I.1 GENERAL INTRODUCTION

The term ‘light metals’ has traditionally been given to both aluminium and magnesium because they are frequently used to reduce the weight of components and structures. On this basis, titanium also qualifies and beryllium should be included although it is little used and will not be considered in detail in this book. These four metals have relative densities ranging from 1.7 (magnesium) to 4.5 (titanium) which compare with 7.9 and 8.9 for the older structural metals, iron and copper, and 22.6 for osmium, the heaviest of all metals. Ten other elements that are classified as metals are lighter than titanium but, with the exception of boron in the form of strong fibres contained in a suitable matrix, none is used as a base material for structural purposes. The alkali metals lithium, potassium, sodium, rubidium and caesium, and the alkaline earth metals calcium and strontium are too reactive, whereas yttrium and scandium are comparatively rare.

I.1.1 Characteristics of light metals and alloys

The property of lightness translates directly to material property enhancement for many products since by far the greatest weight reduction is achieved by a decrease in density (Fig. 1.1). This is an obvious reason why light metals have been associated with transportation, notably aerospace, which has provided great stimulus to the development of light alloys during the last 50 years. Strength: weight ratios have also been a dominant consideration and the central positions of the light alloys based on aluminium, magnesium and titanium with respect both to other engineering alloys, and to all materials are represented in an Ashby diagram in Fig. 1.2. The advantages of decreased density become even more important in engineering design when parameters such as stiffness and resistance to buckling are involved. For example, the stiffness of a simple rectangular beam is directly proportional to the product of the elastic modulus and the cube of the thickness. The significance of this relationship is illustrated by the nomograph
shown in Fig. 1.3 which allows the weights of similar beams of different metals and alloys to be estimated for equal values of stiffness. An iron (or steel) beam weighing 10 kg will have the same stiffness as beams of equal width and length weighing 7 kg in titanium, 4.9 kg in aluminium, 3.8 kg in magnesium, and only 2.2 kg in beryllium. The Mg–Li alloy is included because it is the lightest (relative density 1.35) structural alloy that is available commercially. Comparative stiffness for equal weights of a similar beam increase in the ratios 1:2.9:8.2:18.9 for steel, titanium, aluminium, and magnesium respectively.

Concern with aspects of weight saving should not obscure the fact that light metals possess other properties of considerable technological importance, e.g. the high corrosion resistance and high electrical and thermal conductivities of aluminium, the machinability of magnesium, and extreme corrosion resistance of titanium. Comparisons of some physical properties are made in Table 1.1.

Beryllium was discovered by Vauquelin in France in 1798 as the oxide in the mineral beryl (beryllium aluminium silicate), and in emerald. It was first isolated independently by Wöhler and Bussy in 1828 who reduced the chloride with potassium. Beryl has traditionally been a by-product of emerald mining and was until recently the major source of beryllium metal. Currently more beryllium is extracted from the closely associated mineral bertrandite (beryllium silicate hydroxide).

Beryllium has some remarkable properties (Table 1.1). Its stiffness, as measured by specific elastic modulus, is nearly an order of magnitude greater than that for the other light metals, or for the commonly used metals iron, copper and nickel. This has led to its use in gyroscopes and in inertial guidance systems. It has a relatively high melting point, and its capture cross-section (i.e. permeability) for neutrons is lower than for any other metal. These properties have stimulated much interest by the aerospace and nuclear industries. For example, a design study specifying beryllium as the major structural material for a supersonic transport aircraft has indicated possible weight savings of up to 50% for
components for which it could be used. However, its structural uses have been
confined largely to components for spacecraft, and for applications such as satel-
lite antenna booms. In nuclear engineering it has had potential for use as a fuel
element can in power reactors. Another unique property of beryllium is its high
specific heat which is approximately twice that of aluminium and magnesium,
and four times that of titanium. This inherent capacity to absorb heat, when com-
bined with its low density, led to the selection of beryllium as the basis for the
re-entry heat shield of the Mercury capsule used for the first manned spacecraft
developed in the United States. In a more general application, it has served as

Fig. 1.2 Strength: density relationships for light alloys and other engineering materials.
Note that yield strength is used as the measure of strength for metals and polymers,
compressive strength for ceramics, tear strength for elastomers and tensile strength for
composites (courtesy M. F. Ashby).
a heat sink when inserted in the centre of composite disc brakes used in the landing gear of a large military transport aircraft. Beryllium also shows outstanding optical reflectivity, particularly in the infrared, which have led to its combat use in target acquisition systems, as well as in space telescopes.

Despite much research in several countries, wider use has not been made of beryllium because it is costly to mine and extract, it has an inherently low ductility at ambient temperatures, and the fact that the powdered oxide is extremely toxic to some people. The problem of low ductility arises because of the dimensions of the close-packed hexagonal crystal structure of beryllium. The c/a ratio of the unit cell is 1.567 which is the lowest and most removed of all metals from the ideal value of 1.633. One result of this is a high degree of anisotropy between mechanical properties in the a and c crystallographic directions. At room temperature, slip is limited and only possible on the basal plane, which also happens to be the plane along which cleavage occurs. Furthermore, there has also been little opportunity to improve properties by alloying because the small size of the beryllium atom severely restricts its solubility for other elements. One exception is the eutectic composition Be-38Al in which some useful ductility has been achieved. This alloy was developed by the Lockheed Aircraft Company and became known as Lockalloy. Because beryllium and aluminium have little mutual solid solubility in each other, the alloy is essentially a composite material with a microstructure comprising stiff beryllium particles in a softer aluminium matrix. Light weight (specific gravity 2.09) extrusions and sheet have found limited aerospace applications.

Fig. 1.3 Nomograph allowing the comparative weights of different metals or alloys to be compared for equal levels of stiffness. These values can be obtained from the intercepts which lines drawn from point X make with lines representing the different metals or alloys (courtesy Brooks and Perkins Inc.).

<table>
<thead>
<tr>
<th>Property</th>
<th>Unit</th>
<th>Al</th>
<th>Mg</th>
<th>Ti</th>
<th>Be</th>
<th>Fe</th>
<th>Cu</th>
</tr>
</thead>
<tbody>
<tr>
<td>Atomic number</td>
<td></td>
<td>13</td>
<td>12</td>
<td>22</td>
<td>4</td>
<td>26</td>
<td>29</td>
</tr>
<tr>
<td>Relative atomic mass (C = 12.000)</td>
<td></td>
<td>26.982</td>
<td>24.305</td>
<td>47.90</td>
<td>9.012</td>
<td>55.847</td>
<td>63.546</td>
</tr>
<tr>
<td>Crystal structure</td>
<td></td>
<td>fcc</td>
<td>cph</td>
<td>cph</td>
<td>bcc</td>
<td>fcc</td>
<td>fcc</td>
</tr>
<tr>
<td>a</td>
<td>nm</td>
<td>0.4041</td>
<td>0.3203</td>
<td>0.2950</td>
<td>0.2286</td>
<td>0.2866</td>
<td>0.3615</td>
</tr>
<tr>
<td>c</td>
<td>nm</td>
<td>–</td>
<td>0.5199</td>
<td>0.4653</td>
<td>0.3583</td>
<td>–</td>
<td>–</td>
</tr>
<tr>
<td>Melting point</td>
<td>°C</td>
<td>660</td>
<td>650</td>
<td>1678</td>
<td>1289</td>
<td>1535</td>
<td>1083</td>
</tr>
<tr>
<td>Boiling point</td>
<td>°C</td>
<td>2520</td>
<td>1090</td>
<td>3289</td>
<td>2472</td>
<td>2862</td>
<td>2563</td>
</tr>
<tr>
<td>Relative density (d)</td>
<td></td>
<td>2.70</td>
<td>1.74</td>
<td>4.51</td>
<td>1.85</td>
<td>7.87</td>
<td>8.96</td>
</tr>
<tr>
<td>Elastic modulus (E)</td>
<td>GPa</td>
<td>70</td>
<td>45</td>
<td>120</td>
<td>295</td>
<td>211</td>
<td>130</td>
</tr>
<tr>
<td>Specific modulus (E/d)</td>
<td></td>
<td>26</td>
<td>26</td>
<td>26</td>
<td>160</td>
<td>27</td>
<td>14</td>
</tr>
<tr>
<td>Mean specific heat 0–100 °C</td>
<td>J kg⁻¹ K⁻¹</td>
<td>917</td>
<td>1038</td>
<td>528</td>
<td>2052</td>
<td>456</td>
<td>386</td>
</tr>
<tr>
<td>Thermal conductivity 20–100 °C</td>
<td>W m⁻¹ K⁻¹</td>
<td>238</td>
<td>156</td>
<td>26</td>
<td>194</td>
<td>78</td>
<td>397</td>
</tr>
<tr>
<td>Coefficient of thermal expansion 0–100 °C</td>
<td>10⁻⁶ K⁻¹</td>
<td>23.5</td>
<td>26.0</td>
<td>8.9</td>
<td>12.0</td>
<td>12.1</td>
<td>17.0</td>
</tr>
<tr>
<td>Electrical resistivity at 20 °C</td>
<td>µ ohm cm⁻¹</td>
<td>2.67</td>
<td>4.2</td>
<td>54</td>
<td>3.3</td>
<td>10.1</td>
<td>1.69</td>
</tr>
</tbody>
</table>

Note: Conversion factors for SI and Imperial units are given in the Appendix.
Beryllium is now prepared mainly by powder metallurgy methods. Metal extracted from the minerals beryl or bertrandite is vacuum melted and then either cast into small ingots, machined into chips and impact ground, or directly inert gas atomised to produce powders. The powders are usually consolidated by hot isostatic pressing (HIP) and the resulting billets have properties that are more isotropic than are obtained with cast ingots. Tensile properties depend on the levels of retained BeO (usually 1 to 2%) and impurities (iron, aluminium and silicon) and ductilities usually range from 3 to 5%. The billets can then be hot worked by forging, rolling to sheet, or extruding to produce bar or tube. Lockalloy (now also known as AlBeMet™ 162) is now also manufactured by inert gas atomisation of molten pre-alloyed mixtures and the resulting powders are consolidated and hot worked as described above.

1.1.2 Relative abundance

The estimated crustal abundance of the major chemical elements is given in Table 1.2 which shows that the light metals aluminium, magnesium, and titanium are first, third and fourth in order of occurrence of the structural metals. It can also be seen that the traditional metals copper, lead, and zinc are each present in amounts less than 0.10%. Estimates are also available for the occurrence of metals in the ocean which is the major commercial source of magnesium. Sea water contains 0.13% of this metal so that 1.3 million tonnes are present in each km³, which is approximately equivalent to three times the annual world consumption of magnesium in 2004. Overall, the reserves of the light metals are adequate to cope with anticipated demands for some centuries to come. The extent to which they will be used would seem to be controlled mainly by their future costs relative

<table>
<thead>
<tr>
<th>Table 1.2 Crustal abundance of major chemical elements (from Stanner, R. J. L., American Scientist, 64, 258, 1976)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Element</td>
</tr>
<tr>
<td>Oxygen</td>
</tr>
<tr>
<td>Silicon</td>
</tr>
<tr>
<td>Aluminium</td>
</tr>
<tr>
<td>Iron</td>
</tr>
<tr>
<td>Calcium</td>
</tr>
<tr>
<td>Magnesium</td>
</tr>
<tr>
<td>Sodium</td>
</tr>
<tr>
<td>Potassium</td>
</tr>
<tr>
<td>Titanium</td>
</tr>
<tr>
<td>Hydrogen</td>
</tr>
<tr>
<td>Manganese</td>
</tr>
<tr>
<td>Phosphorus</td>
</tr>
<tr>
<td>Total</td>
</tr>
</tbody>
</table>
to competing materials such as steel and plastics, as well as the availability of electrical energy that is needed for their extraction from minerals.

### 1.1.3 Trends in production and applications

Trends in the production of various metals and plastics are shown in Fig. 1.4 and it is clear that the light metals are very much materials of the twentieth century. Between 1900 and 1950, the annual world production of aluminium increased 250 times from around 6000 tonnes to 1.5 million tonnes. A further eightfold increase took place during the next quarter century when aluminium surpassed copper as the second most used metal.

During this period the annual rate of increase in aluminium production averaged 9.2%. Since the late 1970s the demand for most basic materials has fluctuated and overall annual increases have been much less (Fig. 1.4). These trends reflect world economic cycles, the emergence of China and the Russian Federation as major trading nations, and the greater attention being given to recycling. World

![Fig. 1.4 World production figures for various metals and plastics.](image-url)
production of primary (new) aluminium grew by an annual average of around 4% during the two decades 1980 to 2000 had reached an estimated 29.2 million tonnes in 2004. During this period, the production of secondary (recycled) aluminium is estimated to have risen from 3.8 to 8.4 million tonnes in 2004. Production of iron and steel, which still amounts to about 95% of all metal consumed, and which remained effectively static at around 700 million tonnes for a decade between 1980 and 1990, exceeded 1 billion tonnes in 2004. Here it is interesting to record that the cumulative total of aluminium that has ever been produced (estimated as being around 720 million tonnes up to 2004) is less than one year’s supply of iron! It may also be noted that the production of commodity plastics and other polymeric materials, which enjoyed spectacular growth during the period 1950 to 1980, had reached an estimated 185 million tonnes by 2004.

Two political events have had a major impact on the production and pricing of the light metals in recent years. The first was the change from the former Soviet Union to the Commonwealth of Independent States which was followed by the release of large quantities of metals for sale in Western nations, often at discounted prices. More significant has been the rapid transition of China over the last 20 years from a largely agrarian society to an increasingly industrialised economy. China now leads the world in the production of aluminium with Russia placed second followed by Canada, the United States, Australia and Brazil (Table 1.3). Further major expansion is expected in China where the per capita consumption of aluminium is still low being only 3.4 kg in 2002 compared with 29.3 kg in the United States and 27.9 kg in Japan. China has also become the world’s largest producer of magnesium and is rapidly increasing its capacity to produce titanium sponge.

Metal prices change from year to year and depend on factors that influence supply and demand. For example, the release of large quantities of aluminium by the C.I.S. that was mentioned above, combined with the effects of the world depression in the early 1990s, caused the price of aluminium to fall from an average of $US 1675 per tonne in 1990 to under $US 1100 in 1993.

Table 1.3 World production of primary aluminium for the years 1996, 2000 and 2004 (from International Aluminium Institute, London)

<table>
<thead>
<tr>
<th>Region</th>
<th>1996</th>
<th>2000</th>
<th>2004</th>
</tr>
</thead>
<tbody>
<tr>
<td>Africa</td>
<td>1 015</td>
<td>1 178</td>
<td>1 711</td>
</tr>
<tr>
<td>Asia</td>
<td>1 624</td>
<td>2 221</td>
<td>2 735</td>
</tr>
<tr>
<td>China</td>
<td>?</td>
<td>2 794</td>
<td>6 589</td>
</tr>
<tr>
<td>Europe Western</td>
<td>3 192</td>
<td>3 801</td>
<td>4 295</td>
</tr>
<tr>
<td>Europe Eastern (includes Russian Fed.)</td>
<td>3 185</td>
<td>3 689</td>
<td>4 138</td>
</tr>
<tr>
<td>Latin America</td>
<td>2 107</td>
<td>2 167</td>
<td>2 356</td>
</tr>
<tr>
<td>North America</td>
<td>5 860</td>
<td>6 041</td>
<td>5 110</td>
</tr>
<tr>
<td>Oceania</td>
<td>1 656</td>
<td>2 094</td>
<td>2 246</td>
</tr>
</tbody>
</table>

Thousands of metric tonnes
One consequence of this was the closure of some Western smelters, and cutbacks in others, even though smelters in the C.I.S. were generally regarded as being less efficient. During the decade 1995 to 2004 prices on the London Metals Exchange have ranged from $US 1325 per tonne in 1999 to $US 1806 in 1995, with an average over the whole period of $US 1506.

During the decade 1983–1993, the price of magnesium was relatively constant and, on average, was around twice that of aluminium on the basis of weight. However, the volatility in the price of aluminium has meant that the magnesium:aluminium price ratio has at times ranged from 2.5 to just below 1.5, the level at which magnesium becomes competitive on a volumetric basis. Direct cost comparisons of aluminium with steel are also difficult to make because of density differences, although it may be noted that aluminium alloy sheet is normally three to four times more expensive than mild steel sheet of the same weight. This price differential is reduced with products made from recycled metal, or those for which volume or area are prime considerations.

In most countries, aluminium is used in five major areas: building and construction; containers and packaging; transportation; electrical conductors; machinery and equipment. Patterns vary widely from country to country depending on levels of industrialisation and economic wealth. As examples, Fig. 1.5 compares the consumption of aluminium in 2002 for the current two major users of aluminium, the United States and China. In the United States, transportation (31.6%) had become the major user followed by packaging (20.7%) whereas the reverse was true a decade ago. In China, the market is heavily weighted to building and construction (33%) whereas packaging only accounts for 8%. Transportation consumed 24% of the aluminium used in

Fig. 1.5  Outlets for the consumption of aluminium in (a) United States and (b) China in the year 2002. (from Hunt, W.H., JOM, 56, No. 9, 21, 2004).
China and it is interesting to note that, whereas in the United States in 2003 there were around 800 automobiles for each 1000 people, this figure was less than 20 in China. The largest potential for further growth in the use of aluminium does appear to reside in the automobile which, in the United States has risen from an average of 8 kg per vehicle in 1971 to 90 kg in 1994 and 130 kg in 2004. In Europe, the amount of aluminium in each vehicle was about 100 kg in 2000 and is predicted to increase to 150 kg in 2005 and 200 kg by 2015.

As shown in Fig. 1.4, global production of magnesium was relatively constant at about 250 000 tonnes per annum for the more than two decades from around 1970 until the late 1990s. During this period, some three-quarters of magnesium metal was consumed as alloying additions to aluminium (~55%), as a desulphurising agent for steels (~15%), and to produce nodular cast iron (~6%). Less than 20% was actually used to produce magnesium alloys, mostly as die castings for the aerospace and general transport industries. However, during the decade 1994 to 2004 the increase in the production of magnesium die castings averaged 16% per annum and the amount used for this purpose was estimated to have risen to 152 000 tonnes in 2004. This was approximately one-third of all magnesium produced. Global demand for magnesium, excluding China and Russia, rose by 8% from 2003 to 2004 to reach 410 000 tonnes and is expected to exceed 500 000 tonnes by 2010. To date, wrought magnesium alloys have accounted for relatively little use of this metal because the hexagonal crystal structure makes them less amenable to hot or cold working.

Titanium was not produced in quantity until the late 1940s when its relatively low density and high melting point (1678 °C) made it uniquely attractive as a potential replacement for aluminium for the skin and structure of high-speed aircraft subjected to aerodynamic heating. Liberal military funding was provided in the decade 1947–57 and one of the major metallurgical investigations of all time was made of titanium and its alloys. It is estimated that $400 million was spent in the United States during this period and one firm examined more than 3000 alloys. One disappointing result was that titanium alloys showed relatively poor creep properties bearing in mind their very high melting points. This factor, together with a sudden change in emphasis from manned aircraft to guided weapons, led to a slump in interest in titanium in 1957–58. Since then, selection of titanium alloys for engineering uses has been made on the more rational bases of cost-effectiveness and the uniqueness of certain properties. The high specific strength of titanium alloys when compared with other light alloys, steels, and nickel alloys is apparent in Fig. 1.6. The fact that this advantage is maintained to around 500 °C has led to the universal acceptance of certain titanium alloys for critical gas turbine components. Titanium alloys have also found increasing for critical structural components in aircraft including forged undercarriages, engine mountings and high strength fasteners. Traditionally, aerospace applications have accounted for as much as 75% of titanium mill products, but this level has now fallen below 50%. Much of the remainder is finding increasing applications in the chemical
and power generation industries where the outstanding corrosion resistance of titanium is the key factor.

World production of titanium as sponge (Section 1.4) peaked at around 180 000 tonnes in the mid-1980s when the Cold War had a dominant influence, and had halved by the year 2002. At that time, worldwide production of titanium and titanium alloys mill products was 57 000 tonnes, and is predicted to reach 80 000 tonnes in 2010. In 2002, The USA (28%), CIS (26%) and Japan (25%) supplied nearly 80% of all the world’s titanium mill products. China (7%) was then the fifth largest producer and plans are being implemented to double its output from 7 000 tonnes in 2003 to 14 000 by the year 2010.

1.1.4 Recycling

Due to the realisation that the supply of minerals is finite, much attention is now being paid to recycling as a means of saving metals, as well as reducing both the amount of energy and the output of greenhouse gases involved in their extraction. Based on known reserves, Fig. 1.7 shows one estimate of the dramatic effect that recycling rate may have on the remaining supply of several metals. With aluminium, for example, it has been predicted that a recycling rate of 50% would allow the reserves to last for about 320 years, whereas a rate of 80% would sustain these reserves for more than 800 years. In the United states, Japan and much of Europe, some restrictions have been introduced that require
Fig. 1.7 Estimated relationship between remaining years of supply of some metals and recycling rates based on known reserves of minerals (courtesy of J. Rankin and T. Norgate, Commonwealth Scientific and Industrial Research Organization, Minerals Division, Melbourne).

The incentive for recycling the light metals is particularly strong because their initial costs of production are relatively high. For aluminium, remelting of scrap requires only about 5% of the energy needed to extract the same weight of primary metal from its ore bauxite. Currently the ratio of secondary (scrap) to primary aluminium is around 30% which is well below that recovered from steel or copper. However, as shown in Table 1.4, the relatively long life span of some aluminium-containing products can limit the supply of used metal.

Table 1.4 Global estimates of service lives and recycling rates for products made from aluminium (courtesy International Aluminium Institute, London)

<table>
<thead>
<tr>
<th>Major End Markets</th>
<th>Average Product Life</th>
<th>Average Recycling Rate</th>
</tr>
</thead>
<tbody>
<tr>
<td>Building and construction</td>
<td>25–50</td>
<td>80–85</td>
</tr>
<tr>
<td>Transportation – cars</td>
<td>10–15</td>
<td>90–95</td>
</tr>
<tr>
<td>Transportation – aerospace</td>
<td>15–25</td>
<td>90–95</td>
</tr>
<tr>
<td>Transportation – marine</td>
<td>15–40</td>
<td>40–90</td>
</tr>
<tr>
<td>Transportation – trucks, buses, rail</td>
<td>15–30</td>
<td>50–90</td>
</tr>
<tr>
<td>Engineering – machinery</td>
<td>10–30</td>
<td>30–90</td>
</tr>
<tr>
<td>Engineering – electrical</td>
<td>10–50</td>
<td>40–80</td>
</tr>
<tr>
<td>Packaging – cans</td>
<td>0.1–1.0</td>
<td>30–90</td>
</tr>
<tr>
<td>Packaging – foil</td>
<td>0.1–1.0</td>
<td>20–90</td>
</tr>
</tbody>
</table>
The light alloys in general do present a special problem because they cannot be refined, i.e. alloying elements cannot be extracted or removed. Consequently, unless alloys for particular products are segregated and confined to a closed circuit, remelting tends to downgrade them. With aluminium alloys, the remelted general scrap is used mainly for foundry castings which, in turn, are limited in the amount they can absorb. Thus there is a need to accommodate more secondary aluminium alloys into cast billets that are used to produce wrought materials. In this regard, aluminium provides by far the highest value of any recyclable packaging material and, as an example, the efficient collection of all-aluminium alloy beverage cans to produce canstock is essential for the competitive success of this product (Section 3.6.5). Worldwide, four out of every five beverage cans are made from aluminium alloys and the current recycling rate averages 55%. This figure now exceeds 70% in the United States and is as high as 90% in Sweden and Switzerland.

A schematic that seeks to represent the complex interactions involved in the production, use, and recycling of aluminium is shown in Fig. 1.8. The automotive industry is now the second largest provider of aluminium alloy scrap although sorting from other materials after discarded vehicles have been shredded into small pieces presents technical challenges. Currently flotation methods are commonly used. In the European Union in 2002, some 90% of automotive aluminium alloys were recovered which compares with 80% from discarded building products and electrical appliances, and only 40% from packaging. The aluminium industry in the United States has a vision of 100% recycling of aluminium alloys by the year 2020. Such an ambitious target will require major technological advances, notably the development of a low-cost process for metal purification so that the recycled scrap can become a source of primary aluminium.

In 2004, approximately half the world’s output of magnesium was used as an alloying addition to aluminium and much of this is recycled along with these alloys. Otherwise, the recycling of magnesium alloy scrap is less advanced than for aluminium because of magnesium’s high reactivity with oxygen and nitrogen, and problems with contamination by metals such as copper and nickel that adversely affect corrosion resistance. Clean and sorted magnesium alloy scrap arising from sprues, risers and other discards from casting processes may be recycled within individual foundries. Other scrap is usually remelted under a refining flux that consists of a mixture of alkali and alkaline earth metal chlorides and fluorides. To maximize metal recovery, and reduce toxic gases, it is desirable to remove surface coatings such as paint, lacquer and oil which adds to the cost of recycling. Because of magnesium’s high vapour pressure and relatively low boiling point (1090 °C) distillation may offer a promising alternative solution to the problem of contamination from scrap. It may also provide the opportunity to produce pure magnesium metal from alloys which is much less feasible with aluminium or titanium (boiling points 2520 °C and 3289 °C respectively).
Less attention has been directed to the recycling of titanium alloys despite the fact that, on average, only 0.4 kg of each 1.3 kg of titanium metal sponge ends up in the finished product. High quality titanium alloy scrap can be remelted within a closed circuit to make ingots or slabs. Other scrap, including sponge, can be used to produce ferro-titanium for adding to speciality steels.

Fig. 1.8 Schematic representation of the interactions in aluminium production, uses and recycling (courtesy A. Gesing).
Although aluminium is now the second most used metal, it is a comparative newcomer among the common metals because of the difficulty in extracting it from its ores. Unlike iron, for example, it combines so strongly with oxygen that it cannot be reduced with carbon. An impure form of aluminium was first isolated in 1809 in England by Sir Humphry Davy which he produced from alum, its bisulphate salt. He called this new metal “aluminum”, which is the name still used in the United States, whereas it is now known as “aluminium” in Europe and most other countries.

The first commercial preparation of aluminium occurred in France in 1855 when H. Sainte-Claire Deville reduced aluminium chloride with sodium. As is so often the case, the potential military applications of this new metal led to government support because Napoleon the Third foresaw its use in lightweight body armour. During the period 1855–59, the price of aluminium per kg fell from over $US500 to $US40 but all the Emperor is reported to have received were some decorative military helmets, an aluminium dinner set and some aluminium toys for the children of the Imperial Court. The aluminium produced by Sainte-Claire Deville’s process was less than 95% pure and it proved to be more expensive than gold at that time.

Independent discoveries in 1886 by Hall in the United States and Héroult in France led to the development of an economic method for the electrolytic extraction of relatively high-purity aluminium which remains the basis for production today. By 1888 the price had fallen to less than $US4 per kg and in recent times it has varied between $US1 and $US2 per kg.

Aluminium is extracted from bauxite which was discovered by the French chemist P. Berthier, and named after the town of Les Baux in Provence, Southern France, where the ore was first mined. Bauxite is the end product of millions of years of surface weathering of aluminium silicates (e.g. feldspars) and clay minerals, usually in tropical locations. The principal aluminium-bearing minerals in bauxite exist as several forms of hydrated aluminium oxide, notably gibbsite ($\text{Al}_2\text{O}_3\cdot3\text{H}_2\text{O}$), which is also known as hydragallite or trihydrate, and boehmite ($\text{Al}_2\text{O}_3\cdot\text{H}_2\text{O}$) also known as monohydrate. The actual chemical composition varies with location and the geology of each deposit. Bauxite that is mined simply by open cut methods and the largest known reserves exist in Australia which, in 2004, produced 38% (55.6 million tonnes) of the world’s supply followed by Guyana 10.6%, Jamaica 9.25%, Brazil 9.0% and China 8.6%.

Typically ore bodies contain 30–60% hydrated $\text{Al}_2\text{O}_3$ together with impurities comprising iron oxides, silica and titania. High-grade bauxites with low silica contents are expected to be depleted in 20–30 years time. When it becomes necessary to use grades with higher silica contents, the ore will first need to be processed to remove this impurity which will introduce a cost penalty. One method that has been developed in Russia is called thermochemical alkaline...
conditioning. This involves roasting the bauxite at high temperatures to convert the two most important silica-bearing minerals, kaolinite and quartz, into products that can be dissolved in caustic soda.

Immense amounts of aluminium are also present in clays, shales and other minerals and the amphoteric nature of aluminium provides the opportunity to use acid as well as alkaline processes for its recovery. As one example, some attention has been given to acid extraction of alumina from kaolinite which is widely distributed as a clay mineral and is a major constituent of the ash in coal. The minerals, nepheline, \( \text{Na}_3\text{K(AlSiO}_4\text{)}_4 \), and alunite, \( \text{KAl}_3\text{(SO}_4\text{)}_2\text{(OH)}_6 \), are processed commercially in the Commonwealth of Independent States in plants located in regions remote from sources of bauxite. However, alumina obtained from these and other alternative sources is 1.5–2.5 times more costly than that produced from the Bayer process which is described below.

### 1.2.1 Bayer process for alumina recovery

The Bayer process was developed and patented by Karl Josef Bayer in Austria in 1888 and essentially involves digesting crushed bauxite in strong sodium hydroxide solutions at temperatures up to 240 °C. Most of the alumina is dissolved leaving an insoluble residue known as ‘red mud’ which mainly comprises iron oxides and silica and is removed by filtration. The particular concentration of sodium hydroxide as well as the temperature and pressure of the operation are optimized according to the nature of the bauxite ore, notably the respective proportions of the different forms of alumina (α, β, or γ). This first stage of the Bayer process can be expressed by the equation:

\[
\text{Al}_2\text{O}_3 \cdot x\text{H}_2\text{O} + 2\text{NaOH} \rightarrow 2\text{NaAlO}_2 + (x + 1)\text{H}_2\text{O}.
\]

Subsequently, in the second stage, conditions are adjusted so that the reaction is reversed. This is referred to as the decomposition stage:

\[
2\text{NaAlO}_2 + 2\text{H}_2\text{O} \rightarrow 2\text{NaOH} + \text{Al}_2\text{O}_3 \cdot 3\text{H}_2\text{O}.
\]

The reverse reaction is achieved by cooling the liquor and seeding with crystals of the trihydrate, \( \text{Al}_2\text{O}_3 \cdot 3\text{H}_2\text{O} \), to promote precipitation of this compound as fine particles rather than in a gelatinous form. Decomposition is commonly carried out at around 50 °C in slowly stirred vessels and may require up to 30 h to complete. The trihydrate is removed and washed, with the sodium hydroxide liquor being recycled back to the digestors.

Alumina is then produced by calcining the trihydrate in rotary kilns or, more recently, fluidized beds. Calcination occurs in two stages with most of the water of crystallization being removed in the temperature range 400–600 °C. This produces alumina in the more chemically active γ-form which further heating to temperatures as high as 1200 °C converts partly or completely to relatively inert....
α-alumina. Each form has different physical characteristics and individual aluminium smelters may specify differing mixtures of α- and γ-alumina. Typically the Bayer process produces smelter grade alumina in the range 99.3–99.7% Al₂O₃.

In 2004, the total capacity of alumina refineries worldwide was 65 million tonnes per annum. Approximately one third of the world’s alumina was produced in Australia which had three largest refineries ranging in size from 3.25 to 3.75 million tonnes, one of which was being expanded to a capacity of 4 million tonnes per annum. In most countries, alumina is costs for US$180–200 per tonne to produce.

### 1.2.2 Production of aluminium by the Hall–Héroult Process

Alumina has a high melting point (2040 °C) and is a poor conductor of electricity. The key to the successful production of aluminium lies in dissolving the oxide in molten cryolite (Na₃AlF₆) and a typical electrolyte contains 80–90% of this compound and 2–8% of alumina, together with additives such as AlF₃ and CaF₂. Cryolite was first obtained from relatively inaccessible sources in Greenland but is now made synthetically.

An electrolytic reduction cell (known as a pot) consists essentially of baked carbon anodes that are consumed and require regular replacement, the molten cryolite–alumina electrolyte, a pool of liquid aluminium, a carbon-lined container to hold the metal and electrolyte, and a gas collection system to prevent fumes from the cell escaping into the atmosphere (Fig. 1.9). There are also alumina feeders that are activated intermittently under some form of automatic control. A typical modern cell is operated at around 950 °C and takes up to 500 kA at an anode current density around 0.7 A cm⁻². The anode and cathode
are separated by 4–5 cm and there is a voltage drop of approximately 4.5 V across each cell. The cell is operated so that the carbon side-linings are protected with a layer of frozen cryolite and the upper surface of the bath is covered with a crust of alumina. The molten aluminium is siphoned out regularly to be cast into ingots and alumina is replenished as required. The largest and most productive cells operate at a current efficiency of around 95% and have a daily output of aluminium close to 4000 kg. Typically, some 300 cells are connected in series to make up a potline (Fig. 1.10).

The exact mechanism for the electrolytic reaction in a cell remains uncertain but it is probable that the current-carrying ions are Na\(^{+}\), AlF\(_4\)^{−}, AlF\(_6\)^{3−} and one or more ternary complex ions such as AlOF\(_3\)^{2−}. At the cathode it is considered that the fluorosaluminate anions are discharged via a charge transfer at the cathode interface to produce aluminium metal and F\(^{−}\) ions while, at the anode, the oxofluoro-aluminate ions dissociate to liberate oxygen which forms CO\(_2\). The overall reaction can be written simply as follows:

\[
2\text{Al}_2\text{O}_3 + 3\text{C} \rightarrow 4\text{Al} + 3\text{CO}_2.
\]

Figure 1.11 shows a flow diagram for the raw materials needed to produce 1 tonne of aluminium. Commonly some 3.5–4 tonnes of bauxite are needed from which 2 tonnes of alumina are extracted that, in turn, yield 1 tonne of aluminium. Significant quantities of other materials, such as 0.4 tonne of carbon, are also consumed. However, the most critical factor is the consumption of electricity which, despite continual refinements to the process, still required a world average of 15 700 kWh for each tonne of aluminium produced from
alumina in the year 2004. The best performance reported for that year was an electricity consumption of 13 300 kWh per tonne. These values compare with 28 000 kWh per tonne needed shortly after the Hall-Héroult process was first commercialised late in the 19th century, and the theoretical requirement which is about 6 500 kWh. By 2004, it was estimated that the production of aluminium worldwide consumed 266 000 GWh of electricity, about half of which was generated by hydropower.

Of the total voltage drop of 4.5 V across a modern cell, only 1.2 V represents the decomposition potential or free energy of the reaction associated with the formation of molten aluminium at the cathode. The largest component of the voltage drop arises from the electrical resistance of the electrolyte in the space between the electrodes and this amounts to around 1.7 V, or 35–40% of the total. Efficiency can be increased if the anode-cathode distance is reduced and this aspect has been one focus of recent changes in cell design. A modification that shows promise is to coat the cathode with titanium diboride which has the property of being readily wetted by molten aluminium. This results in the formation of a thinner, more stable film of aluminium that can be drained away into a central sump if a sloped cathode is used (Fig. 1.12). Reductions in anode–cathode spacing from the normal 4–6 cm down to 1–2 cm have been claimed permitting a decrease of 1–1.5 V in cell voltage. Predictions on cell performance suggest that electrical consumption can be reduced to an average of 12 500 kWh per tonne of aluminium produced.

Fig. 1.13 summarizes one calculation of the total energy consumed during all stages in the production of the light metals, as well as that for copper, zinc...
and steel. To produce one tonne of primary aluminium from bauxite in the ground was estimated to require between 70 000 and 75 000 kWh (thermal), where the total energy required has been converted back to an equivalent amount of fossil fuel by assuming 1 kWh (electrical) = 3 kWh (thermal). This reduces to 30 000 kWh (thermal) per unit volume of aluminium, but is still much greater than the estimated 13 000–16 000 kWh (thermal) of energy required to produce one tonne of steel from iron ore in the ground to finished product.
1.2.3 Alternative methods for producing aluminium

Because of the large disparity between the theoretical and actual requirements for electrical energy to produce each tonne of aluminium, it is to be expected that alternative methods of production would have been investigated. One example has been a chloride-based smelting process developed by Alcoa which commenced operation in the United States in 1976 with an initial capacity of 13 500 tonnes of aluminium per year and the potential to achieve a 30% cost saving. This process also used alumina as a starting material which was combined with chlorine in a reactor to produce AlCl₃. This chloride served as an electrolyte in a closed cell to produce aluminium and chlorine, the latter being recycled back into the reactor. The process had the advantage of being continuous but the provision of materials of construction that could resist attack by chlorine over long periods of time proved to be difficult. This factor, together with improvements in efficiencies in conventional electrolysis, led to the process being discontinued in 1985.

Several companies have been investigating carbothermic methods for producing aluminium. One process involves mixing aluminium ore with coking coal to form briquettes, which are then reduced in stages in a type of blast furnace operating at temperatures ranging from 500 to 2100 °C. The molten metal product comprises aluminium combined with iron and silicon which is scrubbed and absorbed by a spray of molten lead at the bottom of the furnace. Since aluminium and lead are immiscible, the lighter aluminium rises to the surface where it can be skimmed off. Further purification of the aluminium is required. Although overall cost savings have been predicted, no commercially viable process has so far eventuated.

1.3 PRODUCTION OF MAGNESIUM

One of the novel explanations for the disappearance of the dinosaurs is the Chinese theory that this was due to a magnesium deficiency which had an adverse effect on the strength of egg shells thereby preventing reproduction. Today it is recognized that human beings require a daily intake of 300–400 mg of magnesium which means that some 600 000 tonnes should be ingested annually throughout the world! Until recently, this was more than double the amount of magnesium metal actually produced each year.

In 1808 Sir Humphry Davy established that magnesium oxide was the oxide of a newly recognised element. Magnesium metal was first isolated in 1828 by the Frenchman Antoine-Alexander Bussy who fused MgCl₂ with metallic potassium. The first production of magnesium by the electrolytic reduction of the chloride was accomplished by Michael Faraday in 1833. Commercial production commenced in Paris in the middle of the 19th century but had reached only some 10 tonnes per annum by the year 1890. Output increased to 3000 tonnes during the last year of the First World War, fell again afterwards and
then rose to around 300 000 tonnes per annum late in the Second World War. In 2004 annual production exceeded 400 000 tonnes.

Magnesium compounds are found in abundance in solid mineral deposits in the earth’s crust and in solution in the oceans and salt lakes. The most common surface minerals are the carbonates dolomite (MgCO$_3$·CaCO$_3$) and magnesite (MgCO$_3$). The mineral brucite (MgO·H$_2$O) is somewhat rarer, as are the chlorides of which carnallite (MgO·KCl·6H$_2$O) is one example. Concentrated aqueous solutions in the form of brine deposits occur at several places in the world including the Dead Sea in Israel and the Great Salt Lake in Utah, United States where a total of close to 80 000 tonnes is currently produced annually. Virtually unlimited reserves are present in the oceans which contain 0.13% magnesium and, until recently, sea water provided more than 80% of the world’s supply of this metal which was extracted by the electrolytic reduction of MgCl$_2$. During the 1940’s the largest sea water plant was constructed in the United States at Freeport, Texas, and production peaked at 120 000 tonnes in the 1970s. Output then declined during the next two decades and the plant was closed in 1998 following severe storm damage. At present the extraction of magnesium from sea water has become uncompetitive for reasons that are explained below.

Two processes have been developed to produce magnesium by the direct reduction of dolomite by ferrosilicon at high temperatures. One is the Pidgeon process originating from Canada in 1941 in which this reaction is carried out in the solid state. Until recently it was only economic in rare conditions where there was a natural site advantage. The other is the Magnétherm process that was developed in France and which operates at much higher temperatures so that the reaction mixture is liquid. This process is also producing magnesium at prices that are currently uncompetitive with metal produced in China by the Pidgeon process.

### 1.3.1 Electrolytic extraction of magnesium

Two types of electrolytic processes have been used to produce magnesium that differ in the degree of hydration of the MgCl$_2$ and in cell design. One was pioneered by IG Fabenindustrie in Germany in 1928 and was adopted later by the Norwegian company Norsk Hydro when it was a major European producer of this metal. Known as the IG process, it uses dry MgO derived from minerals in sea water which is briquetted with a reducing agent, e.g. powdered coal, and MgCl$_2$ solution. The briquettes are lightly calcined and then chlorinated at around 1100 °C to produce anhydrous MgCl$_2$, which is fed directly into the electrolytic cells operating at 740 °C. Other chloride compounds such as NaCl and CaCl$_2$ are added to improve electrical conductivity and to change the viscosity and density of the electrolyte. Each cell uses graphite anodes that are slowly consumed and cast steel cathodes that are suspended opposite one another. Typically the cells operate at 5–7 V and currents may now exceed 200 000 A. Magnesium is deposited on the cathodes as droplets which rise to
the surface of the electrolyte, whereas chlorine is liberated at the anodes and is recycled to produce the initial MgCl₂ cell feedstock.

The second electrolytic process for extracting magnesium from sea water was developed by the Dow Chemical Company for use at the Freeport plant mentioned above. Magnesium in MgCl₂ was precipitated as the hydroxide by the addition of lime and then dissolved in HCl. This solution was then concentrated and dried although the process stopped short of complete dehydration of the MgCl₂ which was then available as the cell electrolyte. In contrast the IG-Norsk Hydro process, the cells required external heat with the steel cell box serving as the cathode. These cells operated at 6–7 V and a current of 90 000 A.

The energy consumed per kg of magnesium produced was around 12.5 kWh for the Norsk Hydro cell and 17.5 kWh for the Dow cell. However, each process required the additional consumption of approximately 15 kWh of energy per kg for preparation of MgCl₂ cell feed.

Two new processes for extracting the basic feedstock anhydrous MgCl₂ in a high-purity form for use in electrolytic cells have been developed in Canada and Australia, although neither has reached commercial production. In Canada, the Magnola process uses tailings from asbestos mines to take advantage of magnesium silicate contained in serpentine ore. The tailings are leached in strong HCl in a novel procedure to produce a solution of MgCl₂ which is purified by adjusting the pH. Ion exchange techniques are then used to generate concentrated, high-purity brine that is dehydrated for use in an electrolytic cell. The Australian process was developed to exploit magnesite (MgCO₃) mined from a huge surface deposit in Queensland that is estimated to contain 260 million tonnes of high-grade ore. MgCl₂ is leached from the magnesite with HCl and glycol is added to the solution, after which water is removed by distillation. Magnesium chloride hexammoniate is then formed by sparging with ammonia. Final calcining produces a relatively low cost, high-purity MgCl₂, and the solvent and ammonia are recycled.

### 1.3.2 Thermic processes

Production of magnesium by direct thermal reduction of calcined dolomite with ferrosilicon proceeds according to the simplified equation:

$$2\text{CaO} \cdot \text{MgO} + \text{Si} \rightarrow 2 \text{Mg} + (\text{CaO})_2\text{SiO}_2.$$ 

In the Pidgeon process, briquettes of the reactants are prepared and loaded in amounts of around 150 kg each into a number of tubular steel retorts that are typically 250–300 mm in diameter and 3 m long. The retorts are then evacuated to a pressure of below 0.1 torr and externally heated to a temperature in the range 1150–1200 °C, usually by burning coal. Magnesium forms as a vapour that condenses on removable water-cooled sleeves at the ends of the
retorts that are located outside the furnace. Approximately 1.1 tonnes of ferrosilicon is consumed for each tonne of magnesium that is produced. Advantages of the Pidgeon process are the relatively low capital cost and the less stringent requirement that is placed on the purity of the raw materials. Major deficiencies are that it is a labour-intensive, batch process which only produces around 20 kg of magnesium from each retort, and requires a lengthy cycle time of around 8 h. The retorts must then be emptied, cleaned and recharged in conditions that are usually dusty and unpleasant. The Pidgeon process has been widely adopted in China where labour costs are low relative to Western nations, and there are readily available supplies of low cost ferrosilicon and anthracite. Several hundred plants of varying sizes have been constructed throughout China which, in 2004, supplied more that half the world’s magnesium. Prices have ranged between $US 1200 and $US 1500 per tonne.

The Magnétherm process employs an electric arc furnace operating at around 1550 °C and with an internal pressure of 10–15 torr. Because the reaction takes place in the liquid phase, the time required for its completion is less than that needed for the Pidgeon process. The furnace may be charged continuously and discharged at regular intervals, and alumina or bauxite is added to the dolomite/ferrosilicon charge which keeps the reaction product, dicalcium silicate, molten so that it can be tapped as a slag. Magnesium is again produced as a vapour which is solidified in an external condenser. Batch sizes may be as high as 11 000 kg and plants have operated in France, Japan, the United States and in the former Yugoslavia.

Alternative thermic techniques have been proposed, although none is currently operating commercially. One idea was to use a plasma arc furnace in which pelletized MgO and coke are fed into a pre-melted MgO, CaO, Al₂O₃ slag. The high energy density of the plasma, and the fact that high temperatures (e.g. 1500 °C) are generated at the surface of the slag where the silicothermic reaction occurs, allows it to be sustained at normal atmospheric pressure. This advantage, combined with the efficient and near total silicon consumption were claimed to enhance economic competitiveness of the thermic route to magnesium production. However, this development occurred before use of the Pidgeon process was greatly expanded in China.

1.4 PRODUCTION OF TITANIUM

The existence of titanium was first recognized in 1791 by William McGregor, an English clergyman and amateur mineralogist, who detected the oxide of an unknown element in local ilmenite sand (FeO·TiO₂). A similar observation was made in 1795 by a German chemist, Martin Klaproth, who examined the mineral rutile (TiO₂) and he named the element titanium after the mythological first sons of the earth, the Titans. An impure sample of titanium was first isolated in 1825 but it was not produced in any quantity until 1937 when Kroll, in Luxembourg, reacted TiCl₄ with molten magnesium under an atmosphere of
argon. This opened the way to the industrial exploitation of titanium and the essential features of the process are as follows.

1. Briquette TiO₂ with coke and tar and chlorinate at 800 °C to promote the reaction: TiO₂ + 2Cl₂ + 2C → TiCl₄ + 2CO.
2. Purify TiCl₄ by fractional distillation.
3. Reduce TiCl₄ by molten magnesium or sodium under an argon atmosphere, one reaction being: TiCl₄ + 2Mg → Ti + 2MgCl₂.

Titanium forms as an impure sponge around the walls of the reduction vessel and is removed periodically. The sponge produced by reacting with magnesium must be purified by leaching with dilute HCl and/or distilling off the surplus MgCl₂ and magnesium. The use of sodium has the advantages that leaching is more efficient and the titanium sponge is granular, making it easier to compact for the subsequent melting process.

The mineral rutile (TiO₂) is the most convenient source of titanium and is found mainly in beach sands along the eastern coast of Australia, India, Mexico and in estuaries in Sierra Leone. Most titanium metal is extracted from rutile although the much more plentiful, but more complex mineral ilmenite (FeO·TiO₂) will become the major source in the future. Ilmenite is available in a number of countries with particularly large deposits being found in China and Russia. It is interesting to note that some 93% of the world’s production of titanium minerals is processed into titanium oxide and amounts to more than 4.5 million tonnes annually. This compound is used as a white pigment in paint, in which it replaced white lead, as well as in papermaking, printing ink, ceramics and plastics. Titanium metal currently accounts for only some 3% of minerals production.

The controlling factor in refining titanium sponge is the metal’s high reactivity with other elements, notably its affinity for oxygen, nitrogen, hydrogen, and carbon. As shown in Table 1.5, the solubility of these interstitial elements in titanium is greater by several orders of magnitude than in other commonly used metals. Since quite small amounts of these elements adversely affect the ductility and toughness of titanium, it is clearly impossible to melt in air or in a normal crucible because the metal will both absorb gases and react with any known oxide or carbide refractory. Accordingly, a radically new method of melting had to be devised leading to what is known as the consumable-electrode arc furnace (Fig. 1.14a).

Melting is carried out in a copper crucible cooled internally by circulating water or a liquid sodium-potassium eutectic. Heat is generated by a direct current arc that is struck between an electrode of titanium to be melted and a starting slug of this material contained in the crucible. An advantage of the liquid alloy coolant is that it does not react with titanium should the electric arc perforate the crucible, whereas water and steam can cause an explosion. The electrode is usually made from welded blocks of compressed titanium sponge into which alloying elements are incorporated in powder form (Fig. 1.14b).
**Chapter 1 THE LIGHT METALS**

Table 1.5  Solubility at room temperature of O, N, C and H in titanium, iron and aluminium (from Morton, P.H., The Contribution of Physical Metallurgy to Engineering Practice, Rosenhain Centenary Conference, The Royal Society, 1976)

<table>
<thead>
<tr>
<th>Interstitial element</th>
<th>Oxygen</th>
<th>Nitrogen</th>
<th>Carbon</th>
<th>Hydrogen</th>
</tr>
</thead>
<tbody>
<tr>
<td>Titanium</td>
<td>14.5 wt%</td>
<td>~20 wt%</td>
<td>0.5 wt%</td>
<td>~100 ppm</td>
</tr>
<tr>
<td>Iron</td>
<td>~1 ppm</td>
<td>&lt;5 ppm</td>
<td>100 ppm</td>
<td>&lt;1 ppm</td>
</tr>
<tr>
<td>Aluminium</td>
<td>&lt;1 ppm</td>
<td>&lt;1 ppm</td>
<td>&lt;1 ppm</td>
<td>&lt;1 ppm</td>
</tr>
</tbody>
</table>

ppm = parts per million

Fig. 1.14 (a) Consumable-electrode arc furnace for melting and refining titanium; (b) consumable electrode made by welding together blocks of compressed titanium sponge (courtesy T.W. Farthing).

arrangement is encased in a vessel that can be evacuated and into which an inert gas such as argon is introduced. The capacity of the furnace is increased by using a retractable hearth and ingots are now commonly produced with diameters of 700–1200 mm and weights of 3–15 tonnes. Double and sometimes triple melting is carried out to improve the homogeneity of the ingots.

Because titanium alloys are used for critical rotating parts of gas turbine engines, special attention has been given to the elimination of inclusions in castings which may serve as sites for the initiation of fatigue cracks in components.
such as forged compressor discs. Sponge quality may be improved by resorting to vacuum distillation and more costly cold hearth melting furnaces have been tried which offer longer residence times for the molten alloy so that there is a greater opportunity for impurities to be dissolved or removed. In these furnaces, melting is carried out either in a high vacuum using electron beam guns as the heat source, or by a plasma torch operating in a controlled atmosphere.

Overall, the energy consumed in making pure titanium sponge is greater than that required for producing any other common metals in ingot form (Fig. 1.13). It is about 70% higher than that required for extracting an equal weight of aluminium and as much as 14 times the energy needed to produce steel. Production by the carbothermic or plasma reduction of titanium compounds has been shown to be feasible on a laboratory scale but other technical problems have so far prevented commercial exploitation. Similarly, extraction of titanium by electrolysis of fused salts has been investigated for many years with little success until a recent unexpected discovery was made in England. This has resulted in the new Fray, Farthing and Chen (FFC) process which is currently being evaluated for commercial production. This process involves using powder compacts of TiO₂ as the cathode and graphite as the anode in a bath of molten CaCl₂. Through careful selection of voltage, the favoured cathodic reaction is the ionization of O₂ rather than the deposition of Ca. The actual reactions are:

\[
\text{Cathode: } \text{TiO}_2 + 4e^- = \text{Ti} + 2 \text{O}^{2-} \\
\text{Anode: } \text{C} + x \text{O}^{2-} = \text{CO}_x + 2x e^- 
\]

A key factor in this process is that TiO₂, which is normally an insulator, becomes conducting once a small amount of O₂ is removed at the beginning of electrolysis. As the reaction proceeds, the porous oxide powder is converted to a high-purity, metallic sponge that can be melted and alloyed without the need of further refining, or used directly to make sintered powder products. Advantages of the FFC process over the traditional Kroll method are (i) all the electrolytic cell materials TiO₂, graphite and CaCl₂ are inexpensive, (ii) the titanium sponge is produced at a faster rate, and (iii) continuous production is possible. If the FFC process can be successfully scaled up, it is thought to have the potential to halve the cost of titanium which would significantly alter this metal’s competitive position with respect, for example, to stainless steels.

**FURTHER READING**


CHAPTER I  THE LIGHT METALS


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