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Part 1 INTRODUCTION TO MINERAL PROCESSING

Unit 1 Mineral and Ores

Minerals

The forms, in which metals are found in the crust of the earth and as seabed deposits, depend on their reactivity with their environment, particularly with oxygen, sulphur, and carbon dioxide. Gold and the platinum metals are found principally in the native or metallic form. Silver, copper, and mercury are found native, as well as in the form of sulphides, carbonates, and chlorides. The more reactive metals are always in compound form, such as the oxides and sulphides of iron and the oxides and silicates of aluminium and beryllium. The naturally occurring compounds are known as minerals, most of which have been given names according to their composition (e.g. galena--lead sulphide, PbS ; sphalerite--zinc sulphide, ZnS ; cassiterite-- tin oxide, SnO_2).

Minerals by definition are natural inorganic substances possessing definite chemical compositions and atomic structures. Some flexibility, however, is allowed in this definition. Many minerals exhibit isomorphism, where substitution of atoms within the crystal structure by similar atoms takes place without affecting the atomic structure. The mineral olivine, for example, has the chemical composition $(\text{Mg}, \text{Fe})_2\text{SiO}_4$, but the ratio of Mg atoms to Fe atoms varies in different olivines. The total number of Mg and Fe atoms in all olivines, however, has the same ratio to that of the Si and O atoms. Minerals can also exhibit polymorphism, different minerals having the same chemical composition, but markedly different physical properties due to a difference in atomic structure. Thus the two minerals graphite and diamond have exactly the same composition, being composed entirely of carbon atoms, but have widely different properties due to the arrangement of the carbon atoms within the crystal lattice. The term "mineral" is often used in a much more extended sense to include anything of economic value which is extracted from the earth. Thus coal, chalk, clay, and granite do not come within the definition of a mineral, although details of their production are usually included in national figures for mineral production. Such materials are, in fact, rocks, which are not homogeneous in chemical and physical composition, as are minerals, but generally consist of a variety of minerals and form large parts of the earth's crust. Granite, for instance, which is the most abundant igneous rock, i.e. a rock formed by cooling of molten material, or magma within the earth's crust, is composed of three main mineral constituents, feldspar, quartz, and mica. These three homogeneous mineral components occur in varying proportions in different granites, and even in different parts of the same granite mass.

Coals are not minerals in the geological sense, but a group of bedded rocks formed by the accumulation of vegetable matter. Most coal-seams were formed over 300 million years ago by the decomposition of vegetable matter from the dense tropical forests which covered certain areas of the earth. During the early formation of the coal-seams, the rotting vegetation formed thick beds of peat, an unconsolidated product of the decomposition of vegetation, found in marshes and bogs. This later became overlain with shales, sandstones, mud, and silt, and under the action of the increasing pressure and temperature, and time, the peat-beds became altered, or metamorphosed, to produce the sedimentary rock known as coal. The degree of alteration is known as the rank of the coal, the lowest rank (lignite or brown coal) showing little alteration,

while the highest rank (anthracite) is almost pure graphite (carbon).

Mineral n.矿物, 矿石(1,A naturally occurring, homogeneous inorganic solid substance having a definite chemical composition and characteristic crystalline structure, color, and hardness.2,Any of various natural substances, as: An element, such as gold or silver; An organic derivative, such as coal or petroleum.3,A substance, such as stone, sand, salt, or coal, that is extracted or obtained from the ground or water and used in economic activities.4,A substance that is neither animal nor vegetable; inorganic matter.5,An inorganic element, such as calcium, iron, potassium, sodium, or zinc, that is essential to the nutrition of human beings, animals, and plants.6,An ore.)

Seabed n.海底, 海床

Platinum n.白金, 铂

Sulphide n.[化] 硫化物

Galena n.[矿] 方铅矿

Sphalerite n.[矿] 闪锌矿

Cassiterite n.[矿] 锡石

Isomorphism n.同形, 类质同像

Polymorphism n.多形性, 多态现象

Olivine n.橄榄石, 黄绿

Chalk n.粉笔, 白垩; vt.用粉笔写(1, A soft, compact calcite, CaCO_3 , with varying amounts of silica, quartz, feldspar, or other mineral impurities, generally gray-white or yellow-white and derived chiefly from fossil seashells.2,A piece of chalk or chalklike substance in crayon(蜡笔) form, used for marking on a blackboard or other surface.)

Granite n.花岗岩

Feldspar n.[矿] 长石

Magma n.(有机物或矿物的)稀糊, 岩浆, [药]乳浆剂

Marsh n.湿地, 沼泽, 沼泽地;

bog n. 沼泽

shale n.[地]页岩, 泥板岩

silt n.淤泥, 残渣, 煤粉, 泥沙 v.(使)淤塞, 充塞

metamorphose v.变形, 变质, 使变成;

lignite n.泥炭, 褐煤;

anthracite n.无烟煤;

Metallic Ores

The essential need for metals in civilized society was emphasized by Professor J. Nutting in his Presidential Address to the Metals Society. His reference to the continuing need for metals as "carriers of loads and transmitters of energy" immediately raises the question of the future availability of metals. The metallurgical industry has changed the natural distribution by using energy to treat ores and so produce pure metal.

There is, with the possible exception of iron, a continuing reduction in the metal content of ores being processed, and availability in the future will be related to the concept of what constitutes a "commercially viable ore".

This will be in contemporary commercial terms and will be at such cost levels as will permit exploitation of very low-grade deposits in the ocean floor and in the continental crustal rocks.

It has been verified that eight elements account for over 99 % of the earth's crust: 74.6% is silicon and oxygen and only three of the industrially important metals (aluminium, iron, and magnesium) are present in amounts above 2%. The others of the eight are calcium, sodium and potassium. All the other useful metals occur in amounts below 0.1%; copper, for example, which is the most important non-ferrous metal, occurring

only to the extent of 0.0055%. It is interesting to note that the so-called common metals, zinc and lead, are less plentiful than the rare-earth metals (cerium, thorium, etc.).

It is immediately apparent that if the minerals containing the important metals were uniformly distributed throughout the earth, they would be so thinly dispersed that their economic extraction would be impossible.

However, the occurrence of minerals in nature is regulated by the geological conditions throughout the life of the mineral. A particular mineral may be found mainly in association with one rock-type, e.g. cassiterite, mainly associates with granitic rocks, or may be found associated with both igneous and sedimentary rocks, i.e. those produced by the deposition of material arising from the mechanical and chemical weathering of earlier rocks by water, ice, and chemical decay. Thus, when granite is weathered, cassiterite may be transported and re-deposited as an alluvial deposit. Due to the action of these many natural agencies, mineral deposits are frequently found in sufficient concentrations to enable the metals to be profitably recovered. It is these concentrating agencies and the developments of demand as a result of research and discovery that convert a mineral deposit into an ore. Most ores are mixtures of extractable minerals and extraneous rocky material described as gangue.

An ore can be described briefly as an accumulation of mineral in sufficient quantity as to be capable of economic extraction. This establishes the market price of the metal as a critical criterion in the definition, and this will vary according to the commercial demands. With the passage of time and the depletion of richer or more readily accessible material, a mineral deposit may be upgraded to an ore. Improvements in metallurgical extraction and the introduction of new methods also become factors in making-available deposits hitherto regarded as uneconomic. Thus the introduction of the flotation process in mineral processing permitted the extraction of copper from material containing less than 0.5% of the metal and formerly regarded as waste. Similarly the introduction of solvent extraction has enabled Nchanga Consolidated Copper Mines in Zambia to treat 9 million tonnes per year of flotation tailings, producing 80 000 tonnes of finished copper from what was previously regarded as waste material.

The grade (metal content) of ore mined and processed will depend upon a number of factors, and generally the lower-grade ores are treated in larger capacity plants than higher-grade ores.

The factors that control the suitability of a deposit for economic mining and processing can be summarized as:

- (1) Location and size of the deposit.
- (2) The ore-feed grade, mineralogy, and texture of the ore. The texture refers to the aggregation (size) and dissemination (distribution) of the valuable minerals within the ore. In some cases the mineral may be coarsely aggregated, so that the particles may be seen by eye. Often, however, there is fine dissemination, and microscopic examination is to study the occurrence. The nature of the minerals is important as a pronounced difference in some physical property between the values and gangue minerals is necessary for processing.
- (3) Financial aspects-investment requirements, available capital and costs of borrowed funds, taxes, and royalty payments.
- (4) Mining cost. Deep mining is expensive compared with open-pit and alluvial operations, and is economic only on higher ore grades. Typical grades for tin ores mined underground are in the range 0.3-1.5% Sn, whereas alluvial deposits are invariably of much lower grade, a large dredging operation in Malaysia, for example, may be processing feed containing 0.02% Sn or less.
- (5) Cost of ancillary services, such as power supply, water, roads, and tailings disposal.
- (6) Amenability of ore to treatment, such as the process flowsheet, operating costs, concentrate grades, and recoveries obtainable.
- (7) The demand for and value of the metal, the metal concentrate prices, and the value of the concentrate placed at the mine.

It should be realised that the market price for the metal does not apply fully or directly to the concentrates. The purchase terms quoted by the smelter are usually based on a nominal concentrate grade, lower concentrate grades being penalized in direct relation to their metal content. The costs for smelting and refining of the metal are deducted, together with a percentage for processing losses. Impurities in the concentrate, which are present above specified limits, are penalised although bonuses are given for precious metals which may be extracted from the material. The market price for the concentrate therefore varies according to its grade and impurity content.

The minimum metal content required for a deposit to qualify as an ore varies from metal to metal according to the factors listed above. Many non-ferrous ores contain, as mined, as little as 1% metal, and often much less. Gold may be recovered profitably in ores containing only 5 parts per million (ppm) of the metal, whereas iron ores containing less than about 20% metal are regarded as low grade.

There are many plants where minerals are recovered in secondary circuits, treating tailings, where the feed grades are much lower than would be economic on a mined ore. Typical ore grades for tungsten ores are in the range 0.5--1.5% WO_3 , but the Climax Molybdenum plant in the United States treats 45 000 tonnes per day (t d^{-1}) of tailings, containing less than 0.1% WO_3 , and is one of the two major producers of tungsten concentrate in the United States.

Tin is recovered from about 10 000 t d^{-1} each of tailings from the lead-zinc, and copper-lead-zinc plants of the Sullivan concentrator in British Columbia and from the Kidd Creek plant of Texasgulf, near Timmins, respectively. The plant feeds are of about 0.06 and 0.15 % Sn respectively.

The working costs for treating old tailings dumps are much lower than conventional mining costs, and operations have been set up in many old mining areas where large tonnages of mill tailings are amenable to processing. The East Rand Gold and Uranium Company (ERGO) retreats old mill tailings from mines of South Africa's East Rand gold-fields to produce gold, uranium, and sulphuric acid. The slimes dams are treated by monitoring, i.e. diverting high-pressure jets of water at the working face. The resultant slurry gravitates to transfer pump stations adjacent to each dam, and is pumped to a central processing plant. The plant treats 1.5 million tonnes per month of slimes, grading 0.53 ppm gold, 40 ppm of U_3O_8 , and 1.04% of S.

The world is now becoming aware of the finite nature of its resources at a price, and of the ever-increasing development costs of large new mines. Reprocessing of old tailings on a large scale must be worth examining very seriously by those with access to sufficient material of this type.

Ores are frequently classed according to the nature of the valuable mineral. Thus in native ores the metal is present in the elementary form; sulphide ores contain the metal in the form of a sulphide, and in oxidised ores the valuable mineral may be present as oxide, sulphate, silicate, carbonate, or some hydrated form of these. Complex ores are those containing profitable amounts of more than one valuable mineral. Metallic minerals are often found in certain associations, within which they may occur as mixtures of a wide range of particle sizes or as single-phase solid solutions or compounds. Galena and sphalerite, for example, commonly associate, as do copper sulphide minerals and sphalerite to a lesser extent. Pyrite (FeS_2) is very often associated with these minerals.

Ores are also classified by the nature of their gangues, such as calcareous or basic (lime rich), or siliceous, or acidic (silica rich).

Constitute vt. 制定(法律), 建立(政府), 组成, 任命

Viable adj. 可行的, 能养活的, 能生育的;

Deposit n. 堆积物, 沉淀物, 存款, 押金, 保证金, 存放物; vt. 存放, 堆积; vi. 沉淀

Cerium n. [化] 铈

Thorium n. 钍

Accessible adj.易接近的, 可到达的, 易受影响的, 可理解的

Gangue n.脉石

Zambia n.[国名]赞比亚(位于非洲)

Tonne 公吨(=1,000 公斤或称 metric ton)

Texture n.(织品的)质地, (木材, 岩石等的)纹理, (皮肤)肌理, (文艺作品)结构

Grade n.品位, 等级, 级别 vt.评分, 评级 v.分等级

Alluvial adj.冲积的, 淤积的

Dredging n.挖泥, 捕捞

Ancillary adj.补助的, 副的

Flowsheet 流程图, 流向图, 程序框图, 操作程序图; process flowsheet 工艺流程图, 生产流程图

Tailing n.残渣, 尾矿, [建]嵌入墙中砖石突出部, 屑

Dump n.(煤, 弃物, 垃圾)堆, 库房, 渣坑, 抑郁

Bonus n.奖金, 红利

Slime n.(尾)矿泥, 粘土, 粘液(A thick, sticky, slippery substance. A mucous substance secreted by certain animals, such as fish. Vile or disgusting matter.) vt.涂泥 vi.变粘滑

Calcareous adj.石灰质的, 含钙的

Siliceous adj.硅酸的, 硅土的

at a price 付出很大代价, 昂贵; at any price 无论花多大代价; 无论如何; above price 宝贵的, 无价的; beyond price 宝贵的, 无价的; without price 宝贵的, 无价的; at the price of 以...的代价; 以高于正常的价格, 以高于市场的价格; filling at the price 物美价廉; give a long price for 高价购买

Non-metallic ores

Ores of economic value can be classed as metallic or non-metallic, according to the use of the mineral. Certain minerals may be mined and processed for more than one purpose. In one category the mineral may be a metal ore, i.e. when it is used to prepare the metal, as when bauxite (hydrated aluminium oxide) is used to make aluminium. The alternative is for the compound to be classified as a non-metallic ore, i.e. when bauxite or natural aluminium oxide is used to make material for refractory bricks or abrasives.

Many non-metallic ore minerals associate with metallic ore minerals and are mined and processed together, e.g. galena, the main source of lead, often associates with fluorite (CaF_2) and barytes (BaSO_4), both important non-metallic minerals.

Diamond ores are the lowest grade of all ores presently mined, the diamond content usually being between 0.03 and 0.15 ppm. The ores are mined chiefly for industrial diamonds, which constitute about 80% of the total production, but also for gems. It is the mining of the very expensive gem diamonds, which are used for jewellery, which subsidises the costs of mining the relatively cheap industrial diamonds from such low-grade ore deposits.

The lowest grade diamond ore presently mined is at the Letseng-La-Terai mine in Lesotho, where the grade is only 0.007 ppm. The ore is economic to work, as occasionally very large diamonds of gem quality are found.

Bauxite n.矾土, 铁铝氧石

Fluorite n.[矿]萤石, 氟石

Barytes n.重晶石 (= barite n.重晶石)

Category n.种类, 别, [逻]范畴

Jewellery n.(=jewelry) [总称]珠宝, 镶嵌有宝石之饰物

Subsidise v.消退, 下沉, 平息

Reading Material Macerals in Coal

Coal is an organic rock. Just as an inorganic rock is composed of petrological components known as minerals, coal consists of petrological components called macerals. But there is a difference. Whereas a mineral is characterized by a fairly well-defined chemical composition, the uniformity of its substance, and the fact that most minerals are crystalline, a maceral of coal varies widely in chemical compositions and physical properties and is not crystalline. The heterogeneous structure of coal is perceptible to the naked eye when examining a coal face or a sample, but only the microscope reveals the details of the constitution of coal.

Coal components can be identified microscopically by either of two techniques. The first is the thin-section or transmitted-light technique, developed mainly in the United States and Great Britain; the second is the polished-section or reflected-light technique, developed mainly in Europe.

The thin section technique (Thiessen method), although it is considered the standard reference for characterizing the petrographic composition of American coals, is not used in all petrography laboratories. This is because the preparation of the thin section takes much time and demands considerable skill, and it is almost impossible to avoid oxidation of the coal during the process. Reflected light petrography was developed in 1913 by the German scientist Winter. This method could not compete at the outset with the older thin section method as far as the degree to which the detail of the coal sample was made visible. The reflectivity method was greatly improved between 1920 and 1935. First, special techniques were devised for etching the coal sample and after that the oil immersion technique was developed.

Marie C. Stopes is credited for developing a system of classification utilizing the reflected light technique. Stopes recognized four coal constituents (lithotypes) that could be seen with the unaided eye (macroscopic) in banded bituminous coals. To these lithotypes she nomenclatured (gave the names) vitrain, clarain, durain, and fusain. Bright coal was distinguished as vitrain and clarain, dull coal as durain, and fossil charcoal as fusain.

Microscopic studies of the four different bands of ordinary coal revealed that two of them, namely the vitrain and fusain, are homogeneous, showing the same kind of material throughout. The clarain and durain bands, on the other hand, are heterogeneous, consisting of different kinds of easily determinable microscopic features. These features are the same as those encountered in the attritus of Thiessen, but appear differently when observed under reflected light.

The fact that these two different microscopic methods (thin-section and reflected-light) were developed separately meant that hardly any investigators employed and thoroughly understood both methods. Furthermore, the European nomenclature and classification of the ingredients of coal developed from the examination of polished sections was carried out along two different lines. In the one, the microscopic features were studied relative to their macroscopic and physical appearance; in the other, they were examined entirely from the point of view of their microscopic characteristics, regardless of existing macroscopic divisions. Consequently, great confusion was generated in applying petrographic nomenclature. At the International Stratigraphic Congress held in Heerlen in 1935, a new nomenclature proposed by Stopes was accepted in an attempt to establish order to the existing situation. The classification system that evolved was referred to as the Stopes-Heerlen (S-H) system.

Besides the lithotypes, the Stopes-Heerlen (S-H) nomenclature distinguishes between microlithotypes and their more or less homogeneous microscopic constituents, which are called macerals based upon the nature of coal as a rock. Three maceral groups have been established: vitrinite, exinite, and inertinite. To these maceral groups, the microlithotypes col-finite, telinite, cutinite, alginite, resinite, sporinite, micrinite, sclerotinite, semifusinite, and fusinite have been assigned. Krevelen provides a concise description of the grouping and

characteristics of these constituents.

Maceral 煤岩组分

petrological adj.[地]岩石学的

heterogeneous adj.不同种类的,异质的

petrographic adj.[地]岩相学的,岩类学的

vitrain n.[矿]镜煤,闪炭

clarain 亮煤

durain 暗煤

fusain n.丝炭,炭画笔, 炭笔画

vitritinite 镜质组

exinite 壳质组

inertinite 惰性组

fusinite 丝质组

Unit 2 Scope of Mineral Processing

"As-mined" or "run-of-mine" ore consists of valuable metallic minerals and waste (gangue). Mineral processing, sometimes called ore dressing, mineral dressing, or milling, follows mining and prepares the ore for extraction of the valuable metal in the case of metallic ores, but produces a commercial end product of non-metallic minerals and coal.

Apart from regulating the size of the ore, it is a process of physically separating the grains of valuable minerals from the gangue minerals, to produce an enriched portion, or concentrate, containing most of the valuable minerals, and a discard, or tailing, containing predominantly the gangue minerals.

This concentration, or enrichment process, considerably reduces the volume of material that must be handled by the extractive metallurgist, so reducing to economic quantities the amounts of energy and reagents required to produce the pure metal.

It has been predicted that the importance of mineral processing of metallic ores may decline as the physical processes utilised are replaced by the hydro and pyrometallurgical routes used by the extractive metallurgist, because higher recoveries are obtained by some chemical methods. This may certainly apply when the useful mineral is very finely disseminated in the ore and adequate liberation from the gangue is not possible, in which case a combination of chemical and mineral processing techniques may be advantageous. However, in the majority of cases the energy consumed in direct smelting or leaching of low-grade ores would be *so enormous as to* make the cost prohibitive. Compared with these processes, mineral processing methods are inexpensive, and their use is readily justified on economic grounds.

Mineral processing reduces the bulk and weight of material which must be transported to the smelter, thus reducing shipping and handling costs; smelting costs are greatly reduced due to the reduced tonnage which must be treated; the lack of undesirable waste materials increases the smelter recovery, due to the production of less metal-bearing slag. Against these advantages, of course, must be charged the losses incurred during milling, and the cost of the milling operations. Taggart has analyzed the balance *between* production costs *and* returns from smelting a typical metallic ore, *with* and *without* mineral processing. The economic significance of milling is most apparent from his results.

Nevertheless, the energy consumed in mineral processing operations can be a considerable proportion of the total energy required to produce the primary metal, especially if the ore is low grade. For a typical copper ore, containing about 0.6% metal, the total energy required to produce the primary metal is about 33 x 1000 kwh per tonne of metal. Almost a third of this total energy requirement is consumed in the mill. On the other hand, the total energy requirement for primary iron from an ore of 24% metal is about 7x1000kWh per tonne of metal, of which the milling requirement is about 10% of the total. It is significant, therefore, that as ore grades decrease, mill energy consumptions could become the most important factor in deciding whether the deposit is developed or not.

Apart from economics, energy consumption is of paramount importance, since, although the world has ample mineral deposits to meet the demand for most common metals for the rest of the century, it has been predicted that there will not be enough energy to produce these metals.

Barsotti studied three metals--aluminium, copper, and iron which are basic to any country seeking economic growth, and which require vast amounts of energy to produce them.

Petroleum, coal, and hydroelectric power are the major energy sources for the production of metals, and Barsotti concluded that: "within three to four decades, petroleum will be substantially depleted, and replacement by other forms of energy, including nuclear energy, may not be sufficient to offset the resultant impact". He indicated that major improvements in energy efficiency and conservation were essential.

If the ore contains worthwhile amounts of more than one valuable mineral, it is usually the object of mineral processing to separate them; similarly if undesirable minerals, which may interfere with subsequent refining processes, are present, it may be necessary to remove these minerals at the separation stage.

Ore dressing is a necessary prelude with most non-ferrous ores. However, there are so many high-grade iron ore deposits that, until relatively recently, there has been little incentive to upgrade them, with the possible exception of Swedish magnetite (Fe_3O_4) containing apatite. Milling of iron ores is, however, continually gaining importance with the working of lower-grade deposits, and most of the iron now produced in the United States is from low-grade taconite (n.[地] 铁燧岩) ores (*finely disseminated iron minerals in silica*).

Part, or all, of a mineral-processing plant is often referred to as a mill, this usually being situated at the mine, although a custom mill treats ore from a number of mines in the locality.

There are two fundamental operations in mineral processing, namely the release, or liberation, of the valuable minerals from their waste gangue minerals, and separation of these values from the gangue, this latter process being known as concentration.

Liberation of the valuable minerals from the gangue is accomplished by comminution, which involves crushing, and, if necessary, grinding, to such a particle size that the product is a mixture of relatively clean particles of mineral and gangue.

The correct degree of liberation is the key to success in mineral processing. The valuable mineral should be freed from the gangue, but only just freed. A process which overgrinds the ore is wasteful, since it needlessly consumes grinding power and makes efficient recovery more difficult to attain. So important is it to avoid overgrinding, that, as will be seen later, some ores are comminuted to a size coarser than their liberating size before initial concentration.

After the minerals have been liberated from the gangue, the ore is subjected to some process of concentration, which separates the minerals into two or more products. Separation is usually achieved by utilizing some specific difference in physical or chemical properties between the valuable minerals and gangue minerals in the ore.

Mineral processing is concerned mainly with the physical methods of separation, which may be:

1. Separation dependent on optical and radioactive properties, etc. This is often called sorting, which commonly included hand selection of high-grade ores until relatively recently.
2. Separation dependent on specific gravity differences. This utilizes the differential movement of minerals due to mass effects, usually in hydraulic currents. Although the method declined in importance with the development of the froth flotation process, it is now being increasingly used due to improved techniques and its relative simplicity compared with other methods. It also has the advantage of producing less environmental pollution.
3. Separation utilizing the different surface properties of the minerals. Froth flotation, which is undoubtedly the most important method of concentration, is affected by the degree of affinity of the minerals *for* rising air-bubbles within the agitated pulp. By adjusting the "climate" of the pulp by various reagents, it is possible to make the valuable minerals air-avid (aerophilic) and the gangue minerals water-avid (aerophobic). This results in separation by transfer of the valuable

minerals to the air -bubbles which form the froth floating on the surface of the pulp.

4. Separation dependent on **magnetic properties**. Low intensity magnetic separators can be used to concentrate ferromagnetic minerals such as magnetite (Fe_3O_4), while high-intensity separators are used to separate paramagnetic minerals from their gangue. Magnetic separation is an important process in the beneficiation of iron ores, but also finds application in the treatment of paramagnetic non-ferrous minerals. It is widely used to remove paramagnetic wolframite ($(\text{Fe}, \text{Mn})\text{WO}_4$) and hematite (Fe_2O_3) from tin ores, and has found considerable application in the processing of non-metallic minerals, such as those found in beach sand deposits.

5. Separation dependent on **electrical conductivity properties**. High-tension separation can be used to separate conducting from non-conducting minerals. This method is interesting, since theoretically it represents the "universal" concentrating method; almost all minerals show some difference in conductivity and it should be possible to separate almost any two by this process. However, the method has fairly limited application, and its greatest use is in separating some of the minerals found in heavy sands from beach or stream placers (n.冲积矿). Minerals must be completely dry and the humidity of the surrounding air must be regulated, since most of the electron movement in dielectrics takes place on the surface and a film of moisture can change the behavior completely. The biggest disadvantage of the method is that the capacity of economically sized units is very low.

Heat treatment is sometimes used to make the ore more suitable for subsequent processing. Roasting can be used to effect major chemical changes, such as the conversion of non-magnetic iron minerals to a ferromagnetic form. Calcination can be used to destroy the colloidal bond of clay minerals and to decompose hydrates and carbonates, making the ore easier to handle and treat.

Comminution and concentration are the two primary operations in mineral processing, but many other important steps are involved, among which are sizing of the ore at different stages in treatment, by the use of screens and classifiers, and dewatering of the mineral pulps, using thickeners, filters, and driers.

The Flowsheet

The flowsheet shows diagrammatically the sequence of operations in the plant. In its simple form it can be presented as a block diagram in which all operations of one character are grouped. In this case comminution deals with all crushing, grinding, and initial rejection. The next block "separation", groups the various treatments incident to production of concentrate and tailing. The third, "product handing", covers the disposal of the products.

The simple line flowsheet is for most purposes sufficient, and can include details of machines, settings, rates, etc.

Paramount adj. 极为重要的

Ample adj. 充足的, 丰富的

Deplete vt. 耗尽, 使衰竭

Offset n. 偏移量, 抵销, 弥补, 分支, 平版印刷, 胶印; vt. 弥补, 抵销, 用平版印刷 vi. 偏移, 形成分支

Incur v. 招致

Magnetite n. 磁铁矿

Apatite n. 磷灰石

Taconite n. [地] 铁燧岩

Disseminate v. 散布

Ferromagnetic adj. 铁磁的, 铁磁体

Beneficiate vt. 选(矿), (冶炼前) 对(矿石) 进行预处理

Paramagnetic adj. 顺磁性的

Wolframite n. [矿] 钨锰铁矿, 铁锰重石

Placer n. 冲积矿, 放置者

Dielectric n. 电介质, 绝缘体; adj. 非传导性的

Comminution n. 粉碎

Aerophilic

Aerophobic

Wolframite n. [矿] 钨锰铁矿, 铁锰重石

Calcinations n. 煅烧

Reading Material Coal Impurities

The impurities occurring in coal may be classified broadly into those that form ash and those that contribute sulfur. Other impurities occur (for example, phosphorus and salt) but they are of little importance in US coals. From the standpoint of coal cleaning, both the ash-forming and the sulfur-containing impurities may be subdivided into two categories----inherent and extraneous. The inherent impurities are inseparably combined with the coal. The extraneous impurities are segregated and can be eliminated by available cleaning methods to whatever extent is economically justified.

Mineral matter

All coals contain noncombustible mineral matter. The residue from these minerals after coal has been burned is called ash. The average ash content of the entire thickness of a coal seam is at least 2 or 3%, even for very pure bituminous coals, and 10% or more for most commercial mines. Material that is too high in ash for ordinary use is called bone coal, carbonaceous shale, or black slate.

Minerals vary widely in coal seams with respect to kind, abundance, and distribution. Their practical importance is evident in nearly every phase of the industry, from mining through preparation and use. As examples, one may call attention to diverse problems such as silicosis, breakage and grindability characteristics, washability problems, coke oven wall attack, and fireside deposits in boilers.

bone coal 骨煤; 骨碳

carbonaceous shale 碳质页岩

Silicosis n. [医] 硅肺病

Grindability 可磨性

Washability 可选性

Detrital adj. 由岩屑形成的

Ash-forming material organically combined with the coal is considered inherent mineral matter; this portion came from the chemical elements existing in the plants from which coal was

formed. Generally, the inherent mineral matter is about 2% or less of the total ash. Extraneous mineral matter is ash-forming material foreign to the plant material from which the coal was formed; the bulk of this material is from detrital matter that settled into the deposit, crystalline deposits from water that penetrated into the coal through fracture cracks and cleavages during and after coal formation, or saline deposits from water before and during the formation of peat from which the coal was formed. It consists usually of slate, shale, sandstone, or limestone and includes pieces ranging from microscopic size to thick layers. Mined coal may also include shale, sandstone, clay, or other extraneous mineral matter from the roof and floor of the mine. The inherent matter is a useful classification when applied to the practical problems of coal cleaning. Generally speaking, it defines the lower limit to which coal can be separated physically from its mineral impurities. This is true because, theoretically at least, all extraneous matter can be removed. In practice, however, this is not always the case.

The first determinations of the mineral constituents of coal were made using optical microscopy. Several more recently introduced techniques for mineralogical examination have provided a means to obtain additional information. These include X-ray diffraction, radiography, thermal and differential thermal analysis, and electron microscopy. The techniques and optical methods involved are described by Gumz, Kirsch, and Mackowsky.

Several empirical formulas have been developed for the calculation of mineral matter from the ash content and other analytical data. The best known is the Parr formula used in the ASTM system of coal classification to convert parameters of volatile matter, fixed carbon, and calorific value to a mineral-matter-free coal basis. The Parr formula is as follows:

$$\text{Mineral matter} = 1.08 \text{ ash} + 0.55 \text{ sulfur.}$$

Variations of the Parr formula have been used for either simplification or improving the accuracy. The ASTM classification system provides a simplified approximation formula where high accuracy is not required as follows:

$$\text{Mineral matter} = 1.1 \text{ ash} + 0.1 \text{ sulfur.}$$

The King, Maries, and Crossley formula developed from a survey of minerals in British coals is the most comprehensive of those proposed because, in addition to correction for water of hydration, corrections are made for the forms of sulfur, carbon dioxide in the carbonates, sulfur retained in the ash, and chlorine. This formula is:

$$\text{Mineral matter} = 1.09 \text{ ash} + 0.5S_{\text{pyr}} + 0.8 \text{ CO}_2 - 1.1 \text{ SO}_{3\text{ash}} + \text{SO}_{3\text{coal}} + 0.5 \text{ Cl.}$$

Because of the large amount of analytical work required to make the corrections, Brown et al made a statistically study that showed certain components in the formula to be interrelated. A simplified formula required fewer analytical determinations was developed as follows:

$$\text{Mineral matter} = 1.06 \text{ ash} + 0.53S + 0.74\text{CO}_2 - 0.32.$$

In this formula, organic sulfur is considered part of the coal substance. The calculation of mineral matter from the chemical composition of coal also has been proposed. This method is based on the assumption that when the chemical composition of the organic coal substance is known, the mineral content is the difference between this value and 100% (all determined on dry-coal basis):

$$\text{Mineral matter} = 100 - (\text{C} + \text{H} + \text{N} + \text{O} + \text{S})$$

Such a method requires direct determination of oxygen, a determination not made normally. Also the method is subject to error since the water of constitution of the mineral matter is included with the hydrogen and oxygen to an unknown extent.

One method that has been used with considerable success is the removal of the organic coal substance by incineration at low temperatures, leaving most of the mineral matter substantially

unaltered. This technique, called low temperature ashing, is based on work of Nagelschmidt. The method consists of completely oxidizing the coal at 375 degree centigrade in a specially designed flask through which a controlled mixture of oxygen and nitrogen is passed. The total time required for complete oxidation is 100-250hr, depending on the rank of coal.

Detrital adj. 由岩屑形成的

ASTM abbr. American Society of Testing Materials, 美国材料实验协会

incineration n. 烧成灰, 焚化

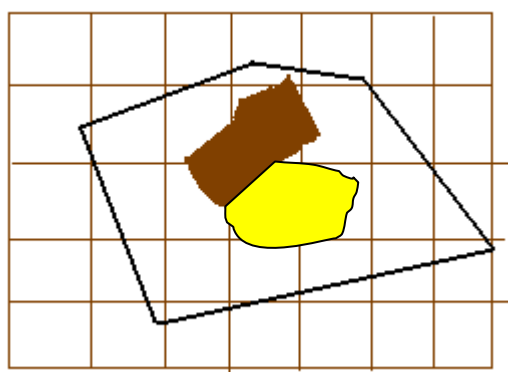
Unit 3 Efficiency of Mineral Processing Operations

Liberation

One of the major objects of comminution is the liberation, or release, *of* the valuable minerals *from* the associated gangue minerals at the coarsest possible particle size. If such an aim is achieved, then not only is energy saved by the reduction of the amount of fines produced, but any subsequent separation stages become easier and cheaper to operate. If high-grade solid products are required, then good liberation is essential; however, for subsequent hydrometallurgical processes, such as leaching, it may only be necessary to expose the required mineral.

In practice complete liberation is seldom achieved, even if the ore is ground down to the grain size of the desired mineral particles. This is illustrated by following figure, which shows a lump of ore which has been reduced to a number of cubes of identical volume and of a size below that (=the size) of the grains of mineral observed in the original ore sample.

It can be seen that each particle produced containing mineral also contains a portion of gangue; completed liberation has not been attained; the bulk of the major mineral—the gangue—has, however, been liberated from the minor mineral—the value.



The particles of “locked” mineral and gangue are known as middlings, and further liberation from this fraction can only be achieved by further comminution.

The **degree of liberation** refers to the percentage of the mineral occurring as free particles in the ore in relation to the total content. This can be high if there are weak boundaries between mineral and gangue particles, which is often the case with ores composed mainly of rock-forming minerals, particularly sedimentary materials. Usually, however, the adhesion between mineral and gangue is strong and, during comminution, the various constituents are cleft across. This produces much middlings and a low degree of liberation. New approaches to increasing the degree of liberation involve directing the breaking stresses at the mineral grain boundaries, so that the rock can be broken without breaking the mineral grains. This can be achieved by heating the rock to a high temperature, although this has not, as yet, been exploited in practice.

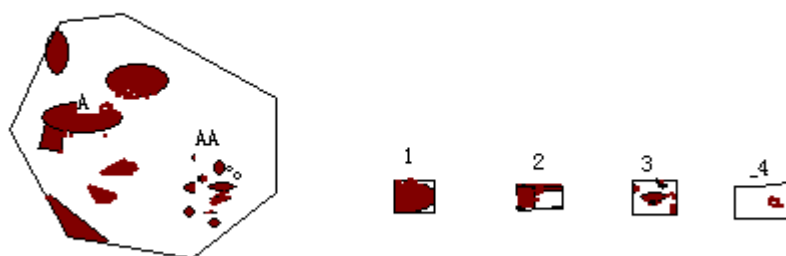
Up till recently it has been impossible for mineralogists to give a quantitative answer to the question: How fine must a particular ore be ground to produce a certain degree of liberation of the valuable mineral particles? The first attempt at the development of a model for the calculation of mineral liberation was made by Gaudin, but this model suffers from many highly

unrealistic assumptions that must be made with respect to the grain structure of the minerals in the ore, and, as a result, the model has not found much practical application.

King has developed an exact expression for the fraction of particles of a certain size that contain less than a prescribed fraction of any particular mineral. The expression is obtained entirely *in terms of* the distribution of linear intercept lengths of the minerals in the ore, these being obtained by line traverses across a section of the ore. The theory is completely free of empirical constants or other parameters, and in particular no assumptions are made regarding the shape of the mineral grains in the ore or of the particles. The model is applicable to *binary isotropic mineral assemblies*, where preferential fracture along grain boundaries does not occur. The theory has been confirmed experimentally for the liberation of pyrite from quartzite, and the ease of application of the model makes it very attractive for use in routine ore-dressing applications. The data needed for the model, namely a sufficiently large sample of intersection lengths, can easily be obtained using modern automated microscopic image analysis equipment, which is becoming increasingly used in well-equipped metallurgical laboratories.

In practice, ores are ground to an optimum mesh of grind, determined by laboratory and pilot scale testwork, to produce an economic degree of liberation. The concentration process is then designed to produce *a concentrate* consisting predominantly of valuable mineral, with an accepted degree of locking with the gangue minerals, and *a middlings* fraction, which may require further grinding to promote optimum release of the minerals. The tailings should be mainly composed of gangue minerals.

Figure x.x is a cross-section through a typical ore particle, and illustrates effectively the liberation dilemma often facing the mineral processor. Regions A represent valuable mineral, and region AA is rich in valuable mineral, but is highly intergrown with the gangue mineral. Comminution produces a range of fragments, ranging from fully liberated mineral and gangue particles, to those illustrated. Particles of type 1 are rich in mineral, and are classed as concentrate as they have an acceptable degree of locking with the gangue which limits the concentrate grade. Particles of type 4 would likewise be classed as tailings, the small amount of mineral present reducing the recovery of mineral into the concentrate, particles of types 2 and 3, however, would probably be classed as middlings, although the degree of regrinding needed to promote economic liberation of mineral from particle 3 would be greater than that in particle 2. (Please figure out images of the four kinds of particles and the original ore fragment structure)



During the grinding of a low-grade ore the bulk of the gangue minerals is often liberated at a relatively coarse size. In certain circumstances it may be economic to grind to a size much coarser than the optimum in order to produce in the subsequent concentration process a large middlings fraction and tailings which can be discarded at a coarse grain size. The middlings

fraction can then be reground to produce a feed to the final concentration process.

This method discards most of the coarse gangue early in the process, thus considerably reducing grinding costs, as needless comminution of liberated gangue is avoided. It is often used on minerals which can easily be separated from the free gangue, even though they are themselves locked to some extent with gangue. It is the basis of the heavy medium process of pre-concentration.

Cleft n. 裂缝, 隙口; adj. 劈开的

Dilemma n. A situation that requires a choice between options that are or seem equally unfavorable or mutually exclusive. 进退两难需要在相等或相互排斥的选择物之间进行选择处境, 进退两难之境; 困境; 进退维谷. I am in a dilemma whether to do maths or English. The doctor was in a dilemma as to whether to tell the patient the truth about his condition or not.

Concentration

The object of mineral processing, regardless of the methods used, is always the same, i.e. to separate the minerals into two or more products with the values in the concentrates, the gangue in the tailings, and the "locked" particles in the middlings. Such separations are, of course, never perfect, so that much of the middlings produced are, in fact, misplaced particles, i.e. those particles which ideally should have reported to the concentrate or the tailings. This is often particularly serious when treating ultra-fine particles, where the efficiency of separation is usually low. In such cases, fine liberated valuable mineral particles often report in the middlings and tailings. The technology for treating fine-sized minerals is, as yet, poorly developed, and, in some cases, very large amounts of fines are discarded. For instance, it is common practice to remove material less than 10 micron in size from tin concentrator feeds and direct this material to the tailings, and, in the early 1970s, 50% of the tin mined in Bolivia, 30% of the phosphate mined in Florida, and 20% of the world's tungsten were lost as fines. Significant amounts of copper, uranium, fluor spar, bauxite, zinc, and iron were also similarly lost.

Figure 1.6 shows the size range applicability of unit concentration process. It is evident that most mineral processing techniques fail in the ultra-fine size range. Gravity concentration techniques, especially, become unacceptably inefficient, and even flotation, the most important concentrating technique used today, usually fails when used alone in the ultra-fine size range.

It should be pointed out that the process is also limited by the mineralogical nature of the ore. For example, in an ore containing native copper it is theoretically possible to produce a concentrate containing 100% Cu, but, if the ore mineral was chalcopyrite (CuFeS_2), the best concentrate would contain only 34.5% Cu.

The **recovery**, in the case of the concentration of a metallic ore, **is the percentage of the total metal contained in the ore that is recovered in the concentrate**; a recovery of 90% means that 90 % of the metal in the ore is recovered in the concentrate and 10 % is lost in the tailings. It is possible to speak of the recovery of metal in tailings, middlings, and other products, but normally the term refers to the amount of metal recovered in the concentrates--the valuable products. Essentially **the purpose of calculating recovery is to determine the distribution** of the metal in the feed (or heads) among the various products of the mineral-dressing operations. The recovery, when dealing with non-metallic ores, refers to the percentage of the total mineral contained in the ore that is recovered into the concentrate, i.e. recovery is usually expressed in terms of the valuable end product.

The **ratio of concentration** **is the ratio of the weight of the feed to the weight of the concentrate**. It is a measure of the efficiency of the concentration process, and it is closely related

to the grade or assay of the concentrate; the value of the ratio of concentration will generally increase with the grade of concentrate.

The grade, or assay, usually refers to the content of the marketable end product in the material. Thus in metallic ores, the per cent metal is often quoted, although in the case of very low-grade ores, such as gold, the metal content may be expressed as parts per million (ppm). Some metals are sold in oxide form, and hence the grade may be quoted in terms of the marketable oxide content, e.g. %W₂O₃, %U₃O₈, etc. In non-metallic operations, grade usually refers to the mineral content, e.g. %CaF₂ in fluorite ores; diamond ores are usually graded in carats per 100 tonnes (t), where 1 carat is 0.2 g. Coal is graded according to its ash content, i.e. the amount of incombustible mineral present within the coal; most of the coal produced in Britain is consumed in power stations, which require a feed with an ash content of between 15 and 20%.

The enrichment ratio is the ratio of the grade of the concentrate to the grade of the heads, and again is related to the efficiency of the process.

Ratio of concentration and recovery are essentially independent of each other, and in order to evaluate a given operation it is necessary to know both. For example, it is possible to obtain a very high grade of concentrate and ratio of concentration by simply picking a few lumps of pure galena from a lead ore, but the recovery would be very low. On the other hand, a concentrating process might show a recovery of 99% of the metal, but it might also put 60% of the gangue minerals in the concentrate. It is, of course, possible to obtain 100 % recovery by not concentrating the ore at all.

There is an approximately inverse relationship between recovery and grade of concentrate in all concentrating processes. If an attempt is made to attain a very high-grade concentrate, the tailings assays are higher and the recovery is low. If a high recovery of metal is aimed for, there will be more gangue in the concentrate and the grade of concentrate and ratio of concentration will both decrease. There is always, therefore, an economic optimum in the combination of grade and recovery. It is impossible to give figures for representative values of recoveries and ratios of concentration. The cost of milling, value of the metal, use for which the concentrate is intended and the nature of the ore, must all be considered; and each ore is a separate problem. A concentration ratio of 2 to 1 might be satisfactory for certain high-grade non-metallic ores. But a ratio of 50 to 1 might be considered too low for a low-grade copper ore; ratios of concentration of several million to one are common with diamond ores. The aim of milling operations is to maintain the values of ratio of concentration and recovery as high as possible, all factors being considered.

Chalcopyrite n. 黄铜矿

Reading Material WASHABILITY STUDIES

To determine the preparation method and the equipment used to clean the coal the preparation engineer must conduct tests to obtain data on size and specific-gravity distribution, type of fracture, petrographic description of materials, moisture, ash and sulfur contents, and ash-fusibility characteristics. The test procedure may embrace all or only several of these characteristics as well as others not so common, depending on the

information required. Washability studies are conducted primarily to determine how much coal of what quality can be produced at a given specific gravity with what separation difficulty; or, conversely, what separation density should be for a desired coal quality.

The minimum quantity of coal required for washability tests, other than those performed on drill-core samples, is dependent on the top particle size in the sample to be tested. Table 4-2 on page 4-15 (Table 2 from ASTM D 2234-72, "Standard Methods for Collection of a Gross Sample of Coal") specifies the number and weight of increments for general purpose sampling from lots of 1000 tons (910 mt) or fractions thereof. Experience dictates that as the particle size increases the weight of the gross sample should increase. When testing a coal deposit of unknown sampling characteristics, it is desirable to obtain gross coal samples from as many locations as possible but, in any event, no less than three gross samples should be tested.

A washability study of a coal is made by testing the coal sample at preselected, carefully controlled densities (specific gravities). This is commonly termed float-and-sink analysis and/or specific-gravity fractionation. The specific-gravity fractions are dried, weighed, and analyzed, generally for ash content. Other analyses, such as sulfur content, are also conducted, depending on the end use of the washed coal. A table is compiled showing the weight percent of each specific-gravity fraction together with the analyses of each fraction. The data are mathematically combined on a weighted basis into cumulative float and cumulative sink and used to develop the washability curves that are characteristic for the coal.

Heavy organic liquids and solutions of zinc or calcium chlorides in water are commonly used 'to obtain the desired specific gravities. However the latter, although less expensive than organic liquids, cannot be practically **concentrated to** specific gravities greater than 1.92. Furthermore, these solutions, especially at the higher gravities, become viscous and are neither accurate nor satisfactory for separating fine sizes of coal. Also, the solution tends to be absorbed into the pores of the coal thereby changing its apparent specific gravity, and the viscous solution interferes with the separating process. When either zinc or calcium chlorides are used it is imperative that the coal be thoroughly washed with warm water to ensure that' all traces of the chloride compound are removed since these compounds distort both the weight of recovered coal and the subsequent ash analysis. Likewise, the hydrocarbon solutions must be carefully dried off until the samples are relatively odor-free. Traces of the solvents have been known to affect the results of carbonization tests.

The properties of several organic liquids used in float-and-sink testing are shown in Table 4-3. Previously, carbon tetrachloride and benzene had been recommended as a solution for float-and-sink testing; however, because of their high toxicity and cumulative physical effects on the human body the use of carbon tetrachloride and benzene is discouraged.

Suitable tanks for holding the specific-gravity solution with provision for removal of the sink material are required. Tests on coals passing approximately the No. 8 (2.36 mm) or No. 16 (1.18 mm) sieve are best performed with organic liquids by using special funnel-flask glass bottles joined by standard ground-glass taper joints. Some laboratories prefer to conduct a static float-sink test on fractions coarser than about No. 50 sieve (0.30 mm) and a dynamic test using a centrifuge for sizes finer than about No. 50 sieve (0.30 mm). The froth-flotation cleaning potential of fine coal is best evaluated by use of the timed-release analysis technique or a combination of flotation, washability, and the error-curve method.

An example of data obtained by means of a float-and-sink analysis is shown in Table 4-4. This coal was separated at specific gravities of 1.27, 1.30, 1.38, 1.50, 1.70, and 1.90. The dry weight of each fraction is determined and each is analyzed for ash content (Table 4-4, columns 2 and 3). The weight percent of each fraction (column 2) is multiplied by the ash content (column 3) to obtain the ash product (column 4). To obtain the cumulative percent ash for each gravity fraction, columns 2 and 4 are cumulated from top to bottom to give columns 5 and 6, respectively. Column 6 is then divided by column 5 to obtain the cumulative percent ash

(column 7). Cumulative sink ash is obtained in essentially the same manner, except that columns 2 and 4 are cumulated from bottom to top to give columns 8 and 9, respectively. Column 9 is then divided by column 8 to obtain cumulative percent ash sink (column 10). Additional analyses of each fraction, such as sulfur or volatile matter, may be added to the table if desired.

Table 4-4 Float-and-Sink data

	Individual fractions			Cumulative Float			Cumulative Sink			Ordinate
1	2	3	4	5	6	7	8	9	10	11
Density	Wt%	Ash%	Ash Prod	Wt%	Ash Prod	Ash%	Wt%	Ash Prod	Ash%	
-127	34.5	2.8	96.9	34.5	96.6	2.8	100.0	1293.1	12.9	17.25
127x130	28.6	3.9	110.8	62.9	207.4	3.3	65.5	1196.5	18.3	48.7
130x138	16.9	8.8	148.7	79.8	356.1	4.5	37.1	1085.7	29.3	71.3
138x150	5.4	16.9	91.3	85.2	447.4	5.3	20.2	937.0	46.4	82.5
150x170	3.3	30.6	101.0	88.5	548.4	6.2	14.8	845.7	57.1	86.85
170x190	3.0	46.2	138.6	91.5	687.0	7.5	11.5	744.7	64.8	90
+190	8.5	71.3	606.1	100	1293.1	12.9	8.5	606.1	71.3	95.75

There are various methods of presenting washability data. Some preparation engineers prefer to plot the data on semilogarithmic paper, but a commonly employed method is the arithmetic plot as shown in Fig. 4-13. The washability curves shown in Fig. 4-13 are plotted from the data presented in Table 4-4. The five curves generally drawn are specific gravity (yield), cumulative float ash, cumulative sink ash, elementary ash, and ± 0.10 specific gravity distribution.

The specific-gravity (yield) curve is plotted directly from the cumulative percent weight float data and specific gravity fractions (columns 5 versus 1). This curve shows the theoretical yield of washed product from the raw coal for any specific gravity of separation. The cumulative float ash curve is plotted directly from the cumulative percent weight float and cumulative percent ash float (columns 5 vs.7) and shows the theoretical percent ash of the washed product at any given yield of washed product. Additionally, since the specific gravity (yield) and cumulative float ash curves have common ordinate values, it is possible to determine what the

separating gravity should be for a desired ash content. From Fig. 4-13 one can see that to obtain a product having an ash content no greater than 5.8%, the washing gravity must not be over 1.47. The maximum yield for that separating gravity will be 85.5%. The cumulative sink ash shows the theoretical ash content of the refuse at any yield of washed product (columns 8 vs. 10).

The elementary ash curve is a derivation of the cumulative percent ash in the float material and is intended to show the rate of change of the ash content at different specific gravities or yields. The curve is designed to show the highest ash content of any individual particle found in the float-coal product at any specific gravity. The cumulative percent ash-float curve at any specific gravity is the average ash analysis for all particles that float at that specific gravity. The elementary ash curve shows the highest theoretical ash content in-any one

particle. This curve is constructed by finding a new ordinate Z and plotting the new ordinate value against the actual ash value of each specific-gravity fraction as the abscissa. The value of ordinate Z is found by solving the following equation:

$$Z=X + Y/2$$

where X is the cumulative weight percent float of all materials of lower specific gravity and Y is the weight percent of the material at the given specific gravity directly above X. Z is then the new ordinate value, shown in Table 4-4, column 11, and is plotted against the percent ash of each gravity fraction (columns 11 vs. 3). Coe gives a detailed discussion of this calculation.

The elementary ash curve should be smooth. If it is not, an error exists in the cumulative float ash curves. The slope of the elementary ash curve indicates the relative ease of separation of the coal from the refuse. Steep slopes represent relatively small ash differences for large differences in yield, whereas a flat slope indicates an easy separation.

The ± 0.10 specific gravity distribution curve in Fig. 4-13 shows the percentage (by weight) of the coal that lies within ± 0.10 sp gr units at any given specific gravity. The curve is constructed in the following manner. The yield at 1.30 sp gr is subtracted from the yield at 1.50 sp gr as read from the specific-gravity curve in Fig. 4-13. When the amount of sink 1,80 material is large (greater than 10-15%), it is advantageous to adjust the numerical difference in the yields (the near-gravity material) by dividing this calculated difference by the yield (as a decimal) at 2.00 sp gr. The resulting adjusted percentage is plotted at 1.40 sp gr. The reason for dividing the difference in the two yields by the yield at 2.00 sp gr is that the material of higher specific gravity than 2.00, because of its rapid settling rate, does not interfere with the separation between washed coal and refuse at normal separating gravities. Failure to make this correction results in an absurd condition in which the addition of rock to the preparation-plant feed apparently decreases the difficulty of separation because it decreases the percentage of material within the ± 0.10 range. Subsequent points are calculated in an analogous manner at intervals of 0.05 sp gr.

As a guide for determining the lowest practical specific gravity to wash a particular coal, many operators use the ± 0.10 sp gr distribution curve. It is often arbitrarily designated that the point at which 10% of the total raw-coal feed lies within ± 0.10 sp gr of the separating gravity is the lowest specific gravity at which it is practical to operate a washing plant. A tabulation of the relative degree of difficulty in the washing of coal in jigs with various amounts of material that lie within ± 0.10 of the separating gravity is shown in Table 4-5. However, with specific coals and with the development of better float-sink vessels and heavy-medium cyclone, efficient separations can be made with coal that has more than 10% of the raw-coal feed lying within ± 0.10 sp gr of the separating gravity. For example, this criterion was not dependable for comparing results in recent work on Illinois No. 6 seam coal. This coal contained as much as 95% coal within ± 0.10 sp gr of the washing gravity required to produce coal of the desired ash content. The percentage ± 0.02 sp gr proved a more useful criterion. With other particular coals a range such as percentage ± 0.05 may be best.

It should be understood that the usefulness of Fig. 4-13 is limited to providing information on the theoretical results obtainable from the coal being tested. It is now well understood that the efficiency inherent in separating devices (the imperfection factor **) varies with the device or machine used and with the mean particle size of coal treated in a device. This projection of the expected coal quality from washability data must be made from the knowledge of what the ultimate washing scheme will be or should be. Computer programs have been developed to facilitate this task.

In addition to providing data for projecting the results to be expected when cleaning coal, washability tests

may be run on plant products to aid in evaluating the performance of cleaning plants. These tests establish whether the plants meet the performance guarantees of their constructors. At a later time the same tests might be used to determine how efficiently a plant is operating and, if necessary, what might be done to improve its performance.

Part 2 SIZE ANALYSIS AND COMMINUTION

Unit 1 Size Analysis

Introduction

Size analysis of the various products of a mill **constitutes** a fundamental part of laboratory testing procedure. **It is of great importance** in determining the quality of grinding and in establishing the degree of liberation of the values from the gangue at various particle sizes. In the separation stage, **size analysis** of the products **is used to** determine the optimum size of the feed to the process for maximum efficiency and **to** determine the size range at which **any losses are occurring** in the plant, so that they may be reduced.

It is essential, therefore, that methods of size analysis must be accurate and reliable, as important changes in plant operation may be made on the results of the laboratory tests. Since only relatively small amounts of material are used in the sizing tests, it is essential that the sample is (be) representative of the bulk material and the same care should be taken over sampling for size analysis as for assay.

Particle Size and Shape

The **primary function** of precision particle analysis **is to obtain quantitative data** about the size and size distribution of particles in the material. However, the **exact size** of an irregular particle **cannot be measured**. The terms length, breadth, thickness, or diameter have little meaning because so many different values of these quantities can be determined.

The size of a spherical particle is uniquely defined by its diameter. For a cube, the length along one edge is characteristic, and for other regular shapes there are equally appropriate dimensions.

For irregular particles, it is desirable to quote the size of a particle in terms of a single quantity, and the expression most often used is the "equivalent diameter". This refers to the diameter of a sphere that would behave in the same manner as the particle when submitted to some specified operation.

The **assigned equivalent diameter** usually **depends on the method of measurement**, hence the particle-sizing technique should, wherever possible, duplicate the process one wishes to control.

Several equivalent diameters are commonly encountered. For example, the **Stokes' diameter** **is measured** by sedimentation and elutriation techniques; the **projected area diameter** **is measured** microscopically and the **sieve-aperture diameter** **is measured** by means of sieving. **The last refers to the diameter** of a sphere equal to the width of the aperture **through which the particle just passes**. If the particles under test are not true spheres, and they rarely are in practice, this equivalent diameter refers only to their second largest dimension.

Recorded data from any size analysis should, where possible, be accompanied by some remarks which indicate the approximate shape of the particles. Descriptions such as "granular" or "acicular" are usually quite adequate to convey the approximate shape of the

particle in question.

Some of these terms are given below:

Acicular	needle-shaped.
Angular	sharp-edged or having roughly polyhedral shape.
Crystalline	freely developed in a fluid medium of geometric shape.
Dendritic	having a branched crystalline shape.
Fibrous	regularly or irregularly thread-like.
Flaky	plate-like.
Granular	having approximately an equidimensional irregular shape.
Irregular	lacking any symmetry.
Modular	having rounded, irregular shape.
Spherical	global shape.

A short list of some of the more common methods of size analysis, together with their effective size ranges, is given in Table 2.1.

Test sieving is the most widely used method for particle-size analysis. It covers a very wide range of particle size, this range being the one of most industrial importance. **So common is test sieving as a method of size analysis that** particles finer than about 75 micron are often referred to as being in the "sub-sieve" range, although modern sieving methods allow sizing to be carried out down to about 5 micron.

Test sieving is so common as a method of size analysis that particles finer than about 75 micron are often referred to as being in the "sub-sieve" range, although modern sieving methods allow sizing to be carried out down to about 5 micron.

e.g. Oxygen is so important to life that we can not live without it.

Table 2.1 Methods and Analysis Range

Method	Approximate useful range (micron)
Test sieving	100000-10
Elutriation	40-5
Microscopy	50-0.25
Sedimentation (Gravity)	40-1
Sedimentation (centrifugal)	5-0.05
Electron microscopy	1-0.005

elutriation 淘析, 淘洗,[采矿]洗提,[工]淘选

Polyhedral adj.多面的, 多面体的

Dendritic adj.树枝状的

Granular adj. 由小粒而成的, 粒状的

Equidimension n. 等尺寸, 同大小

Sieve Analysis (筛分分析)

Sieve analysis is one of the oldest methods of size analysis and is accomplished by passing a known weight of sample material successively through finer sieves and weighing the amount collected on each sieve to determine the percentage weight in each size fraction. Sieving is carried out with wet or dry materials and the sieves are usually agitated to expose all the particles to the openings.

Sieving, when applied to irregularly shaped particles, is complicated by the fact that a particle with a size near that of the nominal aperture of the test sieve may pass only when (it is) presented in a favorable position. As there is inevitably a variation in the size of sieve apertures, due to irregularity of weaving, prolonged sieving will cause the larger apertures to exert an unduly large effect on the sieve analysis. Given time, every particle small enough could find its way through a very few such holes. The procedure is also complicated in many cases by the presence of "near-size" particles which cause "blinding", or obstruction of the sieve apertures, and reduce the effective area of the sieving medium. Blinding is most serious with test sieves of very small aperture size.

The process of sieving may be divided into two stages; first the elimination of particles considerably smaller than the screen apertures, which should occur fairly rapidly and, secondly, the separation of the so-called "near-size" particles, which is a gradual process rarely reaching final completion. Both stages require the sieve to be manipulated in such a way that all particles have opportunities for passing the apertures, and so that any (particles) which blinds an aperture may be removed from it. Ideally, each particle should be presented individually to an aperture, as is permitted (pass through, possible) for the largest aperture sizes, but for most sizes this is impracticable.

The effectiveness of a sieving test depends on the amount of material put on to the sieve (the "charge") and the type of movement imparted to the sieve.

A comprehensive account of sampling techniques for sieving is given in BS 1017. Basically, if the charge is too large, the bed of material will be too many particles deep to allow each one a chance to meet an aperture in the most favorable position for sieving in a reasonable time. The charge, therefore, is limited by a requirement for the maximum amount of material retained at the end of sieving appropriate to the aperture size. On the other hand, the sample must contain enough particles to be representative of the bulk, so a minimum size of sample is specified. In some cases, the sample will have to be subdivided into a number of charges if the requirements for preventing overloading of the sieves are to be satisfied.

Test Sieves (试验筛)

Test sieves are designated by the nominal aperture size, which is the nominal central separation of opposite sides (两对边标称中心距) of a square aperture or the nominal

diameter of a round aperture. A variety of sieve aperture ranges are currently used, the most popular being the German Standard, DIN 4188; ASTM standard, E11; the American Tyler series; the French series, AFNOR; and the British Standard, BSS 410.

Until relatively recently woven-wire sieves were designated (标明, 标示, indicated, specified) by a mesh number, which referred to the number of wires per inch, which is the same as the number of square apertures per inch. This method was widely used, and until 1962 was the basic designation in BS 410. It has the serious disadvantage that the same mesh number on the various standard ranges corresponds to different aperture sizes, depending on the thickness of wire used in the woven-wire cloth. Sieves are now designated by aperture size, which gives the user directly the information he needs.

Since many workers still refer to sieve sizes in terms of mesh number, Table 2.2 lists mesh numbers for the BSS 410 series against nominal aperture-size.

TABLE 2.2. BSS 410 WIRE-MESH SIEVES

Mesh number	Nominal aperture size(um)	Mesh number	Nominal aperture size (um)
3	5600	36	425
3.5	4750	44	355
4	4000	52	300
5	3350	60	250
6	2800	72	212
7	2360	85	180
8	2000	100	150
10	1700	120	125
12	1400	150	106
14	1180	170	90
16	1000	200	75
18	850	240	63
22	710	300	53
25	600	350	45
30	500	400	38

Wire-cloth screens are woven to produce nominally uniform square apertures within required tolerances. Wire cloth in sieves with a nominal aperture of 75 micron and greater are plain woven, while those in cloths with apertures below 63 micron may be twilled (Fig. 2.1).

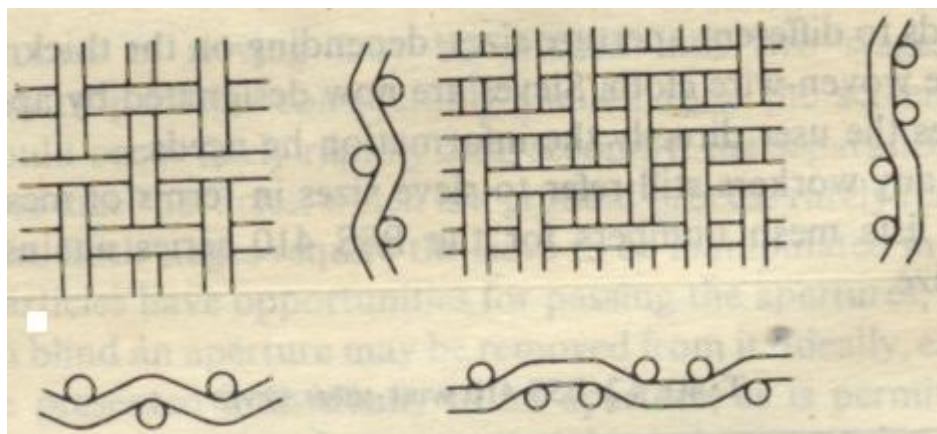


Figure 2.1 plain and twilled sieves

Standard test sieves are not available with aperture sizes smaller than about 37 μm . Micromesh sieves are available in aperture sizes from 5 μm to 150 μm , and are made by electroforming nickel in square and circular mesh. Another popular type is the "micro-plate sieve" which is fabricated by electroetching a nickel plate. The apertures are in the form of truncated cones with the small circle uppermost. This reduces blinding but also reduces the percentage open area, i.e. the percentage of the total area of the sieving medium occupied by the apertures.

It has long been realized that a useful [sieve scale](#)(筛序) is one (代词, 统一, 这样) in which the ratio of the aperture widths of adjacent sieves is the square root of 2. Most modern sieve series are based on a fourth root of 2 ratio or a tenth root of 10, which makes possible much closer sizing of particles.

For most size analyses it is usually impractical and unnecessary to use all the sieves in a particular series. In general, the sieve range should be chosen such that no more than about 5% of the sample is retained on the coarsest sieve, or passes the finest sieve.

Testing procedures

Machine sieve is almost universally used, as hand sieving is long and tedious, and its accuracy depends to a large extent on the operator.

The sieves chosen for the test are arranged in a stack, or nest, with the coarsest sieve on the top and the finest on the bottom. The material to be tested is placed in the uppermost [coarsest sieves](#), and the nest is then placed in a sieve shaker which vibrates the material in a vertical plane or horizontal plane. The duration of screening can be controlled by an automatic timer. After the required time, the nest is taken apart and the amount of material retained on each sieve weighed. Most of the near mesh particles which block the openings, [can](#) be removed by inverting the sieve and tapping the frame gently. Failing this, the underside of the gauze may be brushed gently with a soft nylon brush.

Presentation of particle size analysis Result

There are several ways in which the results of a sieve test can be tabulated. The three most convenient methods are shown in the following table (Table 2.3).

Table 2.3 Results of Typical Sieve Test

Sieve size range	Sieve fractions		Nominal aperture size	Cumulative undersize	Cumulative oversize
micron	Wt(g)	% wt	micron	%	%
+250	0.02	0.1	250	99.9	0.1
-250+180	1.32	0.29	180	97.0	3.0
-180+125	4.23	9.5	125	87.5	12.5
-125+90	9.44	21.2	90	66.3	33.7
-90+63	13.10	29.4	63	36.9	63.1
-63+45	11.56	26.0	45	10.9	89.1
-45	4.87	10.9			

The table shows:

- (1) The sieve size ranges used in the test.
- (2) The weight of material in each size range, e.g. 1.32 g of material, passed through the 250 micron sieve, but was retained on the 180 micron sieve: the material therefore is in the size range $-250+180\text{ }\mu\text{m}$.
- (3) The weight of material in each size range expressed as a percentage of the total weight.
- (4) The nominal aperture sizes of the sieves used in the test.
- (5) The cumulative percentage of material passing through these sieves, e.g. 87.5% of the material, is less than $125\text{ }\mu\text{m}$ in size.
- (6) The cumulative percentage of material retained on the sieves.

The results of a sieving test should always be plotted graphically in order to assess their full significance.

There are many different ways of recording the results, the most common being that of plotting cumulative undersize (or oversize) against particle size. Although arithmetic graph paper can be used, it suffers from the disadvantage that points in the region of the finer aperture sizes tend to become congested. A semi-logarithmic plot avoids this, with a linear ordinate for percentage oversize or undersize and a logarithmic abscissa for particle size. *(please show graphically the results of the sieve test tabulated in Table 2.3)*

It is not necessary to plot both cumulative oversize and undersize curves as they are mirror images of each other. A valuable quantity which can be determined from such curves is the "median size" of the sample. This refers to the mid-point in the size distribution-50% of the particles are smaller than this size and 50% are larger.

Size analysis is very important in assessing the performance of grinding circuits. The "mesh of grind" is usually quoted in terms of one point on the cumulative undersize curve, this often being the 80% passing size. Although this does not show the overall size *distribution* of the material, it does facilitate routine control of the grinding circuit. For instance, if the target mesh of grind is 80% minus 250 μm , then, for routine control, the operator need only screen a fraction of the mill product at this one size. If it is found that, say, 50% of the sample is minus 250 μm , then the product is too coarse, and control steps to remedy this can swiftly be made.

Many curves of cumulative oversize or undersize against particle size are S-shaped, leading to congested plots at the extremities of the graph. More than a dozen methods of plotting in order to proportion the ordinate are known. The two most common methods, which are often applied to comminution studies, where non-uniform size distributions are obtained, are the Gates-Gaudin -Schuhmann (G-G-S) and the Rosin-Rammler(R-R) methods. Both methods are derived from attempts to represent particle size distribution curves by means of equations, which results in scales which, relative to a linear scale, are expanded in some regions and contracted in others.

The Rosin-Rammler method is often used for representing the results of sieve analyses

performed on material which has been ground in ball mills. Such products have been found to obey the following relationship:

$$100-P=100\exp(-bd^n)$$

where P is the cumulative undersize in per cent, b is a constant, d is the particle size, and n is a constant.

This can be rewritten as:

$$\log \left[\ln \frac{100}{100-P} \right] = \log b + n \log d$$

thus, a plot of $\log(100/100-P)$ versus d on log-log axes gives a line of slope n.

In comparison with the G-G-S $\log(\text{cumulative undersize}) - \log(\text{sieve aperture})$, the R-R plots expands the regions below 25% and above 75% cumulative undersize and it contracts the 30-60% region.

Aperture n.孔, 穴, 缝隙, (照相机, 望远镜等的)光圈, 孔径

Tolerance n.公差, 宽容, 忍受, 容忍, (食物中残存杀虫剂的)(法定)容许量

vt.给(机器部件等)规定公差

twill n.斜纹织物,vt.把...织成斜纹,adj.斜纹织物的

electroform vt.电铸,

electroetching n.电蚀刻

truncated adj.切去顶端的, 缩短了,被删节的

ordinate n.[数]纵线, 纵坐标

abscissa n.横坐标

congested adj.拥挤的

gauze n.薄纱, [医]纱布, 薄雾

Reading Material Screening of Coal

Screening is a mechanical process that separates particles by size. Modern coal preparation is made possible by efficient sizing. Sizing of coal is defined in terms of the opening through which an individual particle will or will not pass, or as two openings, the larger of which passes the particle and the smaller of which retains the particle. Screening accomplishes sizing by passing coal of a wide range of sizes through a series of screens, each of a decreasing size. The individual screen discharges are then directed into different screen products for subsequent sale or processing.

Screening in modern preparation plants is done for the following reasons:

1. Removal of fine coal before a size reduction step.
2. Size separation of the coal into a coarse and fine fraction for sized products or processing in coarse and fine coal circuits.

3. Recovery of heavy media from heavy media coal cleaning circuits.
4. Dewatering.
5. Rock and trash removal by handpicking on a screen.

Factors affected on the screen process

Size of coal relies on the mechanical process of presenting a feed material to a screen surface. Commercial screening, which is a continuous process, results in imperfect separations. A perfect separation will have been achieved when all the particles capable of passing through the screen have actually passed through. Commercial screening seldom is designed to achieve perfect separation. A number of factors affect the performance of commercial screens. Factors that influence particle passage through a screen are given below:

1. variables due to the material being screened are the particle shape and size relative to mesh shape, the proportion of near-size, undersize and oversize in feed, the moisture control and stickiness of particles, the resistance to degradation, the bulk density of the material, and electrostatic bunching.
2. variables due to type of screen installation and operation are the length and width of the screen, the amplitude and frequency of vibration, the tautness of the screen mat, the direction of vibration impulses, the feed rate, the type of screen motion, and the angle of incidence and evenness,
3. variables relative to the screen mat are the type of screen mat, the percentage of open screen area, the shape of the screen aperture, the tautness of the screen mat, the corrosion of the screen mat, the feed rate to prevent blinding, the materials of construction to withstand distortion of the mesh with time, and the uniformity of mesh size over the area of the screen.

Because of a multitude of factors, both acting individually and interacting with one another, the principles of screening are not well defined.

Serried adj. 密集的, 林立的, 重叠罗列的

Taut adj. (绳子) 拉紧的, 整洁的, 紧张的

Mat n. 席子, 垫子; v. 铺席子

Bunching [电子] 聚束

Efficiency

Since the purpose of screen is to remove fine material from the feed to a screen, screen efficiency is:

screen efficiency = (Amount of fines in feed reporting to screen underflow) / (Amount of fines in feed)

The size of separation is usually defined by the minimum screen aperture, In practice the screen efficiencies are measured by sampling the feed, screen oversize, and screen undersize.

Screen types

Vibrating screens are the screens of choice in modern practice. The range of screen aperture sizes is 10 in.-100 mesh (254-0.15mm). the slope will vary from 16-26°. The stroke depends upon the size of separation and varies from 12.7-0.8mm. The frequency ranges from 660-3400 for the finest separations.

Dewatering screens are used for dewatering fine coal or refuse.

Sieve bends are used both for sizing and dewatering service. The sieve bend aperture is from 1-0.125mm and the width is from 0.305-1.52m. It has no moving parts and low operating costs and has seen increasing use in fine coal circuits.

Other types of screens also have found application in coal preparation, but do not have as wide an application as the aforementioned types.

Screen surfaces

Woven wire cloth has a wide selection of apertures from 5 in.-500 mesh (125-0.03mm). The screens can be manufactured from a wide variety of metals and alloys. For coarse screen sizes, steel and high-carbon steel are the preferred choices. For finer sizes, stainless steels are often used because of their resistance to corrosion. Square or rectangular- opening screen has less tendency to blind with coal that is difficult to screen.

Perforated screen plate is used for coarse screening where long service life is a requirement. The openings can be square, rectangular, hexagonal, or round. The screens can be made of steel, stainless steel, or rubber-coated steel.

Rod screens constructed of stainless steel are being used to provide long life. Shapes are round or triangular. For coal that is prone to blind on a conventional surface, various screens have been developed.

Plastic screens have been developed using rubber and urethane materials. The screens have an extended life over standard screens, although the cost is higher. Advantages claimed are less tendency to blind and uniformity of screen openings throughout the life of the screen.

Unit 2 COMMINATION

Introduction

Because most minerals are finely disseminated and intimately associated with the gangue, they must be initially "unlocked" or "liberated" before separation can be undertaken. This is achieved by *comminution*, in which the particle size of the ore is progressively reduced until the clean particles of mineral can be separated by *such* (形容词) *methods as* (as 从句, 说明 methods) are available. Comminution in its earliest stages is carried out in order to make the freshly excavated material easier to handle by scrapers, conveyors, and ore carriers, and in the case of quarry products, to produce material of controlled particle size.

Explosives are used in mining to remove ores from their natural beds, and blasting can be regarded as the first stage in comminution. Comminution in the mineral processing plant, or "mill", takes place as a sequence of crushing and grinding processes. Crushing reduces the particle size of run-of-mine ore to such a level that grinding can be carried out until the mineral and gangue are substantially produced as separate particles.

Crushing is accomplished by compression of the ore against rigid surfaces, or by impact against surfaces in a rigidly constrained motion path. This is contrasted with grinding which is accomplished by abrasion and impact of the ore by the free motion of unconnected media such as rods, balls, or pebbles.

Crushing is usually a dry process, and is performed in several stages, with small *reduction ratios* ranging from three to six in each stage. The reduction ratio of a crushing stage can be defined as the ratio of maximum particle size entering to maximum particle size leaving the crusher, although other definitions are sometimes used.

Tumbling mills with *either* steel rods or balls, *or* sized ore as the grinding media, **are used** in the last stages of comminution. Grinding is usually performed "wet" to provide a slurry feed to the concentration process, although dry grinding has limited applications. There is an overlapping size area where it is possible to crush or grind the ore. It has been concluded by Steane that it is cheaper to grind, and that crushing should be carried no further than is necessary to provide suitable feed for the grinding mills.

Principles of Comminution

Most minerals are crystalline materials *in which* the atoms are regularly arranged in three-dimensional arrays. The configuration of atoms is determined by the size and types of physical and chemical bonds holding them together. In the crystalline lattice of minerals, these inter-atomic bonds are effective only over small distances, and can be broken if extended by a tensile stress. Such stresses may be generated by tensile or compressive loading (Fig.1).

Even when rocks are uniformly loaded, the internal stresses are not evenly distributed, as the

rock consists of a variety of minerals dispersed as grains of various sizes. The distribution of stress depends upon the mechanical properties of the individual minerals, but more importantly, upon the presence of cracks or flaws in the matrix, which act as sites for stress concentration (Fig. 2).

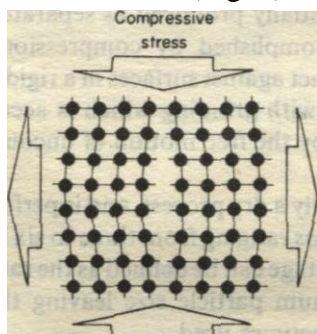


FIG.1. Strain of a crystal lattice resulting from tensile or compressive stresses.

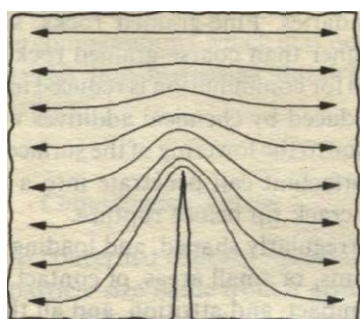


Fig.2. Stress concentration at a crack tip.

It has been shown that the increase in stress at such a site is proportional to the square root of the crack length perpendicular to the stress direction. Therefore there is a critical value for the crack length at any particular level of stress at which the increased stress level at the crack tip is sufficient to break the atomic bond at that point. Such rupture of the bond will increase the crack length, thus increasing the stress concentration and causing a rapid propagation of the crack through the matrix, thus causing fracture.

Although the theories of comminution assume that the material is brittle, crystals can, in fact, store energy without breaking, and release this energy when the stress is removed. Such behavior is known as *elastic*. When fracture does occur, some of the stored energy is transformed into free surface energy, which is the potential energy of atoms at the newly produced surfaces. Due to this increase in surface energy, newly formed surfaces are often more chemically active, and are more amenable to the action of flotation reagents, etc., as well as oxidizing more readily.

Griffith showed that materials fail by crack propagation when this is energetically feasible, i.e. when the energy released by relaxing the strain energy is greater than the energy of the new surface produced. Brittle materials relieve the strain energy mainly by crack propagation, whereas "tough" materials can relax strain energy without crack propagation, by the mechanism of *plastic flow*, where the atoms or molecules slide over each other and energy is

consumed in distorting the shape of the material. Crack propagation can also be inhibited by encounters with other cracks or by meeting crystal boundaries. Fine-grained rocks, such as taconites, are therefore usually tougher than coarse-grained rocks.

The energy required for comminution is reduced in the presence of water, and can be further reduced by chemical additives which adsorb onto the solid. This may be due to the lowering of the surface energy on adsorption providing that the surfactant can penetrate into a crack and reduce the bond strength at the crack tip before rupture.

Real particles are irregularly shaped, and loading is not uniform, but is achieved through points, or small areas, of contact. Breakage is achieved mainly by crushing, impact, and attrition, and all three modes of fracture (compressive, tensile, and shear) can be discerned depending on the rock mechanics and the type of loading.

When an irregular particle is broken by compression, or crushing, the products fall into two distinct size ranges-coarse particles resulting from the induced tensile failure, and fines from compressive failure near the points of loading, or by shear at projections (Fig.3). The amount of fines produced can be reduced by minimising the area of loading and this is often done in compressive crushing machines by using corrugated crushing surfaces.

In impact breaking, due to the rapid loading, a particle experiences a higher average stress while undergoing strain than under slow loading. As a result the particle absorbs more energy than is necessary to achieve simple fracture, and tends to break apart rapidly, mainly by tensile failure. The products are often very similar in size and shape.

Attrition (shear failure) produces much fine material, and is usually

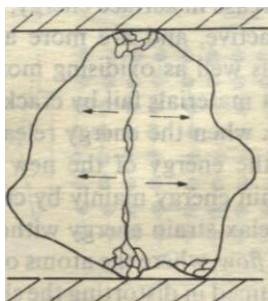


FIG. 3. Fracture by crushing.

undesirable. Attrition occurs mainly in practice due to particle-particle interaction (inter-particle comminution), which may occur if a crusher is fed too fast, contacting particles thus increasing the degree of compressive stress and hence shear failure.

Array n. 点阵, 排列, 编队, 军队, 衣服, 大批 vt. 部署, 穿着, 排列

Tensile adj. 可拉长的, 可伸长的, [物]张力的, 拉力的

Flaw n. 缺点, 裂纹, 瑕疵, 一阵狂风 vt. 使破裂, 使有缺陷, 使无效 vi. 生裂缝, 变的有缺陷

Rupture v. 破裂, 裂开, 断绝(关系等), 割裂 n. 破裂, 决裂, 敌对, 割裂

Strain n. 应变, 过度的疲劳, 紧张, 张力 vt. 扭伤, 损伤 v. 拉紧, 扯紧, (使)紧张, 尽力

Discern v. 目睹, 认识, 洞悉, 辨别, 看清楚

Corrugated adj. 缩成皱纹的, 使起波状的

Comminution Theory

Comminution theory is concerned with the relationship between energy input and the product particle size made from a given feed size. Various theories have been expounded, none of which is entirely satisfactory.

The greatest problem lies in the fact that most of the energy input to a crushing or grinding machine is absorbed by the machine itself, and only a small fraction of the total energy is available for breaking the material. It is to be expected that there is a relationship between the energy required to break the material and the new surface produced in the process, but this relationship can only be made manifest if the energy consumed in creating new surface can be separately measured.

In a ball mill, for instance, it has been shown that less than 1 % of the total energy input is available for actual size reduction, the bulk of the energy being utilized in the production of heat.

Another factor is that a material which is plastic will consume energy in changing shape, a shape which it will retain without creating significant new surface. All the theories of comminution assume that the material is brittle, so that no energy is absorbed in processes such as elongation or contraction which is not finally utilized in breakage.

The oldest theory is that of Rittinger (1867), which states that the energy consumed in the size reduction is proportional to the area of new surface produced. The surface area of a known weight of particles of uniform diameter is inversely proportional to the diameter, hence Rittinger's law equates to

$$E = K \left(\frac{1}{D_2} - \frac{1}{D_1} \right)$$

where E is the energy input, D_1 is the initial particle size, D_2 is the final particle size, and K is a constant. The second theory (1885) is that of Kick. He stated that the work required is proportional to the reduction in volume of the particles concerned. Where f is the diameter of the feed particles and p the diameter of the product particles, the reduction ratio R is f/p . According to Kick's law, the energy required for comminution is proportional to $\log R/\log 2$.

Bond developed an equation which is based on the theory that the work input is proportional to the new crack tip length produced in particle breakage, and equals the work represented by the product minus that represented by the feed. In particles of similar shape, the surface area of unit volume of material is inversely proportional to the diameter. The crack length in unit volume is considered to be proportional to one side of that area and therefore inversely proportional to the square root of the diameter.

For practical calculations the size in microns which 80% passes is selected as the criterion of particle size. The diameter in microns which 80% of the product passes is designated as P , the

size which 80% of the feed passes is designated as F , and the work input in kilowatt hours per short ton is W . Bond's third theory equation is

$$W = \frac{10W_i}{\sqrt{P}} - \frac{10W_i}{\sqrt{F}}$$

where W_i is the *work index*. The work index is the comminution parameter which expresses the resistance of the material to crushing and grinding; numerically it is the kilowatt hours per short ton required to reduce the material from theoretically infinite feed size to 80% passing 100 μm .

Various attempts have been made to show that the relationships of Rittinger, Kick, and Bond are interpretations of single general equations. Hukki suggests that the relationship between energy and particle size is a composite form of the three laws. The probability of breakage in comminution is high for large particles, and rapidly diminishes for fine sizes. He shows that Kick's law is reasonably accurate in the crushing range above about 1 cm in diameter; Bond's theory applies reasonably in the range of conventional rod-mill and ball-mill grinding, and Rittinger's law applies fairly well in the fine grinding range of 10-1000 μm .

Grindability

Ore grindability refers to the ease with which materials can be comminuted and data from grindability tests is used to evaluate crushing and grinding efficiency.

Probably the most widely used parameter to measure ore grindability is the Bond work index W_i . If the breakage characteristics of a material remain constant over all size ranges, then the calculated work index would be expected to remain constant since it expresses the resistance of material to breakage. However, for most naturally occurring raw materials, differences exist in the breakage characteristics depending on particle size, which can result in variations in the work index. For instance, when a mineral breaks easily at the boundaries but individual grains are tough, then grindability increases with fineness of grind. Consequently work index values are generally obtained for some specified grind size which typifies the comminution operation being evaluated.

Grindability is based upon performance in a carefully defined piece of equipment according to a strict procedure. Bond has devised several methods for predicting ball-mill and rod-mill energy requirements, which provide an accurate measure of ore grindability.

Table 2.4 lists standard Bond work indices for a selection of materials.

The standard Bond test is time consuming, and a number of methods have been used to obtain the indices related to the Bond work index. Smith and Lee used batch-type grindability tests to arrive at the work index, and compared their results with work indices from the standard Bond tests, which require constant screening out of undersize material in order to simulate closed-circuit operation. The batch-type tests compared very favorably with the standard grindability test data, the advantage being that less time is required to determine the work index.

TABLE 2.4. SELECTION OF BOND WORK INDICES

Material	Work index	Material	Work index
Barite	4.73	Fluorspar	8.91
Bauxite	8.78	Granite	15.13
Coal	13.00	Graphite	43.56
Dolomite	11.27	Limestone	12.74
Emery	56.70	Quartzite	9.58
Ferro-silicon	10.01	Quartz	13.57

Reading material SIZE REDUCTION OF COAL

INTRODUCTION

Run-of-mine (ROM) coal generally falls into two major groups, that from underground mining (continuous mining machines) and that from strip mining. Continuous miners produce a finer product and strip mining a coarser product that is crushed to produce the *desired* size. Further size reduction depends on the proposed use or uses for the coal. For example, for **pulverized coal power plants** as a rough general rule, the coal is *ground* or *pulverized* to 80% less than 200 mesh (0.075 mm) before it is blown into the boiler furnace as a coal-air suspension.

For **solvent-refined** coal processes, the desired size is about <30 mesh (<0.60 mm); in fluidized bed uses, the coal is typically <1/16 in (<1.6 mm); for **Lurgi pressure gasifiers**, the desired size is 25-63 mm. Coal preparation plants typically work with a coarse size fraction of 150x32 mm.

It seems probable that the next decades will see much wider application of coal cleaning before use and that the stages of crushing and pulverization from ROM coal to end use will be integrated with coal cleaning processes. Since different cleaning processes work better with different ranges of coal size and the degree of cleaning possible is intimately connected with the size of coal, a systems design engineer will need as much information as possible concerning the size distributions produced by different types of machines and how these can be modified by different ways of operating the machines.

In general, the following questions will be of concern:

- 1) What type of **crusher or grinder** is best, or most satisfactory, for reducing a given feed size and type of coal to a desired product size? Frequently, the desired product is smaller than some nominal size, but with as few fines as possible, or the fraction in some size range has to be as high as possible.
- 2) How big does the machine have to be for a given throughput rate?
- 3) How much electrical energy (or its equivalent) is required per ton of product?
- 4) How does the size distribution vary with change in throughput rate, and is a control scheme necessary or desirable?
- 5) Can the size distribution be readily varied if desired?
- 6) What is the optimum way to operate a machine in a given system?

This section emphasizes certain general principles that can act as guides in answering these questions.

PARTICLE SIZING

Since particle size and size distribution are the raw data with which we work, it is worthwhile to review the terminology used. The most widely used measure of particle size for coals is *sieve size*, where a particle is greater than a given sieve size if it does not pass through the holes in the screen on repeated presentations to the holes. Since screens are available in a $2^{1/4}$ progression of sizes, particles are

characterized as being greater than a given size and less than the next screen up--e.g., 18 x 20 mesh (1.0 x 0.85 mm). The lowest size normally used is 400 mesh (0.038 mm), and to get complete sieving through this size usually requires wet-spray sieving (sieving plus wetting agent). If the fractions-by-weight between sizes are measured experimentally, accumulated, and plotted, a *cumulative distribution curve* is obtained.

Types of crushers

ROM coal from strip mining usually contains large lumps of coal, and a number of different types of machines are efficient in crushing these large sizes. A popular piece of equipment is the **single roll crusher**; it is a simple and robust (精力充沛的) machine with low fines production. It consists of a heavy cast iron or steel fabricated frame on which are mounted the crushing roll and the stationary breaker plate. The breaker plate is provided with renewable wear plates bolted to the breaker plate. The roll usually has a series of long teeth spaced at intervals and various short teeth covering the entire crushing surface. The coal is squeezed between the revolving roll and the breaker plate. The long teeth act as feeders and also penetrate the lumps of coal, splitting them into smaller pieces, while the smaller teeth make the proper size reduction. Another popular type of crusher is **double roll breaker**. Instead of the breaker plate in the single roll crusher, the other roll is mounted on the machine frame.

A different type of breakage is used in the **rotary breaker** (Bradford breaker). The breaker essentially consists of an outer cylinder with an inner rotating perforated cylinder fitted with lifters. Coal is fed in one end of the rotating cylinder, where it breaks because of the tumbling action due to gravity fall, and pieces less than the hole sizes pass out to a bottom collection trough, giving a product with positive upper size control.

Breakage is induced by impact force are called **impactor or hammer mills**. The **cage mill** employs multiple selective stages of impact reduction by rotating cages.

Part 3 GRAVITY SEPARATION PROCESSES

Unit 1 Gravity Concentration

Gravity concentration is the method of separating grains of minerals of different specific gravities by reason of their differences in movement *in response to* the joint simultaneous actions *upon* them *of* gravity and one or more other forces. In the great majority of gravity concentration processes and apparatus *the other force used is the resistance to downward penetration* offered by a medium *that has more ore less fluid properties*—a gas, usually air; water or some homogeneous liquids or a mechanical mixture of a fluid and a particulate solid maintained more or less in suspension. The fluid properties used are density and viscosity; the forces which they *bring to bear* (bring to bear v.施加, 运用) are buoyancy and upward impulse.

Gravity methods of separation are used to treat a great variety of materials, *ranging from* heavy metal sulphides such as galena (sp.gr.7.5) *to* coal (sp.gr.1.3), at particle sizes in some cases below 50 micron.

These methods declined in importance in the first half of the last century due to the development of the froth flotation process, which allows the selective treatment of low-grade complex ores. They remained, however, the main concentrating methods for iron and tungsten ores and are used extensively for treating tin ores. Although flotation techniques for cassiterite have been developed in the 1970's, about 85% of the world's tin is still produced by gravity separation. Whenever gravity methods are chosen in preference to flotation, it is usually because relative costs favor the application. Minerals which are liberated at sizes above the normal flotation range may be concentrated even more economically using gravity methods.

In recent years, many companies have re-evaluated gravity systems due to increasing costs of flotation reagents, the relative simplicity of gravity processes, and the fact that they produce comparatively little environmental pollution.

Modern gravity techniques have proved efficient for concentration of mineral having particle sizes in the 50-10 micron range and *when coupled with improved pumping technology and instrumentation*, (modern gravity techniques) have been incorporated in high-capacity plants.

In many cases a high proportion of the mineral in an ore-body can at least be pre-concentrated effectively by cheap and ecologically acceptable gravity systems; the amount of reagents and fuel used can be cut significantly when the more expensive methods are restricted to the processing of gravity concentrate. *Gravity separation of minerals* at coarser sizes as soon as liberation is achieved *can also have significant advantages* for later treatment stages due to decreased surface area, more efficient dewatering, and the absence of adhering chemicals that could interfere with further processing.

Gravity techniques to recover residual valuable heavy minerals in flotation tailings *are being increasingly used*. Apart from current production, there are many large tailings dumps that could be excavated cheaply and processed to give high value concentrates using recently developed technology.

Gravity concentration methods separate minerals of different specific gravity by their relative

movement in response to gravity and one or more other forces, the latter often being the resistance to motion offered by a viscous fluid, such as water or air.

It is essential for effective separation that a marked density difference exists between the mineral and the gangue. Some idea of the type of separation possible can be gained from the concentration criterion

$$(D_h - D_f) / (D_l - D_f)$$

where D_h is the specific gravity of the heavy mineral, D_f is the specific gravity of the fluid medium, and D_l is the specific gravity of the light mineral.

In very general terms, when the quotient is greater than 2.5, whether positive or negative, then gravity separation is relatively easy. As the value below about 1.25 gravity concentration is not generally commercially feasible.

The motion of a particle in a fluid is dependent not only on its specific gravity, but also on its size; large particles will be affected more than smaller ones. The efficiency of gravity processes therefore increases with particle size, and the particles should be sufficiently coarse to move in accordance with Newton's law. Particles which are *so small that* their movement is dominated mainly by surface friction respond relative poorly to commercially high capacity gravity methods. In practice, close size control of feeds to gravity processes required in order to *reduce* the size effect and *make* the relative motion of the particles *specific gravity* is dependent.

It is essential for the efficient operation of all gravity separators that the feed is carefully prepared. Grinding is particularly important in that the feed particles should be as coarse as possible consistent with adequate liberation; successive regrinding of middlings is required in most operations. Primary grinding should be performed where possible in *open-circuit rod mills*, but if fine grinding is required, close-circuit ball milling should be used, preferably with screens closing the circuit rather than hydrocyclones in order to reduce selective overgrinding of heavy friable valuable minerals.

Gravity separators are extremely sensitive to the presence of slimes, which increase the viscosity of the slurry and hence the sharpness of separation, and obscure visual cut-point. It is common practice in most gravity concentrators to remove particles less than about 10 micron from the feed, and divert this fraction to the tailings, and this can account for considerable loss of values. De-sliming is often achieved by the use of hydrocyclones, although if hydraulic classifiers are used to prepare the feed, it may be preferable to de-slime at this stage, since the high shear forces produce in hydrocyclones tend to cause degradation of friable minerals. In conjunction with de-sliming, in most cases, removal of the oversize by screening is necessary.

Many different machines have been designed and built in the past to affect separation of minerals by gravity, and they are extensively reviewed in old textbooks of mineral processing. Main equipment used in modern mills will be dealt with here.

Reading Material Laboratory Heavy Liquid Tests

Laboratory testing may be performed on ores in order to assess the suitability of heavy medium separation on the crushed material and to determine the economic separating density.

Liquids covering a range of densities in incremental steps are prepared, and the representative sample of

crushed ore is introduced into the liquid of highest density. The floats product is removed and washed and placed in the next lower density and so on. The sinks product is finally drained, washed, and dried, and then weighed, together with the final floats product, to give the density distribution of the sample by weight.

Care should be taken when evaluating ores of fine-particle size that sufficient time is given for the particles to settle into the appropriate fraction. Centrifuging is often carried out on fine materials to save the setting time, but this should be done with care, as there is a tendency for the floats to become entrained in the sinks fraction. Unsatisfactory results are often obtained with porous materials, such as magnesite (菱镁矿) ores, due to the entrainment of liquid in the pores, which changes the apparent density of the particles.

After assaying the fractions for metal content, the distribution of material and detail in the density fractions of the sample can be tabulated. Table 3.1 shows such a distribution from tests performed on a tin ore, it can be seen from columns 3 and 6 of the table that if a separation density of 2.75 was chosen, then 68.48% of the material, being lighter than 2.75, would be discarded as a float product, and only 3.81% of the tin would be lost in this fraction. Similarly, 96.19% of the tin would be recovered into the sink product, which accounts for 31.52% of the original total feed weight. For more precise evaluation of the results of laboratory tests the results should be plotted graphically.

Table 3.1 heavy liquid test results

Fraction of density	Weight%	Cumulative weight%	Assay%(Sn%)	Distribution(Sn%)	Cumulative distribution(Sn%)
-2.55	1.57	1.57	0.003	0.04	0.04
2.55-2.60	9.22	10.79	0.04	0.33	0.37
2.60-2.65	26.11	36.90	0.04	0.93	1.30
2.65-2.70	19.67	56.57	0.04	0.70	2.00
2.70-2.75	11.91	68.48	0.17	1.81	3.81
2.75-2.80	10.92	79.40	0.34	3.32	7.13
2.80-2.85	7.87	87.27	0.37	2.60	9.73
2.85-2.90	2.55	89.82	1.30	2.96	12.69
+2.90	10.18	100.00	9.60	87.31	100

Heavy liquid tests are important in coal preparation in order to determine the required density of separation and the expected yield of coal of the required ash content. The ash content refers to the amount of incombustible material in the coal. Since coal is lighter than the contained mineral. The higher the density of separation, the higher is the yield, but the higher is the ash content.

Unit 2 Jig

Jigging is one of the oldest methods of gravity concentration, yet even today the basic principles are not completely understood. The jig is normally used to concentrate relatively coarse material and, if the feed is fairly close sized (e.g. 3-10mm), it is not difficult to achieve good separation of a fairly narrow specific gravity range in minerals in the feed. When the specific gravity difference is large, good concentration is possible with a wider size range. Many large jig circuits are still operated in the coal, cassiterite, tungsten, gold, barites, and iron-ore industries. They have relatively high unit capacity on classified feed and can achieve good recovery of values down to 150 micron and acceptable recoveries often down to 75 micron for metallic ores.

In jig, the separation of minerals of different specific gravity is accomplished in a bed which is **rendered fluid** by a pulsating current of water so as to produce stratification. The aim is to dilate the bed of material being treated and control the dilation so that the heavier, smaller particles

penetrate the interstices of the bed and the larger high specific gravity particles fall under a condition probably similar to hindered settling.

Stratification n.层化, 成层, 阶层的形成

Interstice n.空隙, 裂缝

On the pulsion stroke the bed is normally lifted as a mass, then as the velocity decreases it tends to dilate, the bottom particles falling first until the whole bed is loosened. On the suction stroke it then closes slowly again and this is repeated at every stroke, the frequency usually varying between 55-330 c min⁻¹. Fine particles tend to pass through the interstices after the large ones have become immobile. The motion can be obtained either by using a fixed sieve jig, and pulsating the water, or by employing a moving sieve, as in the simple hand-jig.

Theoretically, the initial acceleration of the mineral grains is independent of size and dependent only on the densities of the solid and the fluid. If the duration of fall is short enough and the repetition of fall frequent enough, the total distance traveled by the particles will be affected more by the differential initial acceleration, and therefore by density, than by their terminal velocities and therefore by size. In other words, to separate small heavy mineral particles from larger, light particles a shortly jiggling cycle is necessary. Although relatively short fast strokes are used to separate fine materials, more control and better stratification can be achieved by using longer, slower strokes, especially with the coarser particle sizes. It is therefore good practice to screen the feed to jigs into different size ranges and treat these separately.

Jigs are classified on the basis of the method of effecting dilation of the bed. Thus, broadly, there are fixed-sieve jigs, in which the dilating fluid, usually water, is caused to pulsate while the bed is stationary in space; and movable-sieve types in which the bed as a whole moves in a stationary body of water.

One of the oldest jig is the *Harz* in which the plunger moves up and down vertically in a separate compartment. Up to four successive compartments are placed in series in the hutch. A high-grade concentrate is produced in the first compartment, successively lower grades being produced in the other compartments, tailings overflowing the final compartment.

Jigs are still the most widely used coal-cleaning devices, and are preferred to the more expensive heavy medium process when the coal has relatively little middlings, or “near-gravity” material, as is often the case with British coals. No feed preparation is required, as is necessary with HMS, and for coals which are easily washed, i.e. those consisting predominantly of liberated coal and denser rock particles, the lack of close density control is not a disadvantage.

Most coal jigs are air-pulsated and are called *Baum* jigs. This standard jig, with slight modifications, has been in use for over 95 years. Air under pressure is forced into a large air chamber on one side of the jig vessel causing pulsations and suction through the screen plates upon which the raw coal is fed, thus causing stratification.

The *Batac* jig is also pneumatically operated, but has no side air chamber like the Baum jig. Instead, it is designed a series multiple air chambers, usually two a cell, extending under the jig for its full width, thus giving uniform air distribution. The jig uses electronically controlled air valves which provide a sharp cut-off of the air input and exhaust. Both inlet and outlet valves are infinitely variable with regard to speed and length of stroke, allowing for the desired variation in pulsation and suction by which proper stratification of the bed may be achieved for differing raw coal characteristics. As a result, the Batac jig can wash both coarse and fine sizes well.

Reading material Installation of the Baum jig(1)

Water pump The pump should have rather more capacity than necessary. In fact, it should have enough excess capacity so that, with due allowance for wear, it will supply the requisite volume against the existing head after one year's operation. It should be driven through a V-belt having a slightly larger motor pulley than necessary. If this practice is followed, the output of the pump can be varied somewhat to compensate for any error in anticipating the water requirement by the mere expedient of changing the size of the drive pulley on the motor. The pump is associated with the re-circulating water system which includes a sump. The settling tank should be equipped with a sludge conveyor to handle the settled fines. Some of the newer installations utilize classifying cyclones rather than settling tanks. In certain systems some of the circulating water is removed from near the bottom of the settling tank. The re-circulating water is collected by skimming troughs in the settling tank and is delivered to the pump sump. A freshwater line is tapped from the main plant source and is used to add make-up water. A regular check on solids content of the circulating water should be maintained. They should not be permitted to exceed 10-15% by weight (1.04 sp gr) and should be controlled by solids removal and the addition of fresh water. Experience has proven that clarified water will result in more efficient cleaning of the finer sizes. When dealing with smaller sized coal feeds (especially with feldspar jigs), a special effort should be made to keep the level of solids in the process water low. The rate of solids removed must be balanced with their introduction. Solids loading in the water circuit can be reduced by operation of the settling tank solids conveyor during start-up of a new shift prior to the introduction of the feed. This practice allows removal of settled fines prior to their re-circulation in the jig system. Often a constant water bleed is essential to prevent the buildup of -200 mesh(-0.075 mm) particles. These particles must be collected in a thickener or large settling basin, sometimes requiring use of flocculants.

Gravity tank Uniformity of water pressure is essential to good jig operation. However, no pump gives a sufficiently steady pressure for really good jigging. A surge tank located between the pump and the jig is most desirable. The pipe leading to the head tank should be large so that the head tank will respond quickly to pick up any variation in the pressure of water from the pump. The tank should have a long overflow lip so that the pulsation from the pump will not cause any appreciable increase or decrease of head to the jig. Experience has shown that an overflow leap of approximately 12 linear ft (3.66 m) is satisfactory, though it will do no harm to have a longer one. If a little water is allowed to overflow this lip at all times, the water pressure on the system leading to the jig will be reasonably uniform. Water requirements usually come within the range of 1000-2500 gal per ton (4159-10,397 L/mt) of feed coal, although more recent designs and flowsheets may permit this volume to be reduced by half.

Water control valves In addition to the valves supplied with the jig on each of the cells, two large valves are needed on the water system. One of these should be a large throttle valve which can be opened and closed quickly when the jig is being started and stopped. In this way it will be unnecessary for the operator to touch the individual control valves on the jig itself. The other valve can be near the pump to regulate the output so that the required volume of water will overflow the head tank at all times. As an alternative to this second valve, some pump manufacturers recommend a bypass line with a valve in it, between the pump and the jig, to carry back any excess water to the sump. The argument for this arrangement is that allowing the water to flow through the pump entails less wear than throttling the output.

Compressor The air compressor should be a positive displacement type and should be driven in the manner just described for the water pump. It should be mounted on a firm support in a dust-free area but with minimum piping direct to the air receiver. In this way the pressure requirements usually do not exceed 3 psig (122 kPa). The receiver should be equipped with a hand-operated valve for exhausting or bypassing the air.

This can be opened or closed by the operator during starting and stopping by means of a series of levers or by means of a solenoid. The valve should be supplied with a stop that can be adjusted to leave a small amount of air on at all times when the jig is idling. In some systems, special control valves are used to close the air supply slowly (taking about 30-45sec) to minimize bed disturbance. This practice will keep the water from building up in the air chambers during short stops that occur from time to time in a day's operation.

Air valve control Cycle frequency control is the most important adjustment of the jig. The control is achieved by adjustment of the rate of movement of the air valve piston. Consequently, some form of variable-speed transmission which is readily adjustable to attain the desired speed is essential. The basic principles of air-valve and stroke-frequency adjustment are discussed subsequently. The detailed adjustment on different devices varies too widely to cite here. All of the valves within a given compartment usually have nearly the same settings. However, the relative timing between compartments is significant. The first compartment usually has the strongest suction. In a multiple-compartment unit, the last compartment should have a slightly longer impulse cycle to allow the air pressure to build in the first compartment where it is needed most.

Interlocked system Every plant, of course, should have an electrical control to stop all machinery in an emergency, but all too often some critical parts of the plant are hooked into an interlocked system that stops them unnecessarily and thus puts undue strain on the equipment.

One of these locations is the jig refuse elevators. These should not be hooked into the regular interlocked system that stops the remainder of the plant. They should be stopped only when the conveyors taking the refuse from the jig are stopped. If this practice is not followed, there is grave danger of shearing pins, thus causing a serious delay. Similarly, the water pump to the surge tank should not be stopped for delays of a minute or two because in many plants a loss of sludge may occur at these times through water overflowing the sludge tank. On the other hand, the raw-coal feeder to the jig should be stopped at any time when any part of the system following the jig is stopped. In order to protect the refuse ejectors and elevators, some form of protection to remove metal in the feed magnetically is also desirable.

Facilities for inspection of refuse In particular, it is important to have good lights over the refuse elevators so that the refuse may be examined regularly. Such examination may be facilitated by having a section of screen plate on which the refuse can be spread out and a freshwater hose that can be used for washing off the fine coal. This screen plate should be hinged so that it can be dropped after examination of any product.

Water piping to the jig The principal point to watch in piping to the jig is to connect the piping in such a manner that the water will not be siphoned from the jig when it stops. However, there is some argument in favor of a piping arrangement that will allow the water to drain down to the screen level on each stop, or at least will draw the water down a matter of 10-12 in. (25.4-30.5 cm). If the operator can see the top of the jig bed at every stop, this provides a check on the uniformity of the jiggling action.

In order to prevent refuse from plugging a gate provided for draining the jig, it is desirable to provide a high-pressure water line that can be opened quickly to free the drain.

Screen plates The most important point during installing screen plates is to have the correct side up. A punch makes a slightly tapered hole and it is very important to have the small end of the hole up; otherwise the plate may plug. Whenever a difficulty arises from plugging of screen plates, they should be examined to ensure the installation is correct. Although 0.25 in. (6.3 mm) openings are normally used, larger openings may be considered as an operating variable.

The larger openings presumably aid in obtaining a more open refuse bed, thus prolonging the suction stroke. Flaky particles are frequently found in Alabama coals and in some coals from Indiana and Illinois. Such

particles can be stratified by the use of an intensified suction stroke. To get the benefit of the good stratification that occurs in a jig, it is necessary to take such flaky material into the hutch compartment, thus requiring larger screen perforations.

An additional point to watch is that the holes be staggered properly. They should be staggered at right angles to the flow so that no refuse can move through the jig without being subjected to some of the upward and downward jets of water. The screen plates for average conditions should be set to provide a downward slope in the direction of refuse flow of about 1 in. per ft (8.33 cm/m). This is a detailed adjustment which is cited later.

A broad system consideration to provide adequate pond storage capacity for draining the jig during maintenance or an emergency is critical to any jig installation. The capacity of such a pond must exceed that of the jig system and should preferably exceed the total in-plant volume, including thickeners. This emergency pond should be utilized only for these purposes, be kept empty, and not be allowed to develop appreciable solids levels. The pond should have no overflow, be physically stable and impervious, provide return capability, and permit solids removal. Current environmental regulations make such facilities essential.

Adjustment of a Baum Jig(2)

Starting the jig in operation:

1) The water valves in each compartment should be opened a number of turns to give a rough initial setting. The water pump should be started and the master valve near the pump set so that a moderate volume of water is running over the overflow of the gravity tank above the jig. Finally, the control valve to the hutch compartment should be opened and the jig filled with water until it overflows the discharge end weir.

2) All motors on the jig should be started.

3) The compressor should be started and the air in the receiver brought up to pressure, usually about 2.5 psig (119 kPa) or 5 in. (127 mm) of mercury.

4) The master valve that controls the air supply to the jig should be opened very slowly so as to ease over any excess of water in the air compartments. This precaution should be followed at all times after a shutdown, so that the bed will not be disturbed.

5) The jig is now ready for coal, but before any is fed, the equipment following the jig should be checked carefully to be sure that all dewatering screens, elevators, conveyors, loading booms, and other equipment are running properly. In an initial start-up the discharge gates should be closed and then adjusted to an initial setting of about 1 in (2.54 cm).

Final adjustment In order to adjust the jig, the operator must have clearly in mind the functions of the pulsion stroke (i.e., the upstroke of the water) and of the suction stroke (the downstroke). The best design and operation is one in which the correct number of impulses can be imparted to make the separation in the least amount of space. Since most jig installations are really multistage washing units, it is important to realize that an adjusted decrease in bed distention should exist from the primary cells to the secondary compartment cells. This variation is essential to achieve maximum cleaning capability for all sizes. In each succeeding stage, with successively reduced impulses, the densimetric stratification of successively smaller particles coincides with the corresponding strata of larger sizes. Once attained in a properly adjusted jig, the stratification will be mechanically sustained throughout all succeeding stages. It is especially imperative that a uniform feed is employed during final adjustment, otherwise the operator may be unable to find the optimum control points.

Air and water The first objective in adjusting the air and water is attaining a fully mobile bed on the

pulsion stroke and a proper degree of suction on the return stroke. The operator can study conditions by sticking his hand down into the mass of coal and refuse being jigged and allowing it to be carried along in a relaxed condition with the coal. On each pulsion stroke he should feel the top 6 in. (15.2 cm) become completely mobile at the peak of the pulsion period and the mass compact, or close, at the end of the suction period. The "feel" of a mobile bed is difficult to describe, it is fluid, but is more than that; the individual particles must feel momentarily free to move with respect to each other. This condition can be detected by wiggling the fingers in the bed at the peak of the pulsion stroke. The feeling of "suction" is, perhaps, less difficult to describe. On the downward movement of the water the bed must, so to speak, grasp the operator's hand. The firmness of the grasp is an indication of the intensity of the suction. The magnitude of these movements can be quantified by inserting gages through the screen plates and measuring the hydrokinetic pressures. However, the practical application of such data has not been demonstrated.

After a few minutes of operation, a bed of refuse will begin to collect on the jib screen. The depth' can be followed if the operator reaches well down into the bed and brings particles to the surface for examination. As soon as the refuse covers the seal gate, the refuse ejector should be started and the automatic control mechanism adjusted to reduce the weight on the float until it rides on a level at which some refuse is withdrawn. When the ejector is started, some coal will appear in the elevator until the system is emptied of what unavoidably passes under the seal gate during the initial period of starting when there is no refuse bed in the jig. During routine operation the refuse elevators should be frequently checked for overload. A major change in the refuse contents of the feed may require a reduction in feed rate, even below design capacity.

Once a bed of refuse is formed on the screen plates, some refinements may be attempted in the adjustment. A wooden stick or handle for insertion in the bed can be helpful for gaging the refuse bed thickness. This can most effectively be accomplished as the suction stroke begins. The discharge gate should be adjusted to a distance of 1.5 times the size of the largest refuse particles. The final adjustment should result in strong pulsations in the primary compartment cells with diminishing impulse force from cell to cell. Since the air and the water both contribute to the pulsion stroke, a proper balance between them is essential. That balance depends upon the conditions required for efficient cleaning. Every degree of mobility is possible, from a pure pulsion stroke in which practically no water returns to the hutch compartment on the suction stroke to one in which a bare minimum of upward current water is used and a large amount returns to the hutch compartment. In the first case, little cleaning is done on the fine sizes; in the second, even very small sizes are efficiently cleaned.

In the average case, medium suction should be sought. This means that the cycle should consist of a fully mobile bed at the peak of the pulsion stroke and a return stroke in which the bed gives a moderate pull on the operator's hand. The general principles involved in obtaining the proper balance between the amount of air and the amount of water are as follows:

increasing the air pressure increases both pulsion and suction, while increasing the water increases pulsion and decreases suction. Vice versa, cutting down on the air, decreases both pulsion and suction, and decreasing the water decreases pulsion and increases suction. The water introduced into a cell during one cycle is a measure of the intensity of impulse.

These general principles are simple but the application requires care. Suppose the bed is not sufficiently mobile. The operator has the alternative of increasing the water or of increasing the air, or of doing both. Before deciding which to increase, he should note whether the top layers are becoming mobile on each stroke. If they are not breaking, he should reach deep down into the bed to determine whether this tendency is associated with lifting of the bed in a mass or with sluggishness. If, on the pulsion stroke, he finds a very fluid condition near the screen while the top layers are tight, the bed is lifting in a mass. When that happens, he

should increase only the water. This is true even though more suction may be desired, for a true suction stroke does not exist unless it is preceded by a fully mobile bed on the pulsion stroke. On the other hand, if the bed is merely sluggish, he will normally increase both air and water.

If the top layers are breaking but the operator wishes to increase the mobility and also to increase suction, he increases the air with little or no addition of water. In fact, he may even reduce the water somewhat. If he wishes to decrease suction while increasing mobility, he will normally leave the air alone and only increase the water. In this way any degree of pulsion or suction can be obtained.

In all adjustments the operator must maintain mobility of the bed at the peak of the pulsion stroke. This is necessary and fundamental, even when strong suction is required. If the operator is in doubt as to the balance between air and water, he should favor the direction of too much water. In particular, he should use more water when the top layers are not opening. There is no merit in trying to save water in jiggling. A certain amount of power is required to jig coal, and that power might as well be supplied via the water pump as via the air compressor. The object is the best adjustment of the jig to clean the coal. If the tonnage of coal to be cleaned is variable, and usually it is, this discussion of complete mobility on each stroke applies to the maximum tonnage to be treated. True, there will be some loss of efficiency when a jig adjusted for a high tonnage is operated at a low tonnage, but the loss is small. The reverse, however, is not true. If the water and the air are adjusted for low tonnage and a peak load occurs to a point where the bed ceases to be mobile, there is a really serious loss of efficiency, in which case coal is lost in the refuse and refuse is passed over into the washed coal.

Jig operators often have difficulty in getting the bed to open completely at the peak of each pulsion stroke. This is particularly likely to happen if the tonnage to be treated is very high. When the top layers fail to open, fundamentally, this means that a high air pressure is sustained for too large a proportion of the pulsion period.

To remedy this condition, use more water and less air. The air may be admitted over a longer period of time by reducing the speed of the jig or by adjusting the air valve to give a longer air-admission period. For instance, the required number of cubic feet can be admitted over a period as short as V_s sec, but in that case the acceleration of the water will be rapid and the tendency for the bed to lift in a mass will be pronounced. In contrast, the air can be admitted over a period of a second or even longer. In a slide-type air valve, the length of the rod connecting the slide valve and eccentric may be lengthened to increase the period of air admission to expand the bed, and subsequently create a stronger suction to improve cleaning the finer sizes. An excessive rod length will make the suction stroke too short.

However, if strong suction is desired, the air pressure must be applied rapidly enough to lift the bed in a mass during the initial phases of the pulsion period. An unnecessarily rapid lifting of the bed accomplishes no useful purpose, and too high an initial air pressure complicates the problem of obtaining complete mobility of the bed toward the end of the pulsion period.

Detailed control of the valve adjustment on the air expansion chamber can be achieved to allow the pressure to drop toward the end of the pulsion period. As the pressure drops, the upward acceleration of the water decreases, thus allowing the top layers of the bed to open. This method is particularly applicable where high capacity and strong suction are requisites for cleaning the smaller sizes. It is also helpful as a means of reducing water consumption on any separation.

Another approach is to combine an increase in the amount of water, which will reduce suction, with an increase in the size of screen perforations. The increased use of water will help to open the top layers while the large perforations will increase the period of mild suction. A long period of mild suction is about equivalent in effectiveness to a short period of strong suction. As to the size of perforations that can be used, there is no rule except to keep increasing the size until a tendency develops for fine coal to be lost in the hutch.

Most jigs have an adjustable weir plate associated with a refuse removal location. If inspection indicates

most of the refuse is being removed at the feed end, the weir should be lowered. However, if large refuse particles are being transported over the weir, it should be raised.

Adjustment of cycle frequency Ideally, to attain an optimum separation, a definite cycle time would be required for each particle, thus requiring an infinite number of frequencies. With this impracticality, a single frequency is selected by trial and error for a given situation employing the variable speed drive on the air valves for modulation. The additional control required is achieved by varying the proportionate part of the total cycle involved in pulsion. Despite extensive studies of wave forms, no jig has been adjusted by direct calculation.

Adjustment of float controls It must be realized that material not ejected through the gate will overflow with the light strata. Accordingly, the corresponding specific gravity point of discharge must be maintained at a sufficient level to preclude loss of float material, but not so high as to permit refuse particles to overflow with the clean coal. After the refuse mechanism is in operation, the proper specific gravity of the float can be determined roughly by watching the refuse elevators. In general, as much refuse should be drawn in a given compartment as practicable. For that reason, the float should be adjusted to pull more and more refuse as long as the quality remains satisfactory. Changes should be made slowly, however, and plenty of time should be allowed for conditions in the jig bed to come to equilibrium after each change. Although most changes will be evident in 5-10 min., it may take several hours for full equilibrium after a change in the jig stroke and 0.5 hr after a change in the float adjustment. Also, from time to time, it is important to check the mobility of the bed, especially if the depth of the refuse bed is being decreased, because the amount of air and/or water can be reduced somewhat as the refuse is lightened. Usually, the float in the first compartment is set to make a separation at a higher gravity than the following compartments. These procedures stress the complication that the specific gravity of separation in a jig is selected at a point where it can be achieved easily and with the best quality demarcation between products, rather than at some "theoretically" preselected value.

Water distribution If jigging is to be efficient, the water distribution over the screen plate must be uniform. This involves, first of all, that the feed be reasonably well distributed across the width of the jig, in regard to both quantity and size. If too much tonnage is fed on one side of the jig, the additional resistance at that point will cause water to escape on the other side. As a result, the heavily loaded part of the jig will be short of water and the bed will become immobile. This same thing will happen if all the coarse coal is fed on one side with the fine coal on the other. In that case, the upward currents of water will escape between the large particles and that part of the bed will become immobile. It follows that any such adverse conditions must be overcome before successful jigging can be obtained.

Two separate problems are involved in securing water distribution: one of distributing the water at right angles to the flow of the coal, i.e., laterally across the jig; and the other of distributing it parallel to the flow, i.e., longitudinally throughout the jig. Lateral water distribution can be controlled by baffles between the air compartment and the screen plate, or by changing the length of the partition wall between the air compartment and the hutch compartment. If the water distribution is unsatisfactory in any of the cells, baffles of some sort are necessary in the hutch compartments. Some jigs are equipped with adjustable baffle plates. Mobility within the jig bed can be studied by immersing one's hand at different points across the entire width of the jig bed and noting the mobility and the degree of suction. Longitudinal distribution of the water on all cells is regulated by having separate air valves and separate sections beneath the screen plate for each interval of 3 or 4 ft (0.9 or 1.2 m) of length. When longitudinal water distribution is correct, the amounts of water added at the feed end and at the discharge end of each compartment usually are appreciably different, decreasing in volume from feed end to discharge end. This change in longitudinal water distribution follows proper adjustment of the cycle impulses, resulting in increasingly strong suction to stratify small refuse particles.

Correction of adverse size distribution in feed The importance of a proper blend of feed sizes and of building up a media density has been noted. In practice, substantial variations in the feed occur. How much variation will the jig take without having the operation upset? The answer is that it will take a marked deviation for a few minutes with little or no apparent loss of efficiency. The jig has the capacity to absorb a fluctuation in the feed without impairment of efficiency. However, every effort should be made to minimize marked changes in the screen analysis of the feed. In particular, wherever surge bins are used there is always a tendency for the fine sizes to run out first, leaving only coarse sizes in the material remaining in the bin. Such an adverse condition, persisting at times for a period as long as 1/2 hr, is detrimental to good jig operation and must be overcome by spreading the coal entering the bin or by installation of vertical partitions in the bin to prevent occurrence of segregation.

Slope of screen plate The slope of the screen plate, should be such that the bed of refuse is uniform over the screen plate when the jig is operating under full load. This means a uniform layer of shale in the first compartment, a uniform layer of bone with a small percentage of shale in the second compartment, and a uniform layer of bone in the third compartment. Refuse tends to take a natural slope in an operating jig, and the screen plate should have this slope. This is determined by a number of factors, of which the tonnage of refuse to be rejected by the jig and the type of stroke used are the most important.

The correct slope of the screen plate can best be determined by stopping the jig under full load, i.e., stopping the jig without allowing it to idle. The water can then be drawn down and a trench made in the bed to show the stratification near the screen plate. If the refuse layer is thicker at the feed end of the compartment than at the discharge end, the screen plate should be steepened proportionately, and if it is thicker at the discharge end than at the feed end, the slope should be decreased proportionately. In an exceptional case, as in the third compartment of a jig that is badly under-loaded, it may actually be necessary to slope the screen upward toward the discharge, end to retain a bone bed.

In addition to actual examination of the bed to determine screen slope, several other indicators are of value. For instance, if pieces of coal of about 1 in. (2.5 cm) diam appear in refuse of 4x0 in. (10x0 cm), that is otherwise uniform, this is almost sure evidence that the screen plate is not sloped sufficiently. On the other hand, a tendency for hutch losses of fine coal is some evidence, though not conclusive, that the screen plate is too steep and that some coal is entering the hutch at the feed end of the compartment. This is a situation that is particularly apt to arise when the feed has been robbed of the 1/4 x 0 in. (6.3 x 0 mm) or other small sizes. To reduce the loss it may be necessary to adjust the air valve to give less suction.

Power requirements The energy requirements in jigging are primarily related to material movement. The coal is moved by water flow. Factors varying power requirements are: feed particle size, jig bed screen area and its percentage of open area, cycle frequency, length of stroke, and bed depth. Air pressure should be kept as low as practical, since the power costs are determined by volume and pressure. The jig cycle control is solely a volumetric displacement. A commonly used value for estimation purposes is 0.1 hp per sq ft (0.80 kW/mt) of screen area.

Final performance evaluation When the several procedures described indicate a jig operation has achieved satisfactory equilibrium performance, an evaluation test should be performed. A detailed sampling over at least 8 hr should provide a composite of the feed and each solid product. At minimum, a detailed particle size and washability analysis should be made on each sample, including ash and sulfur analysis on each fraction. These data permit development of the several performance criteria considered elsewhere. In addition, other localized grab samples within the jig bed (depth, width, and length) can provide base information for diagnosing subsequent operating problems. Unfortunately, these evaluations are seldom performed after initial installation acceptance tests. Any major change in the operation, as feed source or

characteristics and jig adjustment, suggests repeated evaluations.

Unit 3 Heavy medium separation

Introduction

Heavy medium separation (or dense medium separation, or the sink-and-float process) is applied to the pre-concentration of minerals, i.e. the rejection of gangue prior to grinding for final liberation. It is also used in coal preparation to produce a commercially graded end-product, clean coal being separated from the heavier shale or high-ash coal.

In principle, it is the simplest of all gravity processes and has long been a standard laboratory method for separating minerals of different specific gravity. Heavy liquids of suitable density are used, so that those minerals lighter than the liquid float, while those denser than it sink.

Since most of the liquids used in the laboratory are toxic, or give off harmful vapors, the heavy medium used in industrial separation is a thick suspension, or pulp, of some heavy mineral in water, which behaves as a heavy liquid.

The process offers some advantages over other gravity processes. It has the ability to make sharp separation at a required density, with a high degree of efficiency even in the presence of high percentages of near-density material. The density of separation can be closely controlled, within a relative density of $\pm 0.005\text{kg l}^{-1}$ and can be maintained, under normal conditions, for indefinite periods. The separating density can, however, be changed at will and fairly quickly, to meet varying requirements. The process is, however, rather expensive, mainly due to the ancillary equipment needed to clean the medium.

The efficiency of this separation process decreases with size due to the slower rate of settling of the particles. Particles should preferably be larger than about 3 mm in diameter, in which case separation can be effective on a difference in specific gravity of 0.1 or less. Separation down to 500 micron in size can be made by the use of centrifugal separators.

Heavy medium separation is applicable to any ore in which, after a suitable degree of liberation by crushing, there is enough difference in specific gravity between the particles to separate those which will repay the cost of further treatment from those which will not. The process is most widely applied when the density difference occurs at a coarse particle size, as separation efficiency decreases with size due to the slower rate of settling of the particles. Particles should preferably be larger than about 3, in diameter, in which case separation can be effective on a difference in specific gravity of 0.1 or less.

Separation down to 500 micron in size can, however, be made by the use of centrifugal separators. Providing a density difference exists, there is no upper size limit except that determined by the ability of the plant to handle the material.

The heavy medium

As mentioned above, the heavy medium could be liquids or suspensions. Heavy liquids, such as tetrabromoethane(TBE), carbon tetrachloride, have been widely used in the laboratory for the appraisal of gravity separation techniques on ores. Finely ground suspensions in water with a concentration less than 30% by volume behave essentially as simple Newtonian fluids. An example of heavy medium suspension is magnetite in water for coal preparation. The recovery of medium from the concentrate is by magnetic separation.

Separating Vessels

Several types of separating vessel are in use, and these may be classified into gravitational and

centrifugal vessels.

Gravitational units comprise some from of vessel into which the feed and medium are introduced and the floats are removed by paddles, or merely by overflow. Removal of the sinks is the most difficult part of separator design. The aim is to discharge the sink particles without removing sufficient of the medium to cause disturbing downward currents in the vessel.

An example of gravitational units is the Wemco cone separator. The feed is introduced on to the surface of the medium by free-fall, which allows it to plunge several centimeters into the medium. Gentle agitation by rakes mounted on the central shaft helps keep the medium in suspension. The float fraction simply overflows a weir, while the sinks are removed by pump or by external or internal airlift. The separator is widely used for ore treatment, since it has a relatively high sinks capacity. The cone, which has a maximum diameter of up to 6m, accommodates feed particles of up to 10 cm in diameter, with capacities of up to 500t/h.

Cyclone heavy medium separators have now become widely used in the treatment of ores and coal. They provide a high centrifugal force and a low viscosity in the medium, enabling much finer separations to be achieved than in gravitational separators.

The DSM cyclone was developed by the Dutch State Mines, and is used to treat ores and coal in the size range 40-0.5mm. The principal of operation is very similar to that of the conventional hydrocyclone. The ore is suspended in a very fine medium of ferrosilicon or magnetite, and is introduced tangentially to the cyclone under pressure, normally being gravity fed via a constant head of about 2.5-3m. Gravity feeding reduces the degradation which can occur by pumping. The sinks product leaves the cyclone in the apex, while the floats product is discharged via the central vortex finder.

Heavy Medium Separation Circuits

The feed to a heavy medium separator must be screened to remove fine ore, and slimes should be removed by washing, thus alleviating any tendency which such slime content may have for causing sharp increases in medium viscosity.

An important part of in any heavy medium circuit is the provision for reclaiming and cleaning the medium which leaves the separator with the sink and float products.

Reading Material Shaking tables

Shaking tables are concentrating devices that consist of substantially plane surfaces, inclined slightly from the horizontal, shaken with a differential movement in the direction of the long axis, and washed at right angles to the direction of motion by a stream of water.

When a flowing film of water flows over an inclined flat surface, the water closest to the surface is retarded by the friction of the water absorbed on the surface. Therefore, the velocity increases towards the water surface. If mineral particles are introduced into the film, small particles will not move as rapidly as large particles, since they will be submerged in the slower-moving portion of the film. Particles of high specific gravity will move more slowly than lighter particles, and so a lateral displacement of the material will be produced.

Tables are called sand or slime tables according to the size of material that they treat, and are classified as roughing or finishing tables according to the character of the service. Sand tables usually have relatively deep riffles, over a majority of the surface at least, and the space between cleats is usually not more than 2-3 centimeters; slime tables are not riffled so deeply, and the space between cleats is much wider than on sand

tables in order to form pools of relatively quiescent pulp to induce settlement of solids. Slime tables always have a portion of the deck unriffled. Roughing tables are usually riffled full length and riffles are comparatively deep. These tables are thus enabled to treat large tonnages and yet save fine mineral with the coarse in the form of a low-grade concentrate, at the same time rejecting an impoverished tailing. Finishing tables, with a few exceptions, have an unriffled portion for cleaning out fine gangue from the concentrate streak; the riffles are shallower than on roughing tables, and, in general, less resistance is offered to cross travel of solids than on a rougher.

The table decks are usually constructed of wood, lined with materials with a high coefficient of friction, such as linoleum, rubber, and plastics. Decks made from fiberglass are also used which, although more expensive, are extremely hard wearing. The riffles on such decks are incorporated as part of the mould.

Particle size plays a very important role in table separations; as the range of sizes in a table feed increases, the efficiency of separation decreases. If a table feed is made up of a wide range of particle sizes, some of these sizes will be cleaned inefficiently. Beside, the shape of particles is another important factor affecting the separation efficiency.

Shaking tables and other gravity processes require large amounts of water. In certain circumstances, especially where water is at a premium, dry separation techniques may have to be used. Pneumatic tables are sometimes used in conjunction with magnetic and high-tension separation techniques in the treatment of beach sand deposits. In such treatments, not only is water conservation necessary, but the cost of dewatering the gravity concentrate may be prohibitive. Pneumatic tabling is used on some coal plants and has an important use in the upgrading of asbestos.

asbestos n.[矿]石棉

Part 4 FROTH FLOTATION

Unit 1 Principles of Flotation

Originally patented in 1906, flotation has permitted the mining of low-grade and complex ore bodies which would have otherwise been regarded as uneconomic. In earlier practice the tailings of many gravity plants were of a higher grade than the ore treated in many modern flotation plants.

Flotation is a selective process and can be used to achieve specific separations from complex ores such as lead-zinc, copper-zinc, etc. Initially developed to treat the sulphides of copper, lead, and zinc, the field of flotation has now expanded to include the oxidised minerals and non-metallics, including fine coal.

Flotation is undoubtedly the most important and versatile mineral-processing technique, and both use and application are being expanded to treat greater tonnages and to cover new areas.

Froth flotation utilises the differences in physico-chemical surface properties of particles of various minerals. After treatment with reagents, such differences in surface properties between the minerals within the flotation pulp become apparent and, for flotation to take place, an air-bubble must be able to attach itself to a particle, and lift it to the water surface. The process can only be applied to relatively fine particles, as if they are too large the adhesion between the particle and the bubble will be less than the particle weight and the bubble will therefore drop its load.

In flotation concentration, the mineral is usually transferred to the froth, or float fraction, leaving the gangue in the pulp or tailing. This is direct flotation as opposed to reverse flotation, in which the gangue is separated into the float fraction.

The air-bubbles can only stick to the mineral particles if they can displace water from the mineral surface, which can only happen if the mineral is to some extent water repellent or hydrophobic. Having reached the surface, the air-bubbles can only continue to support the mineral particles if they can form a stable froth, otherwise they will burst and drop the mineral particles. To achieve these conditions it is necessary to use the numerous chemical reagents known as flotation reagents.

The activity of a mineral surface in relation to flotation reagents in water depends on the forces which operate on that surface. The forces tending to separate a particle and a bubble are shown in following Figure.

$$\gamma_{S/A} = \gamma_{S/W} + \gamma_{W/A} \cos \theta$$

The tensile forces lead to the development of an angle between the mineral surface and the bubble surface. At equilibrium,

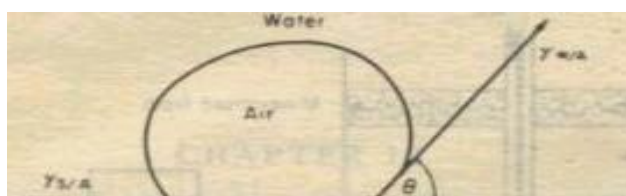


FIG.xx Contact angle between bubble and particle in an aqueous medium-

where $\gamma_{s/a}$, $\gamma_{s/w}$, and $\gamma_{w/a}$ are the surface energies between solid-air, solid-water and water-air, respectively, and θ is the contact angle between the mineral surface and the bubble.

The force required to break the particle-bubble interface is called the work of adhesion. $W_{s/a}$ and is equal to the work required to separate the solid-air interface and produce separate air-water and solid-water interfaces, i.e.

$$W_{s/a} = \gamma_{w/a} + \gamma_{s/w} - \gamma_{s/a} \quad (2)$$

Combining with equation (1) gives

$$W_{s/a} = \gamma_{w/a} (1 - \cos\theta). \quad (3)$$

It can be seen that the greater the contact angle the greater is the work of adhesion between particle and bubble and the more resilient the system is to disruptive forces. The floatability of a mineral therefore increases with the contact angle; minerals with a high contact angle are said to be aerophilic, i.e. they have a higher affinity for air than for water. Most minerals are not water repellent in their natural state and flotation reagents must be added to the pulp. The most important reagents are the collectors, which adsorb on mineral surfaces, rendering them hydrophobic (or aerophilic) and facilitating bubble attachment. The frothers help maintain a reasonably stable froth. Regulators are used to control the flotation process; these either activate or depress mineral attachment to air-bubbles and are also used to control the pH of the system.

TABLE 4.1. CLASSIFICATION OF POLAR MINERALS

Group 1	Group 2	Group 3	Group 4	Group 5
Galena	Barite	CerrusHe	Hematite	Zircon
Covdite	Anhydrite	Malachite	Azurile	Magnetite Goethite
Bonite	Gypsum Anglesite	Wulfenile	Chromile	Hemimorphite Beryl
Chalcopyrite			Ilmenite	Feldspar
Stibnite		Group 3(b)	Corundum	Sillimanite
Argentite		Fluorile	Pyrolusite	Garnet
Bismuthinite		Calcite	Limonite	Quartz
Millerite		Witherite	Borax	
Cobaltite		Magnesite	Wolframite	
Arsenopyrite		Dolomite	Columbite	
Pyrite -r \		Apatite	Tantalite	
Sphalerite '		Scheelite	Rutile	
Orpiment		Smithsonite	Cassiterite	
Psilondite		Rhodochrosite		
Realgar		Siderite		
Native Au, Pt, Ag,		Monaate		
Cu				

Collector molecules may be ionizing compounds, which dissociate into ions in water, or non-ionizing compounds, which are practically insoluble, and render the mineral water-repellent by covering its surface with a thin film.

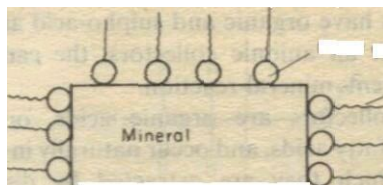
Ionizing collectors have found very wide application in flotation. They have complex molecules which are asymmetric in structure and are heteropolar, i.e. the molecule contains a non-polar hydrocarbon group and a polar group which may be one of a number of types. The non-polar hydrocarbon radical has pronounced

water-repellent properties, whereas the polar group reacts with water.

Ionizing collectors are classed in accordance with the type of ion, anion or cation that produces the water-repellent effect in water.

The structure of sodium oleate, an anionic collector in which the hydrocarbon radical, which does not react with water, constitutes the non-polar part of the molecule, is shown in Fig. 12.5.

Because of chemical, electrical, or physical attraction between the polar portions and surface sites, the collectors adsorb on the particles with their non-polar ends orientated towards the bulk solution, thereby



imparting hydrophobicity to the particles .

Generally speaking, collectors are used in small amounts, substantially those necessary to form a monomolecular layer on particle surfaces (starvation level), as increased concentration, apart from the cost, tends to float other minerals, reducing selectivity. It is always harder to eliminate a collector already adsorbed than to prevent its adsorption. An excessive concentration of a collector can also have an adverse effect on the recovery of the valuable minerals, possibly due to the development of collector multi-layers on the particles, reducing the proportion of hydrocarbon radicals oriented into the bulk solution. The hydrophobicity of the particles is thus reduced, and hence their floatability. The flotation limit may be extended without loss of selectivity by using a collector with a longer hydrocarbon chain, thus producing greater water-repulsion, than by increasing the concentration of a shorter chain collector. However, chain length is usually limited to two to five carbon atoms, since the solubility of the collector in water rapidly diminishes with increasing chain length and, although there is a corresponding decrease in solubility of the collector products, which therefore adsorb very readily on the mineral surfaces, it is, of course, necessary for the collector to ionize in water for chemisorption to take place on the mineral surfaces. Not only the chain length, but also the chain structure, affects solubility; branched chains have higher solubility than straight chains.

The collector added to the head of the flotation system is normally relatively strong and non-selective to promote maximum recovery, and a weaker, more selective collector is often added to the primary concentrate to upgrade the final concentrate produced.

Tensile adj. 可拉长的, 可伸长的, [物]张力的, 拉力的

Hydrophobicity 疏水(性)

Oleate n. [化]油酸盐

Reading Material Laboratory Flotation Testing

In order to develop a flotation circuit for a specific ore, preliminary laboratory testwork must be undertaken in order to determine the choice of reagents and the size of plant for a given throughput as well as the flowsheet and peripheral data. Flotation testing is also carried out on ores in existing plants to improve procedures and for development of new reagents.

It is essential that testwork is carried out on ore which is representative of that treated in the commercial plant. Samples for testwork must be representative, not only in chemical composition, but also relative to mineralogical composition and degree of dissemination. A mineralogical examination of drill cores or other individual samples should therefore be made before a representative sample is selected. Composite drill core samples are ideal for testing if drilling in the deposit has been extensive; the cores generally contain ore from points widely distributed over the area and in depth. It must be realised that ore-bodies are variable and that a representative sample will not apply equally well to all parts of the ore-body; it is used therefore for development of the general flotation procedure. Additional tests must be made on samples from various areas and depths to establish optimum conditions in each case and to give design data over the whole range of ore variation.

Having selected representative samples of the ore, it is necessary to prepare them for flotation testing, which involves comminution of the ore to its optimum particle size. Crushing must be carried out with care in order to avoid accidental contamination of the sample by grease or oil, or with other materials which have been previously crushed. Even in a commercial plant, a small amount of grease or oil can temporarily upset the flotation circuit. Samples are usually crushed with small jaw crushers or cone crushers to about 0.5 cm and then to about 1 mm with crushing rolls in closed circuit with a screen.

Storage of the crushed samples is important, since oxidation of the surfaces is to be avoided, especially with sulphide ores. Sulphides should be tested as soon as possible after obtaining the sample and sulphide ore samples must be shipped in sealed drums in as coarse a state as possible. Samples should be crushed as needed during the test work, although a better solution is to crush all the samples and to store them in an inert atmosphere.

Wet grinding of the samples should always be undertaken immediately prior to flotation testing to avoid oxidation of the liberated mineral surfaces. Batch laboratory grinding, using ball mills, produces a flotation feed with a wider size distribution than that obtained in continuous closed-circuit grinding; to minimise this, batch rod mills are used which give products having a size distribution which approximates closely to that obtained in closed-circuit ball mills. True simulation is never really achieved, however, as overgrinding of high specific gravity minerals, which is a feature of closed-circuit grinding, is avoided in a batch rod mill.

A soft, dense mineral such as galena will be ground finer in closed circuit than predicted by the batch tests, and its losses due to production of ultra-fine particles may be substantial. Some sulphide minerals, such as sphalerite and pyrite, can be depressed more easily at the coarser sizes produced in batch grinding, but may be more difficult to depress at the finer sizes resulting from closed-circuit grinding. Predictions from laboratory tests can be improved if the mineral recovery from the batch tests is expressed as a function of mineral size rather than overall product size. The optimum mineral size can then be determined and the

overall size estimated to give the optimum mesh of grind. This method assumes that the same fineness of the valuable mineral will give the same flotation results both from closed-circuit and batch grinding, irrespective of the differences in size distributions of the other minerals.

It must be appreciated that the optimum grinding size of the particles depends not only on their grain size, but also on their floatability. Initial examination of the ore should be made to determine the degree of liberation in terms of particle size in order to estimate the required fineness of grind. Testwork should then be carried out over a range of grinding sizes in conjunction with flotation tests in order to determine the optimum mesh of grind. In certain cases, it may be necessary to overgrind the ore in order that the particles are small enough to be lifted by the air-bubbles. If the mineral is readily floatable a coarse grind may be utilised, the subsequent concentrate requiring regrinding to further free the mineral from the gangue, before further flotation is performed to produce a high-grade concentrate.

Initial floatability tests are often made on the liberated mineral particles, as a means of assessing a range of suitable collectors. A useful laboratory method is that of contact angle measurement, where a clean smooth surface of mineral is placed in distilled water, and a bubble of air from the end of a capillary tube is pressed down upon it. If after a short time no adhesion is visible on withdrawal of the bubble, the mineral surface is assumed to be clean, and the collector is then added. If the mineral surface now becomes hydrophobic, adherence of the introduced bubble to the surface results. The contact angle produced across the water phase is a measure of the floatability of the mineral. The method suffers from many disadvantages; it is extremely difficult to obtain a truly representative surface of the mineral of the required size (at least 0.5 cm²). The mineral may not be representative of the naturally liberated surface after the intense polishing required to produce a completely clean, flat surface. The method is static, whereas true flotation is dynamic, particles adhering after impact with bubbles rising in the pulp.

In the bubble pick-up method, small quantities of the mineral are crushed and sized, placed into a beaker of distilled water containing the test reagents, and pressed on by an air-bubble held in the concave end of a glass rod. The loaded bubble is then removed from the solution and the attached grains counted under a microscope. A less tedious method has been developed by Lee, who noted that the mineral load invariably assumed a regular shape comprising a single layer of closely packed particles at the base of the bubble. By placing a graticule behind one eye-piece of a binocular microscope and observing a silhouette of the bubble and its load against the graticule, the number of squares covered by the load could be estimated and hence the cross-sectional area of the visible part of the pickup surface could be derived. This value corresponded closely to the number of grains on the bubble as determined by counting and the use of closely sized samples permitted the construction of calibration lines. The pick-up technique is more sensitive to reagent concentration and pulp pH than is the contact angle method, and is far less tedious, although it still suffers from the drawback of being an essentially static test.

In the Hallimond tube technique (Fig. 12.11), dynamic conditions prevail, similar to those in a commercial flotation cell. The mineral particles are held on a support of sintered glass inside the tube containing the distilled water and the collector under test. Air-bubbles are introduced through the sinter and any hydrophobic mineral particles are lifted by the bubbles, which burst at the water surface, allowing the particles to fall into the collecting tube. By treating a small weighed sample of mineral, the weight collected in the tube can be

related to the floatability.

The bulk of laboratory testwork is carried out in batch flotation cells, usually with 500-g, 1-kg, or 2-kg samples of ore. The cells are mechanically agitated, the speed of rotation of the impellers being variable, and simulate the large-scale models commercially available. Introduction of air to the cell is normally via a hollow standpipe surrounding the impeller shaft. The action of the impeller draws air down the standpipe, the volume being controlled by a valve and by the speed of the impeller. The air stream is sheared into fine bubbles by the impeller, these bubbles then rising through the pulp to the surface, where any particles picked up are removed as a mineralized froth. By collecting the froth at set time intervals it is possible to relate the recovery of metal to the concentrate against the time and the resultant plot can be used to determine the size of plant required.

Unit 2 Flotation Reagents

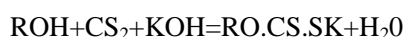
Anionic Collectors

These are the most widely used collectors in mineral flotation and may be classified into two types according to the structure of the polar group. Oxyhydryl collectors have organic and sulpho-acid anions as their polar groups and, as with all anionic collectors, the cation takes no significant part in the reagent-mineral reaction.

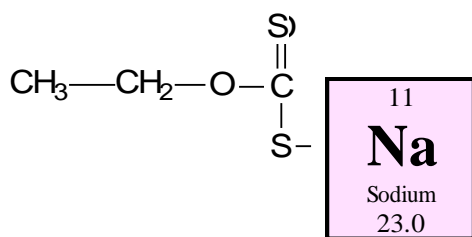
Typically, oxyhydryl collectors are organic acids, or soaps. The carboxylates are known as fatty acids, and occur naturally in vegetable oils and animal fats from which they are extracted by distillation and crystallization. The salts of oleic acid, such as sodium oleate and linoleic acid, are commonly used. As with all ionic collectors, the longer the hydrocarbon chain length, the more powerful is the water-repulsion produced, but solubility decreases. Soaps (the salts of fatty acids), however, are soluble even if the chain length is long. The carboxylates are strong collectors, but have relatively low selectivity. They are used for the flotation of minerals of calcium, barium, strontium, and magnesium, the carbonates of non-ferrous metals, and the soluble salts of alkali metals and alkaline earth metals.

The sulphates and sulphonates are used more rarely. They possess similar properties to fatty acids, but have lower collecting power. However, they have greater selectivity and are used for floating cassiterite, barite, kyanite, and scheelite.

The most widely used collectors are of the sulphydryl type, where the polar group contains bivalent sulphur. They are very powerful and selective in the flotation of sulphide minerals. The most widely used of these anionic collectors are the xanthogenates (technically known as the xanthates) and the dithiophosphates (Aerofloat collectors). The xanthates are the most important for sulphide mineral flotation. They are prepared by reacting an alkali hydroxide, an alcohol and carbon disulphide:



where R is the hydrocarbon group and contains normally one to six carbon atoms, the most widely used xanthates being ethyl, isopropyl, isobutyl, amyl, and hexyl. Sodium ethyl xanthate is typical and has the structure shown in Fig. 12.7. The anion consists of a hydrocarbon non-polar radical and a connected polar group.



The dithiophosphates have pentavalent phosphorus in the polar group, rather than tetravalent carbon (Fig.

12.8).

The reaction between sulphide minerals and sulphhydryl collectors is complex. Xanthates are assumed to adsorb on sulphide mineral surfaces due to chemical forces between the polar group and the surface, resulting in insoluble metal xanthates, which are strongly hydrophobic. Mechanisms involving the formation and adsorption of dixanthogen, xanthic acid, etc., have also been proposed. It has been shown that the sulphide is not joined to the collector anions without the previous action of oxygen. The reaction of xanthate with oxidation products of the sulphide surface through an ion-exchange process is considered to be the major adsorption mechanism for the flotation of sulphides. However, a high degree of surface oxidation, accompanied by the formation of sulphates, which readily react with xanthates, prevents or impedes flotation, since the metal xanthates so formed scale off the mineral.

The solubilities of the hydrophobic xanthates of copper, lead, silver, and mercury are very low, but the xanthates of zinc and iron are much more soluble. Typically, ethyl xanthates are only weak collectors of pure sphalerite, but replacement of the crystal lattice zinc atoms by copper improves the flotation properties of the mineral. The alkali earth metal xanthates (calcium, barium, magnesium) are very soluble and xanthates have no collector action on the minerals of such metals, nor on oxides, silicates, or aluminosilicates, which permits extremely selective flotation of sulphides from gangue minerals.

Xanthates are normally used in weakly alkaline pulps, since they decompose in acid media and, at high values of pH, hydroxyl ions can displace xanthate ions from the mineral surface.

Dithiophosphates are not as widely used as the xanthates, but are still important reagents in practice. They are comparatively weak collectors, but give good results in combination with xanthates. It appears that the water repulsion imparted to the mineral surface is due to the formation of an oxidation product of the dithiophosphate collector which adsorbs on to the mineral surface. Thus, as with xanthates, the presence of oxygen, or another oxidising agent, is essential for flotation. Strong oxidising conditions destroy the hydrophobic substances and are thus undesirable, while oxidation of the mineral surface itself may impede collector adsorption. It is considered that knowledge of the oxidation characteristics of mineral and collector and their electrical properties is vital for the further advance of flotation theory.

Cationic Collectors

The characteristic property of this group of collectors is that the water repulsion is produced by the cation where the polar group is based on pentavalent nitrogen, the amines being the most common. The anions of such collectors are usually halides, or more rarely hydroxides, which take no active part in the reaction with minerals.

Unlike the xanthates, the amines are considered to adsorb on mineral surfaces primarily due to electrostatic attraction between the polar head of the collector and the charged electrical double layer on the mineral surface. Such forces are not as strong or irreversible as the chemical forces characteristic of anionic collectors, so these collectors tend to be relatively weak in collecting power.

Cationic collectors are very sensitive to the pH of the medium, being most active in slightly acid solutions and inactive in strongly alkaline and acid media. They are used for floating oxides, carbonates, and alkali

earth metals such as barite, carnallite, and sylvite.

Frothers

When mineral surfaces have been rendered hydrophobic by the use of a collector, stability of bubble attachment, especially at the pulp surface, depends to a considerable extent on the efficiency of the frother.

Ideally the frother acts entirely in the liquid phase and does not influence the state of the mineral surface. In practice, however, interaction does occur between the frother, mineral, and other reagents, and the selection of a suitable frother for a given ore can only be made after extensive test work.

In sulphide mineral flotation it is common practice to employ at least two frothers and more than one collector. Specific frothers are chosen to provide adequate physical properties to the froth, while the second frother interacts with the collectors to control the dynamics of the flotation process.

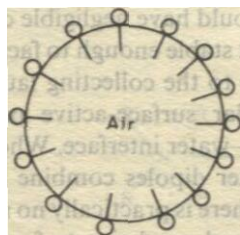
Frothers are in many respects chemically similar to ionic collectors, and, indeed, many of the collectors, such as the oleates, are powerful frothers, being, in fact, too powerful to be used as efficient frothers, since the froths which they produce can be too stable to allow efficient transport to further processing. Froth build-up on the surfaces of thickeners, and excessive frothing of flotation cells, are problems occurring in many mineral processing plants. A good frother should have negligible collecting power, and also produce a froth which is just stable enough to facilitate transfer of floated mineral from the cell surface to the collecting launder.

Frothers are generally heteropolar surface-active organic reagents, capable of being adsorbed on the air-water interface. When surface-active molecules react with water, the water dipoles combine readily with the polar groups and hydrate them, but there is practically no reaction with the non-polar hydrocarbon group, the tendency being to force the latter into the air phase. Thus the heteropolar structure of the frother molecule leads to its adsorption, i.e. the molecules concentrate in the surface layer with the non-polar groups orientated towards the air and the polar groups towards the water (Fig. 12.10).

Frothing action is thus due to the ability of the frother to adsorb on the air-water interface because of its surface activity and to reduce the surface tension, thus stabilising the air-bubble.

Frothers must be to some extent soluble in water, otherwise they would be distributed very unevenly in an aqueous solution and their surface active properties would not be fully effective. The most effective frothers include in their composition one of the following groups:

The acids, amines, and alcohols are the most soluble of the frothers. The alcohols (-OH) are the most widely used, since they have practically no collector properties, and in this respect are preferable to other frothers, such as the carboxyls, which are also powerful collectors; the presence of collecting and frothing properties in the same reagent may make selective flotation difficult. Frothers with an amino group, and certain sulpho group frothers, also have weak collector properties.



Pine oil, which contains aromatic alcohols, the most active frothing component being terpineols, $C_{10}H_{17}OH$, has been widely used as a frother. Cresol (cresylic acid), $CH_3C_6H_4OH$, has also had wide use.

A wide range of synthetic frothers, based mainly on high molecular-weight alcohols, are now in use in many plants. They have the important advantage over industrial products such as pine oil and cresol in that their compositions are much more stable, which makes it easier to control the flotation process and improves performance. A widely used synthetic Alcohol frother is methyl isobutyl carbinol (MIBC). Another range of synthetic frothers are based on polyglycol ethers, and have been found to be very effective. They are marketed under various names, such as Dowfroth 250, Cyanamid R 65 and Union Carbide PG 400.

Although frothers are generally surface-active reagents, it has been shown that surface-inactive reagents, such as diacetone alcohol and ethyl acetal, behave as frothers in solid-liquid-air systems, although not in two-phase liquid-air systems. Molecules of these reagents have two polar groups and are readily soluble in water. They adsorb on solid surfaces but do not appreciably change their hydrophobicity. When the mineral surface, on which the surface inactive frother is adsorbed, is approached by an air-bubble, the molecules reorientate and produce a sufficiently stable three-phase froth. Being surface inactive, these reagents do not reduce surface tension, and apart from the slight reduction due to collectors, the forces available for flotation are maintained at their maximum.

Oxyhydryl

Carboxylate n. 羧酸盐, 羧酸酯

linoleic acid 亚油酸, 罂酸

xanthate n. 黄原酸盐 (黄药)

dithiophosphate 二硫代磷酸盐 (黑药)

carbinol n. [化] 甲醇, 原醇

carbonate n. [化] 碳酸盐; vt. 使变成碳酸盐, 使充满二氧化碳

sylvite n. [矿] 钾盐

carnallite n. 光卤石 (钾的原料), 砂金卤石

terpineol n. [化] 萜品醇, 松油醇

polyglycol 聚乙二醇或有关的醚-二醇类化合物

cresylic adj. [化] 甲酚的, 杂酚油的

Regulators

Regulators, or modifiers, are used extensively in flotation to modify the action of the collector, either by

intensifying or reducing its water-repellent effect on the mineral surface. They thus make collector action more selective towards certain minerals. Regulators can be classed as activators, depressants, or pH modifiers.

Activators. These reagents alter the chemical nature of mineral surfaces so that they become hydrophobic due to the action of the collector.

Activators are generally soluble salts which ionize in solution, the ions then reacting with the mineral surface. A classical example is the activation of sphalerite by copper in solution. Sphalerite is not floated satisfactorily by a xanthate collector, since the collector products formed, such as zinc xanthate, are relatively soluble in water, and so do not provide a hydrophobic film around the mineral. Floatability can be improved by the use of large quantities of long-chain xanthates, but a more satisfactory method is to use copper sulphate as an activator, which is readily soluble and dissociates into copper ions in solution. Activation is due to the formation of molecules of copper sulphide at the mineral surface, due to the fact that copper is more electro-negative than zinc and therefore ionizes less readily:



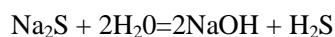
The copper sulphide deposited on the sphalerite surface reacts readily with xanthate to form insoluble copper xanthate, which renders the sphalerite surface hydrophobic.

Very careful control of reagent feeding must be observed when copper sulphate is used as an activator in conjunction with a xanthate collector. It is essential that the minerals are conditioned with the activator separately, before the collector is added, and that when the conditioned pulp enters the collector conditioner, there is little residual copper sulphate in solution. Xanthates react readily with copper ions, needlessly consuming both reagents.

The main use of copper sulphate as an activator is in the differential flotation of lead-zinc ores, where after lead flotation the sphalerite is activated and floated. To some extent, copper ions can also activate galena, calcite, and pyrite.

Oxidised minerals of lead, zinc, and copper, such as cerrusite, smithsonite, azurite, and malachite, float very inefficiently with sulphhydryl collectors and require an extremely large amount. Such minerals are activated by the use of sodium sulphide or sodium hydrosulphide.

In solution, sodium sulphide hydrolyses and then dissociates:



Depressants. Depression is used to increase the selectivity of flotation by rendering certain minerals hydrophilic (water avid), thus preventing their flotation.

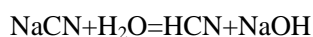
There are many types of depressant and their actions are complex and varied, and in most cases little understood, making depression more difficult to control than the application of other types of reagent.

Slime coating is an example of a naturally occurring form of depression. Slimes in a crushed and ground ore make flotation difficult, as they coat the mineral particles, retarding collector adsorption. They can sometimes be removed from the surfaces by vigorous agitation, but a more reliable method is to use a slime dispersant. Sodium silicate in solution increases the double-layer charge on particles, so that the slime layers which have

formed readily disperse. The clean mineral surface can then interact with the collector. In this respect, therefore, sodium silicate is used as an activator, preventing depression by the slimes. Sodium silicate is also used as a depressant in some systems, depressing silicates, and calcite in fluorite-calcite, and scheelite-calcite separations.

Cyanides are widely used in the selective flotation of lead-copper-zinc and copper-zinc ores, as depressants for sphalerite, pyrite and certain copper sulphides.

It is fairly well established that pure clean sphalerite does not adsorb short-chain xanthates until its surface is activated by copper ions. However, copper ions resulting from very slight dissolution of copper minerals present in the ore may cause unintentional activation and prevent selective separation. Cyanide is added to the pulp to desorb the surface copper and to react with copper in solution forming soluble cyanide complexes. Sodium cyanide is most commonly used, which hydrolyses in aqueous solution to form free alkali, and relatively insoluble hydrogen cyanide:



Since the production of even small traces of hydrogen cyanide is extremely dangerous, cyanides must always be used in an alkaline medium, since free alkali in solution will force the hydrolysis reaction to the left.

Cyanides are, of course, extremely toxic and must be handled with great care. They also have the disadvantage of being expensive and they depress and dissolve gold and silver, reducing the extraction of these metals into the froth products.

Flotation is usually carried out in an alkaline medium, as most collectors are stable under these conditions and corrosion of cells, pipework, etc., is minimized. The control of pH is extremely important, especially in selective flotation, and alkalinity is controlled by the addition of lime, sodium hydroxide, sodium carbonate, and sulphuric acid.

Lime, being cheap, is very widely used to regulate pulp alkalinity and is used in the form of milk of lime, a suspension of calcium hydroxide particles in a saturated aqueous solution. Lime prevents the adverse effects of soluble salts on flotation, by precipitating them as metal hydroxides. It is also used extensively as a depressant for pyrite and arsenopyrite. Both the hydroxyl ion and the calcium ion participate in the depressive effect of lime on pyrite by the formation of mixed films of $\text{Fe}(\text{OH})$, $\text{FeO}(\text{OH})$, CaSO_4 , and CaCO_3 , on the surface, reducing the adsorption of xanthate. Lime has no depressant effect with copper minerals, but does depress galena to some extent. In the flotation of galena, therefore, pH control is often effected by the use of soda ash (Na_2CO_3), pyrite and sphalerite being depressed by cyanide.

Potassium dichromate ($\text{K}_2\text{Cr}_2\text{O}_7$) is used to depress galena in copper-lead separations. The depressive action is due to the chemical reaction between the galena surface and CrO_4^{2-} anions, which produces insoluble lead dichromate which increases wettability and prevents flotation.

Organic reagents such as starch, tannin, quebracho, and dextrin do not ionize in solution, but form colloidal particles in the pulp which can be deposited on the mineral surfaces, preventing flotation in a similar manner to a slime coating. Large quantities of these reagents will depress all minerals and they are not as selective as the electrolytic depressants. They are used in small amounts to depress talc, graphite, and calcite. Starch is also used to depress molybdenite in copper-molybdenum separations.

Arsenopyrite n. 毒砂, 含砷黄铁矿

Dichromate n.重铬酸盐

Tannin n.[化]丹宁酸

Quebracho n.白坚木, 南美产漆树科乔木

Dextrin n.[化]糊精

Molybdenite n.[矿]辉钼矿

Molybdenum n.[化]钼

Reading Material Factors affecting coal flotation

In coal flotation, the flotation rate and cell productivity in terms of tons per kilowatt-hour and tons per cubic foot of cell volume are of greater importance than in most other minerals due to the low unit value of coal. The requirements for successful flotation are sufficient bubble encounter followed by sustained adhesion of the coal particles to the bubble until they reach the upper froth layer and can be removed by the machine.

The major factors affecting coal flotation can be listed as follows:

1. Particle size
2. Oxidation and rank of coal
3. Pulp density
4. pH and other characteristics
5. Flotation reagents
6. Flotation equipment

Typically, feed particle size in coal flotation is -28mesh although some newer coal preparation plants classify or screen at 65 and even 100 mesh prior to flotation. In any case, compared to other methods of coal cleaning, the particle size range in the feed to flotation is very broad, ranging from 0.5mm to submicrometer material. It is not unusual for a flotation feed to contain over 25% -325 mesh material.

The pulp density is the solid concentration to coal flotation cells. It might range from two or three percent to 10% or more, depending on the quantity of water used in the screening or classification step and the rate of coal feed to the plant. In current US practice, the solids concentration is quite variable but probably averages about 7%. The coarser the coal particles, the higher the pulp density, and the finer the particles, the lower the pulp density. Generally no attempt is made to monitor or control the feed solids content. The solids concentration for optimum coal flotation is much lower than that used for metallic ore flotation. This can be readily appreciated when one considers that the specific gravity of ores may over 3.0 while that of coal is about 1.5. Another factor is that in coal flotation, the bulk of the material is normally floated while in ore flotation usually only a small fraction is floated.

The rank and oxidation of coals affect their floatability. Low volatile coals are easier to float than most high volatile coals. Of the petrographic constituents, vitrain is more readily floatable than durain. Anthracite is more difficult to float than high volatile bituminous coal. Lignite is the least floatable form of coal. Further,

even a highly floatable coal will become difficult to float if highly oxidized.

Nevertheless, with the proper use of flotation reagents these various degrees of floatability may be greatly adjusted. Sometimes, changing from MIBC or an alcohol frother to a kerosene-pine oil combination can provide effective result. The use of amyl or butyl xanthated as a collector or promoter has been largely overlooked as a possible reagent additive to coal floatation.

The use of cationic amine collectors should be investigated as a means of floating oxidized coals. Whereas most highly floatable coals require only one or two minutes, the lower rank and oxidized coals require three to four minutes or more. The practice of using a conditioning tank ahead of the cells is recommended. Also, in this connection a high-speed scrubber may be of help in cleaning the coal surfaces if the coal is superficially weathered and/or covered with clay slimes.

The quality of water has to be considered in efficient flotation. It is known that flotation can be difficult in certain ranges of pH. Both recovery and quality of products can be affected. Tests conducted demonstrate that recovery is highest when the pH is between 6-7.5. Low pH value can, of course, be controlled by additions of lime. High pH helps to inhibit the floating of pyrite. The amount of soluble salts in the water also affects flotation results, but little is known of their effect on coal flotation. Colloidal clays or slimes in the water inhibit froth flotation. These may be controlled by the proper use of chemical agents to flocculate them out in the tailings thickener or by the use of fresh water in the flotation circuit, and in severe conditions it may be necessary to hydro-classify slimes out ahead of the flotation cells.

Since flotation reagents are the most expensive item in the process, their optimum use should be carefully ascertained. A readily floatable coal, such as coking coal, usually requires only a small amount of frother. In treating other coals, frothers, collector, depressors, and other agents may be required.

Generally, for most coals, it is desirable to use both a frother and collector together, such as an alcohol and kerosene or fuel oil. Some coal tar oils are advantageous and cresylic acid is an exceptionally good frother and collector for coal. Emulsified pine oil and kerosene are often used effectively.

The quantity of reagents used will vary widely with conditions. Frothers usually are used at rates of 0.1-0.5 lb per ton (50-250 g/mt) of feed. Collectors, when used, may vary from 0.5-2.1b per ton (250-1000 g/mt) of feed.

Unit 3 Flotation Circuits and Equipments

Flotation circuit

Most froth flotation circuits in American coal preparation plants are relatively simple. They produce a primary or rough concentrate and primary tailings with no recleaning of either product.

The slurry generally comes to the froth cells direct from dewatering screen or sieve bend underflows or may be from classifier tank overflows. Hydrocyclone overflows or underflows also may be feed. There is usually no serious attempt to control pulp density and it usually varies from 4-12% solids.

The size consist is widely variable but generally is the product through 0.5mm wedge wire screens or their equivalent. Reagents are usually added to the feed of the first cell. The number of cells varies from four to eight in a single battery with the number of batteries depending upon the total tonnage to handle.

Froth concentrates generally pass by gravity direct to a vacuum filter with the filtrate returned to the cells. Tailings may discharge directly to a waste pond or a settling thickener with the clarified water reused in the plant. The thickener underflow goes to a waste pond or a tailings vacuum filter. This is the typical American straight line froth circuit .

Some circuits , of course, are more complicated, and may utilize special circuits for slimes and coarse sizes; others may employ density control equipment and may reclean froth concentrates. Although it certainly may not be necessary in all cases, there is no doubt that grater attention should be given to the kind of flotation circuit used and that the operators could improve both quality and recovery by customizing their circuit to their particular needs.

With the importance of sulfur in the form of pyrite removal becoming more critical for many coals, special emphasis should be given to a circuit designed for this problem. Such a circuit generally will require closely controlled pulp densities, proper choice of reagents, conditioning of reagents, especially depressors, and at least one recleaning of the froth. In connection with pyrite removal some success has been reported by placing hydrocyclones and sieve vends in the circuit.

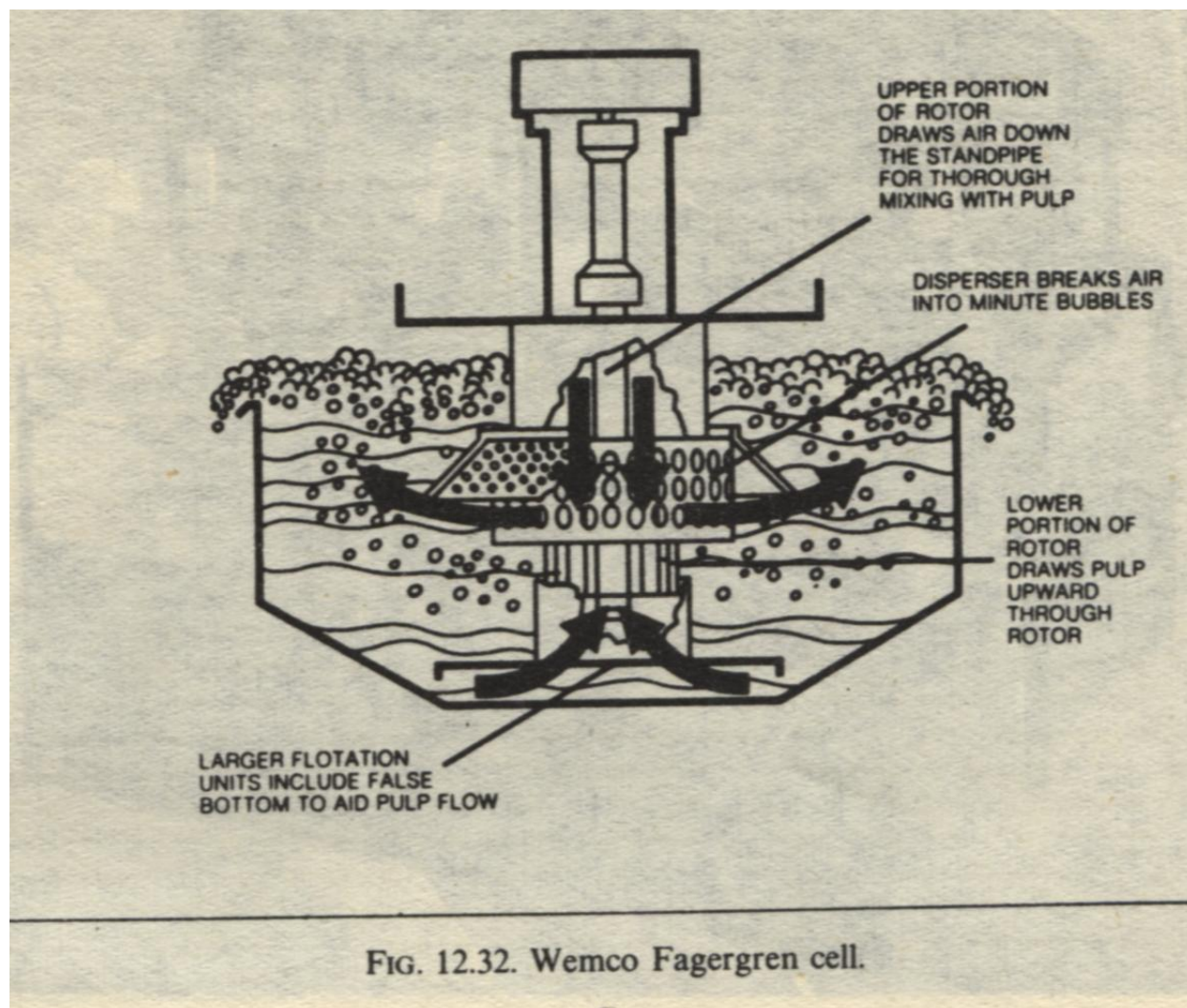
Flotation equipment

Wemco Cells

Wemco flotation cells are based upon the Fragergren Rotor-Stator principle. This principle has a great facility for producing minute air bubbles and thoroughly mixing the pulp. The rotor-rator is the key part of this machine. The rotor consists of rubber or neoprene-covered top and bottom impeller rings separated by rubber or neoprene-covered rotor tubes. The stator is a stationary close-fitting cage which fits over and around the rotor. Slurry is drawn upward into the rotor by the lower impeller while air is pulled down the standpipe by the upper impeller. The mixture of air and pulp is expelled by centrifugal force through the stator tubes.

The high shearing action facilitates maximum formation of small bubbles which quickly rise to the

surface carrying floatable coal. Average retention time in this type cells is between one and two minutes.



Heyl and Patterson Cyclo-cells

The cell are developed by Heyl and Patterson. There are no moving or mechanical parts. Agitation is achieved by submerged vortex chambers which discharge a high velocity jet of pulp and air in the form of a hollow cone.

The principle of operation came from the hydrocyclone. The cyclone, because of its centrifugal action, was experimented with as a means of shearing air, with water under pressure admitted to the feed orifice of the cyclone and air supplied through the overflow orifice. When the cyclone underflow spray is immersed in water the only exit of air is by way of the vortex within the cyclone into the high-velocity spray, which shears the air causing dissemination into minute bubbles.

From studying this action a vortex chamber was especially designed for maximum shearing of air into air bubbles. The vortex chamber has the capacity for shearing more air than it will aspirate. In practice air at about 2psi(115kPa) is supplied through a tube passing down through the vertical axis of the vortex chamber. The cell is very simple. The vortexes are easily installed in almost any number. The standard cell consists of a tank or trough with two or more vortex chambers in series.

The power required is that needed to pump the pulp through the vortex and to supply low pressure air. It will generally range from 0.6-1.2kW per ton.

Denver cells

Denver Equipment Co. was one of the earliest pioneers in establishing their machines in coal flotation. Their standard cell used for many years has been the “sub-a”, No.30 cell. It is used in many countries. This cell is of the mechanical agitation type and may or may not use additional low-pressure air. Pulp comes into the impeller by means of a pipe connected to the side opening from a weir box into which the feed pulp has been inserted. A diffusion plate directly above the impeller causes aeration of the pulp.

Flotation columns

Different kinds of flotation columns have been developed in recent years. They have been widely used in ore and coal processing. Column flotation achieves superior cleaning of fine coal as compared to conventional flotation due to its ability to support a large froth depth and use of wash water in the froth zone which allows removal of entrained impurities from the froth. In addition, the external bubble generators used in column flotation produce smaller bubbles than conventional flotation which results in an increase in the rate of coal recovery from the pulp. Utilizing these basic principles, several column technologies have been developed which produce a cleaner coal product with greater recovery values than those achieved by conventional flotation cells. Flotaire column, Jameson cell, Microdel column, Canadian column, packed-column, and Turboair column are representative types. A significant difference between them is the mechanism by which the microbubbles are produced.

Reading Material Pollution Aspects of Fine Coal Cleaning and “Black Water” Disposal

The effluents from coal washeries and waters draining from plant-site surfaces inevitably contain fine coal and coal refuse materials in suspension. For over 60 years, the coal industry has witnessed a gradual decline in the use of dry methods of coal preparation in favor of wet techniques. Dedusting is becoming increasingly difficult, as more coal is being produced by mechanical means which results in an increase of fines in run-of-mine coal. In many locations the need for water economy has been created because it is difficult to obtain suitable feed water for preparation plants. In other areas increasingly stringent regulation of effluent discharge has necessitated recirculation. The coal preparation plant operator has found it necessary to economize on water. The size of coal going into preparation plants has been getting finer; consequently, the problems facing preparation plant personnel are growing proportionately. With new plants, clarification facilities are generally becoming more complex. At older plants the disposal of effluent continues to present a serious problem. It is becoming increasingly difficult to comply with the standards required by many water authorities and pollution control agencies. Today practically all new washeries are designed to operate on a closed circuit to satisfy pollution requirements.

Characteristics of water for coal preparation The literature indicates that there are few rules for makeup water in coal preparation processes. It is generally agreed, however, that solid matter in the wash water should

be less than 5 %. In coal preparation plants, the solids concentration in plant water varies between 0.25-0.9 lb per gal (30-110 g/L) and may reach concentrations of 1.7 lb (200 g).

Chemical characteristics of water such as neutral pH, low conductivity, and low bicarbonate content are most desirable for plant usage. Clean streams void of contamination from sewage, organic matter, or acid drainage usually are acceptable as sources of water. In most cases, however, the water is obtained at the lowest cost including any necessary treatment. For example, one operator using acid drainage for preparation use is neutralizing with lime because of corrosion problems.

The physical characteristics of water used in coal preparation plants vary considerably from country to country. In Russian, for example, solid matter must not exceed 0.01 lb per gal (1.5 g/L). A common European consideration is the size of the particles suspended in the makeup water. It is agreed that particles from 0.1-74 micron determine the properties of water. These particles are absorbed by the coal, which may increase the ash content. Raybould of the British National Coal Board stated that the maximum concentration of clay should not exceed 1.83 lb per gal (220 g/L) for effective washing at a density of less than 1.5. The disadvantages of using waters charged with solids are: (1) they cause wear, chiefly of pumps and cyclones by erosion; (2) they alter the density of the bath and increase viscosity of heavy media used for separation; and (3) they do not rinse washed products properly. It is generally concluded that the concentration of solid matter in wash water should be less than 5% or between 0.25-0.9 lb per gal (30-110 g/L). Brunard claimed that overloading with the 1.5 lb per gal (180 g/L) of solids or working with too little solids both have their inconveniences. Cyclones, with a high proportion of slurries, work more efficiently than jigs. Recycling of water charged with very high concentrations of finely dispersed clay is to be avoided in heavy medium washing.

Change in water quality during coal preparation Fine coal and mineral particles, such as clays, remain suspended in most plant waters. These particles vary in size from 30 mesh to colloidal dimensions, and most of them can be eliminated by thickeners, cyclones, and filters. Nearly all new washeries are designed to operate on a closed circuit.

Various methods of water clarification applied to slurry and froth flotation tailings are flocculation, thickeners, settling ponds, vacuum filtration, and pressure filtration, among others. The action of dissolved minerals or salts and various promoting agents that are added to enhance flotation, flocculation, and filtration significantly affect the properties of water. Run-of-mine coal contains varying amounts of minerals and soluble salts. Nelson identified the minerals commonly found in coal and, for convenience, Spackman and Moses classified them

into various groups. Sprunk and Donnell stated that the most common minerals in coal were pyrite, calcite, quartz, and siderite, supplemented by clays consisting mostly of hydrosilicates of aluminum.

Some minerals and salts, such as chlorides and sulfates of the alkalis and the alkaline earth metals, dissolve easily in water. In certain circumstances the salts will significantly change the pH of the circulating water. For example, calcite, aragonite (霏石, 文石) and dolomite are slightly soluble to the extent of 14 ppm in pure water at 77°F (25°C). The influence of additional salts present in solution increases the solubility of the carbonates. Thus, sodium chloride, in concentrations up to 7% by weight, can increase the solubility of calcite by 0.03 lb per gal (3.8 g/L). However, if the water contains carbon dioxide or if any additional acid is present, the carbonate will neutralize the acid to a value proportional to its concentration. Soluble clays may also exhibit basic properties. It is conceivable for pyrite, marcasite, and other sulfides that are normally insoluble in water to oxidize and form ferrous sulfate and sulfuric acid. The oxidation of iron sulfide has serious effects on the pH, normally lowering it to between 2.8 and 5. Iron sulfate is sometimes used as an agent to promote the action of flocculent electrolytes. The addition of salts, through either artificial or natural means, will increase the conductivity of the solution. Zelders studied corrosion of coal preparation plants and related that corrosion was proportional to the electrical conductivity of the water as well as the oxygen content, even at constant pH. Merrman found that the conductivity increases 250 micromhos per cc during coal preparation.

Brunard and others found that the hardness of the wash water decreases in some European preparation plants while the chlorides, sulfates, and conductivity increase.

The rapid increase in froth flotation has introduced a new aspect to water treatment requirements. Methods presently in use for preparing fine coal do an acceptable job down to 50 or 100 mesh (0.3 or 0.15 mm). A large percentage of the -50 or -100 mesh (-0.3 or -0.15 mm) fraction of the coal now produced is being sent to settling ponds or sold at low cost due to low quality. Consequently, more operators have resorted to froth flotation.

Flotation separates fine coal from refuse and fine clay. However, an excess of flocculating agents, which may remain in the return water from the water clarification circuit, is detrimental to efficient froth flotation. Calcium hydroxide, often present in recycled filtrates, seriously impedes flotation because it increases the consumption of reagents and often causes a high pH. For reagents, several combinations are used. Some plants use as much as 1.5-3% light oil while others use a mixture of light oil and a frother. Still other plants use only a frother and get excellent results. The oils used are generally light diesel oil or kerosene, and the frothing agents are pine oil or alcohol. Most newer plants use the alcohol alone. The quantities needed vary but amount to about 0.05-0.30 lb per st (0.025-0.15 kg/mt) of solids treated. Where substantial quantities of light oil are used, it is claimed that dewatering of the coal is simplified.

The introduction into the water of organic reagents, such as cresylic acid, pine oils, primary and secondary aliphatic alcohols, methyl isobutylcarbinol, kerosene, light oil, fuel oil, gas oil, and creosote, has of some concern. This is minimized in part by water recirculation. Several operators, both in Europe and the United States, are utilizing water from coking plants, which exhibits frothing and collecting properties. The phenols and ammonia which appear in these liquids are often absorbed by the coals, thereby removing phenols and reducing pollution. For example, Hall showed that ammoniacal waste liquor, containing approximately 1000 ppm of phenols with a biological oxygen demand (BOD) of 3500 ppm, can be utilized in a cell-type flotation plant to replace normal flotation oil without affecting operating results. In this case, phenols were reduced to between 50-100 ppm and the BOD value to less than 100 ppm. In general, reagents that remain in the water after clarification of the froth flotation tailings and the coal product may contain from 0-30 ppm of pollutants.

The average consumption of flocculating reagent used in treating slurry is 3.25 lb per st (1.7 kg/mt) dry weight of solids treated, together with 2.25 lb (1.01 kg) of lime. The average consumption of flocculating reagent used in treating froth flotation tailings is 4.39 lb per st (2.0 kg/mt) together with 2.61 lb (1.18 kg) of lime. Flocculation by electrolytes involves the use of reagents which adhere to the particle but partly remain in the water, thus contaminating the recirculating water. The conventional electrolytes are alum, lime, calcium and sodium chloride, ferric and ferrous sulfate, and hydrophilic materials such as starch or other soluble starch derivatives. Lime, iron sulfate, and starch are probably the most common flocculents, although aluminum sulfate and ferric chloride are sometimes used. Much attention has been devoted recently to high molecular-weight, water-soluble, synthetic polymers which also contribute to the variation in water quality. In general, flocculating agents such as electrolytes, coagulants, and synthetic polymers introduce active ionic species in the solution which can influence the pH and critical micelle formation.

Effects of water quality on unit operations in coal preparation Water quality directly affects many of the separation processes in coal preparation. The variation in feed and the content of solid matter can change significantly as the coal flows through the plant. The water quality has some effect on all of the operations in the preparation plant.

The unit operations of flotation, flocculation, and filtration may be affected by water quality. Coal flotation is a method to clean "black water," and can be considered for use when the cost of cleaning the water circuit and the credits for the process indicate that it has substantial advantages for "black water" clarification. The following paragraphs discuss the theory and practice of "black water" clarification by froth flotation.

Coals, especially those of lower rank, have considerable capacity for the absorption of electrolytes by

direct or exchange absorption, the ion exchange and electrostatic charge on the coal surface, and the altering of the hydrophobicity. Whether the electrolytes have an effect on flotation depends upon the nature of the electrolyte, their concentration, and the original condition of the coal.

Klassen found that flotation pulp which contained 0.02 lb per gal (2.02 g/L) of anhydrous gypsum at 64~F (18~C) increased the yield by 6.5%, while a saturated solution of the same salt increased the yield by 10.7%. During this test sulfonated kerosene was used as the collector. Dilute solutions such as sodium chloride, aluminum chloride, and aluminum sulfate have been found to promote flotation while concentrated solutions depressed flotation activity. El'Yashevich claimed that the presence of dissolved salts depresses the flotation of fusain. Coal flotation was performed industrially using sodium chloride and waste from soda manufacture. Studies indicated that the rate of the process increased considerably; consequently the recovery time increased 50% when sodium chloride was used and 70% when condenser liquid (a mixture of sodium chloride, calcium chloride, and sodium sulfate) was used in concentrations ranging from 0.8-1%. This process produced a concentrated vitrinite froth with little or no fusain.

Klassen, Nevskaya, and Vlasova, with the aid of radioisotopes, noted that the salts were more strongly absorbed by coal than refuse. However, because the absorption was basically physical, the salt was easily desorbed on washing. Adherence of salt to refuse was more stable and gave rise to chemical interaction characteristics. Opsipow stated that surfactants are capable of being absorbed as oriented films, and the air-water interface and the solid surface can function as a frother or collector. With micelle formation, the absorbed layer becomes hydrophilic. The effect is the non-adherence of an air bubble to a solid surface resulting in no contact angle, and flotation is depressed. High salt concentrations decrease the critical micelle concentration of the solution. The concentrations of ions of the same charge produce no effect, but ions of opposite charge are responsible for the decrease.

In conclusion, it can be said that the presence of salts may either increase or decrease the yields of a given flotation circuit depending on the nature of the dissolved salts, the rank of the coal, throughput temperature, and other synergistic properties.

It is generally accepted that optimum coal flotation occurs near the neutral point of pH 7.0. However, there are many instances where coal has been floated over a wider range of pH. This was necessary to decrease the floating of pyrites by depressing them in the pH range of 8.0-8.5 and to hinder the passage of clean minerals into the froth utilizing both an acid reference and alkaline circuit. Pennsylvania anthracite silts were found to yield maximum recovery in the range 6.5-10.07. Consequently, the best recovery of gangue minerals should occur in an acid solution. Brown related that near pH 7 a coal particle possesses a small negative charge. If the coal slurry pulp is made more acid, hydrogen is absorbed, and the charge approaches zero and the hydrophobicity is at a maximum. In a more acid or more basic pulp, the particle will acquire by absorption a negative charge to a hydroxyl ion or positive charge due to a hydrogen ion, resulting in a reduction in hydrophobicity. The frothability of pine oil and its constituents are only slightly affected by a pH change from 3.4-8.0. With an increase in pH from 8.5-11, the frothability of the reagents is increased according to Sun. The theoretical explanation offered is that at a high pH value, the solubilities of reagents increase, as well as their frothing power in water.

Temperature is also a variable which may occur in feed water of flotation circuits. The variation of temperature may be caused by the use of the plant circuit water, feed water coming from cooling water or from other unit operations, or cool water from underground sources. It has been observed that slight increases in recovery can be obtained as the temperature is increased with the optimum temperature ranging between 59-104~F (15-40~C). Above 104~F (40~C) and below 59~F (15~C) the recovery gradually decreases. Sun has indicated that the volumes of froth produced by pine oil increase with the temperature of the liquid, but the time of aeration decreases with temperature. Low temperature results in high viscosity and high surface tension, and therefore, low solubility of pine oil. Oayle and Eddy reported that temperature variations seriously

affect flotation recovery when simple aliphatic alcohols are used. With other reagents such as kerosene, the recoveries appeared independent of temperature. In general, the temperature recovery relationship is influenced mostly by the particular reagent used but is substantially independent of the particular coal sample.

There is a definite interaction of reagents used in other unit operations with those used in the froth flotation circuit. Starch and lime are considered to depress the flotation of coal. Merrman stated that lime in the concentration of one gram per liter seriously hindered coal flotation. However, Hook and Myers revealed that starch flocculants in quantities above 1.5 lb per st (0.75 kg/mt) of solids completely depress flotation. Thus, while it is possible for starch flocculants to affect froth flotation of coal adversely, Hook and Myers believe that by maintaining good plant practice the effects were not significant. Kelsey remarked that polyelectrolytes are totally absorbed on the surface of the particles, are not reflected back in the solution, and can be effective over a wide pH range. Therefore, the possibility of water contamination is not significant. However, if excess reagent is used, long chain organic compounds produce a decrease in the concentration required for critical micelle formation. This is similar to the effect of dissolved salts on flotation.

Part 5 DEWATERING

Unit 1 Introduction

Most mineral separation processes involve the use of substantial quantities of water and the final concentrate has to be separated from a pulp in which the water/solids ratio may be high.

Dewatering, or solid-liquid separation, produces a relatively dry concentrate for shipment. Partial dewatering is also performed at various stages in the treatment, so as to prepare the feed for subsequent process.

Dewatering methods can be broadly classified into three groups—Sedimentation, filtration and thermal drying

Sedimentation is most efficient when there is a large density difference between liquid and solid. This is always the case in mineral processing where the carrier liquid is water. Sedimentation cannot always be applied in hydrometallurgical processes, however, because in some cases the carrier liquid may be a high-grade leach liquor having a density approaching that of the solids. In such cases, filtration may be necessary.

Dewatering in mineral processing is normally a combination of the above methods. The bulk of the water is first removed by sedimentation, or thickening, which produces a thickened pulp of perhaps 55-65% solids by weight. Up to 80 % of the water can be separated at this stage. Filtration of the thick pulp then produces a moist filter cake of between 80 and 90% solids, which may require thermal drying to produce a final product of about 95% solids by weight.

Terminology

Coal slurry A coal slurry can be defined as a watery suspension carrying particles of fine coal and other assorted material ranging upward in size from the finest clay particles. Some particles may be of colloidal size and exhibit characteristics of suspended colloids. The dilution of the system generally is such that it is in a condition that cannot be economically dewatered by filtration.

Coal slime A coal slime may be regarded as a slurry containing particles of such size that 50 or more (by weight) will pass a 200 mesh (0.075mm) sieve. Slime suspensions generally are difficult to handle without special treatment in conventional sedimentation equipment.

Coal sludge is defined as a slurry that has been partly dewatered by sedimentation, usually to a thick or viscous consistency where further dewatering can be readily facilitated by filtration.

Culm and **silt** are characteristic coal preparation terms that should be defined. In anthracite terminology, culm is the generally coarse residue or waste accumulation of coal, bone, and rock from preparation plants. Culm has a wide range of sizes and, except to differentiate it from silt, the term is of limited interest in this discussion. In anthracite terminology, silt is the accumulation of waste fine coal, bone, and slate settled out from preparation plant water in excavated basins. Silt is generally made up of particles ranging in size from those passing a 2.38mm round opening screen to the very finest sizes. Silt is also referred to at times as sludge, fines, slush, or mud and is the partly dewatered derivative of what has been previously defined as slurry. In the terminology of the bituminous coal producing areas, the word culm commonly corresponds to slurry or slime, depending upon the size distribution of the suspended solids.

This Terminology is only meaningful in this paper to facilitate communication.

so as to adv.使得, 以致, 如此...以致于

sedimentation n.沉淀, 沉降

filtration n.过滤, 筛选

drying v. 烘干

Reading material Surface Moisture of Fine Coal Particles

Dewatering of fine coal by continuous filtration involves the filtercake formation and removal of surface moisture by drawing air through the capillaries of the cake. This continuous dewatering operation is actually a cyclic process in that a batch process with continuous cake discharge is repeated every revolution or cycle of the filter operation. In fine coal filtration, the cycle will generally range from 1.5 to 5 min/rev. Thus, both cake formation and cake dewatering occur in relatively short periods of time.

To fully understand how surface moisture can be removed from fine coal, it is necessary to identify how water exists in filter flitercake. Normally, the forms of water are:

1. Interior adsorption water. Contained in micropores and microcapillaries within each coal particle, deposited during coal formation
2. Surface adsorption water. Forms a layer of water molecules adjacent to coal molecules, but on particle surface only
3. capillary water. Contained in capillaries and small crevices found in the surface of the particles
4. interparticle water. Contained in capillaries and small crevices found between two or more particles
5. Adhesion water. Forms a layer or film around the surface of individual or agglomerated particles

The water which can be readily removed using mechanical fine coal dewatering devices, such as vacuum filters, is categorized by types4 and 5; and is generally termed surface moisture. Water type 3 can be removed partially, depending upon the size of the openings in the coal surface and the drying time available in the filter cycle. Inherent moisture is the general term used in a typical proximate analysis of coal to describe the water as defined by types 1 and 2.

In coal processing, the surface moisture is eliminated by first removing the interparticle water formed in a filter cake; this is done by applying a pressure differential across the cake so the water flows from the compacted particles. Next, air is used to remove the adhesion water. In this step, momentum is imparted to the water as the air passes over the water surface. Limited evaporation also occurs; the amount of water removed is dependent on the available energy or temperature of the coal and on the relative humidity of the air passing through the cake.

The amount of the adhesion and interparticle water found in a filter cake depends on the size and number of particles. Particle shape and size distribution of particles determine the void space between particles which contains water. This can be better understood by looking at the voidage expression for a bed of granular solid particles of mixed sizes given below:

$$\varepsilon = 1 - \frac{S\phi_s D_p}{6}$$

$$\frac{1}{D_p} = \sum \frac{X}{D_{pX}}$$

where:

Dp – average particle diametere, defined as diameter of sphere of same volume as particle, ft

E—voidage (fractional free volume), dimensionless

S—specific surface, or area of particle surface per unit volume of bed, ft^2/ft^3

Φ_s —shape factor of solid defined as quotient of area of sphere equivalent to volume of particle divided by actual surface of particle, dimensionless (for natural coal powder ≈ 0.65 ; for pulverized coal particle ≈ 0.73)

Since the voidage is directly proportional to the average particle diameter, a greater amount of relatively large particles will create more interparticle water when the particles are formed as in a filter cake. The quantity of the adhesion water found in a filter cake or packed bed of solids is directly proportional to the total surface area of the solids. Equation shows that surface area is inversely proportional to particle diameter. Removing the adhesion water from the surface of the coal is thought to be affected predominantly by the mass flow of air sweeping across the surface of the particles. The effect of a smaller average particle diameter increases the total surface area of the solids per unit volume, because the voidage is decreased. This increases adhesion water and at the same time limits the mass flow rate of both water and air through the filter cake. The net effect of particle size distribution on coal surface moisture is that the average particle size for a given volume of solids will control the rate of water removal and govern the amount and type of water present.

Ash content found in the clean fine coal dewatered by a vacuum disk filter is an indicator of the amount of micrometer-sized particles found in the sample. Generally, the ash is the remainder of clay, shale, or rock dust that water not removed in the cleaning process. Fine refuse found along with the fine coal usually is smaller in size than coal dust. Some clays can have particle diameters between essentially zero to 10 micron. If the cleaning operation of fine coal is not designed properly to remove the ultrafine refuse material, the moisture content in the filter cake can increase due to the decrease of the average particle size.

Coal has certain physical properties which are used for classification according to a system that ranks the coal from a high to low rank, i.e., anthracite to bituminous to lignite. Basically, this system describes the rank using measurements such as volatile matter, hardness, moisture, carbon content, and heating value of the coal. The number of capillaries within and crevices on the surface of a coal particle are a function of the coal age, density, and hardness. Older or higher-rank coals are harder and denser than those in lower ranks. Because of this relationship, the amount of the inherent moisture increases with decreasing rank since more internal capillaries exist.

The use of surfactants has been studied on several occasions in laboratory-scale as well as pilot-plant-scale experiments designed to demonstrate the effectiveness of different types of surfactants in reducing the surface moisture of fine coal. There are two general types of surfactants that improve the removal of water from coal:

- (1) additives that change the surface tension of water and (2) hydrophobic or oily substance that are surface active agents.

Surfactants that change the surface tension of water are further divided into three groups:

- I. anionic, for example, sulfonated aliphatic esters, aliphatic sulfates, aliphatic sulfonates, alkylaryl sulfonates, esters, and ethers
- II. cationic, for example, dodecylammonium bromide or octadecylammonium chloride
- III. nonionic, for example, the product of condensation of alcohols or phenols with a polymer from ethylene oxide

Baker (name) used an anionic surfactant at dosages up to 1.2 lb/ton (600 g/ton) and demonstrated a reduction in surface moisture from 20 wt% without surfactant to 10 wt% with surfactant. Work done by Nicol showed that the residual moisture of an Australian coal could be reduced seven percentage points by the addition of about 75 ppm of an anionic surfactant, where the addition of 500 ppm of a cationic surfactant only reduced the residual moisture less than five percentage points. Similar results have been demonstrated by Silverblatt and Dahlstrom, as well as Miekecki and Kurzeja.

The effectiveness of any surfactant to remove more surface water from coal depends on the chemistry of the water as well as the type of water present on and in the coal, which is related to the type of coal. Adhesion water and interparticle water, as discussed earlier, are the predominant forms of water affected by surfactants. The amount of water that surfactants are capable of removing changes from coal to coal and must be evaluated for each possible application.

Surfactants are effective in reducing surface moisture in fine coal dewatering. However, the full scale use of surfactants has not been popular in coal preparation plants because of cost and the secondary effects on other unit operations in the plant such as flotation and refuse thickening and water clarification. More developmental work needs to be done to demonstrate the benefits of surfactants on a full scale so that more accurate economic analyses can be completed, including secondary effects of surfactants on plant operation.

Unit 2 Sedimentation

Rapid settling of solid particles in a liquid produces a clarified liquid which can be decanted, leaving a thickened slurry, which may require further dewatering by filtration. The settling rates of particles in a fluid are governed by Stokes' or Newton's laws, depending on the particle size. Very fine particles, of only a few microns diameter, settle extremely slowly by gravity alone, and centrifugal sedimentation may have to be performed, alternatively, the particles may be agglomerated, or flocculated, into relatively large lumps, called flocs, that settle out more rapidly.

Gravity sedimentation

Gravity sedimentation, or thickening, is the most widely applied dewatering technique in mineral processing, as it is a relatively cheap, high-capacity process, which involves very low shear forces, thus providing good conditions for flocculation of fine particles.

The thickener is used to increase the concentration of the suspension by sedimentation, accompanied by the formation of a clear liquid. In most cases the concentration of the suspension is high and hindered settling takes place. Thickeners may be batch or continuous units, and consist of relatively shallow tanks from which the clear liquid is taken off at the top, and the thickened suspension at the bottom. The clarifier is similar in design, but is less robust, handling suspensions of much lower solid content than the thickener.

The continuous thickener consists of a cylindrical tank, the diameter ranging from about 2 m to 200 m in diameter, and of depth 1-7 m. Pulp is fed into the center via a feed-well placed up to 1m below the surface, in order to cause as little disturbance as possible. The clarified liquid overflows a peripheral launder, while the solids which settle over the entire bottom of the tank are withdrawn as a thickened pulp from an outlet at the center. Within the tank are one or more rotating radial arms, from each of which are suspended a series of blades shaped so as to take the settled solids towards the central outlet. On most modern thickeners these arms rise automatically if the torque exceeds a certain value, thus preventing damage due to overloading. The blades also assist the compaction of the settled particles and produce a thicker underflow than can be achieved by simple settling, the solids in the thickener move continuously downwards, and then are raked towards the thickened underflow outlet, while the liquid moves upwards and radially outwards. In general, there is no region of constant composition in the thickener.

Thickener tanks are constructed of steel, concrete, or a combination of both, steel being most economical in sizes of less than 25m in diameter. The tank bottom is often flat, while the mechanism arms are sloped towards the central discharge. With this design, settled solids must bed-in to form a false sloping floor. Steel floors are rarely sloped to conform with the arms because of expense. Concrete bases and sides become more common in the larger-sized tanks. In many cases the settled solids, because of particle size, tend to slump and will not form a false bottom. In these cases the floor should be concrete and poured to match the slope of the arms. Recently earth bottom thickeners have entered service, and are generally considered to be the lowest cost solution for thickener bottom construction.

The method of supporting the mechanism depends primarily on the tank diameter. In relatively small thickeners, of diameter less than about 45m the drive head is usually supported on a superstructure spanning the tank, with arms being attached to the drive shaft. Such machines are referred to as bridge or beam thickeners. The underflow is usually drawn from the apex of a cone located at the center of the sloping bottom.

High-capacity thickeners

Conventional thickeners suffer from the disadvantage that large floor areas are required, since the throughput depends above all on the area, while depth is of minor importance. In recent years, machines known as “high-capacity” thickeners have been introduced by various manufacturers. Many varieties exist, and the machines are typified by a reduction in unit area requirement from conventional installations.

The “hi-capacity” thickener developed by Envirotech Corporation is typical, called tray thickener. The feed enters via a hollow drive shaft where flocculant is added and is rapidly dispersed by staged mechanical mixing. This staged mixing action is said to improve thickening since it makes most effective use of the flocculant. The flocculated feed leaves the mixing chambers and is injected into a blanket of slurry where the feed solids are further flocculated by contacting previously flocculated material. Direct contact between rising fluid and settling solids, which is common to most thickeners, is averted with slurry blanket injection. Radially mounted inclined plates are partially submerged in the slurry blanket; the settling solids in the slurry blanket slide downwards along the inclined plates, producing faster and more effective thickening than vertical descent. The height of the slurry blanket is automated through the use of a level sensor.

An investigation by Sala International in co-operation with the Axel Johnson Institute for Industrial Research into the possibility of better utilization of the settling area of the thickener, resulted in the construction of the lamella thickener. It utilizes a nest of inclined parallel plates which reduce settling distance and at the same time increase effective area. The floor-space requirement of the lamella thickener is only about 20% of that of the conventional thickener.

The inclined parallel trays allow the settled solids to slide by gravity into a hopper. The effective settling area is therefore the horizontal projection of these trays.

The whole lamella pack can be vibrated intermittently or continuously when treating sticky sludges. The feed enters the thickener through a bottomless feed-box at a point which determines the relationship between clarification and thickening areas. The area below the feed-box outlet is for thickening while the area above is for clarification. The discharge liquid flows upward and is removed through specially designed boxes, thus providing a uniform flow distribution between the lamella plates.

Compression in a conventional thickener is provided by surface load, depth of solids, and retention time. As the retention time in a lamella thickener is short, low-amplitude vibrations are applied to the hopper, which play an important part in providing solids compression and flow.

Centrifugal sedimentation

Centrifugal separation can be regarded as an extension of gravity separation, as the settling rates of particles are increased under the influence of centrifugal force. It can, however, be used to separate emulsions which are normally stable in a gravity field.

Centrifugal separation can be performed either by hydrocyclones or centrifuges.

The simplicity and cheapness of the hydrocyclone (Chapter 9) make it very attractive, although it suffers from restrictions with respect to the solids concentration which can be achieved and the relative proportions of overflow and underflow into which the feed may be split. Generally the efficiency of even a small-diameter cyclone falls off rapidly at very fine particle sizes and particles smaller than about 10 micron diameter will invariably appear in the overflow, unless they are very heavy. Flocculation of such particles is not possible, since the high shear forces within a cyclone rapidly break up any agglomerates. The cyclone is therefore inherently better suited to classification rather than thickening.

By comparison, centrifuges are much more costly and complex, but have a much greater clarifying power and are generally more flexible. Much greater solids concentrations can be obtained than with the cyclone.

Various types of centrifuge are used industrially, the solid bowl scroll centrifuge having widest use in the minerals industry due to its ability to discharge the solids continuously.

The basic principles of a typical machine are shown in Fig.15.16. It consists essentially of a horizontal revolving shell or bowl, cylindroconical in form, inside which a screw conveyor of similar section rotates in the same direction at a slightly higher or lower speed. The feed pulp is admitted to the bowl through the center tube of the revolving-screw conveyor. On leaving the feed pipe the slurry is immediately subjected to a high centrifugal force causing the solid to settle on the inner surface of the bowl at a rate which depends on the rotational speed employed, this normally being between 1600 and 8500 rev min⁻¹. The separated solids are conveyed by the scroll out of the liquid and discharged through outlets at the smaller end of bowl. The solids are continuously dewatered by centrifugal force as they proceed from the liquid zone to the discharge. Excess entrained liquor drains away to the pond circumferentially through the particle bed.

Reading material Coagulation and flocculation

Little more than 40 years ago the word flocculant meant very little to the mineral processor. It was more likely to be used with reference to either an inorganic salt, in which case it was really a coagulant, or alternatively a naturally occurring polymeric substance such as starch or guar gum. In both cases the substance was applied as a settling aid. Nowadays the term flocculant is more likely to be used in describing high molecular weight, water-soluble polymers of the type generically known as polyacrylamides.

Coagulation causes extremely fine colloidal particles to adhere directly to each other, all particles exert mutual attraction forces, known as London-Van der Waal's forces, which are effective only at very close range. Normally, the adhesion due to these forces is prevented by the presence around each particle of an electrically charged atmosphere, which generates repulsion forces between particles approaching each other. There is, therefore, in any given system, a balance between the attractive forces and the electrical repulsion forces present at the solid-liquid interface.

In any given system the electrical charges on the particle surfaces will be of the same sign, aqueous suspensions of pH 4 and above generally being negative. Positively charged surfaces occur mainly in strong acid solutions.

The repulsion forces not only prevent coagulation of the particles, but also retard their settlement by keeping them in constant motion, this effect being more pronounced the smaller the particle. Coagulants are electrolytes having an opposite charge to the particles, thus causing charge neutralization when dispersed in the system, allowing the particles to come into contact and adhere as a result of molecular forces. Inorganic salts have long been used for this purpose, and as counter-ions in aqueous systems are most frequently positively charged, salts containing highly charged cations, such as Al³⁺, Fe³⁺, and Ca²⁺ are mainly used. Lime, or sulphuric acid, depending on the surface charge of the particles, can also be used to cause coagulation. Most pronounced coagulation occurs when the particles have zero charge in relation to the suspending medium, this occurring when the zeta potential is zero. The nature of the zeta potential can be seen from model of the electrical double layer at the surface of a particle. The magnitude of the zeta potential depends on the surface potential, and the concentration and charge of the counter-ions. In general, the greater the counter-ion charge and counter-ion concentration, the lower is the zeta potential, although ions of high charge may cause complete charge reversal. Therefore optimum doses of electrolyte are critical.

Flocculation involves the formation of much more open agglomerates than those resulting from coagulation and relies upon molecules of reagent acting as bridges between separate suspended particles. The reagents used to form the bridges are long chain organic polymers, which were formerly natural materials, such as starch, glue, gelatine, and guar gum, but which are now increasingly synthetic materials, loosely termed polyelectrolytes. The majority of these are anionic in character but some of them are non-ionic, and some cationic, but these form a minor proportion of the commercially available products of today's flocculant market. Inorganic salts are not able to perform this bridging function, but they are sometimes used in

conjunction with an organic substance as a cheaper means of charge neutralization, although an ionic polyelectrolyte can and often does perform both functions.

The polyacrylamide, which vary widely in molecular weight and charge density, are extensively used as flocculants. The charge density refers to the percentage of the acrylic monomer segments which carry a charge. For instance, if the polymer is uncharged it comprises n similar segments of the acrylic monomer, the polymer is thus a homopolymer-polyacrylamide. If the acrylic monomer is completely hydrolysed with NaOH, the product comprises n segments of sodium acrylate--an anionic polyelectrolyte, having charge density of 100%. Charge density may be controlled in manufacture between the limits 0-100%, to produce a polyacrylamide of anionic character, weak or strong, depending on the degree of hydrolysis. By similar chemical reactions, polymers of cationic character can be produced. It would be expected that, since most suspensions encountered in the minerals industry contain negatively charged particles, cationic polyelectrolytes, where the cation adsorbs to particles, would be most suitable. Although this is true for charge neutralization purposes, and attraction of the polymer to the particle surface, it is not necessarily true for the bridging role of the flocculant. For bridging, the polymer must be strongly adsorbed, and this is promoted by chemical groups having good adsorption characteristics, such as amide groups. The majority of commercially available polyelectrolytes are anionic, since these tend to be of higher molecular weight than the cationics, and are less expensive.

The mode of action of the anionic polyacrylamide depends on a segment of the very long molecule being adsorbed on the surface of a particle, leaving a large proportion of the molecule free to be adsorbed on another particle, so forming an actual molecular linkage, or bridge between particle. The maximum effect of a flocculant is achieved at an optimum dosage rate; excess polymer can cause dispersion of the particles due to floc breakdown, due to fragile nature of the flocs, flocculating agents are not successful with hydrocyclones, while success with centrifuges can only be achieved with special techniques for a limited range of applications. Even pumping of the flocculated slurry may destroy the flocs due to rupture of the ion-chain molecules. Polyelectrolytes are normally made up to stock solutions of about 1%, which are diluted to about 0.01% before adding to the slurry, the diluted solution must be added at enough points in the stream to ensure its contact with every portion of the system. A shower pipe is frequently used for this purpose.

Mild agitation is essential at the addition points, and shortly thereafter, to assist in flocculant dispersion in the process stream. Care should be taken to avoid severe agitation after the flocs have been formed.

Selective Flocculation

The treatment of finely disseminated ores often results in the production of ultra-fine particles, or slimes, which respond poorly to conventional separation techniques, and are often lost in the process tailings. Selective flocculation of the desired minerals in the pulp, followed by separation of the aggregates from the dispersed material is expected to become an important technique in the future, although plant applications are at present rare. Although attempts have been made to apply selective flocculation to a wide range of ore types, the bulk of the work has been concerned with its application to the treatment of clays, iron, phosphate, and potash ores. A prerequisite for the process is that the mineral mixture must be stably dispersed prior to the addition of a high molecular weight polymer, which selectively adsorbs on only one of the constituents of the mixture. Selective flocculation is then followed by removal of the flocs of one component from the dispersion.

The greatest amount of work on selective flocculation has been concerned with the treatment of fine grained non-magnetic oxidized taconites, which has led to the development of Cleveland Cliffs Iron company's 10 million tonne per year operation in United States. The finely intergrown ore is autogenously ground to 85% minus 25 micron with caustic soda and sodium silicate, which act as dispersants for the fine silica. The ground pulp is then conditioned with a corn-starch flocculant which selectively flocculates the hematite. About one-third of the fine silica is removed in a de-slime thickener, together with a loss of about 10 % of the iron values. Most of the remaining coarse silica is removed from the flocculated underflow by reverse flotation, using an amine collector.

Unit 3 Filtration and Drying

Filtration is the process of separating solids from liquid by means of a porous medium which retains the solid but allows the liquid to pass. Factors affecting the rate of filtration include:

1. The pressure drop from the feed to the far side of the filter medium. This is achieved in pressure filters by applying a positive pressure at the feed end and in vacuum filters by applying a vacuum to the far side of the medium, the feed side being at atmospheric pressure.
2. The area of the filtering surface.
3. The viscosity of the filtrate.
4. The resistance of the filter cake.
5. The resistance of the filter medium and initial layers of cake.

Filtration in mineral processing applications normally follows thickening. The thickened pulp may be fed to storage agitators from where it is drawn off at a uniform rate to the filters. Flocculants are some times added to the agitators in order to aid filtration. Slimes have an adverse effect on filtration, as they tend to blind the filter medium; flocculation reduces this and increases the voidage between particles, making filtrate flow easier. The lower molecular weight flocculants tend to be used in filtration, as the flocs formed by high molecular weight products are relatively large, and entrain water within the structure, increasing the moisture content of the cake, even though pick-up may be improved. Smaller flocs are formed with the lower molecular weight flocculants, which have a higher shere- resistance, and the resultant filter cake is a uniform porous structure which allows rapid dewatering, yet prevents migration of the finer particles through the cake to the medium. Other filter aids are used to reduce the liquid surface tension, thus assisting flow through the medium.

The filter medium

The choice of the filter medium is often the most important consideration in assuring efficient operation of a filter. Its function is generally to act as a support for the filter cake, while the initial layers of cake provide the true filter. The filter medium should be selected primarily for its ability to retain solids without blinding. It should be mechanically strong, corrosion resistant, and offer as little resistance to flow of filtrate as possible. Relatively coarse materials are normally used and clear filtrate is not obtained until the initial layers of cake are formed, the initial cloudy filtrate being recycled.

Filter media are manufactured from cotton, wool, linen, jute, nylon, silk, glass fiber, porous rubber. Cotton fabrics are by far the most common type of medium, primarily because of their low initial cost and availability in a wide variety of weaves. They can be used to filter solids as fine as 10 micron.

Types of filter

Cake filters are the type most frequently used in mineral processing where the recovery of large amounts of solids from fairly concentrated slurries is the main requirement. Those where the main requirement is the removal of small amounts of solid from relatively dilute suspensions are known as screening, or clarification filters.

Cake filters may be by pressure, vacuum, batch, or continuous types.

Pressure filters. Because of the virtual incompressibility of solids, filtration under pressure may have advantages over vacuum. Higher flow rates and better washing and drying may result from the higher pressures that can be used. However, the continuous removal of solids from the pressure-filter chamber can be extremely difficult and consequently, although continuous filters do exist, the vast majority operate as batch units.

Filter presses are the most frequently used type of pressure filter. They are made in two forms—the plate and

frame press and the recessed plate or chamber press.

Vacuum filters. There are many different types of vacuum filter, but they all incorporate filter media suitable supported on a drainage system, beneath which the pressure is reduced by connection to a vacuum system. Vacuum filters may be batch or continuous. Typical batch vacuum filter is the leaf filter, which has a number of leaves, each consisting of a metal framework or a grooved plate over which the filter cloth is fixed.

The most widely used filters in mineral processing applications are continuous vacuum filters, which fall into three classes—drums, discs, and horizontal belt filters.

Drying

The drying of concentrates prior to shipping is the last operation performed in the mineral processing plant. It reduces the cost of transport and is usually aimed at reducing the moisture content to about 5% by weight. Dust losses are often a problem if the moisture content is lower. Rotary thermal dryers are often used. These consist of a relatively long cylindrical shell mounted on rollers and driven at a speed of up to 25 rev min⁻¹. The shell is at a slight slope, so that material moves from the feed to discharge end under gravity. Hot gases, or air, are fed in either at the feed end to give parallel flow or at the discharge to give counter current flow.

The method of heating may be either direct, in which case the hot gases pass through the material in the dryer, or indirect, where the material is in an inner shell, heated externally by hot gases.

Reading material General Methods and Equipment for Coal Dewatering

The degree of difficulty associated with a dewatering problem increases as the surface area of the particles to be dewatered increases. Fine coal is considerably more difficult to dewater than coarse coal because the surface area of coal particles increases as the average particle size decreases. A number of machines and processes are available to the coal industry for efficient dewatering of various sizes of coal and refuse.

Generally, coal sizes larger than 1.5 in (37.5mm) present no particular dewatering problem. Shaker screens of the very crudest types and designs are capable of producing these sizes at low surface moisture. High-speed vibrators are used for the intermediate sizes and can be used for smaller sizes down to 0.02in(0.5mm) if high moistures (15% surface moisture or greater) can be tolerated in the product. Centrifuges of various type can be used practically in the 9.5-0.5mm size range. Solid bowl and screen bowl centrifuges can be used on size smaller than 0.02 in, although a higher product moisture will result. Following the centrifuges, vacuum filter application becomes increasingly appropriate with further size reduction. The disk filter is used most often for dewatering –30 mesh solids.

Considerable expenditures of time and money are made every year by the coal and related industries to determine new and improved methods of cleaning coals. A reduction of ash or moisture in clean coal produced by a cleaning plant is of considerable value to the coal producer. Water on coal loaded from the preparation plant is as contaminating as ash. Water in coal product reduces the heating value, increases the freight cost, contributes to difficulty in handling and shipping and reduces the input and coke yield of metallurgical charges. It therefore behooves the coal operator to remove as much water from his coal as economically possible before loading.

Coal dewatering is required not only prior to loading the clean coal product but may also be required on the feed to some cleaning units. For example, before feeding raw coal to heavy media circuit, the coal should be sized, prewetted, and dewatered; fine coal fed to Deister tables must be prepared to a given solids-water ratio, usually two parts water to one part coal; fine coal fed to heavy-media cyclones should be sized by wet screening methods and dewatered before being mixed with the heavy media. **These dewatering operation on raw coal can be as important to producing a clean product as proper dewatering of coal products is to delivering a dry product.**

Vibration screens usually can dewater coal larger than 1/4 in (6.3mm) to the extent necessary to meet market requirement. Better dewatering results are obtained on a single deck screen than on the bottom deck of a double deck screen because fine coal is delayed in reaching the bottom deck and because water drops from the top deck onto the dewatered bottom-deck coal.

Vibrating screens for the heavy media process

To illustrate the screens used in a heavy media coal preparation plant, consider the following typical installation. Ahead of the heavy media vessel vibrating screens are used for prewetting and fines removal. Prewetting the incoming coal controls the amount of water introduced into the heavy media vessel and assists in the maintenance of the desired specific gravity. Removing the fine material ahead of the vessel prevents contamination of the separating media with fine coal, which has tendency to remain in suspension.

Following the heavy media vessel, the refuse, clean coal, and in some cases a middlings product are handled separately to remove the water and recover the media. The openings in the screen surface range from 0.02-0.1in (0.5-2.5mm), depending upon the requirements of the process. The types of screen surfaces used include stainless steel cloth or plate and parallel rod-type screens. The use of wire cloth is declining because of the short life obtained.

A media recovery screen drains the media, rinses, and then dewateres the coal, middlings, or refuse. To perform these three operations 16ft(4.88m) or longer screens are usually selected, although in some installations two shorter screens in tandem are used. The drain section is usually 4-6 ft at the feed end of the screen and the media drained off at this point is returned directly to the vessel since it contains a high media concentration. After it has passed the drain section, the material is rinsed using spray water, and the media recovered is concentrated before being returned to the heavy media system. Approximately 4-6ft at the feed end of the screen length is used for washing where approximately 5-10 L/min of spray water per ton of coal is applied. The remaining length of the screen is used to dewater the coal, middlings, or refuse to remove the maximum amount of free water before processing or loading. If the quantities to be handled are small, all products can be processed on one screen by using longitudinal partitions to keep the products separated.

Media recovery screens are selected on the basis of the depth of bed that can be successfully drained and rinsed. In general, the finer the coal, the more difficult it is to drain and the thinner the bed must be.

Centrifuges

centrifugal dryers are machines which effectively create high gravity forces for purposes of dewatering coal. Centrifuges are presently used to dewater materials from 1.5 in to 0 in size.

They find application in virtually every wet washing coal plant in existence. Even in plants where the fine fractions are dry screened from the raw coal before entering the wet washing plant. Degradation and screen inefficiency will require installation of some fine coal dewatering equipment. The centrifuges developed for the coal industry are reliable, efficient machines. Their products are consistent, uniform, and easily handled. Properly centrifuged coal can be further dewatered only by evaporation of the moisture remaining on the coal. In general, centrifuges can be classified into two types: perforate basket and solid or screen bowl. There are three types of perforate basket machines:

Basket without transporting device,

Basket with positive-type discharge,

Vibrating basket type.

And the other type includes:

Solid bowl centrifuges,

Screen bowl centrifuges,

The sieve bend

The hydraulic cyclone

Reading Material Treatment of Mine Water

Nature and Formation of Coal Mine Drainage

The nature of acid mine drainage, which may be used as makeup water in many coal preparation plants, varies from site to site. The nature of the dissolved salts and their concentration determine the usefulness of this water for makeup purposes.

Virtually all coal contains pyrites of varying amounts. This iron-sulfide mineral, when exposed to air and water in the course of mining, is oxidized to ferrous sulfate. This salt in most cases leads to the formation of acidic iron-bearing water when dissolved in water. In a few instances, "ferrous bicarbonate" is formed and an alkaline iron-bearing water is produced.

Pollution of mine water cannot be completely prevented since presently there is no reasonably practical method of eliminating mine water. One of the serious difficulties is the wide variation in both composition and volume which occurs not only between different mine water in the same coalfield but also in the same water. Some mine water is excellent in quality and is taken by water companies and used for domestic and industrial purposes. Other water is less pure and innocuous and suitable only for industrial purposes. In some localities mine water can be found which contains acid and appreciable quantities of iron, is highly saline, and contains excessive amounts of suspended solids. Work sponsored by the Pennsylvania Department of Mines and Mineral Industries has indicated that in certain locations mine water can be utilized in the coal preparation plant as a portion of the makeup water, so at certain other locations, the volume and quality of the acid mine drainage is of such a nature that excessive corrosion would result in the coal preparation plant, and supplemental treatment must be undertaken in order to prevent or minimize pollution from this source.

Treatment of Mine Water

There are various methods used for water clarification. These are applied to slurry or froth flotation tailings, singularly or in combination with one or two of several methods including flocculation, concentration, vacuum filtration, and pressure filtration.

Several preparation plants are using coal mine discharge for makeup water. Although it is not most desirable, it is often the lowest cost and sometimes the only water available. The objectional characteristics of mine water are its high conductivity and sulfuric acid content which causes corrosion problems. As mentioned earlier, pollution of mine water arises primarily from the oxidation of pyrite, marcasite, and other sulfides to form ferrous sulfate and sulfuric acid. However, in some cases the mine water contains iron and the water is alkaline due to carbonate solution. The use of mine water greatly reduces demand for both surface water and public water. Most coal preparation plants use large quantities of mine water of quality suitable for industrial use and could use more. In many cases, the largest supplies of surplus mine water occur in, localities where there is least demand for industrial water. As a result, it is doubtful whether full use could be made of these resources at low cost.

It might be mentioned that there are large quantities of rainwater at collieries, such as runoff from roofs and roadways, which are discharged to sewers. At some collieries surface drainage is salvaged for use in the coal preparation plant. Where water shortages exist it is advantageous to segregate surface drainage from the sewage system and provide storage facilities.

After settlement or other treatment, the water can be used as a supply of industrial water for cooling purposes, boiler feed, and as makeup water for the main industrial supply.

Of the many treatment techniques for acid mine drainage, the most technically sound and economically feasible method at present involves lime neutralization and aeration followed by preparation plant usage. Much

work has been devoted to this technology and to the development of engineering parameters for widespread industrial application. With pollution abatement law, in addition to the recurring drought and water shortages in coal producing areas, consideration is being given to the treatment and reuse of mine drainage as makeup water for preparation plants and as potential feed water to industrial and domestic supplies.

This part is concerned mainly with the contamination aspects of fine coal cleaning and "black water" disposal. Attention is given also to the nature and formation of water from coal mine drainage systems and the treatment of these waters for industrial use. Some attention is devoted to the cost of installing and operating the various beneficiation systems for the removal of suspended solids. Minimum operating and maintenance costs are functions of the proper selection and geometry of refuse disposal areas. Disposal procedures are varied but must be rigidly pursued or difficulties will result. It should be recognized that fine solid refuse disposal systems must be carefully designed to minimize contamination.

Part 6 Advanced Coal Processing Technologies

Unit 1 Chemical Comminution for Coal Cleaning

Chemical comminution is a size reduction process that involves the exposure of raw coal to certain low-molecular-weight chemicals that are relatively inexpensive and recoverable. The chemical breakage is unique in two ways: (1) microscopic studies have demonstrated that fragmentation is strongly controlled by boundaries between maceral and mineral matter resulting in greater mineral matter liberation than mechanical crushing to a similar size, and (2) bimodal breakage occurs resulting in the generation of a small amount of fine particles. Both these features make chemical comminution attractive as a substitute or an adjunct to mechanical crushing, which is commonly incorporated into coal preparation plants. The selective liberation allows for lower sulfur value at a given recovery or a better recovery at a given sulfur value.

Chemical comminution of coal with ammonia has been studied by Syracuse Research Corporation (SRC) since 1971. This work led to the coal fragmenting techniques that have been patented by SRC. These patents describe the basic process as it applies to the chemical fracturing of coal after it has been mined in the normal manner. Two other patents based on this technology apply to in situ mining of coal by chemical comminution in 1977, Catalytic, Inc. joined with SRC to undertake the continued development and commercialization of the chemical comminution process.

Chemical comminution, as a promising cost-effective approach of meeting increasingly stringent pollution standards while using cheaper lower-grade coals, warrants continued development.

In combination with conventional coal cleaning processes, chemical comminution offers a means of increasing the removal of pyritic sulfur in some coals. The special economic value of the process lies in its ability to improve the yield of desulfurized coal product without, at the same time, reducing particle size and creating troublesome excessive fines.

Bench-scale data indicate that the process will largely eliminate the expensive equipment needed for processing fine coal by permitting the use of relatively simple hydrocyclone-type equipment which effectively cleans coarser coal. These savings largely offset the initial construction cost of the chemical comminution processing unit.

Realistic cost analyses of coal-cleaning plants utilizing chemical comminution for size reduction have shown that the increased product yield or decreased sulfur content, or both, would make the final preparation system more profitable than an equivalent plant processing mechanically crushed coal. A further advantage is that chemically comminuted coal can be readily stored and easily shipped, thus eliminating problems associated with mechanically ground fine coal.

For certain eastern coals, chemical comminution may be the key to producing a fuel sufficiently clean for direct combustion, for any particular coal, environmental acceptability will depend on its calorific value and proportions of pyritic and organic sulfur. For "cleaned coals" not meeting environmental standards, the process promises to reduce the amount of post combustion desulfurization required to meet future air pollution standards. Furthermore, cleaned coals should result in increased boiler reliability and availability by reducing power plant maintenance costs.

In applying chemical comminution in a coal preparation plant, it has been proposed first to crush the coal by mechanical means to about 1.6 in. (4 cm) top size. The crushed coal may be dried next and treated with liquid ammonia, or alternatively it may be placed in a closed vessel which is subsequently evacuated and then pressurized with gaseous ammonia. In either case the coal is exposed to the comminuting agent for 1-2 hr,

depending on the kind of coal, treatment conditions, and degree of size reduction needed. Following the chemical comminution step, the solids are separated from the ammonia and washed with water to remove any traces of free ammonia. The coal can then be beneficiated by any appropriate physical separation method. However, since the comminuted coal is likely to be in the 0.4 in. x 0 (1 cm x 0) size range, the most appropriate beneficiation methods are likely to include the use of heavy media cyclones, hydrocyclones, wet concentration tables, and froth flotation cells.

Unit 2 The OTISCA Process

--An Anhydrous Heavy-liquid Separation Process for Cleaning Coal

Raw coal in the natural state is a physical mixture of carbonaceous material with a density range of 1.2 to 1.7 g/cm³ and mineral matter with a density range of greater than 2.0 g/cm³ and extending up to that of iron pyrite with a density of 5.0 g/cm³. There is reasonable evidence that a fraction, if not all, of the mineral matter in coal is distributed throughout all of the size ranges down to the submicrometer range. The mining and or crushing of raw coal to some maximum diameter usually results in a size distribution of particles which is reasonably well approximated by a Rosin-Rammler distribution function. Crushed raw coal, therefore, consists of discrete particles of coal, mineral matter, and mixtures of the two bound together as they originally existed in their natural state. Physical separation of these components based on the density differential between the carbonaceous material and mineral matter has employed separation devices utilizing air fluidization, vibratory motion, or liquids of an intermediate density such as organic liquids (usually referred to as heavy liquids), salt solutions, and fine heavy minerals dispersed in flowing water (called heavy media). Other separation devices, such as tables, jigs, spirals, and hydrocyclones, utilizing a combination of frictional and/or gravity or centrifugal forces, are used to affect an apparent density differential between the coal and mineral matter. Since air fluidization, vibratory motion, heavy media and other combined processes have been discussed fully in coal preparation articles and textbooks, this article will be primarily concerned with physical separations involving heavy liquids. Particular emphasis of the discussion will be placed on an anhydrous, heavy-liquid separation process for cleaning coal, called the Otisca process.

The use of a heavy liquids, such as carbon tetrachloride, of an intermediate specific gravity to separate two or more phases of specific gravities, one above and the other below that of the parting fluid, has been recognized of about 100 years as the near perfect means of achieving an absolute separation of the solids. The exciting prospects of using a heavy liquid separation system to beneficiate coal were studied intensely in the 1930s by the Dupont Company and put into a pilot plant for cleaning anthracite at the Western breaker, Sheandah, Pennsylvania, shortly thereafter. With the exception of this pilot plant and a small number of continuous float-and-sink separators used for quality control, heavy liquids have not been used commercially in coal preparation. The most recent application of the heavy-liquid separation process for cleaning coal is called the Otisca process, which has been under continued laboratory investigation and pilot plant development since 1972. The Otisca process is a water less, heavy liquid separation process employing CCl₃F as a parting liquid with certain physical and chemical characteristics which make it particularly advantageous for expanding the familiar laboratory float-and-sink test into the realm of commercial coal cleaning. In this process, a raw coal of any maximum particle diameter and its complete size distribution and with surface moisture up to 10 wt% is subjected to separation in a static bath of the parting liquid. The products of separation, namely, coal and its mineral matter, are transported directly to their respective evaporators where essentially all of the parting

liquid is recovered for reuse. The two primary advantages of this process are that the raw coal feed does not have to be screened or sized in any manner; and even so separation produces a product coal with its chemistry and weight, and Btu yield, which closely approximate the results of the classical washability test for the same raw coal and size distribution throughout the entire distribution of size.

The parting liquid used in this has two key characteristics; (1) It does not react to any known extent with the coal product or refuse material and (2) it permits complete dispersion of the coal product particles throughout the separation both, even though surface liquid coal reaction, may be cited as one of the major reasons for the failure of previous attempts at commercializing the heavy liquid process for cleaning coal. For example, if a reaction does occur between the heavy liquid and coal or its mineral matter, complete heavy-liquid recovery becomes a very expensive, complex process. The cost of the heavy liquid alone prohibits its loss from the process. The complete dispersion of the coal particles from the refuse material will ensure near theoretical recovery of the coal product with a minimum amount of mineral matter, which is the objective of the process.

The separation behavior in a heavy liquid is generally defined by Stokes' law where the velocity of separation in a static bath is proportional to the square of the particle radius times the density differential between the particle and parting liquid, and is related inversely to the viscosity of the liquid phase. In the case of coal separation, with a suitable parting liquid and given sufficient time in a static bath, all but a very small fraction of the particle will either float or sink irrespective of size above the limit for Brownian motion. The need, therefore, does not exist for the screening of a raw coal into special size ranges to perform a heavy liquid separation. The use of certain additives in very small concentrations in the parting liquid provides a dramatic control over the middling concentration and the concentration of water and included slimes on the coal product. Separation mechanism and velocities can also be controlled by the presence of certain other additives.

In this article, the basic principles and recent developments of the density differential separation utilizing anhydrous, heavy construction and testing of a pilot plant, which demonstrates the commercial viability and advantages of coal cleaning using the Otis process, are then discussed. The commercial viability of the process depends upon producing more coal product of a better quality at a lower cleaning cost than can be achieved presently by commercial hydrobeneficiation techniques. The advantages of the process include very low noise levels and dust-free coal product, a refuse material of less than 12% moisture which is nearly free of coal and can be readily compacted, the elimination of all water treatment streams such as "black water," settling ponds, water refinement procedures, and in general, a low plant maintenance effort. The projected capital investment for fine coal cleaning utilizing the process in the range of \$21,000 per raw coal ton per hour, and the estimated operating costs total about \$1.47 per raw coal ton. Details of the process cost estimation along with the large-scale commercial facilities under construction using the process are also described.

The complexities of preparation of a coal with a size distribution of essentially 4-0mm with 15wt% of that size distribution smaller than 0.075 mm and 5 wt % smaller than 37 micron is a well-established fact in the coal industry. Materials handling, dusting, excessive both viscosities, and very inefficient separations are typical of these problems. The solution to these problems always results in an increase of coal preparation costs, often dramatically. On the other hand, it is also well accepted that as the size distribution of most coals is decreased, the potential for pyritic sulfur and ash reduction increases.

The Otis process appears to offer a solution to the pose dilemma in that the process is able to recover more fine coal with less misplaced material at a lower processing cost than alternate processes which are currently available. An important feature of the Otis process is that both bench-scale process simulation and pilot-plant test results closely duplicate the theoretical washability data. In contrast, when transferring the theoretical washability data to the conventional wet beneficiations systems, such as the heavy media separation by a water magnetite suspension, a significant loss in separation efficiency can be experienced.

Further, in this line of comparison and, while it may have already become obvious to the reader, it should be emphasized that an otisca coal cleaning plant is significantly different from the typical hydrobeneficiation plant. The otisca process plant is completely enclosed; and consequently, noise and dust levels are considerably less than those encountered in a conventional plant. The chemicals used in the otisca plant are non-flammable, nontoxic, virtually odor free, and non-corrosive, which allows for inexpensive construction of material handling and electrical equipment. Thermal or mechanical driers necessary to remove process water from the coal water. Hence, the following unit operations are not required: clarifiers, settling ponds, water treatment, bag houses, vacuum filters, filter presses, centrifuges, cyclones, screens, magnetic separators, fluidized bed driers, venturi scrubbers, multistory structures, and dewatering screens, finally, by comparison to a conventional hydrobeneficiation plant, the otisca plant flowsheet is extremely simple in that the raw coal does not have to be classified in order to provide a suitable feed to multiplicity of separating unit operations.

Plant atmospheric emissions of the nontoxic parting liquid can be reduced to extremely low values through proper seal design, product liquid removal, and non-condensable gas incineration. The parting liquid losses under these conditions could be reduced to much less than 0.05 lb/raw coal ton. The nontoxic additives constitute a fixed loss which may vary between 0 and 1 lb/raw coal ton, depending upon the particular coal being separated.

The raw coal feed to an otisca plant may vary from any reasonable top size by zero, as the ash and pyritic sulfur releases may dictate. For example, batch investigation has indicated that separations of settling-pond coal fines, essentially 100 wt%-200 mesh are practical. The lack of classification equipment and size sensitive separation operations in an otisca plant leads to the obvious fact that the otisca process can routinely tolerate wide fluctuations in raw coal feed rates, size distributions and chemical analyses.

Unit 3 Electrostatic Beneficiation of Coal

Small particles can be charged by triboelectrification, conductive induction, or corona charging. The charges developed depend on the particle characteristics such as composition, crystal structure, and surface states, as well as environmental factors such as temperature and humidity. After charging, particles of different types can be separated according to charge to mass ratio using forces in an electric field. Changes in the method of charging, the surface preparation, or the environmental factors can be used to alter the degree of separation of particles of different types. Experimental techniques and apparatus have been developed for application of electrostatic beneficiation principles to separation of ash and pyrite from coal as well as separation of the coal macerals themselves.

The electrostatic separation methods are dry processes and therefore offer many advantages, both in terms of energy efficiency and environmental pollution, over conventional wet separation processes. Only the moisture presenting the mined coal need be driven off and the energy loss associated with the evaporation of large quantities of water absorbed during a flotation separation is avoided. The Hat Creek coals in western Canada contain a significant quantity of clay minerals which form a gelatinous mass during any coal washing process, presenting a formidable water pollution problem. This kind of problem can be completely avoided by electrostatic separation techniques. The environmental problem becomes one of containing dust particles using conventional electrostatic precipitators.

The basis of any electrostatic separation or beneficiation process for finely divided matter is the interaction between an external electric field and the electric charges acquired by the various particles.

Unit 4 High-gradient Magnetic Separation for Coal Desulfurization

High gradient magnetic separation (HGMS) is a relatively new technology which promises to be a practical means for separating micrometer-sized, weakly magnetic materials on a large scale and at much faster flow rates than are possible in ordinary filtration. The technology is also applicable to separating nonmagnetic materials which can be made to associate with magnetic seeding materials such as magnetite. It was developed in 1969 for the wet separation of weakly magnetic contaminants (iron pyrite, titanium oxide, etc.) from kaolin clay to upgrade its quality and brightness. A typical HGMS unit used in this application is illustrated schematically in fig 4.1. the electromagnetic structure consists of energizing coils and a surrounding iron enclosure. The coils in turn enclose a cylindrical working volume packed with fine strands of strongly ferromagnetic materials such as ferritic stainless steel wools. With this design, a strong field intensity of typically 20 kilo-oersted or 1.591 million ampere per meter can be generated and distributed uniformly throughout the working volume. Additionally, by placing in the uniform field the ferromagnetic packing materials

Unit 5 Selective Oil Agglomeration in Fine Coal Beneficiation

the most important feature of oil agglomeration as applied to wet coal preparation is its ability to recover virtually quantitatively the finest coal particles encountered in plant fines circuits and waste streams. Inorganic impurities and moisture are simultaneously rejected from the agglomerated product.

fine particles in liquid suspension can be agglomerated in a number of ways. One of the oldest procedures involves the addition of electrolytes to the suspension to cause a reduction in the zeta potential and allow colliding particles to cohere. A second method involves the use of polymeric flocculants to bridge between particles. A third method, which is our subject here, involves the addition of a second immiscible liquid preferentially to wet the particles and cause adhesion by capillary interfacial forces. While the bonding forces in the first two methods are small and result in rather weak and voluminous agglomerates, the third method can produce denser and much stronger agglomerates.

In the case of fine coals, the Carbaceous constituents can be agglomerated and recovered from an aqueous suspension with many different oils as collecting liquids. Inorganic or ash forming constituents remain in suspension and are rejected. As with froth flotation, oil agglomeration relies on differences in the surface properties of coal and dirt to effect a separation. Froth flotation, however, becomes less effective where extremely fine particles of coal must be treated or if there is considerable clay slime present. By contrast, there appear to be virtually no lower limit on the particle size suitable for oil agglomeration. For example, colloidal particles of silica have been collected by the technique. If desirable, size distributions containing particles 1/8 in in diameter or larger can also be treated. In addition to this ability to treat a broad size range of coal particles, oil agglomeration can produce a dense, coarse granular product of acceptable strength and low moisture content without the need for vacuum or thermal drying.

There is an increased interest today in oil agglomeration because of the larger quantity of fines which must be dealt with in coal preparation. These fines are produced by natural degradation, increasingly mechanized mining methods and by the grinding necessary to liberate finely disseminated impurities from the lower-quality coals now being processed. Much of the current examination of processes based on preferential wetting by immiscible liquids is due to studies at the national research council of Canada. This work resulted from interest in a densified rounded form of agglomerates formed when suitable amounts of bridging liquid are added under appropriate agitation to a solids suspension. A family of techniques, generally known under the original label of spherical agglomeration processes, has subsequently been developed.

Selective oil agglomeration can be carried out over a broad range of conditions. For a given feed material, product ash and combustible recovery are essentially constant from oil concentrations of 5 or 10% to 30 or 40% by weight of solids. Pulp densities from very low levels to as high as 40 or 50 wt% solids can be treated. Intensive agitation over short periods of time (less than 1 min) or less intensive mixing over prolonged periods can be used. Coal and oil properties are important system variables in selective agglomeration. The ash level of agglomerates attained with a given coal depends on the extent of liberation of the impurities and hence on the fineness of ash dissemination and the level of grinding. Grinds down to a few micrometers in diameter can be agglomerated successfully, if desirable. With bituminous coals, many different oils yield excellent coal recovery levels although the refined, light oils give the best ash rejection. Lower-rank coals require special oils for agglomeration because of their more hydrophilic character.

Moisture content in the agglomerates depends on the quantity of oil used. As larger and more compact agglomerates are produced at higher oil levels, both the internal and surface moisture held in the product is reduced. Excessive amounts of oil, however, may trap relatively large amounts of moisture in the resulting coal

oil amalgam. Waste tailings rejected in the oil agglomeration process have good settling characteristics. Reduced waste fines loadings resulting from the technique extend tailings pond life. The treatment of waste fine coal slurries and recovery of coal from tailings ponds are probably the most attractive applications for selective oil agglomeration at the present time.

Unit 6 Leaching with Chemicals

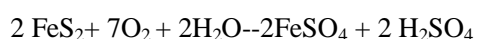
Leaching with Dissolved Oxygen Solutions

Hot aqueous solutions containing dissolved oxygen will extract pyritic sulfur from coal. This has been demonstrated in numerous laboratory tests carried out over a wide range of experimental conditions. Although the rate of extraction is slow when coal is leached under atmospheric pressure and at temperatures below 212F (100C), the rate is fairly rapid at elevated pressures and temperatures. Thus, when -100 mesh (-0.15 mm) Illinois No. 6 coal was leached at 266~F (130C) with a solution containing oxygen dissolved under a pressure of 300 psia (2070 kPa), more than 90% of the pyritic sulfur was extracted in 2 hours. Reasonably fast rates of extraction also have been realized while leaching with solutions containing air dissolved under pressures between 300 and 1000 psia (2070 and 6895 kPa) and at temperatures between 302 and 392F. (150 and 200C).

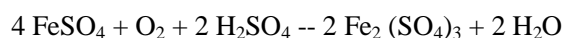
Under some conditions part of the organic sulfur is also extracted from coal by hot, oxygen-bearing solutions. The removal of up to 30 or 40% of the organic sulfur has been accomplished by leaching at 266F (130C) with a basic solution containing ammonium hydroxide, or by leaching at temperatures above 302F (150C) with an unbuffered solution which became acidic as a result of the leaching reactions.

When coal is leached with hot oxygen-bearing solutions, some of the oxygen reacts with the organic matter, resulting in an increase in the oxygen content of the coal and in the formation of some carbon dioxide and a lesser amount of carbon monoxide. Consequently, the treatment can destroy the caking properties of the coal.

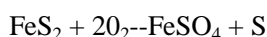
The chemistry of the dissolved oxygen leaching process is complex and seems to change with temperature, concentration, and pH. When pyrites are leached at relatively high temperatures with a solution of low acidity, the pyrites appear to react as follows:



A large portion of the ferrous sulfate produced by the preceding reaction is further oxidized to ferric sulfate, as follows:



Also, at moderate pH (but not at low pH) the ferric sulfate is hydrolyzed and ferric oxide is precipitated. As a result of these reactions, water soluble sulfates are produced which are readily extracted. If, on the other hand, pyrites are leached at somewhat lower temperature with a more acidic solution, part of the sulfur will be converted to the elemental form which is not water soluble. The formation of elemental sulfur appears due to the following reaction which competes with the preceding one:



The production of elemental sulfur complicates the extraction of sulfur from coal since its removal requires another treatment step, such as extraction with a hot organic solvent.

Agarwal et al. proposed several different versions of a coal leaching process which utilizes oxygen-bearing solutions. In one version, ground coal is leached in an aqueous solution containing relatively little acid so the pyritic sulfur is converted to soluble sulfates. After separating the coal and leach solution, the solution is treated with lime to precipitate gypsum and iron oxide or hydroxide, which are discarded. The water is

recovered and returned to the start of the process. In a second version, coal is leached under more acidic conditions so part of the pyritic sulfur is converted to soluble sulfates and part to elemental sulfur. The spent leach solution is treated as before while the coal is subsequently extracted with a hot organic solvent to remove elemental sulfur. In a third version, coal is leached with a solution containing ammonium hydroxide so the pyritic sulfur and part of the organic sulfur are converted to soluble sulfates. The spent leach solution is treated as before with the result that most of the ammonium hydroxide is recovered along with the water, which is then recycled.

The economic feasibility of a sulfur extraction process depends on a number of factors, some of the more important ones being: the yield of clean coal, the consumption of oxygen or other agents, and the rate of extraction which bears on the capital cost. Since oxygen reacts with coal as well as with pyrite at elevated temperatures and pressures, some oxygen is consumed unproductively with a corresponding loss in coal heating value. Sateen et al. measured the oxygen uptake of Illinois No. 6 high volatile B bituminous coal, lignite, and semi-anthracite in aqueous slurries and measured the oxygen, consumed in converting coal carbon into carbon monoxide and carbon dioxide. For the conditions required to extract 90% of the pyritic sulfur from Illinois No. 6 coal containing 2% pyritic sulfur, the total oxygen consumption was estimated to be 0.125 lb O₂ per lb (0.125 kg O₂/kg) coal. Of this total, 28% would be consumed by the pyrite reaction, 43% would be taken up by the coal, 25% would be converted to carbon dioxide, and 4% would be reacted in other ways or lost. If lignite were treated under similar conditions, the oxygen consumption would be several times greater, whereas if semi-anthracite were treated, the oxygen consumption would be much lower.

Sareen also measured the oxygen uptake of Illinois No. 6 coal in ammonium hydroxide solutions at 266F (130C) and 300 psia (2070 kPa) and found it to increase with reaction time. For a reaction time of 2 hr about 0.033 lb O₂ per lb (0.033 kg O₂/kg) coal was taken up by the coal and about an equal amount of oxygen was consumed in converting coal carbon into carbon dioxide or carbonates. For the ammonium hydroxide reaction system, the loss in both coal carbon and heating value increased with ammonium hydroxide concentration as well as with reaction time. For a reaction time of 2 hr at the indicated temperature and pressure, from 5-11% of the coal carbon was lost and from 7-13% of the heating value, depending on the ammonium hydroxide concentration which varied between 0.5 and 3.0 molar.

The rate of extraction of pyritic sulfur from Illinois No. 6. coal in acidic solutions was also measured by Sareen et al. at different temperatures and oxygen pressures. The rate was found to increase greatly between 176 and 266F (80 and 130C) while employing a solution containing oxygen dissolved under a pressure of 300 psia (2070 kPa). In addition, the rate was found to be proportional to the square root of the oxygen pressure and to increase as the particle size was reduced. On the other hand, the rate was about the same for lignite and semi-anthracite as for Illinois No. 6 coal. Other experiments with ammonium hydroxide solutions resulted in about the same rate of extraction of pyritic sulfur, but in addition, organic sulfur was also extracted at a significant rate. Although the rate of extraction of pyritic sulfur can be increased by employing higher and higher temperatures and pressures, the rate of oxidation of coal also is increased by these measures. Hence, for any given coal there is a limit beyond which it becomes uneconomical to increase the temperature and oxygen pressure.

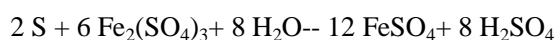
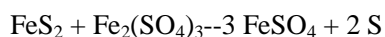
Other alternatives for increasing the rate of extraction of pyritic sulfur are to reduce the particle size, alter the composition of the leach solution, and change the pH of the solution. In this regard McKay and Halpern found the rate of oxidation of relatively pure pyrite in aqueous suspension to be proportional to the pyrite surface area, and Majima and Peters observed the oxidation rate to be greater in a 1 molar sodium hydroxide solution (pH 14) than in solutions buffered with phosphates or ammonia and having a lower pH.

Leaching with Solutions of Ferric Sulfate

The extraction of pyritic sulfur from coal by hot solutions of ferric sulfate has been demonstrated in laboratory tests. Thus, in one series of experiments, from 83-98% of the pyritic sulfur was removed from different samples of coal collected in the Appalachian, Interior, and Western coal bins. For these tests 100 mesh (0.15 mm)x0 coal was treated with 1 normal ferric sulfate solution under refluxing conditions [216F (102C)] for 4-6 hr. The solids and liquid were separated subsequently and the solids were mixed with fresh leaching solution and treated as before for several more hours. The solids were recovered, washed, and then extracted with hot toluene to remove elemental sulfur produced by the ferric sulfate treatment.

Although the time required to extract all of the pyrites from a given piece of coal may be quite long, a large fraction can be removed in a reasonable period. For example, in leaching a number of different coals at 212F (100C), it required nearly 24 hr to extract 95% of the pyrite but only 1 hr to extract 68%, on the average. The leaching time can also be reduced by precleaning to remove as much coarse pyrite as possible and by employing higher temperatures. Moreover, there is a wide variation in the rate of extraction of different coals. Consequently, much of the pyrite can be removed from some coals in 2 hr whereas others may require considerably longer time. Even though fine-size coal can be leached more quickly than coarse coal, Meyers indicated that promising results were also obtained in laboratory tests with coarse coal having a top size of 0.1-0.4 in.(3-10 mm).

When pyrites are leached with ferric sulfate, the following two reactions appear to take place, according to Stokes:



In dilute solutions the second reaction seems to be slower than the first, which leads to an accumulation of sulfur. By reacting relatively pure pyrite with ferric sulfate solutions of different concentration for 1 hr at 212~F (100~C), Yurovskii attributed the slow rate of extraction in dilute solutions to blinding of the pyrite surface by an adhering film of elemental sulfur, and also to adsorption of ferrous sulfate and aluminum sulfate on the pyrite surface. The aluminum sulfate would be derived from the clay minerals present in coal. Garrels and Thompson concluded that the rate of reaction of pure pyrite is controlled by the differential adsorption of ferric and ferrous ions on the pyrite surface, with the rate being proportional to the fraction of the surface covered by ferric ions. Sasmojo also reached this conclusion, and suggested that the rate-controlling step of the reaction mechanism is an electron transfer reaction between an adsorbed ferric ion and the pyrite surface.

Some of the other mineral matter in addition to pyrite is extracted from coal by hot ferric sulfate solutions? However, the free-swelling index and other coking properties of the coal seem to remain intact.

Elemental sulfur deposited on the coal by the first reaction given previously can be extracted with a suitable organic solvent, or alternately it can be removed through heating and vaporization.

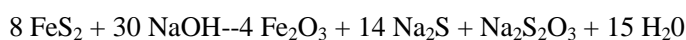
Caustic Leaching

A method of extracting most of the inorganic sulfur and part of the organic sulfur from coal by leaching it with dilute hot caustic solutions has been demonstrated in laboratory experiments. It also has been shown that subsequent washing or treatment of the product with dilute mineral acids will remove much of the mineral matter. Thus, when -200 mesh (-0.075 mm) Illinois No. 6 coal containing 9.8% ash, 1.1% pyritic sulfur, and 0.2% sulfate sulfur was leached with a 10% sodium hydroxide solution at 437F (225C) for 2 hr and then treated with a dilute hydrochloric acid solution, the product contained 0.7 % ash, 0.1% pyritic sulfur, and 0.01% sulfate sulfur? Similar results were obtained with Indiana No. 5 coal. For these treatment conditions the organic sulfur content seemed to remain about the same or to increase slightly. Moreover, the free-swelling

index was only slightly affected.

By using more rigorous treatment conditions, Stambaugh and his coworkers succeeded in removing from 24-60% of the organic sulfur from various samples of coal along with most of the pyritic sulfur. Apparently this was accomplished by employing the following implied treatment conditions: temperatures in the range of 437-662F (225-350C), pressures in the range of 350-2500 psia (2410-17,240 kPa), a leach solution containing 4-10% sodium hydroxide and about 2% calcium hydroxide, coal ground so that 70% passed a 200 mesh (0.075 mm) screen, and an extraction time of up to 30 min. The treated coal was reported to be completely noncoking and nonswelling and more reactive for gasification with steam or hydrogen. The greater reactivity was probably due to the retention of some sodium and/or calcium and alteration or modification of the coal structure.

Although the chemistry of caustic leaching is not well understood, the early work of Stokes and somewhat later work of Stevens suggest that the principal reaction between pyrites and sodium hydroxide is the following:



This reaction produces insoluble iron oxide and soluble sodium sulfide and thiosulfate, with the relative proportion of sulfide to thiosulfate being very large. These products have been observed in leaching coal in about the proportions indicated along with some soluble iron and sulfate. The sulfate may have resulted from exposure of the spent leachant to atmospheric oxygen.

Caustic leaching also extracts some of the trace metals and affects the clay minerals. Reggel et al. suggested that the clay minerals are probably dissolved by the alkali and then reprecipitated as insoluble sodium aluminum silicate ($3\text{Na}_2\text{O} \cdot 3\text{Al}_2\text{O}_3 \cdot 5\text{SiO}_2$). This material dissolves when the coal is treated subsequently with acid.

A method of regenerating the spent caustic leach solution has been proposed by Stambaugh and his co-workers. The method first involves treating the solution with carbon dioxide, which converts sodium sulfide into hydrogen sulfide and converts unreacted sodium hydroxide into sodium carbonate/bicarbonate. The hydrogen sulfide is separated and converted into elemental sulfur by the Claus process. Lime is added next to the spent solution to regenerate sodium hydroxide and precipitate calcium carbonate and dissolved impurities. Finally, the precipitated calcium carbonate is calcined to regenerate lime and carbon dioxide which are recycled to treat more spent solution.

During World War II, I. G. Farbenindustrie in Germany proposed to produce de-ashed coal for conversion into coke to make carbon electrodes for aluminum smelting by chemically leaching coking coal which had first been beneficiated by jig washing and froth flotation. The ash content of the coal was to be reduced from 12 to 0.8 % by physical beneficiation and from 0.8 to 0.28% by chemical extraction. The extraction step involved leaching coal with a 2.5 % solution of caustic soda for 20 min at 482F (250C) and 120 atm (12,160 kPa) pressure followed by washing with a 5% solution of hydrochloric acid and then water. The method was demonstrated with laboratory apparatus which could leach 441 lb per day (200 kg/day) of coal and produce 265 lb per day (120 kg/day) of coke. A total of 22 tons (20 mt) of coke was produced with this apparatus for evaluation. It was reported that the leaching step "destroys at least half the caking properties for a good coking coal." Construction of a plant which could leach 10 tph (9.1 mt/hr) of coal was undertaken but apparently was not completed because of bomb damage at the mine site where it was to be located.

It has also been demonstrated through laboratory experiments that sulfur can be extracted from bituminous coal by employing molten or fused caustic rather than aqueous solutions. For this demonstration a mixture containing equal parts of sodium and potassium hydroxide was used. When -40 mesh (-0.425 mm) Pittsburgh seam coal was contacted for 30 min with the fused mixture in a series of experiments at different temperatures,

increasing amounts of pyritic sulfur were removed as the temperature was raised from 302 to 437F (150 to 225C). At 437F (225C) all of the pyritic sulfur was extracted, and further increases in temperature resulted in removal of organic sulfur with the quantity removed increasing as the temperature was raised up to 752F (400C). The total sulfur content of the coal was reduced from 1.6 to 0.5% by the treatment at 752F (400C). However, the treatment at 752F (400C) also reduced the volatile matter content of the coal from 36.1 to 23.6% and the free-swelling index from 6.5 to 0. By limiting the treatment temperature to 482F (250C), the pyritic sulfur was extracted without greatly affecting the volatile matter content or free-swelling index. Somewhat similar results were realized when Illinois high volatile B coal was extracted with molten caustic, but when Wyoming subbituminous coal was extracted, it decomposed extensively.

COMBINED PHYSICAL AND CHEMICAL CLEANING

Yurovskii described the pilot plant demonstration of a method of cleaning coal which combined physical separation with chemical leaching. Coal having a size consist of 0.12 in. x0 (3 mm x 0) was mixed with a concentrated dense solution of ferric sulfate and fed to a centrifugal separator, which produced an impure float fraction and a very concentrated refuse (sink) fraction. The float coal was separated from the dense solution by a centrifuge and conveyed to a leaching reactor, where the disseminated pyrites were extracted with a hot and less concentrated solution of ferric sulfate and nitric acid. The leached coal was subsequently recovered from the leach solution by a second centrifuge, washed, and dewatered by a third centrifuge. When coal from the Yasinovka mine No. 3/5 of the Donets Basin having an initial sulfur content of 5.82% and ash content of 25.8% was cleaned in the pilot plant, a product having a sulfur content of 2.17% and ash content of 4.5% was produced in 60% yield.

This operation was unusual in several respects. It not only combined physical and chemical methods for cleaning coal but utilized ferric sulfate solutions both as a heavy medium and as a leaching agent. Moreover, the leaching solution also contained nitric acid, and, therefore, it comprised two agents known to attack pyrite instead of just one. Yurovskii credited this approach with several advantages: (1) the mixture of leaching agents provided a more rapid rate of extraction than would have resulted from the use of ferric sulfate alone; (2) the mixture did not seem to affect adversely the coking properties of the coal, whereas the use of nitric acid alone would have; (3) with nitric acid present in the leach solution, all of the dissolved iron was present in the ferric state; and (4) all of the ferric sulfate needed for the various steps in the process was produced by the process itself. A further advantage of this approach was that it did not leave the coal contaminated with elemental sulfur as would have been the case if only ferric sulfate had been used.

Unit 7 Bacterial Removal of Sulfur from Coal

Studies have been reported on the effects of bacteria oxidation of sulfur in coal. In general, research has centered around the use of *Thiobacillus ferrooxidans* and *Ferrobacillus ferrooxidans*. These bacteria are acidophilic, chemosynthetic, and autotrophic. They are acidophilic because they thrive in an acidic environment, chemosynthetic since they derive their energy from inorganic chemical reactions, and autotrophic in that they utilize inorganic material in the synthesis of living matter. Recent work has investigated the existence of these bacteria in acid mine drainage, their natural growth medium, and the use of acid mine drainage in the bacterial removal of sulfur.

Ferrobacillus ferrooxidans was first isolated from acid mine drainage and described in 1954. It is nearly always present in acid mine drainage from bituminous coal mines. The microorganism oxidizes ferrous iron to

ferric iron in a pH range of 2.0-4.5. Silverman and Lundgren reported that the optimum conditions for the iron oxidizing mechanism involving *F. ferrooxidans* are a pH of 3.0-3.6 and temperature of 98.6°F (37°C), although no bacterial growth occurs at this temperature. The optimum pH and temperature for growth of the bacteria are 3.5 and 68-77°F (20-25°C),

respectively. The cells are rod-shaped and 1.0-1.6 μ long. It is the inability of this bacterium to oxidize thiosulfate ($S_2O_3^{2-}$) that distinguishes it from the genus *Thiobacillus*. Margalith et al. showed that pure cultures of *F. ferrooxidans* will oxidize elemental sulfur, but preferentially oxidize ferrous iron because of its greater solubility. Ferrous iron can reach the active site of the iron oxidizing system within the bacterium more rapidly than the slightly soluble sulfur. Silverman and Lundgren also found that the bacteria could slowly oxidize elemental sulfur, but were unable to oxidize ammonium, thiosulfate, di- and trivalent manganese, cobalt, or nickel.

Thiobacillus ferrooxidans was first isolated from acid mine drainage and described by Colmer and Temple et al. in 1951. It too is nearly always present in acid mine drainage, and has been shown to oxidize ferrous iron, sulfate, and metallic sulfides, but its responses to thiosulfate characterize the genus. *Thiobacillus* can oxidize thiosulfate; *Ferrobacillus* cannot. A pH of 2.5 is optimum for growth on iron, sulfur, and metal sulfides. Landesman et al. found the optimum temperature and pH for oxidation of ferrous sulfate 104°F (40°C) and 1.75, respectively. Thiosulfate ($S_2O_3^{2-}$) was shown to be inhibitory to iron oxidation by *T. ferrooxidans*. Thiosulfate concentrations as low as 0.005% significantly slowed the rate of bacterial ferrous iron oxidation. Cations were shown to be generally innocuous to the iron oxidation process, but salts of halides and nitrates were inhibitory. Concentrations of chloride or nitrate of 0.14 M were toxic to the bacteria. Inhibitory substances do not kill the bacteria but significantly reduce its ability to oxidize iron. Toxic substances, however, kill the bacteria. Lazaroff recognized that *T. ferrooxidans* must have sulfate present to oxidize iron, and that the higher the concentration of chloride or other inhibitory anions, the higher the sulfate concentration must be for iron oxidation to continue.

Several authors have studied bacterial oxidation of pyrite in coal from the standpoint of acid mine drainage production and methods of its abatement.

Presently, studies are being conducted to examine the acceleration of bacterial action in order to remove pyritic sulfur from coal prior to its utilization. Previous works along these lines include Zarubina et al., Ashmead, and Silverman et al.

Silverman et al. studied the effects of various physical factors on the rates of pyrite oxidation, the role of *F. ferrooxidans* and *T. thiooxidans* (not *T. ferrooxidans*) as oxidizing agents, and the susceptibility of various forms of pyritic sulfur to oxidation by bacteria. The pyrites used by Silverman et al. were two pyrite concentrations from washed coals and two sulfur balls from coal seams. The results of their study showed that the smaller the pyrite particle size, the faster the oxidation. *Thiobacillus thiooxidans* did not oxidize pyrite of any size. Coarsely crystalline pyrite was not appreciably oxidized by either bacteria in their study, but coarsely crystalline marcasite was oxidized by both, though *T. thiooxidans* did so to a small extent. A pyrite sample containing the mineral calcite ($CaCO_3$) did not oxidize with either bacteria, but with the calcite removed by HCl, *F. ferrooxidans* readily oxidized the pyrite. This is attributed to the acid-neutralizing capacity of the carbonate which increased the pH to exceed the upper limit for the activity of this acidophilic bacterium. *Ferrobacillus ferrooxidans* were shown to accelerate the oxidation of coal pyrite and sulfur ball pyrite but not of coarsely crystalline pyrite, even though X-ray diffraction patterns indicated that all three were the same.

Later, Silverman et al. expanded their study to experiments involving whole coal. They reported that pyrite within lignite, subbituminous, and bituminous coals were susceptible to bacterial oxidation, provided acid conditions were maintained acceptable to the acidophilic requirements of *F. ferrooxidans*. The maintenance of

these acid conditions was dependent on the neutralizing capacity of the coal, which appears to be a function of the rank (lignite > subbituminous > bituminous). Lignites, subbituminous coals, and some bituminous coals high in calcite require an acid pretreatment before the bacteria can oxidize the pyrite. Ferric sulfate supplementation of acid-treated coals, regardless of rank, increased the removal of pyrite. Silverman also reported that the bacterial oxidation of pyrite in whole coal was most effective on the smallest particle sizes due to their relatively greater surface areas.

It still should be pointed out that some of the difficulties in using bacteria for the removal of pyrite are the handling of necessary small sizes of the coal, the handling of acid mixtures, and waste disposal without adding to water pollution.

Part 7 INSTRUMENTATION AND PROCESS CONTROL

Unit 1 Introduction

What is automation? Why automate? Webster's New Collegiate Dictionary defines automation as "the automatically controlled operation of an apparatus, process, or system by mechanical or electronic devices that take the place of human organs of observation, effort, and decision." This definition gives an implied answer to the second question, "Why automate?" Automation, automatic control devices, and process control equipment receive their justification in supplementing or replacing manpower, which results in decreased operating costs.

In the past, processes were operated with the minimum amount of instruments needed merely to guide operators. Processes were manually controlled by the operator on the basis of simple temperature, pressure, flow, and level indications on a panel. Today the tendency is to design a new plant to operate in a fully automatic manner according to a preset program utilizing a minimum of manpower. In addition to this direct cost reduction in manual labor, increased equipment capacity results from uniformity of flow and uniform control of process variables. This control eliminates or minimizes surges and permits equipment to be operated at or near its ultimate capacity and, for a given material throughput, permits the use of smaller process equipment. Automatic control, therefore, be it in the field of coal processing or in any industrial field, is simply a problem of economics. Control equipment is justified in terms of its contribution toward increasing productivity, increasing profit margins, or producing a new and salable product at a profit.

Instruments and control devices can be divided into two general groups: (1) units which measure and (2) units which both measure and control a process variable.

In coal processing, the first class of instruments, those which measure only, include analytical devices such as automatic ash, moisture, and sulfur measuring units and the simple pressure, temperature, flow, and level indicating devices mentioned previously. The second group of instruments, those which both measure and control, includes a wide variety of units such as continuous weighing devices on conveyor belts and photoelectric cells which gage the position of railroad cars on the track and insure that the cars are in the proper location for loading (see Chapter 15).

In one aspect the latter units as used in the coal industry differ significantly from those used in other industries. In coal processing these instruments need not be as highly sophisticated and accurate as in other industries; hence their cost is proportionately lower than in other industries. Because of the economic fact that coal sells for only one or two cents per pound compared to dollars per pound for many chemical products, and because quality control requirements are not as rigid in the coal industry as in other industries, measurement errors of 1 or 2% can be tolerated. Their effect can be calculated in terms of a few dollars as compared to many hundreds or even thousands of dollars in the other industries. These less sophisticated instruments generally have the additional advantage of being more rugged and requiring less maintenance.

It is unusual to find a control instrument in a coal processing plant, which is so sophisticated and sensitive that it must be kept in a controlled atmosphere to protect it from dust and dirt or vibration as is frequently the case in other industries. These differences in instrumentation requirements are illustrated by the fact that the total plant cost in the chemical industry, for example, can be increased by more than 20% with the addition of adequate instrumentation. On the other hand, instrumentation costs in a coal processing plant amount to only a few percent of the total capital cost. For example, a simple one point temperature recording controlling instrument which requires a controlled atmosphere for proper operation might cost \$2500 or more, whereas a simpler, less sophisticated model required in a coal processing plant might cost as little as \$500. Therefore,

investment in instrumentation and process control devices can be much more readily justified in the coal processing industry than in other industries. A relatively small dollar investment can pay large dividends in more economic operations.

Despite the advantages of instrumentation and control devices, caution must be used in selecting and installing such equipment. A control instrument must justify itself. A poorly installed, improperly selected, or unnecessary unit can be less desirable and less beneficial than no unit at all. Selection of suitable control equipment can be a relatively time-consuming and complex task.

Instruments that are the most appropriate for any given set of requirements must be chosen from the many available types. Careful consideration must be given to the suitability and compatibility of the control device with existing instrumentation and processing equipment. No generalized recommendation can be made as to the advisability of using a specific type of control equipment in a given coal processing plant. Equipment must be selected within the framework of each individual set of circumstances existing at a given plant or in a given geographic area. For these reasons this chapter does not concern itself with the specific types of instrumentation necessary for any given flowsheet in a coal preparation installation. The information given here should be considered as a guide for the selection of the proper components in a process control system and not *construed* (v.解释, 分析, 直译) as a recommendation to use any component provided by a specific manufacturer. This chapter attempts to provide, in one convenient location, the greatest possible amount of information for the evaluation and preliminary selection of process instrumentation. It is not intended to be a substitute for the literature, for the recommendations of instrument manufacturers, or for proper engineering design and evaluation of control needs.

Because of the wide variety of available control and analytical instruments which are currently in use, or have potential application, in the coal processing industry, this chapter is organized into sections containing a general description and discussion of specific groups of instruments followed by a list of typical, commercially available pieces of equipment within that group.

The groupings are made according to the function of the equipment (e.g., chemical analysis, temperature measurement), rather than by their application (e.g., thermal dryer control) or by their manufacturer. In general, the equipment lists are organized with the simpler, least complex items first and the more sophisticated devices last. For the most part, the items appearing first in each list are currently in use, or have been used, in coal processing applications, whereas the later items are not in general use but have potential for future application in the industry. In those cases where the specific applications of an instrument may not be obvious to the reader, examples of such applications or potential applications are given in the discussion of each piece of equipment. Other examples of specific applications of instrumentation and control devices are given as a part of the discussions of preparation equipment in the earlier chapters of this book.

Unit 2 ANALYTICAL INSTRUMENTS

The chemical analysis of coal has been performed for a number of years according to the "Standard Method of Laboratory Sampling and Analysis" set up by the ASTM. This method, bearing ASTM designation D-271, was originally adopted in 1927 and has been revised a number of times over the years, in general, the analytical techniques specified in the standard method are based on relatively slow, manually controlled methods of analyses. For example, although the specified wet chemical or Eschka method of determining sulfur in coal is

very accurate, it takes a number of hours to perform. For the purpose of control in a continuously operating plant, this method of analysis is far too time-consuming.

It may be necessary, for example, to determine in a relatively short time the sulfur and ash content of cleaned coal issuing from a float-sink washing unit in order to adjust the specific gravity of separation to compensate for changes which may occur in the ash and sulfur content of the run-of-mine coal feeding the washery. To eliminate this type of problem, automatic analytical devices have been developed permitting the rapid analysis of coal and coal ash. For example, there are a number of resistance- and induction-type furnaces on the market which can be used to determine the sulfur content of coal in a matter of minutes rather than hours. Similarly, spectrographic techniques can be used for the rapid analysis of coal ash.

In addition to these analytical devices, equipment has been developed to analyze coal continuously for its ash and moisture content while it is moving through the coal preparation plant on conveyor belts. The US Bureau of Mines (USBM) has developed a unit for determining coal moisture by using nuclear radiation techniques. In this device, coal moving on a conveyor is bombarded with fast neutrons which rebound as slow neutrons upon striking molecules of water in the coal. The percent of neutrons so changed has been found to be directly proportional to coal moisture content. Similar techniques have been used by Bethlehem Steel to measure the moisture content of coke being fed to their blast furnaces.

Devices are also available to monitor coal ash continuously by using X-rays. In one such device, developed at the Dutch State Mines and marketed in the United States by American Minechem Corp. under the trade name CendreX, two X-ray beams are used. One beam strikes the coal; the other strikes a standard object, usually a Plexiglas plate. The difference in intensity of the two reflected beams is measured by using an automatic compensator, a fluorescent screen, and a photomultiplier. This equipment gives off an electric current which is proportional to coal ash content and can be used to operate automatic control equipment. The unit has a claimed reproducibility of 0.1% coal ash.

The National Coal Board and the Atomic Energy Authority of the United Kingdom have developed an ash monitoring device to continuously measure ash content of coal. In the unit, radiation from a plutonium 238 source bombards the coal and is absorbed or backscattered depending on the elements present in the coal. Combustible elements backscatter radiation well while ash constituents absorb radiation. Thus, by measuring the backscattered radiation, ash content of the coal can be determined and readings can be converted to control signals for the regulation of feeding and blending equipment. The ash monitor is marketed in the United States by Sortex Co. of North America and is claimed to have an accuracy of $\pm 2\%$ ash over an ash range of 15-35%. Higher accuracies are claimed on low-ash washed coals.

The preceding discussion gives only a few examples of the numerous types of automatic analytical equipment available for use in the coal industry. The following list of some specific types of analytical instruments is far from complete, but it does represent a fairly wide sampling of the many types of equipment available.

Figures for equipment accuracy given in this list and subsequent lists are average expected error as percentage of full scale, unless otherwise stated. In general, well-designed, carefully installed, and properly maintained instruments can be expected to provide accuracy as stated or even better. Information given on the accuracy of each instrument assumes that a uniform, truly representative sample of the material to be analyzed has been obtained.

Following is a list of device to measure chemical properties

1. Induction furnace sulfur analyzers (Leco) A small coal or coke sample is placed in a crucible in an induction furnace. The sample is combusted in a stream of prepurified oxygen. Acid sulfur gases generated

(SO₂) are selectively measured by an infrared detector; percent sulfur in the sample is read directly. Range: 0.01-3% sulfur with about 0.002 oz (0.05 g) sample; higher sulfur contents with smaller samples. Accuracy: +/-3% of sulfur present. Remarks: very rapid method; results available in about 2 min; technique suitable for coke and metallurgical coals. For coal containing more than about 2% sulfur, a resistance furnace sulfur analyzer is preferred. Induction furnaces

are also available for rapid (10-15 min) determination of coal ash.

2. Resistance furnace sulfur analyzers (Leco) These are similar in principle to the induction furnace analyzers described in the previous paragraph.

3. bomb calorimeter

4. gas chromatograph

5 x-ray fluorescence spectrometer

Unit 3 PROCESS MONITORING AND CONTROL INSTRUMENTATION

As discussed previously, industrial control and monitoring equipment falls into two general classes, those instruments which measure (sensing devices) and those which both measure and control. Generally, measuring devices for temperature, pressure, flow, etc. may be used by themselves or in conjunction with a control device to form an integrated automatic control system. In effect, most sensing or measuring devices, although they may not be directly connected to a control unit, are used for control purposes because they are employed as a guide for manual operation. In many instances, the measuring instruments are also used in conjunction with a recorder to provide a permanent record of a given process variable for quality control purposes, material and energy balances, and cost accounting.

The measuring instrument itself must be carefully designed and selected for the conditions and ranges of values expected to be encountered. Therefore, various measuring instruments differ radically from one another according to their specific uses.

Conversely, controller mechanisms control almost any variable automatically and do not differ significantly in design and construction. Controllers are actuated by changes in process variables as expressed by an electric, pneumatic, or other type of impulse transmitted from the sensing instrument. Therefore, it is conceivable that a given control valve, for example, could be actuated by an impulse from a temperature measuring device, a pressure measuring device, a flowmeter, or a level indicator.

In addition to measuring process variables and actuating controllers, measuring instruments have a number of other useful functions within a plant. In conjunction with an appropriate control unit, they can alter and adjust control points to fit a given work schedule or a given schedule of changes desired in a particular process variable. They can start and stop parts of a process at preset times. They can sound alarms in the event hazardous conditions occur, and, if the conditions become severe, can totally shut down a plant to prevent accidents. They can be used in a circuit to control a variable in proportion to a change in some other variable. And they can transmit data or impulses over long distances (for example, from a mine to the tipples many miles away) without sacrificing a significant amount of time or accuracy.

A true measuring instrument or sensing device must have three distinct functional parts: a sensing or primary element to detect changes in the measured variable, a device for transmitting the sensed change, and some type of device to receive and measure or indicate the transmitted data. The measuring instrument consisting of these three parts can be self operated or operated by an external source of energy such as electric power. To illustrate these three elements, consider a standard mercury-in-glass thermometer. The sensing element is the mercury reservoir contained in the base of the thermometer. The mercury expands or contracts in accordance with changes in the bulb temperature. The transmitting device, a fine capillary tube, is filled to varying heights as the mercury changes in volume. The measuring device, a scale painted or etched on the tube surface, in turn indicates temperature.

A more complex example is a temperature recorder; the sensing device is a thermocouple which, in response to changes in temperature, generates a varying voltage. The transmitting device is a series of wires which carry the voltage to a potentiometer. This in turn indicates changes in temperature on a chart in direct proportion to the voltage being transmitted by the thermocouple. The measuring element is the potentiometer in combination through a complex electromechanical linkage with a movable pen and pen arm tracking on a chart or strip of paper. The chart itself may be stationary or moving at a constant speed in order to introduce

time as a factor in the recorded data.

The principles by which measuring devices operate have been well-defined and expressed in mathematical terms and are fairly well understood. However, a mathematical analysis of such devices is beyond the scope of this book. Information on the mathematical relationships by which measuring and control devices operate is readily available in the literature.

A list of density and specific gravity measuring devices:

1. Hand hydrometer A weighted float, with a small diameter stem at the top, sinks in the liquid to a depth proportional to the specific gravity. The scale is read at liquid level. Range: adaptable to any specific gravity. Accuracy: third or fourth decimal place. Remarks: widely used where automatic operation is not needed.

2. Balanced-flow vessel (Halliburton) is a fixed volume vessel through which liquid flows continuously. Liquid is weighed automatically by a scale or a force-balanced transmitter. Range: any specific gravity of liquids. Accuracy: 4-1%. Remarks: generally applicable to automatic density control.

3. Industrial specific-gravity displacer This unit is similar to a displacer for level measurement except that the float is submerged and is externally counterbalanced. Range: any liquid density. Accuracy: second or third decimal place. Remarks: a widely used method for industrial recording and control of liquid density.

4. Chain-balance-float density recorder (Precision Thermometer and Instrument Co.) The bob in a continuous-flow vessel is weighted to carry half the weight of a light chain, when submerged in medium-density liquid at the midpoint in its range. The chain is attached to the bottom of the float and to the vessel wall at half-height. Any increase in density causes the float to rise, supporting more chain; any decrease causes it to sink, supporting less chain. Float position is transmitted by a differential transformer.

5. Gamma-ray density gage (Industrial Nucleonics, Ohmart, others) Gamma radiation from an appropriate source (e.g., cesium 137) passes through the liquid. The amount of energy reaching a detector varies inversely with the density of the process stream. Accuracy: $\pm 0.2\%$. Remarks: the unit takes continuous density measurements of solutions, liquids, slurries, and divided solids. This type of unit is also gaining acceptance in the control of medium specific gravity in heavy medium cyclone circuits.

Unit 4 THE AUTOMATIC CONTROL MECHANISM

Up to this point, discussion has been primarily limited to descriptions of devices for measuring a process variable and has not dealt to any great extent with the control of a process based on these types of measurements. As mentioned before, there are several basic types of automatic controllers which can be used in conjunction with instruments measuring any desired variable.

A controller can be defined as any unit which detects a change from the set point of a measured variable and takes corrective action to restore the process to the set point. This restoration to the set point, unfortunately, is not an instantaneous change as might be desired. The manner in which the process variable returns to the set point and the time which is required to bring the variable back into control is a function of the type of system involved, the type of measuring instrument used, the type of control instrument used, and the time lags which may exist within the control circuitry or in the process itself.

In many instances a significant amount of time may pass before the system begins to respond to a signal from a controller. This time lag may be a function of the process itself, as, for example, the length of time required for a conveyor belt to empty itself of its current load before beginning to transfer a new material, or the inertia of conveyor drive motors which must be overcome before the speed of the conveyor can build up to a new desired level. Other examples include the time required to empty a hopper or bin before changing materials within a bin, the finite amount of time the final control element itself requires to complete the necessary operation, and dead time---a lag which occurs when the metering element is located some physical distance away from the control point.

An example is illustrated by a pH meter located in a water effluent pipe of a washery to measure the acid content of waste water issuing from the plant. When the pH meter senses an overly high increase in water acidity, it may relay a signal back to the washery to actuate a control valve adding an alkaline material to the water to decrease its acidity. The dead time is the amount of time required for the neutralized water introduced at the control point to arrive at the pH sensing element in the effluent pipe. During this period of time, control action has actually been taken, and the water has been neutralized, but the pH meter itself is still sensing a highly acid condition and is calling upon the controller to continue adding a neutralizing agent to the water stream.

Such dead-time lags can be extremely detrimental to process control. In this example, by the time the neutralized water reaches the pH meter, the control has been calling for an excessive amount of alkali and may have raised the pH of the water above the desired level. The pH meter will then sense the high alkalinity of the water and cause the controller to stop adding alkaline material, thereby causing the water to become acid again and initiating a series of oscillations in the acid and alkaline content of the water which may, due to excessive dead time, continue infinitely. To avoid such a cyclic type of control, modifications must be made to the control system to limit the rate at which it can respond to the signal for a change or to build a delay equivalent to the dead time in the controller itself.

To eliminate or to minimize the types of cyclic problems which arise from time lags in a control circuit, a number of different types of controllers have been devised, and a number of different control actions have been incorporated into them. The most rudimentary of these is the so-called on-off or two-position control. This type of control is simply a system which chooses between one of two possibilities in control. In the case of the neutralization of acidic water, it would be a system which either closes the neutralizing stream valve completely or opens it fully with no intermediate positioning possible.

An improvement on two-position control is floating control. In floating control, the final control element--in this case the valve--moves at a constant speed in either direction, whenever the controlled variable

deviates from its set point. Motion of the valve continues until the controlled variable returns to its set point, or until the valve reaches a fully open or fully closed position.

However, neither of the foregoing systems can be used if significant time lags are present and cycling is undesirable. In such cases, proportional control might be preferred. In this type of control, the final element operates in direct proportion to the change in the measured variable.

In the water neutralization system with proportional control, the valve would open or close a preset amount in direct proportion to a specific pH change. Such a control system is usually fitted with a band or control range called the proportional band. In this case the system might have a proportional control band of, for example, 6-7 pH value. At pH 7, the valve adding alkali might be totally closed. At pH 6 it might be totally open, and at intermediate values of pH could be partially open in direct proportion to the pH value. At any value of pH above 7, the valve would always be closed. At any value of pH below 6, the valve would always be fully open. Thus, the proportional control would exist only in the preset proportional band, that is, between pH 6 and pH 7.

From the preceding example, it can be seen that if the pH change exceeds the preset band, and the band is not manually changed, a cyclic operation could again result because the fluctuation is greater than the controller is capable of handling. To minimize this problem, another type of control can be implemented, reset control, in very simple terms, with reset control the required manual change in the proportional band is made automatic. As soon as the controlled variable deviates from the set point, there is a gradual and automatic shift in the proportional band to bring the variable back to the set point. In other words, proportional plus reset action controls the final position of the valve and its rate of movement in proportion to the pH of the liquid.

A third type of control action can be incorporated with the proportional and reset actions. This is called rate (or derivative) action. In rate action, again using the example of the acid stream, the position of the valve is made a function of the rate at which the controlled variable, the stream acidity, is changing. In effect, rate action provides an initially large temporary overcorrection when a fluctuation occurs to counteract the effect of process time lags. Therefore, in the final combined proportional plus reset plus rate action, the valve position is a function of three things: (1) the absolute value of the acidity of the stream (proportional control); (2) the rate at which the acidity of the stream is changing (rate control); and (3) the rate of movement of the valve stem, which in turn is proportional to the stream acidity (reset control).

The latter type of control is the most desirable type of control for processes with large time lags. With proportional plus reset alone, there can be wide deviations of the controlled variable from its set point if large time lags exist (greater than 2 mt). In addition, the time required to restore the variable to its set point may be quite long, particularly if a wide proportional band and a slow reset rate is used. With the addition of the rate action, the effect of the proportional action upon the final control element is advanced, frequently eliminating this problem.

Part 8 PRELIMINARY CAPITAL AND OPERATION COST ESTIMATION

Prior to constructing a new plant or expanding existing facilities, a detailed preliminary cost estimate is necessary to determine the capital investment required and to project future operating costs of the new facility. Undertaking construction of such facilities is senseless unless sufficient funds will be available to complete the work and a reasonable degree of certainty exist that the new venture will produce adequate revenue to justify the investment required. The following discussion outlines one possible method of performing such a preliminary estimate to guide the investment decisions. It must be emphasized that the cost guidelines presented here are not intended as a substitute for actual cost data developed through experience and based on sound accounting procedures.

Undertaking n.任务; 事业; 企业 计划, 企图, 承办, 承包 担保, 保证, 许诺 殡仪业

welfare undertakings 生活福利事业

Project n.计划, 方案, 事业, 企业, 工程 v.设计, 计划, 投射, 放映, 射出, 发射(导弹等), 凸出

Revenue n.收入, 国家的收入, 税收

Justify vt. *To demonstrate or prove to be just, right, or valid*

证明...有道理[应该], 为...辩护

为...提供法律根据, 宣誓证明(自己)有财力作保

【刷】整(版), 装(版), 调整(铅字)的间隔使全行排满

justified each budgetary expense as necessary; anger that is justified by the circumstances.

证明每一项预算费用都是必要的; 被事实证明是有正当理由的愤怒 Capital Costs

The fine quality of the cloth justifies its high price.

布的质量好, 价钱高是应该的。

Can you justify yourself in saying that?

你能证明你有着充分理由可以那样说吗?

Capital Costs

Before beginning an estimate of capital costs, several factors must be determined. The proposed plant flowsheet must be prepared in as detailed a manner as possible. Also, it is desirable to determine the exact geographical location of the proposed facility. The projected production rates of all products, byproducts, and major wastes must be known. In a coal preparation plant these would include the various coal products (i.e. metallurgical and steam coal) and refuse materials and, in some instances, by products from sale of coal-associated minerals.

When this information has been determined, a decision must be made as to whether a *battery limits* plant or a *grass roots* plant is required. A battery limits estimate is based on the assumption that all necessary utilities, railroad sidings, haulage roads, etc. are already available at the proposed plant site and that the land has been developed. The grass roots estimate assumes the proposed site is completely undeveloped. A grass roots estimate may total two to three times as much as a battery limits estimate and represents the more conservative approach to cost estimation.

Haulage n. 拖运; 拖曳 拖运费//运输; 输送; 搬运//牵引量; 搬运量//牵引力; 拖力 运输距离运输方式

haulage man

运输工

belt haulage
 (用)胶带输送
 cable haulage
 钢索运输
 electric(al) haulage
 电机车运输; 电气牵引
 face haulage
 回采工作面运输
 gathering haulage
 汇集调车
 gravity haulage
 重力运输
 inland haulage
 内陆运输费用
 mine haulage
 矿山运输
 primary haulage
 【采矿】回采工作面运输
 rail haulage
 轨道运输
 tractor haulage
 拖拉机拖运
 trolley haulage
 架空线电机车运输
 underground (belt) conveyor haulage
 井下(皮带)输送机运输

After selecting the type of estimate to be applied, a detailed listing of all equipment, materials, land, etc., should be prepared from the plant flowsheet. Next, the cost of each individual item should be estimated, using any of a number of methods, depending on the accuracy required, the urgency of the project, and the degree of completion of each design. Equipment costs, including structures, should be determined on an installed basis. Obviously, the most accurate estimating technique is to solicit firm quotations from equipment suppliers, construction firms, etc. Usually, however, design data is insufficient to permit such quotations to be made until construction plans have been finalized. Therefore, less accurate techniques generally must be employed. Preliminary equipment quotations can be obtained from vendors. Alternately, published cost data can be used, and this data can be corrected and updated to provide for cost increases that may have occurred since publication or that may be anticipated prior to actual plant construction.

Finalize v. 把(计划,稿件等)最后定下来, 定案

Quotation n. 引用语, 价格, 报价单, 行情表

Anticipated 预先的, 预期的

Tabulation n. 作表, 表格

Solicit vt, vi (对金钱、帮助或协助的) 恳求, 乞求, 索要 appeal /ask/ beg / request

Vendor n. 卖主

Generally, a published tabulation of cost data will include, in addition to the cost figures themselves, a value of the Engineering News-Record (ENR) construction cost or building cost indices, the Marshall and Swift (M&S) equipment cost index, or some other suitable index. These indices are used to date cost information and permit adjustment for the effects of inflation. For example, a tabulation of installed equipment costs published in 1974 might have included a value of 431 for the M&S index. A similar tabulation published in 1969 might have reported a value of 285. The increase in the index from 1969 until 1974 reflects the proportionate increase in equipment costs over this five-year interval. In other words, equipment costs in 1974 were approximately 431/285 times greater than those in 1969 (about 50%). Therefore, these indices are, in effect, measures of the cost-of-living at the time the data was originally obtained.

Indices may also be used to project future costs by extrapolating the value of the index to future time. Such projections, however, should be made with extreme caution as they assume a fixed rate of inflation that may not prove valid. In general, projections of this nature should be used only for approximation purposes and should preferably be limited to no more than one or two years.

It is important to note that the various cost indices have different bases that must be understood before using the index. For example, the two engineering news-record indices are based on construction and building costs while the Marshall and Swift index is based upon installed equipment costs. Some of these indices also have subindices for specific industries. The M&S index, for example, has a subindex specifically for the installed costs of mining and milling equipment.

In order to use the cost data correctly it is necessary to determine the current value of the cited index (or the projected value at the time of purchase or construction). Values of the various indices are available in both tabular and graphical form in a number of regularly published magazines and trade journals. Typical data, taken from Chemical Engineering, is shown in Fig. 5-6. (omitted)

Once the current (or projected) value of the appropriate index is obtained, the cost data can be updated merely by multiplying the obsolete cost figures by the ratio of the current (or projected) value of the index to the value published with the cost data. It should be noted that, if no index value is included with the cost data, an appropriate value may be estimated from graphs such as those in Fig. 5-6 by selecting the value from the graph that corresponds to the date of publication of the cost information.

Using one or more of these procedures (firm bids, preliminary quotations, or projections from published data) the cost of all major items of equipment should be determined. Costs of auxiliary equipment of a minor nature should be estimated by using the same technique or simply multiplying the cost of major equipment by a ratio of minor to major equipment costs based on company records for past jobs.

Once equipment costs have been estimated, installation costs must be determined if not included in the original estimate. Installation costs include both the cost of installation materials (piping, wiring, etc.) and labor. Again, direct quotation or published data may be used. Published labor data, which is generally included with published data on equipment prices, is usually given in terms of man-hours required to perform a given job and can be converted to dollars merely by multiplying by the prevailing wage rates in the area. Installation materials are normally cited as a percentage of equipment costs, or they may be based on the ratio of installation materials costs to equipment costs from records of past company projects. Total estimated installed equipment cost is equal to the sum of major equipment costs, auxiliary equipment costs, and the cost of installation materials and labor.

To complete the estimate of capital investment it is necessary, for a battery limits estimate, to add the costs associated with installation of utilities on the site, cost of design and engineering, land costs, etc. Generally, engineering and design and minor or unforeseen items can be included by adding approximately 20% to the capital investment (10% for engineering and 10% for contingencies 或有费用). Instrumentation, if not included

in equipment costs, can be accounted for by adding an additional 5%. Alternately, different percentages may be used based on prior company experience. Grass roots estimates would also require the addition of site preparation costs, access costs for utilities and transportation, auxiliary off-site facilities, etc. The final figure then may have to be adjusted upward or downward, depending on working conditions at the job site. Generally, however, the procedure outlined should be adequate for preliminary purposes. In addition, if the work is to be performed by an outside contractor, an allowance must be added for his fee. A breakdown of the various capital cost items is given in following table.

Table x.xx Some Fixed Capital cost Items

Battery limits estimation
Buildings and constructions
Major equipment
Installation of equipment
Piping
Instrumentation
Engineering and design
Construction supervision
Contractors fees (if performed by outside contractor)
Contingencies(unforeseen costs)
Grass roots estimates (above items plus)
Land and site improvement
Auxiliary facilities(e.g access roads,power substations,fire control,etc.)
Outside utilities (e.g., piping to site,electric feeders,etc.

The many variable factors existing in the coal preparation industry prohibit establishment of a firm general set of estimation guidelines applicable to any coal company in any given area. To gain further insight into estimation techniques, the reader is urged to refer to the published literature and to company records.

Working Capital

An additional capital investment factor that cannot be depreciated and hence is frequently overlooked in preparing cost estimates is working capital. Working capital is simply the amount of money and materials that must be available at all times to keep the plant operating. These funds must be provided when the plant is built and therefore must be included in the required capital investment.

For a coal preparation plant working capital would include:

- 1) Product inventories---coal (and byproducts, if any) in storage awaiting sale or shipment;
- 2) Supplies inventory--spare parts, chemicals, media, construction materials, etc.;
- 3) Accounts receivable (approximately 30 days' sales in most cases)--in other words, the average amount of money normally owed to the company by customers; and
- 4) Current assets, including cash on hand; less
- 5) A credit for current liabilities (approximately 30 days' operating costs in most cases)--the amount of money normally owed by the company to workers and suppliers.

Depreciate v.折旧, (使)贬值, 降低, 贬低, 轻视

Inventory n.详细目录, 存货, 财产清册, 总量

Liabilities 债务,负债

Operating Costs

Once capital investment has been determined, production costs can be rapidly assessed. It is suggested that a standard form be set up for such estimates. One typical form that is applicable to virtually any process including coal preparation is shown in Fig. 5-7.

In preparing such forms it is imperative that the basis of the estimate in terms of production rate, location, process used (heavy media, tables, etc.), annual operating days, and cost index at the time of estimation be clearly stated. These factors have a major effect on cost and must be clearly understood to prevent misuse of the estimate.

When the basis of estimation is established the depreciable portion of capital investment must be determined. This is equal to the total investment less working capital and salvage value of the plant. Working capital and salvage value are theoretically recoverable at any time upon future plant liquidation and hence cannot be included in depreciation allowances. It should be noted that cost of land is usually excluded as a depreciable item unless the construction and operation of the plant would reduce the land value upon plant liquidation.

Depreciable adj. 可贬值的, 可蔑视的

salvage value 折余值, 废品价值, (保险)海难后的残余财产价值

Liquidation n. 清算

Raw Materials Costs

Raw materials costs are calculated by itemizing each raw material required, the annual quantity of each material, and its unit cost---dollars per ton or dollars per gallon, etc. (See lines 14 of Fig. 5-7.) Total annual cost is determined by multiplying the quantity of each material by its unit cost, and adding these figures (line 5). If desired, the annual cost of any item in the estimate may be reduced to a unit basis (e.g., dollars per ton of clean coal produced) by dividing the annual cost by the annual production rate. It should be noted that the run-of-mine coal delivered to the preparation plant is, in fact, the major raw material for the plant and must be included in the estimate as such. Its unit cost is represented either by the actual production cost or by the particular value assumed by the firm for accounting purpose. although the preparation plant is an integral part of the mining complex and run -of-mine coal is not actually a purchased commodity, in the estimate it must be treated as such in order to obtain a realistic economic appraisal.

Miscellaneous Credits and Debits

These credits and debits (lines 6-9) are the various byproduct values associated with the plant. The primary byproduct in coal preparation is a debit--coal refuse. It is included in terms of the annual tonnage (based on washability data) multiplied by the estimated cost per ton for disposal. If possible, this cost should be determined from company records due to the wide variance in refuse disposal methods and haulage distances. In the absence of such information, a cost of 0.5-1\$ per ton may be assumed based on individual circumstances. If disposal costs are handled in this manner, however, care must be taken not to include labor and other costs associated with disposal in the plant costs. Otherwise double counting of disposal costs is possible.

Also included in miscellaneous credits would be any credit expected from sale of refuse material or coal-associated minerals or any materials (e.g., road fill) used by other company departments and transferred to them at some book value.

Direct Expense

Utilities Utility costs (lines 11-16) (i.e., steam, water, electricity, and natural gas) are included in the estimate by calculating the annual quantity required and multiplying by the unit cost. Utility services are usually purchased from a regulated company and unit costs can be obtained directly from the company or the regulatory body. If the utility services are provided in house--for example, from a company-owned generator or water filtration plant--the unit costs should be based upon the incremental cost to the company to produce the additional utility requirement or on the company's book value for the utility service, depending upon

established accounting practice.

Labor Labor costs (line 17) are computed by multiplying the number of man-years required to operate the new facility by the prevailing wage rates in the area under consideration. This item includes direct labor costs only and does not include the cost of fringe benefits, supervision, and nonproducing support personnel (i.e., clerical staff, maintenance, sales, etc.). It should also be remembered that for around-the-clock, 168-hr per week operations, allowance must be made for 4.2 normal 40-hr man-weeks for each job each week, not 3 or 4. The additional 0.2 shifts should be included at overtime rates along with any other anticipated overtime costs. Depending upon company experience it may also be necessary to include an allowance for overtime costs and productivity decreases associated with employee absenteeism. Many estimates in the past have been in error because the estimator overlooked these factors.

Absenteeism n. 旷课, 旷工

Supervision Costs of supervision (line 18) can be estimated by taking a fixed percentage of direct labor costs based on past experience. In the absence of prior data a factor of 15% may be assumed.

Maintenance Maintenance costs (line 19) are usually calculated as a fixed percentage of depreciable capital investment per year. This percentage also can be based on past company records or, alternately, a factor of 6% may be used. Of this 6%, 2.4% represents maintenance labor and 3.6% represents maintenance materials.

Factory supplies Factory supplies (line 20), those miscellaneous items such as lubricating oil, instrument charts, wiping cloths, etc., can be estimated at 0.5% of the depreciable capital investment per year.

Indirect overhead Indirect overhead costs (line 21) include a prorata share of the cost of administration and support services (e.g., purchasing, receiving, accounting, clerical, etc.). These costs generally exceed 50-60% of direct labor plus supervision plus maintenance labor and are quite variable depending on the size of the company and on corporate policy. The figure chosen thus should be based on past company records using care to avoid counting items considered elsewhere in the estimate.

Prorata n 按比例分配, 摊派

Clerical adj. 牧师的, 书记的 n. 牧师

Payroll Overhead Payroll overhead (line 22) can be estimated at 30-50% of direct labor plus supervision plus indirect overhead, depending upon the cost of employee fringe benefits to the company and union labor agreements. If mining systems are included in the estimates, the United Mine Workers of America (UMWA) royalty must be added to this figure.

fringe benefits 附加福利

Laboratory costs Approximately 5% of direct labor can be allowed for this item (line 23).

Contingencies It is difficult to be certain that all cost items have been included in any given estimate. Hence it is wise to include an allowance for contingencies (line 24). It is suggested that the following amounts, expressed as a percentage of the direct cost items described previously plus raw materials and miscellaneous debits and credits, be included:

"Installations similar to those currently used by the company,	
for which standard costs are available	1%
Installations common to the industry, for which reliable data	
are available	2%
Novel installations that have been completely developed and	
tested	3%
Novel installations that are in the development stage	5%"

Indirect Expense

Depreciation Depreciation allowances (line 26) are based on tax laws and vary from industry to industry

depending on many factors. For coal preparation plants an annual allowance of 6.67 % of depreciable capital investment can be used. This factor assumes straight line depreciation over a 15-year period.

Real estate taxes and insurance For a coal preparation plant an allowance of 2% of capital investment should be adequate per year (line 27). In addition, a credit should be taken for any investment tax credit allowable under current law, if applicable.

Depletion allowances Depletion allowances (line 28) generally will not apply to a coal preparation plant. If, however, the estimate also includes mining cost, this factor should be included at the current allowable rate. Depletion allowances are tax credits granted by law to compensate for eventual exhaustion of a natural resource such as coal. They are computed as a fixed percentage of the market value of the resource in the first usable, salable form into which it is processed.

Amortization Again, this factor (line 29) does not normally apply to coal preparation plants. It is applicable to any intangible asset, such as a patent license, that has a limited life and should be included if relevant.

Packing and Shipping Expense

Because coal is generally loaded and shipped in bulk quantities and is sold f.o.b. the preparation plant, shipping costs (line 33) are absorbed by the purchaser and this item is of little consequence to preparation costs. However, if these costs are paid by the preparation plant they should be included in the estimate.

Total Production Cost

To determine the total production cost, the various cost items described (and tabulated in Fig. 5-7, lines 1-33) must be summed and compared. As indicated in Fig. 5-7, net raw materials cost (line 10) equals the gross raw materials cost (line 5) plus any raw materials debits (or minus any credits) (line 9). Total direct conversion cost (line 25) equals the sum of all direct items of expense (lines 11-24). Similarly, total indirect conversion cost (line 30) equals the sum of the indirect items of expense (lines 26-29). Total conversion cost (line 31) equals the sum of the direct and indirect total costs. Total operating cost (line 32) equals the total conversion cost plus the net raw materials cost. The final total production cost, f.o.b. the preparation plant (line 34), is then equal to the total operating cost plus the packing and shipping expense (line 33), if any.

Profitability

After completing this estimate, the total projected net plant income should be calculated and compared to the total manufacturing cost. Gross income can be estimated by multiplying the anticipated annual sales of each coal product by the sales price and summing these figures. Net annual income is equal to gross income after deduction for sales and distribution expenses. Potential profitability can then be estimated as follows:

- 1) Gross annual profit (profit before taxes) = gross income less total production cost.
- 2) Net annual profit (profit after taxes) = gross annual profit less federal and state or local income taxes (generally about 51%).
- 3) Percent return on sales = (net annual profit - gross income) x 100%.
- 4) Plant payout time -- (total capital investment less working capital) / (gross annual income plus depreciation).
- 5) Net annual return on investment = (net annual profit) - (total capital investment).

Obviously it is most desirable for plant payout time to be as low as possible with the other factors as high as possible. As a rough rule-of-thumb, a potential investment is worthy of consideration if the plant pay out time is two to three years or less. It should be carefully noted that this analysis assumes the plant will be operated at its full design capacity from the time it is first put into operation--that is, capital will be put to its best use. However, this is frequently not the case and the estimate must be adjusted accordingly to obtain a true

picture of potential profitability. This correction is made by adjusting the figures for net income by the reduced production rates that may be anticipated in the first years of operation or for reduced production rates due to operations on other than an around-the-clock basis.

Comparing Alternatives

Frequently, several potential preparation plant capacities or sites are under consideration and several possible washed coal products. In such instances, a detailed cost estimate should be made based on each alternative to determine the most profitable choice. Although one alternative may appear very attractive, other less obvious alternatives will frequently be the most beneficial in the long run. As is discussed earlier in this chapter, such decisions are normally made on the basis of discounted cash flow (DCF) analysis. Thus, for example, if the minimum acceptable DCF rate of return is 15%, any alternative with a lower DCF will normally be rejected. Similarly, for those alternatives with a DCF in excess of 15%, the alternative or combination of alternatives yielding the highest overall return normally will be selected.

The DCF comparison is made by comparing the various investments with the cash flows generated by the investment over the life of the project.

In mathematical form the DCF rate of return, r , is defined by

$$P = \sum_{i=0}^n \frac{CF_i}{(1+r)^i}$$

where P , the venture worth = 0; n is project life in years; and CF_i is cash flow at the end of the i th year.

In this equation investments are represented by negative cash flow figures at the end of the year in which the investment is made; the initial investment is assumed to be made at the end of year 0--i.e., before startup.

The cash flow figures for each year are simply the net income after taxes for the year in question plus any depreciation and depletion allowances claimed during the year minus any additional investments made during the year.

Thus, for example, the foregoing equation for a project life of three years becomes:

$$P = -I + \frac{CF_1}{1+r} + \frac{CF_2}{(1+r)^2} + \frac{CF_3}{(1+r)^3}$$

where I equals the initial investment.

Once the cash flows are estimated for each year it becomes a simple matter to substitute them into the equation and then, by trial-and-error and interpolation, determine the value of r at which P equals 0.

At this point r is equal to the discounted cash flow rate of return.

CONCLUSION

In view of the increased value of coal, it is imperative that coal preparation be analyzed, taking into consideration the economic parameters outlined in this chapter. Optimum plant circuits must be designed based upon sound cost engineering judgment in order to maximize the financial return in light of current and projected market values for both metallurgical and steam coals.

Reading Material Evaluation and Prediction of Optimum Results

INTRODUCTION

Washability characteristics derived from float-and-sink analysis in Chapter 4 demonstrate that coal is intrinsically heterogeneous, and this heterogeneity is also evident in the proximate, ultimate, and size analyses. A range of particles from high to low specific gravity is usually seen in the run-of-mine coal. One objective of coal preparation is to recover the high quality float material (the particles of lower specific gravity) such that the recovered product meets certain quality specifications, as well as to reject low quality particles of higher specific gravity. The maximum recovery of cleaned coal at the required quality is of major concern to plant operators. However, the quantity and quality of products of coal cleaning, determined in the operation of actual commercial cleaning plants and by theoretical calculation based on run-of-mine washability studies, are known to be different with each coal. Therefore, during the last half century, coal preparation engineers have repeatedly sought methods to predict the technical and economical efficiency of separation of cleaning equipment and systems. Many formulas have been proposed by international investigators to evaluate the efficiency of various coal cleaning operations, primarily based on the yield and/or ash of the various products of coal cleaning. Performance criteria in use today are capable of defining all of the important aspects of a given concentrating operation; however each criterion has certain restrictions or limitations. It is important, therefore, to understand how such formulas can be used and the problems involved in their construction.

Technical Measure of Plant Performance

Conventional means for evaluating a preparation plant's performance are generally approached through the use of equipment performance measures and plant performance measures. Measures involving separation efficiencies are influenced by the following three broadly encompassing primary factors: (1) density composition of the feed coal, (2) design characteristics of the cleaning unit, and (3) specific gravity of the separation. Since separation efficiencies are dependent criteria, it means that the evaluation is dependent on factors involving both the nature of the coal as well as the design of the equipment; and, as such, dependent evaluation represents a very broad technical indicator of an individual process or plant's performance. Such evaluators as recovery efficiency, ash error, and total correctly placed material, in both quantitative and qualitative forms, are used to implement this evaluation. Separation efficiencies or dependent criteria are used to evaluate different equipment for application to different reserves. Equipment performance measures known as independent criteria are influenced mainly by quantitative errors inherent in the design and operation of the processing equipment. Such criteria as the "Ecart Probable Moyen," the "Imperfection," and the "Sharpness Index" are used to implement this evaluation. Hence, sharpness of separation or other independent criteria may be used to evaluate different equipment for application to a specific coal reserve.

Economic Measure of Plant Performance

True assessments of coal cleaning methods require reference to economic measures such as profit or profitability. Pricing generally is determined by quantity, quality, amount of extraneous contamination, and production losses. During the years in which methods, such as collecting technical efficiency criteria for characterizing plant performance, were being improved, coal preparation engineers were also seeking appropriate economic criteria to assess plant performance.

Technical and economic effectiveness of plant performance are important parameters for coal preparation flowsheet design. The relationship of the technical and economic parameters can be formulated in an economic and mathematical model with the aid of computers. The computerized models not only can simulate plant performance but also forecast optimum cleaning results. Coal preparation flowsheet alternatives need to be examined continuously, and a substantial reduction in labor and time can be achieved through the use of computerized models for these repetitive computations.

To select a best flowsheet among alternatives is an ultimate goal, but it is not a simple procedure. A profit function to be incorporated in an economic and mathematical model must be developed which has a relationship to recovery of clean coal, sale price, and production cost. Coal preparation engineers can approach an optimum flowsheet through the use of simulation techniques, optimum seeking methods, and a cost engineering point estimate or probabilistic method to approach profit maximization or yield optimization, subject to quality and safety constraints.

CHARACTERISTICS OF COAL CONCENTRATION PROCESSES

Coal concentration may be viewed as two subdivided processes involving liberation and coal cleaning. In the liberation process the bonds between coal and impurities are fractured, and in the separation process the resulting liberated particles are sorted in water, dense-media, or other chemical media into two or more products. One is a float product that is rich in coal and the other is a reject that is lean in coal. For example, float and sink data, contrasting a perfect and imperfect separation, are shown in Fig. 18-1. Separation processes of this type, which are based on differences in specific gravity, constitute the main processes of coal preparation. The theoretically attainable separation is presented in Fig. 18-1 (A-1). By contrast, in any actual coal cleaning processes, the separation may be incomplete. This is illustrated in Fig. 18-1 (B-1). A proportion of the total feed coal consisting of the misdirected particles is divided randomly between the products being separated. The phenomenon can be better understood by referring to the individual graphs. In Fig. 18-1 (A-2) and (B-2) the Tromp distribution curve for perfect and imperfect sharpness of separation is shown. In Fig. 18-1 (A-3) and (B-3) the probability density functions of feed coal, clean coal $f(g)$, and refuse $s(g)$ are shown. In Fig. 18-1 (B-3) the "triangles of error" (these areas depict the misplaced material in the clean coal and refuse), and "error distance" are shown. The triangles of error and error distance both provide a quantitative measure of the deviation from a perfect separation. They are indicators of the effectiveness of coal cleaning operations. In Fig. 18-1 (A-4) and (B-4) two distribution functions df/dg and ds/dg for clean coal and refuse are presented; they provide an easier visualization of the inherent difficulty of separation. The overlap area of the functions (error area) in Fig. 18-1 (B-4) reflects the fact that the distribution of misplaced material in clean coal and refuse is due to the inherent specific gravity composition of feed coal and also the imperfect separation of the process. Various technical criteria, proposed in the literature over the last 60 years, for evaluating the performance of coal cleaning units and operations are shown later in this chapter in Tables 18-8 and 18-9. The mathematical expressions are written in terms of the standardized symbols which are defined in Table 18-7.

EQUIPMENT PERFORMANCE MEASURES

(Equipment Efficiencies, Sharpness of Separation, Independent Criteria)

Often the desired criterion of equipment performance is one which reflects the inherent ability of the cleaning unit to make a sharp separation between coal and impurities, and this criterion must be as free as possible from the influence of the effects of coals of dissimilar character and separations at different densities. It assists in predicting the actual clean coal yields and ash and sulfur contents of a cleaning operation from standard

float-and-sink washability analysis. Hence, such criteria are useful in formulating performance guarantees for new plants and in calculating the quantity and quality of products obtainable by treating a new coal in an existing unit of known performance characteristics.

Methods of evaluating performance that are substantially independent of the density composition of the feed coal will be better understood and used more intelligently if their derivation is clear. In commercial coal cleaning, coal of density well below the density of separation and impurities of density well above the density of separation report largely (or entirely) to their proper products, clean coal and refuse, respectively. As coal approaches the density of separation, however, more and more material tends to report to an improper product. Finally, an infinitesimal increment of coal, at the density of separation, is divided equally between the clean coal and refuse. The density of this increment can logically be defined as the density of separation. The curve in Fig. 18-2 has the shape of a Gaussian error distribution curve, and a mathematical analysis of gravity separation demonstrates that it is indeed governed by the laws of probability. Specifically, it is based on the random distribution laws and is calculated from the standard deviation.

The inner curve in Fig. 18-2 represents the type of sharp separation that can be obtained in good dense-medium operations. Both limbs of the curve reach the specific-gravity axis, and the range in density between complete recovery of coal and complete rejection of impurity is quite narrow. The outer curve is characteristic of a good jig operation. The right-hand limb does not reach the axis, and the spread between the specific gravity of separation and the specific gravity at which there was complete recovery of coal is rather large.

The Tromp Distribution Curve

A bar chart showing the percentage of each density fraction of the feed coal that was recovered in the clean coal was devised in 1912 by Hancock. In 1938 Tromp observed that, if these percentage recoveries are plotted against the mean densities of the density fractions, the type of curve illustrated in Fig. 18-3 is obtained. By curious coincidence the same curve was developed independently by Fraser and Yancey for a publication that appeared in 1938. The curve has borne many names, such as, appropriately, Tromp curve, error curve, partition curve, distribution curve, and recovery curve. There still is no uniform terminology. In Europe the term partition curve is widely used, while distribution curve appears to be favored in the United States. The International Standards Organization (ISO) recognizes both terms. Note that if the left-hand limb of the curve in Fig. 18-2 is rotated 180° about the 50% ordinate, its shape becomes that of the distribution curve in Fig. 18-3.

Derivation of the Tromp Distribution Curve To determine the distribution curve for a particular cleaning operation, three sets of coal analytical data are required: (1) recovery of clean coal (Y_c), (2) float-and-sink washability analysis of the clean coal product, and (3) float-and-sink washability analysis of the refuse.

The recovery of clean coal (Y_c) can be determined either by direct weight, or it can be predicted by using material balance equations, if the average ash of feed coal, clean coal, and refuse are known.

A model for a simple cleaning system is shown in Fig. 18-4.

A mathematical representation of the mass balance in this cleaning system can be written as follows:

$$A_c * Y_c + A_r * Y_r = A_f * Y_f \quad (1)$$

$$Y_c + Y_r = Y_f = 100 \quad (2)$$

where A_f is average ash of feed coal, %; Y_f is recovery of feed coal, % = 100; A_c is average ash of clean coal product, %; Y_c is recovery of clean coal product, %; A_r is average ash of refuse, %; and Y_r is recovery of refuse, %.

This set of equations which includes ash balance (Eq. 1) and yield balance (Eq. 2) can be solved simultaneously. By multiplying the yield balance (Eq. 2) by $(-A_r)$, and adding this equation to the ash balance (Eq. 1), and rearranging the result, the recovery of clean coal, Y_c , is determined as follows:

$$Y_c \% = \frac{(A_f - A_r)}{(A_c - A_r)} 100$$

The result may be used for preparing the Tromp distribution curve calculation. The general procedure for preparing the Tromp distribution curve is summarized in Table 18-1.

Samples of the feed coal and products were previously analyzed to obtain the individual ash content. In this example, the ash content in the clean coal product is 7.0%, the refuse 60.20%, and the feed 17.20%. By modeling a simple cleaning system and employing material balances (Eqs. 1 and 2), the recovery of clean coal was found to be $Y_c = (17.20 - 60.20) / (7.0 - 60.20) = 0.8083 = 80.83\%$, as shown in Table 18-1. The difference between the average specific gravity of the fraction and the specific gravity of separation (dso) are obtained and shown in column J.

Strictly speaking, the distribution data should be plotted against the mean specific gravity of the fraction—the specific gravity at which half of the fraction would float and half would sink. In practice, however, they are plotted against the midpoint of the specific gravity range of the fraction. Assumptions are required in plotting the lightest and heaviest fractions because they have no exact limiting specific gravities. If 1.30 is the lowest specific gravity used in the analysis, as frequently is the case, the point for the float should be plotted at a specific gravity that is midway between that of the lightest particle present and 1.30. A figure of 1.26-1.28 generally is used. Any error involved in making this assumption generally has very little influence on the shape and position of the curve; it becomes important only when the specific gravity of separation is unusually low. If 1.80 is the highest specific gravity in the analysis the sink is usually plotted at 2.20-2.30, depending on what is known about its composition. If the highest specific gravity is 1.60 the proper position of the point must be lowered accordingly. An error made in selecting the proper specific gravity at which to plot the sink may sometimes have a significant influence on the shape of the curve.

The general principle in choosing a specific gravity interval for washability testing is that the narrower the range employed, the better. In practice, 0.05 is frequently used. However, it is recommended that in all new float-and-sink washability analyses for the purpose of forecasting cleaning results, consideration should be given to testing at 0.02 intervals. This is particularly important for plants making low ash and low sulfur clean coal products.

If the quantity and quality of clean coal product is the main object of cleaning, the distribution curve may be constructed on a clean coal basis, as given in Fig. 18-3.

The Straight Line Concept of the Tromp Distribution Curve Generally the distribution curve is plotted directly against specific gravity, as shown by the lower scale in Fig. 18-3. In comparing curves having different specific gravities of separation, however, they may be plotted against the difference between the specific gravity of the fraction and that of the separation, as illustrated by the upper abscissa scale in Fig. 18-3. This will be recognized as the abscissa scale used in Fig. 18-2.

In Europe it is common practice to plot the distribution curve on either log probability or arithmetic probability paper in an effort to obtain a straight line. The ordinate employed is always percentage recovery on a probability scale, but the specific gravity abscissa scale varies with the type of cleaning unit involved. For dense-medium cleaning units the abscissa scale is arithmetic. For processes that employ water it is $\log d_{50} - 1$ (specific gravity of separation - 1), and for pneumatic processes it is $\log d_{50}$.

The slope of the line is a measure of the sharpness of the separation, and the slope plus the specific gravity of separation combine to characterize the complete curve. In principle, only two points are required to plot the curve; thus a great deal of costly laboratory work would be eliminated. In practice, distorted tails on Tromp distribution curves for commercial coal cleaning units are frequently observed.

As illustrated by the curves in Fig. 18-5, the middle portion of the curve often does approximate a straight line quite closely (as is often the case when conventional scales are used, for that matter) but the tails usually exhibit curvature. Terra, an early advocate of the straight-line plot, argues that the tails are of limited significance, and imputes the results to possible experimental and sampling errors.

The distorted tails can also possibly be attributed to poorly designed, poorly operated, or overloaded coal cleaning units, or possible sizing problems prior to coal cleaning. In such cases, the mathematical modeling of the Tromp distribution curves should not be restricted to the straight line relationship of the probability scale of ordinates, but should also include the tails. Theoretically, however, the Tromp distribution curves for all well-designed and well-operated commercial coal cleaning units should show a straight line relationship on the probability scale of ordinates. Thus, it may be prudent for coal preparation engineers to eliminate such a distortion of tails through proper design and operation of the coal cleaning units as well as other unit operations, such as the integral parts of the coal cleaning system within the coal preparation plant.

Parameters Derived from Tromp Distribution Curve It is common practice to use parameters which are derived from the distribution curve, instead of using the whole distribution curve for measuring the technical efficiency of a cleaning unit. The following parameters are widely used to describe the characteristics of the Tromp distribution curve:

1) *The Specific Gravity of Separation or Tromp Cut Point's d_{50}* The specific gravity of separation, or Tromp cut point, is normally defined as the specific gravity that is coincident with the 50% distribution factor (d_{50}). In Fig. 18-3, the specific gravity of separation d_{50} is 1.53. The point of separation, d_{50} , is that specific gravity where a particle has an equal chance of appearing in the clean coal or in the refuse. This provides a means of locating the specific gravity of separation of a cleaning unit.

2) *Ecart Probable Moyen, E_{pm}* The concept of the straight-line distribution curve led Terra to propose "Probable Error" or "Ecart Probable Moyen" as a criterion to represent the sharpness with which the coal and impurities are separated. This term, usually abbreviated to E_{pm} (Ecart Probable Moyen), is a measure of the

deviation of the distribution curve from a perfect separation. Numerically, the E_{pm} is arbitrarily chosen at one-half the specific gravity interval spanned by the distribution curve in passing from the 25-75% recovery ordinates. Mathematically, it can be expressed as follows:

$$E_{pm} = d_{25} - d_{50}, \text{ or}$$

$$E_{pm} = (d_{25} - d_{75})/2$$

The latter way of expressing the E_{pm} is more common as it takes into account a nonsymmetrical distribution. Generally, the steeper the distribution curve, the lower the E_{pm} . In other words, a low E_{pm} designates a sharp separation, and a large E_{pm} denotes a separation that is not sharp. The usual range of E_{pm} is from about 0.020 to perhaps 0.30 or more. The example given in Table 18-1 and Fig. 18-3 is used to illustrate the preparation of the distribution curve. The particular cleaning unit used to develop this example yields specific gravities of separation of 1.61 for d_{25} and 1.47 for d_{75} . This results in an E_{pm} value of 0.07. [$E_{pm} = (1.61 - 1.47)/2 = 0.07$].

3) Error Area, A_{er} Since the distribution curve for a perfect separation would show all material lighter than the specific gravity of separation reporting to clean coal and heavier material reporting to refuse, it would be composed of three straight-line segments. One segment would be on the 100% recovery axis; one would comprise the specific gravity of separation abscissa; and the third would lie on the 0% recovery axis. The area lying between this curve for a perfect separation and the distribution curve actually obtained is, like E_{pm} , a measure of the sharpness of the separation.

This area is shown hatched in Fig. 18-3 and is termed A_{er}

Driessen proposed standardizing the error area by uniformly plotting the distribution curve to a scale on which a unit of length that represents 2% on the ordinate or weight scale will represent 0.1 on the abscissa or specific gravity scale. A_{er} , it should be noted, is dimensionless. A_{er} ranges from lows of around 10 for a good separation to over 100 for an inferior separation.

4) Imperfection, I Imperfection involves the influence of the specific gravity of separation on the shape of the distribution curves. When first developed, the distribution curve was believed to be independent of the density of separation. As more experience with the curve was accumulated, however, it became apparent that, other factors being equal, there is a tendency for the curve to steepen as the density of separation decreases. In other words, separations at low density tend to be sharper than those at high density. This concept was difficult for some to accept because it appeared to contradict the well-established principle that efficiency increases with increase in the density of separation.

Apparently the French research organization, Cerchar, was the first to recognize the relationship between sharpness of separation and density of separation. In an effort to develop a criterion for sharpness of separation that would be independent of the density of separation, they originated the term "Imperfection," for which the symbol is I . The formula for the Imperfection is:

$$I = E_{pm} / (d_{50} - 1) \quad (\text{Jig Cleaning Unit})$$

where I is Imperfection; E_{pm} is Ecart Probable Moyen, or probable error; and d_{50} is specific gravity of separation.

Cerchar contends that d_{50} increases in proportion to $d_{50} - 1$, and that 1 therefore is a constant for a particular

cleaning device. They originally proposed the use of I only for jigs. More recently others have used I formulas with other water cleaning process units such as tables and water-only cyclones. For other types of cleaning processes such as dense-medium cleaning, Aso suggested using d_{50} as the denominator in the I formula.

$$I = E_{pm} / d_{50} \quad (\text{Dense-Medium Cleaning Unit})$$

The example given in the Table 18-1 and Fig. 18-3 for the Tromp distribution calculation is used to illustrate how the value of the I of a proposed cleaning unit is calculated. Suppose that the proposed cleaning unit is a dense-medium device and the I can be derived from E_{pm} and d_{50} . In this example, the result of the calculation is given as $I = E_{pm} / d_{50} = (0.07)/(1.53) = 0.046$.

With most sets of performance data for the same cleaning unit, or for units of the same type operating under comparable conditions, the Imperfection varies over about as great a range as does the Ecart Probable Moyen or error area. Sometimes the range of the I values is even greater. Nevertheless, it is a matter of general observation that separations at low density generally are sharper than those at high density. However, with dense-medium units the change in sharpness of separation with change in density of separation is so small that it often can be neglected. Tests made within 0.2 sp gr units can be considered directly comparable. With the dense-medium cyclone the effect of separating density is more pronounced. The determination of E_{pm} , A_{er} and I vary with the specific gravity of separation d_{50} . Such a determination was conducted by the US Bureau of Mines (USBM) in a dense-medium cyclone pilot plant operation under carefully controlled conditions.

Applications and Limitations on the Sharpness of Separation Criteria The inherent clumsiness and inconvenience of using a curve instead of a number to characterize a gravity separation imparts importance to measures like Ecart Probable Moyen, E_{pm} , Error Area, A_{er} , and Imperfection, I . These measures of separation are defined as sharpness of separation criteria.

It is important, therefore, to examine these criteria and establish both their usefulness and their limitations.

Simplified Criteria A_{er} has the distinct disadvantage of requiring the measurement or calculation of the area under a curve. A large error area for a separation can result from poor recovery of the clean coal, poor elimination of impurity, poor performance in the midgravity range, or any combination of these difficulties. Tromp suggested that part of this ambiguity could be eliminated by specifying two error areas—one for the portion of the curve to the left of the specific gravity of separation and one for the right-hand segment. Although this suggestion has merit, it has never been followed to any extent. Many instances can be found in the literature where a faulty separation is indicated by the error area but not by the Ecart Probable Moyen. Nevertheless, there is a general relationship between these two criteria. Generally, when recovery of the lightest coal is complete and there is nearly complete elimination of the impurity (the conditions that provide a low A_{er}), the middle section of the curve is fairly steep and therefore the probable error also is low.

E_{pm} appears to have gained in popularity over A_{er} in recent years. This is understandable because of the ease with which it can be determined from the distribution curve. However, the fact that E_{pm} is determined entirely by the middle section of the curve makes it less reliable than A_{er} . The curves shown in Fig. 18-6, which represent three performance tests of dense-medium washers, have substantially the same E_{pm} of 0.052, 0.053, and 0.059, but they represent entirely different types of separation. The poor recovery of light coal, shown by the curve for the Roundup Seam, fails to influence the value of the E_{pm} . The error areas for these three Curves are quite different: Surmi, 40; Pittsburgh, 25; and Roundup, 57.

Neither A_{er} nor E_{pm} adequately portrays the entire distribution curve; the entire curve is important. Therefore, blind reliance on the sharpness of separation criteria instead of consideration of the entire curve can be misleading.

Importance of Near-Gravity Material The amount of material within ± 0.10 sp gr unit of the specific gravity of separation has long been used as an indication of the difficulty of a separation. The greater the percentage (by weight) of this material, at the specific gravity of separation (d_{50}), the more difficult will be the separation. When preparing washability characteristic curves, the ± 0.10 sp gr distribution curve is frequently constructed to determine that point of specific gravity of separation where it is practical to operate the cleaning unit. The method of calculation is detailed in Chapter 4.

Part 9 Supplementary Reading

Paper1

CLEAN COAL TECHNOLOGY OPPORTUNITIES WITHIN THE INDUSTRIAL SECTOR OF CHINA

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ABSTRACT

“Clean Coal Technology” is a term used to describe various coal utilisation systems which can show efficiency advantages and improved environmental performance compared to conventional mature systems. While commonly applied to power generation plant, it is also applicable to the significant advances made in technology for the domestic and industrial sectors.

China is an industrialising country in transition from a centralised to a market economy. The industrial sector is, at present, the major coal using sector in China where coal is and will remain the prime fuel and a major feedstock. Increasing industrialisation means, for example, that subsectors such as steel, construction materials and chemicals production are growing rapidly. This has to be underpinned by increased coal production and utilisation. However, the need to improve coal utilisation capacity significantly has to be balanced by the need to ensure that there are acceptable environmental options. Thus there is a need to determine the optimum strategy for technology selection and transfer in terms of cost, efficiency and environmental impact. The options include:

- introduction of new technology both to ensure better coal utilisation and acceptable control of pollutants
- upgrading and improving existing technology allied to extensive training programmes
- enhanced capacity building initiatives to ensure sustainability
- improved definition of research and development needs together with a focused implementation of such programmes to meet industrial requirements.

In all cases there will be a need to attract financial support from a number of sources. Consequently there is a need to fully assess and evaluate the investment requirements and opportunities. This reflects the need for the industries within the various subsectors to make the transition towards operating within an international market economy.

This paper outlines and considers these various issues associated with such technology opportunities in China. In order for R&D to have maximum commercial impact, it is initially necessary to look at the steps involved, illustrated below for a power generation scenario based on exported Australian coal.

R&D targeted at cost savings should focus on mining and transport (from the producers perspective), and on the last three elements of the chain (from the end users perspective). This highlights the need for a systems approach integrating geological (deposit evaluation) mining and preparation functions - the "pit to port" concept, and extending this through coal quality, impact assessments and the design of combustion and emission control systems - the "pit to market" concept. Coals will eventually move from being undifferentiated commodities to specific purpose end-users with full awareness of technical, economic, and environmental performance.

1. INTRODUCTION

China is the major coal producer and user in the world. Proven national reserves were recorded at 1,006 billion tonnes in 1994, representing over 95% of the total fossil fuel reserves in China. In 1995 the raw coal production was 1.3 billion tonnes with planned increases in output to 1.4 billion tonnes by 2000 and 1.8 billion tonnes in 2010. In the longer term, the indicative production level for 2050 has been set at 4 billion tonnes.

Coal is the main energy resource in China. Some 72% of industrial and steam fuel, 52% of chemical feedstock raw material and 92% of domestic fuel requirements are met by coal. Table 1 provides a breakdown of coal consumption in China.

Table 1 The Structure of Coal Consumption in China

Use	Percentage of total consumption%(1995)
Power generation	31
Industrial and commercial boilers	30
Metallurgical use	8
Industrial kilns	3
Domestic use	20
Export and others	8

China is an industrialising country and, as has been indicated, energy usage will continue to grow at a fast rate. However, such use will result in a significant worsening of the environmental situation unless compensating steps are taken. National statistics indicate that in 1994 some 28 million tons of dust were discharged into the atmosphere throughout the country, of which 22 million tons arose from coal combustion, (namely 79% of the total). About 15 million tons of SO₂ and 4 million tons of NO_x were also emitted into the air, with some 89% of SO₂ and 67% of NO_x originating from coal combustion. Thus there is a need for China to consider very carefully how best to meet its needs for increased coal utilisation in terms of cost, efficiency and environmental impact.

2. PRIORITY NEEDS TO IMPROVE ENERGY EFFICIENCY AND ENVIRONMENTAL PERFORMANCE

A number of points are listed below which need consideration in any attempt to introduce general energy efficiency and environmental improvements for China.

* One of the key needs is price reform. Where energy prices have been held at artificially low levels there is no real incentive to improve end-use efficiency.

Power generation may be the dominant sector where major growth is expected and where large efficiency gains can be made. However, this will be a slow process because of the large investments and long lead times. A balanced policy will also encompass significant improvements in coal use in domestic, industrial and coking markets, these remaining significant and major sectors for the foreseeable future.

Coal preparation and washing is an enabling technology without which the benefits available in efficiency improvements cannot be fully realised, especially in the domestic and industrial markets. Apart from preparation for coke production, most coals for other markets are not prepared. In 1995, 280 million tonnes of raw coal were washed and according to the 9th Five Year Plan of the Ministry of Coal Industry it is intended that this will increase to 419 million tonnes in 2000 and 720 million tonnes in 2010. Washing the coal controls the ash content and to some degree the sulphur content. This has significant benefits in terms of subsequent improved utilisation of the product, reduced transport costs and reduced atmospheric pollution. At the same time, coal washing can also produce a slime of very fine suspended solids in the waste water. This material represents a major environmental hazard as well as an energy loss. There is also the issue of price. Washed coal is more expensive than raw coal. In overall terms its use should be cost effective. However, most industrial companies are reluctant to even evaluate the options, preferring to purchase their coal solely on the basis of fuel price.

Optimum investment strategies are likely to indicate that high priority should be given to upgrading and improving what is already there, allied to training programmes. This can include improving operations and/or retrofitting either a bolt-on clean-up technology to, say, an industrial process or introducing an alternative improved technology with integral pollution control.

Technologies which can be retrofitted to existing industrial plants include low-NO_x burners, flue gas desulphurisation and particle control systems. Whilst reduction of pollution from existing plant is very important, the systems used widely in the West are capital intensive and may not be the most appropriate for China where capital resources are limited. In the longer term, for, say, the power generation sector it may be better to introduce advanced technology in which the higher efficiency cycle reduces the polluting potential of the power station in addition to reducing the demand on fuel resources. Indeed in China there is considerable technical interest in advanced systems such as pressurised fluidised bed combustion (PFBC), integrated gasification combined cycle (IGCC) and Topping Cycles. However, a major technology actually being introduced for large industrial and some utility applications is circulating fluidised bed combustion (CFBC). This technology is attractive as it can handle low grade coal, is relatively low capital cost and can be constructed relatively quickly. There are two options for both utility and industrial applications, a new purpose built plant or a retrofit to an older plant. The former approach can be tailored to exactly meet the local requirements but requires a greater capital investment and a longer build time prior to operation. On the other hand, the retrofit route of replacing, say, the combustion process and boiler of an old unit with new CFBCs is much cheaper than a new build. It offers a cost effective way of replacing an old unit with lower capital outlay, where, for example, coal supply systems, buildings, components such as the steam turbine cycle, can all continue to be used provided reasonable maintenance and upgrading has been practiced.

3. INDUSTRIAL SECTOR TECHNICAL OVERVIEW

In China the steel, chemicals and building materials subsectors are the major coal using subsectors, predominantly through gasification to produce a gas either for heating purposes or as a chemical raw material.

For example, China is the world's second largest producer of steel, making 94 million tonnes in 1995. Production is expected to rise to 140 million tonnes per year by 2010, as the economy expands. A large proportion of iron and steel production plant is old, relatively small in scale and there is a low degree of integration. Specific energy consumption in China (1993) was 1.00 tonnes of coal equivalent per tonne of steel (tce/ts) compared with 0.61-0.89 tce/ts in advanced steel companies

overseas. The iron and steel industry is one of the largest sources of industrial pollution in China. In 1993 about 1.8 million tonnes of smoke, dust and SO₂ were released to air and about 2.8 billion tonnes of wastewater discharged. About 77% of these emissions met standards. Plans call for an increase in compliance to 90% by 2010.

Coal gasifiers are a major source of fuel gas. Most of these gasifiers were built in the 1950s and 1960s. An important item in the 9th Five Year Energy Plan is the intention to upgrade 200 coal gasifiers producing fuel gas, for use mainly at these special steel plants. The predominant type of gasifier is a former Russian design of fixed bed gasifier, the AR-13. These have a coal gasification efficiency of some 60% and severe environmental problems. Hence, the Ministry of Metallurgical Industries and the enterprises are committed to replacing them with modern two stage fixed bed designs with an efficiency of 75-80% and a better quality gas.

The drive would be energy conservation and improved process performance, i.e.

increased gasification efficiency;

improved control leading to more efficient gas use;

increased calorific value of the gas leading to more efficient use;

improved coal preparation leading to smoother operation;

better energy efficient components - variable speed drives, high efficiency motors, etc.

In addition, there is a need to introduce improved waste water/tar treatment plants. At present the procedures used would not be acceptable in western countries and would not meet the standards required by the aid agencies (see below).

The Chemical Industry in China is highly diverse with 20 different subsectors producing some 30,000 different products. The industry is a major energy consumer, accounting for around 9% of the total national energy consumption, of which some 60% of that energy is provided by coal. National policy goals within the 9th Five Year Plan include increasing agricultural chemicals production and developing the pharmaceuticals and fine chemicals industries. It is also planned to introduce more advanced technology, introduce new facilities for production of high added value products and concentrate production on chemical products that meet the requirements of the market and can therefore be readily sold.

There are a number of significant differences between the Chinese chemical industry and the industries of other developed countries. In China, coal is used as the prime energy source (both as a fuel and a feedstock) whereas elsewhere (apart from South Africa) it is gas or oil. Production is generally located throughout the country as predominantly medium to small scale plant unlike elsewhere where production is generally centralised and large scale. Thus nearly every region has a small/medium sized fertiliser plant located near to the agricultural demand. The use of smaller scale production plant means that energy efficiency in the industry is low.

The Chemical Industry is a major polluter producing some 40 million tonnes of waste solids per year as well as 700 million m³ of gaseous wastes and 7 billion m³ of liquid waste. The chemical fertiliser industry is one of the largest individual sources of pollution in the Chemical Industry. As with other industries, the national environmental regulations and policies apply throughout the Chemical Industry, although there are a number of regulations with specific reference to particular

manufacturing processes. During the 9th Five Year Plan, the principle of "emphasising pollution prevention in conjunction with pollutant abatement techniques" will continue to be applied. The aim is to strengthen environmental management at factories and promote pollution prevention and control. In addition, environmental protection will be emphasised as an important link in the restructuring of the chemical industry. Additional efforts will be made in order to limit the generation of pollution as the industry expands, and promote more environmentally acceptable production techniques. The aim is to improve the integrated utilisation of resources and control of the three wastes, to reduce effectively pollutant generation and emissions to various environmental media, so that the serious pollution problems associated with the Chemical Industry can be progressively eliminated.

With regard to clean coal technology options for the Chemical Industry, the chemical fertilizer subsector provides an interesting example. It is one of the largest individual sources of pollution and the main consumer of coal. The chemical fertiliser industry is particularly important because of its impact on food production and there is a strategic need to increase food production per unit area of land in China to meet increased demand. Thus the objective for the nitrogen fertilizer chemical industry is to increase output of fertilizer, without increasing pollution levels, leading to a 30% reduction of ammonia nitrogen discharges by the year 2000. For newly constructed chemical plant, the objective is to minimize waste production at source. For Chinese technology, total emissions of pollutants must attain the level achievable by the best domestic equipment. For new processes based on imported technology, the emissions standards should attain the advanced levels of the industrialized nations.

The technology generally used to convert the coal into gas for chemical synthesis is based on a design imported in the 1950s. This design is not only antiquated but inefficient and highly polluting. The coal utilisation efficiency (energy basis) is about 60% compared to modern western gasification technology with a conversion efficiency of >85%. Since it is based on a cyclical operation some 50% of the gas produced is vented to the atmosphere because it is of too low a quality for chemical upgrading. This vent gas represents a

significant release of CO₂ to the atmosphere. The design also restricts the coal that can be used to anthracite and coke, both of which are less freely available than other coals in China. The result is that fertiliser production is constrained and the necessary increased outputs cannot be obtained.

As in the steel industry, the aim is to introduce Western gasification technology. The type of technology will vary with scale of operation. In the fertiliser industry, which is typically termed as medium scale, the preferred route is to introduce atmospheric fluidised bed gasification plant. Such an approach will reduce energy consumption per unit of ammonia production with very significant reduction of gaseous and liquid emissions.

The Building Materials Industry is one of the most important industrial subsectors in the Chinese economy. It covers the manufacture of construction materials, non-metallic mineral products and inorganic materials. These represent over one thousand different products, the principal ones being cement, plate (flat) glass, bricks, clay tiles, lime and ceramics (sanitary ware and ceramic tiles). The rapid expansion of China's economy and infrastructure in recent years has led to a corresponding increase in demand for and output of building materials for use in engineering construction and housing. China is now the largest producer in the world of building materials such as cement and plate glass.

The nature of the processes used in these industries, such as sintering, high temperature firing and drying, means that they are very energy intensive. The total energy consumption in 1993 was 179 mtce, representing some 16% of the total for the industrial sector of China. Out of this total energy consumption, the actual tonnage of coal used was 156 million tonnes. Energy efficiency is in general poor, being some six to twelve times lower than in advanced industrialised countries. Coal is the predominant primary energy source for the Building Materials Industry, and is used in a wide range of individual processes including gasification plant, boilers, furnaces and kilns. There is considerable diversity in the age, size and efficiency of individual facilities.

The State Administration of Building Materials has a policy of improving energy efficiency which includes the aim to decrease specific energy consumption by improvements to technology and institutional procedures. Specific targets have been established for this subsector whereby energy consumption per 10000 yuan of output value will decrease from 14.1 tce in 1990 to 10.5 tce by 2000, representing a year-on-year reduction of some 3 percent in specific energy consumption. In the sanitary ware industry, there is a further target to reduce specific energy consumption from 1.81 tce/tonne in 1990 to 1.50 tce/tonne in 2000, which represents an annual improvement of some 2% in energy efficiency. There is also the intention to adjust the structure of the sector by increasing the proportion of output which arises from large and medium scale enterprises while simultaneously upgrading and modernising small enterprises.

As in the other subsectors, the proposed approach is to introduce new gasifier plant (based in this instance on two stage fixed bed technology) together with new gas treatment plants to deal with waste tar and contaminated water. Besides the supply side energy efficiency improvements, there are direct process benefits through more stable and better quality gas production.

In all subsectors coal is fired in small industrial boilers to raise steam for the process applications and for heating purposes. There are over 450,000 such industrial boilers, most of them stoker fired with thermal efficiencies of 50-60%. These low efficiencies are a result of an almost total lack of coal preparation prior to combustion, inadequate boiler operating procedures and little or no economic driving force to introduce energy saving measures. There are various initiatives, attracting Aid Agency support, to both introduce improved boiler packages with improved combustion systems, and to build up Chinese capabilities to ensure better, more effective use of the existing units.

4. REGULATORY AND INSTITUTIONAL ISSUES

Besides the technical issues, there are a number of barriers that can prevent the implementation of energy

efficiency and environmental improvement measures. Such barriers have a broader impact than on energy efficiency measures alone and can impact on resource allocation between new projects, refurbishment(refurbish vt.重新擦亮[磨光],再刷新; 整修) of existing plant and machinery, energy efficiency measures, and general management of an enterprise. This is because the introduction of energy efficient measures in China must be taken in the context of an economy undergoing fundamental change while moving very rapidly to a market economy. Fewer prices are now controlled. Enterprises tend to be responsible for many output and pricing decisions, with, most importantly, enterprise managers being provided with profit related incentives. In addition, managers of enterprises are now the principal economic agent (in conjunction with the industry bureaux) in determining management practices and alternative investment activities. Thus a manager will decide whether to allocate investment resources to energy efficiency measures, new projects, or refurbishment project based on their respective contribution to the profitability of the factory.

That said, alternative investment activities may not conflict with the overall objectives of improving energy efficiency. Management changes and refurbishment typically yield a very high rate of return. This is due to the unit cost reduction achieved at a low investment cost. A significant factor in that cost reduction is the reduction in the resource cost associated with energy. Thus the role of energy efficiency must be considered as a part of general cost reduction and performance enhancing activities.

There are a number of specific issues to be considered under the following subheadings.

Government Policy, Supervision and Regulation:

Multiplicity of organisations and offices leading to unclear responsibilities and poor communications;

Poor enforcement of environmental regulations and low levels of fees and fines, providing inadequate incentive for the introduction of energy efficiency measures.

Internal Management of Enterprises:

Lack of training on approaches to improve energy efficiency;

Lack of funding for demonstration projects and information transmission

Access to finance for investment:

Limitation of funds (at the provincial level) for viable projects.

Lack of a flexible and commercial banking system.

Lack of financial expertise at the enterprise level.

External Environment and markets:

Energy efficiency projects need to effectively compete for funds with new and refurbishment projects.

5. OPTIONS FOR IMPROVEMENT

Within the growing economy, there are a number of options which need to be pursued to ensure that China can both increase industrial output while addressing the environmental issues associated with coal use.

5.1 Introduction of new Technology

In many cases, (e.g., the chemicals subsector), the way forward needs to be replacement of obsolete equipment (e.g., the gasifiers) with more modern systems equipped with adequate gas clean up systems. Such an approach will not just result in more efficient coal usage and better pollution control it will usually also result in more stable, reliable and efficient operation of the industrial production process. The introduction of new equipment will require major capital investment and in most cases the introduction of western technology. Sometimes the equipment will be imported directly. However, it is of greater benefit to the economy of a receiving country to have local manufacturing capabilities established to produce equipment of Western design.

For such technology transfer to work, both provider and recipient have to benefit. The provider will usually be a private, rather than a government, organisation and must obtain appropriate financial returns (with funding agency aid where appropriate) and must be able to protect intellectual property. The recipient needs to

obtain the most appropriate, sustainable technology and will need contracts that include appropriate training and follow-up service. From the point of view of the country benefiting from the technology transfer, it is important the technologies are both appropriate to the needs and capable of being introduced effectively. In addition, it is essential that the transferred technology is capable of being supported by the indigenous population. This can be achieved by training of the workforce or by the introduction of suitable institutional arrangements.

5.2 Improvements to existing processes and procedures.

While there is an understandable wish to introduce new (western) technology, there is much that can be done to improve both existing processes and the associated operating procedures. Indeed such an approach has the potential to encourage industry to introduce practical, cost effective, affordable and immediately applicable energy efficiency measures.

Indeed the cement/bricks subsectors and small/medium industrial enterprises appear to be promising in that small investments can result in major energy savings. Here there appear to be significant direct supply side savings, through improvements to industrial boilers (these being used in all subsectors) and kilns. There also appear to be further indirect supply side savings through the improvement and further introduction of briquetted fuels. On the demand side, there appears to be significant scope for generic energy saving improvements for most of the industrial manufacturing and process equipment used in all the various industrial subsectors.

In order to be effective, such improvements require a significant adjustment to the current approach to energy utilisation in China. Thus to ensure that industry will adopt such incentives will require a range of measures, to introduce technical improvements while also dismantling various barriers to implementing such improvements on a sustainable basis. The needs are wide ranging and comprise various parts:

- Introduction and demonstration of practical low cost technical solutions with a fast payback.

- Development and enhancement of dissemination routes to ensure adequate appreciation of the improvements throughout China.

- Establishment of improved managerial awareness and capability within industrial enterprises.

- Introduction of more effective procedures for developing and supporting industrial energy efficiency improvements.

- Introduction of extensive capacity building initiatives and training programmes to introduce new ideas and awareness to key staff within industrial enterprises and the associated governmental organisations.

5.3 Capacity Building Initiatives

As noted above, the need is to ensure that sustainable improvements are introduced to China and as such there is a need to encourage a change of attitude to energy efficiency, environmental protection, health and safety at work, and business management within industrial enterprises. There are a wide range of training activities which would be beneficial:

- Application of environmental impact assessment techniques

- Environmental management

- Energy auditing

- Energy efficiency targeting and control

- Introduction to modern air and water pollution monitoring equipment

- Health and safety at work awareness

- Business management training (including financial appraisal techniques) to encourage movement towards greater business autonomy.

5.4 Improved Research and Development Needs

Within the industrial coal utilisation sector of China the role of R&D should be to improve the nominated coal utilisation technologies through higher efficiencies, greater availability, lower capital and operating costs

and reduced environmental impact. The R&D should be of short to medium term duration and would need to attract funds from various sources. In some instances, for example in the steel subsector with its high intensity of energy usage where such development projects should have payback periods of less than two years, then funding should be sought primarily from Chinese sources. However, in most instances the R&D would have medium term horizons and there may be scope to also gain funding from various aid agencies. For example, the UNDP will support R&D if it is structured within an institutional development and capacity building framework.

In such cases, the need to introduce Western partners to fulfil the role of mentors, advisors, reviewers and, in some cases, research co-workers would be regarded as a key requirement. This would ensure that the R&D initiatives would, in themselves, become the form of capacity building necessary for providing a measure of sustainability in establishing Chinese capabilities to improve industrial coal utilisation.

5.5 Improved Business Planning and Awareness

To achieve successful technology transfer generally needs facilitators and/or funders. Facilitators assist the matching of technology sellers to recipients and a key function is the determination of the correct technologies to be transferred. Funding agencies, such as the European Union aid schemes and the UK Government's Know How Fund (KHF) and Department for International Development, play an important role in this aspect by funding technical assistance projects and training programmes and so help developing nations, and nations in transition, in making judgements about their requirements. The funders of the technology transfer are pivotal to the whole process and organisations such as the World Bank and the Asian Development Bank (ADB) perform a vital role providing financing to assist developing nations and those in transition with specific projects.

Multilateral donors such as the ADB and most governments are interested in capacity building, institutional development and overall economic benefit to the nation. Within this framework the Bank's approach to suitable investment projects in China is to provide loans, not grants, for the provision of imported equipment and services. The limited funds that they have available are best directed to projects which will have a relatively large impact and where the private sector can not be expected to find all of the funds.

In all such cases there is a need for the seeker of funds to be fully aware of the Bank's lending criteria and the need for robust comprehensive business plans. While the state owned industrial enterprises are making the transition to operating within an international market economy, there appears to be a need to enhance their capabilities to prepare comprehensive business plans reflecting the need to attract investment from a number of sources.

6. THE WAY FORWARD

In China, coal will be the major source of primary energy and a prime feedstock for the foreseeable future. The likely population growth and relative magnitude of coal reserves in relation to other fossil fuels make it important that the most efficient, environmentally friendly ways of using coal are introduced at the earliest opportunity. The technologies needed are not confined to the power market, there are significant needs in the industrial and domestic sectors as well.

There is a wide range of coal technology and expertise appropriate for technology transfer. Key priorities should be improvements in energy efficiency and environmental clean-up technologies, whether it be by rehabilitation of existing equipment or by the introduction of newer technology. There is a need for the appropriate organisations in China to become familiar with the ranges of technology available through training programmes, study tours and technical assistance projects. There is also a need for the companies or institutions in the industrialised nations providing technology to understand fully the needs of the receiving organisations in China.

ADVANCES IN COAL PREPARATION AND UTILISATION

Neville C Lockhart and John K Wright***

CSIRO, Division of Coal and Energy Technology

The future competitiveness of coal will be maintained and enhanced through minimizing costs, improving product quality, adopting more efficient and environmentally acceptable utilisation processes, implementing maximum practical environmental controls, and widely promoting a clean, high technology image.

Australia is the world's largest coal exporter and a leader in coal mining and preparation R&D and technologies, also in aspects of coal utilisation for power generation, metallurgical applications and other uses. CSIRO is Australia's dominant, and one of the world's largest, R&D organisations and it works in close collaboration with industry, which provides a significant proportion of the funding and implements the outcomes of R&D. CSIRO has a substantial R&D effort embracing coal, gas, and renewables. This paper summarises the R&D in coal preparation, utilisation, environmental impacts, and technology transfer.

Many of the technologies are applicable to Asia and we are looking for closer contact with interested production companies, mining/processing equipment manufacturers, and governments.

1. INTRODUCTION

Australia is the world's largest coal exporter and a leader in coal mining and preparation R&D and technologies, also in aspects of coal utilisation for power generation, metallurgical applications and other users. The Commonwealth and Industrial Research Organisation (CSIRO) is Australia's dominant, and one of the world's largest, R&D bodies. The organisation employs over 7,000 people and is organised in 22 sectors reflecting different areas of the Australian economy and natural resources. It is part-funded by industry, and in the Energy Sector the average industry support is 40%. The Energy sector covers a broad range, with coal the major component.

The future competitiveness of coal will be maintained and enhanced through minimizing costs, improving product quality, adopting more efficient and environmentally acceptable utilisation processes, implementing maximum practical controls, and widely promoting a clean, high technology image. The theme of this conference is Clean Coal Technologies (CCT's)

comprising: (a) Cleaning of coals before end use such as combustion, (b) Conventional combustion, (c) Advanced combustion or other energy conversion processes, and (d) Cleaning of gaseous and particulate emissions after combustion/energy conversion.

Combinations of (b) or (c) with (a) and/or (d) are typical.

This paper summarises CSIRO R&D from these perspectives.

In order for R&D to have maximum commercial impact, it is initially necessary to look at the steps involved, illustrated below for a power generation scenario based on exported Australian coal.

R&D targeted at cost savings should focus on mining and transport (from the producers perspective), and on the last three elements of the chain (from the end users perspective). This highlights the need for a systems approach integrating geological (deposit evaluation) mining and preparation functions - the "pit to port" concept, and extending this through coal quality, impact assessments and the design of combustion and emission control systems - the "pit to market" concept. Coals will eventually move from being undifferentiated commodities to specific purpose end-users with full awareness of technical, economic, and environmental performance.

CSIRO's approach involves breaking down each element of the chain (above) into all unit operations involved (for example, preparation involves crushing, sizing, splitting, beneficiation of coarse and fine streams,

dewatering of products and tailings and process control. The leverage points and technology issues/constraints are thereby identified. One can then target improvements to existing, and development of new, procedures, technologies and products.

3. COAL PREPARATION

As-mined coal is of variable quality and rarely meets the desired physical and chemical specifications of end-user industries - power generation, steel-making, cement manufacture and in some countries industrial/commercial/domestic uses. Coal preparation is thus necessary to meet quality standards and product consistency, and is an important element of clean coal technologies. Nearly all Australian export coal is washed, and Australia is a world leader in coal preparation. Continued technological advancement is a key element of an internationally competitive coal industry.

Coal preparation is a low cost element of the coal chain. However the simplistic analysis of the Table above hides the fact that coal preparation (including blending) can have a disproportionately positive impact on all downstream costs. Moreover it may well be cheaper to increase recovery in the preparation plant than by mining additional coal. The issues in terms of outcomes include: increased recovery of saleable coal, enhanced quality i.e. lower ash, moisture, fines phosphorus, etc., and better quality control and assurance.

In process terms, coal preparation involves a complex sequence of unit operations, many of which (particularly the fine coal circuits) are relatively inefficient. These must also be matched and integrated for overall plant operations. CSIRO adopts a holistic approach aimed at addressing breakage and liberation, washability, slurry distribution, size predication and classification, beneficiation of coarse and fine coal, dewatering of coal and tailings, sensors, process control and, ultimately total plant optimisation.

Recent successful applications of this R&D include:

- Demonstration of higher yields via selective breakage in coal crushers
- new design criteria for enhanced flow and splitting of coal slurries
- plant trials of three stage classification circuits for more accurate sizing
- proof of concept of a new high capacity flotation machine
- commercial sales of energy efficient flotation cells.
- procedures for reducing moisture content in coarse coal centrifuging
- commercial development of air-purged centrifugal de-watering
- improved dewatering in fine coal centrifuges
- additives that substantially reduce moisture in vacuum filtration and in some cases agglomerate the fines giving easier handling
- commercial on-line coal analysis instrumentation

Many of these advances should be of interest to Asian countries, particularly China and India, where much greater quantities of raw coal will need to be cleaned. Much of the work has been supported technically and financially through the Australian Coal Association Research Program (ACARP). More details of the CSIRO work and other ACARP-supported R&D will be given by the ACARP representative.

4. OTHER COAL PROCESSING

Although coal preparation in water based, dry processing can also be relevant. For example in China these are areas where water supply is a problem and/or the temperature is well below freezing part of the year. CSIRO has significant expertise in dry processing alternatives and is aware of the relatively poorly performing air tables and jigs. However, practical work has focussed on vibrating tables, magnetic (REMS) and electrodynamic separation in association with dry size classification. CSIRO has successfully demonstrated dry magnetic separation at coal preparation plants and power station sites as a lower cost, higher performance (combined ash-moisture context) alternative to conventional wet processing with greatly reduced environmental impacts (Figure 6). China has demonstrated a dry fluid bed density separation using magnetite. It may be worth examining specific opportunities for these technologies. CSIRO also has expertise and pilot

scale facilities for thermal drying, fines agglomeration, and sub-bituminous coal stabilisation (against moisture absorption and self heating) techniques that may be applicable, together with or separate from any dry processing approach. A binderless briquetting process has been successfully demonstrated at a cost well below that of conventional binder processes.

Although most of the work is targeted at coal preparation for the conventional power and steel industry, future markets are also considered. An example is the development of a chemical coal cleaning process that gives products with an ash content of less than 1%. These products can be made into coal/water or coal/oil mixtures and handled, and used, as oil substitutes. A multimillion \$ investment by CSIRO, industry, the Australian government and possibly overseas interests now seeks to demonstrate this ultraclean coal (UCC) process on a large scale.

CSIRO is also examining the development of solid smokeless fuels based on coal. Much of the less-developed world relies on solid fuels such as wood and wood charcoal or poor quality coal for heating and cooking. Alternatives such as gas, LPG, and kerosene are expensive and lead to foreign exchange difficulties. Yet deforestation (and greenhouse impact) is a major and accelerating problem, compounded by the low energy utilisation efficiencies of the open fires and appliances used. Furthermore, electric power is not the panacea, especially considering the capital costs, plus the cost of distribution and of appliances to utilize the electricity.

INTRODUCTION to Vision 21

The Department of Energy's (DOE) Office of Fossil Energy (FE), through the Coal & Power Systems (C&PS) program, has begun a new approach for developing 21st century energy plants that would have virtually no environmental impact.

A new approach is timely because of the unprecedented changes in the electric power industry — electric utility deregulation and restructuring, the availability of relatively low-cost natural gas, and environmental issues surrounding power production, including global climate change.

The new approach is called Vision 21. Vision 21 is a government/industry/academia cost-shared partnership to develop the technology basis for integrated energy plants that will, early in the 21st century, result in the deployment of ultra-clean plants that produce electricity and “opportunity” products, including clean transportation fuels, high-value chemicals, synthesis gas, and hydrogen.

The overall objective of the Vision 21 program is to effectively eliminate, at competitive costs, environmental concerns associated with the use of fossil fuels for producing electricity and transportation fuels. Vision 21 is based on three premises: (1) that the Nation will need to rely on fossil fuels for a major share of its energy needs well into the 21st century; (2) that it makes good sense to rely on a diverse mix of energy resources, including coal, gas, biomass and other renewables and nuclear, rather than on a limited subset of these resources; and (3) that research and development (R&D) directed at resolving energy and environmental issues can find affordable ways to make energy conversion meet ever stricter environmental standards.

Today, a typical power plant uses one type of fuel, usually coal, and produces only one thing — electricity. A Vision 21 plant would be fuel flexible, meaning it could use one or more of several different feedstocks including, coal natural gas, or petroleum coke. Any of these could be mixed with biomass. In turn, the plant could produce one or more high-value products such as electric power, clean fuels, chemicals, or hydrogen. Secondary products such as heat/steam for industrial use could also be produced. A Vision 21 plant will be capable of a variety of configurations to meet differing market needs, including both distributed and central power generation.

Vision 21 builds on the Clean Coal Technology (CCT) demonstration program experience and a portfolio of advanced technologies already being developed within the C&PS program —including integrated gasification combined-cycle, pressurized-fluidized bed combustion, advanced gas turbines, fuel cells, and fuels synthesis —and adds other critical technologies and system integration techniques. Endorsed by the President's Committee of Advisors on Science and Technology (PCAST) in 1997 and further supported by the National Research Council in 2000, C&PS' Vision 21 program serves as a “roadmap” for future electric power and fuels R&D. Vision 21 technologies, once achieved, will offer the United States, and the world, a new method of coal-based power generation that would have significant advantages over current methods.

Paper 4

Coal-Water Fuels

Coal-Water fuels (CWF), or Coal-Water Mixture (CWM), or Coal-Water Slurry (CWS) have significant advantages in terms of in-plant storage and handling, as they can be stored in tanks, and pumped. Early Swedish developments have already been described. CWF introduce the possibility of long distance pipeline transport, and/or can be carried in ocean tankers or river barges with relatively easily loading and unloading. Fugitive dust problems during transport and storage would be eliminated. Their use would also eliminate the need for milling as an integral part of boiler operation, as the milling would be done separately.

The development of CWF also introduces additional possibilities of a final coal cleaning operation at the utility site, with fine milling, advanced cleaning, and the preparation of a CWF with a minimal amount of additives which is used with only a short intermediate storage time.

Some commercial-scale work has been carried out on CWM technologies, and a 0.25 Mt/y plant has been built and operated at Rizhao city by a consortium of Japanese and Chinese companies in China's Shandong province. A 0.5 Mt/y plant has been operated in Sardinia by the Italian Enichem Company. The technology has not yet, however, been widely used because of unfavorable economics.

A particular opportunity offered by the use of CWF is that because much of the coal is milled to below 500 micron size, various advanced coal-cleaning processes can be used to reject some of the sulphur and mineral matter. Depending on the application, the top size of the particles can range from 500 to 30 micron. The coal is often milled to two separate size fractions in order to maximise the particle packing density. It is kept in suspension by the use of surfactant-dispersant additives. It can be held in a slurry which has 60--70% of coal, and contains only 1-2% of additives. There is thus some 30--40% of water present. While this represents a penalty in terms of combustion efficiency, NO_x formation in a pulverised coal fired boiler is suppressed, as the flame temperature is lower. For PFBC applications, the feed is sometimes made into a slurry or paste. The additives include dispersants, stabilisers, foam suppressants and biocides.

A recently commissioned plant at Fano in Italy is designed to beneficiate Colombian coal with an ash content of 12% and 0.9% sulphur to a product with an ash content of 3.5% and 0.5% sulphur.

The preferred parent coals for CWF are the high volatile bituminous coals with high specific energy, low inherent moisture, low ash content, low sulphur and good beneficiation characteristics. Easy size reduction is another desirable characteristic. In addition, a considerable amount of work has been done on low rank coals, investigating the possibilities of partial pyrolysis. Japan plans to expand its process range by developing a lignite-based CWF, and a 350 kg/h plant started up in 1994 is intended to develop a less expensive fuel than one based on bituminous coals.

Preliminary grinding is followed by wet ball-mill which is normally used to achieve the necessary size distribution. Oversize is recycled. The suspension is filtered to achieve a wet filter cake typically containing 20-30% of water. The various additives are added in stages, and this operation is carefully controlled to achieve the required product viscosity.

Coal cleaning may be carried out after the first grinding stage, but the cleaned coal may well need additional milling to achieve the necessary size distribution. In some circuits the additives are added directly to the feed coal. Cleaning may be required in any case, as the presence of liberated ash can affect coal slurryability, associated with its hydrophilic characteristics.

The requirements for CWF storage and handling systems are dependent on the fuel type, scale of operation and on the particular application. For use in a large utility boiler, for example, the fuel could be prepared on-site. The main buffer storage of coal could be in the traditional form of stockpiles (or solids). After preparation of

the CWF, the suspension could be maintained principally by agitation, with minimum use of stabilising additives, provided that the CWF could be supplied consistently to the burners and satisfactorily atomised. This would reduce materials costs significantly.

In a recent assessment of the most appropriate (and profitable) products from a mining operation in Alabama, USA, consideration was given to turning the fines product from a cleaning plant into a CWF. The selected coal from the Blue Creek seam was a medium volatile coal with a Hardgrove Grindability of 90+. Thus it was extremely soft with considerable breakage through the mining, handling, storage and preparation processes. Up to 40% was below 600 μm size. The raw coal was broken to below 50 mm. It was sized onto three fractions. The -50+6 mm fraction was treated in dense-medium baths. The -6+0.6 mm fraction went to Deister tables and heavy medium cyclones. The -600 μm material was treated and separated using flotation.

Drying the -600 μm material below 20% moisture content was regarded as difficult and/or costly. The advantages of developing a new product/market for this material in the form of a CWF include the increased value of the +600 μm product at 6% moisture.

The development has reached the stage of pilot plant testing (Hamilton and Arnold, 1992). It represents an interesting attempt to produce the most valuable products by an operation akin to 'refining' the coal. The intention is to establish whether the preparation of a CWF product would be cost-effective compared to drying and back-adding the fines to the coarse cleaned product.

The atomisation characteristics of the fuel are important for combustion, as are its slagging and fouling behaviour. In terms of coal cleaning, it is important not to alter the ash chemistry so much that the deposition tendencies increase, even though the quantity present has been reduced.

While the emphasis in work on CWF applications has been in possible retrofits to replace oil in oil-burning equipment, current interest is in taking advantage of the inherent possibilities of removing some of the impurities during a pre-cleaning stage. This would reduce the amount of residues formed during combustion in proportion to the amount removed, and might significantly reduce boiler operating costs. The possible applications are discussed further by Thambimuthu (1994).

Because of the possibilities of using advanced cleaning techniques on the coal, there has been interest in the reductions in SO_2 emissions achieved by using a sorbent loaded CWF. The results have been somewhat mixed. Reductions in the range 20--70% have been reported in pilot-scale flame tests. The use of sorbent loaded fuels in larger industrial flames proved relatively ineffective for sulphur capture. It may be possible to achieve acceptable SO_2 emission levels by using a combination of deep cleaning and in-duct sorbent injection. CWFs have been extensively tested, but the economic implications of their use, and the fact that they involve new and innovative technology, have meant that they have not yet been widely used. In the light of current developments the technology should be considered carefully as a potential contributor to optimising the coal-to-electricity chain, for particular coals. They may, however, prove to be more applicable in small rather than large applications.

Reading and Searching a Patent

Patent is an official license from the government giving one person or business the exclusive right to make use, and sell an invention for a limited period. Ideas are not eligible, neither is anything not new. The earliest known patent for an invention in England is dated 1449.

Reviewing patent documents requires the skill of understanding the significance of what is being disclosed. Legal counsel should always assist in interpreting the legal effect of any patent on commercial activity. However, a patent attorney or agent often must seek the assistance of technical personnel to gain a full understanding of the technology disclosed and claimed in a given patent. Further, an understanding of the form, content, and function of the various sections of a U.S. patent assists the non-lawyer in understanding the commercial importance of any issued patent.

The cover or front page of an U.S. patent must follow the form requirements placed on issued patents by the U.S. PTO. Specifically, the front cover discloses the inventor in two locations. The first named inventor is generally used as a head note for the patent. A given patent may often be referred to in an informal sense by this inventor's name.

Once the patent is issued, the inventor is referred to as the patentee. The first named inventor, if there is more than one, is printed prominently in the upper left-hand corner of the front page of the patent. All of the inventors or patentees are listed beneath the invention title along with the inventors' full names, addresses, and citizenship if other than the United States.

The title of the invention is generally written so as to use the shortest possible accurate description of the invention described fully in the patent and found in the claims. The patent application number and filing date are printed beneath the title. The application number and filing date are important because the patent application filing date may be used to eliminate other publications of third parties that might be used to limit the legal scope of the applicant's rights.

Also printed on the front page of the patent is a coded classification listing. This coding is complex and largely unnecessary to a lay person's understanding of a patent. This classification stems from the specific technology area to which the patent application was assigned during processing in the U.S. PTO. The classification also results from the search or review of prior patents completed by the Patent Examiner.

Apart from the technical classification information, the front page of the patent also contains a listing of publications or references cited during examination, including "United States Patent Documents," "Foreign Patent Documents," and "Other Publications" such as trade literature, journal articles, and product descriptions.

The front cover of the patent generally also identifies the U.S. Patent Examiner who reviewed and allowed the patent application, as well as the patent attorney, agent, or firm who worked with the Patent Examiner on the application.

Also provided is an abstract, which describes the invention, specifically highlighting its most valuable

properties and distinguishing features. By doing so, the abstract assists those searching for prior patents which disclose developments relevant to an invention or patent application presently under examination. Another aid to patent searchers is the listing of claims and drawing sheets. A representative drawing may also often be found on the front page of the patent, if figures are provided by the inventor. Figures or drawings are not required to receive a patent. However, where figures are essential to a full and complete understanding of the invention, they must be included. Further, the figures should show those elements of the invention which are found in the claims.

Within the body of the issued patent, the title is generally repeated to maintain clarity. A field of invention is then provided. The field of invention should direct the reader to the general area of technology to which the invention relates, and to specific improvements in the identified areas of application.

A description or explanation of the background of the invention may also be provided by the inventor. This background section discusses previous developments of inventors working in the same area of technology and may also list publications or patents that have discussed these developments and predate the filing date of the patent application. The background section may also point to deficiencies in the prior developments that the inventor intends to overcome.

To complement the discussion of problems and prior publication in the background of the invention, the inventor may generally provide a summary of the invention disclosed in instant patent. The summary of the invention should provide an explanation of the invention in the broadest and simplest terms and should also discuss how the invention disclosed in the patent solves problems remaining in prior work in this area of technology.

The patent should also provide a brief description of any drawings or figures. This brief description is often given in the technical terms used by engineering draftsmen to explain the various views illustrated in the figures.

The next section of the patent is titled "The Detailed Description of the Preferred Embodiment", often a multipage work serving several functions. First, the detailed description should provide an illustration of the invention in both its broadest or simplest sense and in its most preferred sense. Any elements of the invention that the inventor believes are crucial to the success or performance of the invention must also be included within this description. Further, this description should provide an explanation of the invention that is definite and illustrative, so as to allow persons having nothing but the patent before them to practice or use the invention in the manner intended. This description should be understood by those who work in the area that covers the subject matter of the patent.

Elements often include a detailed explanation of the various elements of the invention comprising the function of those elements, a written description of those elements, and an analysis of the elements that relies on any figures present in the patent application. The Detailed Description of the Preferred Embodiment may also include one or more working examples, especially if the invention is related to chemical technology. That is, in cases relating to chemistry, biochemistry, and chemical engineering, working examples are more often included than not. These working examples may serve any number of functions, including illustrating the formulation, applicability, and performance of the invention. Working examples may also be used to illustrate how the invention is distinguishable from those inventions previously developed and patented. As such, these working examples may include data such as adhesion and cohesion performance for adhesives, disinfecting

and sanitizing efficacy for cleaners, or data on chemical and physical properties for polymer systems.

The final section of an issued patent is the claims. A United States patent is required by law to have at least one claim. The claims represent the legal definition and boundaries of the rights resulting from the patent grant. Patent claims are analogous to the legal description which one might find on a title to real estate.

When evaluating an issued patent for purposes of determining the patentability of a new invention, the entire patent must be considered. As a result, the figures and The Detailed Description of the Preferred Embodiment are every bit as important to an issued patent as the claims. At certain times any one of these elements may become more relevant than another. For example, claims tend to be more relevant to determinations of patent infringement or violation. However, in determinations concerning the patentability of new inventions, the figures and The Detailed Description of the Preferred Embodiment may be the most relevant aspects of any previous patent.

Patent Searches. Because valid patent claims can only be issued on an invention that is novel and innovative in light of prior art, it is necessary to search the prior art for previous references either to the composition of matter, process, or machine defined in the claims of a patent application, or to any similar composition, process, or apparatus that would render the claimed invention obvious to a person skilled in the field of the invention. Inventions that have been described in a publication or embodied in a product are said to have been anticipated in the prior art and are not patentable. Patentability searches are performed by examiners employed by the national and regional patent offices and are an important step in the examination of patent applications. Patentability searches should also be performed by the representatives of inventors prior to the filing of a patent application so that the claims will not overlap with any publication in the prior art. These searches may encompass the full scope of the published literature, including patents, technical journals, gray literature, and even catalogs. Individuals or organizations who are making plans to introduce a new product or process must conduct infringement searches to ensure that they will not infringe patents that belong to others. Infringement searches need only consider patents in force and pending applications that may result in patents in countries where manufacturing or marketing are contemplated. After a patent application has been published and/or a patent has been granted, organizations that wish to practice the invention may also conduct validity searches to be used as ammunition for opposition proceedings or invalidity lawsuits. Validity searches, like patentability searches, should include all forms of published literature, but are limited to publications with effective dates earlier than the filing date of the patent application being challenged.

Searches of scientific and technical literature are performed using any of the information retrieval tools suitable for searches done for other purposes. Patent offices have devised special classification systems to facilitate searches among the individual patent documents in their collections. These patent classification systems were designed to subdivide patents into groups covering similar inventions were claimed in later applications. All of the existing fields of science and technology were defined and provided with a class code and subdivisions of the fields were given narrower classification designations. Patents belonging to each subclass were stacked together in drawers or on shelves similar to the stacks of boxes in a shoestore, and examiners or members of the public could extract a stack of patents and search for information in the subfield of interest by flipping through paper copies of the patent documents. As new fields of science and technology have developed, each patent classification system has been revised so that the emerging technologies can be searched. Patents are assigned classification codes by the examining office and the relevant primary classification and any cross-reference classifications are printed on the first page of the patent. Although patent classifications originated as tools for manual searches, they can be searched through printed or electronic

indexes as well.

Patent systems were conceived encouraging the dissemination of information on technological developments. Information dissemination is therefore essential for the patenting process. Patent offices have traditionally announced the issuance of new patents in bulletins and gazettes. Other organizations, notably scientific and technical societies and for profit publishers, have produced value-added patent information services. These secondary sources of patent information serve multiple purposes, among which are current awareness alerting, document delivery, and retrospective searching. Traditionally, such products have appeared as printed publications, but increasingly they have found second use in electronic form in on-line databases, and in the 1990s there has been rapid growth of optical storage of information, especially as Compact Disk-Read Only Memory (CD-ROM) products. Patent documentation is a field in considerable ferment, with rapid introduction of new products and capabilities.

Printed Patent Office Gazettes. The issuance of patents is announced by patent offices in publications typically known as gazettes and bulletins, which are published most commonly at the time of the patent's publication, but there are exceptions. Advance information is published in a patent gazette by some countries prior to the publication of patent documents, typically as a notification of filing details. However, some patent gazettes do not appear until well after the effective publication date of the patents they announce. The amount of information included in patent gazettes varies. Typically, they include bibliographic details on published patent applications and granted patents, including patent number, title, inventor, patentee, patent classification, application number and date, and priority application details if relevant. Some gazettes also provide the front page abstract of the patent and a representative drawing. In addition to announcement of new patents and applications, the various gazettes typically include listings of patents that have been rejected, challenged, or disclaimed, patents that have been allowed to lapse, and in some instances even listings of new applications that have been made but that will not be published for some time, if ever. Gazettes often include indexes to the information they contain; the amount of indexing available varies from country to country.

Information from Other Sources. Some of the abstracting and indexing services produced by scientific and technical societies have traditionally included patent information, especially in the field of chemistry. For instance. Chemical Abstracts (CA), produced by the American Chemical Society since 1907, has always covered patents. On the other hand, some notable information services have not included patent coverage. One example, despite the fact that many patents are based on some aspects of engineering, is the Engineering Index. Science Abstracts, covering physics, electricity, and electronics, is another example, which has not covered patents since 1976. However, even where patents are covered, the focus may not be ideal for those concerned with the legal aspects of patents. Thus, CA in its patent coverage documents the new chemistry involved, but shies away from the legal aspect of patents. For these and other reasons, others have stepped in to develop a variety of patent information services, e.g., Derwent information Ltd. of London.