a sill may not 'cave' or subside regularly in longwall workings, which thereby cause problems. In room and pillar workings, such a 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 will also extend as part of an ongoing programme during mine development and then continue during coal mining to ensure a continuing accessible coal reserve. The understanding of the geological parameters is essential to ensure successful high productivity coal mining.

Technical development in underground mining has concentrated principally on coal mining by two systems, longwall mining and room and pillar mining. Both methods have developed to meet the criteria of reductions in the work force 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 coal mining has developed over the past 50 yr. 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 by 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 are

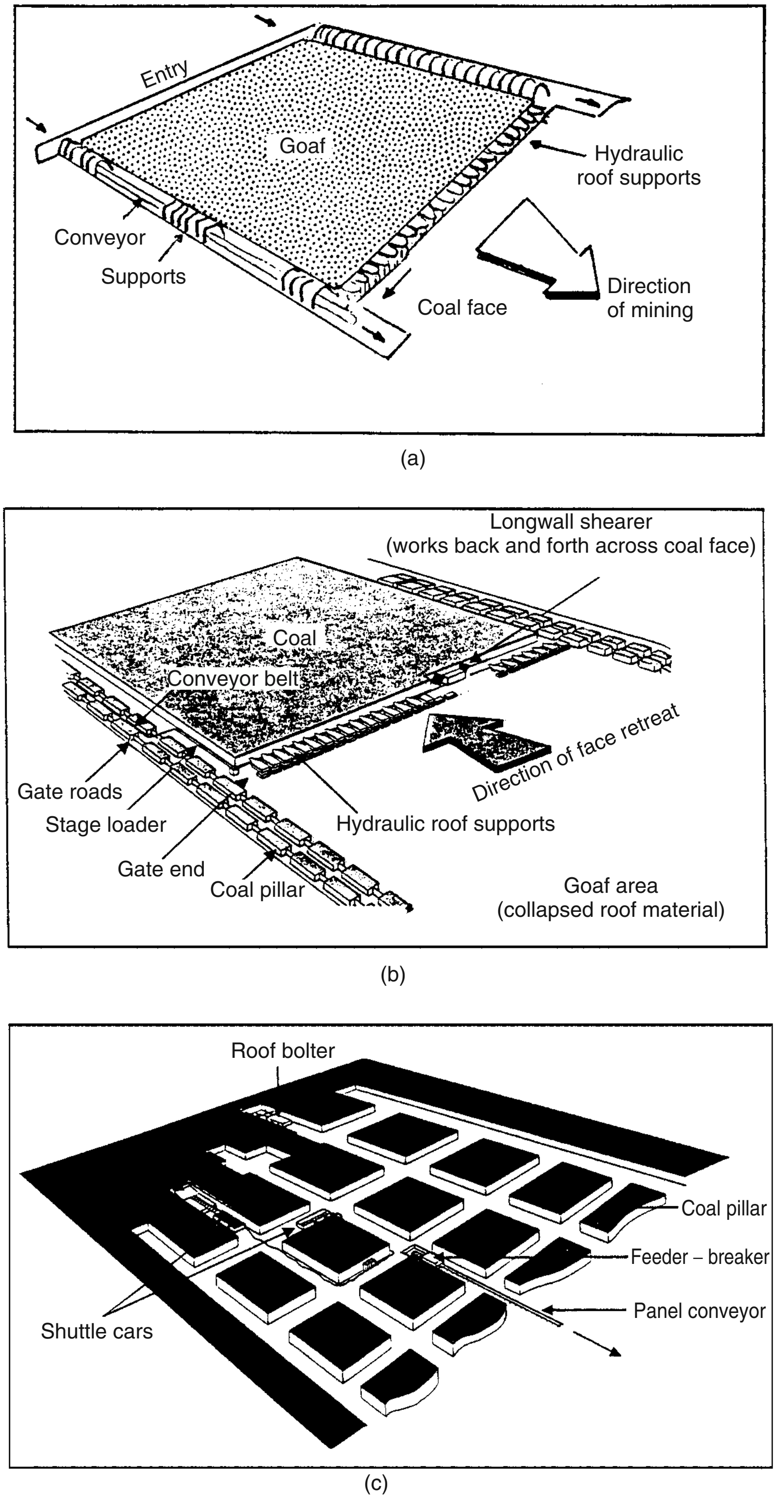



**Figure 10.2** Methods of mining in underground mines. (a) Longwall advance mining; (b) longwall retreat mining; (c) room and pillar mining. ((b) and (c) From Hunt and Bigby, 1999).

(a)

(b)

**Figure 10.3** Longwall mining equipment (a) rotary drum shearer, Huabei mine, PRC. (Photograph by Dargo Associates Ltd.) (b) Coal plough, Friedrich-Heinrich mine, Germany. (Reproduced by permission of DBT.)

attached to the front of the roof supports and run 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 capacities of the transportation, ventilation and power equipment that can be supplied and installed. In the United States, panel width is usually 120–293 m, in United Kingdom it is 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

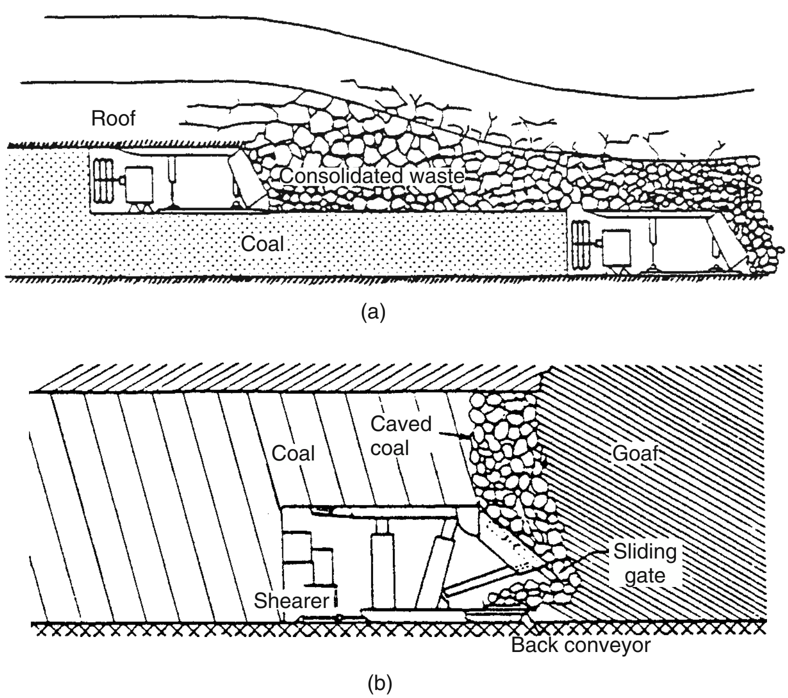



**Figure 10.4** Methods for longwall mining in thick seams. (a) Multiple slicing, using two longwall systems. (b) Single longwall system combined with sublevel caving. (From Ward (1984), by permission of Blackwell Scientific Publications.)

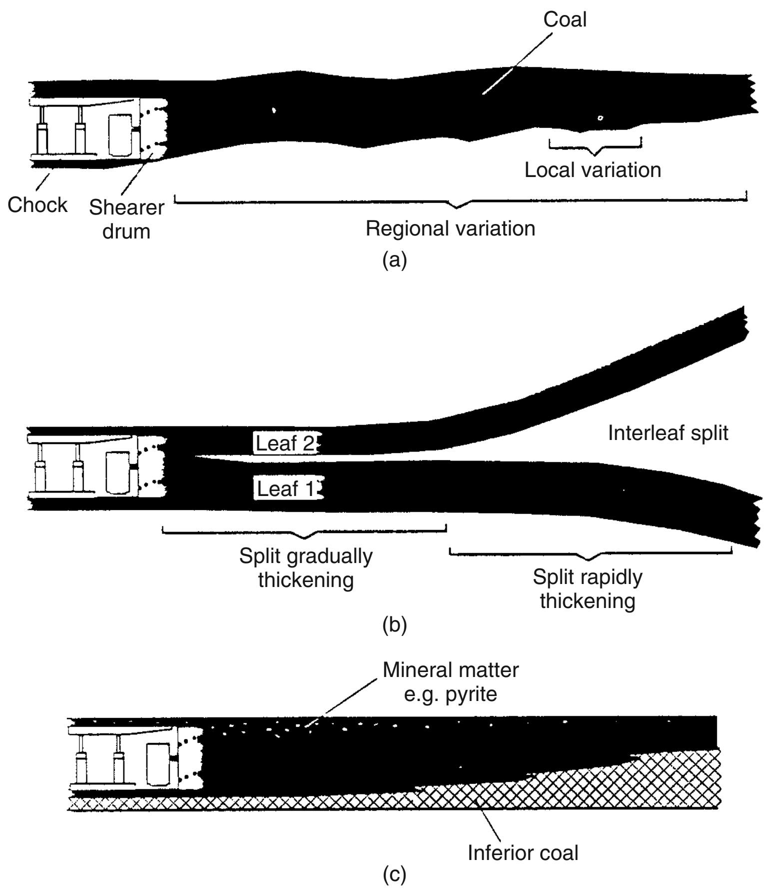



**Figure 10.5** Effects of changes in coal seam development on longwall mining. (a) Local thickness variation affects working of individual faces. (b) Coal seam splits affect quality of face product – rapid thickening may terminate mining. (c) Change in mineral content affects face product and increases wear on mining equipment. (From Fulton, Guion and Jones, 1995.)

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 (ROM) coal. Also the shearer can emit sparks if a quartzose sandstone is encountered, which 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 minimize the loss of longwall faces by identifying conditions in advance of the longwall face, thus reducing failure rates.

Longwall mining is used extensively in the United States and western Europe, and is now becoming established in Australia, South Africa, China, India and 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 either by 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 to a conveyor or shuttle car to be transferred to the main mine haulage system. In continuous mining, a single machine with

**Figure 10.6** The EIMCO Dash Three Continuous Miner in operation in Alabama, United States. (Photograph by courtesy of EIMCO, Bluefield, West Virginia, United States.)

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 coal mines. 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 as the depth of the coal increases, larger pillars are necessary 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 the prevailing geological conditions that facilitate rapid development in seam roadways with the most economic level 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, and this and the mechanical point anchor bolt 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, that is 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.

Mechanized room and pillar mining has long been established in the United States, Australia and South Africa, but it has rarely been used in Europe. In the past in United Kingdom, underground coal workings that extended offshore were worked by room and pillar method as longwall shearers had high levels of vibration. Room and pillar mining is also widespread in Indian and Chinese coal mines, with varying levels of mechanization. In northern China, new fully mechanized 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 systems will be determined by the type of rock strata and the loads acting on it, that is 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, in deeper mines strata control problems are not uncommon. The vertical component of stress is dependent on the depth of mining. In the United Kingdom, Germany and Poland, mining depths are greater than 600 m, whereas in Australia, South Africa and the United States they are less than 300 m (Hunt and Bigby, 1999). At greater depths mining becomes progressively more difficult and expensive. Larger areas of coal (pillars) have to be left between longwall panels or in room and pillar districts. There is also the increased possibility that rock bursts may occur.

Horizontal stress is a critical factor affecting roof stability in underground mines. Mark and Gadde (2010) state that the stress regimes encountered in underground coal mines 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 have created a World Stress Map, which can be used to indicate regional stress fields (Reinecker et al., 2005). Mark and Gadde (2010) have evaluated global trends in coal mine horizontal stress measurements, and have illustrated stress orientations in the United States, 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 United Kingdom and Europe, a major horizontal stress component exists, orientated northwest–southeast, whereas in Australia, horizontal stress orientations in the southern coalfields of New South Wales can vary from north–south in some areas to east–west in others, and that the dominant stress direction in Western Australia is east–west. In the eastern United States, the greatest horizontal stress is east-north-east, whereas in the western United States, horizontal stress directions have a wide variation in direction across the region. The Mark and Gadde research findings have indicated that the calculated depth gradient for the eastern United States was 1.6 times the vertical stress, in the Bowen Basin, Australia it was 1.4 times the vertical stress and for northern Europe (United Kingdom and Germany) 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 (United Kingdom), longwall panels aligned east–west were unsuccessful, but when changed to a north–south orientation 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 related directly 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 United Kingdom mine, a longwall panel was aligned 110° 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.

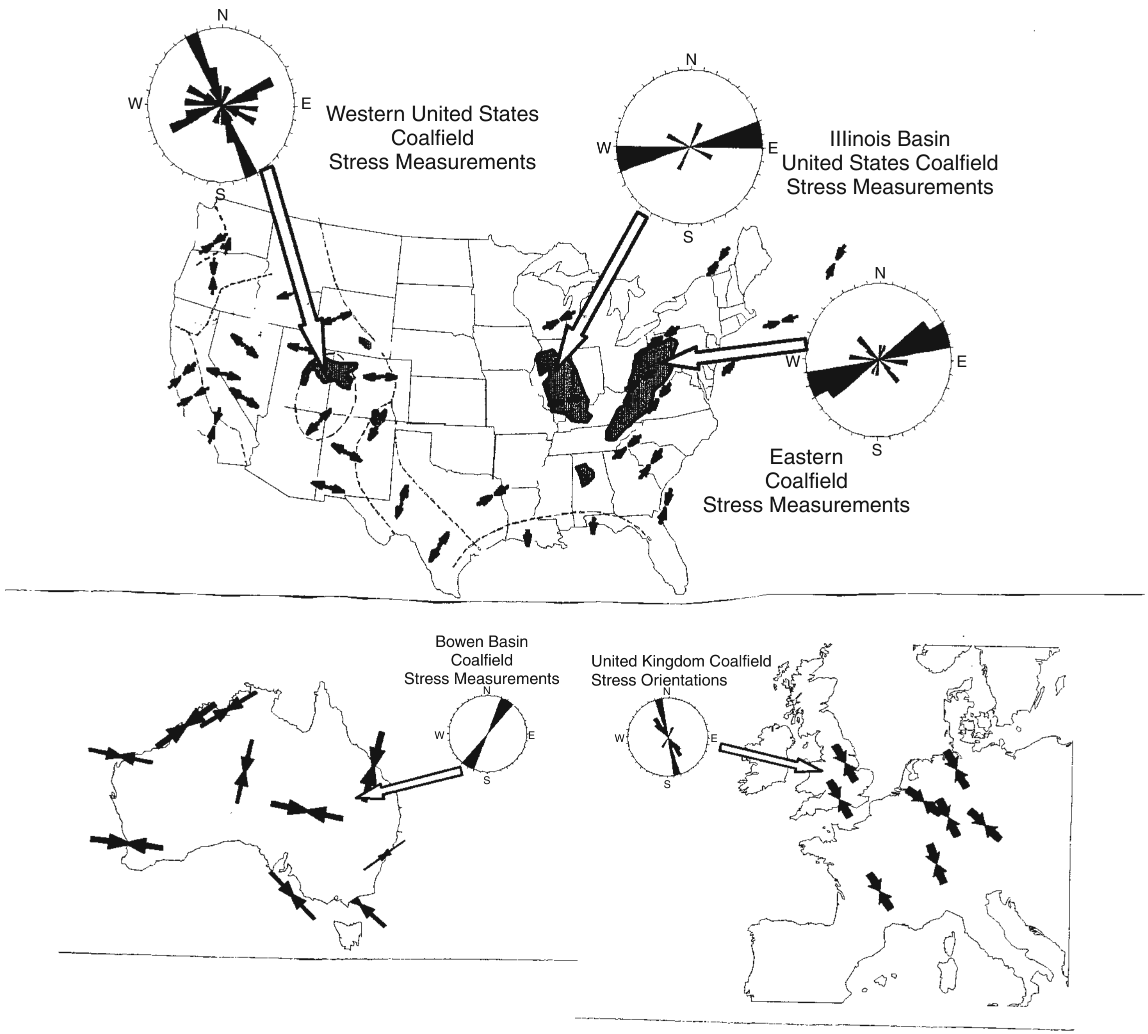



**Figure 10.7** World stress maps of the United States, Australia and northern Europe showing principal stress directions together with stress measurements taken in specific coalfield areas (Mark and Gadde, 2010).

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, and 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 recognized in a number of mines in the United States and South Africa, which has increased the importance of identifying the horizontal stress field as early as possible. Figure 10.9 shows an optimized longwall layout and working sequence for high horizontal stress conditions (Hunt and Bigby, 1999).

The stress field is also affected by multiseam 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 United States, room and pillar mining has become fully mechanized with the use of continuous miners and roof bolting machines.

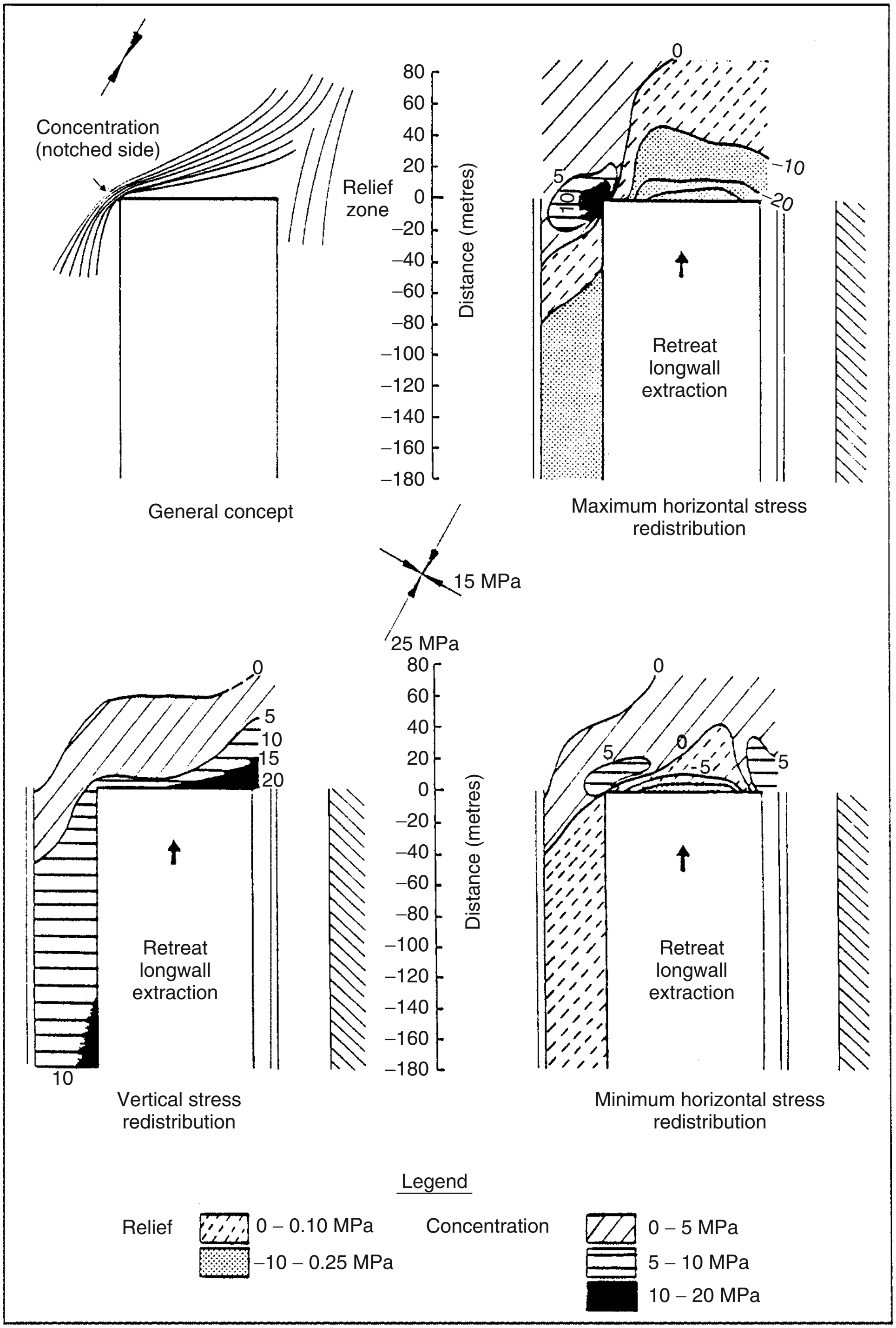



**Figure 10.8** Stress redistribution about a retreating longwall panel. (From Siddall and Gale, 1992.)

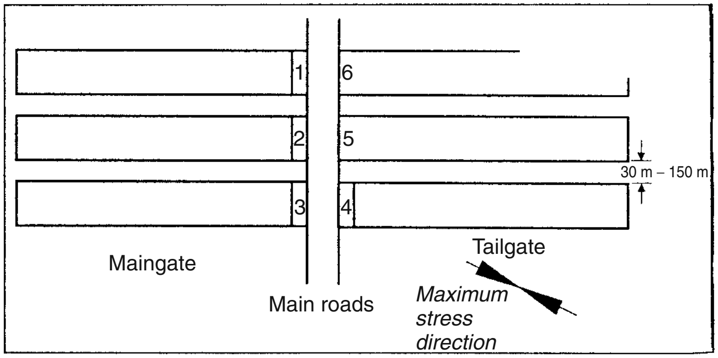



**Figure 10.9** Optimized longwall layout and working sequence for high horizontal stress conditions. (From Hunt and Bigby, 1999.)

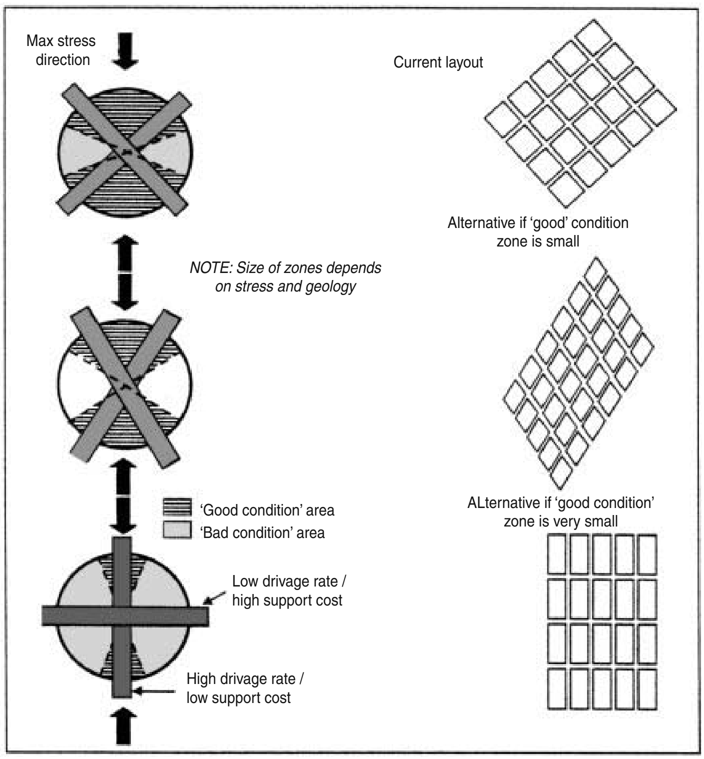



**Figure 10.10** Possible layouts to minimize horizontal stress effects for a room and pillar district. (From Hunt and Bigby, 1999.)

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 and 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, with the amount of expansion allowing the original stress

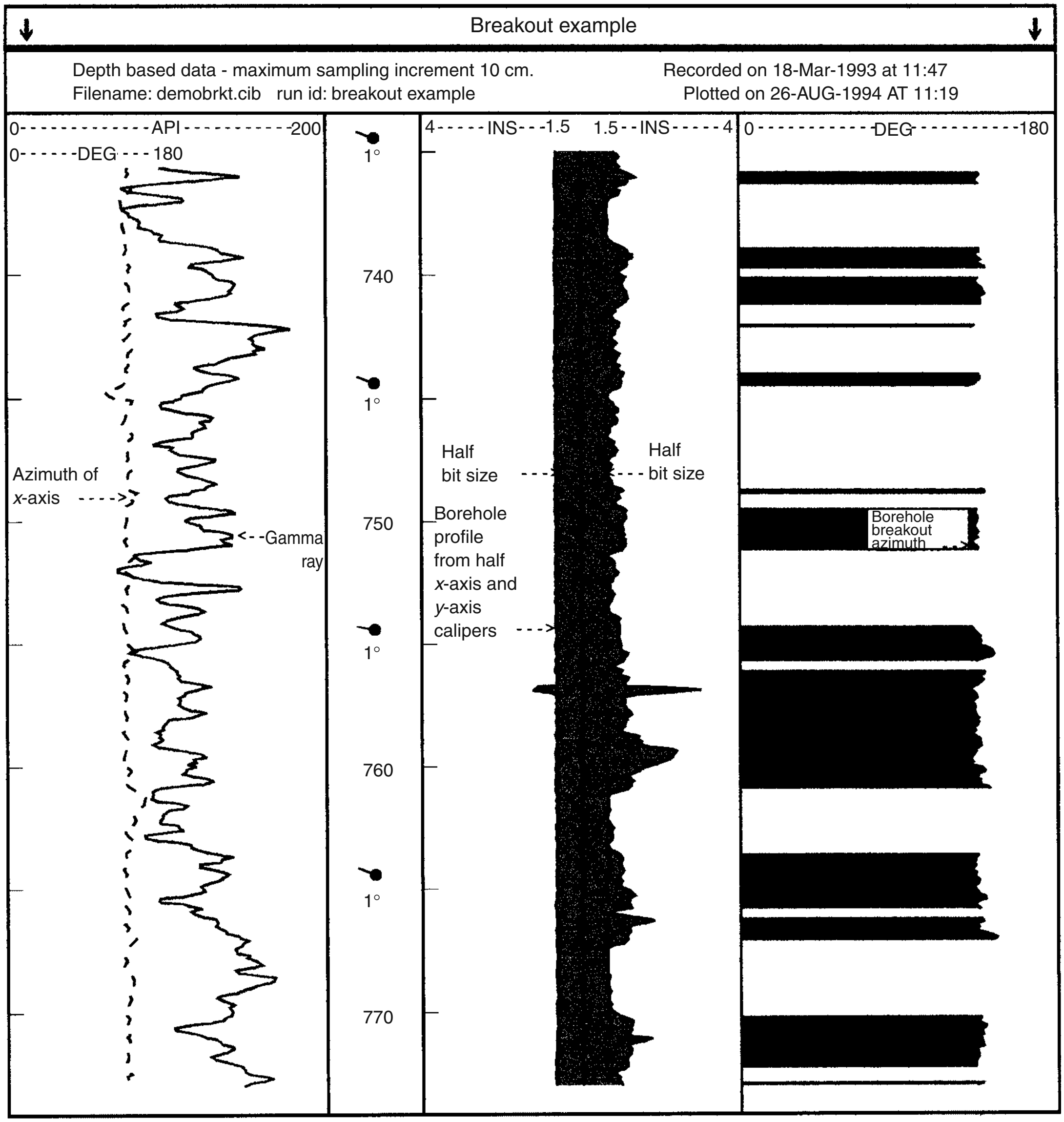



**Figure 10.11** Borehole breakout log showing minimum and maximum calipers. Breakout is identified by rock spalling or an increase in borehole diameter. (Reproduced by permission of Reeves Oilfield Services Ltd.)

field to be measured. Other methods include pressurizing 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 bolt holes, fixed equipment, etc.

The use of down-hole geophysics has led to the realization that horizontal stress regimes may be recognized, 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 calipers. In recent years, the use of acoustic scanning tools has produced high-resolution

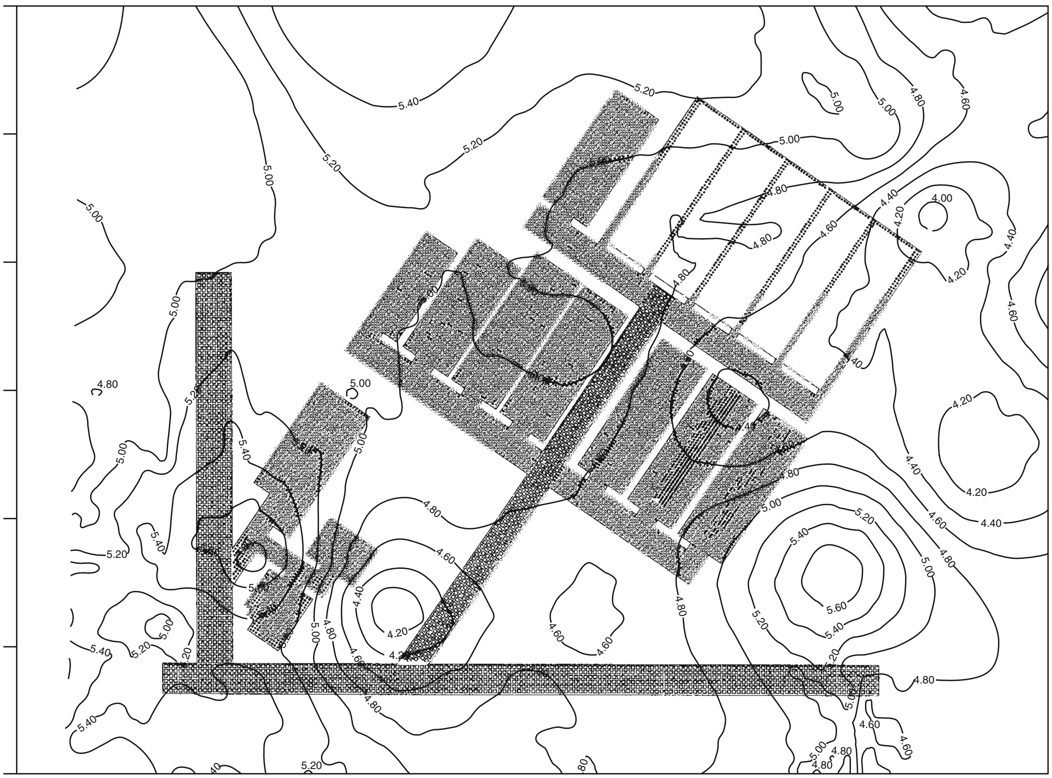



**Figure 10.12** Final layout design for an underground mine development, with superimposed contours. (Reproduced by permission of Society for Mining, Metallurgy and Exploration, Inc. (www.smenet.org).)

formation images in boreholes, and breakouts can be identified and plotted using this technique (see Chapter 8, and Figure 8.31). In the United Kingdom, an average breakout orientation of 54°/234° has been identified in conjunction with minimum stress orientation measurements from other techniques such as hydro-fracturing and overcoring (Brereton and Evans, 1987).

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, and the object is to create a panel design that can be constructed, saved, copied and modified as many times as required, which will be 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 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 digitized 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 digitized, 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 have 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, including strength properties for intact 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

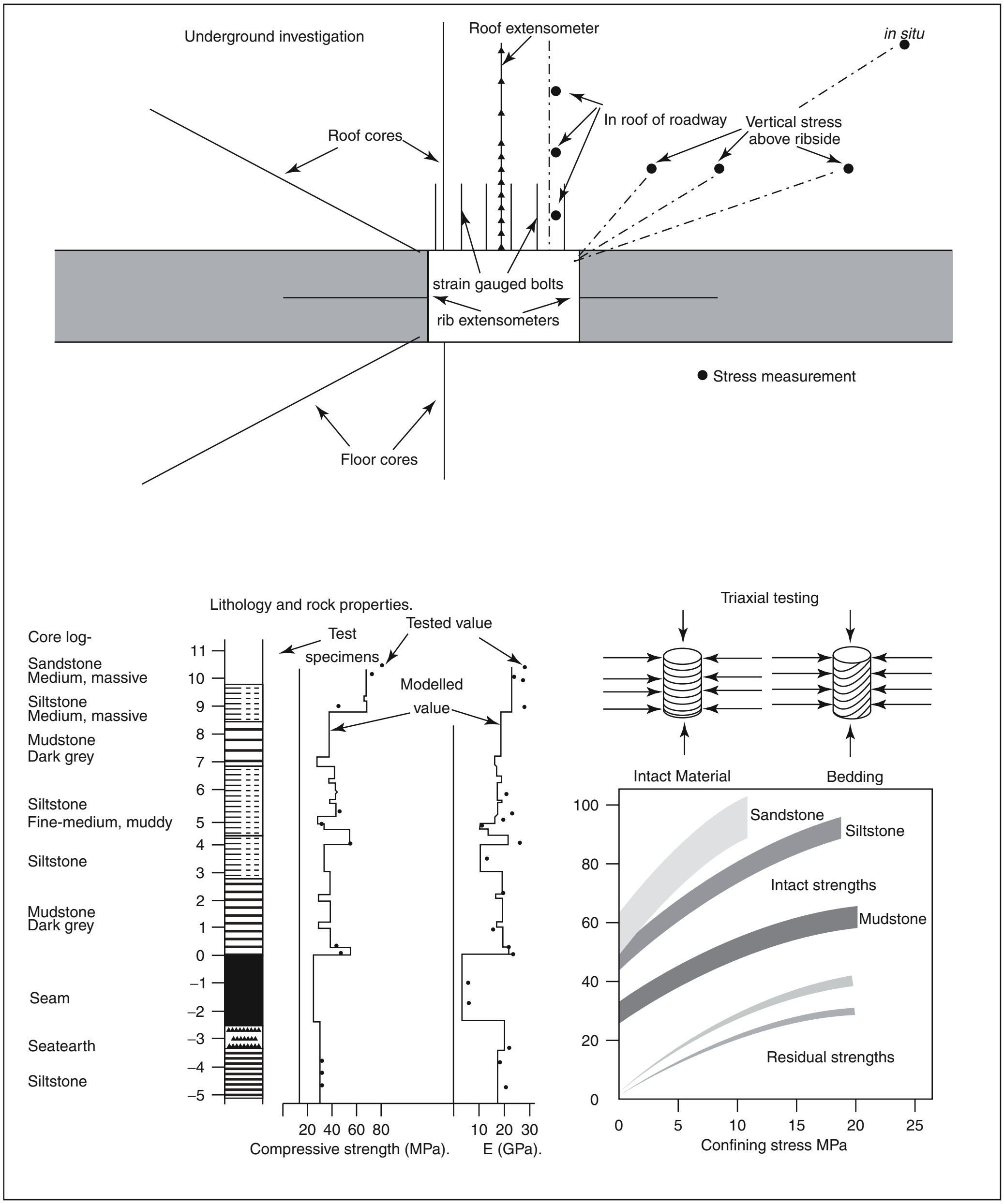



**Figure 10.13** The range of underground measurements and laboratory rock tests normally undertaken for model generation. (Reproduced from Garratt (1999).)

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 how the stresses around the roadway have changed from pre-mining levels as 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 their 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, as well as identify support requirements and optimum pillar size. This improves the accuracy of interpreting coal mine geotechnical properties, which will reduce the risk factor in underground coal mining 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 Strata and air temperature*

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 United Kingdom). In deep coal mines 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, so that 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.

### *10.2.3 Production*

A comparison of production between longwall and room and pillar operations shows that the highest face production is achieved in longwall mines, for example 1.0–2.5 Mt $yr^{-1}$, dependent on seam thickness, compared with 0.25–1.0 Mt $yr^{-1}$ for continuous miners. Typical continuous miner section production per shift for one continuous miner and two to 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 to four shuttle cars.

In the United States longwall production increased from 5000 t per employee-year in 1989 to over 10,000 t per employee-year in 1999 (Sabo, 2000), with a 10 h 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) averages 9000 t per employee-year compared with 6000 t per employee-year for non-longwall mines.

Where high production is essential to remain competitive, for example in the United States 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, that is outputs of +2 Mt $yr^{-1}$ 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 more simple than for longwall mining, and training and organization are 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 £30–40 million ($50–70 million) and a room and pillar continuous mining system is nearer £10 million ($15 million). For the investment in coal mining in developing countries, room and pillar mining is more flexible in changing geological conditions than longwall systems, although the latter is more productive.

## 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 and ground water influences, the direct effect of climate and the commitment to restore the land to meet environmental requirements.

Ward (1984) divides surface mining into two types.

1. 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.
2. 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 coal mine development in the world is 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 United States are also surface mining operations. Brown coal mines are virtually all surface mines, many of which are large-scale operations, as at Bełchatów in Poland and in the Latrobe Valley in Victoria, Australia.

### *10.3.1 Geological factors*

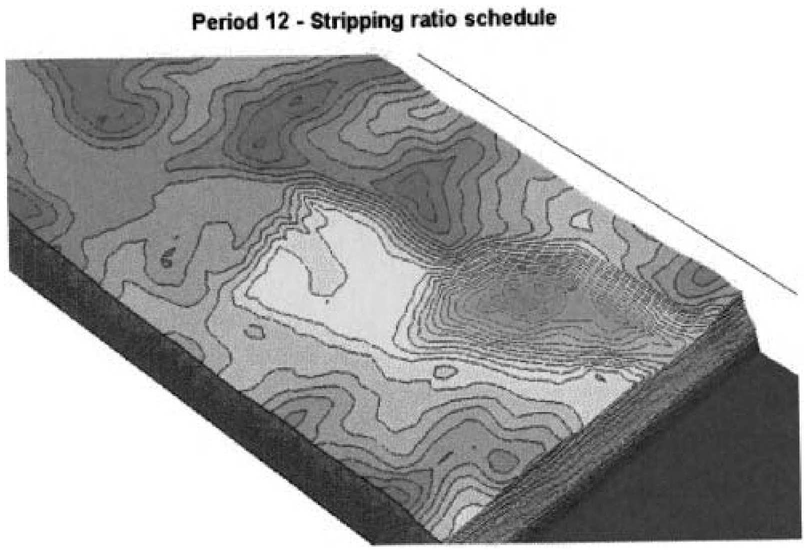
A surface mine will be designed by (i) assessing the basic geological data of the area, together with geographical and economic constraints. This will determine whether further investigation is warranted, or should cease. (ii) The assessment of reserves from more detailed geological data followed by 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.

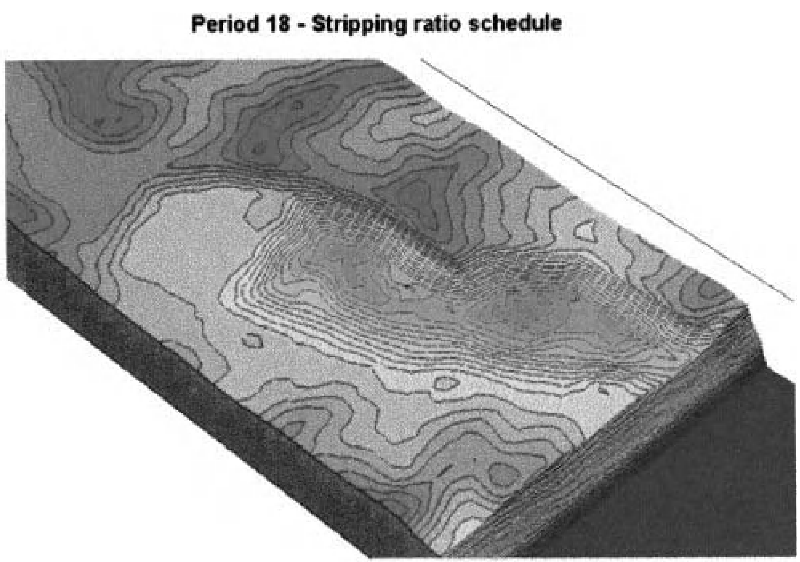
During the second stage, a planned drilling programme will identify the weathering and hardness of the overburden, changes in coal seam thickness, seam splitting, washouts, small-scale faulting and the nature of the non-coal interbeds between coal seams. The groundwater regime should be ascertained and all water levels known, 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.

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 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, is based on the type of equipment to be used. In opencast mining, the selection of equipment such as bucketwheel excavators, 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 three-dimensional 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, stripping ratio and depth cut-off limits, which are built into the model. Figure 10.14a and b shows three-dimensional contour maps of the stripping ratio schedule




(a)




(b)

**Figure 10.14** (a) Three-dimensional contour map showing stripping ratio schedule after 3 yr. (b) Three-dimensional contour map showing stripping ratio schedule after 4.5 yr. (Reproduced by permission of Datamine International.) This figure is reproduced in colour in the Plates section.

for a selected area to be mined after 3 yr and 4.5 yr respectively (Datamine International, personal communication, 2002). Such analysis allows schedules optimized for stripping ratio 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 has 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, 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 developing 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 optimization programmes. The ultimate pit configuration may already be defined by the mine owners or by physical constraints, but of great 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 maximize net product value (NPV), cashflow, stripping ratio and coal quality limits.

### 10.3.2 Mining equipment

In surface mining, the excavation of overburden is by dragline, electric or hydraulic shovels or bucketwheel excavators (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 coal mines, particularly in the United States. Draglines are used to remove overburden in both brown (lignite) and black (bituminous) coal surface mines, and 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 $m^3$ and larger buckets (Table 10.1). In the United States, a single dragline of this size can move $7 \times 10^6$ BCM $yr^{-1}$, and the largest draglines (85–122 $m^3$) can move over $20 \times 10^6$ BCM $yr^{-1}$, and can excavate down to coal seams 45 m below the surface (Pippenger, 1998). Such equipment is only useable in the largest mines; elsewhere dragline capacity may be lower (10–20 $m^3$) when the mining operation is smaller, for example in India.

**Table 10.1** Dragline capacities currently in use.

| Make/model | Boom length (m) | Bucket capacity ($m^3$) |
|---|---|---|
| CAT 8000 | 75–101 | 24–34 |
| CAT 8750 | 109–132 | 76–129 |
| P&H 9010C | 80–105 | 57 |
| P&H 9030C | 100–130 | 85–122 |

CAT = Caterpillar.
Source: adapted from manufacturers' websites.

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 10.1. Figure 10.15 shows a large dragline removing overburden and dumping adjacent to the excavation area. The average cost of a walking dragline with a 30$m^3$ bucket and 90 m boom is \$39.9 million, and with a 60 $m^3$ bucket and 104 m boom is around \$93 million.

Draglines mounted on crawlers are smaller capacity, the largest with a 69 m boom and 17 $m^3$ bucket is operating in New South Wales, Australia. A crawler dragline with 7.6 $m^3$ bucket and 61 m boom is \$5 million.

All of these are major capital items to equip a surface coal mine.

#### 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 $yr^{-1}$. Most of these operations utilize large loading equipment together with large capacity trucks; this style of operation is referred to as 'truck and shovel'. There are three types

**Figure 10.15** Dragline removing overburden in opencast mine, USA. courtesy of Bucyrus International Inc. Reproduced by permission of *World Coal*, Palladian Publications Ltd.

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, traditionally it has been 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 $43.6\,m^3$. All are larger than the typical wheel loader. The United States is currently the largest market for all three types of loader, with large electric shovels and large trucks dominating the larger coal mines.

The manufacture of such equipment is very specialized, and there are a number of major manufacturers of electric shovels, hydraulic excavators and wheel loaders. Table 10.2 shows the size and capacity of a selection of

**Table 10.2** Size and capacity of a selection of electric and hydraulic excavators, wheel loaders and trucks that are currently in use.

| Machine | Make/model | Bucket capacity ($m^3$) |
|---|---|---|
| Electric excavator (shovel) | CAT 7182 | 6.9–17.6 |
| | CAT 7495HD | 19.1–49.7 |
| | CAT 7495HF | 30.6–61.2 |
| | P&H 2300XPC | 19–36 |
| | P&H 4100C | 30.6–61.2 |
| | Liebherr R9350 | 18 |
| Hydraulic excavator (shovel) | CAT 6015 | 7 |
| | CAT 6040 | 22 |
| | CAT 6090 | 37–62 |
| | KOM PC850SE-8 | 4–4.5 |
| | KOM PC3000-6 | 12–20 |
| | KOM PC8000-6 | 28–48 |
| | Liebherr R9250 | 15 |
| | Liebherr R996B | 34 |
| | Liebherr R9800 | 42 |
| Wheel loaders | CAT 988H | 6.4–7.7 |
| | CAT 994H | 14–36 |
| | KOM WA150-5 | 1.5 |
| | KOM WA600-6 | 6.4 |
| | Liebherr L528 | 2–4 |
| Trucks | CAT 777F | 100 (t)* |
| | CAT 793F | 250 (t)* |
| | CAT 797F | 363 (t)* |
| | Liebherr T282 | 363 (t)* |
| | BeLAZ A3-7557 | 90 (t)* |
| | BeLAZ A3-75601 | 360 (t)* |

CAT = Caterpillar, KOM = Komatsu.
*Load capacity.
Source: adapted from manufacturers' websites.

electric and hydraulic excavators and wheel loaders that are currently in use, and Figure 10.16a, b and c shows examples of all three kinds of equipment.

Surface coal mines producing 10 Mt $yr^{-1}$ or more, will probably select the largest trucks and shovels, as is the case in United States and Australia. Smaller mines will require smaller units, as used in the United Kingdom 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 a 'backhoe' configuration, and 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, coal mining has relied on smaller machines, chiefly because surface mining has only recently begun to increase in scale, the older machines are electric, but 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. China, the United States and CIS utilize both large and small electric shovels and are the chief manufacturers.

In considering electric shovels and hydraulic excavators, haul or dump trucks are utilized for both overburden and coal removal. For overburden, large dump trucks are used in the larger mining operations to complement the larger size shovels. Dump truck sizes range from 35–100 t for smaller shovels and 100–363 t for the large shovels (Figure 10.17), most are rear dump models, but bottom dump trucks, usually 100–190 t, are also used. Articulated dump trucks are also in operation in some countries. Table 10.2 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 between \$4–18 million, for hydraulic excavators \$1.5–14.0 million, wheel loaders \$0.1–6.0 million and dump trucks \$1,2–5.7 million. In addition, ancillary equipment such as dozers, graders and backhoe excavators will also be required. Depending on the mine size, stripping ratio and production scheduling, the capitalization is a major item, but has the advantage of flexibility in that 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 7 yr, 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 (BWEs)*

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, which then load onto a main belt conveyor or into trams for transport out of the mine. The normal capacity of these machines is 420–2300 $m^3\ h^{-1}$ (1000–3000 $yd^3$) and they are particularly effective in excavating soft overburden in flat-lying strata. Figure 10.18a shows a BWE cutting overburden in a mine in Bosnia-Herzegovina and 10.18b 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 558 $m^3 h^{-1}$ machine has a capital cost of around \$1.6 million (not including ancillary equipment), and a

(a)


(b)

**Figure 10.16** (a) Electric shovel removing overburden, in central India. This type of shovel is still commonly in use, but has been superceded by larger capacity models in the larger mining operations in the United States and Australia. (Photograph by courtesy of Dargo Associates Ltd.) (b) Hydraulic shovel loading overburden in Spain. Modern shovel capacities are 20–30$m^3$. (Photograph by courtesy of Liebherr Mining Equipment Co., Ltd.)


(c)

**Figure 10.16** (c) Wheel loader removing overburden. (Photograph by courtesy of *World Coal*, Palladian Publications.)

$1550\,m^3\,h^{-1}$ machine a cost of \$5.8 million. 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 have, however, a long mine life.

### *10.3.3 Surface mining methods*

The method of mining and the equipment used in surface mining is 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). Surface mines are typically up to 50 m wide and the deepest level of excavation up to 80 m. Working faces can have an angle of $50^\circ$–$90^\circ$, and mine batters and spoil tips have angles that range from $30^\circ$–$45^\circ$, the lower angles being most common. A brief outline is 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, while 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 (Figure 10.19). 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 stripping ratio 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, the topsoil is then replaced and prepared for appropriate land use. Such large-scale strip mines are operating in the Powder River Basin, Wyoming, United States, for example Jacobs Ranch mine produces $13\,Mt\,yr^{-1}$ of subbituminous, low sulfur coal for electricity generation (Hartman, 1992).

**Figure 10.17** Large dump truck (280 t) being loaded at Fording coal mine, Canada. (Photograph by courtesy of Komatsu Mining Systems.)

*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.

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, which can be on a very large scale, as at Anjaliang, China (Figure 10.20a, or of more modest size, as seen at Pljevlja mine in Montenegro (Figure 10.20b). 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, that is 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 blasted, if necessary, and shovelled directly into waiting trucks by small shovels or wheel loaders. Figure 10.21 illustrates such an operation

(a)

(b)

**Figure 10.18** (a) Bucketwheel excavator (BWE) cutting overburden in Gacko lignite Mine, Bosnia-Herzegovina. (Photograph by courtesy of Dargo Associates Ltd.) (b) Large BWE and conveyor system in operation in Berezovsky opencast mine, Kansk-Achinsk Basin, Russian Federation. (Photograph by courtesy of Dargo Associates Ltd.) Figure 10.18b is reproduced in colour in the Plates section.

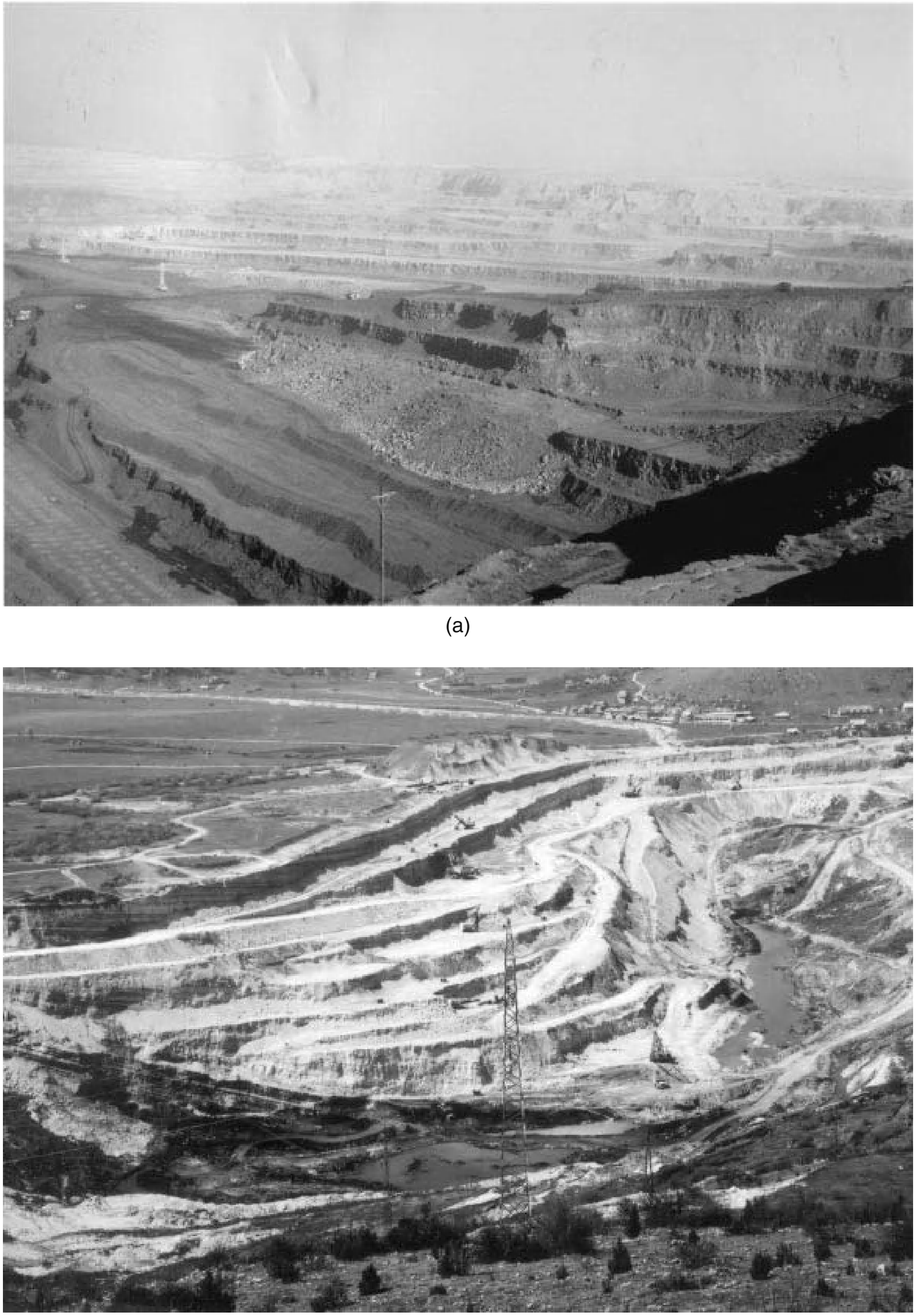
**Figure 10.19** Coal being loaded into trucks by hydraulic excavators, Western Coalfields, India. (Photograph by courtesy of Dargo Associates Ltd.)

at Shotton mine in Northumberland, United Kingdom. Brown coal will either be cut by BWEs and conveyored, 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 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 in China, El Cerrejon Norte in Colombia, Guasare in Venezuela and Grootegeluk in South Africa, as well as numerous similar operations in Australia and the United States. Smaller black coal opencast mines can be found in East Kalimantan, Indonesia, India and the United Kingdom. Brown coal mines dominate central and eastern Europe and produce 50% of the world's total, there are also significant producers in the Ekibastuz coalfield, CIS, in the Latrobe Valley, Victoria, Australia, at Hambach, Germany, at Bełchatów, Poland and in the Big Brown mine in Texas, United States.

Contour mining is carried out in rugged topography, characterized by hill, 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 United States, 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 stripping ratio, or being in an area of the mine that had effectively sterilized



(a)

(b)

**Figure 10.20** (a) Large scale benched mining operation, Anjaliang opencast mine, Peoples Republic of China (photo by courtesy of Dargo Associates Ltd). (b) Smaller scale benched mining operation, Pljevlja brown coal mine, Montenegro (photo by courtesy of Dargo Associates Ltd.)

**Figure 10.21** Truck and shovel operation, Shotton opencast mine, Northumberland, United Kingdom. (Photograph by courtesy of M. C. Coultas.)

**Figure 10.22** Wirtgen surface strip miner in operation in Gacko mine, Bosnia-Herzegovina. (Photograph by courtesy of Dargo Associates Ltd.)

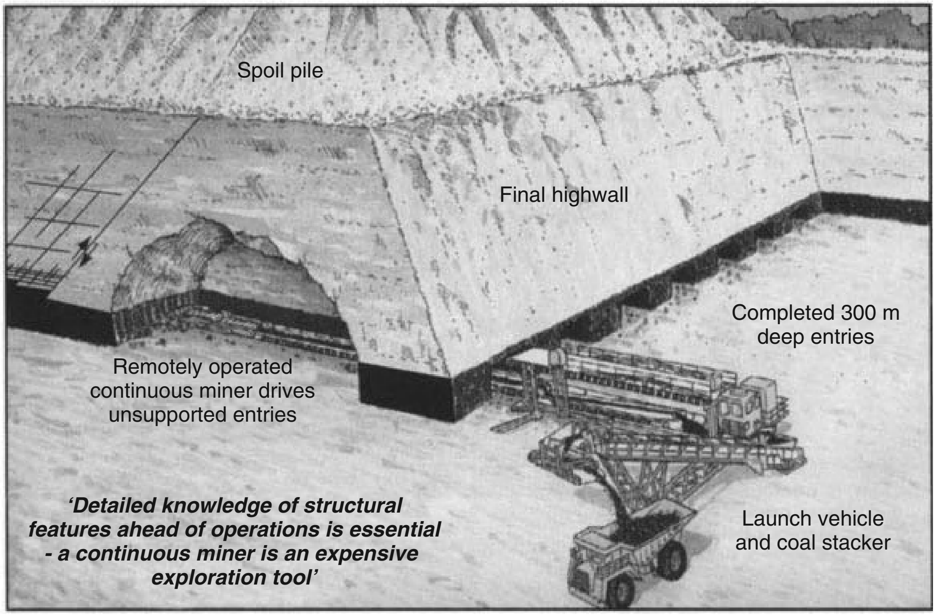



**Figure 10.23** Schematic image of a highwall mining operation. From Shen and Fama (2001) with permission.

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. 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 Australia, the continuous highwall miners have reached 500 m penetration and 124,000 t per month , the auger system reaching 200 m penetration and 60,000 t per month (Shen and Fama, 2001). In the United States, auger systems have produced over 16 Mt of coal from the Appalachian coalfields from coal seams ranging from 0.6–4.9 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 United States in the 1970s and in Australia in the 1980s. It has now been introduced in South Africa and the United Kingdom, and is a means of accessing coal reserves in opencast mines hitherto considered unmineable. Figure 10.24 shows auger mining in the United Kingdom, where the auger holes have a diameter of 0.45 m.

**Figure 10.24** Auger mining in Bottom Busty seam, Shotton opencast mine, Northumberland, United Kingdom. Auger holes have 0.45 m diameter. (Photograph by courtesy of M. C. Coultas.)

**Figure 10.25** Multiple rail system in Ekibastuz mine, Kazakhstan. (Photograph by courtesy of Dargo Associates Ltd.)

### *10.3.4 Production*

A major part of world coal production comes from surface coal mines; Gilewicz (1999) quotes $1.3 \times 10^9$ t produced from the world's major surface mines, which includes all significant brown coal production. Surface black coal mines produce on average 1.0–15.0 Mt $yr^{-1}$, whereas lignite mines can be much larger, producing up to 40 Mt $yr^{-1}$. In surface mining, the capital cost savings when compared with underground mining are 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 coal mines 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 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.

# 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, or 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 contributary source rock. Although the bulk of coal utilization 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, carbon dioxide, carbon monoxide, nitrogen 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 coal mining first began. In underground workings, methane is released from coal exposed at the coal face, plus the broken coal being transported through the mine. Methane is a flammable gas and is explosive between a lower limit of ca. 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 methane released into the workings, so that it is essential that an adequate ventilation system is in operation. A danger is that of methane collecting in roof pockets and in the upper parts of 'manholes' or cuts in the roadway sidewalls where the rock sequence may still be exposed, as well as in goaf (collapsed rock sections) 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 coal miners to measure the concentration of methane 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 methane content has been 1.25% for the use of electrical power. The use of locomotives and shotfiring, that is using explosives underground, must be discontinued if the methane exceeds this limit. At 2% methane, labour must be withdrawn from the workings, until the methane content is diluted to within the statutory limit. Other coal mining countries used the same method of detection as the United Kingdom until

*Coal Geology*, Second Edition. Larry Thomas.

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 United Kingdom, the current statutory limits are 0.25% of methane in air for air entering the working area and 0.5% for methane 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 has been increased to 1.5% in certain longwall installations. In Australia, intake airways are kept to below 0.25% methane, 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% methane. Continuous mining equipment may be required by the Inspectorate to be equipped with automatic methane 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% methane. In New Zealand, the limit for methane in the general body of the air in a coal mine is 1.25%. An Inspector can call for a ventilation survey of the mine if this figure is exceeded. In the United States, electrical shut downs 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 carbon dioxide, due to the fact that coals rich in pyrite absorb more oxygen when moist, and this absorbed oxygen produces not only water by combination with hydrogen, but also carbon dioxide by combination with carbon. Carbon dioxide, 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 methane explosions and underground fires. The gas is combustible and poisonous.

Only a small proportion of the nitrogen found in coal gases has its origin in the nitrogen present in the coal material; the bulk of the nitrogen originates from the surrounding air.

Free hydrogen occurs in small amounts associated with methane, but is not usually found in any great amounts.

Ethane is more prominent in gases derived from oxidized 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 methane 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. Methane is usually referred to as coal-bed methane (CBM) in most literature, however, in Australia CBM is known as coal seam gas (CSG).

### *11.2.1 Coal-bed methane*

#### *11.2.1.1 Generation of coal-bed methane*

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 ($CH_4$), carbon dioxide ($CO_2$), nitrogen ($N_2$) and water ($H_2O$). Coal-bed methane is generated in two ways: (i) during the early stages of coalification, at temperatures below 50°C, biogenic methane is formed by decomposition of the organic material and where biological activity induces reducing conditions, which remove oxygen and sulfate. Where subsidence and burial are rapid, biogenic CBM may be trapped in shallow gas reservoirs (Rightmire, 1984). (ii) Coal-bed methane is generated by means of catagenesis, the process by which organic material is altered as a result of the effect of increasing temperature. Coal-bed methane generated at temperatures in excess of 50°C will be due to this process and is referred to as thermogenic methane. The relative volumes of CBM generated by biogenic and thermogenic mechanisms are shown in Figure 11.1.

During coalification, more than twice as much $CO_2$ as $CH_4$ is generated up to the boundary of high volatile bituminous and medium volatile bituminous coals. Coal-bed methane volumes generated increase rapidly above this point, with the CBM generation peak occurring at about 150°C, or at the boundary of medium volatile bituminous and low volatile bituminous coals (Figure 11.1). The two gases associated with CBM are $CO_2$ and $N_2$. The latter is found only as a minor constituent of thermogenic gas as it migrates readily from the system due to its small molecular

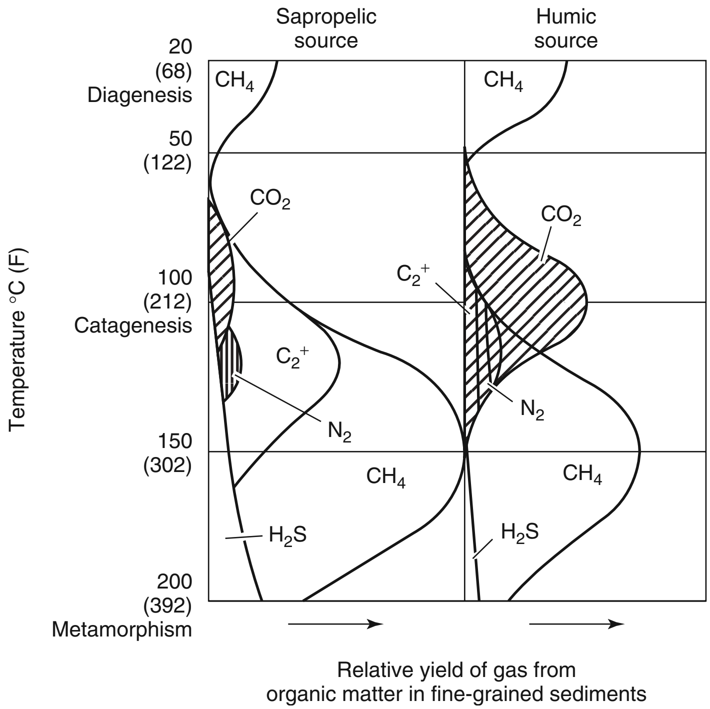



**Figure 11.1** Generation of gases with depth, $C_2^+$ represents hydrocarbons heavier than $CH_4$ in gas phase. $N_2$ is generated initially as $NH_3$. (From Rightmire (1984), based on Hunt (1979).)

size. $CO_2$ is a principal constituent of early thermogenic gas (Figure 11.1), but is only a relatively minor 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 $CO_2$ and $N_2$, and trace amounts of higher hydrocarbons such as ethane, propane, etc., form the remainder.

### *11.2.1.2 Retention of coal-bed methane*

Coal-bed methane is retained in coals in three ways: (i) as a free gas within the pore space or fractures in the coal; (ii) as adsorbed molecules on the organic surface of the coal; and (iii) 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). 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 $CH_4$, as is understanding 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 utilize pore surfaces 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 content, studies have shown that

**Table 11.1** Gross open-pore distribution in coals.

| Rank | C (% daf) | Porosity distribution (%) <12Å | 12–300Å | >300Å |
|---|---|---|---|---|
| Anthracite | 90.8 | 75.0 | 13.1 | 11.9 |
| Low volatile bituminous | 89.5 | 73.0 | Nil | 27.0 |
| Medium volatile bituminous | 88.3 | 61.9 | Nil | 38.1 |
| High volatile A bituminous | 83.8 | 48.5 | Nil | 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 | Nil | 59.1 |
| Lignite | 63.3 | 12.3 | Nil | 87.7 |

Source: based on Rightmire (1984) and Gan *et al*. (1972).

coals with carbon contents of <76% and >83% generally have surface areas <1 $m^2/g$, whereas coals within that range have areas >10 $m^2/g$. An exception to this is anthracite with >92% carbon, which also has high areas of 5–8 $m^2/g$ (Gan, Nandi and Walker, 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, pores are 20–200Å diameter, and inertinite is the most porous, with pore diameters of 50–500Å (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 $CH_4$ 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. Gas content and reservoir pressure are properties of the coal and vary widely, and coals may have different Langmuir parameters despite having other similar coal properties (Fenniak, 2004).

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 large-scale commercial enterprises. Huy *et al*. (2010) studied permeability in relation to fracture width and 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, such that 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 in order to determine the relationship between cleat spacing, cleat height and coal banding texture for coals of different rank were undertaken by Dawson and Esterle (2010; see section 2.3). 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 but increases in anthracite coal. Fracture permeability in most coals found in the United States lies in the range of 0.1–50 millidarcys (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

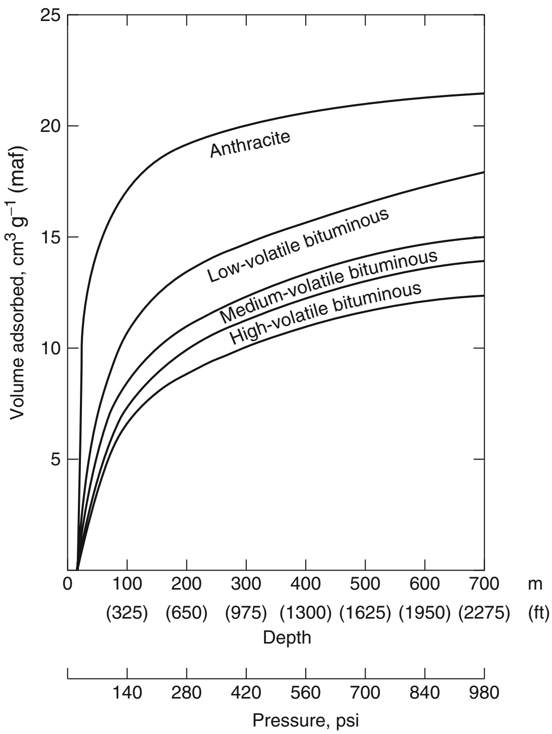



**Figure 11.2** Adsorptive capacity of coal as a function of rank and depth compiled by Dargo Associates Ltd from various sources. (From Rightmire (1984), based on Kim (1977).)

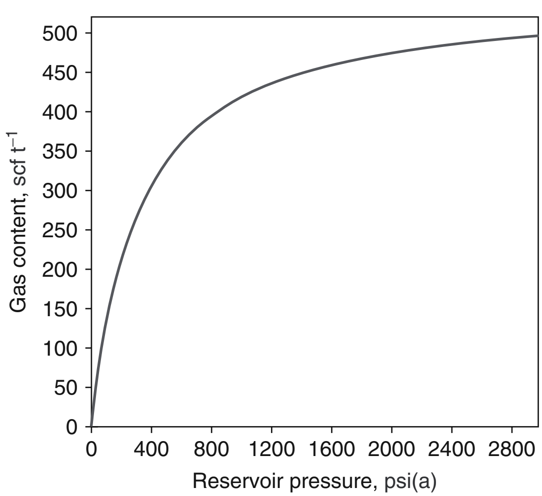



**Figure 11.3** Typical coal-bed methane isotherm characteristic of San Juan Basin coal. (From mattieu.fenniak.net (2004).)

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, United States. Here a domain of east-northeast striking face cleats in the south in Cretaceous Mesaverde Group coals is replaced in the northern part of the basin by west-northwest 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 22 myr after deposition, whereas in the north $R_o$ 0.5% was not reached until 31 myr 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 Paleogene–Neogene coals in the same area.

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 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 optimize 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 Production of coal-bed methane*

The production of CBM from underground sources is either by draining old and current mine workings (CMM) or by production from wells sunk into virgin or unmined coal seams (CBM) (Figure 11.5).

#### ***Coal mine (active and abandoned) methane***

*Active coal mine methane.* Methane ($CH_4$) is released as a direct result of the process of coal mining, in many mines the preferred mining method is by longwall extraction and this, as with other underground mining techniques, releases $CH_4$ previously trapped within the coal seam into the air supply of the mine, thus creating a safety hazard. In addition, $CH_4$ emissions arise from the collapse of the

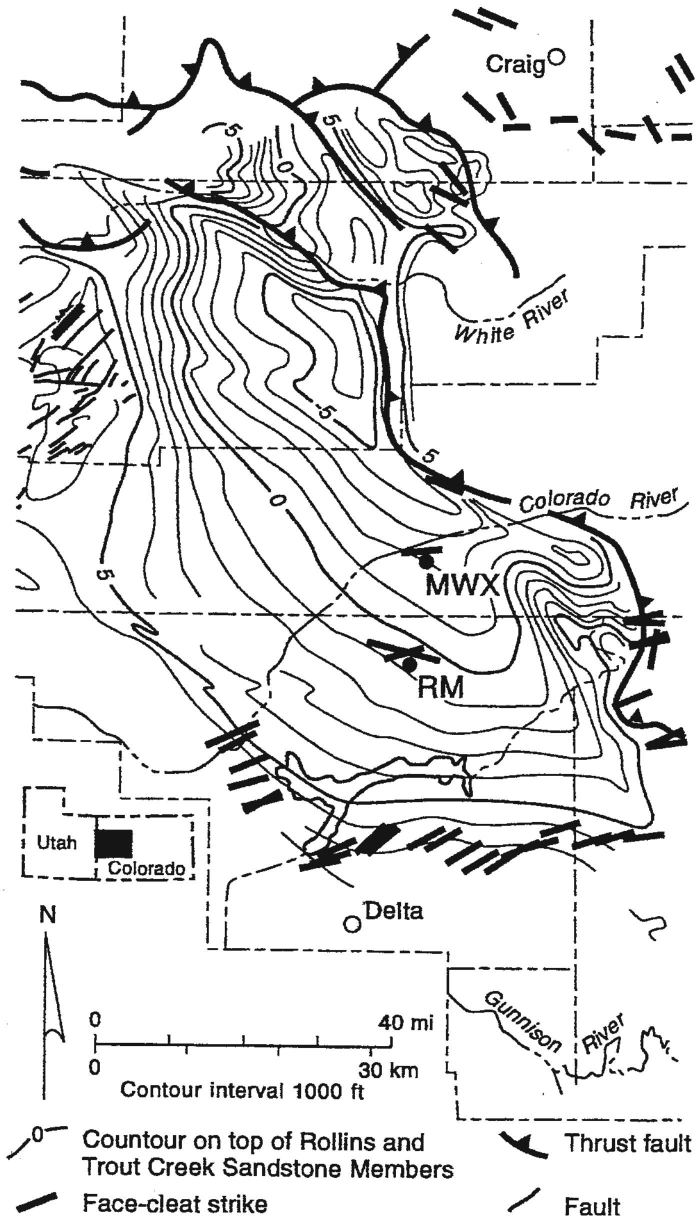



**Figure 11.4** Map of Cleat System in the Piceance Basin, Colorado, USA (Laubach *et al.* 1998). (Reproduced with permission, Elsevier Publications.)

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 $CH_4$, 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 $CH_4$ content and this puts pressure on mine ventilation systems, which have to be increased as worldwide safety standards require that mines cease operation when methane-in-air levels exceed a predetermined percentage. Lost production due to excessive methane-in-air content can have serious economic implications. When faced with this phenomenon, mines should

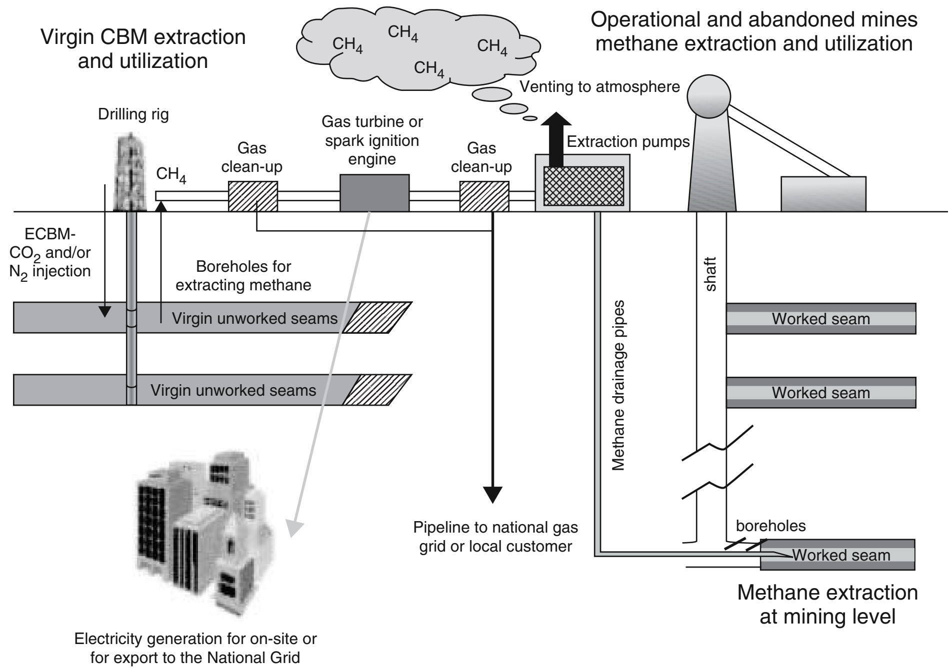



**Figure 11.5** Coal-bed methane and current mine methane options for methane ($CH_4$) extraction and utilization. (From DTI, 2001.)

recover and use the $CH_4$. In United States, those mines with high gas contents vent up to 75% of all gas liberated. Worlwide, mines vent 1.23 Tcf (35 Bcm) to the atmosphere, and use around 0.07 Tcf (2 Bcm) (Schultz 1997); see Appendix 5 for all unit conversion values. Depending on the drainage method selected, a mine can remove between 20 and 70% of $CH_4$ 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 $CH_4$ out of the working areas of the mine into shafts leading to the surface, this technique is known as ventilation air methane (VAM). The VAM is released through ventilation shafts and can then be destroyed or captured for utilization rather than allowing it to be released directly into the atmosphere. Ventilation air methane has the lowest concentration of $CH_4$ 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 premine), gob wells (vertical gob), long-hole horizontal boreholes and horizontal and cross-measure boreholes (Figure 11.6). 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 $CH_4$. 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 $CH_4$–air mixture, but as gob gas is exposed to lower volumes of air than VAM, it displays much higher $CH_4$ concentration levels, typically between 35 and 75% (WCA, 2011). Horizontal and cross-measure boreholes are drilled within the mine to drain $CH_4$ from seams in production and from surrounding gob areas. Again, the holes in-seam produce pure $CH_4$ and the holes in the gob areas produce a $CH_4$–air mixture. Significant advances in directional drilling systems have been made in recent years that have led to a larger range of $CH_4$ drainage options in gassy mining operations. Long, directionally steered in-seam boreholes can effectively reduce *in situ* $CH_4$ contents of large coal volumes in advance of mining in low- to high-permeability coals, as well as to drain faults and fissures containing free $CH_4$. As well as the coal currently mined, coals both above and below this seam can be drained, and gob gas can be drained from mined areas that lie above future longwall panels in a lower seam.

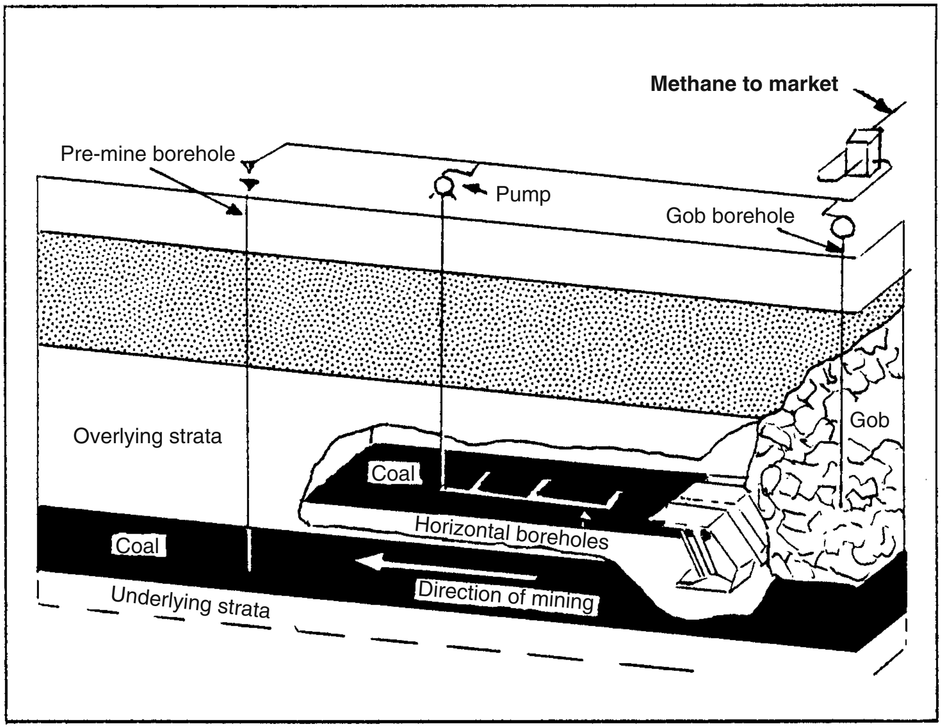



**Figure 11.6** Methane extraction from active mine workings. (Adapted from Schultz, 1997.)

*Abandoned mine methane.* Abandoned mine methane (AMM) can be recovered from previously worked but now disused underground coal mines. Such mines are either sealed, that is 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 $CH_4$ 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, and 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 2 yr post-mine closure. The United Kingdom, United States and Germany have been leaders in AMM projects, but there is huge potential for AMM in China and other countries with coalfields areas now no longer worked.

*Production of coal-mine and abandoned-mine methane.* In Australia, 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 11.7 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 11.8 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. BHP Billiton Illawara Coal has been capturing and utilizing $CH_4$ gas drained from coal seams in this way. Their WestVAMP project uses the dilute $CH_4$ in the ventilation air to generate electricity.

At the present time, China has become a world leader in CMM recovery and use. In China, 95% of the coal mined comes from underground workings. In 2007, China had an estimated 40 CMM projects dealing with pre-mining $CH_4$ drainage from within mines or from surface, or

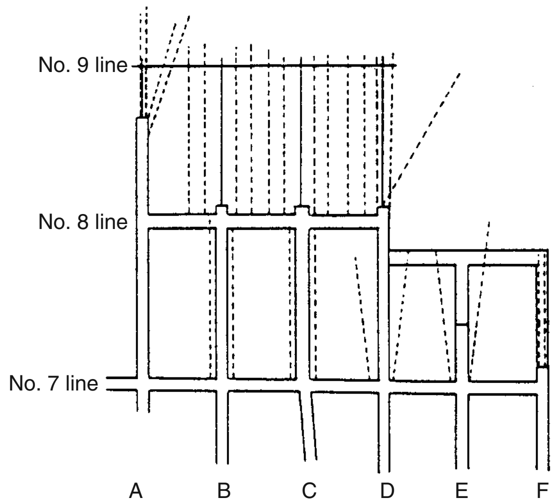



**Figure 11.7** Pattern of drainage holes ahead of a panel front, Metropolitan Colliery, Sydney Basin, Australia. (From Hargreaves and Lunarzewski, 1985.)

post-mining $CH_4$ drainage by degassing worked out goaf areas. By 2002 there were 193 mines with $CH_4$ drainage systems, draining ca. 1.1 Bcm of gas of which 0.6 Bcm was used (Vrolijk *et al.*, 2005), this has now increased to 4.7 Bcm in 2007, of this, 44% was from Shanxi Province. The concentration of $CH_4$ in many Chinese coal mines is low, usually a concentration of 10–35%, with a maximum concentration of 60% and 70% of the recovered CMM having a concentration of less than 30% (IEA, 2009). The principal type of CMM project in China is for power generation, in 2006, 3.0 Bcm of $CH_4$ was drained and 1.2 Bcm was used to generate 460 MW (IEA, 2009). Although china has an enormous potential for CMM recovery, most Chinese coal mines have drainage systems that produce low quality CMM and, in particular, have the problem of the transport of gas in or near the explosive range of $CH_4$ close to underground workings. Improvements in mine degasification will help to avoid potential risks. As China gains experience with VAM systems and gas recovery at coal mines, utilization of CMM will increase. In many parts of China, coal mines are located in isolated rural areas where the transportation of $CH_4$ 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 China in the future.

In Russia CMM is concentrated in the Kuzbass, Pechora and Donbass regions. In 2005 CMM from underground mines totalled 1.8 Bcm (UNFCCC, 2009). The Kuzbass region accounts for 78% of CMM with 47 active underground mines, and the Pechora Basin contributes 12%. In recent years, the rate of $CH_4$ recovery from CMM drainage in active mines has been 25–30% on average (CMM Country Profiles, 2010), and abandoned coal mines are monitored for $CH_4$ concentrations. To improve

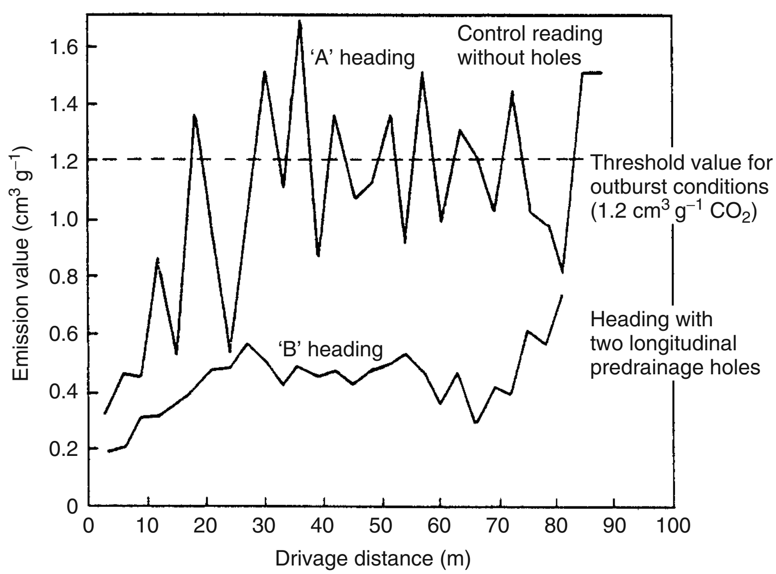



**Figure 11.8** Reduction of gassiness resulting from pre-drainage. (From Hargreaves and Lunarzewski, 1985.)

this general situation a number of projects have been set up to study the utilization of CMM from both active and abandoned mines.

In the adjoining Ukraine, 95% of the $CH_4$ produced from underground coal mines is vented directly to the atmosphere. Projects designed to utilize CMM are ongoing at the Bazhanov and South Donbass #3 coal mines. Current-mine methane emissions reductions from degasification of these two Donbass mines is anticipated to be ca. 2.5 Bcf (0.7 Bcm) per year.

In Poland, coal-mine emissions in 2003 were 16.5 Tcf (0.47 Tcm). There are now a number of projects underway to utilize CMM for power generation, boiler fuel and other industrial use.

In Germany, mine gas has been successfully produced in the Ruhr area for decades. With the successive closure of underground coal mines, gas production will decrease, to be replaced by AMM and then CBM from unmined coal seams.

In the Czech Republic, 3.5 Tcf (0.1 Tcm) of CMM is utilized each year for domestic purposes and for sale to European markets. The OKR mining basin accounts for 99.8% of CMM from Czech mines. In 1998, 4.2 Tcf (0.12 Tcm) of $CH_4$ was drained from the mines, and in addition AMM is produced from 10 abandoned shafts and four wells totalling 880 Mcf (25 Mcm) per year (CMM Country Profiles, 2010).

In Mongolia, CMM emissions totalled 123 Mcf (3.5 Mcm) in 2005. The Nalaikh underground mine was closed due to gas explosions but is now under review as a possible CMM supplier.

In the Karaganda Basin in Kazakhstan, the coal mines are extremely gassy. The average $CH_4$ content of coal seams is 10–12 $m^3\,t^{-1}$ at depths of 100–200 m, 16–20 $m^3\,t^{-1}$ at depths of 350 m and 20–25 $m^3\,t^{-1}$ at depths of 500–700 m. This can result in the release of gas at 150 $m^3\,min^{-1}$ on a coal face producing 5000 t $day^{-1}$. 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 permeability and the permeability of coal beds 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. The gas extracted by degassing methods totals some $12 \times 10^6\ m^3yr^{-1}$, and has been utilized for heating in the above-ground mine facilities (Baimukhametov, 2006).

In the United Kingdom, conventional methane drainage has concentrated on boreholes that give access to methane 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 methane 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, that is 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 (IMC, 1997).

As well as the presence of gas in the operating mines, the problem of surface gas emissions associated with coal mining has greatly increased in the United Kingdom, due to the rapid run down of the United Kingdom coal industry. The large volumes of gas in abandoned mines causes not only surface emissions, but gas from abandoned workings migrates into adjacent operating mines resulting in high gas levels, particularly during falls in barometric pressure (IMC, 1997). Operating mines have exercised direct control of mine gas by means of ventilation and water pumping, often over large areas, however, these methods cannot be continued and maintained for abandoned workings indefinitely. 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 eight CMM drainage schemes are in operation in the United Kingdom from abandoned underground mines. These are privately run and sell gas to generate electricity, totalling approximately 40 MW. New CMM sites are being investigated so an increase in the amount of CMM drainage in the United Kingdom is likely over the next few years.

Where CMM drainage has been established successfully, a mine can utilize its CMM on site or transport it

**Table 11.2** Estimated coal-mine methane (CMM and AMM) projects and utilization.

| Country | CMM projects | AMM projects | Utilization |
|---|---|---|---|
| Australia | 10 | 5 | Power generation, VAM oxidation |
| China | 40 | 0 | Town gas, power generation, industrial use, vehicle fuel, pipeline injection |
| Czech Republic | 1 | 0 | Pipeline injection |
| Germany | 9 | 36 | Heat and power generation |
| Kazakhstan | 1 | 0 | Boiler fuel |
| Poland | 21 | 0 | Power generation, coal drying, industrial use, boiler fuel |
| Russia | 7 | 0 | Power generation, boiler fuel |
| United Kingdom | 8 | 8 | Power generation, boiler fuel |
| Ukraine | 9 | 0 | Power generation, heating, industrial use |
| United States | 13 | 26 | Power generation, coal drying, heating, pipeline injection |
| Total | 104 | 49 | |

Source: various, including US Environmental Protection Agency (http://www.epa.gov) and United Kingdom Coal Authority (http://www.coal.gov.uk).

to a dedicated user. In the United States, a number of mines sell their CMM through existing gas pipelines, this is dependent on the quality of the recovered CMM meeting set standards and that the production, processing and transportation are competitive with other gas sources. The United States has accounted for $CH_4$ emissions from its abandoned underground mines, with 9% (13.5 Bcf) of all coal-related CMM emissions coming from this source between 1998 and 2006 (WCA, 2011). Table 11.2 summarizes a number of CMM and AMM projects worldwide and their utilization of mine gas.

***Production of coal-bed methane in unmined areas***

*Well-production methods.* The production of CBM from surface wells penetrating virgin or unmined coal has experienced phenomenal growth worlwide over the past 20 yr. Studies of the major coal-bearing basins in the world suggest that more than 50% of the estimated *in situ* 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). Major concerns are, first, the effect of horizontal and vertical stress components on deep-lying coal beds, where 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, where 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 United States, in the San Juan Basin, New Mexico, a CBM well drilled in 1953 has produced 150–180 Mcf day$^{-1}$ (4.2–5.1 Mcm day$^{-1}$) for 30 yr. 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 (Fenniak, 2004).

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 11.9a). Reservoir pressure is reduced by dewatering the coal bed, 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.

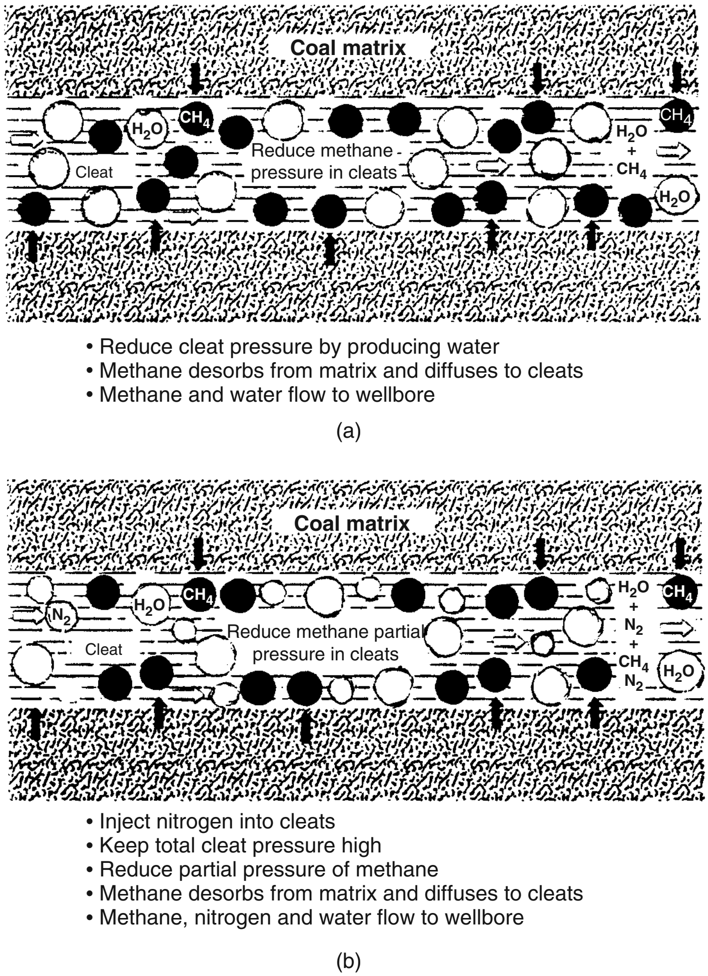



**Figure 11.9** (a) Coal-bed gas recovery by reservoir pressure depletion. (b) Enhanced coal-bed gas recovery by use of nitrogen injection. (From Murray (1996), based on Puri and Yee (1990).)

However, this method does not recover much more than 50% of the CBM in place. To overcome such low recovery rates, a series of experiments has been carried out in the United States on reservoir enhancement. One method is the injection of an inert gas such as nitrogen into the coal, which lowers the partial pressure in the coal and allows a greater percentage of CBM to be recovered (Figure 11.9b). 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, United States, and has been tested in Alberta, Canada. The use of dynamic openhole cavity completion techniques has also been developed in the United States. 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 over pressuring 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 United States leads the field in such experimentation, driven by the fact that CBM has become an increasingly important energy resource available to large energy consumers.

*World production.* Current estimates of global in-place CBM resources range from 4000 to 5000 Tcf (113–143 Tcm) of which 1500 Tcf (42 Tcm) is deemed recoverable. Those countries with the largest CBM

**Table 11.3** Major coal resources (BP, 2011) and estimated coal-bed methane resources in the world.

| Country | Coal resource ($10^6$ t) | Methane resource (Tcf, in-place) |
|---|---|---|
| Russia | 157,010 | 2824 |
| China | 114,500 | 1100 |
| United States | 245,088 | 700 |
| Canada | 6518 | 100–550 |
| Australia | 76,400 | 35 |
| Germany | 40,699 | 100 |
| United Kingdom | 228 | 100 |
| Kazakhstan | 33,600 | 25 |
| Poland | 5709 | 424 |
| India | 60,600 | 70 |
| Zimbabwe | 502 | 40 |
| Ukraine | 33,873 | 60 |
| Indonesia | 5529 | 453 |
| France | No longer exploited | 368 |
| Total | 780256 | 6399–6849 |

Tcf = trillion cubic feet.
Source: multiple (2011).

**Table 11.4** Coal-bed methane (CBM) proved reserves and production from principal United States producing areas.

| Area | CBM proved reserves 2007 (Bcm) | CBM production 2007 (Bcm) |
|---|---|---|
| Alabama (Black Warrior Basin) | 2127 | 114 |
| Colorado (Piceance, San Juan) | 7869 | 519 |
| New Mexico (San Juan) | 4169 | 395 |
| Utah (Uinta Basin) | 922 | 73 |
| Wyoming (Powder River Basin) | 2738 | 401 |
| Virginia (central Appalachians) | 1948 | 85 |
| Eastern States (IL,IN,OH,PA,WV) | 393 | 31 |
| Western States (AR,KS,LA,MT,OK) | 1709 | 136 |
| Total United States | 21,875 | 1754 |

Source: US Energy Information Administration (EIA, 2010).

resource potential are shown in Table 11.3. Of these, China, Russia and the United States have the largest resources, with other significant resources in Australia, Canada, Europe, India, Indonesia, Kazakhstan, South Africa and Ukraine.

Over the past 27 yr, the United States has led the world in developing CBM production on a large scale. The key United States producing areas are the Powder River Basin and San Juan Basin in the Rocky Mountains area, together with the Black Warrior Basin and Appalachian region in the eastern United States. There are over 15,000 producing wells in the United States, and the annual production in 2007 was 1.75 Tcf (EIA, 2009). Figure 11.10 shows the growth of CBM production in the United States from its beginnings in 1984 to reaching 1.0 Tcf by 2000. During the 1990s the San Juan Basin in Colorado and New Mexico accounted for the bulk of United States CBM production, since then other areas such as the Powder River Basin in Wyoming have significantly increased both their resources and production. Table 11.4 shows proven CBM reserves and production for the principal producing states in the United States, and Figure 11.11 shows the geographical distribution of the principal CBM resources in the United States.

In Canada it is estimated that there are 100–550 Tcf (2.8–15.5 Tcm) of CBM in the Province of Alberta, with additional resources in British Columbia. Development of CBM in Canada lags behind the United States, in part due to lack of permeability in coals and a number of environmental issues (Stanley, 2005).

In China there are large CBM resources, estimates at 1100 Tcf (30 Tcm) are considered the second largest CBM resource in the world. The estimated recoverable CBM resources are 350 Tcf (10 Tcm), and 95% of the CBM resources are located in four main areas, namely, Shanxi–Shaanxi–Inner Mongolia, Hebei–Henan–Annui, Yunnan–Guizhou–Sichuan–Chonqing and in the Xinjiang region in western China (Figure 11.12). Of these, the Shanxi–Shaanxi–Inner Mongolia region has the most CBM resources, estimated at 600 Tcf (17.25 Tcm). In this area, the Ordos and Qinshui Basins have the highest potential. Geological conditions, with the exception of the Ordos Basin, are structurally more complex than the commercial areas in the United States, and become increasingly more complex towards the east. The Ordos Basin has coals at 300–1500 m with simple structure, and has an estimated CBM resource of 50–100 Tcf (1.4–2.8 Tcm). The Qinshui Basin also has a moderately shallow perimeter suitable for CBM development, but there are numerous structural elements present. The Hebei–Henan Basin comprises a number of discrete fault-bounded coalfields containing coals of bituminous rank,

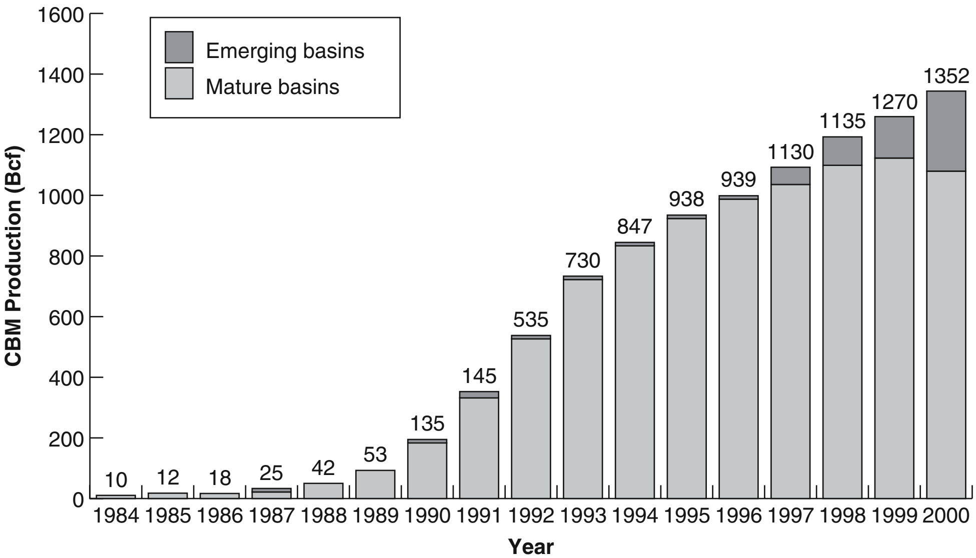



**Figure 11.10** Coal-bed methane production from United States (lower 48 States) coal basins. (From EIA, 2009.)

and has a number of gassy coal mines. Texaco have drilled wells in this region and obtained a gas content of 12 m3 $t^{-1}$ at 610 m depth (Stevens, 1999). The North China Bureau of Petroleum Geology has carried out CBM exploration at the Liulin CBM pilot site in Shanxi Province. Three coal beds with a combined thickness of 8–10 m have been selected, at a depth of 340–400 m, dipping at 3–8°. The coals contain 20% ash and 80% vitrinite and the $CH_4$ content ranges from 10 $m^3$ $t^{-1}$ to 20 $m^3$ $t^{-1}$ and the estimated resource is 1.6 Tcm $km^{-2}$ (Xiaodong and Shengli, 1997). Both Phillips Petroleum and Lowell Petroleum have also carried out tests in the Liulin and Hedong region of the eastern Ordos Basin, and Arco have also drilled a number of test wells in the eastern Ordos Basin and have obtained favourable results (Stevens, 1999). The results of the Liulin tests indicate that China has coal-bearing areas with high CBM potential, comparable to those in the United States. Although historically a number of technical and logistical problems have been encountered, the potential in China for CBM production on a commercial scale is still great, and advances can be expected to exploit this energy resource. Figure 11.12 shows CBM interests in China.

Russia has an estimated CBM resource of 2824 Tcf (80 Tcm). The Kuzbass Basin provides one of the largest CBM resource development opportunities with resources of 460 Tcf (13 Tcm) at 1800–2000 m depth (Figure 11.13). The Pechora Basin has a CBM resource of 77–120 Tcf (2.2–3.4 Tcm), but this area's harsh climate may limit exploitation of this resource. In the Kuznetsk Basin, all of the underground coal mines are gassy, and more than 1.2 Tcm was liberated in 1991 (Marshall, Pilcher and Bibler, 1996), with only 17% being removed by mine drainage systems. In 2003, Gazprom launched a project to determine the potential of CBM production in Kuzbass Basin (Gazprom, 2011). In this area, in 2008–2009, eight wells were drilled in the Taldinskoye field, and forecasted resource estimates are 3.35 Tcf (95 Bcm). Further investigation may well prove that a much greater volume is present in the Basin. On the west Russian and south Ukraine border, the Donetsk Basin has CBM resources of 430–790 Tcm of which 100–200 Tcm are in virgin coal reserves. These figures also may be underestimated. Until recently, Russia has lacked the technology and, more significantly, the capital to develop further CBM production. It is clear that CBM usage would benefit the industrialized regions by replacing brown coal and low-quality black coal combustion in terms of air quality. It is expected that CBM production will become a significant contributor to the Russian energy needs.

In Australia, the identification of CBM resources has, and is being, assessed, notably in the Gunnedah Basin, Gloucester Basin and Clarence-Moreton Basin (Figure 11.14). In the Gunnedah Basin, exploration has identified over 35 Tcf (1.0 Tcm) of CBM *in situ*, over an

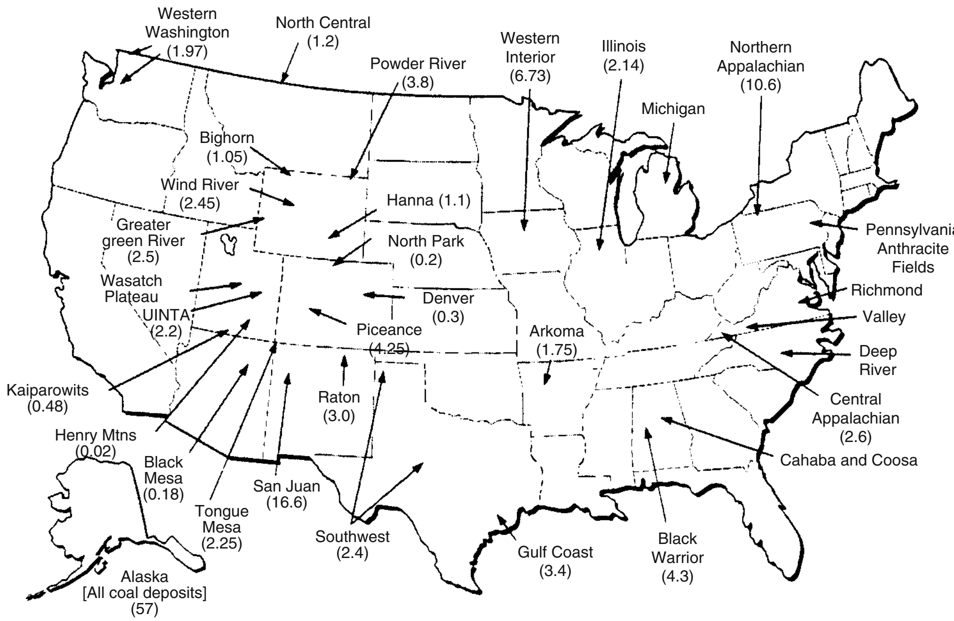



**Figure 11.11** Coal-bed gas resources of the United States (in trillion cubic feet, Tcf); includes probable, possible and speculative potentially producible supply. (From Murray, 1996.)

area of 9500 km$^2$. In the Gloucester Basin, exploration has shown that favourable geological conditions exist for CBM exploitation. Coal-seam permeabilities are high, seams are higher rank and have low *in situ* stress. Conditions in the Clarence-Moreton Basin suggest that there is high potential for CBM but as yet the basin has not been explored extensively. There have been low levels of exploration in the Surat, Sydney and Darling Basins, but a new approach to gas exploration and open access to markets should encourage further exploration for gas in these areas.

In India, tests have been carried out in the Jharia Coalfield, Bihar, where recoverable resources are estimated to be 8.8–17 Tcf (0.25–0.5 Tcm; at a 50% recovery rate), and a limited amount of exploration has taken place in the Raniganj Coalfield, West Bengal. Current estimations suggest a conservative resource of 70 Tcf (2 Tcm) of CBM is present in the coal basins of India (Bhaskaran and Singh, 2000). Further extensive exploration for CBM in India will give a more accurate reflection of the country's CBM producing potential, to this end a number of contracts for potential CBM block areas have been leased.

In Poland, CBM exploration has been carried out in the Lower and Upper Silesian Basins, the latter being the main focus of interest. An estimated resource of 424 Tcf (12 Tcm) has been given for the Upper Silesian Basin (Wojcicki, 2007). With the closure of the Lower Silesian coal mines, and rationalization of the mines in the Upper Silesian Basin, production of CBM will be critical to the maintenance of energy supplies in these industrialized areas.

Coal-bed methane production has commenced in the Lorraine Coalfield in northeastern France, where in 2011 CBM resources of 8.5 Tcf (240 Billion m$^3$) were estimated by European Gas (2011). This project has rekindled interest in CBM in the neighbouring region in Germany.

In Kazakhstan the Karaganda Basin has a high gas content as mentioned above, with $CH_4$ resources being estimated at 19.4 Tcf (550 Bcm). The Ekibastuz Basin has CBM resources of 2.6 Tcf (75 Bcm), and the estimated total CBM resource within Kazakhstan is 25.2 Tcf (713 Bcm).

In the United Kingdom the past decade has seen a significant increase in exploration for CBM. Initial targeting of CBM prospects has been 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

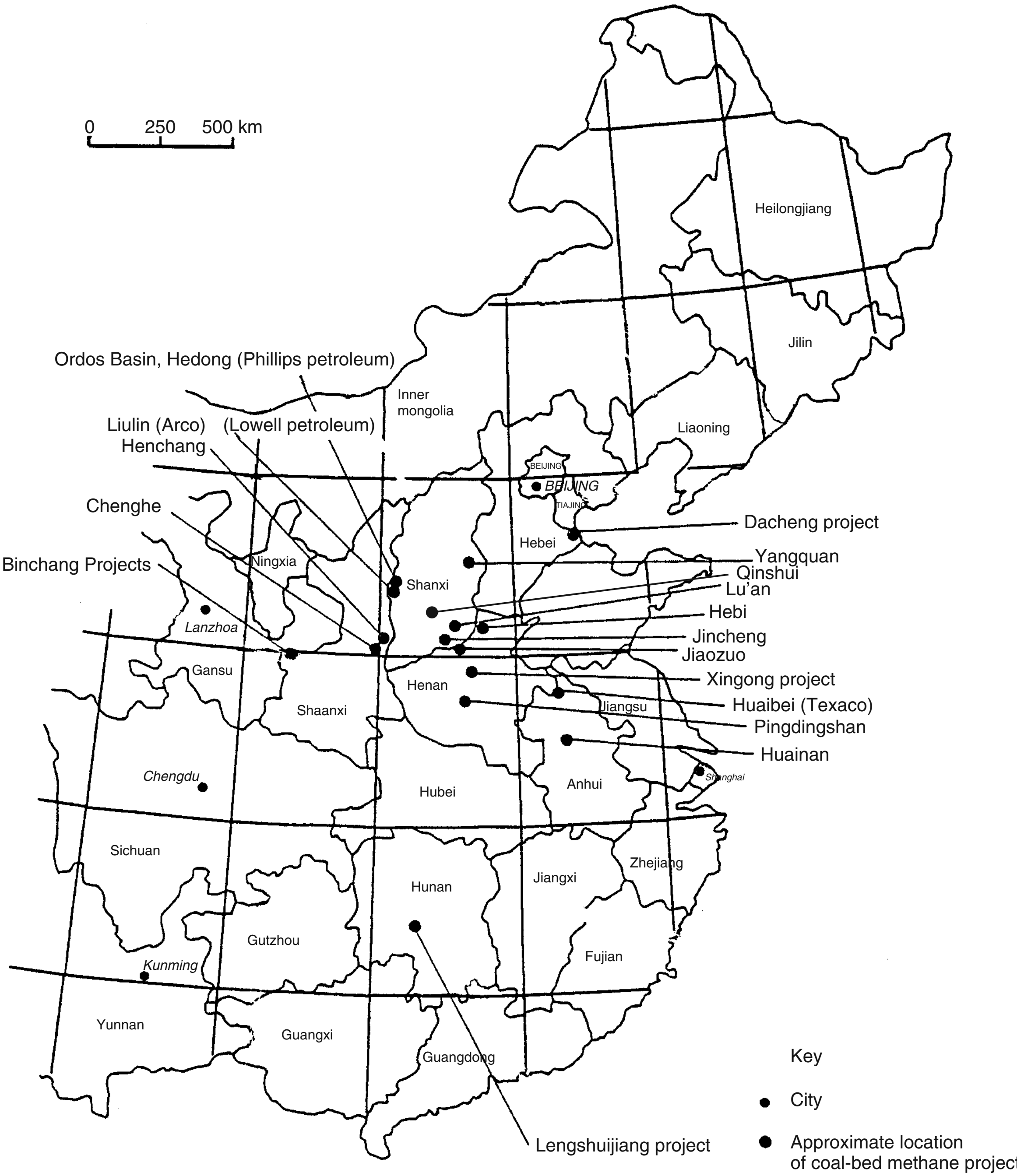



**Figure 11.12** Coal-bed methane investigations in People's Republic of China, including areas of western oil company participation. (Dargo Associates Ltd, 2011.)

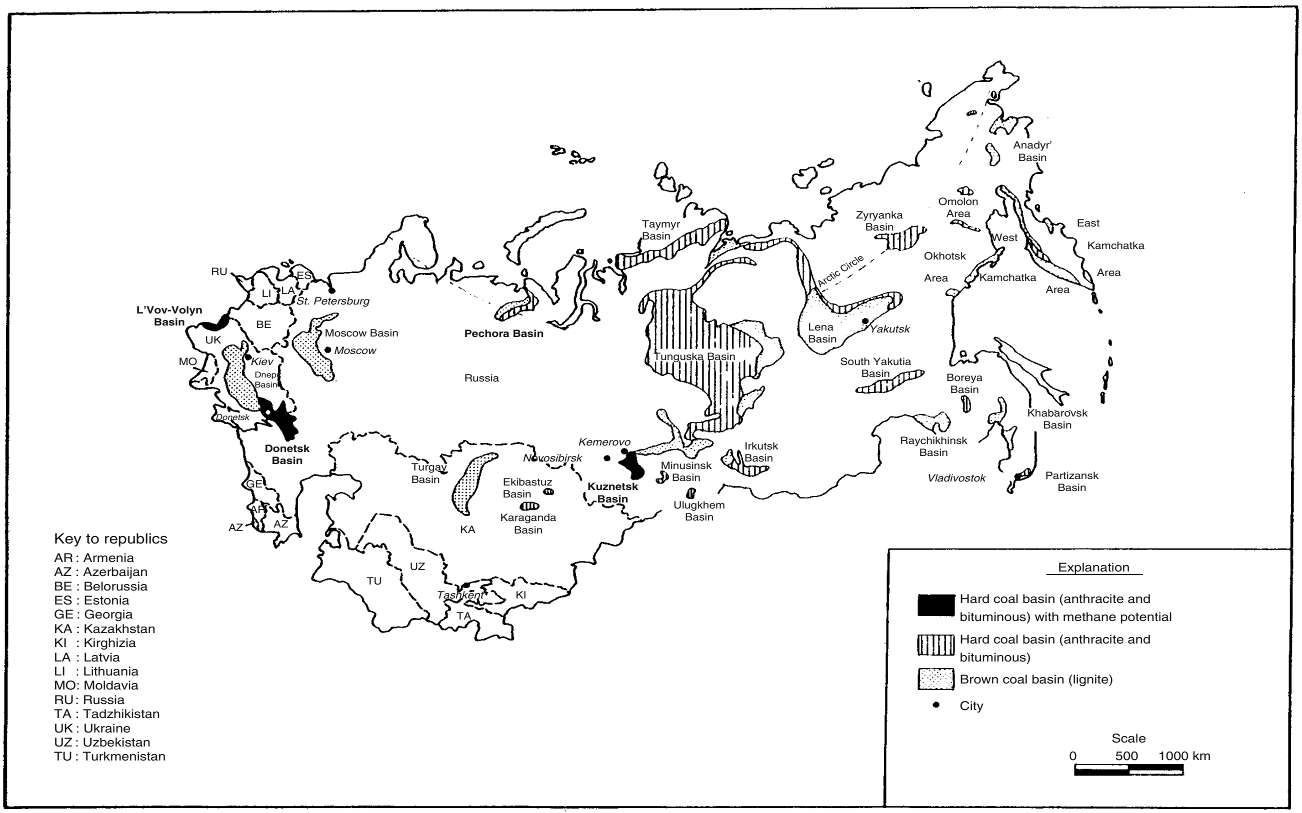



**Figure 11.13** Major coal basins of the former USSR, including those with the highest coal-bed methane potential. (From Marshall, Pilcher and Bibler, 1996.)

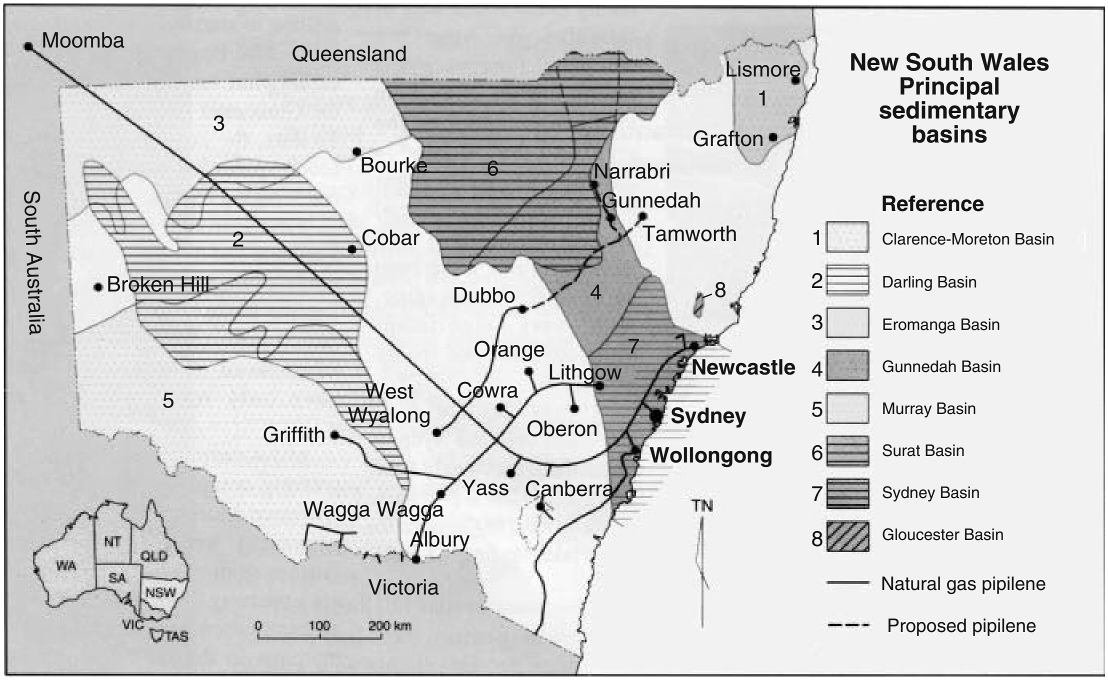



**Figure 11.14** Principal sedimentary basins with oil and gas potential in New South Wales, Australia. (Reproduced with permission from MINFO®, 2000.)

content (Bailey *et al.*, 1995). The coalfield areas of the United Kingdom (Figure 11.15) vary in rank, in confining pressure and in gas content, which along with low coal-seam permeability will constrain CBM production in many parts of the United Kingdom (Creedy, 1999). The South Wales Coalfield has high rank coal combined with significant confining pressure, and other areas have lower rank coal preserved at greater depth. These areas appear the most attractive for CBM exploration. Coal-bed methane development is still at an early stage in United Kingdom, however, a number of licences have been granted for both conventional CBM wells and also for gob gas and mine drainage wells. Coal-bed methane 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, using 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.

In Zimbabwe, 40 Tcf (1.1 Tcm) of potentially recoverable CBM is estimated to be in place in the Lupane-Lubimbi area. Expressions of interest to develop this resource are actively being sought.

In Brazil, the Charqueadas-Santa Rita Basin, Rio Grande do Sul, has been under intensive study. Evaluation of the CBM potential of this area is ongoing (Martins *et al.*, 2010). In Japan, on the island of Hokkaido, a coal seam at 900 m depth is being studied with a view to extract $CH_4$ and store $CO_2$ (Fujioka, Yamaguch and Nako, 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. It is clear that in the future the utilization of both CBM and CMM will become a major contributor to the energy needs of numerous countries worldwide.

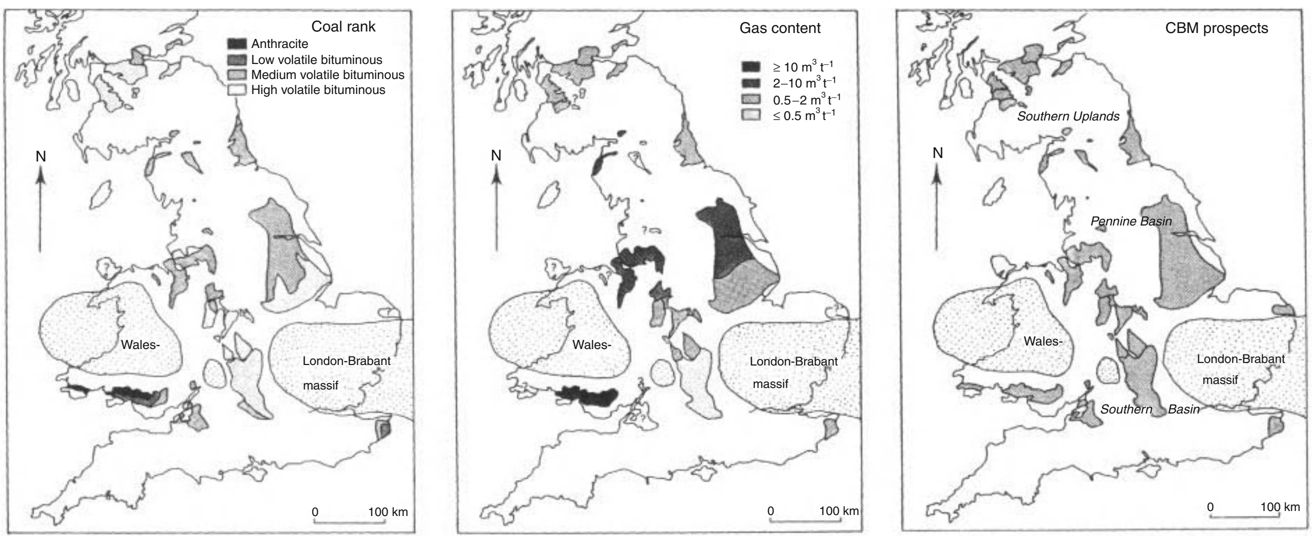



**Figure 11.15** Rank map of Carboniferous coals, map of hypothetical gas content of Carboniferous coals and the principal coal-bed methane prospects in Great Britain. (From Bailey *et al.*, 1995.)

## 11.3 Underground coal gasification (UCG)

The concept of the underground gasification of coal was first envisaged by Sir William Siemens (1868) and Mendelev (1888). A patent was issued in United Kingdom 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 United States and Europe during the 1970–1980 energy crisis. Commercialization never took place due to a 20 yr 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 United Kingdom. China has conducted a number of trials and has several commercial UCG projects in operation, and a number of other countries are carrying out field and laboratory testing.

### 11.3.1 Underground coal gasification – the case for and against

The case for developing UCG has advantages and disadvantages, as outlined by Burton *et al.* (2007). Underground coal gasification has numerous advantages over conventional underground or opencast mining and surface gasification, namely:

1. conventional coal mining is eliminated with UCG, reducing operating costs, surface damage and eliminating mine safety issues;
2. coals that are unmineable (too deep, low quality, too thin) are exploitable by UCG, thereby increasing domestic resource availability;
3. surface transportation of coal at the surface is eliminated, reducing cost, coal stockpiling and shipping;
4. no surface gasification facilities are required, reducing capital costs;
5. ash in coal remains underground, avoiding ash disposal at the surface;
6. a reduction in $SO_x$ and $NO_x$ and other pollutants;
7. UCG produces less greenhouse gas compared with conventional mining and surface combustion – the

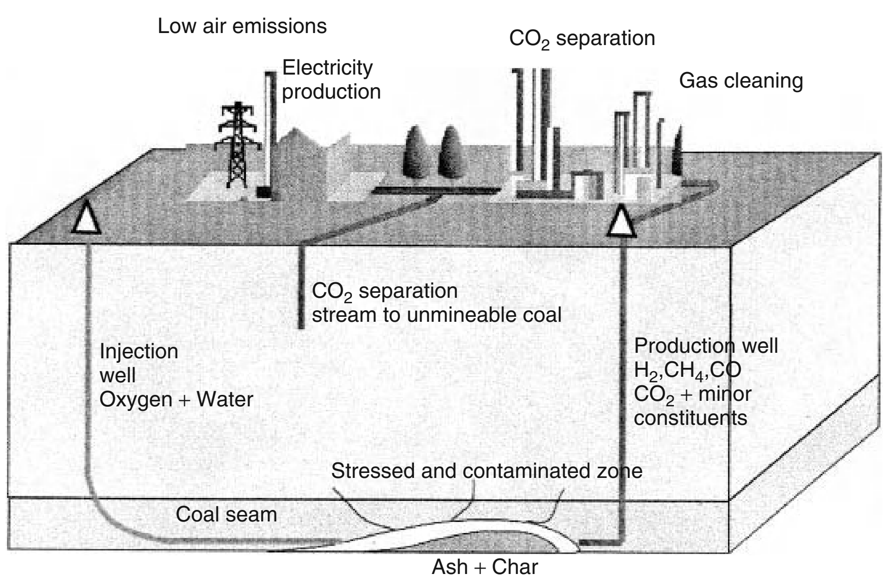



**Figure 11.16** The main components of a commercial underground coal gasification site for power generation. (From UCG Engineering, 2006.)

well infrastructure for UCG can be used subsequently for $CO_2$ sequestration operations;
8. coal seam gas technology can extract 3–5% of the energy (as methane) in the coal, whereas UCG extracts more than 80% of the coal's energy.

These advantages are illustrated in Figure 11.16 which shows the layout of a commercial UCG site for power generation (UCG Engineering Ltd, 2006).

The UCG technology is still not perfect and there are a number of limitations, namely:

1. UCG can have serious environmental implications, in particular, aquifer contamination and ground subsidence – site selection and operation need to be carefully assessed;
2. 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;
3. UCG operations cannot be controlled to the same extent as surface gasifiers – water influx, distribution of reactants in the gasification zone and the growth of the cavity can be estimated only from measurements of temperatures and product gas quality and quantity;
4. until a reasonable number of UCG-based power plants are built and operated, the economics of UCG has major uncertainties;
5. 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 involves the following carbon gasification reactions, as outlined by ETSU (1993), Burton, Friedmann and Upadhye (2007) and Paul Ahner (personal communication, 2011).

1. The primary oxidation reaction: when oxygen ($O_2$) is passed over/through hot coal it combines with carbon (C) to form carbon monoxide (CO), which is a flammable gas. This is an exothermic reaction (i.e. it produces heat). If too much oxygen is supplied, then carbon dioxide ($CO_2$) is produced which is inert, therefore careful regulation is required.
2. The steam-char reaction: when water or steam ($H_2O$) is passed over/through heated coal, the oxygen in it combines with carbon to form CO, and releases the hydrogen ($H_2$) 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 $O_2/H_2O$ ratio.
3. The $CO_2$ reduction reaction (reverse Boudard Reaction): when $CO_2 + C$ produces CO.
4. The water gas shift reaction: when $CO + H_2O$ produces $H_2$ and $CO_2$.
5. The methane synthesis reaction: at high pressures achievable in the gasification of deep coal, $H_2$ combines with C to form methane ($CH_4$). This reaction is beneficial in that it increases the CV of the product, and since it is exothermic, it reduces the amount of input $O_2$ required to gasify a unit mass of coal.
6. Pyrolysis: when coal + heat produces $CH_4 + CO + H_2$ + light hydrocarbons.

Since the desirable reactions require high temperatures and excess carbon (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-carbon contact and decreasing excessive water influx into the reactor. To keep the potential contaminants inside the reactor, some groundwater influx is required, and 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 (Paul 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, and coal at the base of one well is 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 'hydrofraccing', 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 enters the seam only at the area to be gasified, where it intersects the production well (Figure 11.17). Another method developed by the Lawrence Livermore National Laboratory in the United States 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 the process start-up, after which the horizontal production well is used. A liner is inserted in the injection well and

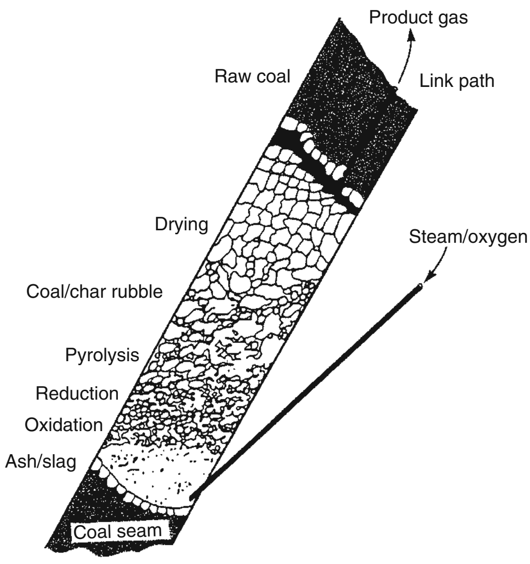



**Figure 11.17** Typical configuration for steeply dipping bed (SDB) gasification. (From Oliver and Dana, 1991.)

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 11.18a and b 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 oxygen 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 11.19). 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 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 11.20 (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.

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 11.5 (Paul 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

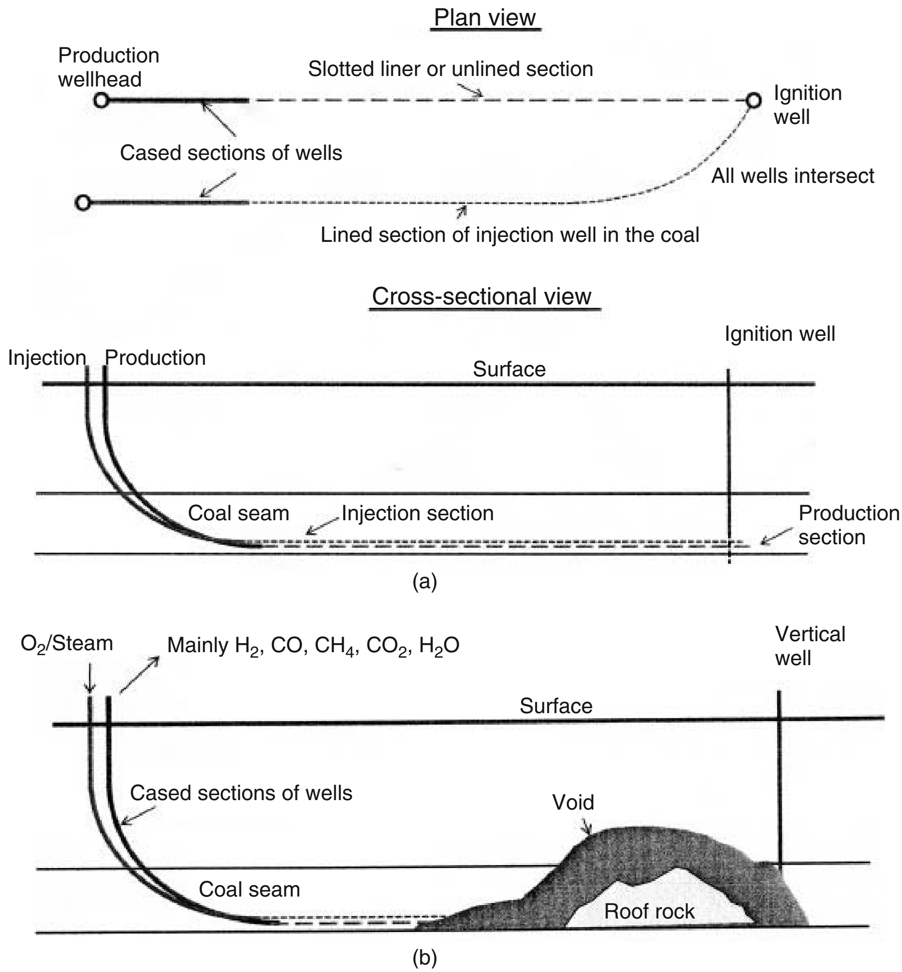



**Figure 11.18** (a) Plan and cross section views of a UCG panel before gasification. (b) Cross section view of an operating UCG panel showing roof fall. (Reproduced by permission Paul Ahner pfaconsulting@sbcglobal.net (2011) personal communication.)

is then to extend the area of combustion by drilling additional injection wells. The final phase of the UCG operation is to extinguish the fire. This is done by injecting nitrogen ($N_2$) 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 the UCG field trials require an efficient control and monitoring system. This will include the development of a series of boreholes to 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 'break through' occurs.

Well documented larger United States field tests yielded the thermal efficiencies shown in Table 11.6. 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*

In the former USSR, in the 1950s a number of UCG stations were built in Tula, Yushno-Abinsk, Shatsky and Angren, all were designed to burn lignite. In addition,

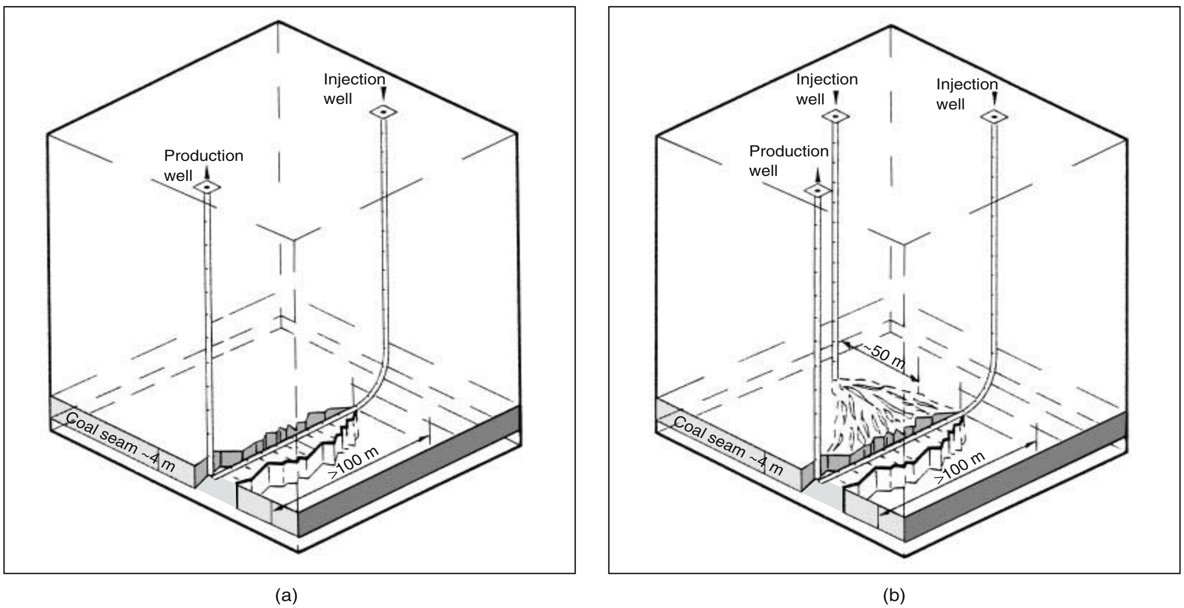



**Figure 11.19** (a) Injection and production wells using the controlled retracting injection point (CRIP) technique. (b) Addition of an offset well designed to extend the combustion zone laterally. (From ETSU, 1993.)

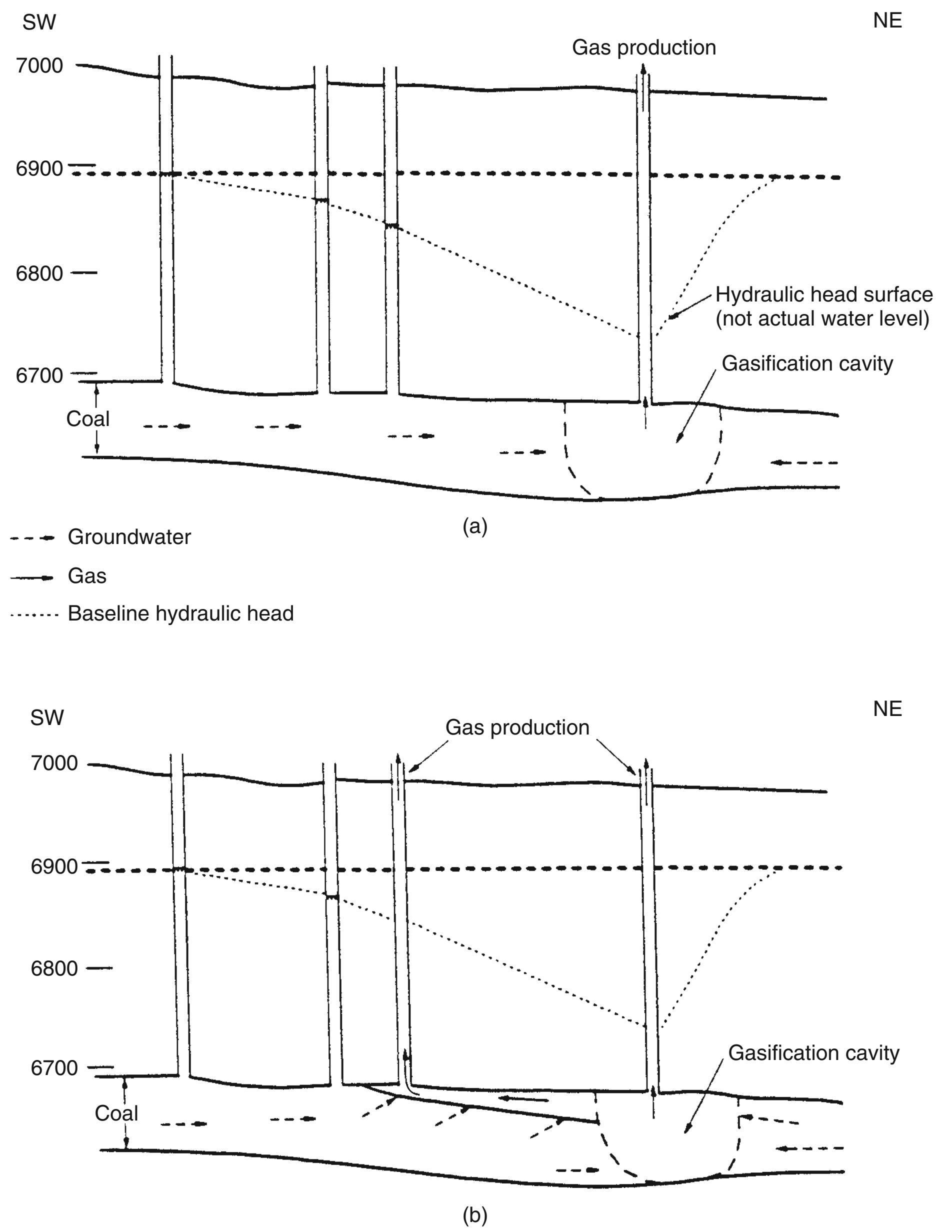



**Figure 11.20** Schematic diagram of hydrogeological relationships under high- and low-pressure operating conditions during the Rocky Mountain 1 underground coal gasification test. (a) Under typical operating conditions (cavity pressure less than hydrostatic), groundwater moves toward the cavity where 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 southwest part of the site. (From Beaver *et al.*, 1991.)

**Table 11.5** Desired underground coal gasification site characteristics Paul Ahner pfaconsulting@sbcglobal.net (2011) personal communication.

| 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 yr assuming that 62% of the coal is gasified and a power generation efficiency of 43% |
| Coal rank | Lignite to bituminous coals are amenable to UCG. A swelling bituminous coal with a FSI $>3$ can present plugging problems which will require special consideration. The folded and faulted nature of anthracite coals precludes it from UCG |
| Coal structure | Steeply dipping strata at $63^\circ$ have been very successfully gasified in the United States. The increased cost of drilling as the process moves further down dip must be considered. UCG in coal seams dipping $<5^\circ$ 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: Paul Ahner, personal communication (2011).

**Table 11.6** Efficiencies from typical large underground coal gasification pilot test in the United States.

| | Rocky Mountain 1 | Rawlins 2 | Hanna | Hoe Creek |
|---|---|---|---|---|
| Tons of coal gasified* | 11,227 | 8560 | 14,182 (sum of five tests) | 3251 |
| Energy in dry gas** (%) | 79.3 | 81.9 | 79.3 (average from five tests) | 73 |
| Energy in tars (%) | 4.9 | 1.8 | – | – |
| Energy in sensible heat (%) | 3.7 | 5.5 | – | – |
| Energy in steam produced (%) | 6.1 | 5.3 | – | – |
| Energy lost to formation (includes product gas) (%) | 6 | 5.5 | – | – |

*All public United States tests that gasified more than 2000 tons of coal.
**% basis – total energy produced/total energy in the coal gasified, determined from a carbon balance.
Source: Paul Ahner, personal communication (2011).

**Table 11.7** Underground coal gasification production in the former USSR.

| Installation | Production ($10^6$ $m^3$ gas $yr^{-1}$) | Calorific value (Kcal $m^{-3}$) |
|---|---|---|
| Tula (Russia) | 400 | 750–850 |
| Yushno-Abinsk (Kuznetsk Basin) | 100 | 1000 |
| Shatsky (Russia) | 200 | 800 |
| Kamenskaya (Donets basin) | 730 | 900 |
| Lisichansk (Donets Basin) | 120 | 850 |

Source: based on Douchanov and Minkova (1997).

UCG installations were constructed at Lisichansk and Kamanskaya in bituminous and anthracite coals. These installations have produced gas at varying CVs (Table 11.7), but only the stations at Yushno-Abinsk and Angren remained in operation after 1980 (Douchanov and Minkova, 1997).

Tests in the United States, at the Energy Center in Wyoming, involved hydrofracturing of coal at 120 m depth to enhance the permeability, and in 1973 8.5 Mcf (0.24 Mcm) gasper day were produced. The Morgantown Energy Center has produced gas at 3.5 Mcf (0.1 Mcm) per day with a CV of 1100 Kcal $m^{-1}$ from coal seams 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 $m^3$) 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 ca. 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 pyrolitic 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 two of the 30 United States field tests and lessons were learnt.

At the same time tests have been carried out in Europe, in the United Kingdom, France, Czech Republic, Italy, Hungary and Spain. The coal seams in Europe are characterized 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 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 (EWG) 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 11.19). 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 United States, 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 Department of Trade and Industry (DTI) in the United Kingdom identified UCG as one of the potential technologies for the development of United Kingdom'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 1997–2003. The long-term goals of the project are power production and liquid fuels production using gas-to-liquids technology. Nine process wells were drilled and are producing syngas from a 10 m thick coal seam at a depth of 140 m at a rate of 80,000 $Nm^3\,h^{-1}$ at LHV of 5.0 $mJ\,Nm^{-3}$ 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.

China has had the largest UCG programme, including 30 projects in different phases of development. These include the Xinhe #2 mine test, the industrial trial at Liuzhuang mine in Tangshan, XinWen's tests at Suncun in Shangdong and the Caozhuang mine in Feicheng. The method utilizes the abandoned galleries of coal mines 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 hydrogen. China is also developing an industrial pilot UCG project in the Northern Inner Mongolia Autonomous Region at Gonggou mine, targeting a coal seam at 200 m depth. This is expected to produce 1.5 $Mcm\,day^{-1}$ of syngas, and to generate $32.4 \times 10^6$ $Kwh\,yr^{-1}$ of power by 2010 (International Energy, 2007).

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, these include 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, there remain several key scientific and technical gaps, which further research should address. One important possibility is that the challenge of managing $CO_2$ emissions can link UCG with carbon capture and sequestration (CCS).

## 11.4 Coal as a liquid fuel

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.

### *11.4.1 Coal liquefaction technology*

There are various methods of coal liquefaction, the main problem being the deficiency of hydrogen in coal compared with liquid fuels. This can be overcome by adding hydrogen to the coal by a number of processes:

1. direct liquefaction by hydrogenation;
2. indirect liquefaction by the Fischer–Tropsche (FT) synthesis;
3. removal of part of the carbon 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 hydrogen deficiency, the H:C ratio is increased by adding a hydrogen donor. Coal is dispersed in a thermally stable 'solvent' and/or 'hydrogen-donor' and passed into a pressurized autothermal reactor at temperatures between 400° and 500°C. If additional hydrogen is not supplied to the reactor, the hydrogen-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 $t^{-1}$ 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 a hydrogen additive.

The Fischer–Tropsche (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.

The FT synthesis involves the gasification of the coal, carried out in a Lurgi-gasifier to produce the synthesis gases carbon monoxide, hydrogen and methane. The methane is treated in a gas reformer and synthesized in a Kellog reactor. The carbon monoxide and hydrogen 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 $yr^{-1}$ is used by Sasolburg in plants designed to produce 50,000 bbl $day^{-1}$ of gasoline and other products for chemical feedstocks from the processing of 30,000 t $day^{-1}$ of coal (Sage and Payne, 1999).

Pyrolysis involves the heating of pulverized 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.

### 11.4.2 Coal properties for liquefaction

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 hydrogen contents enhance the liquid yields. The required high H:C ratio is closely linked to rank and petrographic composition, the latter requiring 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 then decreases with increasing rank. Inertinite shows varying degrees of reactivity in coal liquefaction (Taylor *et al.*, 1998).

In hydrogenation, the use of coals containing high amounts of oxygen, nitrogen and sulfur, means that hydrogen is consumed in the removal of these heteroatoms. Because the supply of hydrogen 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 reaction throughout and increase problems of solid and liquid separation, and may deactivate any catalyst. Some inorganic material can have catalytic effects of their own, for example pyrite has favourable catalytic properties (Taylor *et al.*, 1998).

The gasification processes are comparatively insensitive to coal properties, and can utilize coals that would be unsuitable for other processes. For example, the FT synthesis used for 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) summarize those characteristics favourable for coal hydrogenation as:

| | |
|---|---|
| Vitrinite reflectance | $<0.8\%$ |
| H:C atomic ratio | $> 0.75\%$ |
| vitrinite + liptinite | $>60\%$ |
| volatile matter (daf) | $>35\%$ |
| low concentration of heteroatoms. | |

### 11.4.3 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 United States, South Africa, Germany, Japan and the United Kingdom. Other major coal producers have the potential to consider upgrading coal to supplement their own oil reserves and/or to reduce their oil imports, for example China, India and Poland. Additional smaller coal producers such as Indonesia, 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 United Kingdom, research and development into coal liquefaction has concentrated on the direct liquefaction process known as liquid solvent extraction (LSE), developed by British Coal. To date no process has been demonstrated at a commercial scale (Robinson, 1994). The model plant is designed to produce approximately 50,000 bbl day$^{-1}$ of liquid transport fuels from 17,000 t day$^{-1}$ of coal. In the first stage, bituminous coal is digested in a hydrogen-donating solvent in the absence of hydrogen, 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, Cloke and Belghazi, 1997).

In the United States, a variation on the pyrolysis technique is the international liquids from coal (LFC) process (Weber and Knottnerus, 2000). Its development has been influenced by the electricity generators in the United States changing from high-sulfur coals from eastern United States to low-sulfur coals from western United States. This change is 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 high-moisture coals (25–32%) with relatively low heating values (7900–8800 Btu lb$^{-1}$) from the Powder River Basin. 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 oxygen added to produce a stable solid fuel. The volatile matter driven off during pyrolysis is partially condensed in a multistep 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 northwest 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 (Piedad-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 produced high quality hydrocarbon products. The company is confident that it can use this gas source to make synthetic liquid hydrocarbons with a focus on producing diesel (ABN Newswire, 2011).

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, concerns over gas supplies and resistance to the development of nuclear energy, together with the greater availability of coal, the ideal economic scenario for future development of coal liquefaction technology could be provided.

## 11.5 Coal as an oil-prone source rock

In addition to the treatment of coal to produce alternative sources of energy, geological processes acting upon coal-bearing sequences have produced hydrocarbon reserves, which have been preserved as oil, condensate, and wet and dry gas.

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, China 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 Southeast 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 because recognition of such oils in a basin can indicate the presence of coals not previously identified.

### *11.5.1 Suitability of coal 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 and distinguish oil-prone coals from other coals, and these indicate that oil-prone coals are richer in hydrogen relative to carbon. It is generally accepted that sediments must contain moderate to high concentrations of hydrogen-rich kerogens in order to have significant oil-source potential. The progenitors of hydrogen-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, Cooper and Barnard, 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 hydrogen than vitrinite, which in turn is richer than inertinite. 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 11.21).

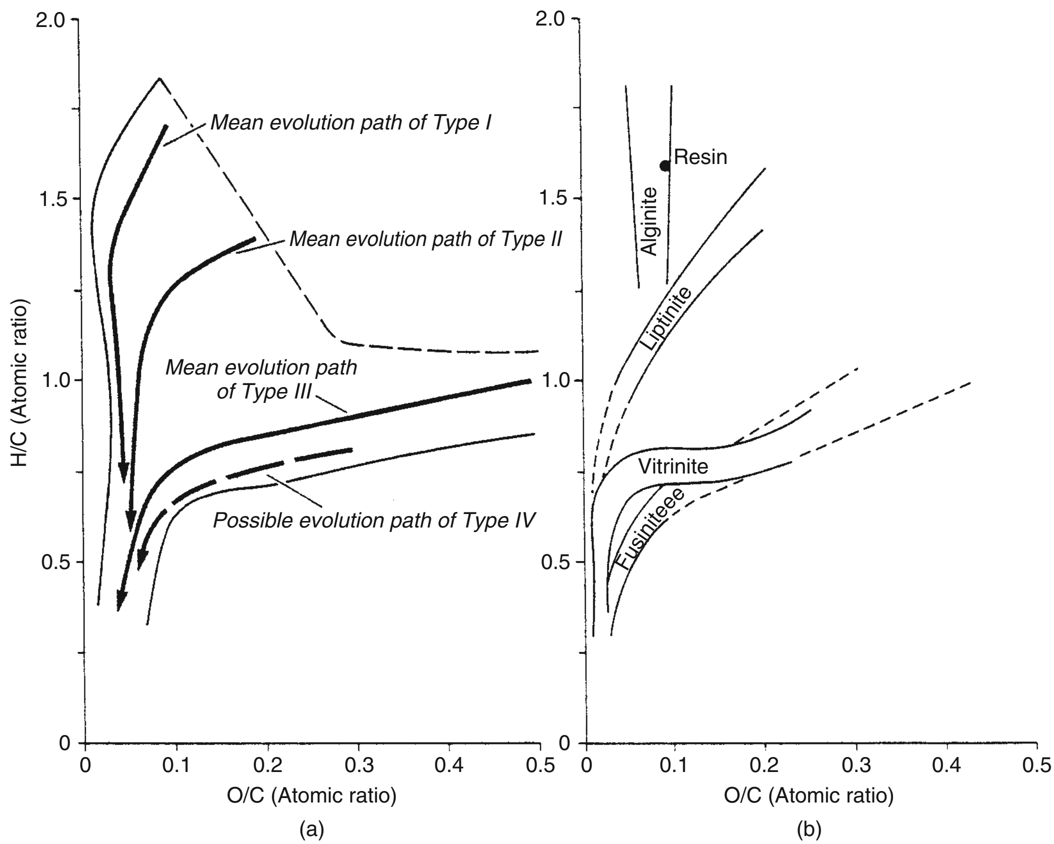



**Figure 11.21** Comparison of kerogen types, evolution paths and petrographic components of coal based on atomic ratios. (From Powell and Boreham, 1994.)

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 Hydrogen Indices in Rock-Eval analysis would be above 220–300 mg HC $gC^{-1}$ 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 Paleogene–Neogene oil-prone coal sequences of Southeast 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–Paleogene–Neogene basins of Australasia and the tropical Paleogene–Neogene basins of Southeast Asia (Macgregor, 1994).

Because hydrogen 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 hydrogen-poor and hydrogen-rich components. Rock-Eval pyrolysis records the release of hydrocarbons and $CO_2$ with increasing temperature, and determines the temperature of maximum hydrocarbon generation

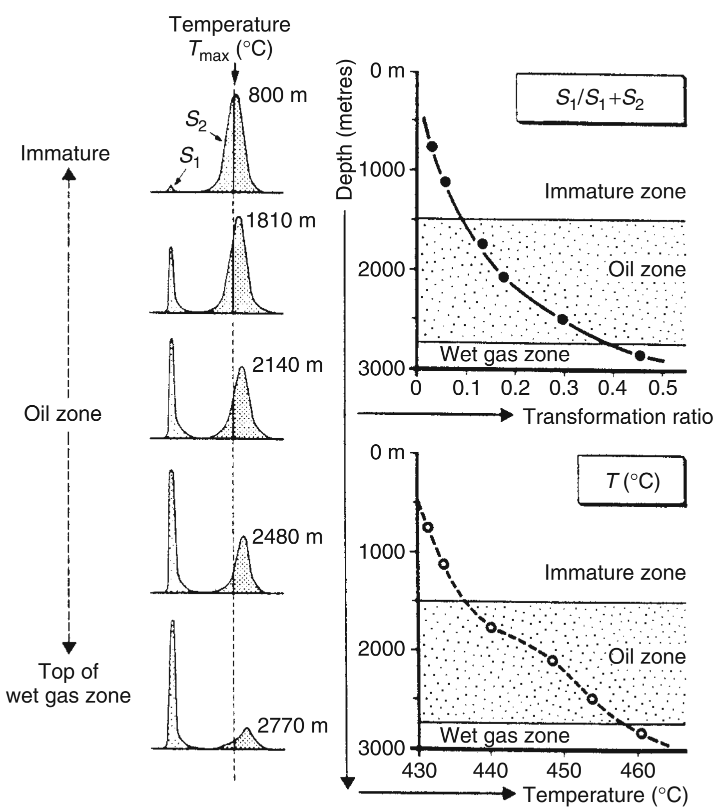



**Figure 11.22** Characterization of source-rock maturity by pyrolysis methods. Transformation ratio and/or peak temperature $T_{max}$ may be used as indicators of thermal evolution. (From Taylor *et al.*, 1998.)

($T_{max}$). 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 given as $S_1 : (S_1 + S_2)$, and $T_{max}$ both increase with increasing maturity, as shown in Figure 11.22, which illustrates the characterization of source rock maturity by pyrolysis methods (Taylor *et al.*, 1998). The ratio $S_2$/TOC (total organic carbon, expressed in weight %) or Hydrogen Index correlates with the atomic H:C ratio measured by elemental analysis on kerogen.

Isotopes can be used to characterize the total carbon or bulk fraction of a kerogen, oil or gas. Isotope analysis is considered most useful for characterizing 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.

### 11.5.2 Coal-sourced oil and gas occurrences

The principal oil basins sourced from coal-bearing sequences are listed in Table 11.8. The Gippsland Basin in Australia, and the Kutei Basin in East Kalimantan, Indonesia are the largest, most fully documented and least disputed cases of coal-bearing sequences to source oil provinces (Macgregor, 1994). Some 80% of Australian oil is attributed to a coal-bearing sequence source, namely from the Gippsland Basin. The Cooper, Eromanger and Taranaki Basins also contribute hydrocarbons, chiefly as large amounts of gas. In Southeast Asia, the proportion of oil reserves sourced from coal-bearing sequences is estimated at 10–30%. Other significant coal-sourced contributions are from Mid-Jurassic coals in China and Egypt. Minor reserves of oil may be present that are sourced from Paleogene–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 11.23 shows the hydrocarbon reserves in relation to geological age, and in particular, the change from gas-prone coals in

**Table 11.8** Examples of case studies of petroleum systems derived from terrigenous sediments, that is coal or terrestrially sourced organic matter.

| Country | Basin/Province | Source rock | | |
|---|---|---|---|---|
| | | Age | Hydrogen indices* | Reference |
| Australia | Gippsland Basin | Late Cretaceous–Paleogene–Neogene | 200–350 | Moore *et al.*, 1992 |
| | Cooper/Eromanga Basin | Permian | 150–300 | Vincent, Montimore and McKirdy, 1985 |
| | | Jurassic | 200–400 | |
| | Bowen/Surat Basin | Permian | 150–250 | Boreham (unpublished) |
| Canada | Beaufort-Mackenzie Basin | Eocene–Paleocene | 130–250 | Issler and Snowdon, 1990 |
| China | Turpan Basin | Jurassic | 200–500 | Zhao and Wu, 1997 |
| Indonesia | Ardjuna Sub-basin, Java | Late Oligocene | 250–400 | Noble *et al.*, 1986 |
| | Kutei Basin, Kalimantan | Middle Miocene | 200–350 | Durand and Oudin, 1979 |
| New Zealand | Taranaki Basin | Late Cretaceous–Paleogene–Neogene | 230–360 | Curry, Emmett and Hunt, 1994 |
| Nigeria | Niger Delta | Late Cretaceous–Paleogene–Neogene | $<200$ | Bustin, 1988 |
| Norway | Haltenbanken area, North Sea | Jurassic | 275 | Forbes *et al.*, 1991 |

*Hydrogen Indices are measured on immature samples.
Source: part based on Powell and Boreham, 1994.

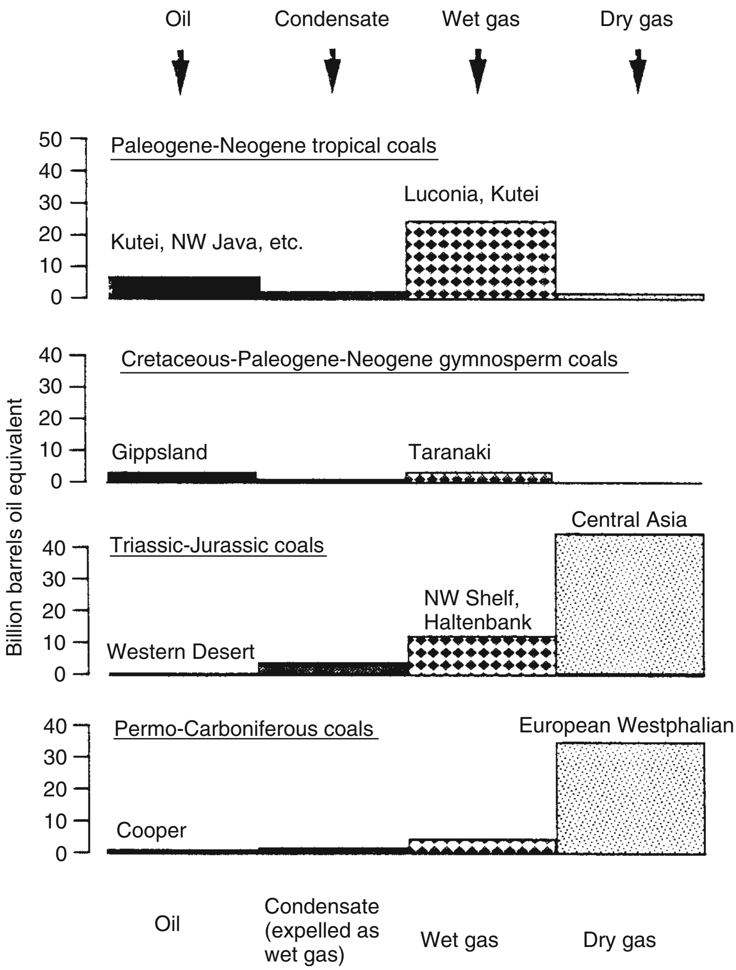



**Figure 11.23** Hydrocarbon reserves tied to coal-bearing source sequences by age. Volumes of different hydrocarbon phases are plotted in billion barrels oil equivalent. Oil-prone coals plot to the left; gas-prone coals plot to the right. In pre-Cretaceous times gas-prone coals predominate, whereas later times see the appearance of significant oil-prone coals. (Partly based on Powell and Boreham, 1994.)

pre-Cretaceous times followed thereafter by oil-prone coals (Macgregor, 1994).

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 characterized by high concentrations of inertinite and low levels of liptinite, a common 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 Hydrogen Index values for coals in the Cooper Basin range from 116–300 mg HC $gC^{-1}$, and is considered to be derived from terrestrial organic matter co-deposited with algal-derived organic material (Curry, Emmett and Hunt, 1994; Powell and Boreham, 1994).

In northwest China, a thick sequence of Jurassic sediments is located in a number of tectonic basins along the flanks of the Tianshan-Qilan Mountains and one such basin is the Turpan Depression, which contains 4500 m of sediments. The sequence is characterized 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 composed primarily of liptinite and vitrinite, classified as Type I and Type II organic matter. The Hydrogen Index has a range of 200–500 mg HC $gC^{-1}$, and the depth of hydrocarbon maturation and expulsion is given as 3000–4000 m (Zhao and Wu, 1997). These hydrocarbons from coal-bearing source rocks are of major importance in China, and large reserves of oil and gas have been identified.

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 is 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 Paleogene–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).

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 unprospective may be reassessed in the future due to the increasing need for hydrocarbon reserves.

# 12 Coal Use and the Environment

## 12.1 Introduction

In the past 40 yr 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 industrialized nation. Developing nations are being asked to conform to environmental standards conceived in the industrialized 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 remains an emotive issue with environmentalists and their political supporters despite the tremendous improvements in mine rehabilitation and coal-fired power station emissions. Historically, the coal industry 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 ensuring prevention of 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, for example high-sulfur coals, are unlikely to be the prime target of mining companies, who know their limitations in the coal sales market.

Greenhouse gas (GHG) emissions from active and abandoned coal mines and coal fired power plants are being reduced by capture and use of methane ($CH_4$). Carbon capture and storage (CCS) in the form of $CO_2$ produced primarily from power generation is being actively investigated with a number of projects operating worldwide.

## 12.2 Coal mining

The increasingly complex regulatory regimes imposed by governments have 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 utilization 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 as well as remaining cost effective and competitive.

Once the results of the exploration phase are known, and that they indicate that a viable mining operation is possible, it is normal practice to prepare a preliminary

*Coal Geology*, Second Edition. Larry Thomas.

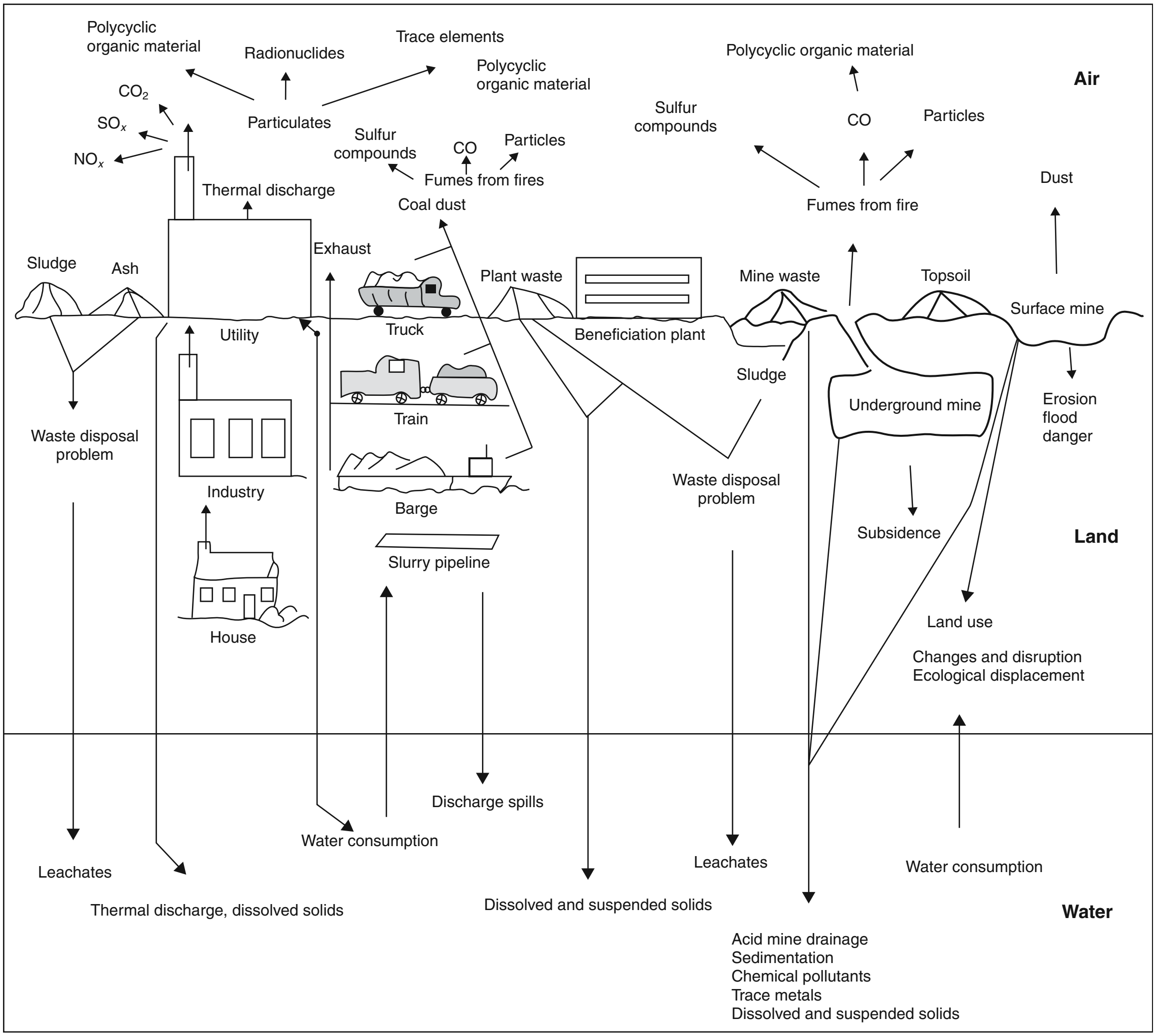



**Figure 12.1** Physical and chemical effects on the environment due to coal mining, transportation and combustion.

environmental report. This report, plus the final details of the coal mine planning, will serve as support documents in the preparation of the official Environmental Impact and Social Assessment (EISA). All potential mine developments require an EISA in order to obtain the necessary legal permits and concessions 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.

### *12.2.1 Effects on water supply*

#### *12.2.1.1 Surface water*

During surface mining operations it is possible that certain drainage divisions 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 where water courses are used for public water supply, that no contamination occurs. Local and regional government 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 (TDS) 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 UCG have shown that contamination of aquifer waters is a major concern and must be safeguarded against.

### *12.2.2 Contamination of mine waters*

One of the most serious effects of underground coal mining 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 industrialized countries such as in western and eastern Europe and the United States exhibited industrial wastelands as a consequence of coal mining. 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.

Acid mine drainage is the principal cause of contaminated water arising from coal mining. It results from the exposure of sulfide minerals, particularly pyrite, to water and oxygen 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 $FeS_2$) are commonly present in coals and coal-bearing sequences, and these are reactive to atmospheric oxygen. 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 characterizes 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 break out 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 oxygen, as in the case of old workings. However, this is greatly reduced in saturated or flooded conditions when the presence of free oxygen is removed. Consequently, once waters flowing through the workings have flushed out all the oxidized 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 United States, acid mine drainage associated with coal mining 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 neutralization of the mine water before discharge. The estimated cost of such chemical

**Table 12.1** Changes in groundwater quality with depth.

| Depth from surface (m) | Concentrations of dissolved compounds (mg $L^{-1}$) | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | $Na^+$ | $Ca^{2+}$ | $Mg^{2+}$ | $Ba^{2+}$ | $Sr^{2+}$ | $Nh_4^+$ | $Mn^{2+}$ | $Cl^-$ | $SO_4^{2+}$ | $HCO_3^-$ |
| 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 |

treatment of mine waters in the United States 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 chloride 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 mineralized with calcium and magnesium salts, and at great depths, concentrations of barium, strontium and ammonium chlorides are characteristic. Saline waters from deep coal mines contain high amounts of chlorides, for example 61, 240 mg $L^{-1}$ $Cl^-$ in waters from deep coal workings in Nottinghamshire, United Kingdom (Downing *et al.*, 1970), together with high amounts of ammoniacal nitrogen. Other contaminated mine waters produced within deeper workings may also contain diffused methane gas. Such saline 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 $L^{-1}$ (Clarke, 1995). In addition, Polish saline waters contain natural radioactive isotopes, mainly $^{226}Ra$ from the uranium series and $^{228}Ra$ from the thorium series. Up to 40% of the total amount of radium remains in the ground but up to 225 megabecquerels (MBq) of $^{226}Ra$ and 400 MBq of $^{228}Ra$ are released daily into rivers along with other mine effluents. To counteract this, technical measures such as induced precipitation in gob areas has 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 past 10 yr (Chalupnik *et al.*, 2001).

Table 12.2 shows some selected analyses of saline and acidic waters from deep coal workings in the United Kingdom. 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, Burke and Gray, 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 coal mines 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 southeast 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 utilized will be recharged back into the groundwater system.

In underground mines hydrogeological understanding 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

**Table 12.2** Selected analyses (in mg $L^{-1}$) of deep waters from Coal Measures strata in the United Kingdom, illustrating compositions of saline brines. Note the acid 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.

| | Eakring 8 | Glentworth 5 | Plungar 4 | Moorgreen Piper Colliery | Pye Hill No. 2 Colliery | Bentinck Colliery |
|---|---|---|---|---|---|---|
| Source | Crawshaw sandstone | Lower coal measures sandstones | Crawshaw sandstone | Pumped water | Pumped water | Initial drainage water |
| $Na^+$ | 8079 | 7005 | 7900 | – | – | – |
| $K^+$ | 96 | 9 | 31 | – | – | – |
| $Ca^{2+}$ | 792 | 1552 | 822 | – | – | – |
| $Mg^{2+}$ | 218 | 192 | 556 | – | – | – |
| $Cl^-$ | 14,555 | 11,786 | 14,910 | 3600–10,800 | 1100–3900 | 31,400 |
| $SO_4^{2-}$ | Nil | 2718 | 342 | – | – | – |
| $HCO_3$ | 73 | 549 | 220 | – | – | – |
| TDS | 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 |

TDS = total dissolved solids.
Source: adapted from Banks, Burke and Gray (1997), reproduced by permission of the Geological Society.

planning of groundwater removal from the mine (see section 9.5) and reduce the potential for groundwater contamination.

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 United States 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 oxygen. 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 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 sites producing most acid. Effective prevention of rainwater inflow is achieved by using asphalt or concrete caps to cover spoil materials.

An interesting development has been the use of bactericides. Bacteria, principally *Thiobacillus ferrooxidans*, catalyse the oxidation of pyrite, greatly increasing the rate of reaction. The use of bactericides in the form of surfactants has proved effective in the treatment of acid mine drainage. Commercially produced, slow-release surfactants are now available in the form of spray or pellets and have an effective life of 2–7 yr (Clarke, 1995), and have been applied to sites in Pennsylvania and Kentucky, United States. The cost of bactericide is offset by savings in reductions in the amount of topsoil required or in the use of other remedial methods.

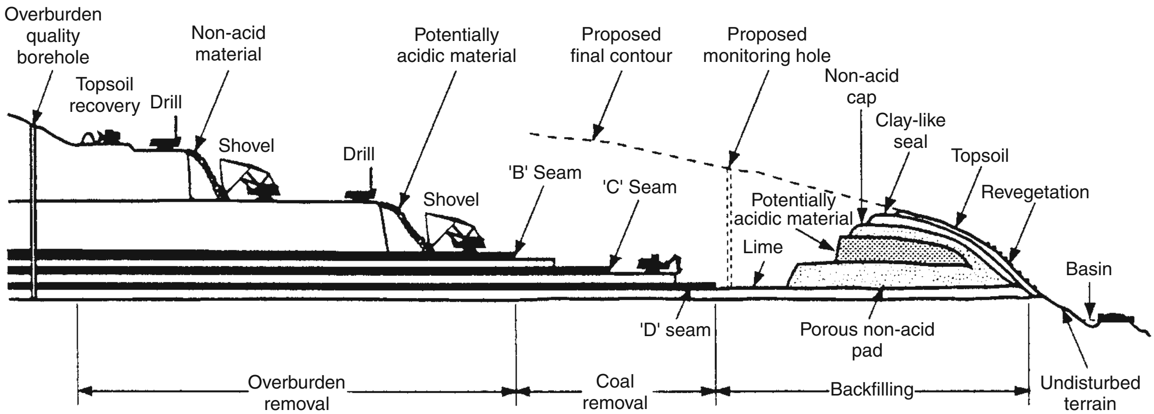



**Figure 12.2** Selective handling and placement of mine spoil to prevent the formation of acid mine drainage at a mine in west Virginia, United States. (From Clarke (1995), reproduced with permission of IEA.)

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 oxygen through their root systems causing aeration of the substrate. Wetland treatment studies are being carried out in Kentucky, United States and in South Wales, United Kingdom (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. Discolouration of the water, although 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 stabilize 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 can still be severely eroded in a single storm. It is therefore recommended 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 mg $L^{-1}$ up to 100,000 mg $L^{-1}$. 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 coal mining. 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 problematical 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 very 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 United Kingdom, at Cilfynydd in South Wales, a large tip fed by an aerial ropeway received up to 500 t $day^{-1}$, and the tip was used for over 50 yr (Bentley, Davis and Gallup, 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, Wright and Hutchinson, 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, which ran downhill for 600 m. These flows released groundwater from a fault zone in the underlying sandstone, which then caused a secondary debris flow (Figure 12.3). In the United Kingdom since the Aberfan disaster, improved tipping practices and rigorous inspections have ensured that no rapid tip failures have occurred 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 United States alone more than $1.5 \times 10^9$ $m^3$ $yr^{-1}$ is moved by dragline, add to this figure that 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 $7.4 \times 16^9$ Bcm of overburden will be moved to mine 1.8 Mt of coal (Figure 12.4). Certain methods of open pit mining ease the problem by back filling 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 topsoil, have adequate drainage and are revegetated with selected local plant species. 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 re-treated or removed, and some are being reworked for their discarded coal content. This not only removes the dump altogether but can yield coal otherwise lost. Tip reclamation operations are well established in the United States, for example in southern Illinois, Indiana and western Kentucky and in the United Kingdom, for example 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

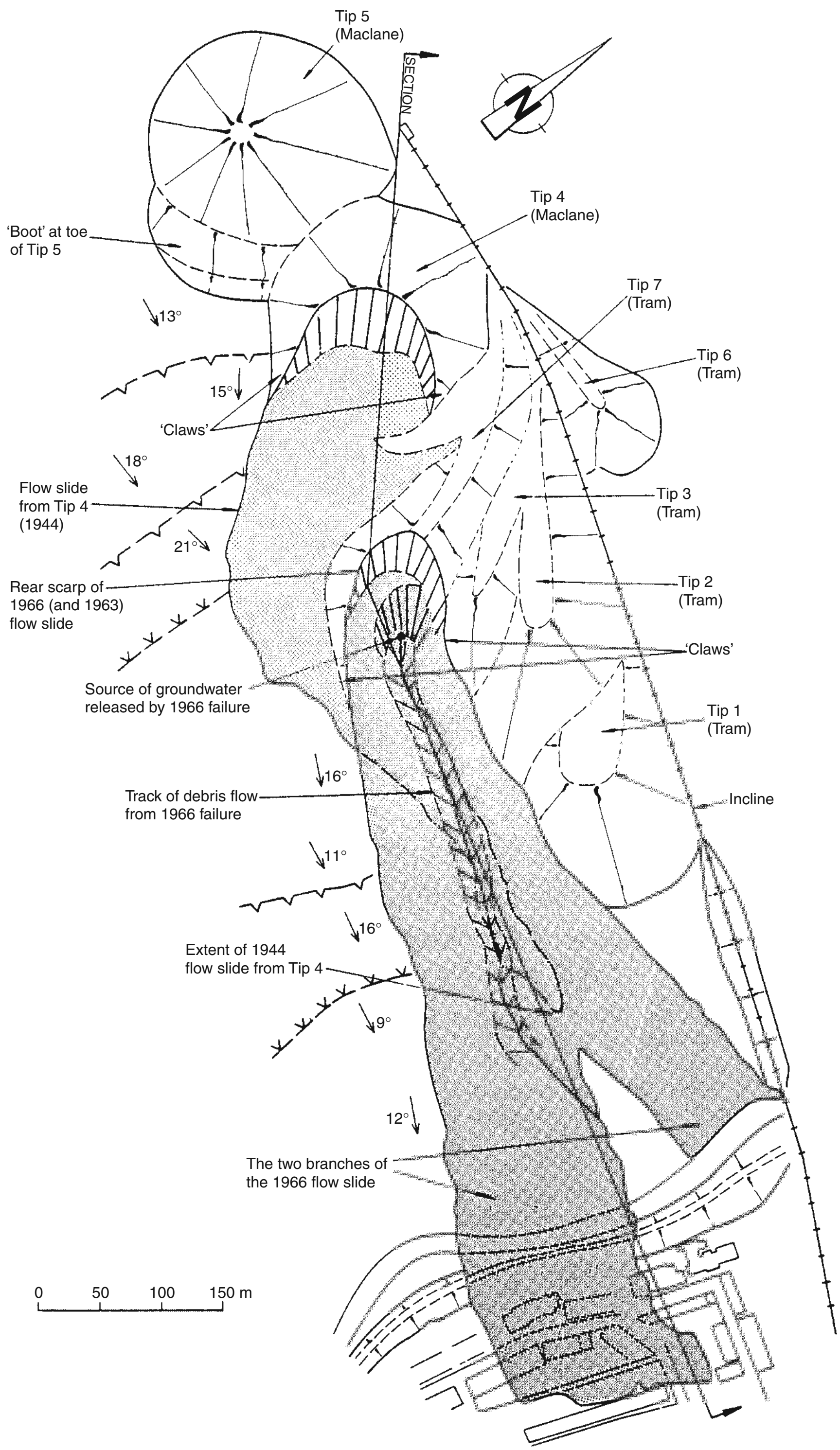



**Figure 12.3** Plan of Merthyr Vale (Aberfan) colliery tip flow slides of 1944, 1963 and 1966. (From Siddle, Wright and Hutchison, 1996.)

**Figure 12.4** Opencast mine in Bosnia-Herzegovina showing large scale overburden removal. Note small scale failures of the high wall caused by surface water action during periods of mining inactivity. (Photograph by courtesy of Dargo Associates Ltd.)

**Figure 12.5** Opencast mine in Maharastra State, India, showing protective bund of overburden to prevent flooding from nearby river. (Photograph by courtesy of Dargo Associates Ltd.)

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 acid mine drainage. 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 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 backfilled 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 to 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.

Oxygen 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 oxygen is small, the reaction is slow with little rise in temperature, but where the quantity of oxygen 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 oxygen determines its propensity to spontaneous combustion. Another contributory factor to spontaneous combustion is the presence of pyrite in the coal, pyrite can be oxidized easily, and on doing so swells and exposes more coal surface to oxygen and therefore assist 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 96 yr. 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 has been lost through fires and a further 1500 Mt isolated from further development (World Coal, 1997). The major problem is how to eliminate the fires, which requires elimination of one or more of the components needed to sustain them, namely, fuel, oxygen and heat. Methods such as excavation (removes fuel), smothering (removes oxygen), quenching (lowers fuel temperature) are currently being applied. Of these, excavation is the only certain means to extinguish the fire, however, in the case of Jharia, this would require $500 \times 10^6$ m$^3$ of material from 60 open pit excavations, which apart from the prohibitive cost would take a very long time. This option is therefore not viable except on a case by case basis (Michalski and Gray, 1997). Although Jharia is an extreme case, underground fires occur elsewhere, for example 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

(a) (b)

**Figure 12.6** (a) Coal burning as a result of ignition by forest fires in Kalimantan, Indonesia. (Photograph by LPT.) (b) Spontaneous combustion of coal in the high wall of an Indian mine. (Photograph by courtesy of Dargo Associates Ltd.)

fires – examples of such fires are shown in Figure 12.6. In the case of stockpiles, remedies are available: (i) avoid having large ranges of coal particle size, which allows a higher rate of oxidation; (ii) 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 that no 'hot-spots' develop, and 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 air line to the coal face are critical. Because of leakage through and 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, which 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, such as 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 damp 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 minimize 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 localized 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 settles or subsides, 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

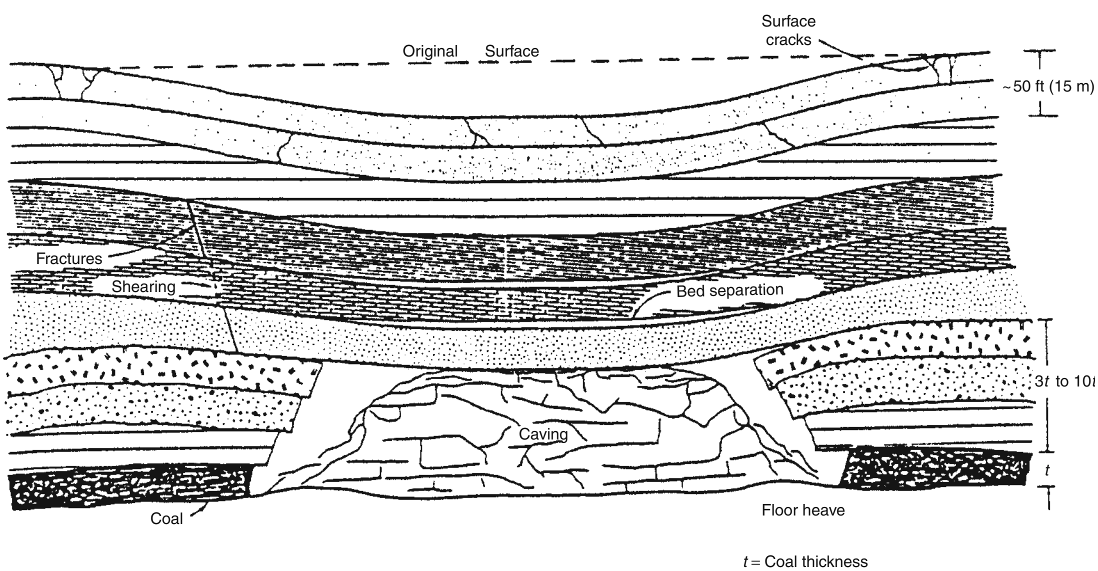



**Figure 12.7** Strata disturbance and subsidence caused by mining. (From Hartman (1992), reproduced with permission of SME.)

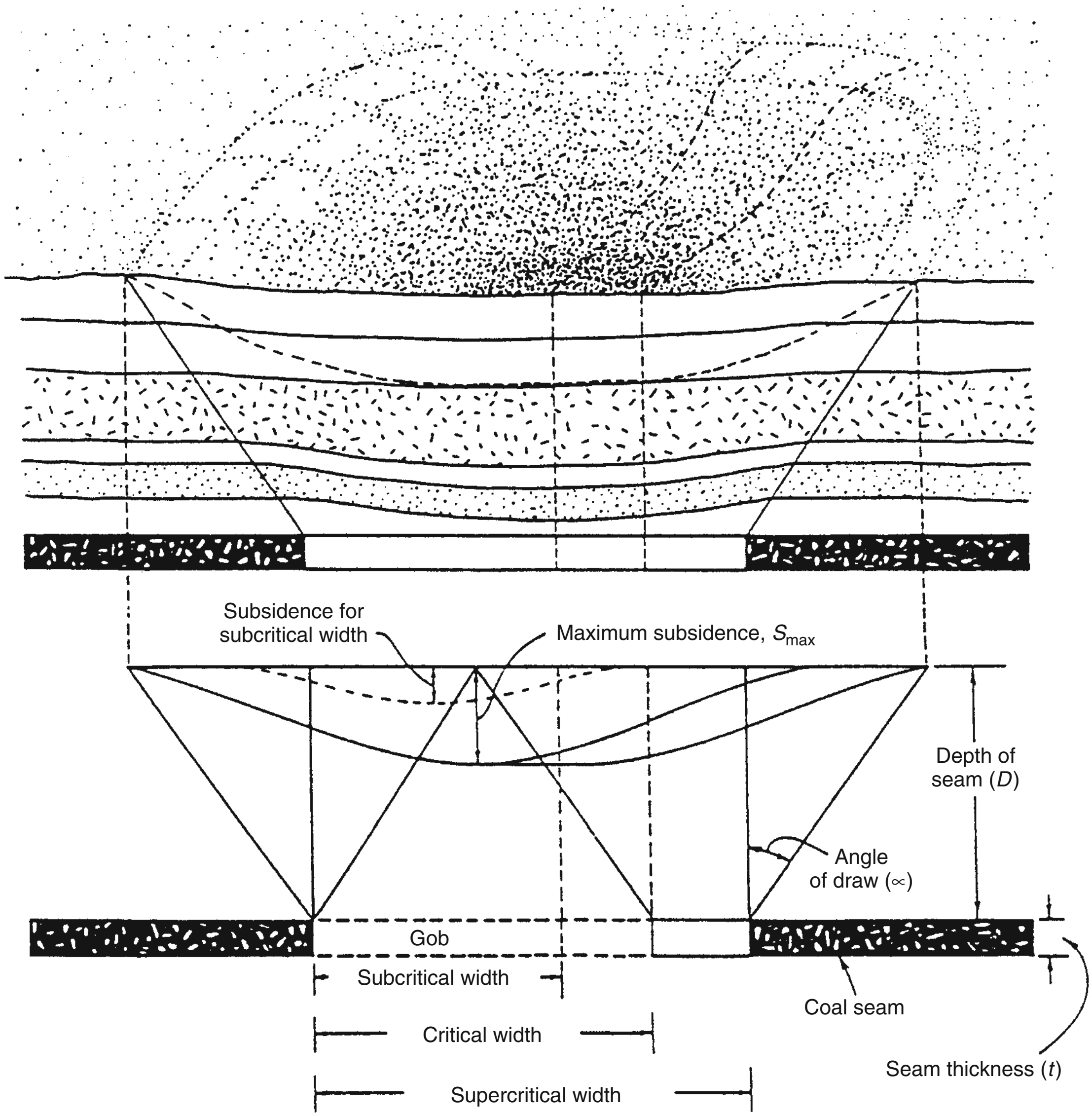



**Figure 12.8** Influence of extraction width on subsidence. (From Hartman (1992), with permission of SME.)

coalfields in China, where in some circumstances they have been flooded and used for fish farming (Figure 12.9). Trough subsidence may be 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 to be not only vertically downward but also horizontal and, in some locations, upward (Singh, 1992).

Old shallow mines have produced small surface subsidence features but modern longwall mining, 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 coal mining has long ceased.

Subsidence can be reduced substantially if large pillars of coal are left in place, or if rock waste from other mining

Figure 12.9 Subsided land overlying underground workings, People's Republic of China, now flooded and used for fish farming. (Photograph by courtesy of Dargo Associates Ltd.)

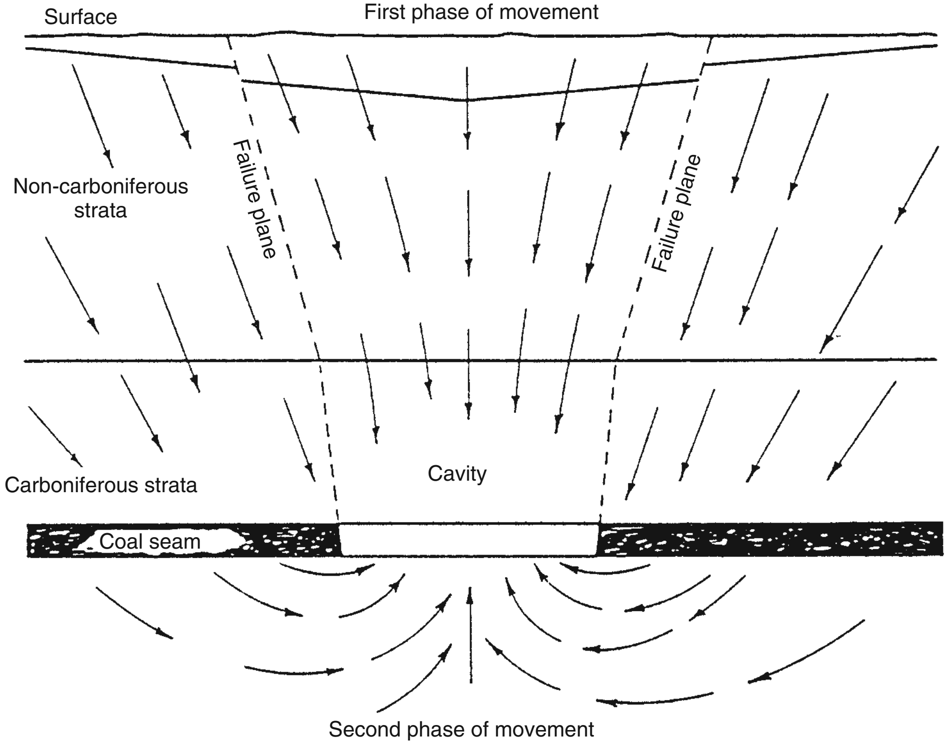



Figure 12.10 Schematic representation of ground movements due to subsidence. (From Hartman (1992), reproduced with permission of SME.)

**Figure 12.11** Collapse of capping material over an old concealed shaft in a modern urban area, United Kingdom. (Photograph by courtesy of IPR/25-36c British Geological Survey. © NERC. All rights reserved.)

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 are 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 United Kingdom 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, United Kingdom.

Mining subsidence and deterioration in the capping of old mine shafts can cause the problem of gas seepage from old workings, particularly methane, which is lighter 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 left between coal pillars. Gas will also migrate upwards in old shafts and if these are inadequately sealed, will reach and escape at the surface. In Figure 12.12, buildings at the surface will be vulnerable to gas invasion. House A is protected by an underlying layer of clay which prevents gas from reaching the surface, whereas house B has no protection. Fatalities have occurred from gas seepage from old mine workings, and as a result, careful investigations should now be made before buildings are erected in old mining areas.

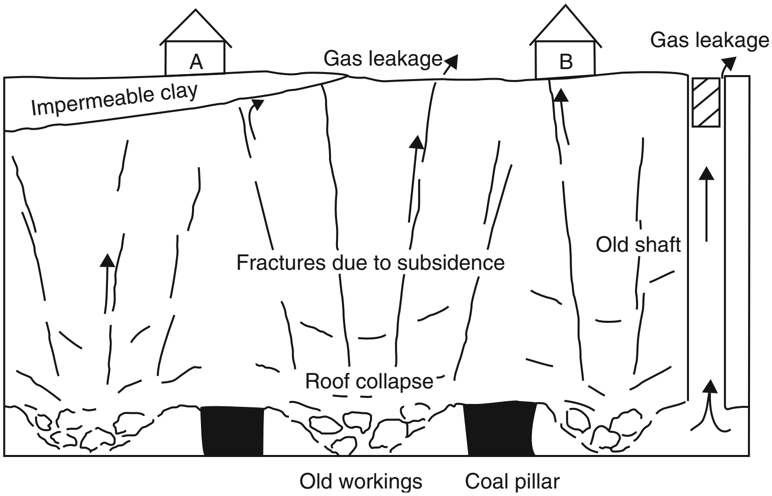



**Figure 12.12** Schematic view of the upward migration of mine gas by means of fractures due to subsidence and from an inadequately sealed shaft. House A is protected by underlying impermeable clay, whereas house B is vulnerable to gas invasion.

## 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 in the iron and steel industry, and in the electricity generation sector. Coal provides around 29% of global primary energy needs and generates about 42% of the world's electricity, generating some 8263 TWh in 2008. Of this figure, the three leading producers of coal fired electricity were People's Republic of China with 2733 TWh, United States with 2133 TWh and India with 569 TWh (IEA, 2011). In addition,13% (717 Mt) of the world's total black coal production is currently utilized by the steel industry, 60% of which is dependent on coal. Recent years have seen an increase in international traded coal, and in particular, the trading of steam coals for electricity generation. In 2000, 381 Mt of steam coal and 192 Mt of coking coal were traded internationally, whereas in 2010, 676 Mt of steam coal and 262 Mt of coking coal were traded internationally (WCA, 2011). By the year 2010, coal-based electricity generation is expected to reach 7400–8000 TWh (WEC, 1998). 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 largest 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), grindability and ash fusion temperature. The performance of different coals in terms of their specifications, 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 utilized in the future. This can have serious repercussions on coal mining. For example, the low sulfur requirements for coals to be used for power generation in the United States has seen a decline in the mining of high sulfur coals in the traditional coalfields in eastern United States, in Illinois, Indiana and western Kentucky,

**Figure 12.13** Modern coal-fired power station, Inner Mongolia, People's Republic of China. (Photograph courtesy of Dargo Associates Ltd.)

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 which are normally used in coal-fired boilers, although coals outside the given ranges can be burned.

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, sulfur and nitrogen oxides and volatile organic compounds. Fly ash removed from the exhaust gases can make up 60–85% of the coal ash residue in

**Table 12.3** Normal range coal specifications for PF-fired boilers.

| Parameter | Range | Comment |
|---|---|---|
| Total moisture | Maximum 15% (ar) | If high, creates handling problems. The limits are higher for lignites and low-rank coals. Reduces net CV |
| Ash | Maximum 20% (ad) | If high, creates fly ash problem. Reduces net CV |
| Volatile matter | Minimum 20–25% (daf) | For conventional PF burners |
| Calorific value (CV) | High | Almost any CV fuel can be used, the higher the better |
| Sulfur | Maximum 0.8–1.0% (ad) | Maximum value dependent on local emission regulations |
| Nitrogen | Maximum 1.5–2.0% (daf) | Various limits apply in some countries because of $NO_x$ emissions |
| Chlorine | Maximum 0.2–0.3% (ad) | Causes ash fouling problems in boiler |
| Hardgrove grindability index (HGI) | Minimum 45–50 | Lower HGI values require larger grinding capacity and more energy |
| Ash fusion temperatures (AFT) | Various | Dry bottom boiler – IDT >1200°C. Wet bottom boiler – FT <1300°C |
| Maximum size | Maximum 40–50 mm | Dependent on capacity of grinding equipment |
| Fines content (<3 mm) | Maximum 25–30% | High fines can increase moisture content and create handling problems |

IDT = initial deformation temperature.
Source: *BP Coal Handbook* (BP Coal Ltd, 1987).

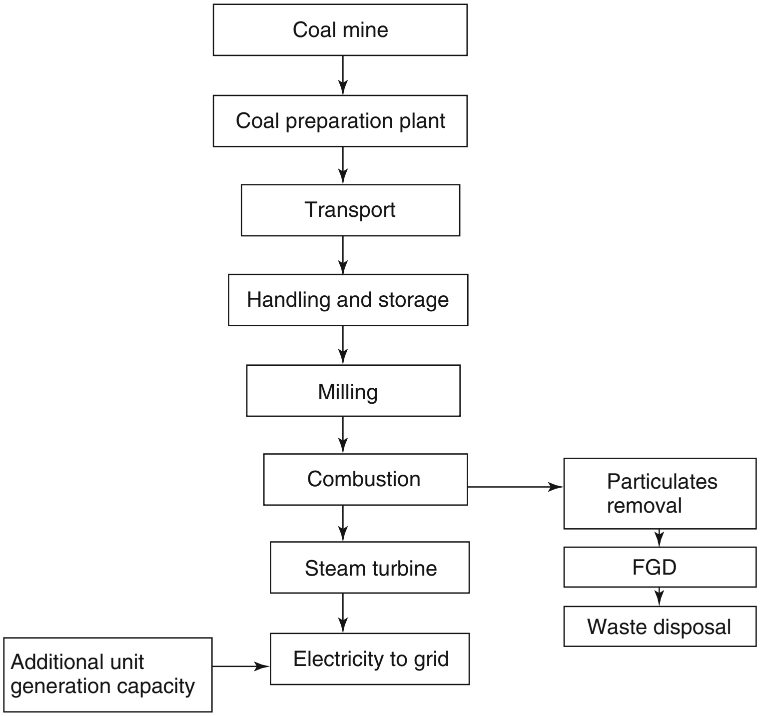



**Figure 12.14** Flow diagram showing stages from coal delivery to electricity generation in a modern power station: FGD = flue gas desulfurization.

pulverized 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 desulfurization (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 demineralization, 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 industrialized 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 particulates less than 10 microns (μm) in size, sulfur dioxide ($SO_2$), on nitrous oxides ($NO_x$), and on fly ash. In addition, carbon dioxide ($CO_2$) and dioxins have also attracted a great deal of attention in deciding on the effects of these pollutants on the atmosphere and global ecology.

To estimate the amount of $SO_2$ produced daily from the power station stack, it is necessary to know the calorific value, hydrogen and moisture content of the coal and the station heat rate. For example, a 500 MWe plant using coal with 2.5% sulfur, 16% ash, and a calorific value of 30 MJ $kg^{-1}$ (12,898 Btu $lb^{-1}$) will emit: 200 t $SO_2$, 70 t $NO_2$, 500 t fly ash, 500 t solid waste, and have 17 gigawatt-hours (GWh) of thermal discharge (World Bank Group, 1998).

$SO_2$ is one of the principal gaseous pollutants, which can be a hazard to human health and damage both the natural and man-made environments. Total global emissions of $SO_2$ from the combustion of coal, oil and oil-derived fuels, refining and smelting amounts to 140 Mt $yr^{-1}$. To control the emissions of $SO_2$ 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 ($FeS_2$), which can be removed in a coal preparation plant and thereby effectively reducing sulfur content in traded coals, as for example 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 $SO_2$ emissions is to remove the $SO_2$ from the flue gases before they are released to the atmosphere (see section 12.3.1.2).

$NO_x$ emissions are produced by the reaction of the nitrogenous compounds in the coal with oxygen. It is considered that nitrogen is released during devolatilization and enters the gas phase as HCN or $NH_3$, where it reacts with air to form $NO_x$ in a complex series of chemical reactions. High temperatures and rapid heating rates maximize the yield of the volatile nitrogen species and the presence of free oxygen favours the formation of $NO_x$. Among the $NO_x$ emissions from power plants is nitrous oxide ($N_2O$), which has a significant effect on the atmosphere as it is a strong absorber of infrared radiation and is considered as a major contributor (20%) to ozone depletion.

Carbon dioxide ($CO_2$) is emitted as a product of coal combustion, and relates to the carbon content of the coal burned at the time. Old and inefficient plant allow higher $CO_2$ emissions than do modern installations.

Dioxins is the general name given to a group of some 210 species consisting of 75 polychlorinated dibenzo-para-dioxins (PCDDs) and 135 polychlorinated dibenzofurans (PCDFs). 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 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. There is little published data on dioxin emissions from coal, and current testing in the United Kingdom has so far recorded only 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, United States it has been demonstrated that concentrations of lead, thallium, antimony, cadmium, selenium, arsenic, zinc, nickel, chromium and sulfur were markedly increased in the size range 0.65–74 μm.

**Table 12.4** Distribution of elements among bottom ash, fly ash and flue gas.

| 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 |

Source: Valkovic (1983).

In Australia, a threefold increase in concentrations of gallium, germanium, mercury and lead have been observed between the coarse (>50 μm) and fine (<2 μm) fly ash fractions. Table 12.4 shows the distribution of elements between the main coal residues in a power plant, that is bottom ash, fly ash and flue gas, taken from an example in the United States. 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 during the current decade. For example, where permitted particulate emissions levels were 50 mg $Nm^{-3}$ (normal cubic metre), they were reduced to 10 mg $Nm^{-3}$ after the year 2000, which meant that the electrostatic precipitators were required to collect up to 99% of particulates from flue gas.

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 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, the fly ash then 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, for example 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 ASTM have 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 $SiO_2$+ $Al_2O_3$+ $Fe_2O_3$; Class C fly ashes are derived from lignite and subbituminous coals and should contain a minimum of 50% of $SiO_2$+ $Al_2O_3$+ $Fe_2O_3$. Both classes of fly ash should possess their own distinctive sets of chemical, physical and engineering properties. This classification is used as the basis for selecting fly ashes as admixtures in cement and concrete.

### *12.3.1.2 Flue gas desulfurization (FGD)*

The control of $SO_2$ emissions has centred on FGD technologies (DTI, 2000a). These are now widely used to control the emissions of sulfur dioxide ($SO_2$) and sulfur trioxide ($SO_3$). Varieties of FGD processes are available, most use an alkali sorbent to recover the acidic sulfur compounds from the flue gas. The alkaline materials most commonly used 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 $SO_2$ 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, that is the process selected will be the one with the lowest overall through-life cost. Technical considerations include the degree of desulfurization 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, and has evolved over 50 yr. 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 $SO_2$. Flue gas desulfurization plant manufacturers claim that over 95% of $SO_2$ can be removed with the absorber. This process also removes almost all of any hydrogen chloride (HCl) in the flue gas.

The calcium carbonate ($CaCO_3$) from the limestone reacts with the $SO_2$ and oxygen ($O_2$ from air) to produce gypsum (hydrated calcium sulfate $CaSO_4.2H_2O$) which precipitates from solution in the sump. HCl is also dissolved in the water and neutralized 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 Wellman–Lord process (using aqueous sodium sulfate solution). Semi-dry processes are the circulating fluidized 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 $SO_2$. The spent sorbent is extracted with the fly ash as a mixture of ash and calcium or sodium compounds.

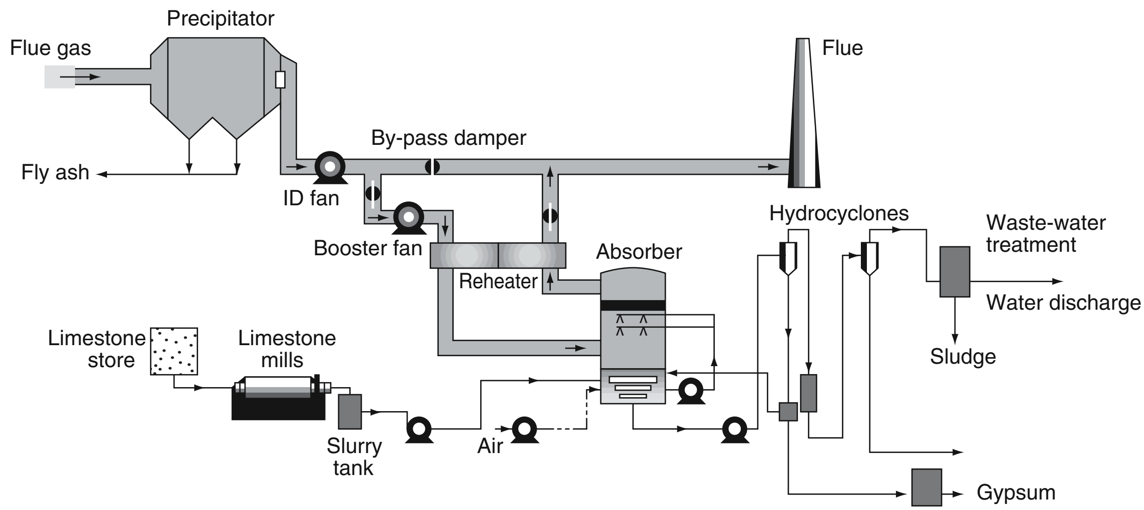



**Figure 12.15** Schematic flow diagram of a limestone/gypsum flue gas desulfurization process. Reproduced by permission of DTI Clean Coal Programme.

### *12.3.1.3 Other emission controls*

The reduction of $NO_x$ emissions requires the fitting of $NO_x$ reduction equipment to both existing plant (retrofitting) and to new plant. Conventional pulverized fuel (pf) burners are designed to achieve rapid intimate mixing of the fuel with the combustion air. Since the fuel devolatilizes in high temperatures, and air-rich conditions, the level of $NO_x$ is high. To counteract this low-$NO_x$ burners are used based on the physical separation of three air streams to the burner, the addition of each air stream helping to reduce $NO_x$ production. The use of low-$NO_x$ 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 $NO_x$ reduction cost effective as well as environmentally desirable.

A post-combustion $NO_x$ 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 NO and $NO_2$ in the gas to form molecular nitrogen and water. The SCR systems can be costly and difficult to retrofit, due to their physical size, but are widely installed in Japan and Germany.

Several combined $SO_2$ and $NO_x$ removal systems have been developed, one such system is the SNOX process developed in Denmark. The flue gas is reheated, then it undergoes SCR, it is then further heated and a second catalytic reactor oxidizes $SO_2$ to $SO_3$. The gas is then cooled to condense out the $SO_3$ as sulfuric acid. The condenser uses glass tubes to prevent excessive acid corrosion. SNOX units have been built in Denmark, Italy and the United States.

Reduction in $CO_2$ emissions at pf plants has been mainly by chemical absorption processes and current research is seeking to improve on these methods. $CO_2$ emissions have been linked to reduced efficiency levels in coal-fired plants, particularly in non-OECD countries. This is due to a number of factors: poor quality coal, small and ageing units, poor maintenance and obsolete technologies.

### *12.3.1.4 Fluidized bed combustion*

In the electricity generating industry, fluidized 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, namely bubbling FBC (BFBC) and circulating FBC (CFBC), both of which can be either atmospheric or pressurized 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 fluidization. Increases in the gas flow above the minimum will produce bubbles, the upwards and sideways coalescing movements of which 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 $SO_2$, the bed particles need to be controlled in the temperature range of 800–900°C. Over the past 40 yr, BFBC technology has been shown to be well suited to the utilization 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, which provides a balance of the combustion air to reduce $NO_x$ levels.

Circulating FBCs 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 fluidized 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 stabilizes the overall temperature. Again additional combustion air is introduced at higher levels to reduce $NO_x$ levels.

Many CFBC units proved capable of achieving lower levels of the primary pollutants, $NO_x$, $SO_2$, $CO_2$ and particulates. Sorbent is added to the system to control $SO_2$ emissions, $NO_x$ levels are minimized by careful bed temperature control and particulate control systems are installed. Pressurized FBC (PFBC) 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 China. Circulating FBC plants are dominant in Asia, again principally in China with 52% of installed capacity; North America has 26% and Europe has 22%. Pressurized FBC is now expanding into the United States and Europe, with the highest level of current activity in Japan (DTI, 2000b).

### *12.3.2 Other major users*

#### *12.3.2.1 Iron and steel production*

The potential source of pollution in the iron and steel making process relating to coal is in the production of coke. Other emissions are part of the steel making process and are not considered here.

In 2010, world crude steel production was 1414 Mt, an increase of 15% compared with 2009. The top steel producers in 2010 were the People's Republic of China with 627 Mt, Japan with 110 Mt and the United States with 80 Mt. Between 2000 and 2010, the People's Republic of China increased its use of steel by 400%. Almost 70% of global steel is produced in basic oxygen furnaces (BOF), which require 770 kg of coal to produce 1 t of steel. A further 29% of steel is produced in electric arc furnaces (EAF) and for this process, about 150 kg of coal is required to produce 1 t of steel (WCA, 2011).

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 carbonized, usually taking 3–4 days. This method is still widespread in China (Figure 12.16), where coke making is still a cottage industry. This method has been replaced by carbonizing a thin vertical layer of coal in less than a day, and 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 $CO_2$ emissions.

Reductions in $CO_2$ 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 pulverized fuel and heavy fuel oil, and by introducing larger blast furnace units to gain from economies of scale (WCI, 2001).

In the United Kingdom, in Yorkshire, the Monckton Cokeworks produces 535 t of coke and 276,000 $m^3$ of coke oven gas per day from 790 t of coal, with an annual coke production of 200,000 t. A new 11 MWe combined

**Figure 12.16** Local coke manufacture in small 'ovens', Guizhou Province, People's Republic of China. (Photograph courtesy of Dargo Associates Ltd.)

heat and power plant (CHP) has been developed as part of an environmental clean-up and redevelopment programme. Of the coal gas produced from the coke ovens, 40% is reused to heat the coke ovens, and 60%is diverted to the new CHP plant. The CHP plant utilizes the coke oven waste gas to produce $31\,t\,h^{-1}$ of superheated steam, this in turn produces 11 MW, with 1.4 MW required for on-site demand and the remainder sold via the local grid. This facility is able to generate more added value from the surplus high calorific value 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 EAF plants. The energy consumption of basic oxygen steel making is very low compared with other production methods. In addition, it is possible to transform this method into a net supplier of energy by utilizing the waste converter gases. The energy input and associated level of $CO_2$ emissions per tonne of steel produced by the international steel making industry could be reduced by over 25% if the status of the production technology used in Europe and the United States was introduced on a global scale. Currently the $CO_2$ emissions per tonne of steel are three times higher in China (3.9 t $CO_2$) compared with western Europe (1.3 t $CO_2$). Apart from utilizing 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 $CO_2$ emissions is a financial one, with the capital expenditure necessary to achieve such reductions in the CIS and China being estimated at $20 billion and $42 billion respectively (WCI, 2001).

#### *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 utilize 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 calorific value. 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 has led to the replacement of coal by gas and oil heating. The thick smogs of large cities are 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 CIS, China and eastern Europe coal is plentiful, 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.

1. Road transport: coal is moved from the mine to the customer by lorry fleets. This means using public roads, which can cause problems such as wear and tear, traffic congestion and dust from coal loads. In countries such as China 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, for example in East Kalimantan, Indonesia, private coal roads transport coal to the loading areas on the major rivers or ports.
2. 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 minimized.
3. 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.
4. Water transport: coal transported by barge or ocean-going vessels has only a dust problem on loading/unloading, but some coals have the propensity for spontaneous combustion (see section 12.2.6).

## 12.4 Health

Worldwide coal mining remains a growth industry, and coal mining, 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, that is 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, coal mining, 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. Pneumoconiosis and silicosis were the legacies of the coal miner together with newer complaints such as vibration white finger.

Coal miners 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 thermo-regulatory system tries to regulate the body temperature at 37°C, but when working in hot and humid conditions this control is not maintained, the body temperature starts to rise and produces various physiological effects such as heat rash, fainting, heat exhaustion, 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 dispel 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. Although 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 coal mines 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 coal mining is the release of naturally occurring toxic organic compounds into the environment. In the former Yugoslavia, researchers believe that a relationship may exist between organic compounds leached by groundwater from shallow lignite deposits and a disease known as Balkan endemic nephropathy. This disease, recognized since 1956, is a progressive kidney disease that leads to death. The disease occurs in villages situated on alluvial deposits overlying Pliocene lignites. The lignites contain large amounts of organic compounds some of which may be water soluble. Current research is studying the drinking water sources of each village, the hydrological regime and the water and lignite chemistry, to determine whether there is a definite link between the disease and the water supply (Finkelman, Feder and Orem, 1991).

## 12.5 Carbon capture and storage (CCS)

Addressing the challenge of climate change, while still meeting the increasing need for affordable energy, will require the implementation of energy efficient and low carbon technologies. Capturing $CO_2$ 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.

Carbon capture and storage is not a new technology, as 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 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). Carbon capture and storage is not eligible for the Kyoto Protocol's Clean Development Mechanism (CDM), which enables clean energy technology to be transferred to developing countries. This limits such countries to contribute to emissions reductions and reduces the global effort to reduce emissions.

The coal industry is committed to minimizing 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 with the intended effect of all the measures included in the Kyoto Protocol of climate change of 5% (WCI, 2010).

The construction of integrated gasification combined cycle (IGCC) power plants is more costly than conventional coal-fired plants, but they are more efficient and produce less pollution. The process facilitates carbon capture and environmental controls. Integrated gasification combined cycle power plants of 300 MW capacity have been built in the United States and Europe, and larger 600 MW units are now being constructed in the United States. In Spain, the 335 MW plant at Puertollano was commissioned in 1998, and 450 MW IGCC plants have been planned for South Wales and Killingholme in the United Kingdom, also a 450 MW IGCC plant with $CO_2$ separation and storage is planned for construction in Germany (Crook, 2006).

Carbon capture and storage projects designed to store large amounts of $CO_2$ are looking for significant capacity in subsurface formations both on land and under the sea floor to sequester $CO_2$ for hundreds if not thousands of years. Capturing the $CO_2$ 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 while maintaining the growth of global energy demands.

Greenhouse gases such as $CH_4$ can and are being recovered from active and abandoned coal mines (section 11.2.1.3). Although $CO_2$ injection either into deep unmineable coal seams or used for enhanced coalbed methane extraction will provide a minor contribution to world $CO_2$ storage capacity, it may be a significant storage option for certain countries. It is likely that only deep disused coal mines or deep unmineable coal deposits would be considered as suitable for CCS. The use of coal seams as storage for $CO_2$ raises a number of issues as to their suitability. The low level of permeability of coals, their ability to swell and the sterilization 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, which may represent leakage risks if $CO_2$ is injected.

In Europe, injection into coal seams may provide a technical option for $CO_2$ 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 freshwater groundwater zone ($<500$ m). The potential contamination of the groundwater system would need to be assessed (EC, 2009). Countries such as the United States have established a prevention programme to protect the underground sources of drinking water (USDW) from potential endangerment caused by $CO_2$ injection wells (Jackson, 2010). The storage of $CO_2$ in abandoned mine workings also has the risk of leakage through fissures and fractures produced during and after mining. Only deep mines with an overlying sequence containing a reliable seal could be considered, thus ensuring that groundwater will not be affected.

Carbon capture and storage technologies could represent a significant percentage of the cumulative effort for reducing $CO_2$ emissions worldwide. Around 32 Mt of $CO_2$ is already stored globally and this is increasing (WCI, 2010). The Sleipner Project in the Norwegian North Sea is the world's longest running $CO_2$ storage operation. Injection commenced in 1996 and has stored ca. 12 Mt of $CO_2$ in the Utsira Sand, a large saline aquifer 900 m below the sea bed. The project is intensely monitored using geophysics, whereby three-dimensional seismic surveys have been undertaken every 2yr in a comprehensive time-lapse monitoring programme known as four-dimensional seismic surveys (Chadwick, 2011). Other large-scale $CO_2$ storage projects include the In-Salah Gas Project in Algeria, where it is estimated that 17 Mt of $CO_2$ will be stored, and Weyburn in Saskatchewan, Canada will store 20 Mt of $CO_2$ from a coal gasification plant in North Dakota, United States. In China, coal-fired power generation is rapidly increasing, and it is now the largest $CO_2$ emitter. Studies are therefore under way to produce an inventory of space available in old or depleting oil fields, naturally occurring underground brine and unmineable coal deposits containing significant percentages of sandstone (Stephenson, 2009).

It is clear that CCS will continue to be an issue particularly affecting coal generated energy and the efforts necessary to both control emissions and to identify suitable secure repositories for $CO_2$.

## 12.6 Environmental regulations

### *12.6.1 Introduction*

The effects of mining and utilizing 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 industrialized countries. In recent years, prominence has been given to the harmful effects of 'greenhouse gases' on the Earth's atmosphere, these gases comprise carbon dioxide, methane, nitrous oxide, hydrofluorocarbons, perfluorocarbons and sulfur hexafluoride. In addition, the emission of sulfur dioxide and particulate matter 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, for example Australia. In some countries, the responsibility for emission standards is divided between the State Ministry and local government, for example Italy and South Korea. Others have emission standards that are the sole responsibility of Central Government, as in the case of the People's Republic of China, France, Germany and the United Kingdom. The United States involves both Federal and State Governments in the regulation of air pollution. Most industrialized countries follow one of these models for air quality legislation. (See relevant literature for details of individual country emission limits, e.g. McConville, 1997.)

Emissions into the atmosphere may traverse individual country boundaries, and because of the international nature of air pollution has led to a number of initiatives to control trans-boundary pollution. Agreements between countries have been made, for example between the United States and Canada, the United Nations Economic Commission for Europe Conventions (UNECE), the European Community (EC) environmental legislation, the Kyoto Protocol, the Bali Action Plan (BAP) and the Copenhagen Accord. In addition, the World Bank environmental guidelines are regularly implemented when projects seek international finance.

### *12.6.2 UNECE Convention*

The UNECE Convention of Long Range Trans-Boundary Air Pollution (LRTAP) was signed in 1979 by 33 countries including the United States and Canada. Now 42 countries

**Table 12.5** UNECE (1994) $SO_2$ emission standards for new coal-fired plants.

| New plant type | Plant size (MWt) | Emission standard* |
|---|---|---|
| Combustion plants | 50–100 | 2000 mg $m^{-3}$ |
| Combustion plants | 100–500 | 2400 – (4 × $P$) mg $m^{-3}$ |
| Combustion plants | >500 | 400 mg $m^{-3}$ |
| Combustion plants; domestic high-sulfur coal | 100–167 | 800 mg $m^{-3}$ or 40% $SO_2$ removal |
| Combustion plants; domestic high-sulfur coal | 167–500 | 800 mg $m^{-3}$ or 15 + (0.15 × $P$)% removal |
| Combustion plants; domestic high-sulfur coal | >500 | 800 mg $m^{-3}$ or 90% removal |

$P$ = plant size in MWt.
*Emission standards figures are given in mg $m^{-3}$ on dry flue gas at 6% $O_2$ and standard temperature and pressure (0°C (273 K), 101.3 kPa).
Source: McConville (1997).

are party to the Convention. The Convention outlines the responsibility of governments to minimize trans-boundary air pollution and 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. A Second Sulfur Protocol was signed in 1994 by 27 European countries, the EC and Canada. The emission standards for $SO_2$ are given in Table 12.5. The Second Sulfur 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 comes into force only after it has been ratified by 16 countries; by 1997, only eight countries had ratified the Protocol (McConville, 1997).

The Protocol on nitrogen oxides ($NO_x$), the 'Sofia Protocol', was signed in 1988 and came into force in 1991, signed by 23 countries and ratified by 16. The Protocol requires that emissions of $NO_x$ be frozen at 1987 levels by the end of 1994 and then maintained.

A Protocol on the control of volatile organic compounds (VOC) emissions was signed in 1991 and came into force in 1997. Volatile organic compounds are defined as all organic compounds of anthropogenic nature, other than methane, that are capable of producing photochemical oxidants by reactions with $NO_x$ in the presence of sunlight (McConville, 1997).

Protocols on persistent organic pollutants and heavy metals are under preparation.

### 12.6.3 European Community

The European Community (EC) puts forward legislation to be adopted as formal proposals by the Commissioners. European Community legislation is presented either as regulations that are binding on member states or as directives that give targets to be achieved and deadlines in which to implement them, but how this is done is left to the member states. Environmental legislation is in the form of directives allowing member states flexibility in trying to achieve environmental objectives.

The Directive on Controlling of Emissions from Large Combustion Plant (LCPD) 1988 sets out emission standards for particulates, $SO_2$ and $NO_x$ (see Table 12.6). The Integrated Pollution Prevention and Control Directive (IPPC) 1996 requires the introduction of an integrated environmental licensing system applied to a range of industrial processes including power plants larger than 50 MWe. This is to be applied to all new and existing plant by 2007, and member states are to implement their own schedules in order to achieve this target.

The EC has since drafted additional directives, one is the European union Acidification Strategy which proposes new limits for $SO_2$ and $NO_x$, and a second is the revision of the LCPD with much more stricter emission ceilings for $SO_2$ and $NO_x$ to be achieved by 2010 (McConville, 1997).

### 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

**Table 12.6** European Community emission standards (1988) for new coal-fired plants.

| Emission type | Plant type | Plant size (MWt) | Emission standard* |
|---|---|---|---|
| $SO_2$ | Combustion plants | 50–100 | 2000 mg $m^{-3}$ |
| $SO_2$ | Combustion plants | 100–500 | $2400 - (4 \times P)$ mg $m^{-3}$ |
| $SO_2$ | Combustion plants | >500 | 400 mg $m^{-3}$ |
| $SO_2$ | Combustion plants Operating <2200 hours per year | >400 | 800 mg $m^{-3}$ |
| $SO_2$ | Combustion plants; domestic high or variable sulfur coal | 100–166 | 40% removal |
| $SO_2$ | Combustion plants; domestic high or variable sulfur coal | 167–500 | $15 + (0.15 \times P)$% removal |
| $SO_2$ | Combustion plants; domestic high or variable sulfur coal | >500 | 90% removal |
| $NO_x$ | Combustion plants | >50 | 650 mg $m^{-3}$ |
| $NO_x$ | Combustion plants; coal volatiles <10% | >50 | 1300 mg $m^{-3}$ |
| Particulates | Combustion plants | 50–500 | 100 mg $m^{-3}$ |
| Particulates | Combustion plants | >500 | 50 mg $m^{-3}$ |

$P$ = plant size in MWt.
*Emission standards figures are given in mg $m^{-3}$ on dry flue gas at 6% $O_2$ and standard temperature and pressure (0°C (273 K), 101.3 kPa).
Source: McConville, 1997.

Bank's environmental policy is Operational Directive OD4.00, 1989, updated in 1991 as OD 4.01. Guidelines for emissions from thermal plants were issued in 1998 (Table 12.7). These establish maximum emission levels for all fossil fuel based thermal power plants with a capacity of 50 MWe or larger. The guidelines focus on emissions of particulates less than 10 μm in size, $SO_2$ and $NO_x$.

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: (i) the specific requirements for the power plant itself, focusing on issues to be addressed in arriving at project-specific emission standards; and (ii) 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 (World Bank Group, 1998).

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 environmental 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 stabilize emissions of GHGs at 1990 levels by the year 2000. Following this, an agreement to cut emissions of greenhouse gases 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 greenhouse gases by 5.2% from 1990 levels by the period 2008–2012. The Kyoto Protocol was endorsed by 160 countries, but will only be legally binding if at least 55 countries sign up to it, including developed nations responsible for at least 55% of GHG emissions from the industrialized world. The cut in emissions is to be achieved by differential reductions for individual countries: the European Union, Switzerland and the majority of central and eastern European nations will deliver reductions of 8%; the United States 7%; Japan, Hungary, Canada and Poland 6%; New Zealand, Russia and the Ukraine are required to stabilize their emissions; while Australia, Iceland and Norway are permitted to increase slightly, although at a reduced rate to current trends.

**Table 12.7** World Bank proposed emission standards.

| Emission type | Plant type | Plant size (MWt) | Emission standards* |
|---|---|---|---|
| $SO_2$ | New power plants | All | 2000 mg m$^{-3}$ maximum emission level of 0.2 t day$^{-1}$ per MWe of capacity up to 500 MWe plus 0.1 t day$^{-1}$ for each additional MWe of capacity over 500 MWe |
| $SO_2$ | Old power plants | All | Emission levels to meet regional load targets |
| $NO_x$ | New power plants | All | 750 mg m$^{-3}$ |
| $NO_x$ | New power plants; coal volatiles <10% | All | 1500 mg m$^{-3}$ |
| $NO_x$ | Old power plants | All | Emission levels as recommended for new plants or at least a 25% reduction in baseline level |
| Particulates | New power plants | All | 50 mg m$^{-3}$ if not possible, then must achieve 99.9% removal |
| Particulates | Old power plants | All | 100 mg m$^{-3}$ but target should be 50 mg m$^{-3}$ |

*Emission standards figures are given in mg m$^{-3}$ on dry flue gas at 6% $O_2$ and standard temperature and pressure (0°C (273 K), 101.3 kPa).
Source: World Bank Group (1998).

It was considered that the majority of nations would sign up to the Kyoto Protocol, but by June 1998 only 41 countries had signed, and there are a number of nations, some of which contribute a large percentage of the GHG emissions, who still have not committed to this reduction, for example the United States.

### *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 United Nations Framework Convention on Climate Change (UNFCCC), and the 5th 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 2 yr negotiating process was agreed, referred to as the Bali Action Plan (BAP), also known as the Bali Roadmap. The Copenhagen Accord is a document that the delegates present agreed to 'take note of'. The Accord is not legally binding and does not commit countries to agree to a binding successor to the Kyoto Protocol, whose present round ends in 2012.

The Accord reinforces the need for emissions reductions together with providing financial assistance to help developing countries cut carbon emissions. It was recognized 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 has come 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 (UNEP) 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, that is an emission level of 44 Gt. Without the Accord emissions reductions this figure might reach 50 Gt 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.

## 12.7 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 coal mining, 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 water 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 industrialized nations, is likely to be a long protracted affair. The underlying reason being 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 coal mining practices and the advances in clean-coal technology, together with the development of alternative uses of coal such as coal-bed methane 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 the environmental lobby will continue to have a profound effect on the feasibility and cost of coal mining and utilization.

# 13 Coal Marketing

## 13.1 Introduction

Although this book is concerned primarily 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:

1. the quality and quantity of the product;
2. the transportation of the product;
3. the contractual terms of the sale and purchase of the product;
4. 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 specialized chemical processes are also commercial outlets for coal. Statistics for coal equivalence and coal usage are given in Appendix 4. Coals are marketed locally, for example 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 by ocean-going vessels to overseas markets.

## 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 (CV) and the properties of coal that affect this; in particular, 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 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

*Coal Geology*, Second Edition. Larry Thomas.

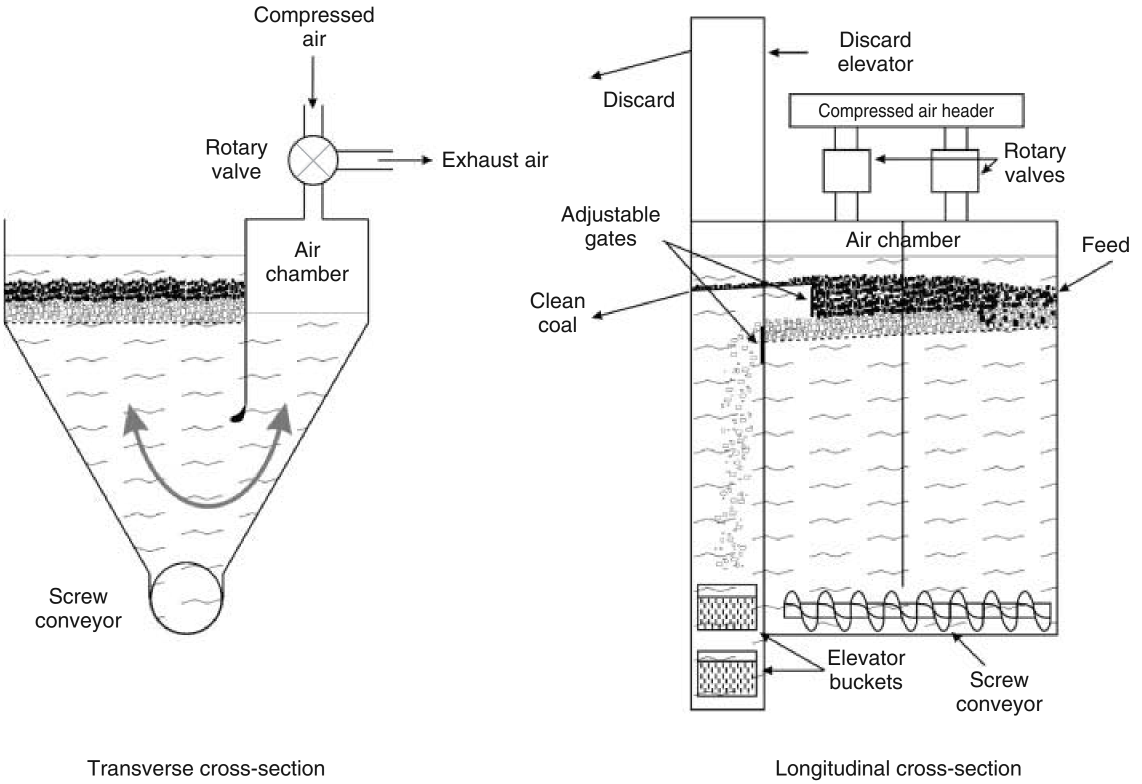



**Figure 13.1** Section througha Baum type jig. Coal is fed into the jig and the water level rises and falls by means of injection of compressed air (see transverse section). The up and down motion of the water effects a separation (see longitudinal section). (Reproduced with permission of Dargo Associates Ltd.)

main process is using 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 float-and-sink analysis.

For fine coal (<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 a coal receival facility.

Many modern mines deliver coal to their dispatch area where it is conveyed to storage silos. Each silo may contain a specific coal quality that 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 which 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 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 stockpiled for a short time only.

**Figure 13.2** Coal silos for loading coal directly to trains, People's Republic of China. (Photograph by courtesy of Dargo Associates Ltd.)

**Figure 13.3** Modern coal stockyard at port of Qinhuangdao, People's Republic of China. (Photograph by courtesy of Dargo Associates Ltd.)

## 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.

**Figure 13.4** Overland conveyor, 14 km in length, from Kaltim Prima Coal mine to the port of Tanjung Bara, East Kalimantan, Indonesia. (Photograph by courtesy of Dargo Associates Ltd.)

### *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 Coal mine to the port at Tanjung Bara (Figure 13.4). A single span steel-cored conventional belt conveyor carries 1100 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 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 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 utilization and reduce costs, so loading from bins (Figure 13.6) or silos is the most effective for dedicated transport routes. Road transport may be used to haul coal to terminals on either rivers, for example in the United States 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 $yr^{-1}$, for example 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


**Figure 13.5** Coal transportation by small capacity 10 t truck, India. (Photograph by courtesy of Dargo Associates Ltd.)

**Figure 13.6** Automatic loading of trucks from overhead bins, Orissa State, India. (Photograph by courtesy of Dargo Associates Ltd.)

to Dalrymple Bay in Queensland, Australia, and in the People's Republic of China 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 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.

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).

**Figure 13.7** Automatic loading of trains, each wagon receives exact tonnage as train moves slowly through loading bay, Alberta, Canada. (From Howland, 1998.)

**Figure 13.8** Train being loaded by payloader, Orissa State, India. (Photograph by courtesy of Dargo Associates Ltd.)

Upon delivery, the coal has to be unloaded, the method of unloading is governed by the type of wagons used, that is 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), as 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, 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 United States. In the United States such train units carry up to 12000 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 tippled to empty them. Trains can be equipped with rotary couplings that permit wagons to be tippled without uncoupling. More commonly, wagons must be uncoupled and tippled separately then recoupled. Using rotary couplings, a 60 t wagon train can be unloaded in 1 h, whereas uncoupling would require 3 h 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, for example the Grand Canal in People's Republic of China, which runs from Shanghai to Beijing. In the United Kingdom canals have been utilized for over 200 yr, 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, in the United States 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 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.

**Figure 13.9** Bottom discharge wagons (60 t), on Eastern Railways, India. (Photograph by courtesy of Dargo Associates Ltd.)

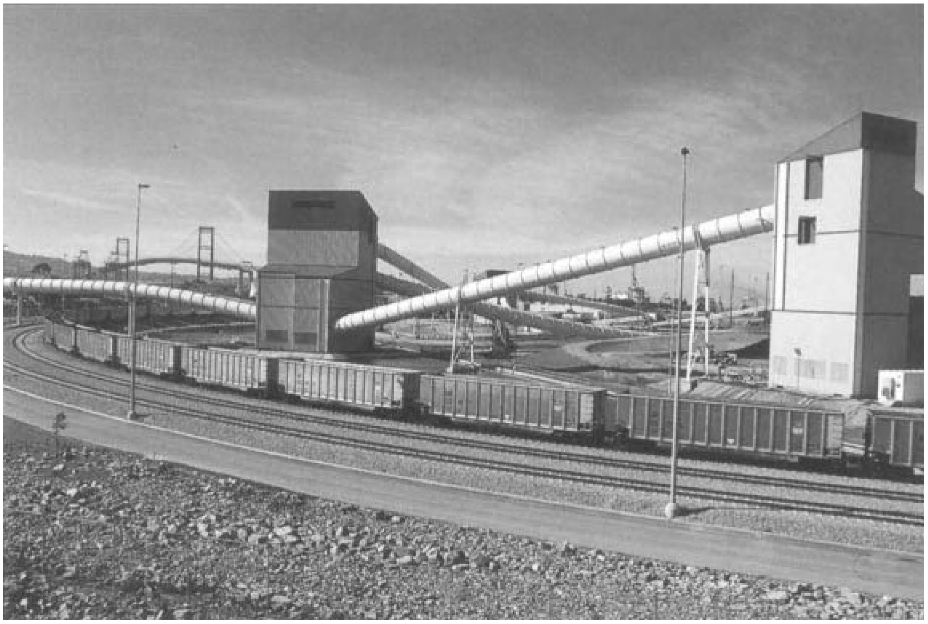


**Figure 13.10** Large train units transporting 12,000 t over 1200 km in western United States. (From Harder, 1998.)

**Figure 13.11** Top discharge wagons with drop down doors for side unloading. Guizhou Province, People's Republic of China. (Photograph by courtesy of Dargo Associates Ltd.)

**Figure 13.12** Coal barges carrying up to 10,000 t, Mississippi River, United States. (With permission of *World Coal*, Palladian Publications.)

### *13.3.2.2 Bulk carriers*

Ocean bulk carriers account for over 500 Mt $yr^{-1}$ of seaborne trade in coal, and this continues to grow. The smallest ships are 10,000 dwt, for example those used between United Kingdom 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, and Cape size vessels are +100,000 dwt, with 200,000 dwt being the largest. The larger the bulk carrier, and 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 coal terminals of the world can load 10,000 t $h^{-1}$, for example Richards Bay in South Africa and Dalrymple Bay, Australia, and Qinhuangdao in the People's Republic of China (Figure 4.13). The throughput capacities of some of the leading coal handling ports are listed in Table 13.1 (The Tex Report, 2010). New port facilities have recently been completed such as at Los Angeles Export Terminal (LAXT), which is capable of handling 7700 t $h^{-1}$, the largest export facility on the United States Pacific Coast.

The unloading of coal is done either with 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 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 Cape size vessels.

Coals are transported around the world, and Figure 13.14 shows the long-haul sea routes taken by Cape size and Panamax vessels. Smaller Handy size vessels are usually used for shorter journeys or for ports that cannot take larger vessels, for example along the east coast of India. The international hard (or black) coal trade totalled 938 Mt in 2010, with 676 Mt being thermal

**Figure 13.13** Coal loading directly into the ship from the stockyard conveyor system, port of Qinhuangdao, People's Republic of China. (From Thomas and Frankland, 1999.)

**Table 13.1** Throughput of major exporting ports.

| Port | Throughput ($Mt\,yr^{-1}$) |
|---|---|
| Dalrymple Bay (Australia) | 85 |
| Gladstone (Australia) | 75 |
| Hay Point (Australia) | 44 |
| Abbott Point | 25 |
| Roberts Bank (Canada) | 22 |
| Qinhuangdao (People's Republic of China) | 75 |
| Puerto Bolivar (Colombia) | 15 |
| Tanjung Bara (Indonesia) | 32 |
| Balikpapan (Indonesia) | 12 |
| Gdansk (Poland) | 9 |
| Vostochnyy (Russia) | 12 |
| Muchka (Russia) | 5 |
| Richards Bay (South Africa) | 76 |
| Baltimore (United States) | 9 |
| Mobile (United States) | 23 |
| Hampton Roads (United States) | 33 |
| Norfolk (United States) | 48 |
| LAXT (United States) | 57 |

Source: The Tex Report, 2010.

or steam coal and 262 Mt being coking coal. Of this the seaborne trade comprises 629 Mt steam coal and 217 Mt coking coal (WCA, 2011).

Table 13.2 shows the black coal tonnage traded by the major coal exporting and importing countries (WCA, 2011). Coal is the major fuel used for generating electricity worldwide. Table 13.3 shows countries heavily dependent on coal for electricity generation (WCA, 2011). Gross coal-fired electricity generation in the European Union in 2007 was 988 MWh or 29% 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.

## 13.4 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.

### *13.4.1 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 he/she has 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.

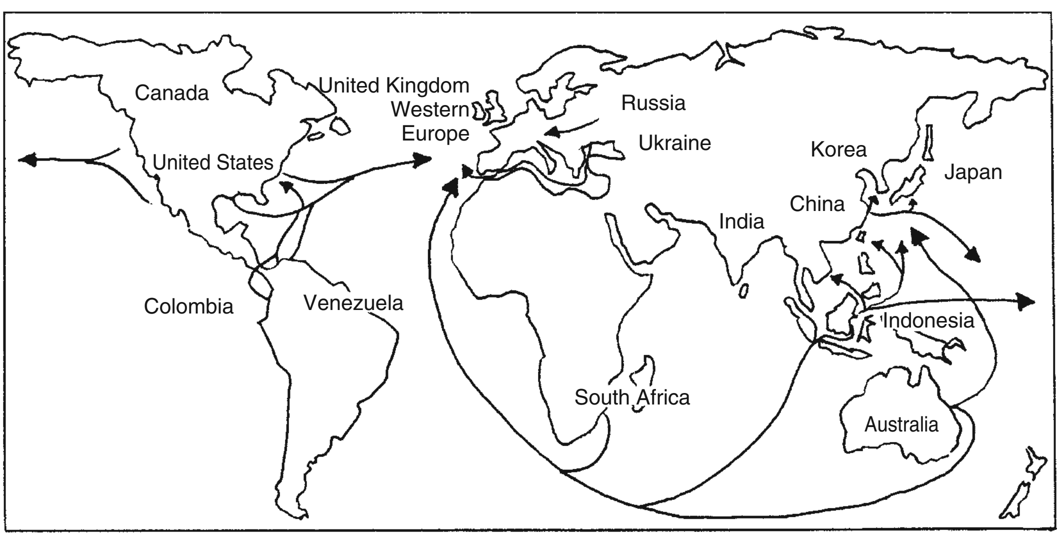



**Figure 13.14** Principal coal export routes to markets in Western Europe and the Far East.

Table 13.2 Major coal exporting and importing countries.

| Coal exporters | Steam coal (Mt yr$^{-1}$) | Coking coal (Mt yr$^{-1}$) | Coal importers | Steam coal (Mt yr$^{-1}$) | Coking coal (Mt yr$^{-1}$) |
|---|---|---|---|---|---|
| Australia | 143 | 155 | People's Republic of China | 129 | 48 |
| South Africa | 68 | 2 | Japan | 129 | 58 |
| People's Republic of China | 24 | 8 | Republic of Korea | 1 | 28 |
| Indonesia | 160 | 2 | Taiwan | 58 | 5 |
| Colombia | 67 | 1 | United Kingdom | 20 | 6 |
| Russia | 95 | 14 | India | 60 | 30 |
| United States | 23 | 51 | Germany | 38 | 8 |
| Canada | 4 | 27 | Turkey | | |
| | 20 | 7 | | | |

Source: WCA (2011).

Table 13.3 Principal users of coal for electricity generation.

| Country | % |
|---|---|
| South Africa | 93 |
| Poland | 90 |
| People's Republic of China | 79 |
| Australia | 76 |
| Kazakhstan | 70 |
| India | 69 |
| Israel | 63 |
| Czech Republic | 56 |
| Morocco | 55 |
| Greece | 55 |
| United States | 45 |
| Germany | 44 |

Source: WCA (2009).

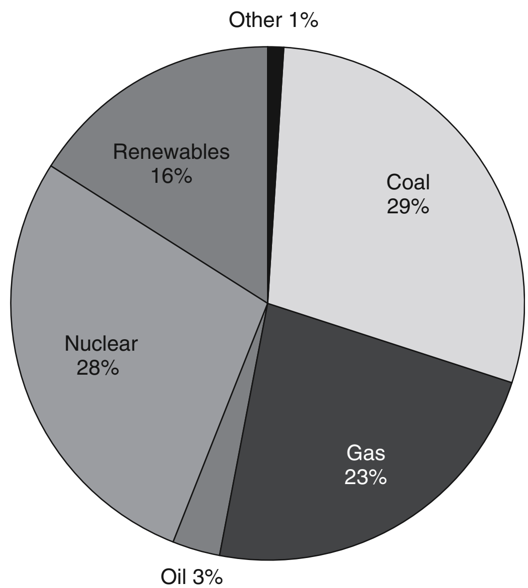



Figure 13.15 Gross electricity production in the European Union 2007. (From EU, 2010.)

### 13.4.2 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 1 yr at which time the tonnage and price of coal to be purchased for the next year is renegotiated, 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 United States.

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, but in general most clauses in international coal contracts include coal type and quality (which has a significant effect on the planned use of the coal, for example 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 minimize risk to either party, such as changes in law, taxes and regulations, arbitration, *force majeur* 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.

### 13.4.3 Indexed contracts

Indexed contracts are usually long-term contracts such as between a coal-fired electricity generating station and a single or group of coal 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 is determined by the specification of the boilers in the power station and is fixed for the duration of the contract, which can be 20 yr. The contract will contain similar clauses to those for a term contract, but 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 supply 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, for example 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 (HGI) and ash fusion temperature (AFT). In the case of coal imports to a power station, the coal supplier(s) normally will 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 organization, that is a suitable thermal coal price index.

## 13.5 Coal price and indexing

The basic commercial property of thermal coal is its net calorific value (NCV) and this parameter together with other properties which may affect commercial and environmental considerations will determine the price paid for the coal. For spot cargoes this is a straightforward 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:

$$P_{\mathrm{CS}} = P_{\mathrm{B}} \times \frac{CV_{\mathrm{CS}}}{CV_{\mathrm{B}}}$$

where $P_{\mathrm{CS}}$ = the price paid for coal deliveries, $P_{\mathrm{B}}$ = base price for coal, $CV_{\mathrm{CS}}$ = NCV of coal delivered and $CV_{\mathrm{B}}$ = 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 (GCV), 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 (C and F) 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 (DES) all freight and risk for loss are with the seller, and 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 be either 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, which are compared to historical rates in the form of a coal price index.

Indexed contracts may include price adjustments based on one, 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, and 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 McCloskey Coal Report (MCIS) publishes spot prices for steam coal collected from co-operating contacts, both suppliers and buyers, and these prices are adjusted to 6000 kcal $kg^{-1}$ net as received basis. The index is available on a monthly basis and accurately illustrates trends in pricing for the European spot market and for the Asian coal market. This index is not considered suitable for pricing long-term contracts. The South African Coal Report (SACR) index was launched in 1998. This index has been back dated to a price in January 1986 and has been limited to a particular type of coal and a particular market. The prices used in the index are actual shipments to the European power generation market, which consumes over 24 Mt $yr^{-1}$ of South African steam coal. The SACR reflects quickly and accurately spot market trends. Colombia and Tai Power also publish indexes, which are of limited use outside their geographical regions.

Other indices are the average price of cargoes already sold. Any indexation of prices to these indexes requires a retrospective adjustment in price. Current indices include the European Union based on the quarterly average CIF prices for coal imported from outside the European Union 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 6 six months after the quarter indexed. This means that all prices for cargoes must be adjusted retrospectively, since price changes will occur each quarter.

A comparison of indexes (Figure 13.16) shows that none of the curves correspond exactly, but that they do indicate similar trends. The volatility of the various indices is a measure of the periodicity, that is monthly indices vary more than indices set annually, and spot prices vary more than term contracts. Current prices are now showing an upward trend after a decline in the late 1990s.

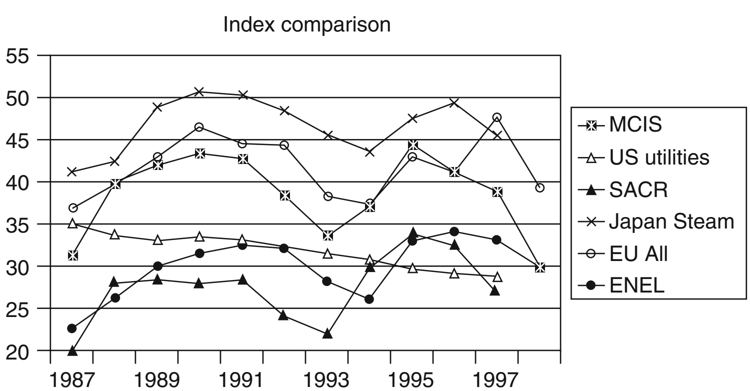



**Figure 13.16** Comparison of selected coal indexes over a 10 yr period. MCIS = McCloskey Coal Report; SACR = South African Coal Report; ENEL = Ente Nazionale per l'Energie eLettrica. (Reproduced by permission of Dargo Associates Ltd.)

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 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.

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# APPENDIX 1 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 United Kingdom/Europe, United States, Australia, China, India and Russia. Other countries 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.

## British Standards Institution (BS)

Breckland, Linford Wood, Milton Keynes MK14 6LE.

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.

| | |
|---|---|
| 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. |
| 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. |

*Coal Geology*, Second Edition. Larry Thomas.

| | |
|---|---|
| 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. |
| 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.

## International Organization For Standardization (ISO)

Casa Postale 56, CH 1211, Geneve 20, Switzerland.
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 334-1975 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 351-1984 | 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 501-1981 | Coal – determination of the crucible swelling number. |
| ISO 502-1982 | Coal – determination of caking power, Gray–King coke test. |
| ISO 540-1995 | Solid mineral fuels – determination of fusibility of ash, high temperature tube method. |
| ISO 561-1989 | Coal preparation plant – graphical symbols. |
| ISO 562-1998 | Hard coal and coke – determination of volatile matter content. |
| ISO 567-1995 | Determination of bulk density of coke (small container). |
| ISO 579-1999 | Determination of total moisture of coke. |
| ISO 587-1997 | Solid mineral fuels – determination of chlorine using Eschka method. |
| ISO 589-1981 | 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 602-1983 | 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 622-1981 | 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 647-1974 | 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 test, expression and presentation of results. |
| 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). |
| ISO 1015-1975 | Brown coals and lignites – determination of moisture content, direct volumetric method. |
| ISO 1017-1985 | Brown coals and lignites – determination of acetone-soluble material (resinous substances) in the toluene-soluble extract. |
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| ISO 1170-1977 | Coal and coke – calculation of analyses to different bases. |
| ISO 1171-1997 | 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 1928-1976 | Solid mineral fuels – determination of gross calorific value by the calorimeter bomb method and calculation of net calorific value. |

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| ISO 5074-1994 | Hard coal – determination of Hardgrove grindability index. |
| ISO 7404.4-1994 | 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 7404.7-1988 | Method of determining microlithotype, carbominerite and minerite composition. |
| ISO 7404.5-2009 | Method of determining microscopically the reflectance of vitrinite. |
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| ISO 8833-1989 | Magnetite for use in coal preparation – test methods. |
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| ISO 10753-1994 | Coal preparation plant – assessment of liability to breakdown in water of minerals associated with coal seams. |
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| ISO 11760-2005 | Classification of coals. |
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| ISO 13909.2-200 | Hard coal and coke – mechanical sampling – Coal: sampling from moving streams. |
| ISO 13909.3-200 | Hard coal and coke – mechanical sampling – Coal: sampling from stationary lots. |
| ISO 13909.4-200 | Hard coal and coke – mechanical sampling – Coal: preparation of test samples. |
| ISO 13909.5-200 | Hard coal and coke – mechanical sampling – Coke: sampling from moving streams. |
| ISO 13909.6-200 | Hard coal and coke – mechanical sampling – Coke: sampling from stationary lots. |
| ISO13909.7-200 | Hard coal and coke – mechanical sampling – Methods for determining the precision of sampling, sample preparation and testing. |

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| ISO13909.8-200 | Hard coal and coke – mechanical sampling – Methods of testing for bias. |
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100 Barr Harbor Drive, Conshohocken, Pennsylvania, United States.

Figures after Standard number give the year of most recent re-approval.

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| D 197-2002 | Sampling and fineness test of pulverised coal. |
| D 293-2004 | Sieve analysis of coke. |
| D 291-2002 | Cubic foot weight of crushed bituminous coal. |
| D 346-2004 | Collection and preparation of coke samples for laboratory analysis. |
| D 388-1999 | Classification of coals by rank. |
| D 409/D409M-2009 | Grindability of coal by the Hardgrove-machine method. |
| D 440-2002 | Drop shatter test for coal. |
| D 441-2002 | Tumbler test for coal. |
| D 720-2004 | Free swelling index of coal. |
| D1412-2004 | Equilibrium moisture of coal at 96% to 97% relative humidity and 30 degrees C. |
| D1756-2002 | Carbon dioxide in coal. |
| D1757-2002 | Sulfur in ash from coal and coke. |
| D1857-2003 | Fusibility of coal and coke ash. |
| D2013-2007 | Samples, coal, preparing for analysis. |
| D2014-1997 | Expansion or contraction of coal by the sole-heated oven. |
| D2234/D2234M-2009 | Collection of a gross sample of coal. |
| D2361-1995 | Chlorine in coal. |
| D2492-2002 | Forms of sulfur in coal. |
| D2639-2007 | Plastic properties of coal by the constant-torque Gieseler plastometer. |
| D2797-2007 | Preparing coal samples for microscopical analysis by reflected light. |
| D2798-2009 | Microscopical determination of the reflectance of the organic components in a polished specimen of coal. |
| D2799-2009 | Microscopical determination of the maceral composition of coal. |
| D2961-2002 | Total moisture, <15% in coal reduced to No.8 (2.36 mm) topsize. |
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| D3172-2007 | Proximate analysis of coal and coke. |
| D3173-2002 | Moisture in the analysis sample of coal and coke. |
| D3174-2002 | Ash in the analysis sample of coal and coke from coal. |
| D3175-2002 | Volatile matter in the analysis sample of coal and coke. |
| D3176-2002 | 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. |
| D3180-2002 | Calculating coal and coke analyses from As-determined to different bases. |
| D3302/D3302M-2009 | Total moisture in coal. |
| D3402-1993 | Tumbler test for coke. |
| D3682-2001 | Major and minor elements in coal and coke ash by atomic absorption. |
| D3683-2000 | Trace elements in coal and coke ash by atomic absorption. |
| D3684-2000 | Total mercury in coal by the oxygen bomb combustion/atomic absorption method. |

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| D3761-2002 | 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. |
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| D4239-2002 | Sulfur in the analysis sample of coal and coke using high temperature tube furnace combustion methods. |
| D4326-2001 | Major and minor elements in coal by X-ray fluorescence. |
| D4371-1998 | Washability characteristics of coal. |
| D4596-2008 | Collection of channel samples of coal in the mine. |
| D4606-2003 | 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. |
| D4702-2004 | Guide for quality management of mechanical sampling systems. |
| D4749-2002 | Sieve analysis for coal, performing and designating coal size. |
| D4916-1997 | Practice for mechanical auger sampling. |
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| D5061-2005 | Microscopical determination of volume % of textural components in metallurgical coke. |
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| D5263-2008 | Method for determining the relative degree of oxidation in bituminous coal by alkali extraction. |
| D5341-1999 | Method for measuring coke reactivity index (CRI) and coke strength after reaction (CSR). |
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| D5865-2004 | Standard test method for gross calorific value of coal and coke. |
| D5987-1996 | 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-1999 | Method for determination of bulk density coal using nuclear backscatter depth density methods. |
| D6349-2007 | 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. |

D6357-2000 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.

D6518-2006 Practice for bias testing a mechanical coal sampling system.

D6542-2005 Practice for tonnage calculation of coal in a stockpile.

D6543-2000 Guide to the evaluation of measurements made by on-line coal analyzers.

D6609-2007 Guide for part-stream sampling of coal.

D6610-2001 Standard practice for manual sampling coal from surfaces of stockpiles.

D6883-2003 Standard practice for manual sampling of stationary coal from railroad cars, barges, trucks or stockpiles.

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D7430-2010 Standard practice for mechanical sampling of coal.

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## Standards Association of Australia (AS)

80–86 Arthur Street, North Sydney, NSW, 2060, Australia.

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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.

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AS 1038.1-2001 Total moisture in hard coal.

AS 1038.2-2006 Total moisture in coke.

AS 1038.3-2000 Proximate analysis of higher rank coal.

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AS 1038.5-1998 Gross specific energy of coal and coke.

AS 1038.5.1-1988 Adiabatic calorimeters.

AS 1038.5.2-1989 Automatic isothermal-type calorimeters.

AS 1038.6.1-1997 Determination of carbon and hydrogen.

AS 1038.6.2-2007 Determination of nitrogen.

AS 1038.6.3.1-1997 Determination of total sulfur (Eschka method).

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| AS 1038.10.4-2001 | Determination of trace elements – Coal, coke and fly ash – determination of fluorine content by pyrohydrolysis method. |
| AS 1038.10.5.1-2003 | Determination of trace elements in coal, coke and fly-ash, determination of mercury content by tube combustion method. |
| AS 1038.10.5.2-2003 | Determination of trace elements in coal, coke and fly ash, determination of mercury content by acid extraction method. |
| AS 1038.11-2002 | Forms of sulfur in coal. |
| AS 1038.12.1-2002 | Determination of crucible swelling number of coal. |
| AS 1038.12.2-1999 | Carbonization properties of higher rank coal, determination of Gray–King coke type. |
| AS 1038.12.3-2002 | Determination of the dilatometer characteristics of higher rank coal. |
| AS 1038.12.4.1-1996 | Plastic properties of higher rank coal by the Gieseler plastometer. |
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| AS 1038.14.1-2003 | Analysis of coal ash, coke ash and mineral matter (borate fusion-flame atomic absorption spectrometric method). |
| AS 1038.14.2-2003 | Analysis of higher rank coal ash and coke ash (acid digestion-flame atomic absorption spectrometric method). |
| AS 1038.14.3-1999 | Analysis of higher rank coal ash and coke ash (wavelength dispersive X-ray fluorescence spectrometric method). |
| AS 1038.15-1995 | Fusibility of higher rank coal ash and coke ash. |
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| AS 1038.18-2006 | Coke – size analysis. |
| AS 1038.19-2000 | Determination of the abrasion index of higher rank coal. |
| AS 1038.20-2002 | Hardgrove grindability index of higher rank coal. |
| AS 1038.21.1.1-2008 | Determination of the relative density of hard coal and coke, analysis sample – density bottle method. |
| AS 1038.21.1.2-2002 | Determination of the relative density of hard coal and coke, analysis sample – volumetric method. |
| AS 1038.22-2000 | Direct determination of mineral matter and water of hydration of minerals in hard coal. |
| AS 1038.23-2002 | Determination of carbonate carbon in higher rank coal. |
| AS 1038.24-1998 | Guide to the evaluation of measurements made by on-line coal analyzers. |
| AS 1038.25-2002 | Durham cone handleability test. |
| AS 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. |
| AS 2434.1-2002 | Determination of the total moisture content of lower rank coal. |
| AS 2434.2-2002 | Determination of the volatile matter in low rank coal. |
| AS 2434.3-2002 | Determination of the moisture-holding capacity of lower rank coals. |
| AS 2434.4-2002 | Determination of the apparent density of dried lower rank coal and its chars (mercury displacement method). |
| AS 2434.5-2002 | Determination of moisture in bulk samples and in analysis samples of char from lower rank coal. |
| AS 2434.6.1-2002 | Ultimate analysis of lower rank coal. |

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| AS 2434.7-2002 | Determination of moisture in the analysis sample of lower rank coal. |
| AS 2434.8-2002 | Determination of ash in the analysis sample of lower rank coal. |
| AS 2434.9-2000 | Determination of four acid-extractable inorganic ions in lower rank coal. |
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| AS 2856.2-1998 | Maceral analysis. |
| AS 2856.3-2000 | Microscopical determination of reflectance of coal macerals. |
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| AS 3980-1999 | 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. |
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| AS 4156.3-2008 | Coal preparation of higher rank coal, magnetite for coal preparation plant use – test methods. |
| AS 4156.4-1999 | Coal preparation – flowsheets and symbols. |
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| AS 4156.7-1999 | Coal preparation – coal size classifying equipment – performance evaluation. |
| AS 4156.8-2007 | Coal preparation – drop shatter test. |
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Figures after standard give the year of most recent re-approval.

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| GB/T 211-1996 | 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 212-2001 | Determination of inherent moisture in coal. |
| GB/T 212-2001 | Determination of ash content in coal. |
| GB/T 212-2001 | Determination of volatile matter in coal. |
| GB/T 213-2003 | Determination of calorific value in coal. |
| GB/T 214-1996 | Determination of total sulfur in coal. |
| GB/T 216-2003 | Determination of phosphorus in coal. |
| GB/T 217-1996 | Determination of true relative density of coal. |
| GB/T 218-1996 | Determination of carbon dioxide content in mineral carbonate associated with coal. |
| GB/T 219-1996 | Determination of ash fusion temperature in coal. |
| GB/T 397-1998 | Technical condition of coal for metallurgical coke. |

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| GB/T 474-1996 | Preparation of coal samples. |
| GB/T 475-1996 | Sampling for commercial coal. |
| GB/T 479-2000 | Determination of plastometric indices of bituminous coals. |
| GB/T 481-1993 | Sampling method of coal sample for production. |
| GB/T 482-1995 | Sampling of coal in seam. |
| GB/T 483-1998 | General rules for analytical and testing methods of coal. |
| GB/T 1341-2001 | 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. |
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| GB/T 2565-1998 | Determination of Hardgrove grindability index in coal. |
| GB/T 2566-1995 | Determination of transmittance for low rank coal. |
| GB/T 3058-1996 | Determination of arsenic in coal. |
| GB/T 3558-1996 | Determination of chlorine in coke. |
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| GB/T 4632-1997 | Determination of moisture holding capacity of coal. |
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| GB/T 5447-1997 | Determination of caking index of bituminous coals. |
| GB/T 5448-1997 | Determination of fusibility of ash in coal. |
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| GB/T 5448-1997 | Determination of Roga index of bituminous coal. |
| GB/T 5448-1997 | Determination of free swelling index(FSI)/crucible swelling number(CSN) in coal. |
| GB/T 5450-1997 | Audibert–Arnu dilatometer test of bituminous coal. |
| GB/T 5751-1986 | China coal classification. |
| GB/T 6948-1998 | Microscopic determination of the reflectance of vitrinite in coal. |
| GB/T 6949-1998 | Determination of apparent relative density of coal. |
| GB/T 7186-1998 | Terms relating to coal preparation of coal. |
| GB/T 7560-2001 | Determination of mineral matter in coal. |
| GB/T 8899-1998 | Determination of maceral group composition and minerals in coal. |
| GB/T 9649.17-2001 | 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 12937-1995 | Terms relating to coal petrology. |
| GB/T 14181-1997 | Specification of anthracite for determination of caking index of bituminous coal. |
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| GB/T 15224.2-2004 | Classification for sulfur content of coal. |
| GB/T 15224.3-2004 | Classification for calorific value of coal. |
| GB/T 15334-1994 | Determination of moisture in coal – microwave drying method. |
| GB/T 15458-1995 | Determination of abrasion index of coal. |
| GB/T 15459-1995 | Determination of shatter strength of coal. |
| GB/T 15460-2003 | Determination of carbon and hydrogen in coal – colormetric and gravimetric method. |
| GB/T 15588-2001 | 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 16417-1996 | Method of evaluating the washability of coal. |
| GB/T 16658-1996 | Determination of chromium, cadmium and lead in coal. |
| GB/T 16659-1996 | Determination of mercury in coal. |

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| 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 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. |
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Manak Bhavan, 9 Bahadur Shah Zafar Marg, New Delhi 110002, India.

Figures after Standard give the year of most recent re-approval.

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| 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 sulfur (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 5) -2001 | Methods of test for coal and coke: Part 5 Special impurities (first revision. |
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| IS1354-2000 | Methods of test for coke special test (second revision). |
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| 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). |
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| IS5062(Part 3)-2000 | 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). |
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| 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. |
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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. |
| IS15439-2000 | Hard coal – determination of oxygen content. |
| IS15440-2004 | Coal sampling of pulverized coal conveyed by gases in direct fixed coal systems. |

## State Standards of Russia (GOST, i.e. Gosudarstvennyy Standart)

Obtainable from: Technormativ Information Systems LLC, 19 Shossa Entuziastov, Moscow 111024, Russia.

| | |
|---|---|
| GOST 1186-1987 | Method for determination of plastometric indices. |
| 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. |
| GOST 10100-1984 | Determination of washability of hard coal. |
| GOST 10175-1975 | Determination of germanium 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 (reinous 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 11762-1987 | Brown coals, hard coals and anthracite. Accuracy of weighing. |
| GOST 14834-1986 | Oxidized brown coals of the Far East. Classification. |
| GOST 15585-2009 | Hard coal. Determination of caking index. |
| GOST 16126-1991 | Hard coal. Determination of caking power – Gray–King method. |
| GOST 17070-1991 | Russian coal – terms and definitions. |
| GOST 17321-1971 | 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 25543-1988 | 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 28823-1990 | Methods for the petrographic analysis of bituminous coal and anthracite: Part 4 Method of determining microlithotypes, carbominerite and mineral composition. |
| GOST 29086-1991 | Hard coal. Determination of mineral matter. |
| GOST 30313-1995 | Classification of coal. |
| 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)

# APPENDIX 2 Tables of True and Apparent Dip, Slope Angles, Gradients and Per Cent Slope

## Table of true and apparent dip

| True dip | Angle between strike and direction of section | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | 80° | 75° | 70° | 65° | 60° | 55° | 50° | 45° | 40° | 35° | 30° | 25° | 20° | 15° | 10° | 5° |
| | 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 above may be calculated from:

$$\tan(\text{apparent dip}) = \tan\ (\text{true dip} \times \sin(\text{angle between strike and direction of section}).$$

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## Table of dips of strata and of land surfaces*

| Angle of slope (degrees) | Gradient of slope | Per cent |
|---|---|---|
| 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 | |

*These can be expressed in either angles, gradients or per cent slope. Those values most commonly encountered are included in this table.

# APPENDIX 3 Calorific Values Expressed in Different Units

| MJ kg$^{-1}$ | Btu lb$^{-1}$* | kcal kg$^{-1}$† | lb lb$^{-1}$‡ |
|---|---|---|---|
| 4.5 | 1,935 | 1,076 | 1.99 |
| 4.6 | 1,978 | 1,099 | 2.04 |
| 4.7 | 2,021 | 1,123 | 2.08 |
| 4.8 | 2,064 | 1,147 | 2.13 |
| 4.9 | 2,107 | 1,171 | 2.17 |
| 5.0 | 2,150 | 1,195 | 2.21 |
| 5.1 | 2,193 | 1,219 | 2.26 |
| 5.2 | 2,236 | 1,243 | 2.30 |
| 5.3 | 2,279 | 1,267 | 2.35 |
| 5.4 | 2,322 | 1,291 | 2.39 |
| 5.5 | 2,365 | 1,315 | 2.44 |
| 5.6 | 2,408 | 1,338 | 2.48 |
| 5.7 | 2,451 | 1,362 | 2.52 |
| 5.8 | 2,494 | 1,386 | 2.57 |
| 5.9 | 2,537 | 1,410 | 2.61 |
| 6.0 | 2,580 | 1,434 | 2.66 |
| 6.1 | 2,623 | 1,458 | 2.70 |
| 6.2 | 2,666 | 1,482 | 2.75 |
| 6.3 | 2,709 | 1,506 | 2.79 |
| 6.4 | 2,752 | 1,530 | 2.83 |
| 6.5 | 2,794 | 1,554 | 2.88 |
| 6.6 | 2,837 | 1,577 | 2.92 |
| 6.7 | 2,880 | 1,601 | 2.97 |
| 6.8 | 2,923 | 1,625 | 3.01 |
| 6.9 | 2,966 | 1,649 | 3.06 |
| 7.0 | 3,009 | 1,673 | 3.10 |
| 7.1 | 3,052 | 1,697 | 3.14 |
| 7.2 | 3,095 | 1,721 | 3.19 |
| 7.3 | 3,138 | 1,745 | 3.23 |
| 7.4 | 3,181 | 1,769 | 3.28 |
| 7.5 | 3,224 | 1,793 | 3.32 |
| 7.6 | 3,267 | 1,816 | 3.36 |
| 7.7 | 3,310 | 1,840 | 3.41 |
| 7.8 | 3,353 | 1,864 | 3.45 |
| 7.9 | 3,396 | 1,888 | 3.50 |
| 8.0 | 3,439 | 1,912 | 3.54 |
| 8.1 | 3,482 | 1,936 | 3.59 |
| 8.2 | 3,525 | 1,960 | 3.63 |
| 8.3 | 3,568 | 1,984 | 3.67 |
| 8.4 | 3,611 | 2,008 | 3.72 |
| 8.5 | 3,654 | 2,032 | 3.76 |
| 8.6 | 3,697 | 2,055 | 3.81 |
| 8.7 | 3,740 | 2,079 | 3.85 |
| 8.8 | 3,783 | 2,103 | 3.90 |
| 8.9 | 3,826 | 2,127 | 3.94 |
| 9.0 | 3,869 | 2,151 | 3.98 |
| 9.1 | 3,912 | 2,175 | 4.03 |
| 9.2 | 3,955 | 2,199 | 4.07 |
| 9.3 | 3,998 | 2,223 | 4.12 |
| 9.4 | 4,041 | 2,247 | 4.16 |
| 9.5 | 4,084 | 2,271 | 4.21 |
| 9.6 | 4,127 | 2,294 | 4.25 |
| 9.7 | 4,170 | 2,318 | 4.29 |
| 9.8 | 4,213 | 2,342 | 4.34 |
| 9.9 | 4,256 | 2,366 | 4.38 |
| 10.0 | 4,299 | 2,390 | 4.43 |
| 10.1 | 4,342 | 2,414 | 4.47 |
| 10.2 | 4,385 | 2,438 | 4.52 |
| 10.3 | 4,428 | 2,462 | 4.56 |
| 10.4 | 4,471 | 2,486 | 4.60 |
| 10.5 | 4,514 | 2,510 | 4.65 |
| 10.6 | 4,557 | 2,533 | 4.69 |
| 10.7 | 4,600 | 2,557 | 4.74 |
| 10.8 | 4,643 | 2,581 | 4.78 |
| 10.9 | 4,686 | 2,605 | 4.83 |

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(*Continued*)

| MJ $kg^{-1}$ | Btu $lb^{-1}$* | kcal $kg^{-1}$† | lb $lb^{-1}$‡ |
|---|---|---|---|
| 11.0 | 4,729 | 2,629 | 4.87 |
| 11.1 | 4,772 | 2,653 | 4.91 |
| 11.2 | 4,815 | 2,677 | 4.96 |
| 11.3 | 4,858 | 2,701 | 5.00 |
| 11.4 | 4,901 | 2,725 | 5.05 |
| 11.5 | 4,944 | 2,749 | 5.09 |
| 11.6 | 4,987 | 2,772 | 5.14 |
| 11.7 | 5,030 | 2,796 | 5.18 |
| 11.8 | 5,073 | 2,820 | 5.22 |
| 11.9 | 5,116 | 2,844 | 5.27 |
| 12.0 | 5,159 | 2,868 | 5.31 |
| 12.1 | 5,202 | 2,892 | 5.36 |
| 12.2 | 5,245 | 2,916 | 5.40 |
| 12.3 | 5,288 | 2,940 | 5.45 |
| 12.4 | 5,331 | 2,964 | 5.49 |
| 12.5 | 5,374 | 2,988 | 5.53 |
| 12.6 | 5,417 | 3,011 | 5.58 |
| 12.7 | 5,460 | 3,035 | 5.62 |
| 12.8 | 5,503 | 3,059 | 5.67 |
| 12.9 | 5,546 | 3,083 | 5.71 |
| 13.0 | 5,589 | 3,107 | 5.76 |
| 13.1 | 5,632 | 3,131 | 5.80 |
| 13.2 | 5,675 | 3,155 | 5.84 |
| 13.3 | 5,718 | 3,179 | 5.89 |
| 13.4 | 5,761 | 3,203 | 5.93 |
| 13.5 | 5,804 | 3,227 | 5.98 |
| 13.6 | 5,847 | 3,250 | 6.02 |
| 13.7 | 5,890 | 3,274 | 6.07 |
| 13.8 | 5,933 | 3,298 | 6.11 |
| 13.9 | 5,976 | 3,322 | 6.15 |
| 14.0 | 6,019 | 3,346 | 6.20 |
| 14.1 | 6,062 | 3,370 | 6.24 |
| 14.2 | 6,105 | 3,394 | 6.29 |
| 14.3 | 6,148 | 3,418 | 6.33 |
| 14.4 | 6,191 | 3,442 | 6.38 |
| 14.5 | 6,234 | 3,466 | 6.42 |
| 14.6 | 6,277 | 3,489 | 6.46 |
| 14.7 | 6,320 | 3,513 | 6.51 |
| 14.8 | 6,363 | 3,537 | 6.55 |
| 14.9 | 6,406 | 3,561 | 6.60 |
| 15.0 | 6,449 | 3,585 | 6.64 |
| 15.1 | 6,492 | 3,609 | 6.69 |
| 15.2 | 6,535 | 3,633 | 6.73 |
| 15.3 | 6,578 | 3,657 | 6.77 |
| 15.4 | 6,621 | 3,681 | 6.82 |
| 15.5 | 6,664 | 3,705 | 6.86 |
| 15.6 | 6,707 | 3,728 | 6.91 |
| 15.7 | 6,750 | 3,752 | 6.95 |
| 15.8 | 6,793 | 3,776 | 7.00 |
| 15.9 | 6,836 | 3,800 | 7.04 |
| 16.0 | 6,879 | 3,824 | 7.08 |
| 16.1 | 6,922 | 3,848 | 7.13 |
| 16.2 | 6,965 | 3,872 | 7.17 |
| 16.3 | 7,008 | 3,896 | 7.22 |
| 16.4 | 7,051 | 3,920 | 7.26 |
| 16.5 | 7,094 | 3,944 | 7.31 |
| 16.6 | 7,137 | 3,967 | 7.35 |
| 16.7 | 7,180 | 3,991 | 7.39 |
| 16.8 | 7,223 | 4,015 | 7.44 |
| 16.9 | 7,266 | 4,039 | 7.48 |
| 17.0 | 7,309 | 4,063 | 7.53 |
| 17.1 | 7,352 | 4,087 | 7.57 |
| 17.2 | 7,395 | 4,111 | 7.62 |
| 17.3 | 7,438 | 4,135 | 7.66 |
| 17.4 | 7,481 | 4,159 | 7.70 |
| 17.5 | 7,524 | 4,183 | 7.75 |
| 17.6 | 7,567 | 4,207 | 7.79 |
| 17.7 | 7,610 | 4,230 | 7.84 |
| 17.8 | 7,653 | 4,254 | 7.88 |
| 17.9 | 7,696 | 4,278 | 7.93 |
| 18.0 | 7,739 | 4,302 | 7.97 |
| 18.1 | 7,782 | 4,326 | 8.01 |
| 18.2 | 7,825 | 4,350 | 8.06 |
| 18.3 | 7,868 | 4,374 | 8.10 |
| 18.4 | 7,911 | 4,398 | 8.15 |
| 18.5 | 7,954 | 4,422 | 8.19 |
| 18.6 | 7,997 | 4,446 | 8.24 |
| 18.7 | 8,040 | 4,469 | 8.28 |
| 18.8 | 8,083 | 4,493 | 8.32 |
| 18.9 | 8,126 | 4,517 | 8.37 |
| 19.0 | 8,169 | 4,541 | 8.41 |
| 19.1 | 8,212 | 4,565 | 8.46 |
| 19.2 | 8,255 | 4,589 | 8.50 |
| 19.3 | 8,298 | 4,613 | 8.55 |
| 19.4 | 8,341 | 4,637 | 8.59 |
| 19.5 | 8,383 | 4,661 | 8.63 |
| 19.6 | 8,426 | 4,685 | 8.68 |
| 19.7 | 8,469 | 4,708 | 8.72 |
| 19.8 | 8,512 | 4,732 | 8.77 |
| 19.9 | 8,555 | 4,756 | 8.81 |

(*Continued*)

| $MJ\ kg^{-1}$ | $Btu\ lb^{-1}$* | $kcal\ kg^{-1}$† | $lb\ lb^{-1}$‡ |
|---|---|---|---|
| 20.0 | 8,598 | 4,780 | 8.86 |
| 20.1 | 8,641 | 4,804 | 8.90 |
| 20.2 | 8,684 | 4,828 | 8.94 |
| 20.3 | 8,727 | 4,852 | 8.99 |
| 20.4 | 8,770 | 4,876 | 9.03 |
| 20.5 | 8,813 | 4,900 | 9.08 |
| 20.6 | 8,856 | 4,924 | 9.12 |
| 20.7 | 8,899 | 4,947 | 9.17 |
| 20.8 | 8,942 | 4,971 | 9.21 |
| 20.9 | 8,985 | 4,995 | 9.25 |
| 21.0 | 9,028 | 5,019 | 9.30 |
| 21.1 | 9,071 | 5,043 | 9.34 |
| 21.2 | 9,114 | 5,067 | 9.39 |
| 21.3 | 9,157 | 5,091 | 9.43 |
| 21.4 | 9,200 | 5,115 | 9.48 |
| 21.5 | 9,243 | 5,139 | 9.52 |
| 21.6 | 9,286 | 5,163 | 9.56 |
| 21.7 | 9,329 | 5,186 | 9.61 |
| 21.8 | 9,372 | 5,210 | 9.65 |
| 21.9 | 9,415 | 5,234 | 9.70 |
| 22.0 | 9,458 | 5,258 | 9.74 |
| 22.1 | 9,501 | 5,282 | 9.79 |
| 22.2 | 9,544 | 5,306 | 9.83 |
| 22.3 | 9,587 | 5,330 | 9.87 |
| 22.4 | 9,630 | 5,354 | 9.92 |
| 22.5 | 9,673 | 5,378 | 9.96 |
| 22.6 | 9,716 | 5,402 | 10.01 |
| 22.7 | 9,759 | 5,425 | 10.05 |
| 22.8 | 9,802 | 5,449 | 10.09 |
| 22.9 | 9,845 | 5,473 | 10.14 |
| 23.0 | 9,888 | 5,497 | 10.18 |
| 23.1 | 9,931 | 5,521 | 10.23 |
| 23.2 | 9,974 | 5,545 | 10.27 |
| 23.3 | 10,017 | 5,569 | 10.32 |
| 23.4 | 10,060 | 5,593 | 10.36 |
| 23.5 | 10,103 | 5,617 | 10.40 |
| 23.6 | 10,146 | 5,641 | 10.45 |
| 23.7 | 10,189 | 5,664 | 10.49 |
| 23.8 | 10,232 | 5,688 | 10.54 |
| 23.9 | 10,275 | 5,712 | 10.58 |
| 24.0 | 10,318 | 5,736 | 10.63 |
| 24.1 | 10,361 | 5,760 | 10.67 |
| 24.2 | 10,404 | 5,784 | 10.71 |
| 24.3 | 10,447 | 5,808 | 10.76 |
| 24.4 | 10,490 | 5,832 | 10.80 |
| 24.5 | 10,533 | 5,856 | 10.85 |
| 24.6 | 10,576 | 5,880 | 10.89 |
| 24.7 | 10,619 | 5,903 | 10.94 |
| 24.8 | 10,662 | 5,927 | 10.98 |
| 24.9 | 10,705 | 5,951 | 11.02 |
| 25.0 | 10,748 | 5,975 | 11.07 |
| 25.1 | 10,791 | 5,999 | 11.11 |
| 25.2 | 10,834 | 6,023 | 11.16 |
| 25.3 | 10,877 | 6,047 | 11.20 |
| 25.4 | 10,920 | 6,071 | 11.25 |
| 25.5 | 10,963 | 6,095 | 11.29 |
| 25.6 | 11,006 | 6,119 | 11.33 |
| 25.7 | 11,049 | 6,142 | 11.38 |
| 25.8 | 11,092 | 6,166 | 11.42 |
| 25.9 | 11,135 | 6,190 | 11.47 |
| 26.0 | 11,178 | 6,214 | 11.51 |
| 26.1 | 11,221 | 6,238 | 11.56 |
| 26.2 | 11,264 | 6,262 | 11.60 |
| 26.3 | 11,307 | 6,286 | 11.64 |
| 26.4 | 11,350 | 6,310 | 11.69 |
| 26.5 | 11,393 | 6,334 | 11.73 |
| 26.6 | 11,436 | 6,358 | 11.78 |
| 26.7 | 11,479 | 6,381 | 11.82 |
| 26.8 | 11,522 | 6,405 | 11.87 |
| 26.9 | 11,565 | 6,429 | 11.91 |
| 27.0 | 11,608 | 6,453 | 11.95 |
| 27.1 | 11,651 | 6,477 | 12.00 |
| 27.2 | 11,694 | 6,501 | 12.04 |
| 27.3 | 11,737 | 6,525 | 12.09 |
| 27.4 | 11,780 | 6,549 | 12.13 |
| 27.5 | 11,823 | 6,573 | 12.18 |
| 27.6 | 11,866 | 6,597 | 12.22 |
| 27.7 | 11,909 | 6,620 | 12.26 |
| 27.8 | 11,952 | 6,644 | 12.31 |
| 27.9 | 11,995 | 6,668 | 12.35 |
| 28.0 | 12,038 | 6,692 | 12.40 |
| 28.1 | 12,081 | 6,716 | 12.44 |
| 28.2 | 12,124 | 6,740 | 12.49 |
| 28.3 | 12,167 | 6,764 | 12.53 |
| 28.4 | 12,210 | 6,788 | 12.57 |
| 28.5 | 12,253 | 6,812 | 12.62 |
| 28.6 | 12,296 | 6,836 | 12.66 |
| 28.7 | 12,339 | 6,859 | 12.71 |
| 28.8 | 12,382 | 6,883 | 12.75 |
| 28.9 | 12,425 | 6,907 | 12.80 |

(*Continued*)

| MJ $kg^{-1}$ | Btu $lb^{-1}$* | kcal $kg^{-1}$† | lb $lb^{-1}$‡ |
|---|---|---|---|
| 29.0 | 12,468 | 6,931 | 12.84 |
| 29.1 | 12,511 | 6,955 | 12.88 |
| 29.2 | 12,554 | 6,979 | 12.93 |
| 29.3 | 12,597 | 7,003 | 12.97 |
| 29.4 | 12,640 | 7,027 | 13.02 |
| 29.5 | 12,683 | 7,051 | 13.06 |
| 29.6 | 12,726 | 7,075 | 13.11 |
| 29.7 | 12,769 | 7,098 | 13.15 |
| 29.8 | 12,812 | 7,122 | 13.19 |
| 29.9 | 12,855 | 7,146 | 13.24 |
| 30.0 | 12,898 | 7,170 | 13.28 |
| 30.1 | 12,941 | 7,194 | 13.33 |
| 30.2 | 12,984 | 7,218 | 13.37 |
| 30.3 | 13,027 | 7,242 | 13.42 |
| 30.4 | 13,070 | 7,266 | 13.46 |
| 30.5 | 13,113 | 7,290 | 13.50 |
| 30.6 | 13,156 | 7,314 | 13.55 |
| 30.7 | 13,199 | 7,337 | 13.59 |
| 30.8 | 13,242 | 7,361 | 13.64 |
| 30.9 | 13,285 | 7,385 | 13.68 |
| 31.0 | 13,328 | 7,409 | 13.73 |
| 31.1 | 13,371 | 7,433 | 13.77 |
| 31.2 | 13,414 | 7,457 | 13.81 |
| 31.3 | 13,457 | 7,481 | 13.86 |
| 31.4 | 13,500 | 7,505 | 13.90 |
| 31.5 | 13,543 | 7,529 | 13.95 |
| 31.6 | 13,586 | 7,553 | 13.99 |
| 31.7 | 13,629 | 7,576 | 14.04 |
| 31.8 | 13,672 | 7,600 | 14.08 |
| 31.9 | 13,715 | 7,624 | 14.12 |
| 32.0 | 13,758 | 7,648 | 14.17 |
| 32.1 | 13,801 | 7,672 | 14.21 |
| 32.2 | 13,844 | 7,696 | 14.26 |
| 32.3 | 13,887 | 7,720 | 14.30 |
| 32.4 | 13,930 | 7,744 | 14.35 |
| 32.5 | 13,972 | 7,768 | 14.39 |
| 32.6 | 14,015 | 7,792 | 14.43 |
| 32.7 | 14,058 | 7,815 | 14.48 |
| 32.8 | 14,101 | 7,839 | 14.52 |
| 32.9 | 14,144 | 7,863 | 14.57 |
| 33.0 | 14,187 | 7,887 | 14.61 |
| 33.1 | 14,230 | 7,911 | 14.66 |
| 33.2 | 14,273 | 7,935 | 14.70 |
| 33.3 | 14,316 | 7,959 | 14.74 |
| 33.4 | 14,359 | 7,983 | 14.79 |
| 33.5 | 14,402 | 8,007 | 14.83 |
| 33.6 | 14,445 | 8,031 | 14.88 |
| 33.7 | 14,488 | 8,055 | 14.92 |
| 33.8 | 14,531 | 8,078 | 14.97 |
| 33.9 | 14,574 | 8,102 | 15.01 |
| 34.0 | 14,617 | 8,126 | 15.05 |
| 34.1 | 14,660 | 8,150 | 15.10 |
| 34.2 | 14,703 | 8,174 | 15.14 |
| 34.3 | 14,746 | 8,198 | 15.19 |
| 34.4 | 14,789 | 8,222 | 15.23 |
| 34.5 | 14,832 | 8,246 | 15.28 |
| 34.6 | 14,875 | 8,270 | 15.32 |
| 34.7 | 14,918 | 8,294 | 15.36 |
| 34.8 | 14,961 | 8,317 | 15.41 |
| 34.9 | 15,004 | 8,341 | 15.45 |
| 35.0 | 15,047 | 8,365 | 15.50 |
| 35.1 | 15,090 | 8,389 | 15.54 |
| 35.2 | 15,133 | 8,413 | 15.59 |
| 35.3 | 15,176 | 8,437 | 15.63 |
| 35.4 | 15,219 | 8,461 | 15.67 |
| 35.5 | 15,262 | 8,485 | 15.72 |
| 35.6 | 15,305 | 8,509 | 15.76 |
| 35.7 | 15,348 | 8,533 | 15.81 |
| 35.8 | 15,391 | 8,556 | 15.85 |
| 35.9 | 15,434 | 8,580 | 15.90 |
| 36.0 | 15,477 | 8,604 | 15.94 |
| 36.1 | 15,520 | 8,628 | 15.98 |
| 36.2 | 15,563 | 8,652 | 16.03 |
| 36.3 | 15,606 | 8,676 | 16.07 |
| 36.4 | 15,649 | 8,700 | 16.12 |
| 36.5 | 15,692 | 8,724 | 16.16 |
| 36.6 | 15,735 | 8,748 | 16.21 |
| 36.7 | 15,778 | 8,772 | 16.25 |
| 36.8 | 15,821 | 8,795 | 16.29 |
| 36.9 | 15,864 | 8,819 | 16.34 |
| 37.0 | 15,907 | 8,843 | 16.38 |
| 37.1 | 15,950 | 8,867 | 16.43 |
| 37.2 | 15,993 | 8,891 | 16.47 |
| 37.3 | 16,036 | 8,915 | 16.52 |
| 37.4 | 16,079 | 8,939 | 16.56 |

# APPENDIX 4 Units of measurement

$1 \times 10^6$ t coal equivalent = $1 \times 10^6$ t of coal at 28.0 MJ $kg^{-1}$
or 6692 kcal $kg^{-1}$ gross calorific value.

$1 \times 10^6$ t oil equivalent = $1.5 \times 10^6$ t of coal (approximately)

= $3.0 \times 10^6$ t of lignite (approximately).

1 t of coal at 25.1 MJ $kg^{-1}$ or 6000 kcal $kg^{-1}$ will produce approximately 2400 kW $h^{-1}$ of electricity.

A 1000 MW power station requires $3 \times 10^6$ t of coal at 25.1 MJ $kg^{-1}$ $yr^{-1}$.

1 t of coal at 25.1 MJ $kg^{-1}$ or 6000 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.

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# APPENDIX 5 Methane Units Converter

| To convert from: | To: | Multiply by: |
|---|---|---|
| Cubic foot | Cubic metre | 0.02832 |
| Pound | Kilogram | 0.4536 |
| Short ton | Metric ton (tonne) | 0.9072 |
| Btu | Joule | 1055 |
| Cubic foot methane | Pound methane | 0.04246 |
| | Btu | 1,014.6 |
| | kW-h | 0.2974 |
| | t $CO_2$ equivalent | 0.000404 |
| | t C equivalent | 0.00011 |
| | Gram | 19.26 |
| Pound methane | Btu | 23,896 |
| | kW-h | 7 |
| | t $CO_2$ equivalent | 0.00953 |
| | t C equivalent | 0.0026 |
| Btu | kW-h | 0.000293 |
| Methane GWP | $CO_2$ GWP | 21 |
| $CO_2$ | C equivalent | 0.27273 |
| Methane | C equivalent | 5.7273 |

GWP = global warming potential.
Source: adapted from US Environmental Protection Agency Coalbed Methane Outreach Program.

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# Glossary

**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.

**ANTHRACITE** Anthracite is the highest rank coal and is characterised by low volatile matter ($<10\%$) and high carbon content. Semi-anthracite is coal midway between low volatile bituminous and anthracite.

**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 sulphurous gases from sulphides.

**ASSIGNED RESERVES** Coal which 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*.

**BILLION** 1,00,00,00,000 (one thousand million)

**BITUMINOUS COAL** Bituminous coal lies between subbituminous coal and semi-anthracite in terms of rank. Usually divided into three sub groups – low volatile, medium volatile and high volatile.

**BROWN COAL** See **Lignite**.

**BUCKETWHEEL EXCAVATOR (BWE)** Large earth-moving machines using a boom with a rotating wheel on which hang a series of buckets with a cutting edge for excavating soft overburden and brown coal or lignite.

**BULK SAMPLE** Large coal sample, 5–25 tonnes, 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** and **Net calorific value**.

**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 which may have originated during coalification.

**COAL BED METHANE (CBM)** Occurs as a free gas (methane $CH_4$) 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.

**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.

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**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.

**DES** Delivered Ex-Ship, cargo considered delivered after unloading at port.

**DRAGLINE** Large earth moving machines with bucket capacities 5–30 $m^3$, 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.

**EMISSIONS** Those gases and particulates vented by power stations and industrial users; in particular, those which 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 which is usually less than 4 mm.

**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 DESULPHURISATION (FGD)** A process designed to recover the acidic sulphur 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 which causes the bed to be suspended resulting in the transference of heat at very high rates.

**FOB** Free on Board; cargoes purchased when loaded at port of embarkation.

**GEOLOGICAL LOSSES** Losses to be deducted from measured reserves due to geological constraints, e.g. faults, washouts, seam splitting.

**GEOPHYSICAL LOGGING** Measurement of the variation with depth of selected physical properties of rocks with geophysical measuring tools (sondes) located in boreholes.

**GOAF/GOB** Once an underground mine area has been mined and no longer required for access, the roof is allowed to collapse into the abandoned area to form 'goaf' or 'gob'.

**GONDWANALAND** Southern segment of Pangaea.

**GPS** Global Positioning Systems; 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.

**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.

**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 which 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)** 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** Those for which the density and quality of the points of measurement are not more than 2.0 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** Resources for which the points of measurement are widely spaced, usually not more than 4.0 km, so that only an uncertain estimate of the resource can be made.

***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 pounds. 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 SURVEY** Measurement of the magnetic susceptibility of rocks.

**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 1.0 km apart and are sufficient to allow a reliable estimate of the coal tonnage.

**MEASURED RESOURCES** The projection of the thickness of coal, rank and quality data for a radius of 0.4 km from a point of measurement.

**MEGAWATT HOUR (MWh)** A unit of electricity denoting the work of one million Watts acting for one hour.

**METALLURGICAL COAL** (See also **Coking coal**.) Coal suitable for metallurgical use because of its coking qualities and chemical characteristics.

**METHANE** 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 yield of the two products are calculated by using an M curve graph.

**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, sulphur oxides and water of hydration lost upon combustion of the coal.

**MIRE** General term for peat forming ecosystems of all types.

**NET CALORIFIC VALUE** During combustion in furnaces, the maximum achievable calorific value is the net calorific value at constant pressure, and is calculated and expressed in absolute joules, calories per gram, or Btu per pound.

**$NO_x$** A group of gaseous pollutants, notably nitrous oxide ($N_2O$) 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 which may be overlain by variable amounts of overburden.

**OXIDATION** When coal is exposed to oxygen and the properties of coal begin to change, in particular the lowering of calorific value.

**PANGAEA** Supercontinent existing until Tiassic Period.

**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**.

**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 which has received no preparation other than possibly screening.

**RECONNAISSANCE** Peliminary 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 which 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.

**RMR** Rock Mass Rating, an indicator of the behaviour of rock mass surrounding an underground excavation using a number of geotechnical parameters.

**ROOM & 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** Production of acoustic or seismic signals when an explosive device is introduced into the ground and which are reflected or refracted back to recording equipment at the surface.

**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 pounds.

**$SO_2$** A principal gaseous pollutant released by flue gases vented to the atmosphere.

**SPECIMEN SAMPLE** Orientated sample of coal collected for studies in the laboratory of optical fabric or structural features of coal.

**SPONTANEOUS COMBUSTION** The adsorption of oxygen on to 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 noncoking 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%.

**SULPHUR** May be a component of the organic and/or mineral fractions of a coal. Forms sulphur dioxide during coal combustion, a serious pollutant. It is also undesirable in coking coals because it contaminates the hot metal.

**THERMAL COAL** See **Steam coal**.

**TONNE** Or metric tonne, 1000 kilogrammes or 2204.6 pounds.

**TOPOGENOUS** Peat whose moisture content is dependent on surface water.

**TRUCK** Large capacity vehicle usually employed for transporting overburden material from the mine to dump areas; size usually tailored to size of earthmoving equipment; not used on public highways.

**UNASSIGNED RESERVES** Coal which 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.

**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 which 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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