Mining Chemicals

HANDBOOK

Revised Edition

CYTEC

Mining Chemicals

HANDBOOK

Revised Edition

CYTEC

Technology ahead of its time™

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Acknowledgment

This latest edition of Cytec's "Mining Chemicals Handbook," a traditional service to our customers and to the Mining Industry, was written and reviewed by our Mineral and Alumina Processing Technical Service staff. Their special effort is a sign of the importance we attach to serving our customers in every way possible.

The contributors were backed up by expert editorial comments from the Mineral and Alumina Processing staff in Cytec's global offices. Much of the credit for this book goes to the following contributors and editors who reviewed the book:

Arnold Day, Chief Editor

David Briggs
Frank Bruey
Frank Cappuccitti
Owen Chamberlain
Jennie Coe
Mark Eichorn
Peter Fortini
Terry Foster
Calvin Francis
Abdul Gorken
Jim Lee
Morris Lewellyn
Lino Magliocco
D. R. Nagaraj
Randy Nix
Donato Nucciarone
Wilfred Perez
Andy Poulos
Peter Riccio
Alan Rothenberg
Don Spitzer
Willard Thomas
Dave Withers

Congratulations to all these contributors for a job well done.

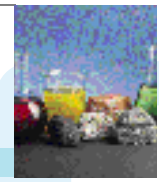
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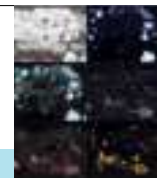
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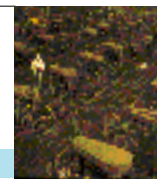
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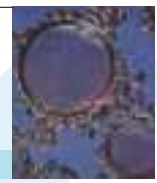
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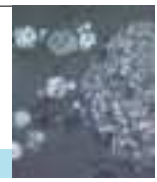
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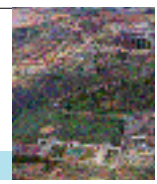
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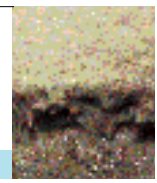
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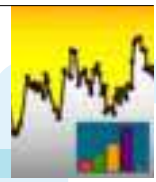
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Introduction

The year 2003 marks Cytec's 87th anniversary as a supplier of chemical reagents to the mining and mineral processing industry. Formerly a part of American Cyanamid Company, Cytec became an independent company in 1993. Starting as a supplier of cyanide to the gold-mining industry, our product line has expanded to over 500 reagents for use in flotation, flocculation, filtration, solvent extraction, and other applications. While most of these products were derived from our own research programs, others were obtained by Cytec's acquisition of OREPREP specialty frothers from Baker Petrolite, Nottingham Chemical's industrial mineral products, and Inspec (Chile) Mining Chemicals product lines in 1998 and 1999. These acquisitions have significantly expanded our product lines in sulfide and non-sulfide collectors, frothers, and defoamers.

The Mining Chemicals Handbook was originally little more than a directory of our products but, over the years, has evolved into a respected manual for use by engineers and plant operators in solving a variety of mineral processing problems. Of course, a manual of this scope can not, and is not intended to, provide in-depth information on all aspects of mineral-processing theory and practice. We hope, however, that it will provide a useful "starting point" for researchers and operators alike when planning a testing program or trying to solve some plant problem. More comprehensive information on all the topics discussed in this handbook can be found in innumerable textbooks, reviews, and technical papers, some of which are referenced in the bibliographies at the end of each section.

This latest edition of the Handbook includes a new section on the safety and handling of chemical reagents (Section 13). Cytec's foremost priority is the health and safety of all its employees and customers; we urge you to make it your priority to read this section and to consult with your nearest Cytec representative if you have any questions or comments regarding this important information.

You will also find a new section on the fundamental aspects of flotation chemistry (Section 5). Again, this is not meant to be a comprehensive review of this complex, and sometimes controversial, subject. Rather, it is intended to explain, and give examples of, the importance of designing or selecting the best collector, or collector combination, for each specific ore type. It demonstrates how seemingly insignificant changes to a collector's chemical structure can have a major impact on the flotation efficiency of different minerals as a function of pH and Ep, the pulp potential.

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New sections have also been added on guidelines for laboratory testing of flotation reagents (Section 4); the effect of selective reagents on the design and operation of flotation plants (Section 4A); and on the use of statistical methods for designing laboratory and plant experiments and the evaluation of the results obtained therefrom (Section 12). The applied mineralogy section (Section 3) and the computations section (Section 11) have been expanded to include some of the more recent developments in analytical instrumentation and automation and computer techniques available for these aspects of mineral processing. The section on solvent extraction (Section 10) has also been expanded to include the many new phosphine-based extractants that have been introduced since the last revision of the Handbook.

The manufacture (from basic raw materials) and the applications know-how of water-soluble polymers has been a core competency of Cytec since first introducing these products in the early 1950s. A complete range of both dry and liquid products is available for the flocculation and dewatering of mineral slurries. The flocculants section (Section 8) has been expanded considerably to cover the composition and use of these water-soluble polymers. Of particular note is the development and widespread acceptance of hydroxamated polyacrylamide (HXPAM) flocculants for use in the Bayer process. This new chemistry provides significant process benefits in red mud settlers and thickeners. A new section (Section 9) has been added which describes these polymers and other Cytec products for use in alumina refineries.

As both we and our customers learn more about the interaction of reagents with various ore-types, the practice of "custom-designing" a unique reagent or reagent formulation for individual ores has become increasingly common. Although there are a host of factors which have a bearing on any plant operation, we believe that the choice of chemical reagents is often under-appreciated. While many problems do not have a "chemical solution", the proper testing and selection of reagents can often have a major impact on plant performance e.g. improved metal recoveries and concentrate grades, better elimination of penalty elements, reduced lime consumption in flotation, the possibility of operating at a coarser primary grind, etc. Cytec's technical representatives are available to work with you in optimizing the use of all our reagents. Since Cytec offers a total range of mineral processing reagents, our technical representatives are in a position to help you take advantage of interactions and synergies among the chemicals used in any particular process. They are backed by an experienced team of researchers, engineers, metallurgists, and chemists.

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As mentioned previously, the range of products which Cytec offers has expanded dramatically over the last several years. Since many of these were custom-designed for a specific orebody, it is not possible to include every single one of them in this Handbook. Rather, we have tried to include the major products from each "chemical family" of reagents. You should also note that, from time to time, certain products may be available only on "special order" in minimum quantities or even discontinued, for a variety of reasons. Your Cytec representative is in the best position to not only advise you on the availability of new or experimental products, but also to make sure that you do not waste time by testing products which are not available.

The concept of "Joint Technical Development Programs" between supplier and user is one which Cytec has employed successfully for many years. We know our reagents (and what they can or can not do) better than anyone, but we are also aware that nobody knows your ore better than you do!

Important note: All reagent dosages in the Handbook are expressed as grams per metric ton of ore (abbreviated as g/t) unless noted otherwise. To avoid confusion, we have not used the term "tonne"; the term "ton" always means a metric ton. To convert from grams/metric ton to pounds per short ton, simply multiply by 0.002, or divide by 500. Similarly, precious metal and other trace elements contents are expressed as grams per metric ton (g/t) or ppm; to convert grams per metric ton to troy ounces per short ton, simply divide by 34.28. For other convenient conversion factors, see Section 14.

Physical properties are given for some of the more common Cytec reagents. For more details, please consult the individual product data sheets and MSDS's.

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2 Usage of Cytec flotation reagents



Reagent	Page	Form	Usual dosage, g/ton	Feeding method
Promoters				
AEROFLOAT 25 promoter	108	Liquid	25-100	Undiluted
31	108	Liquid	25-100	Undiluted
208	111	Liquid	5-50	5-20% solution or undiluted
211	111	Liquid	10-100	5-20% solution or undiluted
238	111	Liquid	10-100	5-20% solution or undiluted
241	108	Liquid	10-75	5-20% solution or undiluted
242	109	Liquid	10-75	Min. 10% solution or undiluted
AERO 7310 promoter	109	Liquid	10-100	5-20% solution or undiluted
Sodium AEROFLOAT promoter	112	Liquid	5-50	5-20% solution or undiluted
AERO (or SF) 203 promoter	107	Liquid	5-100	Undiluted
AERO (or SF) 204 promoter	107	Liquid	5-100	Undiluted
AERO (or SF) 758 promoter	107	Liquid	5-100	Undiluted
AERO 303 xanthate	106	Solid	10-100	10-20% solution
317	106	Solid	10-100	10-20% solution
325	106	Solid	10-100	10-20% solution
343	106	Solid	10-100	10-20% solution
350	106	Solid	10-100	10-20% solution
AERO 400, 404, 407, 412 promoter	115	Liquid	5-50	5-20% solution or undiluted
AERO 3302 promoter	107	Liquid	2-25	Undiluted
3477	111	Liquid	5-25	5-20% solution or undiluted
3501	112	Liquid	5-25	5-20% solution or undiluted
3894	116	Liquid	5-25	Undiluted
4037	120	Liquid	5-100	Undiluted
5100	118	Liquid	5-100	Undiluted
5415	117	Liquid	5-50	Undiluted
5430	111	Liquid	5-100	Undiluted
5460	117	Liquid	5-100	Undiluted
5474	111	Liquid	5-100	Undiluted
AERO 5500, 5540, 5560	119	Liquid	5-100	Undiluted

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Common Sulfide Materials						Precious metals	Non-sulfide base metals	Non-metallics, metallic oxides, etc.
Pb	Zn	Cu	Fe	Mo	Co-Ni			
✸	✸	✸	✸			✸		
✸	✸	✸	✸		✸	✸		✸
		✸				✸		
	✸	✸						
		✸				✸		
✸	✸	✸				✸		
✸	✸	✸				✸		
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✸	✸	✸	✸		✸	✸	✸	
✸	✸	✸	✸		✸	✸	✸	
✸	✸	✸	✸		✸	✸	✸	
✸	✸	✸	✸		✸	✸	✸	
✸	✸	✸	✸		✸	✸	✸	
✸	✸	✸	✸		✸	✸	✸	
	✸	✸		✸		✸	✸	
	✸	✸			✸	✸		
	✸	✸			✸	✸		
	✸	✸				✸		
	✸	✸				✸		
	✸	✸				✸	✸	
	✸	✸			✸	✸		
	✸	✸			✸	✸		
	✸	✸				✸		
	✸	✸			✸	✸		
	✸	✸			✸	✸		

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Reagent	Page	Form	Usual dosage, g/ton	Feeding method
Promoters				
AERO 5688 promoters	111	Liquid	5-100	5-20% solution or undiluted
6682	120	Liquid	5-100	5-20% solution or undiluted
6697	113	Liquid	5-100	5-20% solution or undiluted
7151	120	Liquid	5-100	5-20% solution or undiluted
7249	114	Liquid	5-100	5-20% solution or undiluted
7380	120	Liquid	5-100	Undiluted
7518	120	Liquid	5-100	Undiluted
7640	120	Liquid	5-100	Undiluted
8399	120	Liquid	5-100	Undiluted
Reagent S-8474, S-8475 promoters	120	Liquid	5-100	5-20% solution or undiluted
Reagent S-8718 promoter	120	Liquid	5-100	Undiluted
S-8805	120	Liquid	5-100	Undiluted
AERO 8761	120	Liquid	15-100	5-20% solution or undiluted
8880	120	Liquid	10-50	Undiluted
8985	120	Liquid	10-50	5-20% solution or undiluted
9020	120	Liquid	10-50	Undiluted
Reagent S-9411 promoter	120	Solid	5-100	10-20% solution
AEROPHINE 3418A promoter	114	Liquid	5-50	5-20% solution or undiluted
AERO 6931 Promoter	114	Liquid	5-50	5-20% solution or undiluted
Reagent S-4604	114	Liquid	5-50	5-20% solution or undiluted
AERO 3000C promoter	170	Liquid	100-500	Undiluted
3030C	170	Liquid	100-500	Undiluted
3100	170	Paste	100-500	10-15% dispersion in water
AERO 702, 704, 708, 718 promoters	169	Liquid	250-1500	Undiluted
AERO 722, 728 promoters	169	Liquid	250-1500	Undiluted
AERO 727, 727J	169	Liquid	250-1500	Undiluted
730 promoters	169	Liquid	250-1500	Undiluted

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Common Sulfide Materials						Precious metals	Non-sulfide base metals	Non-metallics, metallic oxides, etc.
Pb	Zn	Cu	Fe	Mo	Co-Ni			
	✸	✸ ✸ ✸ ✸ ✸ ✸ ✸ ✸ ✸				✸ ✸ ✸ ✸ ✸ ✸ ✸ ✸		
✸	✸	✸	✸		✸	✸		
		✸ ✸				✸		
		✸ ✸ ✸ ✸		✸ ✸ ✸		✸ ✸		
✸	✸	✸	✸		✸	✸		
✸	✸	✸			✸	✸		
✸	✸	✸				✸		
✸	✸	✸				✸		
							✸ ✸ ✸	✸ ✸ ✸
							✸	✸
								✸ ✸ ✸

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Reagent	Page	Form	Usual dosage, g/ton	Feeding method
Promoters				
AERO 825 promoter	166	Viscous Liquid	250-1500	10-30% dispersion in water
827	166	Viscous Liquid	250-1500	10-30% dispersion in water
828	166	Liquid	150-250	Undiluted
830	167	Liquid/Paste	150-750	5-10% dispersion in water
845	167	Liquid	150-750	5-10% dispersion in water
847, 848	169	Liquid	25-100	5-10% w/Fatty Acids
850	166	Liquid	250-1500	Undiluted
AERO 851, 852, 853, 854,		Liquid	250-1500	Undiluted
855, 857 promoters	166	Liquid	250-1500	Undiluted
AERO 856 promoters	166	Liquid	250-1500	Undiluted
858	166	Liquid	250-1500	Undiluted
862	166	Liquid	250-1500	Undiluted
865	166	Liquid	250-1500	Undiluted
866, 869	166	Liquid	250-1500	Undiluted
870	169	Liquid	25-100	5-10% dispersion in water
Frothers				
AEROFROTH 65 frother	123	Liquid	5-100	Undiluted, 5-25% solution
70	123	Liquid	15-100	Undiluted
76A	123	Liquid	15-100	Undiluted
88	124	Liquid	15-100	Undiluted
OREPREP 501 frothers	124	Liquid	15-100	Undiluted
507	123	Liquid	15-100	Undiluted, 5-25% solution
515	124	Liquid	15-100	Undiluted
521	124	Liquid	15-100	Undiluted
523	124	Liquid	15-100	Undiluted
533	124	Liquid	15-100	Undiluted
549	125	Liquid	15-100	Undiluted

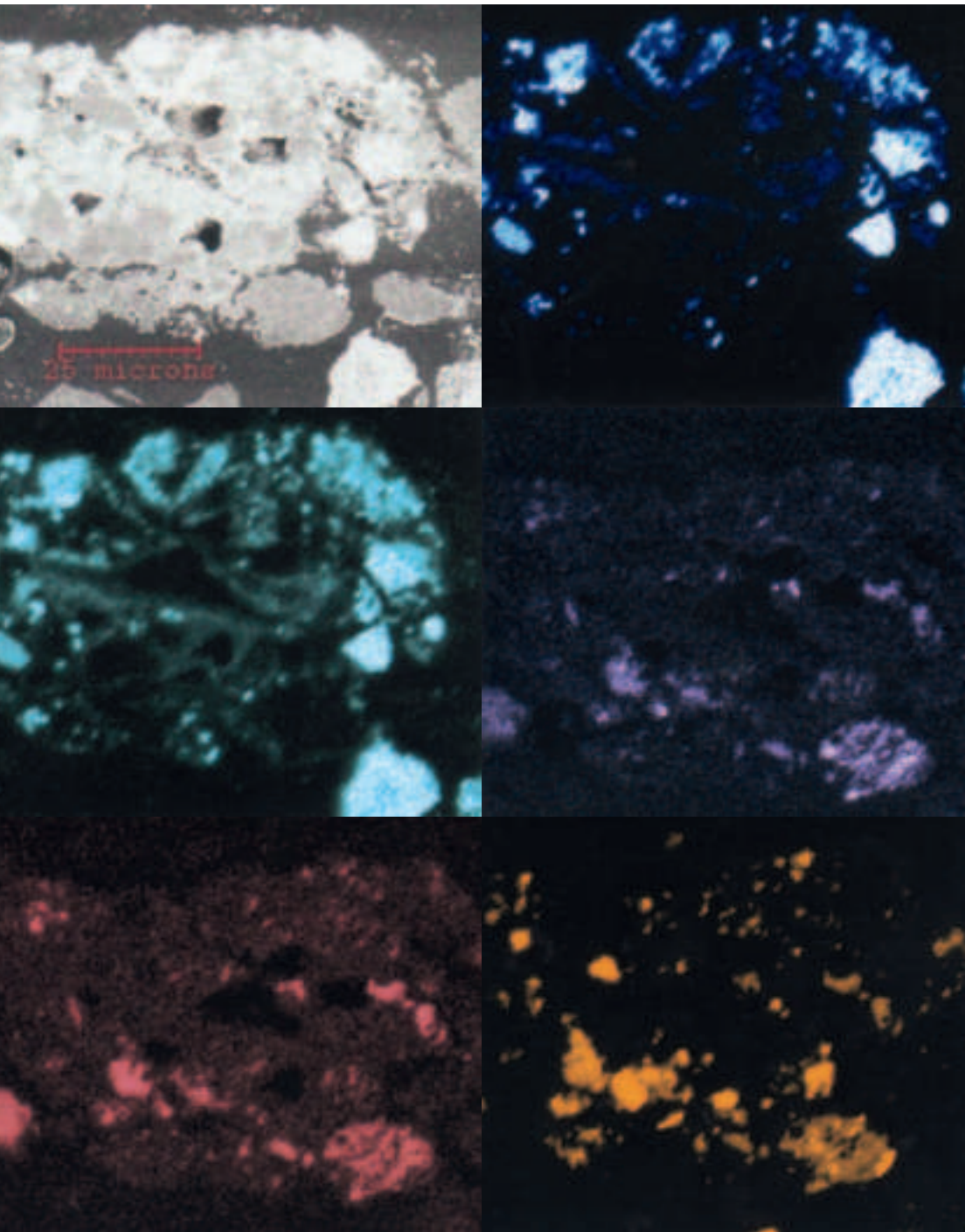
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Common Sulfide Materials						Precious metals	Non-sulfide base metals	Non-metallics, metallic oxides, etc.
Pb	Zn	Cu	Fe	Mo	Co-Ni			
							✷	✷
							✷	✷
							✷	✷
							✷	✷
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✷	✷	✷	✷	✷		✷		✷
✷	✷	✷	✷			✷		✷
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✷	✷	✷			✷	✷		
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✷	✷	✷			✷	✷		
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3 Applied mineralogy and mineral surface analysis



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Section 3 Applied mineralogy and mineral surface analysis

3.1 Applied mineralogy

Applied mineralogy, sometimes called process mineralogy, involves the identification and the mode of occurrence of minerals as they relate to the beneficiation of ores. Even today, in the actual practice of mineral beneficiation, the role of applied mineralogy is often not fully appreciated and utilized. However, in order to optimize the treatment of any particular ore, applied mineralogy must play a prime role.

In developing a process scheme for a new ore, identification of the minerals present in the ore is the essential first step. Some minerals may be considered "valuable" and others "undesirable." These are relative terms, depending upon location, metal or mineral prices, associated minerals, and other circumstances of a particular deposit. Mineral economics must be kept in mind. Calcite, fluorite, hematite, and pyrite, for example, can be valuable minerals in certain deposits and undesirable in others. Simple identification of the constituent minerals is usually not sufficient to guide a beneficiation scheme. Even in simple ores, the amenability of a mineral assemblage to beneficiation depends not only on the nature and abundance of the minerals, but also on their textures, size ranges, surface condition, and modes of occurrence. Many fine-grained or complex ores have remained unexploited for many years because they were not amenable to the beneficiation technology then available, or because their mineralogical characteristics were not adequately understood.

Another important role of applied mineralogy is in maintaining optimum metallurgy and trouble-shooting in an operating plant. This is achieved by routine mineralogical examination of laboratory and mill process streams.

The objectives of mineralogical examinations as they relate to both operating plants and design schemes for new ores are discussed below. The first two items are essential steps in optimizing ore beneficiation. The importance and need for the others depends on the type and complexity of the material under investigation.

Identification of the minerals present in the ore

Mineralogical data from general geological studies and hand specimen identification are inadequate. In order to select the best process scheme for a new ore, or to trouble-shoot effectively in an operating plant, an accurate identification of the minerals and their mode of

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occurrence are necessary. Mineral identification is accomplished using optical, physical, chemical and instrumental methods. Microscopical examination of thin sections and/or polished grain mounts is usually the first step.

Some examples of why detailed mineral information is important to ore beneficiation are:

- Occurrence of the desired element in more than one mineral, particularly if the minerals have different responses to concentration. Examples: gold as native gold and gold in solid solution in pyrite; copper in chrysocolla and chalcopyrite; copper in chalcopyrite, malachite and Cu-bearing goethite; tin in cassiterite and frankeite.
- Variability in mineral composition (substitution, isomorphism). Examples: variability of Ag in solution in gold grains, high-Fe versus low-Fe content in sphalerite.
- The presence of gangue minerals that can have an adverse effect on beneficiation; eg. montmorillonite and talc.
- The presence of rare or unexpected minerals.

Determination of mineral textures and associations with other minerals

This can be either a qualitative or quantitative analysis; in the latter case it is often referred to as a "modal analysis" and involves the determination of the degree of liberation (at various grind sizes) of the valuable from the non-valuable minerals. This information is essential to the selection, modification or operation of a particular beneficiation process. Some important features to look for are:

- Rims or coatings of one mineral around another. Examples include digenite/chalcocite rimming pyrite; pyrite around galena; pyrite with an inner rim of chalcocite and an outer rim of Cu-bearing goethite.
- Extremely fine, intimate intergrowths of two or more minerals. Examples include ilmenite/magnetite/hematite; pentlandite/pyrrhotite; chalcopyrite/sphalerite; sphalerite/chalcopyrite/galena.
- Extremely fine inclusions of one mineral in another, such as 2 micron or smaller gold blebs in quartz; chalcopyrite blebs in sphalerite; fine chalcopyrite grains in magnetite.
- More than one mode of occurrence of a desired mineral. For example, free gold and fine gold inclusions in arsenopyrite; free chalcocite and chalcocite locked with siliceous gangue.

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Identification of minerals diluting a concentrate

Mineralogical examinations can provide insightful data in regards to a low-grade concentrate. An examination can determine if the diluents are free or locked with other minerals. If the diluents are locked, it can be determined what conditions could be changed, if any, to achieve a higher grade. In addition to those mineral which merely lower the concentrate grades and add to smelting costs, certain other minerals need to be identified since they contain toxic penalty elements. Examples include: As in arsenopyrite, tennantite, orpiment, realgar; Sb in stibnite, tetrahedrite, antimonite; Bi in bismuthinite; Cd in sphalerite.

Identification of the cause of mineral recovery difficulties

Mineralogical examination of flotation tail samples can identify the valuable minerals reporting to the tail, determine if they are free or locked, and provide a good indication of whether optimizing flotation conditions in some way could improve recovery. If the value minerals are locked, their grain sizes and degree of locking with other value or gangue minerals can be determined, thereby providing useful information for optimizing the grinding size.

3.1.1 Sampling the ore or mineral sample

The value of a mineralogical examination depends on the relevance of the samples examined as well as on the manner of their investigation. An unrepresentative sample may provide useful mineralogical information, but may not thoroughly define a problem. In many cases, the granular samples submitted for mineralogical examination are intended to represent thousands of tons of ore or perhaps hundreds of tons of concentrate or tailings. Whether the samples are truly representative is beyond the control of the mineralogist. For plant and laboratory products, however, the mineralogist should insist on samples which are as representative as possible.

On the other hand, the mineralogist has a responsibility to assure that the sub-samples which he extracts, treats, and examines from the submitted samples are reasonably representative of that sample. Only a "pinch" of a granular sample is used for a loose-grain mount for the petrographic microscope. Micas may concentrate toward the top of the sample envelope and heavy minerals to the bottom; calcite, jarosites, and clay minerals may concentrate in the fines; highly magnetic minerals may form clusters.

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3.1.2 The tools of mineralogy

The tools of a mineralogical examination range from a hand lens and hand magnet to sophisticated instruments like the x-ray powder camera, the diffractometer, the electron microprobe and the QEM-SEM. Optical microscopes are still in wide use because of the breadth and versatility of observations made with them. They are aided by various separating devices and techniques. Screens and pneumatic sizing devices provide size-fractions for more detailed study. Heavy-liquid, electro-magnetic and electro-static separations, panning machines, and selective dissolution collect or eliminate certain minerals or groups of minerals. Microscopes also help select certain grains or areas for study by more specialized instruments, such as the electron-microprobe.

There are three principal types of optical microscopes used in applied mineralogy: the stereoscopic microscope, the petrographic microscope, and the ore microscope. The stereoscopic microscope is used for examining loose grains and rough surfaces under oblique illumination at magnifications of 5X to as much as 210X. The petrographic microscope is used for examining thin sections and transparent grains by axially-transmitted light at magnifications of about 20X to 1200X. The ore microscope is used for examining polished sections of ores and opaque grains by axially-reflected light at magnifications of about 20X to 1200X. Higher magnifications are possible, but a point is soon reached above which magnification is not desirable because it does not resolve any further detail. For higher resolution, the scanning electron microscope is required. (See 3.1.2.5)

Both ore and petrographic microscopes are polarizing microscopes with the rotating stages graduated in degrees. The images are inverted, and the working distances between objective lens and object are small, particularly for objectives having powers greater than 10X. Because of their higher powers and shallower depths of field compared to the stereoscopic microscope, these instruments require very low relief in the material under observation. In some instruments, sources for both transmitted and reflected light are available, providing the capabilities of both the petrographic and ore microscopes.

For maximum usefulness, ore and petrographic microscopes require more knowledge of optics, crystallography, and microscopy than do stereoscopic microscopes. The use of polarized light permits the determination of several optical properties and their angular relations to certain crystallographic directions such as those of cleavage, edges, and elongation. From these observations, positive

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identification of many minerals can be made, even from particles of only a few microns in maximum dimension.

3.1.2.1 The stereoscopic microscope

Use of a stereoscopic microscope is a vital first step in the mineralogical examination of samples of crushed and ground ores, and of laboratory and mill products. The image is three-dimensional, and physical and crystallographic features are the same as those seen on coarser minerals with the naked eye. Some minerals can be readily recognized by such properties as color, luster, crystal habit, cleavage, fracture, transparency, and magnetic behavior. The microscope has considerable working distance between the lower lens and the object to permit manipulation of grains and simple physical and chemical tests. Free minerals can be picked out by needle or forceps for separate tests. Grain sizes can be measured by the use of scales mounted in one of the eyepieces. Coarse locking between minerals can be observed and followed in a series of decreasing size fractions. Identification of unrecognized or partially obscured minerals is usually difficult unless they can be manipulated to produce easy diagnostic test results.

In addition to permitting an overall view of the mineral assemblage, the stereoscopic examination can indicate the desirability, direction, and scope of further investigation. It is often beneficial to subdivide the sample into two or more fractions using size, magnetic susceptibility, gravity, or other physical properties to obtain products which need more critical evaluation by other techniques. Chemical methods are also useful. An acid-insoluble residue may provide information not easily available otherwise. These separations may be qualitative or quantitative, as the case requires. All granular products of these separations should be examined under the stereoscopic microscope for identification.

Section 3.1.3 provides useful tables of minerals characteristics for identification of minerals by stereoscopic microscopy.

If further identification, greater textural detail, or quantitative mineralogical analysis are needed, recourse should be made to petrographic and ore microscopes.

3.1.2.2 Petrographic microscopy

The petrographic microscope can be used to identify transparent minerals, which constitute the great majority of all minerals. Opaque minerals are seen in silhouette. The microscope is used in

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examinations of thin sections and loose grains in very thin layers. The thin sections are about 30 microns thick and are made from slices of rock, ore, or in some cases, plastic with embedded fragments. Loose grains are examined in oils or similar media. Oils are usually of known index of refraction for comparison with those of transparent minerals. Usually a series of different reference oils are used to match or bracket the indices of refraction of various minerals. All of these preparations are made on microscope slides and covered with a thin cover glass. For more information on the techniques of petrographic microscopy, the reader is referred to the books and articles listed in the bibliography.

3.1.2.3 Ore microscopy

The ore microscope can handle the microscopically opaque minerals and several minerals which are called "semi-opaque." The "semi-opaque" minerals include such common ore minerals as sphalerite, cuprite, hematite, proustite, and pyrargyrite, which are usually studied under the ore microscope because of their associations with more opaque minerals.

Under an ore microscope, the mineralogist examines polished surfaces of ore fragments and mineral grains. In most cases, these objects have been cast in plastic briquettes, which after hardening are abraded to a plane surface and polished to a mirror finish. Care must be taken that the polished surface is perpendicular to the axis of the microscope during examination. Minerals are identified on the basis of reflected color, reflectivity, polishing hardness, internal reflection (if any), cleavage, crystal habit, and optical properties of the mineral surface in the presence of polarized light. With a micro hardness tester, indentation hardness numbers may be obtained by measuring a critical dimension of an impression made in a mineral surface by a shaped diamond under a known load. Relative reflectivities may be judged by eye by comparison with those of several common minerals such as pyrite, galena, tetrahedrite, sphalerite, and magnetite. There also are useful accessories for quantitatively measuring the reflectivities of polished mineral surfaces.

Classical test procedures have been developed to aid the mineralogist. Etch tests may be performed at low power on single minerals to help identify them. Reagents which stain certain minerals diagnostically, may be applied locally or over the entire polished surface.

Individual grains may be worked out of the surface for microchemical tests or x-ray diffraction. With the advent of the electron

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microprobe in many laboratories, these classical tests are used less commonly; but, when properly done, the etch and stain tests can be quick and decisive.

Some 330 minerals are more or less opaque and can be studied to advantage under an ore microscope. Of these, only about 30 are distinctively colored in polished surface; the rest occur in various shades of gray. Fortunately, some of the common minerals, like pyrite, chalcopyrite, covellite, pyrrhotite, and copper have distinctive colors, although they are less intense than those seen in hand specimens with a hand lens or unaided eye. Some "semi-opaque" and transparent minerals may show characteristic internal reflections, as in proustite, malachite, and alabandite. Sphalerite, on the other hand, shows a wide range of body colors in its internal reflections. Further detail in books and articles on the techniques of ore microscopy are contained in the bibliography.

3.1.2.4 X-ray diffraction (XRD)

X-ray diffraction provides the exact identity of crystalline minerals. X-ray beams diffracted off of powdered mineral surfaces give interference patterns that are characteristic of each crystalline phase. In mineralogical studies, X-ray diffraction is often used to, (1) confirm the presence of talc, (2) identify the specific clays or other fine-grained minerals present, (3) identify the specific serpentine minerals and, (4) identify the carbonate minerals.

3.1.2.5 Scanning electron microscope/energy dispersive X-ray (SEM-EDX)

The electron-microprobe is an extremely useful supplement to optical microscopy. Most electron microprobes can accept standard briquettes for examination. The only additional preparation is for an extremely thin coating of carbon or conductive metal to be sputtered over the polished surface to conduct the electrical charge away. A beam of electrons (as small as 1 micron in diameter) can be focussed on a selected point or it can be made to scan a small field to determine the silver content of gold grains, the substituent elements in sphalerite or tennantite, or an analysis of a fine inclusion. It can also map the distribution of specified elements.

The electron microscope enables the viewing of a sample at high magnifications. Energy dispersive X-ray provides an elemental analysis of minerals containing elements with atomic numbers from beryllium to uranium. When the electron beam bombards a sample,

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X-rays, characteristic of each element, are emitted. The SEM-EDX is a valuable tool for the microscopist because, with careful preparation, individual grains in a thin section or polished grain mount can be analyzed for chemical content. SEM-EDX analysis provides, (1) elemental data for unknown phases, (2) identity of trace elements in minerals, (e.g., copper in goethite, silver in galena and silver in gold, substituent elements in sphalerite or tennantite), (3) elemental mapping, (4) identification of small inclusions, and (5) high magnification.

3.1.2.6 Automated image analysis

Several computer-controlled, automated techniques for quantitative image analysis have been developed. The use in this handbook of QEM-SEM (Quantitative Evaluation of Minerals with Scanning Electron Microscope) as an example does not imply or constitute a recommendation of any one system over another.

QEM-SEM[1] is a fully-automated, powerful image analyzer which can determine quantitatively the size distribution and association of minerals or phases in complex mixtures. The system, developed by CSIRO, Australia, uses X-ray and electron signals generated in a scanning electron microscope to produce lineal or two-dimensional representations of the mineral assemblages. In the simplest mode of operation, point identification provides an automated version of conventional volume fraction determination (point counting). This technique provides both the degree of liberation of specified minerals and the intergrowth distribution for unliberated minerals.

QEM-SEM comprises a computer-controlled scanning electron microscope fitted with a multi-element, (up to 4) energy dispersive X-ray detector and a back-scattered electron detector. Samples are prepared in the form of polished sections. The electron beam is positioned automatically at regularly-spaced points in a field of observation. For particles, the line spacing is made the same as point spacing along lines, typically 3 µm, in order to obtain a full 2-D image of each particle. For drill core samples, the line spacing is much greater (up to 200 µm). For determination of volume fractions alone, a widely spaced (40 to 200 µm) grid of points is used. At each sampled point, the signal generated by the back-scattered electrons is used to determine the average atomic number of the small area of material irradiated by the beam and thus identify the mineral phase. More typically, the beam is left in position for 20-30 ms until sufficient X-rays have been collected to allow computer identification of the particular mineral present. The procedure is repeated for successive fields of observation in order to generate mineral maps. The computer software then isolates the individual mineral particles as

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grains from the mineral maps, to determine the amount of each mineral present, its mean grain size or grain size distribution, and its degree of association with other minerals. For visual display, each mineral is color-coded and viewed on a color monitor. Particles in the size range 5 to 500 µm can be readily handled in the analysis. Typically, 500-1000 particles in the size range 53-106 µm can be analyzed in 1-2 hours. For dense minerals present in amounts of less than 1-2%, high-speed back-scattered electron imaging can select, for detailed mapping, only those particles or local areas containing the desired mineral. A relatively large sample can thus be scanned to identify a statistically significant number of occurrences of the mineral of interest. This technique, for example, simplifies the search for value-mineral occurrences in flotation tailings.

[1] *Manufactured by LEO Electron Microscopy, A Carl Zeiss SMT AG Company*

3.1.3 Tables for identification of selected minerals in fine granular samples under a stereoscopic microscope

The three tables at the end of this text list approximately 100 selected minerals and certain properties which may assist in identifying them under a stereoscopic microscope in ground ores, mill products, and natural sands. These minerals have been selected partly because of their abundance or economic importance in the mineral industry and partly because of their potential amenability to sight recognition as fine particles. Unfortunately, abundance and importance do not always go hand-in-hand with such amenability. Many important minerals have been omitted for lack of visual diagnostic properties in fine sizes.

It is not to be expected that these tables will enable the observer to make many positive identifications of unknown minerals; that is not always possible without the aid of instruments more elaborate than the stereoscopic microscope. The primary purpose of the tables is to provide guidance for the recognition, under magnification, of minerals known from previous experience, probably at a coarser size. Several common minerals, such as galena and malachite, can often be recognized under the stereoscopic microscope simply by their obvious similarity to their macroscopic counterparts. With experience, the number of minerals recognizable in fine sizes will continue to grow.

Naturally, some previous knowledge of mineralogy and its terminology is assumed, but a few pertinent definitions are reviewed below. Further details on principles and mineral descriptions are

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available in standard texts on mineralogy. It should be emphasized here, however, that reduction in particle size may obscure, alter, or render indeterminate some properties normally recorded in published mineral descriptions. For example, crystal shapes may have been destroyed; and the colors of transparent minerals may seem unduly pale.

Qualitative determination of the minerals is typically based on direct observations and physical measurements of specific gravity, luster, hardness, color, fracture, cleavage and streak.

The tables are divided on the basis of luster and specific gravity, as follows:

Table 3-1: Minerals with metallic to sub-metallic luster.

Table 3-2: Minerals with non-metallic luster and specific gravities below 2.95.

Table 3-3: Minerals with non-metallic luster and specific gravities above 2.95.

The **luster** of a mineral refers to the quality and intensity of light reflected from a fresh surface. The quality is expressed in such terms as metallic, vitreous, silky, and resinous. Imperfect lusters are designated by the prefix "sub," but such refinement cannot always be made on small grains. Hyphenated terms, like metallic-pearly, refer to a combination of sub-metallic and a second luster; such combinations are rare in the tables.

- Metallic luster is the luster of metals, as seen in gold, copper, and pyrite. All other lusters are grouped as "non-metallic."
- Vitreous luster is the luster of broken glass. Adamantine luster is the luster of diamond. Greasy luster is the luster of oily glass.

Other terms such as pearly, silky, and resinous are self-explanatory.

The dividing point between minerals in Tables 2-2 and 2-3 was chosen at a specific gravity of 2.95 because that is the specific gravity of acetylene tetrabromide (also called symmetrical tetrabromoethane), a heavy liquid commonly used in laboratory sink-float separations. There are so many minerals with non-metallic lusters that it is desirable to split them into at least two gravity fractions.

The tables can be used without the gravity separation, but much more successfully if this separation can be made before the minerals are to be examined. If low-gravity minerals like gypsum and brucite are being sought, a liquid with a specific gravity of about 2.50 would be helpful to concentrate them.

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For purposes of these tables, acetylene tetrabromide is the most important. Very few of the minerals listed have specific gravities close to 2.95. Biotite and tremolite have ranges which straddle 2.95. Biotite is included in the Mica Group in Table 2-2 and listed separately in table 2-3. Tremolite is listed in both tables. Otherwise, a specific gravity of 2.95 makes a relatively clean break between the listed minerals – a break which can readily be sought in a sample of liberated grains by a simple procedure.

In each table minerals are listed alphabetically with their chemical formula. Mineral groups like the feldspars and the skutterudite series are included, but their individual species, except biotite (see above) are not. If further details on group members are needed, they should be sought in mineralogy texts.

Mohs hardness numbers are listed in columns headed by "H." Although hardness is not useful under a stereoscopic microscope as in hand specimens, the numbers will serve as a guide to relative scratch resistance, which may be an observable clue in some cases. Bear in mind that the apparent hardness of a fine-grained aggregate like earthy hematite or kaolinite is not the true hardness of the mineral itself.

Specific gravities are listed in the third columns, under the heading "sp. gr."

Lusters are listed in the fourth columns, often by simple abbreviations. Minerals with a wide range of lusters may appear in two tables. Hematite, for example, with lusters ranging from metallic to dull, occurs in both Table 2-1 and Table 2-3.

Colors are listed separately in the fifth column in Table 2-1 because the colors of those minerals are reasonably constant and characteristic. The colors of the transparent minerals are usually not characteristic (calcite and fluorite, for examples), but when they are helpful for identification, the colors are included under remarks.

Fracture describes the kind of surface obtained when a mineral breaks in a direction which is not a cleavage direction. Fractures are useful diagnostic properties in many cases as they still are apparent in fine sizes when cleavage does not predominate. The principal types of fracture are:

- ***Conchoidal fracture*** (abbreviated "conch") – forms one or more smooth shell-like surfaces, either convex or concave.
- ***Even fracture*** – forms a nearly smooth plane with only gentle depressions and elevations.

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- *Uneven fracture* – forms a rough and irregular surface, but without sharp, jagged points.
- *Hackly fracture* – forms a surface with sharp and jagged elevations and corresponding pits.
- *Splintery fracture* (abbreviated "splint") – produces elongated spikes, usually in fibrous minerals.
- *Earthy fracture* – is the fracture formed in extremely fine-grained aggregates, as in kaolinite and chalk.

Cleavage (abbreviated "Cl.") is the breaking or separating of a mineral along one or more sets of planes which are parallel to definite crystallographic directions. Minerals like mica, galena, and calcite, which cleave along smooth lustrous planes, are said to have perfect cleavage. Minerals with good to perfect cleavage tend to show cleavage surfaces at the expense of fracture surfaces in fine sizes. Some minerals, like graphite and the micas, have one cleavage in one direction only. Others, like the amphiboles and the pyroxenes, have one cleavage parallel to the faces of their normal prism and hence in two directions, intersecting at acute and obtuse angles. Still others, like galena and calcite, have one cleavage in three directions. In each of these cases the cleavage faces are equally smooth and lustrous. Some minerals have more than one cleavage, in which case one cleavage is more perfect than the others. If a mineral has more than one cleavage, only the major one will be mentioned except in special cases.

When present, cleavage is a very important diagnostic property, not only by its geometry and perfection but also because cleavage planes in transparent minerals often carry a luster which is different from that of the rest of the mineral. Indications of cleavage should be looked for carefully.

The **streak** of a mineral is the color of its finest powder or of the mark it makes on unglazed porcelain. The powder can be observed through the microscope by crushing one or more grains of a mineral to a fine flour with a stiff narrow blade or spatula or between microscope slides. Mineral grains coarser than 100 mesh can often be drawn across unglazed porcelain with a very fine-pointed forceps to produce a mark observable through the microscope; with practice even finer grains of some minerals may be streaked. In many cases, the streak of a mineral shows little or no variation and, especially for minerals with a wide range of colors such as like calcite and sphalerite, it is far more characteristic than the color of a coarser grain.

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Table 3-1 Minerals with metallic and submetallic luster*

Name & Composition	H	sp. gr.	Luster	Color
Argentite/Acanthite Ag_2S	2.0-2.5	7.2-7.4	Met	Dark lead-gray
Arsenopyrite FeAsS	5.5-6.0	6.0	Met	Silver-white to steel-gray
Bismuthinite Bi_2S_3	2.0-2.5	6.8	Met	Light lead-gray, often with yellow tarnish
Bornite Cu_5FeS_4	3.0-3.25	5.1	Met	Coppery pink to pinkish bronze
Boulangerite $Pb_5Sb_4S_{11}$	2.5-3	6.2	Met	Bluish lead-gray; may have yel. spots due to oxidation
Bournonite $PbCuSbS_3$	2.5-3	5.8	Met to dull	Steel-gray to dark lead-gray
Calaverite $AuTe_2$	2.5-3	9.1-9.4	Met	Pale brass-yellow to silver-white
Chalcocite Cu_2S	2.5-3	5.5-5.8	Met	Dark lead-gray
Chalcopyrite $CuFeS_2$	3.5-4	4.1-4.3	Met	Brass yellow; may tarnish orange, blue, purple, black
Chromite $FeCr_2O4$	5.5	4.5-4.8	Met to submet	Iron-black to brownish black
Copper Cu	2.5-3	8.95	Met	Light coppery pink, tarnishing redder
Digenite Cu_9S_5	2.5-3.0	5.5	Sub-metallic	Blue to black
Enargite Cu_3AsS_4	3.0	4.45	Met	Grayish black to iron-black
Galena PbS	2.5-2.8	7.58	Met	Lead-gray
Gold Au	2.5-3	19.3	Met	Rich golden yellow, whiter than high silver

*Mostly opaque, even in very thin splinters; all except graphite have specific gravites above 4.0

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Abbreviations: d. = dark sl. = slightly irid. = iridescent
l. = light cl. = cleavage

Fracture	Remarks
Uneven, subconch	Both forms very sectile. Fresh surfaces darken under strong light. Streak dark lead-gray.
Uneven	Granular, compact; crystals columnar with diamond x-section. Brittle. Streak grayish black.
	Slightly sectile; massive, columnar to fibrous; perfect cl. parallel length, 2 other poorer cleavages. Streak dark lead-gray.
Uneven	Tarnishes quickly to iridescent blues and purples. Brittle. Streak grayish black.
	Columnar to fibrous or plumose; good cl. parallel length. Brittle, but thin fibers flexible. Streak brownish gray to brown.
Subconch, uneven	Massive, compact; crystals short columnar or tabular. Rather brittle. Streak dark gray to black.
Subconch, uneven	Bladed to lathlike, columnar. Also massive. Very brittle. Streak yellowish to greenish gray.
Conchoidal	Usually compact massive. Rather brittle; slightly sectile. May be sooty or powdery. Streak dark lead-gray.
Uneven	Usually compact massive. Brittle. Streak greenish-black.
Uneven	Usually massive. Brittle. May be feebly magnetic. Translucent in thin splinters. Streak brown. Forms two series with Magnesiochromite ($MgCr_2O_4$) and Hercynite ($FeAl_2O_4$)
Hackly	Very ductile and malleable.
Conchoidal	Often mistaken for chalcocite. Usually massive and granular.
Uneven	Perf. cl, in 2 directions at 82° and 98°. Brittle. Streak grayish black. Tarnishes dull.
Subconch	Easy and highly perf. cl. in 3 mutually perpendicular directions. Massive cleavable to fine granular. Streak lead-gray.
Hackly	Very ductile and malleable. Often in flakes and flattened grains.

(continued on next page)

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Table 3-1 Minerals with metallic and submetallic luster* (continued)

Name & Composition	H	sp. gr.	Luster	Color
Graphite C	1.0-2.0	2.09-2.2	Met to dull	Steel-gray to iron-black
Hematite Fe_2O_3 (also in Table 2-3)	5.0-6.0	5.26	Met to submet to dull	Steel-gray (cryst); reddish brown to red (earthy to dull compact material)
Ilmenite $FeTiO_3$	5.0-6.0	4.72	Met to submet	Iron-black
Jamesonite $Pb_4FeSb_6S_{14}$	2.5	5.6	Met	Grayish black, may tarnish iridescent
Linnaeite Co_3S_4	4.5-5.5	4.5-4.8	Met	Light gray, easily tarnished
Luzonite Series Cu_3 (As,Sb) S_4	3.5	4.4	Met	Gray, often with coppery tint
Magnetite Fe_3O_4	5.5-6.5	4.9-5.2	Met	Black
Marcasite FeS_2	6.0-6.5	4.9	Met	Pale brass-yellow to nearly white
Millerite NiS	3.0-3.5	5.5	Met	Pale brass-yellow
Molybdenite MoS_2	1.0-1.5	4.6-4.7	Met	Bluish lead-gray
Pentlandite $(Fe,Ni)_9$ S_8	3.5-4.0	4.6-5.0	Met	Pale bronze yellow
Pyrite FeS_2	6.0-6.5	4.8-5.0	Met	Pale brass-yellow, may tarnish iridescent
Pyrolusite MnO_2				
Crystals:	6.0-6.5	5.1	Met	Light steel- or iron-gray
Massive:	2.0-6.0	4.4-5.0	Met to submet	Dark, sometimes bluish-gray or iron black
Pyrrhotite $Fe_{1-x}S$ (x = 0 to 1.7)	3.5-4.5	4.6-4.7	Met	Yellowish to brownish bronze, may tarnish

*Mostly opaque, even in very thin splinters; all except graphite have specific gravites above 4.0

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Abbreviations: d. = dark sl. = slightly irid. = iridescent
l. = light cl. = cleavage

Fracture	Remarks
	Foliated, scaly, granular, earthy. Perf. and easy cl. in 1 direction. Sectile. Flakes flexible. Streak black to dark gray.
Subconch to uneven	Crystals brittle, elastic in thin flakes. Flakes may be translucent or show red internal reflections. Streak red to reddish brown.
Conch to subconch	Tabular to platy; also massive. Brittle. Streak black.
	Fibrous to columnar; also in felted masses of needles. Good cl. pendicular to length. Brittle. Streak grayish black.
Uneven to subconch	Massive, compact; also in octahedra.
Uneven	Usually massive, granular. Brittle. Tarnishes dull. Streak grayish black. Dimorphous with Enargite.
Conchoidal, uneven	Massive and in octahedra. Strongly magnetic. Brittle. Streak black. Oxidizes to hematite and limonite.
Uneven	Compact, stalactitic, radiating, rounded; also spearhead forms. Brittle. Streak grayish to brownish black.
Uneven	Massive, compact, tufted; also in slender to capillary crystals. Brittle. Streak greenish black.
Uneven	Perf. cleavage in 1 direction. Sectile. Laminae flexible but not elastic. Streak greenish gray.
Conch	Massive, granular. Brittle. Non-magnetic but usually assoc. with pyrrhotite. Streak bronze-brown.
Conchoidal, uneven	Usually massive; also in cubes, octahedra, pyritohedra. Brittle. Streak greenish to brownish black.
Splintery	Columnar to fibrous. Brittle. Streak black or bluish black.
Uneven	Granular to powdery massive; sooty. Streak black or bluish black. Also concentrically banded.
Uneven, subconch	Usually massive, granular. Magnetic, much less than magnetite. Brittle. Streak dark grayish black.

(continued on next page)

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Table 3-1 Minerals with metallic and submetallic luster* (continued)

Name & Composition	H	sp. gr.	Luster	Color
Siegenite $(Ni,Co)_3 S_4$	4.5-5.5	4.5-4.8	Met	Light gray, easily tarnished
Silver Ag	2.5-3.0	10.1-11.1	Met	Silver-white to; grey to black tarnish
Skutterudite series (Co,Ni,Fe) As_3	5.5-6	6.5	Met	Tin-white to silvery gray
Stibnite Sb_2S_3	2.0	4.6	Met	Lead-gray to steel-gray
Tetrahedrite-Tennantite $(Cu,Fe)_{12}(Sb,As)_4S_{12}$	3.0-4.5	4.6-5.1	Met	Iron black to gray

*Mostly opaque, even in very thin splinters; all except graphite have specific gravites above 4.0

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Abbreviations: d. = dark sl. = slightly irid. = iridescent
l. = light cl. = cleavage

Fracture	Remarks
Uneven, subconch	Massive compact; also in octahedra.
Hackly	Ductile and malleable. In scales, wires, and branching forms.
Conchoidal, uneven	Dense to granular massive; also in cubes and octahedra. Brittle. Streak grayish black.
Subconch	Columnar to acicular; also in radiating groups, massive. Perf. cleavage parallel length. Slightly sectile. Flexible. Crystals often bent or twisted. Streak lead gray.
Subconch, uneven	Massive compact; also in tetrahedra. May show red internal reflections. Streak black to brown, to cherry-red in high As members. Tetrahedrite also forms a series with Freibergite.

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Table 3-2 Minerals with non-metallic lusters and specific gravities below 2.95*

Name & Composition	H	sp. gr.	Luster
Beryl $Be_3Al_2Si_6O_{18}$	7.5-8.0	2.6-2.9	Vitreous to resinous
Brucite $Mg(OH)_2$	2.0-2.5	2.4	Waxy to vitreous. Pearly on cleavage.
Calcite $CaCO_3$ (May have some (Mg,Fe,Mn)	3.0	2.7	Vitreous to dull. Pearly on some cleavages.
Chrysocolla $(Cu,Al)_2H_2Si_2O_5(OH)_4.nH_2O$	2.0-4.0	1.93-2.4	Vitreous, greasy, dull
Chrysotile $Mg_3\ Si_2O_5\ (OH)_4$	2.5	2.55	Silky
Collophane (Cryptocrystalline variety of apatite; see Table 2-3)	3.0-4.0	2.5-2.9	Dull to subresinous
Dolomite $CaMg\ (CO_3)_2$	3.5-4.0	2.85	Vitreous, pearly
Feldspar Group (K,Na,Ca) Al silicates	6.0-6.5	2.5-2.9	Vitreous, pearly
Gibbsite $Al\ (OH)_3$	2.5-3.5	ca. 2.4	Vitreous, dull; pearly on cl. surfaces
Gypsum $CaSO_4 \bullet 2H_2O$	2.0	2.3	Subvit. pearly, silky
Halite NaCl	2.0	2.1-2.2	Vitreous
Kaolinite $Al_2Si_2O_5(OH)_4$ (Use electron microscope or x-ray diffraction to distinguish from montmorillonite and other clay minerals)	2.0-2.5	2.61-2.68	Dull

*Mostly transparent or translucent in thin splinters, but many very fine-rained varieties appear opaque, even in -200 mesh grains

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Abbreviations: d. = dark sl. = slightly irid. = iridescent
l. = light cl. = cleavage

Fracture	Remarks
Uneven, conch	Brittle. Streak white. Hexagonal columns; granular, massive. Wide variety of usually pale colors.
Conchoidal	Foliated, fibrous, rarely granular. Perf. cl. in 1 direction. Folia flexible. White to pale green or gray. Streak white.
Conch (cl dominant)	Usually in cleavage fragments or fine granular to earthy massive. Perf. cl. in 3 directions at 75° and 105°. Streak white to grayish. Efferv. in cold dilute acids.
Conchoidal	Massive, compact, earthy, fibrous, encrusting. L. green, bluish green, turquoise-blue. Rather sectile; translucent varieties brittle. Streak white when pure.
Splintery	Bundles of parallel fibers. Flexible. White, greenish to yellowish white, pale olive green. Streak white.
Subconch, uneven	Massive hornlike or opaline; may show fossil fragments, micro-banding. Grayish to yellowish white; rarely brown. Streak white.
Conchoidal	Fine granular or in cl. fragments. Perf. cl. in 3 directions at 74° and 106°. Brittle. Often some shade of pink; also white, gray, l. brown. Streak white. Powder efferv. weakly in cold dilute acids.
Subconch, uneven	2 cleavages at or near 90°. Brittle. Usually pale colors. Na-Ca feldspars may show play of color; parallel, closely spaced twin striations. Streaks white or uncolored.
	Usually compact, earthy; fibrous. Crystals tabular, with cl. in 1 direction. White and shades of white.
Conchoidal, splintery	Granular, foliated, fibrous, earthy. 1 perf. cleavage; flakes flexible. 2 other cleavages make flattened rhombic fragments. Colorless; also white, gray, yellowish, brownish when massive. Streak white.
Conchoidal	Granular, cleavable, compact. Perf. cl. in 3 directions at 90°. Brittle. Colorless to faintly tinted. Water-soluble. Crystals cubes, rarely octahedra. Streak white.
Earthy	Earthy aggregates of very fine platelets; rarely in crystals of stacked platelets. Friable. Usually white; may be tinted or stained. Smooth feel.

(continued on next page)

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Table 3-2 Minerals with non-metallic lusters and specific gravities below 2.95* (continued)

Name & Composition	H	sp. gr.	Luster
Mica Group Complex K,Mg,Na,Fe,Al, Li silicates	2.0-3.0	2.7-3.3**	Pearly, vitreous
Montmorillonite*** Hydrated Ca.Mg.Al silicate. (x-ray diffraction usually needed for positive identification)	1.0-2.0	2.3-3.0	Dull
Quartz SiO_2	7.0	2.65	Vitreous
Sulfur S	2.0	2.0-2.1	Resinous, greasy
Sylvite KCl	2-2.5	1.9-2.0	Vitreous
Talc $Mg_3Si_4O_{10}(OH)_2$	1.0	2.6-2.8	Pearly, greasy
Tremolite $Ca_2Mg_5Si_8O_{22}(OH)_2$ (Low-Fe member of actinolite series)	5.0-6.0	3.0	Vitreous pearly, silky

*Mostly transparent or translucent in thin splinters, but many very fine-rained varieties appear opaque, even in -200 mesh grains

**Only biotite ranges above 2.95. See biotite in Table 3-3.

***This refers to montmorillonite species proper, not the Montmorillonite Group

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Abbreviations: d. = dark sl. = slightly irid. = iridescent
l. = light cl. = cleavage

Fracture	Remarks
	Foliated, flaky. Perf. cl. parallel flakes. Flakes tough, elastic. All but biotite are colorless or light-colored in thin flakes. Streak white. Sericite is very fine-grained muscovite in aggregates.
	Earthy, waxy, or porcellanic aggregates. White, pink, buffer stained. Friable when dry.
Conchoidal	Granular, compact; columnar hexagonal crystals with pointed terminations. Fine powder white. No cleavage. Colorless, white, pale rose, pale violet.
Conchoidal, uneven	Granular, fibrous, compact, earthy. Rather brittle. Shades of yellow, greenish, reddish, or yellowish gray. Streak white.
Uneven	Granular, compact; cubic crystals. Perf. cleavage in 3 directions at 90°. Colorless, white, blue, gray, orange. Water soluble; becomes damp in moist air.
Uneven	Foliated, granular, fibrous, compact. Perf. cleavage in 1 direction. C1. flakes flexible. Pale green, pale gray, white. Streak white.
Uneven, splintery	Bladed, columnar, fibrous, asbestiform. Brittle. Perf. cl. in 2 directions at 56° and 124° parallel length. White to gray. Streak white.

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Table 3-3 Minerals with non-metallic lusters and specific gravities above 2.95*
(including a few with submetallic lusters or lusters ranging from metallic to dull)

Name & Composition	H	sp. gr.	Luster
Actinolite $Ca_2 (Mg,Fe)_2Si_8O_{22} (OH)_2$ (An amphibole, grading into tremolite with decreasing Fe)	5.0-6.0	3.0-3.2	Vitreous, pearly silky
Anhydrite $CaSO_4$	3.0-3.5	3.0	Vitreous, pearly
Apatite $Ca_5 (PO_4)_3 (OH,F,Cl)$ (Collophane is a crypto-crystalline variety; Table 2-2)	5.0	3.1-3.4	Vitreous to greasy
Azurite $Cu_3 (OH)_2 (CO_3)_2$	3.5-4.0	3.77	Vitreous
Barite $BaSO_4$	3.0-3.5	4.5	Vitreous inclining to resinous
Biotite $K(Mg,Fe)_3(Al,Fe)Si_3O_{10}(OH,F)_2$	2.5-3.0	2.7-3.3	Vitreous to submet; pearly on cl.
Cassiterite SnO_2	6.0- 7.0	6.6-7.1	Adamant to sl. greasy
Cerargyrite (also called Chlorargyrite) AgCl	1.5-2.5	5.5-5.6	Resinous to adamantine
Cerussite $PbCO_3$	3.0-3.5	6.55	Adamantine to vitreous, or resinous
Cinnabar HgS	2.0-2.5	8.09	Adamantine to dull
Columbite-Tantalite Series $(Fe,Mn,Mg) (Nb,Ta)_2O_6$	6.0-6.5	5.0-7.95	Submetallic, greasy, dull

*Mostly transparent or translucent in at least thin splinters, but many fine-grained varieties appear opaque, even in -100 mesh sizes

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Abbreviations: d. = dark sl. = slightly irid. = iridescent
l. = light cl. = cleavage

Fracture	Remarks
Uneven, splintery	Bladed to acicular to fibrous. Brittle. Pale to dark green. CI. in 2 directions parallel length at 56° and 124°. Streak paler than body color.
Uneven, splintery	Granular, fibrous, cleavable. Brittle 3 cleavages at 90°; l perf. with pearly luster, 2 less perf. Colorless to bluish or brownish gray. Streak white or grayish white.
Conchoidal, uneven	Granular, compact; also in columnar hex. crystals. Green, blue, aquamarine, white, colorless. Streak white. Brittle.
Conchoidal	Usually complex crystalline; also earthy. Brittle. Light to dark blue. Streak blue, lighter than body color.
Uneven	Tabular to columnar crystals; also massive, laminated, earthy. Brittle. l perf. and 2 minor cleavages at 90°. White, gray, pale yellow, brownish. Streak white.
	Foliated; massive scaly aggregates. Perf. cl. in 1 direction. Flakes elastic. Black, green, brown, even thinnest scales usually colored.
Subconch, uneven	Massive, columnar, fibrous. Brittle. Usually yellow to reddish brown; also brownish black and opaque. Streak white, gray, brown.
Uneven	Sectile, ductile, and very plastic; waxy. Usually gray, becoming purple on exposure to strong light. Mostly massive. May have other minerals adhering.
Conchoidal	Massive, compact, earthy; tabular. Very brittle. Colorless, white, gray. Streak colorless, white. Effervesces in dilute $HN0_3$.
Uneven, subconch	Rhombohedral tabular and columnar crystals; also earthy. Perf. cl. in 2 directions at 60° and 120°. SI. sectile. Scarlet to brownish red and lead-gray. Streak scarlet.
Subconch, uneven	Stout columnar, equant, massive. Grayish and brownish black, may tarnish irid. High Mn varieties may show reddish brown internal reflections. Transparent in thin splinters. Streak dark red to black.

(continued on next page)

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Table 3-3 Minerals with non-metallic lusters and specific gravities above 2.95*
(including a few with submetallic lusters or lusters ranging from metallic to dull)
(continued)

Name & Composition	H	sp. gr.	Luster
Corundum Al_2O_3	9.0	4.0-4.1	Adamant to vitreous
Covellite CuS	1.5-2.0	4.6-4.8	Submet to dull
Crocidolite (asbestos form of Riebeckite) $Na_2Fe_5Si_8O_{22}$	5.0	3.0-3.4	Silky, dull
Cryptomelane KMn_8O_{16}	6.0-6.5	ca. 4.3	Submet to dull
Cuprite Cu_2O	3.5-4.0	6.0	Adamant, submet, earthy
Ferberite $FeWO_3$ (High-Fe member of Wolframite series)	4.0-4.5	7.5	Metallic-adamant
Fluorite CaF_2	4.0	3.18	Vitreous
Garnet Group $A_3B_2(SiO_4)_3$ Where A = Ca,Mg,Fe,Mn and B = Al,Fe,Cr, Mn	6.5-7.5	3.5-4.3	Vitreous, resinous
Goethite FeO(OH) (see Limonite below)	5.0-5.5	3.3-4.3	Silky, dull, adamant-metallic
Hematite Fe_2O_3 (See also Table 2-1)	5.0-6.0	5.26	Metallic to submet, to dull
Hornblende Complex Ca,Mg,Fe,Al silicate (an amphibole)	5.0-6.0	2.9-3.45	Submet, vitreous, pearly

*Mostly transparent or translucent in at least thin splinters, but many fine-grained varieties appear opaque, even in -100 mesh sizes

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Abbreviations: d. = dark sl. = slightly irid. = iridescent
l. = light cl. = cleavage

Fracture	Remarks
Uneven, conch	Stout columnar to barrel-shaped crystals; in rounded grains; massive granular. Brittle. Usually grayish, but many other colors, sometimes gem quality.
Uneven	Massive or spheroidal; rarely in hex. plates. Perfect cl. in 1 direction. Luster slightly pearly on cleavage surfaces. Streak lead-gray to black.
	Finely fibrous. Blue to bluish gray, leek-green, lavender. An amphibole. Forms a series with magnesioriebickite.
Conchoidal	Fine-grained compact masses; concentrically banded spheroids; cleavable masses. Steel gray to black. Apparent hardness may be as low as 1 in fibrous and cleavable masses.
Conchoidal, uneven	Massive, granular, earthy. Also in octahedra, cubes (often elongated). Brittle. Shades of red to nearly black. Streak brownish, red, shining.
Uneven	Columnar to bladed groups; massive. Perf. cl. in 1 direction. Black. Weakly magnetic. Streak brownish black to black.
Uneven	Granular, massive earthy. Perf. cl. in 4 directions at 70-1/2° and 109-1/2°. Brittle. Usually colorless, white, or pale green, blue, purple, yellow.
Conchoidal, uneven	Complete crystals dodecahedral or trapezohedral; also granular, lamellar, compact. Usually red, pink, yellow, white, or brown. No cl. but may have parting at 60° and 90°. Streak white. For details on individual species, see texts.
Uneven	Massive, fibrous, columnar; earthy to ocherous. Crystals blackish brown; brittle. Massive varieties yellowish to reddish brown. Earthy varieties brownish yellow. May form pseudomorphs after pyrite. Streak brownish to orangish yellow.
Subconch, uneven	Crystals steel gray, brittle. Flakes may be translucent or show red internal reflections. May form pseudomorphs after pyrite, magnetite. Streak red to reddish brown.
Uneven, splintery	Columnar to fibrous. Perf. cl. in 2 directions parallel length, at 56° and 124°. Brittle. Dark green, black, brown. Translucent in thin splinters. Streak paler than body color.

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Table 3-3 Minerals with non-metallic lusters and specific gravities above 2.95*
(including a few with submetallic lusters or lusters ranging from metallic to dull)
(continued)

Name & Composition	H	sp. gr.	Luster
Huebnerite $MnWO_4$ (High Mn member of Wolframite series)	4.0-4.5	7.12	Submet, resinous
Kyanite Al_2SiO_5	4.5 lengthwise 6.5 crosswise	3.5-3.7	Vitreous to pearly
Limonite A mixture of hydrated iron oxides.	4.0-5.5	2.9-4.3	Vitreous to dull
Magnesite $MgCO_3$	4.0-4.5	2.98-3.4	Vitreous to dull
Malachite $Cu_2CO_3(OH)_2$	3.5-4.0	3.6-4.1	Adamant to vitreous; silky dull
Monazite Rare earth phosphate	5.0-5.5	4.6-5.7	Resinous, waxy vitreous
Orpiment As_2S_3	1.5-2.0	3.49	Resinous to greasy; pearly
Psilomelane Hydrated Ba-bearing manganese mineral - mainly Romanechite.	5.0-6,0	4.4-4.7	Submet to dull
Pyromorphite $Pb_5 (PO_4)_3 Cl$	3.5-4.0	6.5- 7.0	Resinous to greasy
Pyroxene Group Complex Ca,Mg,Fe,Mn,Al Silicates, some with Na, Ti	5.0-6.5	3.0-3.96	Vitreous, pearly, dull; some submet
Realgar AsS	1.5-2.0	3.5-3.6	Resinous to greasy, dull
Rhodochrosite $MnCO_3$	3.5-4.0	3.4-3.6	Vitreous to pearly

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Abbreviations: d. = dark sl. = slightly irid. = iridescent
l. = light cl. = cleavage

Fracture	Remarks
Uneven	Columnar, in radiating or parallel groups. Yellowish to reddish brown, rarely brownish black. Perf. cl. in 1 direction parallel length. Streak yellow to reddish brown.
Splintery	Bladed to columnar. 2 lengthwise cleavages at 74° and 106°. Usually white to blue, gradational. Rarely pale green. Streak white.
Uneven, earthy	Very brittle in vitreous forms. Compact, earthy, ocherous. Yellowish to reddish brown to brownish black. May be pseudomorphous after pyrite, siderite. Streak yellowish to reddish brown.
Conchoidal	Granular, cleavable, compact like unglazed porcelain. Usually light-colored. Effervesces in hot dilute HCl. Streak nearly brown.
Uneven, subconch, splintery	Massive, fibrous, concentrically banded. l perfect cleavage. l. to d. green to blackish green. Efferv. in cold dilute acids. Streak pale green.
Conchoidal, uneven	In sands, usually well rounded. Brittle. 2 cls. at 90°. Yellow, yellowish to reddish brown. Streak white or faintly colored.
	Granular, foliated. 1 perf. cleavage. Cleavage lamellae flexible, show pearly luster. Lemon to golden and brownish yellow. Streak pale lemon-yellow.
	Often in concentric layers in rounded particles. Black. Streak black. In some specimens apparent H is down to 2. X-ray diffraction needed to distinguish from cryptomelane.
Subconch to uneven	Crystals hex. prisms, often with hollow ends, or barrel-shaped. Granular, subcolumnar. Usually green, olive green, yellow, brown. Streak white,
Uneven	Massive, granular, lamellar, fibrous. Cl. in 2 directions near 90°. Brittle. Shades of gray, yellow, green, and brown. Streak grayish.
Conchoidal	Granular, compact, encrusting. Sectile. Transparent when fresh. Cleavage in 1 direction. Red to orange-yellow. Streak orange-red.
Uneven	Granular to compact. 1 perf. cl. in 3 directions at 73° and 107°. Brittle. Usually in shades of pink to rose-red and reddish brown. Effervesces in hot dilute acids.

(continued on next page)

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Table 3-3 Minerals with non-metallic lusters and specific gravities above 2.95*
(including a few with submetallic lusters or lusters ranging from metallic to dull)
(continued)

Name & Composition	H	sp. gr.	Luster
Ruby Silver **Proustite** (Ag_3AsS_3) and **Pyrargyrite** (Ag_3SbS_3)	2.0-2.5	5.5-5.9	Adamant
Rutile TiO_2	6.0-6.5	4.2-4.6	Metallic-adamant
Scheelite $CaWO_4$	4.5-5.0	5.9-6.10	Vitreous to adamant
Siderite $FeCO_3$	3.5-4.5	3.8-4.0	Vitreous to pearly, dull
Sillimanite Al_2SiO_5	6.0-7.5	3.2-3.3	Vitreous, silky
Smithsonite $ZnCO_3$	4.0-4.5	4.3-4.5	Vitreous, pearly
Sphalerite (Zn,Fe) S	3.5-4.0	3.9-4.1	Resinous to adamant
Spodumene LiAl Si_2O_6	6.5-7.0	3.0-3.2	Vitreous, pearly, dull
Tremolite $Ca_2Mg_5Si_8O_{22}$ $(OH)_2$ (Low-Fe member of actinolite series)	5.0-6.0	2.9-3.1	Vitreous pearly, silky
Uraninite UO_2	5.0-6.0	6.5-10.6	Submet, pitchlike to dull
Willemite Zn_2SiO_4	5.0-6.0	3.9-4.2	Weak vitreous to resinous
Wolframite (Fe,Mn) WO_4 (series between Huebnerite and Ferberite)	4.0-4.5	7.0-7.5	Metallic-adamant

*Mostly transparent or translucent in at least thin splinters, but many fine-grained varieties appear opaque, even in -100 mesh sizes

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Abbreviations: d. = dark sl. = slightly irid. = iridescent
l. = light cl. = cleavage

Fracture	Remarks
Conchoidal, uneven	Rhombohedral cl. distinct. May show red internal reflections. Scarlet to deep red or brownish red. Streak scarlet to purplish red.
Conchoidal, uneven	Usually slender columnar to acicular. Brittle. Usually reddish brown, red, black. Streak pale brown to yellowish.
Uneven to subconch	Massive, granular. Brittle. Usually white, yellowish or brownish white. Fluoresces blue-white in short U.V. radiation. Streak white.
Conchoidal, uneven	Granular, cleavable, compact. Pert. cl. in 3 directions at 73° and 107°. Usually grayish and yellowish brown to brown and reddish brown. Effervesces in hot dilute acids. Streak white.
Splintery, uneven	Fibrous, columnar. Lengthwise cleavage in 2 directions at 88° and 92°. Brittle. Light brown, grayish brown, near-white, rarely pale green. Streak white.
Uneven, splintery	Granular to compact; earthy and friable. Perf. cl. in 3 directions at 72° and 108°. Brittle. Shades of gray, greenish to brownish white, yellow. Effervesces in cold dilute acids. Streak white.
Conchoidal	Perf. cleavage in 6 directions at 60°. Cleavable masses; granular, fibrous, cryptocrystalline. Brown, black, red, yellow, rarely green, white to nearly colorless. Streak brownish yellow to white.
Splintery, uneven	Cleavable, compact, columnar. Cleavage in 2 directions at 87° and 93°. Greenish, grayish, and yellowish white; rarely pale green or purple. Streak white.
Uneven, splintery	Bladed, columnar, fibrous, asbestiform. Brittle. Perf. cl. in 2 directions at 56° and 124° parallel length. White to gray. Streak white.
Conchoidal, uneven	Massive, granular; cubic and octahedral crystals. Brittle. Steely to velvety and brownish black. Colloform varieties (pitchblende) may show banding. Streak brownish black, grayish.
Conchoidal, uneven	Columnar, massive, granular. Brittle. Greenish yellow, apple green, flesh red, grayish white, brown. Streak white or faintly colored.
Uneven	Columnar, lamellar, massive; granular. Dark grayish to brownish black. Brittle. 1 pert cl. parallel length. May be slightly magnetic. Streak reddish brown to black.

(continued on next page)

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Table 3-3 Minerals with non-metallic lusters and specific gravities above 2.95*
(including a few with submetallic lusters or lusters ranging from metallic to dull)
(continued)

Name & Composition	H	sp. gr.	Luster
Zincite (Zn,Mn)O	4.0	5.4-5.7	Subadamant
Zircon $ZrSiO_4$	7.5	4.5-4.7	Adamant

*Mostly transparent or translucent in at least thin splinters, but many fine-grained varieties appear opaque, even in -100 mesh sizes

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Abbreviations: d. = dark sl. = slightly irid. = iridescent
l. = light cl. = cleavage

Fracture	Remarks
Conchoidal	Massive, foliated, compact, granular. Cleavage in 1 direction. Orange-yellow to deep red. Streak orange-yellow.
Uneven	Crystals square prisms with pointed ends. Commonly shades of brown, also colorless and orange. Brittle. Sometimes cloudy from its own radioactivity. Streak white.

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3.2 Mineral surface analysis

Many separations in minerals processing are based on modifications of surfaces of minerals using chemicals. The success of such separations depends entirely upon the nature and composition of mineral surfaces involved and how the chemicals are interacting with those surfaces. In this context, the bulk phase composition might often be almost irrelevant. For example, the success of a flotation separation depends upon the surface composition of minerals that are targeted for either flotation or depression. Even if the best possible collector reagent is designed for a given value mineral, it can fail to perform if under a given set of pulp conditions either the value mineral surfaces are not optimal for reagent adsorption or the gangue mineral surfaces favor reagent adsorption. The converse applies for depressants and activators. An understanding of the composition of mineral species under process conditions and the mechanism of interactions of reagents with mineral surfaces is of great importance in reagent design/selection and the optimization of mineral separation processes.

Significant efforts have been made in the past to obtain knowledge of mineral surface composition, and numerous techniques have been investigated. Until three decades ago most of these techniques provided only indirect information about mineral surface composition. **Infrared spectroscopy** was perhaps the most successful technique until the advent of **X-ray Photoelectron spectroscopy (XPS)** and related **electron spectroscopy** (or vacuum) techniques. Although the vacuum techniques (typically using ~10^{-10} torr) are *ex-situ*, one of the major advantages is the ability to analyze individual mineral particles from a complex mixture containing a variety of mineral grains, such as those from actual plant flotation streams.

Infrared spectroscopy (IR) had been the workhorse in studying mineral-reagent interactions until early 1970s. It can be performed in transmission, reflection and emission modes. Transmission mode is the simplest, but it is an *ex-situ* technique. A small amount of the sample in the form of a fine powder is worked into KBr pellets or Nujol and this mixture is then pressed to form a thin disk. Information on mineral-reagent interactions can be obtained by monitoring changes – such as peak shifts or formation/disappearance of peaks – in the IR spectrum before and after reagent adsorption. The main advantage is that information on identity of adsorbed species and molecular bonding is obtained. Also IR technique can be quantitative. Major disadvantages are (a) presence of any water masks many important peaks; (b) the low sensitivity of IR requires the use of reagent concentrations that far exceed those of

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relevance in flotation; (c) only very fine powders can be used. These disadvantages can be overcome to a large extent in IR spectroscopy used in the reflection mode. The most commonly used technique is the **Attenuated Total Reflection (ATR)**. The sample is placed in contact with a large crystal (such a Ge, TlBr/TlI, or AgCl) whose refractive index is higher than that of the sample. The radiation is oriented on the crystal such that total reflection occurs at the crystal-sample interface and, therefore, information is obtained from the surface layers (typically < 2µm). The major advantage is that it can be used in the presence of water thereby making this an *in-situ* technique. Another technique in the reflection mode is **DRIFT**, which analyzes diffuse reflectance. Sensitivity is, however, still fairly low. Many of the major drawbacks of conventional IR have been overcome in the **Fourier Transform Infrared Spectroscopy (FTIR)**, which uses interferometers and a laser source. Sensitivity is improved significantly (at least two orders of magnitude), as also accuracy and reproducibility in wavelength determination.

Raman Spectroscopy is potentially a useful technique in aqueous systems to study mineral-reagent interactions *in-situ*. It uses an intense laser beam to induce Raman scattering and, consequently, traces of impurities or the sample itself emit fluorescent background irradiation upon which the very weak Raman spectrum is superimposed. This presents a serious obstacle to Raman measurements. The laser source often destroys the adsorbed species or causes chemical changes. The possibility of using resonance Raman or Surface-enhanced Raman has been considered, but these are limited to certain unique systems only.

Nuclear Magnetic Resonance (NMR) can, in theory, provide information about the chemical environment of the nuclei in the adsorbed molecules and how this is affected by the adsorption process and molecular dynamics in the adsorbed layer. It is also an *in-situ* technique. Unfortunately the poor sensitivity of the technique has prevented its use in flotation systems.

Two important *in-situ* techniques that use molecular probes to investigate chemical environment and molecular dynamics at solid-solution interfaces are **Fluorescence spectroscopy and Electron Spin Resonance (ESR) spectroscopy**. In the former a fluorescent label (or a dye) is used either as an independent probe or attached to the adsorbing molecule itself, whereas in the latter a spin probe is used. In theory both techniques possess reasonably good sensitivity. Extensive studies in flotation systems have been conducted using these techniques.

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Fluorescence spectroscopy is a well-developed technique for investigating the formation of hydrophobic domains in solution and at solid-liquid interfaces. In this study, probes such as pyrene and dansyl are used. Pyrene and dansyl can both be attached to the adsorbing molecules. Pyrene fluorescence can also be used as an independent probe. Through monitoring the ratios of intensities of two characteristic peaks (pyrene) or the shift of specific peak (dansyl), both probes give information on the hydrophobic domain formation that helps to develop the adsorption mechanism, particularly the role of hydrophobic force in causing adsorption. The techniques can also provide valuable information about conformation of adsorbed polymers.

In ESR, a study of the electron spin and associated magnetic moment are measured in the presence of a magnetic field. Only molecular species possessing an unpaired electron (e.g. transition metal ions, free radicals, defect centers etc.) can be detected. ESR technique can give information on both the formation of hydrophobic domains and their nature. More importantly, it is a powerful technique that can yield information also on the orientation of the molecules, which is often the critical parameter in determining wettability or hydrophobicity of particles. The same reagent at the same adsorption density can yield hydrophobicity (or hydrophilicity) and flocculation (or dispersion), depending on the orientation of the functional groups on the molecules. Commonly used probes contain nitrosyl (or nitroxide) groups. The major disadvantage is that most common collectors and other flotation reagents do not possess unpaired electrons, which necessitates the introduction of spin probes. The underlying assumption is that the spin probe itself neither interacts with the mineral nor affect interaction of the molecule under study. It is not certain whether this condition can be met in a system as complex as that of flotation. Paramagnetic centers in flotation reagents can interfere with measurements and interpretation of spectra. Also sensitivity appears to be insufficient for the low surface areas found in flotation systems.

Mirage spectroscopy or photo-thermal deflection spectroscopy gives information on light absorbing species present as a thin layer at the surface of a less absorbing sample surface. On illumination by a pump beam at a wavelength where light is absorbed and converted exclusively to heat, the temperature of the sample increases. This heat is transmitted to the surrounding aqueous phase, leading to a decreasing gradient of temperature, and the associated gradient of refractive index, from the surface sample. The gradient of refractive index can be measured as a bending of a probing laser beam parallel to the surface of the sample. The deflection of the probing laser

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beam can be correlated to the absorbance of the adsorbed species or to its thickness if the layer is homogeneous. Measurements can be made by either changing the wavelength of the pump beam to record absorption spectrum or measuring the deflection of the beam at a fixed wavelength to obtain dynamics of the formation of adsorbed layer. The major advantages of this technique are that it is carried out *in-situ* and almost real-time measurements can be made. The major disadvantages are that the system has to be quiescent (no stirring) throughout measurements, measurements are carried out in the absence of electrolytes and that no chemical compositional information is obtained.

XPS and Auger Electron Spectroscopy (AES), which have been used extensively often with much success for the past two decades, are two of the techniques that can provide quantitative direct elemental composition of mineral surfaces and oxidation states.

In XPS, the mineral sample is irradiated with monochromatic X-ray photons, and the kinetic energy of the ejected electrons from the sample is measured with an electron energy analyzer. Binding energies of the electrons are then calculated from kinetic energies using the energy of the exciting radiation and the work function of the spectrophotometer. The binding energies are characteristic of the elements comprising the sample surface and the chemical environment of the elements in question. The sampling depth of conventional XPS is 20-30 atomic layers or less, and the surface sensitivity is dictated by the kinetic energy of the X-rays from the source (which is limited by the X-ray tube used; for ex. ~1487 eV for AlKα). A more advanced XPS technique is one where **synchrotron radiation (SR)** is used instead of X-ray tube. SR provides a wide and continuous energy spectrum thereby affording tunable, sufficiently low kinetic energies and the resultant high resolution and reasonable measurement times. By using several different excitation energies, SR-XPS provides the possibility to obtain a depth profile.

Auger Electron Spectroscopy (AES) is a non-destructive high-vacuum method of surface chemical analysis. In this technique, the mineral sample is bombarded with a beam of electrons (energy ~2000-3000 eV), which results in ejection of Auger electrons from elements in the top atomic layers of the surface. The energy of the Auger electrons (typically <2000 eV and independent of the energy of the primary electron beam), which is characteristic of its source element, is then measured. A variation of the conventional AES is the **Scanning Auger Microscopy (SAM)** which combines physical imaging of the surface, as in scanning electron microscopy, with surface chemical analysis of particles. SAM uses a focused electron beam with energies in the 5-50 keV range to cause ionization of

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core levels in surface atoms. The energy of the Auger electrons is then measured. Focus of the electron beam can be achieved down to 20 nm and scanning (or rastering) can be used as in the electron microscope.

Since their invention in the 1980s, **scanning tunneling microscopy (STM)** and **atomic force microscopy (AFM)** have become popular tools in surface science. These techniques allow observation of the topography of solid surfaces at atomic resolution in ambient environments. Both methods, however, suffer from limitations that prevent their use in studying natural mineral surfaces under flotation related conditions.

Secondary Ion Mass Spectroscopy (SIMS) is a relatively new technique in mineral surface analysis of relevance to flotation as evidenced by the limited published literature, and it offers several advantages over most other surface analytical techniques.

In the SIMS technique, a beam of energetic ions, such as those of Ga, Xe or Ar, is directed at the mineral particle surfaces under high vacuum conditions. The ion beam transfers some of its momentum to the sample surface, causing desorption of surface species as positive and negative ions (secondary ions), and neutral fragments. The secondary ions are then separated and collected according to their respective masses using a mass spectrometer. The result is a mass spectrum, similar to those obtained in conventional Mass Spectroscopy that is used for bulk phase analysis of solids, liquids and gases. SIMS by nature is a destructive technique, i.e. the surface is being continually eroded by the incident ion beam and is changing with time. By tuning and focusing the primary ion beam current, a very controlled surface depletion in the sub-monolayer range **(Static SIMS)** and in the multi-layer range **(Dynamic SIMS)** is possible for all types of materials. The low ion beam doses used in static SIMS result in minimum disruption of chemical bonds and a minimum amount of surface being removed. Thus static SIMS is necessary for analyzing surfaces containing organic species such as flotation reagents adsorbed on mineral surfaces if molecular information is desired. High ion beam doses (i.e. depleting multilayers) are used in dynamic SIMS for sputtering and depth profiling to determine whether certain species are present only on the surface (such as Cu-activated pyrite or sphalerite) or are also present in the bulk.

SIMS instrumentation is commercially available with a quadrupole mass spectrometer, a magnetic sector mass spectrometer or a time of flight (ToF) mass spectrometer. The attributes of the ToF which makes it particularly well suited for static SIMS measurements are:

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(a) high transmission, (b) unlimited mass range, (c) parallel detection (i.e. all masses are measured virtually simultaneously), (d) high mass resolution, (e) static imaging as a result of the above, and (f) the ease of charge compensation for insulators.

The unique advantages of SIMS over other techniques are: (a) high sensitivity (generally more sensitive than XPS), (b) molecular composition of surface species (not just elemental composition) which facilitates an unambiguous identification of the surface species, (c) spatial distribution of surface species (imaging or mapping), (d) shallow depth of penetration (as small as one monolayer) which is of direct relevance to flotation, (e) ability to detect both inorganic and organic species, (f) depth profile, and (g) high resolution (with the use of micro-focusing liquid metal ion guns, SIMS images with submicron resolution may be obtained). The disadvantages of SIMS are, (a) difficulty in quantifying surface species, (b) large differences in sensitivities for different surface species, and (c) possible ion-induced surface reactions under certain conditions.

Much of the pioneering work on the use of SIMS in flotation systems was conducted in Cytec's Research Laboratory. SIMS and XPS have been used in a variety of flotation systems, including plant and laboratory flotation products and pure minerals. These studies have been successful in detecting, identifying and mapping collector species on mineral surfaces, as well as in investigating metal ion activation of sulfide and gangue minerals in order to either explain or solve plant related problems.

Laser Ion Mass Spectroscopy (LIMS) is a variation of SIMS and uses two laser sources, such as Nd-YAG. The first laser, called the ablation laser, hits the sample at a 90° angle and removes (or ablate) material from the surface layers. The second laser is perpendicular to the first laser (or parallel to the sample) and is positioned about 600 microns above the sample surface. The second laser is also coupled with the first laser with delay times in the range of 700-1400 nanoseconds. The function of the second laser is to ionize the ablated neutral material from the sample surface. The ions are then focused using an electrostatic lens and analyzed by mass/charge ratio using a time-of-flight drift tube. The major advantages of LIMS are small analysis area (spot size typically in the range of 5-10 microns) and rapid analysis times (of the order of minutes). The main disadvantage is greater sampling depths (of the order 500-1000 Å) in LIMS (it is 1-3 monolayers in SIMS). Other differences between LIMS and SIMS are, (a) spatial resolution is 1-3 microns in LIMS (it is 1500 Å in SIMS), (b) imaging is not possible in LIMS and (c) organic species and polymers cannot be analyzed by LIMS.

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4 Laboratory evaluation of flotation reagents



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Section 4 Guidelines for laboratory evaluation of flotation reagents

Laboratory flotation testing is a costly and time-consuming process. The need to produce quality results and, more importantly, accurate and concise conclusions from the resources invested is vitally important. Therefore, to produce meaningful and useful data in the lab, a systematic investigation using good experimental techniques and consistent laboratory testing procedures must be followed. The information presented here is not meant to be exhaustive and should be used only as a guideline. Experience and intuition play an important role in the evaluation of a flotation process. The following procedures are discussed in this section:

- *Sampling* – samples should be representative of plant feed/ore type
- *Microscopic analysis* – to determine mineralogical associations and degree of liberation.
- *Ore preparation* – representative sub-sampling and handling of ore for flotation evaluation
- *Grinding* – to achieve desired liberation of value minerals
- *Test design* – to incorporate clear, measurable objectives. Statistical vs. traditional approach.
- *Flotation* – screening of reagents and other variables for improved metallurgical performance
- *Handling of flotation products* – sub-sampling to provide samples for assays.
- *Assaying* – to generate mass balances to evaluate flotation performance
- *Data analysis/Interpretation of results* – to determine if objectives have been met and provide direction for additional tests.

A. Sampling

When ore samples are taken directly from the mine or a stockpile, it should be borne in mind that no two ore bodies are the same, and that variations within an ore body are also common. Close consultation among the milling, mining and geology departments is essential to ensure that the sample is as representative as possible. Reproducibility of flotation test work is paramount to the evaluation of flotation reagents. Generally, the sample should be sufficiently

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large so that an entire investigation can be completed on one sample without having to re-sample the deposit.

In the case of operating plants, samples may be taken from the conveyor belt feeding coarse ore to the grinding section (e.g. rod mill feed). Samples should be taken over a sufficient period of time so that the ore will be representative of current mill feed.

When taking pulp samples it is advisable to verify that the plant is operating under normal conditions. It is recommended that fresh pulp samples be taken daily, since the ground ore is subject to aging effects. The objectives of the test work will dictate the sampling point and whether to turn off reagent additions prior to sampling.

B. Microscopy

Microscopical examination of the feed samples, which is often neglected, is essential in the design of the test program and reagent selection. The feed samples should be examined by a qualified microscopist/mineralogist, using the appropriate techniques, to identify the type and mode of occurrence of minerals and their degree of liberation from each other (see Section 3).

C. Ore preparation

Dry ore

The dried ore sample must be transported to the test laboratory as quickly as possible and preferably in a coarse state (≥1-2 cm) to keep oxidation to a minimum. The sample is then typically stage-crushed to minus 1-2 mm then split manually using a riffle or a rotary splitter to obtain flotation charges of the desired weight. The ore charges should then be sealed in plastic bags and stored in a freezer (preferably -15°C or lower) to retard oxidation/aging effects. Several randomly chosen samples should be submitted for assay to confirm that sample splitting has been conducted properly and that the samples are representative.

Pulp samples

The amount of pulp sample taken at any one time is dependent upon many factors. These include percent solids of the pulp, the size of the laboratory flotation cell, the number of flotation tests to be conducted in a particular series, and the degree to which the pulp is known to be sensitive to aging effects. Sub-sampling of the pulp into flotation charges can be done either volumetrically or, preferably, gravimetrically while the pulp is being adequately agitated. When the situation is such that the pulp has to be used for an extended test series, then the test charges should be placed in sealed containers and stored in a freezer.

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D. Grinding

Laboratory grinding tests are conducted primarily to establish the size distribution of the solids, which is dictated by the objectives of the test work.

Mesh of liberation

This is estimated by examining various screen size fractions of the ground ore (usually the coarser fractions) using reflected light microscopy. This provides information on the modes of occurrence and the degree of liberation of the desired minerals i.e. sulfide-gangue mineral associations.

When microscopical facility or expertise is not available, the optimum liberation size can be estimated from a granulometry vs. flotation recovery curve (see F).

Granulometry versus grinding time relationship

By graphically plotting the cumulative weight percent passing (or retained on) a screen size vs. the log grinding time, a relatively straight line will result between about 15% and 85% cumulative weight for that screen size. It is then a simple matter to change the grinding times during the test program in order to change the flotation feed granulometry.

Experience at Cytec favors the use of a rod mill for laboratory batch grinding to minimize tramp oversize and sliming. The pulp density for grinding is generally in the range of 60% to 70% solids, depending on the ore's pulp viscosity and the specific gravity of the dry solids.

The ground pulp should be wet screened on a 200 mesh (74 µm) or 325 mesh (44 µm) sieve and the oversize and undersize (slimes) material filtered and dried separately. The oversize is then dry-screened on a series of sieves generally from about 500 µm through 74 µm or 44 µm (depending on the original size used for the wet screening). Any material passing through the finest sieve should be added to the undersize from the wet screening operation. The weights of the various screen fractions are then used to determine the size distribution of the ground ore. Stainless steel sieves are recommended for most routine screening.

E. Test design

Prior to undertaking any extensive reagent-screening program, the objectives for such a program should be clearly defined. The variables (i.e. collector type, collector dosage, frother type, pH etc.) to be studied should be well thought out along with the levels

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of treatment to use in order to observe the desired response and to determine the relative importance of these variables. A thorough investigation of all the variables involved in a process is not practical. The variables selected for study will depend on the response under investigation as well as feedback as the investigation progresses. Variables not under investigation should be kept as constant as possible.

In some cases the traditional approach of changing one variable at a time is adequate, but in most cases an experimental design based on statistical principles is recommended. This enables the researcher to investigate the effects of several variables simultaneously. Carefully planned experiments conducted in this manner will provide more information than the traditional approach and with a smaller number of tests.

There are many references to statistical experimental designs in the literature. Cytec's field representatives have been appropriately trained in developing experimental designs and can assist the customer in this respect. For additional information, refer to Section 12.

F. Flotation testing

In designing a flotation test program, experience plays an important role in minimizing the number of variables and the range over which these variables need to be tested. Knowledge of how other plants are treating similar ores is a valuable tool for the metallurgist. Cytec personnel offer this experience and knowledge as a result of metallurgical investigations conducted at many plants and with many ores from around the world.

A number of factors will require evaluation in a flotation test program

1. **Grind-granulometry**
   The grinding range to be evaluated will be largely influenced by the microscopical examination of various screen fractions, referred to previously. Because of the operating costs associated with grinding, a common plant practice is to grind as coarsely as possible without sacrificing rougher recovery; the rougher concentrate then requires regrinding for adequate mineral liberation prior to cleaner flotation. Evaluation of regrinding should be conducted using the information presented in Section D. Proper selection of collector combinations may allow utilization of a coarser grind without loss of rougher recovery.

   In the case of complex ores where recovery of two or more mineral values into separate concentrates is desired, coarse

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primary grinding may not be practical. Due to the resulting complex regrinding and cleaning circuits, with large and sometimes unstable circulating loads, circuit control on a plant scale may not be manageable. In such cases, it may be preferable to grind finer for adequate mineral liberation ahead of the rougher stage, thereby simplifying circuit design and control.

We recommend grinding out the mill with quartz silica (200-500 g) prior to each day's testing to remove rust and residual reagents.

2. **Conditioning time and points of reagent addition**
The conditioning time and points of reagent addition usually have a large influence on metallurgy, particularly under plant operating conditions. For plants currently in operation, the reagent points-of-addition and conditioning times should be adhered to for the standard or control test, but changing the reagent addition point could produce better metallurgy and should be part of any test program. The effect of collector stage-addition and the use of different collectors at varying points in the proposed circuit will also need to be evaluated. Oily collectors are generally, but not always, added in the grinding circuit, and water-soluble collectors can usually be added to the pulp after grinding.

Addition points of frothers, activators and depressants can vary widely, depending on the mineral associations, water quality and types of collector being evaluated. Optimum points of addition for these reagents usually become more apparent after conducting some tests and evaluating the metallurgical results.

3. **pH-alkalinity**
The usual practice is to float at natural pH or in an alkaline circuit adjusted with lime or milk of lime. In some cases, the use of sodium carbonate, sodium hydroxide or ammonia may have an advantage. Acid circuits are utilized if the metallurgical advantages outweigh the higher equipment and operating costs.

pH adjustment is best made in the grinding mill with minor adjustments in the flotation cell. The amount of pH modifier to add is usually based on trial and error and, once established should remain constant for all the tests unless it is a variable under investigation. The recovery vs. pH of certain minerals is documented in the literature. Typical pH operating ranges for various ore types are discussed under separate headings for those ores.

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4. **Water quality**
Water quality from one plant to another can vary greatly. For example in Papua New Guinea the tropical rain produces water of low dissolved salts content, TDS ~100-500 ppm, while on the other hand in arid regions of Australia bore water with a dissolved salts content of >300,000 TDS is used. Water quality can have a substantial effect on metallurgy. Soluble salts can cause undesired activation or depression of various minerals, significantly affect froth structure and frother consumption, as well as the consumption of other reagents. Salts of magnesium, iron and copper are particularly troublesome. It is preferable, therefore, to conduct flotation studies using process water from the plant flotation circuit to more closely simulate actual plant conditions. In cases where this is not practical, simulated process water can also be made after analyzing the plant water and adding the correct amount of minerals or salts.

Routine laboratory flotation screening tests may be conducted using local tap water but results should be confirmed on-site using fresh pulp and plant process water.

5. **Pulp density**
Pulp density, affecting the pulp viscosity, is a significant factor influencing flotation results. High pulp viscosities inhibit air dispersion and good bubble formation, thereby adversely affecting recoveries. Different flotation machine mechanisms are subject to this effect to varying degrees. It is usual practice in laboratory testing to conduct rougher flotation on pulps of 25% to 40% solids. Cleaner flotation is normally conducted at lower pulp densities compared to rougher flotation. The lower pulp density tends to produce higher concentrate grades by promoting better froth drainage.

Higher pulp densities are usually acceptable with increasing specific gravity of the ore solids. When the outcome of flotation experiments will influence plant design, the upper pulp density limit which does not adversely affect rougher recovery, should be determined.

6. **Pulp potential**
Pulp potential can play a key role in sulfide flotation. For a given pH value, the potential range for optimum flotation of a specific mineral can be determined. Such potential ranges have been published for both xanthate and non-xanthate systems. Pulp potentials can be modified electrochemically or chemically with the latter being more practical especially for sulfide minerals.

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Sodium sulfide (Na_2S), sodium hydrosulfide (NaHS), sulfur dioxide (SO_2), nitrogen and air are commonly used to this end. The use of sulfide ion addition requires careful control which is critical to the success of potential controlled flotation or depression.

Potential measurements may be taken with a sulfide ion electrode (SIE) or Ag_2S (vs. Ag/AgCl) electrode when using sulfide ions to adjust pulp potential. A Pt electrode or Au electrode is recommended for potential measurements in all other systems.

7. **Pulp temperature**
Typically the flotation temperature is not studied in base metal sulfide separations but never the less should be maintained as constant as possible. However, the effect of pulp temperature on complex mineral separation should not be ignored. The use of ambient temperature process water stored in a large tank is recommended. Temperature plays a key role in some non-sulfide, non-metallic separations and is discussed under separate headings for those industrial minerals.

8. **Flotation time - rate kinetics**
The practical flotation time required for an ore can be determined by producing incremental concentrates. Separate concentrates are removed at timed intervals, until the froth is completely barren. Using the weights and assays for each incremental concentrate, the metal distribution in each can be determined. This information is then graphically plotted as cumulative recovery versus cumulative flotation time and used for the guidance in subsequent flotation tests. Different collector systems will often show significant differences in flotation rates, which will be apparent by comparing their individual recovery versus time curves. It is also good practice to microscopically examine the incremental concentrates to determine the relative flotation rates of the variously associated minerals and the necessity for regrinding.

The rate at which the mineralized froth is removed and the position of the air valve will also have an influence on flotation kinetics. Therefore it is advised that a consistent froth-scraping pattern at timed intervals, say every 15 seconds, be maintained. If a compressed gas cylinder (air or nitrogen) is to be used for flotation, a flowmeter can be installed between the gas source the air inlet of the flotation machine. The impeller shaft and walls of the cell should also be periodically washed with process water from a wash bottle to return adhering minerals to the pulp and to maintain the pulp level.

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For plant design purposes, it is usual practice to allow at least double the laboratory flotation time for the actual plant operation.

9. Collectors

Establishing the best collector combination is generally regarded as one of the most important aspects of a metallurgical investigation. Although there are many individual collectors for sulfide minerals, the most widely used belong to the general chemical families such as monothiophosphates, dithiophosphates, thionocarbamates, thioureas, allyl xanthate esters, xanthogen formates, mercaptobenzothiazole and xanthates. Within each of these chemical families there are many variations of alkyl or aryl groups which, particularly in the case of the dithiophosphates, can demonstrate significant differences in metallurgical performance on an ore. The prudent metallurgist, therefore, should test at least a few variations within a particular chemical classification before making a judgment on its effectiveness. Likewise, judgment of a collector's performance should not be made hastily based on its use alone. Combinations of different collector types, such as thionocarbamates with dithiophosphates, may demonstrate better metallurgical performance (synergism) than either collector used on its own.

10. Frothers

Selection of a suitable frother for plant operation, by means of laboratory testing, is more difficult than for other reagents to be used in the plant. Of particular interest is the ability of the frother to improve flotation kinetics, recovery and selectivity. The ideal frother or frother combination selected should produce frothing conditions suitable for mineral transport to the froth phase and subsequent cell overflow, while also allowing drainage of entrained gangue particles. The type of flotation cell used in the plant, ore granulometry, the minerals present and their associations, and the presence of slimes will all have an influence on the frothing conditions and the froth character. It is usual practice to make the final frother choice by actual plant testing. For laboratory batch flotation tests, a froth depth of 1.5 to 3.0 cm is adequate.

Where selectivity in flotation is essential, the first choice of frother should be an alcohol type (i.e. AEROFROTH 70, 76A, 88 or OREPREP 501 frothers). Where stronger frothing conditions are required, use of a polypropylene glycol frother such as AEROFROTH 65, OREPREP 507, and OREPREP 786 frothers is recommended. In addition, Cytec Technical representatives will provide assistance in designing or recommending

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custom-formulated frothers to provide optimum frothing conditions and metallurgical performance. For further information on the selection and use of frothers, please see Section 6.2.

11. Depressants

The presence of easily floating gangue minerals such as talc, chlorite, sericite, and pyrophyllite may require depressants such as AERO 633 depressant, CYQUEST 3223, AERO 8842 depressant, AERO 8860 depressant, and various natural polysaccharides. Sodium silicate is sometimes used in sulfide mineral flotation. Carbonaceous matter can be depressed with AERO 633 depressant or Reagent S-7107 depressant. The polymeric depressants used in the selective depression and separation of various sulfide minerals will be discussed under the headings for those ores and in Section 6.3.

12. Separate treatment of sands and slimes

In the case of ores with a high clay (such as kaolin), dolomite, clinochlore or phlogopite content, it may be advantageous to separate the ground pulp into a sand fraction and a slime fraction for separate flotation treatment.

For example, clay slimes increase pulp viscosity and interfere in the recovery of the coarser particles. The fine sulfides (minus 10 µm) often float more slowly than the plus 10 µm particles, requiring a longer flotation circuit residence time.

In actual practice, there are two treatment schemes generally used. In the first method, the ground ore is separated into a sand fraction and a slime fraction for separate rougher flotation. In the second method, the ground ore is subjected to rougher flotation, followed by cycloning the rougher tails into sand and slime fractions. The sand and slime fractions are then treated separately by scavenger flotation. The coarse scavenger feed may require regrinding before flotation.

The use of a dispersant such as sodium silicate, CYQUEST 3223, CYQUEST DP-3 or CYQUEST DP-6 will also help to disperse slimes, reduce pulp viscosity, thereby improving recovery and selectivity.

13. Stages of flotation - rougher, cleaner and scavenger

Laboratory flotation is a batch process that may consist of the following separation stages: rougher, scavenger, and cleaners.

Rougher: The first stage of separation and concentration whereby recovery of the desired minerals is maximized while minimizing gangue flotation. The proper collector selection is critical in this respect.

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Scavenger: Tailings from rougher and, in some cases, recycled cleaner flotation tailings are floated, often with additional collector and frother, to maximize the recovery. The objective is to recover particles (i.e. middlings) not recovered during rougher flotation.

Cleaners: The second stage of concentration whereby the products of rougher and scavenger flotation are re-floated to maximize grade. In most cases, the rougher and scavenger concentrate are reground before cleaner flotation. Multiple cleaning (re-cleaning) stages may be necessary to obtain a marketable concentrate. Small amounts of collector are usually added and aid recovery in the cleaning stages.

In most cases, simply conducting rougher flotation tests is not adequate to fully judge the performance of a collector, reagent scheme or the variable under study. Basing collector selection on rougher flotation recovery alone can be extremely misleading. For example, a collector which gives the highest rougher recovery may be so unselective as to lead to high circulating loads and inferior recovery and concentrate grades in the cleaning stages. At the very least, rougher flotation collector evaluation should include a minimum of three stages, taking separate concentrates over time to produce grade-recovery curves as shown in Figure 4.1. Selection of collectors for further testing should then be based on the relative positions of the grade-recovery curves.

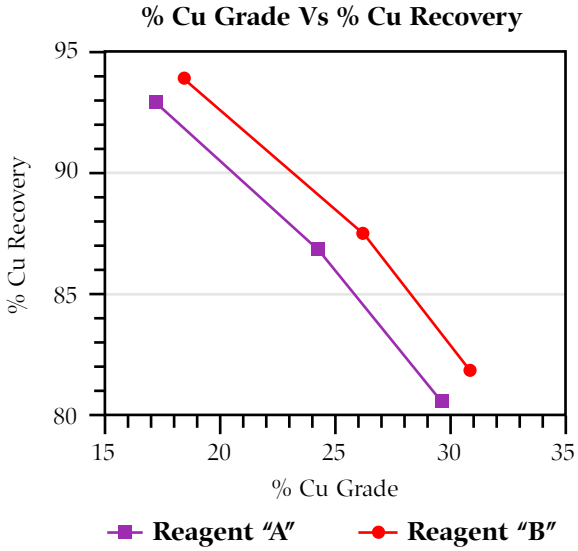


Figure 4.1

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It is good practice to carry rougher flotation into the cleaning stages to produce the final product and to completely evaluate the influence of the variable(s) on the total process. In order to have enough concentrate to conduct cleaner flotation, two or more rougher floats should be conducted. An alternative is to conduct rougher flotation using a larger pulp volume (2-3 kg of ore) and then to clean the concentrate in a smaller volume cell (0.5 to 1 kg). The downside to conducting batch rougher and cleaner tests is that the cleaner tails and process water can not be recirculated as they are in the plant and thus, locked cycle flotation testing would more closely simulate plant practice.

14. Locked cycle flotation testing

To complete the testing of an ore for flowsheet development and to obtain metallurgical data on expected plant performance, locked cycle flotation tests should be carried out. Prior to conducting such tests, the need for and necessary conditions for regrinding of rougher or scavenger concentrates and intermediate products (cleaner tailings) should be established. The need for regrinding is determined by microscopical examination of the various flotation products, as described previously.

In each complete cycle test (Fig. 4.2), middlings (in the form of cleaner tailings or scavenger concentrates) are recirculated to one or more processing steps in the subsequent test cycle. The disposition of these middlings streams should be determined during prior laboratory testing and by optimization during the locked-cycle test work, depending on the results obtained therein.

From each cycle test, a final concentrate and final tailings are obtained. Except for the very last cycle test, middlings will be circulated. An estimate of middlings weights can be made by filtering the middlings products and obtaining their weights as damp filter cakes. In this manner it can he seen if middlings weights stabilize after a few complete cycles. It may take from four to seven cycles to reach equilibrium conditions.

Equilibrium is reached when for at least two consecutive cycles:

- The combined weights of the final concentrate plus the final tailings stabilize and approximate the weight of fresh ore charged to each new cycle.
- The assays of the final concentrate and the final tailings stabilize and the calculated head assay, based on these two products, are similar to the original fresh feed assay.
- Metallurgical distribution between the final concentrate and the final tailings stabilizes.

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If equilibrium conditions are not established after six or seven cycles, the flotation products must again be examined microscopically to determine the cause. Addition of a small amount of collector to the cleaners or further regrinding of middlings products may be required. The use of recycled process water can be simulated by clarifying the tailings by sedimentation to recover the water. Water from the concentrate or intermediate products can be recovered in the same way or by filtration. The effect of reagents and soluble salts in a re-circulating water system can also be assessed in this manner.

Where more than one valuable metal is to be recovered, each into a separate concentrate, the complexity of the cycle test and calculations involved increase considerably.

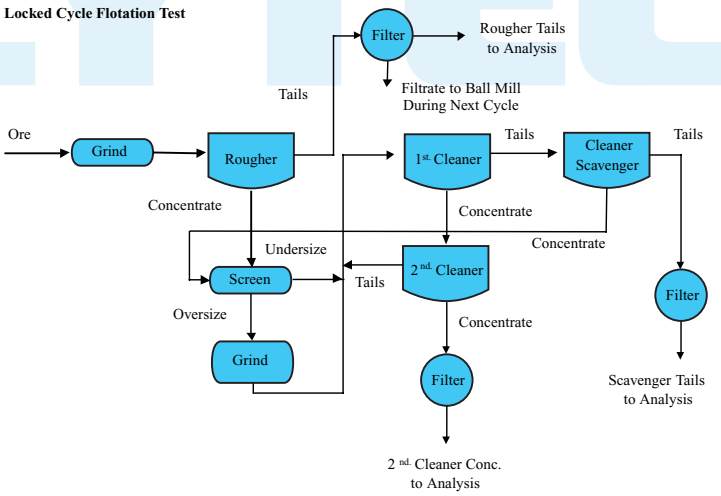


Figure 4.2

G. Handling of flotation products

Flotation products are filtered using vacuum filtration for the concentrates and a large volume pressure filter for the tailings. We suggest using filter paper of high wet strength such as sharkskin filter paper or craft paper. Filtration can further be enhanced by flocculating the products, which is extremely helpful if the products contain a large amount of slimes.

The filtered products are then dried at 70-100°C. It is important that the oven temperature does not exceed 100°C so as to avoid roasting the sulfide minerals and driving off sulfur. The concentrate

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and tails should be dried separately either in separate ovens or, if in the same oven, by placing the low grade tails on the upper shelves and the higher grade concentrates on the lower shelves. After drying, the net weight of the flotation products is recorded for calculating the metallurgical balance. The products may be brushed through a screen (35 Tyler mesh for example) to break up aggregates, then mixed by rolling on a rubber sheet before representative cuts are taken for chemical analysis. It is common practice to pulverize the samples prior to analysis.

H. Interpretation of results

The assay results and recorded weights are then used to generate mass balances from which graphs can be created.

- Rate kinetic curves can be generated, % cumulative recovery versus time.
- Grade recovery curves, % cumulative grade versus % cumulative recovery. (See figure 4.1)
- Selectivity curves, % cumulative recovery of valuable metal versus % cumulative recovery of a gangue element. (See Figure 4.3)

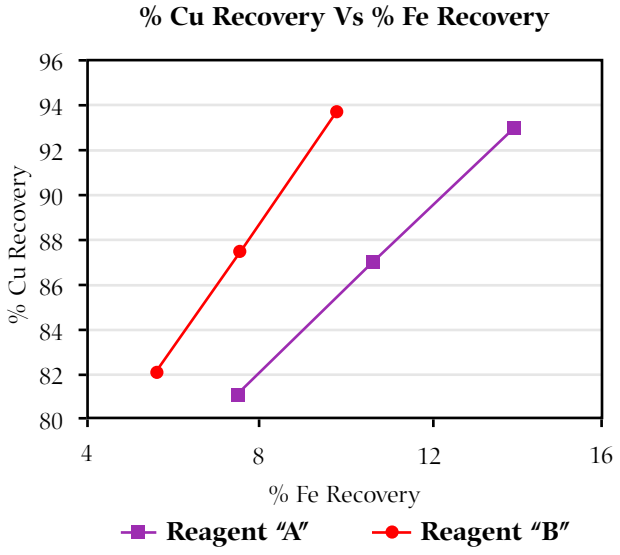


Figure 4.3

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Section 4A The effects of reagent choice on flotation circuit design and operation

When testing a new orebody, the potential impact of reagent choice on equipment selection and circuit configurations is often not fully appreciated. During preliminary feasibility testing, it is not uncommon to evaluate only one or two collectors (usually a xanthate and/or a dithiophosphate), an arbitrarily selected frother, and a pH modifier such as lime. This is particularly true in the case of relatively simple ores such as a copper or copper-gold ore containing iron sulfides such as pyrite. The assumption is that this will provide sufficient information for flowsheet design and a preliminary economic/metallurgical analysis. "Fine tuning" of reagents is left to a later stage of the investigation, or even until after the plant has started operating. We believe that, even for simple ores, this approach has potentially serious pitfalls, which are discussed in this section.

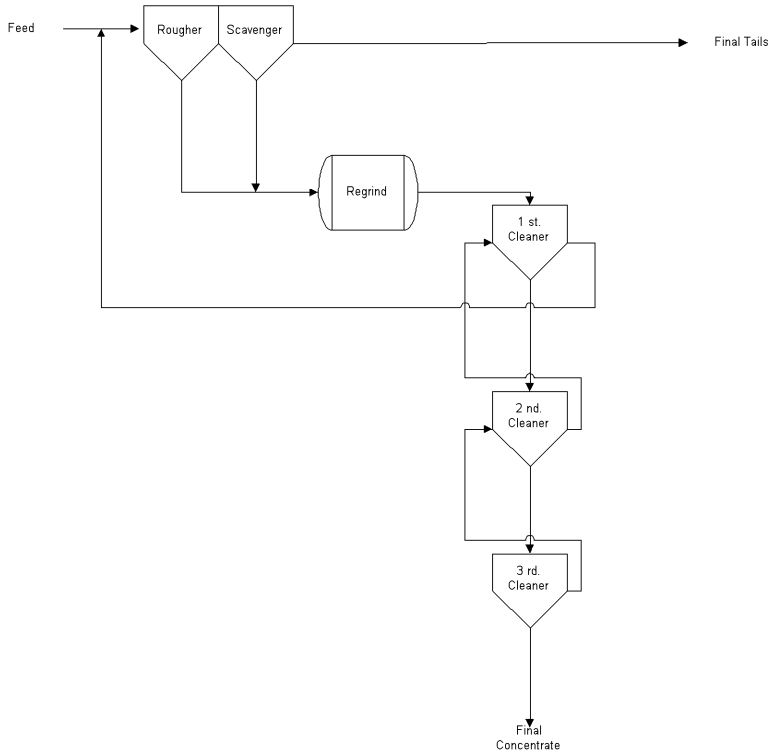
Different reagents (including collectors, frothers, pH modifiers, and depressants) can have a significant effect on flotation kinetics, the grade-recovery relationship, the amount and type of froth, the mass of rougher and scavenger concentrates, and rejection of penalty elements, etc. Optimization of these variables at an early stage of the testing process can have a significant effect on flowsheet design, as well as on capital and operating cost estimates. Consider a situation where Reagent combination A gives the highest rougher-scavenger recovery, but with a lower concentrate grade (and hence a greater mass of rougher-scavenger concentrate) than Reagent combination B. If combination B is then eliminated from further consideration because it gives lower rougher recovery, its following potential benefits of better rougher selectivity may be overlooked:

- The greater selectivity of Reagent B and the lower mass pull in the rougher-scavenger circuit will reduce the required regrinding and cleaning capacity which may reduce both capital and operating costs.
- The reduced load in the regrind and cleaning circuit may well result in an increase in final concentrate grade and/or recovery compared to Reagent A.
- Reduced circulating loads in the cleaner circuit usually mean the cleaner circuit is easier to control and operate.

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- The use of a more selective reagent or reagent combination in the rougher-scavenger circuit usually enables operation of that circuit at a lower pH, thus reducing the amount of lime or other depressant required.
- The use of a selective collector may produce a sufficiently high-grade concentrate in the early stages of the rougher circuit, that this product can bypass the regrinding stage and be sent directly to the feed to the first or second cleaner. This not only further reduces the load on the regrind circuit, but also minimizes the risk of overgrinding already liberated value minerals. Such overgrinding can lead to "sliming" and subsequent loss of overall recovery. Flowsheets 1 and 2 are traditional, simple flotation circuits. Flowsheet 3 indicates the kind of circuit which may be possible when using more selective reagents.

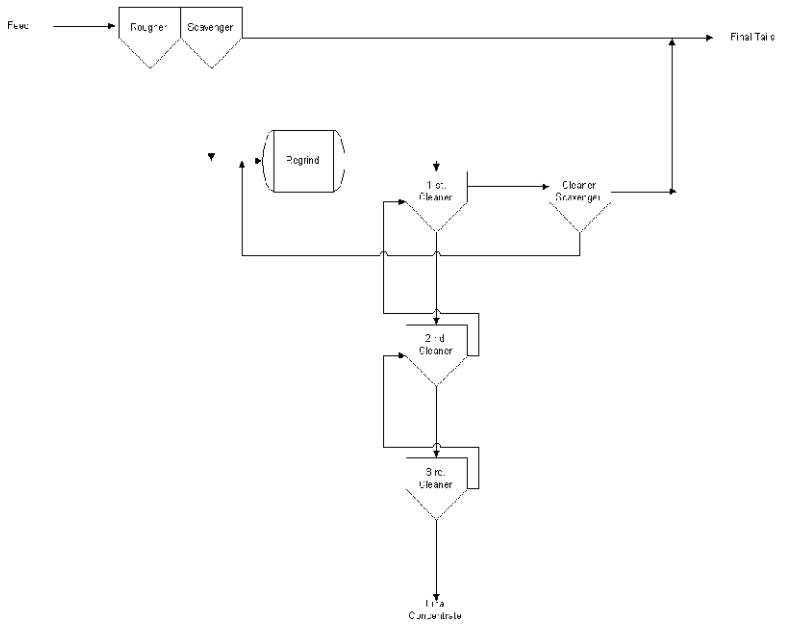
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Flowsheet 1 – Conventional

Flowsheet 1 is typical of early base-metal flotation flowsheets. The cleaning circuit is totally "closed" with the 1st. cleaner tails being returned to the head of rougher-scavenger flotation. In some cases, the scavenger concentrate was also returned to the head of rougher flotation. Such a flowsheet is typified by high circulating loads in both the rougher-scavenger and cleaner stages.

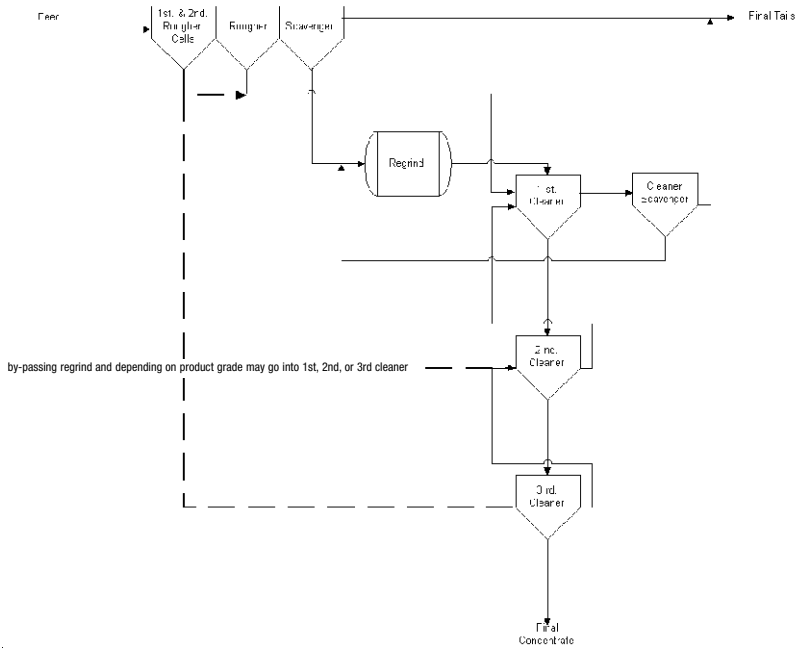
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Flowsheet 2 – Modified Conventional

Flowsheet 2 is probably the most typical of current base-metal flotation circuits. The 1st. cleaner tailing is sent to a cleaner-scavenger stage, the concentrate of which is returned to the regrind mill. The cleaner-scavenger tailing joins the rougher-scavenger tailing to form the final plant tailings. This design reduces the circulating loads in both the rougher-scavenger and cleaner stages, thereby reducing the flotation capacity required for a given mill tonnage.

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Flowsheet 3 – Selective Rougher

Flowsheet 3 represents the type of design which may be made possible by the use of more selective collectors in the rougher-scavenger stage. Samples of the concentrate are taken from successive cells down the rougher bank for both chemical assay and mineralogical examination. In most cases, it will be found that the concentrate from the early stages of rougher flotation will be of high enough grade and sufficiently liberated to bypass the regrind mill. Whether this concentrate is sent to the first, second, or final cleaner stage will depend upon its grade and mineralogical characteristics. This flowsheet design further reduces the circulating load in the cleaners as well as minimizing overgrinding of already-liberated value mineral.

The advantages described above for simple ores are even more important when treating complex ores containing two or more value minerals. With these ores, separation efficiency between the individual value minerals is often more critical than the selectivity between the value minerals and the gangue minerals.

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In the case of already existing flotation circuits, many of the described advantages could still be obtained if suitable circuit and piping changes were made. Furthermore, since many plants are already operating at or above design tonnages, greater selectivity in the rougher circuit and the consequent reduction of the load on the regrind and cleaning circuit, can have major benefits, such as eliminating circuit bottlenecks.

To summarize, the selection of collector and other reagents should not be based on rougher-scavenger evaluation only, and certainly not solely on reagents that give the highest recovery therein. Rather, reagents should be evaluated on the grade-recovery relationships they produce throughout the whole process, including regrinding and cleaning. This will inevitably entail at least locked-cycle testing in the laboratory, preferably followed by pilot-scale testing.

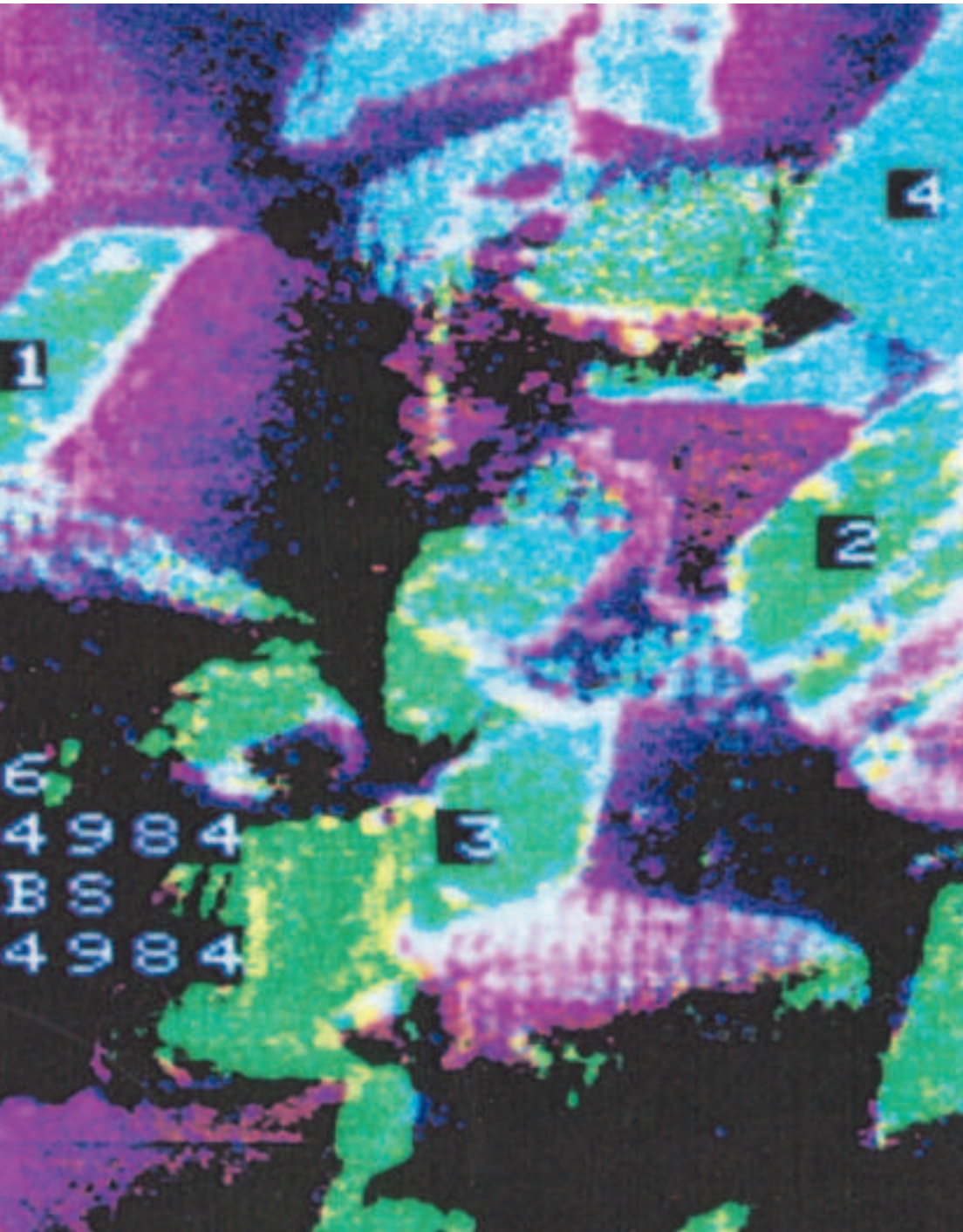
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5 Flotation reagent fundamentals



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Section 5 Flotation reagent fundamentals

Flotation is a physico-chemical process. This statement clearly indicates that both physical and chemical factors are equally important in flotation. In other words, it would be naïve to proclaim that one set of factors is more important than the other set, which is sometimes done in research or practice. Chemical factors include the interfacial chemistry involved in the three phases that exist in a flotation system, viz. solid, liquid and gas. Interfacial chemistry is dictated by all the flotation reagents – such as collectors, depressants, frothers, activators, and pH modifiers – used in the process, water chemistry, and the chemistry of the minerals. Physical (or more accurately, physical-mechanical and operational) factors comprise equipment components (cell design, hydrodynamics, bank configuration, and bank control) and operational components (feed rate, mineralogy, particle size, and pulp density). Thus flotation, while simple in concept, is an extremely complex process in practice involving many scientific and engineering phenomena.

In most flotation systems, physical and chemical factors are not independent, i.e. there are significant interactions among the many variables. In theory, when all physical factors are optimized, a change in a chemical factor should clearly record a measurable change in flotation efficiency (either recovery or grade or both), and vice versa. In practice, however, this may not be immediately obvious because of certain operational restrictions, and metallurgists have to revert to statistical tools to demonstrate significant changes. A further complication is that neither physical nor chemical factors can always be fully or satisfactorily optimized since there can be significant changes occurring routinely in mineralogy, feed rates and particle size distribution. Nevertheless, flotation plant operators still achieve impressive separations and performance by managing controllable factors.

In general, in a fully commissioned plant it is more difficult to change physical-mechanical factors than operational or chemical factors. Indeed, in most plants considerable attention is, therefore, focused on changing or optimizing chemical and operational variables.

The importance of chemical factors in achieving target performance has been widely recognized. In many circuits, a mere change in pH of the pulp can cause dramatic differences in flotation efficiency. This is true of flotation reagents as well.

In this section an attempt is made to highlight how changes in the chemistry of flotation reagents can have marked influence on flotation efficiency. The chemistry of collectors is used to illustrate

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structure-activity aspects, though the principles are applicable to depressants as well.

A brief, simplified description of terminology will be necessary to appreciate the structure-activity aspects of flotation reagents. *Donor Atoms* or *donors* or *ligand atoms* are those atoms in the reagent molecule that bond directly with the metal atom on the mineral surface. *Ligands* are the functional groups containing the donor atom(s) on the reagent molecule that participate in bond formation with metal atoms on the mineral; donor atoms are also often referred to as ligands. *Functional Groups* are a well-recognized group of atoms containing the donor atoms in the reagent molecule. *Acceptors* are atoms or groups of atoms that accept electrons from donors. A metal atom on the mineral surface is the acceptor in most instances. Acceptors are generally positively charged, while donors or ligands or functional groups are often negatively charged. Note, however, that in cationic flotation reagents, the functional group of the molecule carries a positive charge, and this can interact with a mineral surface that has negative sites. Functional groups are generally *polar* (i.e. carrying a charge, partially or fully). *Non-polar* moieties of a flotation reagent molecule are generally a hydrocarbon chain (linear or branched, aliphatic or aromatic or a combination).

For a vast number of flotation reagents, adsorption at the solid-liquid interface is of critical importance. Frothers, which adsorb significantly at the liquid-air interface and alter its properties, can also adsorb at the solid-liquid interface and influence flotation outcome. However, interfacial chemistry of frothers is largely characterized by non-specific adsorption processes. Most commonly used frothers belong to the classes of short-chain alcohols and polyglycols (and their monoethers). Consequently, the scope of structure-activity relationships is rather limited.

The driving force for, and the mechanism of, adsorption of flotation reagents on minerals comprises chemical (chemisorption, surface reaction or complexation, and chemical adsorption), electrostatic (physisorption or physical adsorption), and non-specific forces (such as Van der Waal's forces, hydrogen bonding, and the so-called hydrophobic force). Chemical interactions have the highest adsorption energies followed by electrostatic and non-specific interactions. In many cases, more than one driving force is in operation. Overall adsorption energy is, therefore, a sum of all energies associated with various adsorption processes.

In the case of non-specific adsorption processes, structural aspects of the reagent molecule that can be changed include the nature and type of the hydrocarbon chain, moieties capable of hydrogen bonding etc. In general, such changes in the molecule can only cause

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small changes in interfacial properties (for example, hydrophobicity) of the solid-liquid interface. Hydrophobicity imparted by a reagent on the mineral surface increases with an increase in the reagent's hydrocarbon chain length.

When the adsorption process is predominantly electrostatic in nature, a change in the charge density of the molecule (or the functional group), or of the mineral surface, causes a noticeable change in adsorption energy or interaction energy. Pulp chemistry plays a significant role in these systems; for example, the presence or addition of inorganic ions. Reagents that carry positively-charged functional groups are called "cationic" reagents; these are typically amines – primary, secondary, tertiary or quaternary. Reagents that carry negatively-charged functional groups are called "anionic" reagents; examples of these are fatty acids (carboxyl groups), hydroxamates and alkyl or aryl sulfonates (or sulfates). Reagent molecules that can potentially have both cationic or anionic sites (depending upon pH for example) are called "amphoteric" (zwitter ionic) reagents. In general, for cationic reagents, adsorption is predominantly electrostatic. Similarly, in the case of sulfonate or sulfate-containing reagents, the electrostatic component is usually the predominant one (there can, however, be a chemical component also). In the case of anionic collectors containing carboxyl or hydroxyl groups, there is often a significant chemical component in the overall adsorption energy in addition to the electrostatic component. Under certain conditions, for these reagents the electrostatic component can be completely overridden by the chemical component.

Structure-activity aspects become very important, and offer a wide scope for reagent design and control, in systems where the driving force for adsorption of flotation reagents on minerals is chemical. Since chemical interactions between reagent molecule and mineral surfaces have the highest adsorption energies, changes in structure of the reagent molecule can potentially result in large changes in the strength of adsorption, the resultant interfacial properties, and flotation response. This has been clearly demonstrated in a large number of reagent families in flotation research and practice. A few examples are given later in this section.

Several models have been proposed to explain chemical adsorption of reagent molecules on mineral surfaces. Some examples of these include chemisorption, surface reaction, and surface complexation. Irrespective of the model or the process of chemical interaction of reagents with minerals, the basic requirement is that a chemical bond – covalent or partially covalent – be formed between the donor atoms of the reagent and the metal atom of the mineral, at

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least in the first adsorbed layer. Further, in the first adsorbed layer, the metal atom is still a part of the mineral lattice. Subsequent layers of metal-reagent complexes can, and often do, exist, but in these layers the metal is obviously not part of the mineral lattice. The first adsorbed layer is quite stable on the mineral surface, and often requires chemical changes for desorption (the common notion that high turbulence can dislodge adsorbed species is a myth). In the case of sulfide minerals and certain thiol reagents, an electrochemical mechanism of adsorption via formation of a metal-reagent complex is now widely accepted. Many sulfide minerals are excellent conductors and exhibit properties that are similar to those of metals. Electrochemical reactions are quite facilitated, and are similar to reactions in batteries or corrosion processes. Furthermore, many thiol reagents exhibit redox reactions. Extensive studies and plant observations have established that redox conditions of flotation pulps do influence flotation efficiency.

In discussing the chemistry of flotation reagents it is most convenient to classify them into two distinct groups: a) those used specifically for sulfide minerals, and b) those used for non-sulfide minerals. With the exception of a few elements such as the base and precious metals, most elements or their minerals are obtained from non-sulfide ores. It is well recognized that separation schemes for non-sulfide minerals are distinctly different from those for base metal sulfide minerals. Such distinctions can be readily understood by the fundamental differences that exist in physical and chemical properties between sulfide and non-sulfide minerals. These differences arise, for the most part, from differences in the chemistry between S and O. The base-metal sulfide minerals are characterized by mostly covalent or metallic bonding, low solubility in water, weakly hydrated surfaces and poor hydrogen bonding, a high degree of natural hydrophobicity, strong affinity for S-containing ligands, and pulp chemistry dominated by electrochemical reactions. Conversely, the non-sulfide minerals are generally characterized by ionic bonding, higher solubility in water, strongly hydrated surfaces and strong hydrogen bonding, strong affinity for O-containing ligands, and pulp chemistry dominated by ion exchange reactions. Plant practice is often consistent with the major differences between sulfides and non-sulfide minerals.

The sulfur atom on either a carbon or a phosphorous atom is the key donor and the center of activity in sulfide collector chemistry. Its bonding properties are readily modified by neighboring atoms and groups, especially by the two other major donor atoms N and O. Sulfide minerals can be floated by almost any collector, including those that do not contain sulfur. However, in order to obtain selectivity

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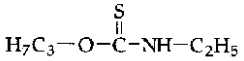
that is meaningful in industrial flotation at economic levels, a sulfur-containing collector is invariably preferred. This statement is amply supported by the fact that all of the commercially used sulfide collectors, since the introduction of xanthate, contain sulfur.

In addition to the basic functional groups containing the major donor atoms, substituents attached to them provide a unique character to the collector molecule. These groups essentially modify the affinity of the collector for a given sulfide surface, the hydrophobicity conferred, kinetics of adsorption, and the pKa of the molecule which, in turn, has a direct influence on the solution properties of the collector and its interaction with sulfide surface. Substituents can also participate in bond formation with the mineral, which may either reinforce or counter the interactions of the basic functional group with the sulfide surface.

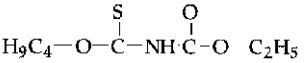
Thus, seemingly minor changes to the structure of a collector molecule can have a very significant effect on the collector's performance in the flotation process. This is illustrated in the examples which follow.

Example 1

In the case of the traditionally-used dialkyl thionocarbamates, such as O-isopropyl N-ethyl thionocarbamate (IPETC, AERO 3894 promoter, structure 5-I), the basic functional group is -O-C(=S)-NH-. An interesting modification of the basic dialkyl thionocarbamates is the substitution of an alkoxycarbonyl group on the N atom (as shown in structure 5-II). The use of the strongly electron-withdrawing alkoxycarbonyl substituent introduces an additional active donor, O, in the form of C=O attached to the alkoxy group. Thus, the functional group is not solely restricted to the thionocarbamate; instead, it is the more complex -O-C(=S)-NH-C(=O)-O, which has quite different properties from the basic thionocarbamate group. The pKa of the molecule is directly affected; for example, the pKa of IBECTC (structure 5-II) is 10.5 compared with a pKa of >12 for IPETC. These attributes make the new thionocarbamates strong copper sulfide collectors at low pH values (<11), for example, while still maintaining the selectivity against pyrite characteristic of the thionocarbamates.



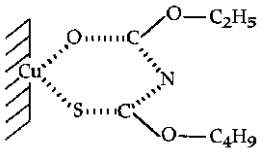
(Structure 5-I)
Isopropyl Ethyl Thionocarbonate
(IPETC)



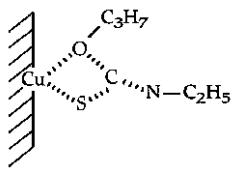
(Structure 5-II)
Isobutyl Ethoxycarbonyl Thionocarbonate
(IBECTC)

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Fundamental studies have shown that the new alkoxycarbonyl thionocarbamates form a highly favored, six-membered chelate (see 5-III) with Cu atoms on a copper sulfide mineral surface. In the case of IPETC, however, such a favorable chelate is not possible (instead a less favorable four-membered chelate involving the O and the S is formed, (see 5-IV). External reflectance FTIR studies using copper foils have indicated that when a copper foil was first treated with IBECTC and then with IPETC, the IBECTC adsorbed on copper foil could not be displaced by IPETC. When the copper foil was treated in the reverse order, IBECTC was able to adsorb on copper by displacing IPETC. Similar results were obtained when a xanthate was used instead of IPETC.



(5-III)
Schematic of Cu-IBECTC surface Complex



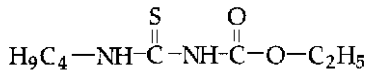
(5-IV)
Schematic of Cu-IPECTC surface Complex

Example 2

The alkoxycarbonyl thioureas (Structure 5-V) are structurally similar to the alkoxycarbonyl thionocarbamates, except that the former class has the basic thiourea functionality and exhibits collector properties that are characteristic of both the thiourea group and the alkoxycarbonyl substituent. Due to the presence of the second N, instead of O, however, the modified thioureas are found to have collector properties that are often quite significantly different from those of the alkoxycarbonyl thionocarbamates. The alkoxycarbonyl thioureas have been found to enhance the recovery of silver and gold from ores. Adsorption measurements on pure minerals, laboratory flotation tests, microscopic examination of flotation products, and plant usage experience have all confirmed that the modified thioureas show a stronger capacity than the corresponding thionocarbamates for floating chalcopyrite. The alkoxycarbonyl thionocarbamates, on the other hand, have been shown to float the copper-rich minerals such as bornite, covellite and chalcocite more effectively than do the corresponding thiourea collectors. These differences appear to be kinetic in nature, and the equilibrium recovery of the minerals may sometimes be the same for both classes of collectors. The reasons for such minerals differentiation by collectors should be related to both the bonding states of the metal on the sulfide surfaces

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and to the electron density distribution on the donor atoms of the collectors, as also to the effect of redox conditions on the collector properties.



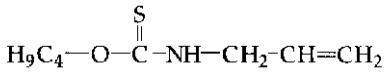
(Structure 5-V)
n-Butyl Ethoxycarbonyl Thionourea (NBECTU)

Cytec introduced ethoxycarbonyl thionocarbamate in collectors in 1985 and ethoxycarbonyl thiourea collectors in 1989 (the 5000 series of AERO promoters) and their commercial use is now widespread for the flotation of copper, gold, silver and PGM minerals. These modified thionocarbamates and thioureas are stable compounds and quite resistant to oxidation. They are more selective against iron sulfides than the simple dialkyl thionocarbamates even at pH < 10. The alkoxycarbonyl thionocarbamates and thioureas were both developed as selective collectors for operation at reduced pH values and, as such, afford substantial lime savings. They have excellent shelf life, hydrolytic stability in a wide pH range, and they are readily dispersed in water.

Microscopical examination of flotation tails from porphyry copper plants using the modified thionocarbamates and thioureas has demonstrated that part of the performance advantages obtained with these collectors can be attributed to the efficient recovery of coarse sulfide particles, including middlings.

Example 3

Another interesting modification of the dialkyl thionocarbamate structure is obtained by incorporating an allyl group, $-CH_2-CH=CH_2$ on the N donor atom (see structure 5-VI). The allylic double bond modifies the adsorption and collector properties quite significantly in comparison to the dialkyl thionocarbamates such as IPETC. The double bond in allyl thionocarbamates can be expected to form a π-complex with Pt, Pd, and possibly Cu. Adsorption studies have shown that there is a strong tendency for the allyl thiono-carbamates to interact with copper and platinum surfaces. Cytec introduced the allyl thionocarbamates in 1980, and they were fully commercialized



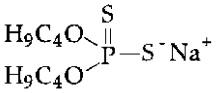
(Structure 5-VI)
Isobutyl Allyl Thionocarbamate (IBATC)

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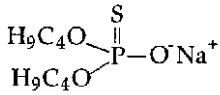
in 1989 (the 5000 series of AERO promoters). One of their main attributes is the rapid flotation kinetics that they provide at quite low dosages. Laboratory and plant tests conducted on platinum ores have shown that the allyl thionocarbamates improve recovery of PGMs, again at low dosage levels.

Example 4

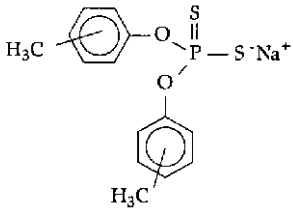
An important modification of the basic dithiophosphorous group, >P(=S)S, as found in the dithiophosphate collectors (structure 5-VII & 5-IX), is that of replacing one of the S donors in the functional group by an O donor to give the corresponding monothio derivative (structure 5-VIII & 5-X). This single change in the nature of the donor atoms in the dithioacid is sufficient to alter its collector property dramatically in view of the quite different properties of the donor atoms O and S.



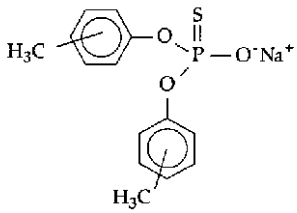
(Structure 5-VII)
Diisobutyl Dithiophosphate (DTP)



(Structure 5-VIII)
Diisobutyl Monothiophosphate (MTP)



(Structure 5-IX)
Dicresyl Dithiophosphate (DTP)



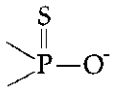
(Structure 5-X)
Dicresyl Monothiophosphate (MTP)

Extensive studies of the solution and collector properties of the monothio and dithio acids in a wide pH range have indicated that the monothioacids are more stable, stronger acids, and stronger collectors than their dithio analogs under certain pH conditions. The dialkyl monothiophosphate, for example, is found to be a truly acid circuit collector (effective in the pH range 2-7 in contrast to the dithiophosphate, which is a better collector in the alkaline pH range (pH > 9).

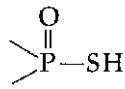
The differences in the collector properties between the mono and dithiophosphates are attributed to the rather interesting tautomerism

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that exists in monothiophosphate (structures 5-XI and 5-XII). The available evidence suggests that, in aqueous solutions, the thiol form, P(O)SH, may be stable in the acid pH range and the thione form, $P(S)O^-$, stable under alkaline conditions. The thiol form is understandably favorable for sulfide flotation. In the thione form, the very electronegative O tends to retain much of the electron density at the expense of the less electronegative sulfur. The reduced electron density on the thione S is probably responsible for weak bonding with sulfides above pH 7.



(Structure 5-XI)
Thione Tautomer (basic pH)



(Structure 5-XII)
Thiol Tautomer (acid pH)

Monothiophosphates, introduced in 1989, are now used widely on copper and gold ores. The monothiophosphates are used for bulk sulfide flotation in acid circuits where they are more stable and stronger than xanthates, dithiophosphates, and xanthogen formates. They have also found application for selective gold flotation from primary Au ores or for improving Au recovery in base metal sulfide flotation in alkaline circuits.

Example 5

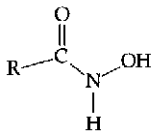
Often, enhanced performance can be realized by merely changing the hydrocarbon part of the reagent molecule while keeping the functional group intact. For example, a slightly branched hydrocarbon group in a collector molecule can provide a greater selectivity in flotation than a linear hydrocarbon group. It is well known in flotation practice that an aryl dithiophosphate floats galena far better than an alkyl dithiophosphate.

Example 6

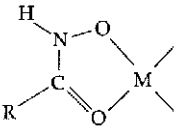
It is well-known that fatty acids (Structure 5-XV), which are used extensively in flotation of non-sulfide minerals, are inherently non-selective. Hydroxamic acids (Structure 5-XIII), which are structurally similar to fatty acids, are considerably more selective. They differ from fatty acids by a nitrogen which does not participate directly in bonding with a metal atom, but has an effect on the electron density on the O donor attached to it. The O donors in hydroxamic acids are weaker donors (more selective) than those in fatty acids. There is considerable covalence in the bonds formed with metals (compared

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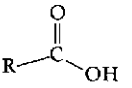
with the ionic character of the bonds formed with fatty acids). These factors impart considerable selectivity in the hydroxamate interaction with metals, and hence in flotation. They form five-membered metal chelates (shown in Structure 5-XIV) because the hydroxyl attached to N is appreciably acidic; this is in contrast to the fatty acids which, under certain conditions, can form a less stable four-membered chelate (structure 5-XVI).



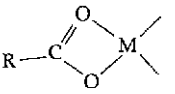
Structure 5-XIII
Alkyl Hydroxamic Acid



Structure 5-IXV
Metal chelate



Structure 5-XV
Fatty Acid



Structure 5-XVI
Metal chelate

On the basis of differences in stability constants of many metal complexes hydroxamic acid, it can be predicted that hydroxamic acids should be more selective than commonly used fatty acids, and indeed this has been found to be the case in practice. Recently a new manufacturing process was developed and alkyl hydroxamate was introduced by Cytec in 1989 under the trade name AERO 6493 promoter which is currently used for the removal of colored impurities from kaolin and for oxide copper recovery. It has also been shown recently that alkyl hydroxamates improve the recovery of precious metals that are associated with pyrite, marcasite, pyrrhotite and goethite. In kaolin beneficiation, alkyl hydroxamates have been found to be much more effective than fatty acids; they produce higher brightness clays with better yields from a variety of kaolin clays. No activators are required, and retention times in flotation are shorter than those for fatty acids.

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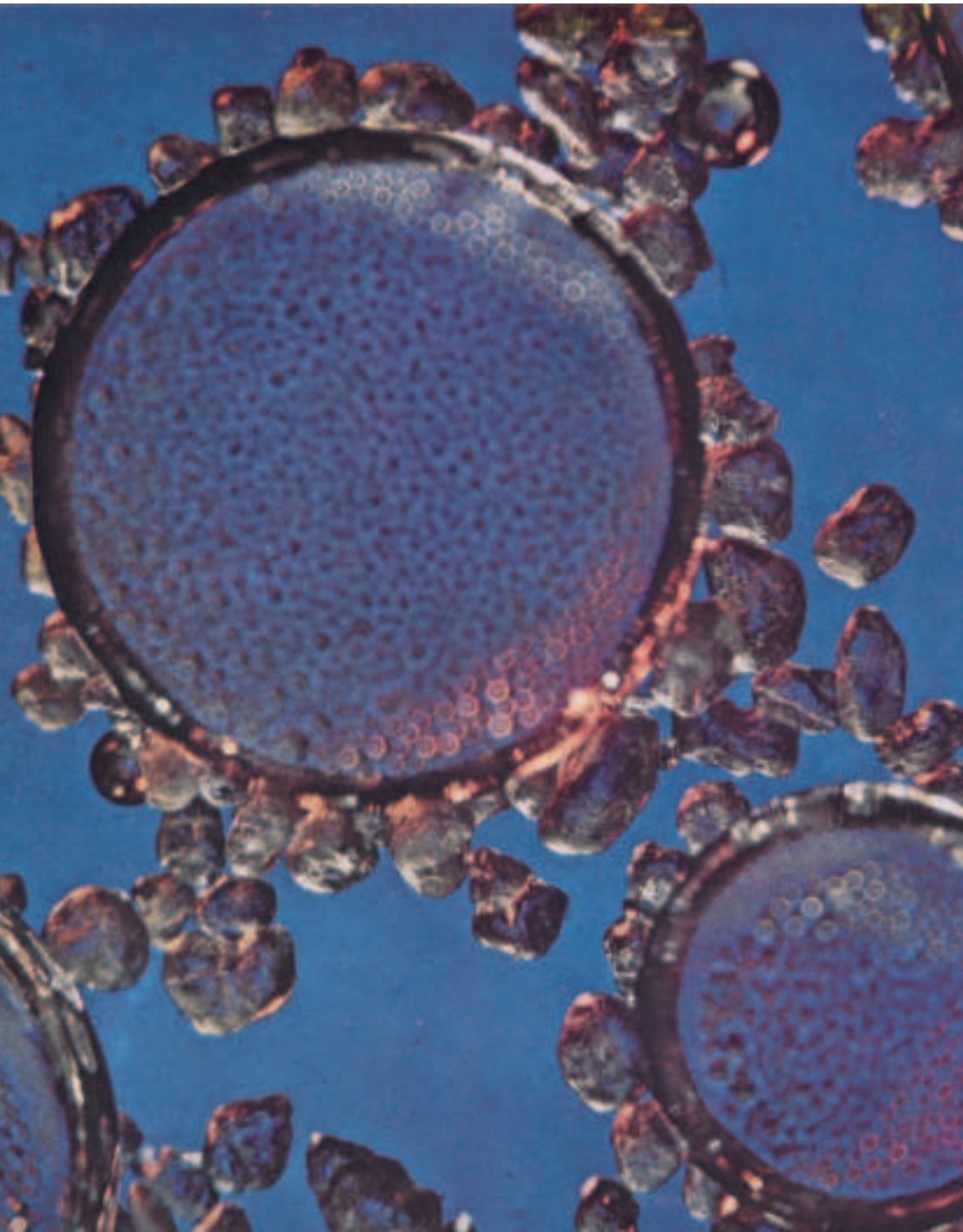
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6 Flotation of sulfide ores



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Section 6 Flotation of sulfide ores

Many collectors and frothers are in use in the flotation treatment of sulfide and metallic ores containing such metals as copper, lead, zinc, nickel, cobalt, molybdenum, iron, precious metals (including platinum-group metals) and such penalty elements as arsenic, antimony and bismuth. The principal factors affecting the choice of collectors are the mineral forms (sulfide, oxidized and/or metallic species) and their associations with each other and the gangue minerals.

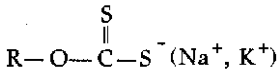
Recent trends in flotation practice have shown that, in many cases, a combination of two or more different collectors provides better metallurgy than a single collector. This is not surprising when one considers that, even in such a simple case as copper ores, there may be a variety of copper minerals present (eg. chalcopyrite, chalcocite, covellite, bornite, native copper, tetrahedrite, and oxidized or tarnished copper minerals) each of which responds differently to different collector chemistries. Obviously, this aspect is even more important when making a bulk float of minerals of two different metals (eg. lead and copper). For many decades, the most commonly-used collector combinations were those of xanthate and dithiophosphate, or of xanthate and dialkyl thionocarbamate. However, in the past 10-15 years, a large number of new collector chemistries has been developed and introduced by Cytec. Whilst increasing the complexity of the reagent testing process, this has undoubtedly greatly expanded the opportunity of establishing the optimum reagent combination for any specific ore. This aspect of collector selection is addressed in more detail in Section 6.4.

6.1 Cytec's sulfide collectors (promoters)

There are many possible ways of categorizing sulfide collectors; eg. copper collectors, lead collectors, soluble collectors, oily collectors, thiol collectors, etc. We feel that none of these classifications adequately distinguishes the actual functionality of the collectors. Consequently we have chosen to classify the collectors based on their chemical structure, functional groups, and the important donor atoms. Please note that Cytec has always used the terms "collector" and "promoter" synonymously. Other reagents which assist the adsorption of a collector on the mineral surface are referred to as "activators", and their use is discussed later.

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6.1.1 AERO xanthates



xanthate

Xanthate collectors were introduced in 1925, and are still widely used, especially for easy-to-treat ores where selectivity (especially against iron sulfides and penalty elements) is not an issue. They are usually supplied in the powder or pellet forms and are readily soluble in water, and could be made up to any strength for convenience in dosing. Xanthate solutions have relatively poor long-term stability and, therefore, are supplied in liquid form only when the manufacturing plant is in close proximity to the use location. Xanthates are available in a range of carbon chain lengths, generally from C_2 to C_5. The collecting power generally increases with increase in chain length, but the selectivity decreases. Xanthates are relatively unstable at low pH and, therefore, are not suitable for flotation in acid circuits.

AERO 303 xanthate – Potassium ethyl xanthate. Shortest carbon chain of the available AERO xanthates. Particularly useful where maximum selectivity is desired.

AERO 325 xanthate – Sodium ethyl. Used on complex ores for maximum selectivity. Most frequently used to float galena with Pb/Zn ores.

AERO 343 xanthate – Sodium isopropyl. Most widely used in the flotation of sulfide minerals of copper, molybdenum and zinc. Good compromise between collecting power and selectivity.

AERO 317 xanthate – Sodium isobutyl. A relatively strong collector used in the flotation of Cu, Pb, Ni, Zn, and PGM ores.

AERO 350 xanthate – Potassium amyl. The most powerful and least selective xanthate. Often used as a scavenger collector following a more selective rougher collector. Used widely in the flotation of Cu, Ni, Zn, and Au-containing iron sulfides.

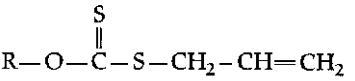
6.1.2 Xanthate derivatives

Two classes of xanthate derivatives are in common use, xanthogen formates and xanthate allyl esters. Both are oily collectors, more

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selective than the corresponding xanthate, and can be used over a wide pH range. Since they are insoluble in water, point of addition and conditioning time may be important. Xanthate allyl esters are among the most selective of all the available sulfide collectors.

AERO 3302 promoter

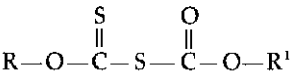


Xanthate Allyl Ester

Comments

- Oily collector, not soluble in water, therefore, usually fed to grinding mill.
- Effective copper collector in both alkaline and acid circuit. Also good for zinc flotation in lime circuit. Usually used in conjunction with xanthate. Very selective against pyrite.
- Excellent collector for molybdenite and is, therefore, often used on copper/molybdenite ores.
- Often increases recovery of gold and silver.
- Used for flotation of sulfidized copper-oxide minerals.
- Improves selective recovery of platinum group metals.

AERO 203, 204, and 758 promoters



Dialkyl Xanthogen Formate

Note: *In some regional markets, these products are known as SF 203, 204, and 758 promoters.*

Comments

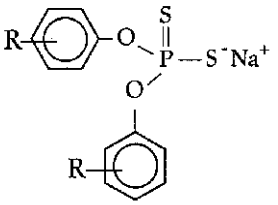
- Oily collector, not soluble in water, therefore, usually fed to grinding mill.

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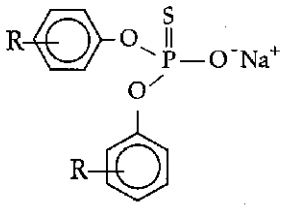
- Originally developed specifically for flotation of copper ores in acid circuits (pH 3-5). They are now used in both acid and alkaline circuits for copper-molybdenum ores, and in alkaline Zn circuits.
- In alkaline circuits, they are more selective than their corresponding xanthates.
- *AERO 204* promoter is a stronger collector than AERO 203 promoter, and is often used to improve coarse particle recovery.
- *AERO 758* promoter is a formulated product that is designed to improve flotation kinetics and froth characteristics/properties.

6.1.3 Phosphorous-based collectors

A. Aryl AEROFLOAT and AERO promoters



Diaryl Dithiophosphate



Diaryl Monothiophosphate

A.1 Dithiophosphates

AEROFLOAT 25 promoter – Acid form. Good for Ag, Pb, Cu and activated Zn sulfides.

AEROFLOAT 31 promoter – This is based on AEROFLOAT 25 promoter, but contains a secondary collector to improve silver flotation. Widely used for flotation of Pb from Pb/Zn ores and Cu/Pb from Cu/Pb/Zn ores. Improves Ag recovery from these ores.

AEROFLOAT 241 promoter – This is the ammonium salt of AEROFLOAT 25 promoter. Water soluble in all concentrations. Most selective of all liquid AEROFLOAT promoters. Widely used for flotation of Pb from Pb/Zn ores, and as a secondary collector for some copper ores.

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AEROFLOAT 242 promoter – This is the ammonium salt of AEROFLOAT 31 promoter. It is water soluble, but should be made up at minimum 10% strength to avoid precipitation of the secondary collector. Widely used for flotation of Pb from Pb/Zn ores and Cu/Pb from Cu/Pb/Zn ores. Improves Ag recovery from these ores.

AERO 7310 promoter – This is similar to AEROFLOAT 241 promoter but with a higher activity.

Comments

- ***AEROFLOAT 25*** and ***31*** promoters have considerable frothing properties, much more so than their ammonium salts, ***AEROFLOAT 241*** and ***242*** promoters.
- In alkaline circuit, the aryl AEROFLOAT promoters have a much lower tendency than xanthates to float pyrite, pyrrhotite, and unactivated sphalerite.
- Unlike xanthates, the aryl AEROFLOAT promoters are stable in acid circuit; however, lose their selectivity against iron sulfides. Consequently, ***AEROFLOAT 25*** and ***31*** promoters can be used as strong, non-selective sulfide promoters for bulk flotation in acid circuit.
- ***AEROFLOAT 25*** and ***31*** promoters should be added to the pulp full strength. Because they are in the free acid form, pre-mixing with water or ***AEROFLOAT 241*** or ***242*** promoters, or any other aqueous product could release toxic H_2S gas. This precaution does not apply to the addition of these reagents to pulps in the amounts normally used for flotation.

Physical characteristics

AEROFLOAT promoters	Color	S.G.	Viscosity (cps) 25°C**
25	Dk. Brown --- Blk.	1.19	100-200
31	Dk. Brown --- Blk.	1.19	250-500
241*	Dk. Brown --- Blk.	1.13	300-800
242*	Dk. Brown --- Blk.	1.13	300-600
7310	Yellow --- Brown	1.14	80-100

*Water Soluble -- Solution strength of AEROFLOAT 242 promoter should never be less than 10%.

**Brookfield Model LVF No.2 spindle, 30rpm

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A.2 Monothiophosphates

AERO 5688 promoter is a novel collector based on monothiophosphate chemistry. In commercial use at a number of operating locations around the world, AERO 5688 promoter is particularly effective for selective flotation of precious metals in alkaline circuits (pH > 7.0). It is also effective in the flotation of sulfide minerals and precious metals in acid circuits. In moderately alkaline circuits (pH 7-10), it can be used for selective flotation of copper sulfide minerals and precious metals from ores in which the presence of highly activated iron sulfide minerals precludes the use of other sulfide collectors; in fact with respect to iron sulfides, AERO 5688 promoter is one of the most selective of the available sulfide collectors in alkaline circuits.

Typical properties	**AERO 5688 promoter**
Appearance	Clear amber to red liquid
Specific Gravity, @ 20°C (68°F)	1.20
pH	>13
Viscosity, Brookfield LVT, cps @ 20°C (68°F) Spindle#2 @ 60 rpm	15-35
Freezing Point	
Crystallization begins, °C (°F)	2 (36)
Pourable Slurry forms, °C (°F)	-10 (14)
Product Solidifies, °C (°F)	-16 (3)
Freeze-thaw Stability	Good
Conductivity (µmhos)	23.6-24
Solubility in Water	Infinite

Comments/Primarily used in the flotation of:

- Base metal sulfides, gold/silver and PGMs from ores in acid circuit (pH 3-7).
- Selective gold/silver and copper sulfides flotation in mildly alkaline circuits (pH 7-10).
- Used in conjunction with traditional sulfide collectors to improve precious metals recovery in alkaline circuits.
- Flotation of cement copper in LPF process.
- In acid circuits, dosage requirements for *AERO 5688* promoter are significantly lower than those for the more traditional sulfide collectors. Experience also indicates that these collectors improve flotation kinetics, especially of slow floating gold particles.

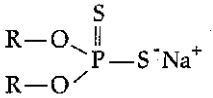
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- Dosage rates are usually in the range of 5 to 50 g/t for base metal sulfide ores and up to 100 g/t for precious metal ores.
- *AERO 5688* promoter can be fed directly to the circuit, or can be diluted with water to any strength. For ease of metering, it is often diluted to 5-10 % strength.
- *AERO 5688* promoter exhibits some frothing properties.

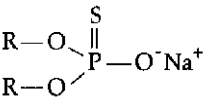
A.3 Formulated P-based product

AERO 8985 promoter is a formulated product that is used for Cu-Au Ores, where it provides optimum recovery of both Cu and Au by combining the advantages of dithiophosphates and monothiophosphates.

B. Alkyl AEROFLOAT and AERO promoters



Dialkyl Dithiophosphate



Dialkyl Monothiophosphate

B.1 Dithiophosphates

Sodium AEROFLOAT promoter – (R=ethyl). Used mainly for selective flotation of Cu from Cu/Zn ores where Zn minerals tend to float readily; for flotation of activated Zn sulfides where selectivity against iron sulfides presents a problem. Very selective against iron sulfides.

AEROFLOAT 208 promoter – (R=ethyl + sec. Butyl). Selective collector for copper ores. Excellent collector for native Au, Ag and Cu.

AEROFLOAT 211 promoter – (R=isopropyl). Selective collector for Cu and activated Zn minerals. Stronger collector than Sodium AEROFLOAT promoter.

AEROFLOAT 238 promoter – (R=sec. Butyl). Widely used in Cu flotation and for increasing by-product Au recovery. Combines good collecting power with good selectivity against iron sulfides.

AERO 3477 promoter – (R= isobutyl). A strong, but selective collector for Cu, Ni and activated Zn minerals. Improves recoveries of precious metals, particularly those of the platinum group metals.

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AERO 3501 promoter – (R=isoamyl). Used for flotation of Cu and activated Zn minerals, especially for coarse middlings. Applications are similar to those of AERO 3477 promoter, but tends to generate more froth.

AERO 5430 promoter – (R=isobutyl). A "low-frothing" version of AERO 3477 promoter. Used when maximum froth control is desired.

AERO 5474 promoter – (R=isoamyl). A "low-frothing" version of AERO 3501 promoter. Also used when maximum froth control is desired.

Physical properties

AEROFLOAT promoters	Sodium	208	211	238
Appearance	Colorless to yellow liquids			
pH	13.0 - 13.7			
sp.gr., 30°C	1.20	1.15	1.15	1.12
Viscosity (cps)				
0°C	22	25	31	45
30°C	6	7	8	12
Boiling Point, °C	103	103	103	103
Crystallization Starts, °C	-4	-12	-10	-12
Pourable Slurry Forms, °C	-9	-15	-10	-13
Solidification, °C	-13	-29	-20	-26
Freeze-Thaw Stability	Good			

Physical properties

AERO promoters	3477	3501	5430	5474
Appearance	Colorless to yellow liquids			
pH	13.0 - 13.7			
sp.gr., 30°C	1.12	1.08	1.07	1.05
Viscosity (cps)				
0°C	41	38	2000	2200
30°C	11	10	750	550
Boiling Point, °C	103	103	107	107
Crystallization Starts, °C	2	4	<-20	<-20
Pourable Slurry Forms, °C	-13	-4	-	-
Solidification, °C	-25	-9	-	-
Freeze-Thaw Stability	Good			

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Comments

- The alkyl AERO and AEROFLOAT promoters are more selective against iron sulfides in alkaline circuit than the corresponding xanthates.
- Sodium AEROFLOAT and *AEROFLOAT 208, 211* and *238* have minimal effect upon froth generation.
- Sodium AEROFLOAT and *AEROFLOAT 208, 211* and *238* are poor collectors for galena, making them the ideal choice for selective flotation of Cu from Pb.
- For many ores, the alkyl AERO and AEROFLOAT promoters are used as the principal collector, in conjunction with a xanthate as a secondary or scavenger collector. The longer chain ones are, however, often used as the sole collector to insure maximum selectivity.

B.2 Monothiophosphates

AERO 6697 promoter is a novel collector based on monothiophosphate chemistry, similar to AERO 5688 promoter in many of its collector properties. AERO 6697 promoter is in commercial use at a number of operating locations around the world. The choice between AERO 5688 and AERO 6697 promoters depends on the mineralogy/ore type, gangue mineralization, and frothing characteristics. On any particular ore, both products should be tested. For a description of typical applications, refer to Section on AERO 5688 promoters.

Physical properties

AERO 6697 promoter

Appearance	Clear yellow to amber liquid
Specific Gravity, @ 20°C (68°F)	1.14
pH	>13
Viscosity, Brookfield LVT, cps @ 20°C (68°F) Spindle#2 @ 60 rpm	15-35
Freezing Point	
Crystallization begins, °C (°F)	2 (36)
Pourable Slurry forms, °C (°F)	-10 (14)
Product Solidifies, °C (°F)	-16 (3)
Freeze-thaw Stability	Good
Solubility in Water	Infinite

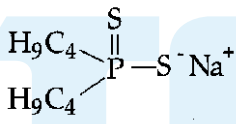
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B.3 Formulated P-based product

AERO 7249 promoter is a formulated product that is used extensively in many Cu-Au plants, where it provides optimum recovery of both Cu and Au by combining the advantages of dithiophosphates and monothiophosphates, and provides excellent selectivity against iron sulfides.

C. Dialkyl dithiophosphinates

AEROPHINE 3418A



AEROPHINE 3418A promoter is a unique, P-based sulfide collector. It was originally developed for the flotation of copper and activated zinc minerals. It has since been found to be an invaluable (and often irreplaceable) collector in the beneficiation of complex, polymetallic, and massive sulfide ores. On these ores it provides very selective separations. It is highly effective for galena and precious metals, especially silver. Its main attributes are strong collecting power but with excellent selectivity against iron sulfide minerals, unactivated sphalerite and penalty elements. On many ores, the dosage required may be considerably lower than that needed for traditionally-used non-selective collectors such as xanthates. Other characteristics include:

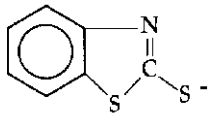
- Low frothing contribution, even on ores containing clay minerals.
- Fast kinetics.
- Good collection of coarse middling particles.
- Excellent collector for precious metals, PGM, galena, and copper sulfides from complex, polymetallic or massive sulfide ores.

AERO 6931 and Reagents S-4604 and S-7583 promoters

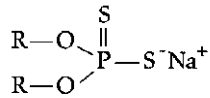
These collectors were developed recently as lower-cost versions of AEROPHINE 3418A promoter. Comparative testing should always be conducted, to ensure that metallurgical results are equivalent to those obtained with AEROPHINE 3418A promoter.

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6.1.4 The 400 series of AERO promoters



and



Mercaptobenzothiazole *Dithiophosphate*

AERO 400 promoter – Used mainly for flotation of gold-bearing pyrite in acid and neutral circuits.

AERO 404 promoter – Widely used for the flotation of tarnished and secondary Cu minerals, tarnished Pb and Zn minerals, and precious metals in alkaline circuit. Excellent collector for pyrite and auriferous pyrite in acid and neutral circuits.

AERO 407 promoter – A stronger collector than AERO 404 promoter. May substantially replace xanthates in many applications, while being more selective against iron sulfides in alkaline circuit. Useful for treating a wide range of precious and base-metal ores, particularly those of Cu, Ni and Zn. Excellent for bulk flotation of poly-metallic ores and pyritic gold ores in acid circuits.

AERO 412 promoter – A stronger collector than AERO 407 promoter with substantially the same applications.

Physical properties

AERO promoters	400	404	407	412
Appearance	← Colorless to Yellow Liquid →			
Boiling Point, °C	103	104	103	103
Freezing Point, °C	N/A	-2	-7	9
pH	>12	← 11.5 - 13.0 →		
sp.gr., 25°C	1.26	1.15	1.17	1.16
Viscosity (cps)				
0°C	N/A	21	20	–
30°C	N/A	6	6	7
Solubility	← Completely Water Soluble →			

N/A= Not Applicable

Comments

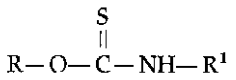
- Generally stronger collectors than the corresponding alkyl AERO and AEROFLOAT promoters, but still more selective than xanthates against iron sulfides in alkaline circuit. Use of xanthate as a secondary collector is sometimes helpful in providing maximum recovery.

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- Compared to alkyl dithiophosphates, longer conditioning times or addition to grinding mill is sometimes beneficial.
- Although originally developed mainly for the flotation of tarnished Pb ores, the 400 series of AERO promoters are now widely used in the flotation of most base-metal and precious metal ores. For the flotation of "oxide" Cu, Pb and Zn minerals, pre-sulfidization is usually required.

6.1.5 Nitrogen-based collectors

A. Dialkyl thionocarbamates

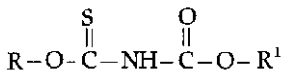


Dialkyl Thionocarbamate

AERO 3894 promoter

This oily collector was originally developed for, and is still used in, the selective flotation of copper ores in alkaline circuits. However, due to its high selectivity, it generally requires the conjoint use of a xanthate to insure maximum recovery of middling (composite) particles. Being water-insoluble, addition to the grinding circuit is often beneficial.

B. The Functionalized Thionocarbamates



Alkyl Alkoxycarbonyl Thionocarbamate

In view of the limitations of the dialkyl thionocarbamates mentioned above, Cytec in the mid 1980's developed a series of funtionalized thionocarbamates with the intention of producing collectors that combine the selectivity of the dialkyl thionocarbamates and the collecting power of xanthates. The other objective was to develop collectors which would allow selective flotation of copper ores containing iron sulfides under mildly alkaline conditions (pH 8-10)

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in contrast to the higher pH values required to depress pyrite when using xanthate and other collectors. Essentially this was achieved by the incorporation in the collector molecule of an O-containing (ethoxycarbonyl) functional group, thereby augmenting the role of the S functional group. The introduction of this second functional group lowers the pKa of the molecule by several orders of magnitude compared to that of dialkyl thionocarbamates. This allows the collector to be effective at lower pH values. (for further discussion, see Section 5) Further, the second functional group provides for the formation of more favorable and stronger metal complexes and, therefore, stronger adsorption. This has been demonstrated by sequential adsorption studies. For example, AERO 5415 and AERO 5460 promoters have been shown to replace previously adsorbed dialkyl thionocarbamate from the mineral surface but, on the other hand, dialkyl thionocarbamate does not replace previously adsorbed AERO 5415 or AERO 5460 promoters. They are especially effective for copper-rich minerals such as chalcocite, digenite, covellite and bornite. They are poor galena collectors, as all thionocarbamates are.

AERO 5415, AERO 5460 promoters

These two collectors are structurally similar, but AERO 5460 promoter being the higher homologue is the more powerful of the two and, therefore, especially suitable for the recovery of coarse middlings particles, whilst being only slightly less selective. Both of these collectors are now in wide commercial use (both as-is or as components of customized formulations) for the flotation of Cu, Cu-Mo and Cu-Au ores. In most cases, the dosage required of these collectors is lower than that for the traditional collectors, in addition to providing considerable savings in lime costs.

Comments

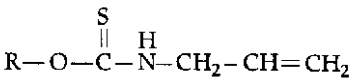
- Being insoluble in water, addition to the grinding circuit or a conditioning step ahead of flotation may be beneficial. However, in many cases *AERO 5415* and *AERO 5460* promoters are more readily dispersible than the dialkyl thionocarbamates and allyl alkyl thionocarbamates (depending upon pH and other conditions). Consequently, in many cases, addition to the head of flotation is possible and indeed may be preferable. The best point of addition should be determined by laboratory and plant testing.
- Because of their high collecting power in moderately alkaline circuits, and their high selectivity against iron sulfide minerals,

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the preferred rougher flotation pH for these collectors is usually in the range of 8 to 10, compared to the typical range of 10 to 12 required with other collectors. Similarly, in the cleaner circuits, the pH required is lower than that necessary with other collectors.

- Operating in the lower pH range not only provides a considerable reduction in lime costs but, on ores containing significant amounts of clay and other slimes, also reduces pulp viscosity. This usually enhances flotation efficiency or permits operating the circuit at higher % solids.
- It has been well established in practice that the use of ***AERO 5415*** and ***5460*** promoters generally enhances the recovery of precious metals.
- They are stable hydrolytically in a wide pH range.

C. Allyl Alkyl Thionocarbamates



Allyl Alkyl Thionocarbamate

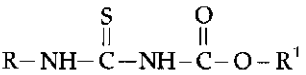
AERO 5100 promoter

AERO 5100 promoter is a modified version of IPETC, with incorporation of an allyl group attached to the nitrogen, which increases its collecting power but retains its known selectivity against iron sulfide minerals. Due to its very low solubility in water, it sometimes has a flattening effect on the froth, especially if overdosed. The optimum point of addition – to the grind, to a conditioner, or staged-addition – should always be determined by experiment. If a flat, dry froth is still a problem, the conjoint use of a small amount (10% to 20% of the AERO 5100 dosage) of a short-chain dithiophosphate such as Sodium AEROFLOAT or AEROFLOAT 208 promoter, is often helpful.

The principal uses of AERO 5100 promoter are in the flotation of copper, activated zinc, and precious metals. It is an extremely poor collector for galena and is therefore an excellent choice for floating ores which contain only nuisance amounts of lead, or for selective flotation of copper in Cu-Pb-Zn ores.

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D. The functionalized thioureas



Alkyl Alkoxycarbonyl Thionocarbamate

The only thiourea used commercially prior to 1989 was thiocarbanilide (diphenyl thiourea). Its use was confined mainly to that of a secondary collector for enhancement of Ag recovery in Pb/Ag and Ag ores. Its availability only as a dry and difficult-to-disperse powder (extremely insoluble in water) severely restricted its use for other applications. Research by Cytec in the 1980's led to the development of an easy-to-use liquid thiourea collector with a wide range of applications. This was achieved by the incorporation of an alkoxycarbonyl group in the thiourea molecule, similar to that used for functionalized thionocarbamates (see Section 6.1.5.B). The functionalized thiourea is now used commercially as a formulated product.

Although they are similar to the functionalized thionocarbamates in their collector properties on most ores, they have been found to be the preferred collectors for chalcopyrite and coarse chalcopyrite middlings in some ores. Laboratory and plant tests have indicated that they are particularly effective for Au and Ag minerals. Excellent for activated sphalerite. They are poor galena collectors. Thus they can be used for float copper minerals selectively from complex sulfides containing lead. Selective against iron sulfides and unactivated sphalerite in a wide pH range.

In contrast to the analogous thionocarbamates, the functionalized thiourea is quite effective at pH > 10.5; this is attributed to the higher pKa and the stability of the thiourea functional group.

They are hydrolytically stable in a wide pH range, perhaps more so than the analogous thionocarbamates because of the enhanced basicity imparted by the additional nitrogen and because of the higher stability of the C-N bond. Laboratory tests and plant usage indicate that they do not have much influence on froth characteristics.

AERO 5500 promoter

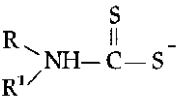
This functionalized thiourea-based oily collector, is an excellent collector for copper minerals, especially chalcopyrite. It is also a good collector for metallic gold and silver.

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AERO 5540, 5560 promoters

These combine the performance attributes of both functionalized thionocarbamates and thioureas. As a result they have a more general applicability. AERO 5560 promoter, being the higher homologue, is stronger than AERO 5540 promoter.

E. Dithiocarbamates



Mono and Dialkyl Dithiocarbamates

The use of dithiocarbamates in sulfide flotation is as old as that of xanthates. Their collector properties are similar to those of xanthates in many respects. They are excellent collectors for Pb, Zn, and Ni minerals

They are much more stable than xanthates, even in acid circuits. Consequently, they are particularly effective for the flotation of most sulfides and precious metals in acid and neutral pH circuits.

They are more expensive than xanthates and are usually used as secondary collectors.

Reagent S-8474, S-8475 promoters

These are liquid products. Easy-to-handle. Stable. Can be fed as-is or as a solution in water (can make solutions of any strength).

Reagent S-9411 promoter

This is a solid product. Readily soluble in water like xanthates. Aqueous solutions are much more stable that those of xanthate.

6.1.6 Special formulations

AERO 4037, 6682, 7518 and ***reagents S-7151, 7380, 7640, 8399, 8718, 8761, 8880, 8985, 9020 promoters***.

These collectors have all been custom-formulated to meet the requirements of individual copper, gold and zinc ores, and are based on the Cytec collector chemistries discussed in previous sections. The applications of some of these products are described

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later in this section. For more information on these products, or to plan a test program to optimize a product for your particular application, please contact your Cytec representative.

6.1.7 Important notice

Some batches of products containing alkoxycarbonyl thionocarbamates and thioureas may contain more than 0.1% ethyl carbamate as a side-reaction product. As a result, these products are classified as potential carcinogens. Please refer to the exposure control and personal protection sections of the relevant Material Safety Data Sheets for the appropriate safe handling and personal hygiene procedures.

As a result of continuing research and development by Cytec, new and improved versions of these products, AERO 5700 and 5800 promoters, have been added to this product line. New products that provide longer shelf life, greater stability, improved environmental friendliness, and superior performance levels are currently in the later stages of development. Please keep in close contact with your local Cytec representative for the latest developments.

Section 6.2 Frothers

Frothers were among the first reagents developed for mineral concentration by froth flotation; they remain a critical part of the suite of reagents used today. As a class, they are relatively low molecular weight organic compounds containing oxygen bound to carbon. They must have the property of generating a froth that is capable of supporting and enriching a mineral. The froth formed by these compounds must have certain characteristics, such as:

1. It must have the correct film properties so that the valuable mineral will attach to the bubble surfaces but the gangue minerals will not.

2. It must be stable enough to support a considerable weight of mineral and mobile enough to carry that mineral to the lip of the cell and then to the launder for recovery.

3. It must be sufficiently transient for the bubbles to break down and re-form continuously, so that the water and gangue minerals drain back into the pulp.

4. It must not be so stable that it does not break down in the launders and sumps, yet it must be capable of forming again when air is introduced in subsequent flotation stages. The importance

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of achieving an optimal froth bed can not be overemphasized, since this is where all the enrichment of the valuable minerals occurs as a result of hydrophilic gangue particles draining back into the pulp while the hydrophobic valuable minerals remain in the froth.

There are many subjective terms used to describe the characteristics of a flotation froth e.g., "stable", "effervescent", "persistent", "sticky", "brittle", "free-flowing", "mobile", "selective", "unselective", "loose", "tightly-knit", "dry", "wet or watery" and so on. From the operator's point of view, it is probably sufficient to consider froths as falling into two categories:

1. Froths in which the bubble membrane is relatively thin. Such froths tend to carry less water (i.e. are dry), to entrain less gangue slimes (i.e. they are selective), and to be relatively less stable and persistent.

2. Froths in which the bubble membrane is relatively thick. Such froths tend to carry more water (i.e. are wet), to entrain more gangue slimes (i.e. they are less selective) and to be relatively stable and persistent.

Pine-oil and cresylic acid were among the earliest commonly-used frothers, but these have now mostly been replaced by synthetic alcohols and glycols.

6.2.1 Alcohol frothers

The alcohol frothers currently in use consist of branched or cyclic hydrocarbon chains containing between five and eight carbon atoms. They may also contain a variety of other compounds formed during their manufacture. The type and amount of these secondary compounds can have a significant effect on their performance and the type of froth they produce. They are only sparingly soluble in water so are fed "as-is" to the flotation circuit. Because of their low persistence, they are often stage-added to the flotation circuit. They tend to produce the type of froth described in the first category above.

6.2.2 Glycol frothers

The ones in common use consist of polypropylene or polyethylene glycols and their ethers. They are readily soluble in water so can be diluted to any given strength. Besides their particular structure, their molecular weight plays a significant role in their performance. The

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glycol frothers tend to produce the type of froth described in the second category above. Because of their persistence, stage-addition may not be necessary. Due to their solubility and low vapor pressure, they have a greater tendency to be returned to the flotation circuit in the recycle water.

6.2.3 Cytec's frothers

The following frothers have a sufficiently wide range of applicability to fulfill any flotation requirement. Relatively broad recommendations are given for each frother. These recommendations are based on practical experience and should be used only as a guide when selecting frothers for testing.

AEROFROTH 65 frother

A polyglycol that exhibits strength and longevity in flotation circuits. AEROFROTH 65 frother has been used extensively over the world in many hard-to-froth flotation circuits to provide a froth at low consumption.

OREPREP F-507 frother

A water-soluble polyglycol consisting of a blend of three dissimilar molecular weights to provide a wide range of tolerance to different ore types and pH. Especially useful in conventional flotation cells for the flotation of coarse particles at high pH, as well as in column flotation cells.

AEROFROTH 70 frother

A low molecular weight alcohol frother is used when selectivity is important for feed containing a higher than normal percentage of fines. It has found a high degree of acceptance in coal, lead sulfide, and graphite flotation at neutral to slightly alkaline circuits.

AEROFROTH 76A frother

A frother that has a wide range of utility in the flotation of various types of circuits. It is the preferred frother when a slightly more stable and persistent frother is required as compared to either AEROFROTH 70 or MIBC.

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AEROFROTH 88 frother

This alcohol-based frother has found wide use in coal and industrial minerals flotation, especially where clays and other types of slime minerals are present.

OREPREP F-501 frother

A frother which generally provides faster kinetics and lower consumption in metallic sulfide flotation circuits than other alcohol frothers. F-501 is noted for a more rapid flotation of minerals in the first bank of conventional rougher flotation circuits and has been credited with increasing recovery if the operators do their part in removing the mineral-laden froth.

OREPREP F-521 frother

A frother formulated to lower consumption, improve longevity in the rougher float row, and improve pH tolerance as compared to conventional alcohol frothers. F-521 is designed to do this without a loss of operating control that often accompanies many formulated frothers which are designed to be stronger.

OREPREP F-523 frother

A frother that is considered by many operators as the best compromise frother for use in high pH, medium to coarse particles in the rougher feed, high solids, and requirement for longevity. This frother is especially noted for use in large sulfide flotation plants at high pH that have less than 60% recycle water from the flotation process.

OREPREP F-533 frother

A formulated product developed for specific customers who found OREPREP F-521 frother to be too weak in a high pH system, yet found OREPREP F-523 to be too strong when the plant practiced 100% process water recycle.

OREPREP F-515 frother

A frother that is applicable to the same conditions as OREPREP F-507, except when the feed rate is increased above the design of the plant and an increase in kinetics is required while maintaining a strength that is approximately to slightly less than that of OREPREP F-507 frother. OREPREP F-515 frother has been used to replace OREPREP F-507 at 10%-15% higher dosages while increasing the

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kinetics in order to handle the increased coarse particles that accompany feed tonnage that exceeds plant design.

OREPREP F-549 frother

A frother that provides a different approach. Instead of developing a formulated product to provide the different properties of strength versus selectivity, this is accomplished by providing a specific molecular family group that exhibits the properties of alcohol joined with a polyglycol, often used when the alcohols are not persistent enough, and the polyglycols are too persistent.

6.3 Modifying agents

In addition to collectors and frothers, a large number of other reagents usually referred to as "Modifying agents" are used in the flotation of sulfide ores. This is especially true in the case of complex ores, where two or more valuable minerals have to be separated from each other, e.g. Pb/Zn ores, Cu/Zn ores Cu/Pb/Zn ores, Cu/Mo ores, Cu/Ni ores etc.

These modifying agents cover a variety of functions; for example, pH modifiers, depressants, activators and dispersants.

6.3.1 pH modifiers

Most minerals exhibit an optimum pH range for a given collector. While some minerals can often be floated at the natural pH of the ores, in most cases the pH has to be adjusted for maximum recovery and selectivity. The most commonly used reagents for alkaline circuits are lime and soda ash. For acid circuit flotation, the most commonly used reagent is sulfuric acid. These three modifiers are generally the most cost effective. Other pH modifiers are also used occasionally when difficult separations are involved.

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6.3.2 Depressants

A. Inorganic depressants

The principal ones used and their typical applications are as follows:

Cyanide	Depression of iron sulfide minerals such as pyrite, pyrrhotite and arsenopyrite. Depression of Zn minerals during Pb flotation from Pb/Zn ores.
Ferrocyanide	Depression of Cu and Fe sulfide in Cu/Mo separation.
Sulfoxy species	Depression of Zn and Fe sulfides during flotation of Cu and Pb minerals, and depression of Pb minerals in selective flotation of copper minerals. Also used in conjunction with starch for the depression of Pb minerals during Cu/Pb separation.
Zn Sulfate	Used alone, or in combination with cyanide, for depression of Zn minerals in the flotation of Pb/Zn, Cu/Zn, and Cu/Pb/Zn ores.
Dichromates	Used for the depression of Pb minerals during Cu/Pb separation.
Sodium sulfide & Hydrosulfide	Used for the depression of Cu and Fe sulfide minerals in Cu/Mo separation.
Nokes Reagent & Anamol D	Used for the depression of Cu and Fe sulfide minerals in Cu/Mo separation
DETA (Diethylene triamine)	Used for the depression of pyrrhotite in Cu/Ni ores.
Permanganates & other oxidizing agents	Can be useful in the separation of pyrite from arsenopyrite

B. Natural organic depressants

Quebracho & Lignin sulfonates	Depression of Fe sulfide minerals.
Dextrin, Starches	Used in the depression of weathered silicates and carbonaceous matter.
CMC & Guar gum	Used in the depression of magnesium silicates such as talc and pyroxene. Especially useful in the flotation of PGM and Ni ores.
AERO 633 depressant	Used for the depression of carbonaceous minerals in the flotation of base metal sulfide ores.

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C. Synthetic polymeric depressants

Over the past several years, Cytec has conducted extensive research on the development of synthetic polymeric depressants to address some of the drawbacks associated with the aforementioned traditional depressants. These new products offer many potential advantages: better dosage-performance and lower treatment costs, ease of handling, lower toxicity, ease of structural modifications to suit different applications and ore variability, and consistency from batch to batch.

Reagent S-7260 depressant

This product has shown considerable promise in both laboratory and plant tests for the depression of Cu and Fe sulfides in Cu/Mo separation. The dosages required are often one-tenth of those required for traditional depressants such as NaHS and Nokes reagent. Under certain conditions a combination of AERO 7260 depressant and NaHS has given the best performance. In these cases a small amount of NaHS is used to provide the initial ideal pulp potential range of −450 to −500 mV (Au electrode vs. Ag/AgCl). One of the important advantages of using this combination is that the depressant effect is not adversely affected by aeration, as it is in the case of NaHS alone.

Other applications include: depression of iron sulfides and sphalerite in Cu and Pb circuits; depression of penalty elements, such as Sb, As and Bi, in Cu and Cu/Pb circuits; depression of sulfide minerals during the flotation of talc and other non-sulfide gangue minerals from sulfide ores or concentrates.

Reagent S-7262 depressant

The applications of this depressant are similar to those of AERO 7260, but this product is recommended where maximum selectivity is required.

Reagent S-7261A depressant

This functionalized polymer is used for the depression of pyrrhotite in Cu, Ni, Pb, and Zn circuits.

Reagent S-8860 and S-9349 depressants

These functionalized polymers are used for the depression of Mg silicates such as talc, pyrophyllite, serpentines, olivines and pyroxenes. The benefits of these depressants have been demonstrated on

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a plant scale on ores as those of PGMs, Ni and Pb. As general replacements for natural polysaccharides such as guar, dextrin, and CMC, the full benefits of these depressants on other ores are still being investigated. Indicated advantages include lower dosages and treatment costs, ease of handling, and improved metallurgy.

S-7260, S-7262, S-7261A, S-8860 and S-9349 are available as low-viscosity solutions with little or no odor and can be diluted further to any strength required for ease of handling and feeding. The best addition point can be determined only by careful laboratory testing and is dependent on the type of separation in question. The order of addition of collector and synthetic depressant is also dependent on the type of separation and the metallurgical objectives. However, both laboratory and plant experience to date suggest that the addition of polymer **after** collector addition provides the best selectivity and control.

These new polymeric depressants are fully compatible with the typical collectors in use and do not alter or require any adjustment or control of pulp redox potentials.

In addition to the five products mentioned above, various modifications of these products for use in specific applications are in the experimental stage. For more information check with your nearest Cytec representative.

6.3.3 Activators

Certain minerals do not float well with the use of only a collector, but require prior activation.

The most commonly used activators are:

$CuSO_4$	Activation of Zn sulfide and Fe sulfide minerals such as pyrite and pyrrhotite when the latter contain values such as Au, Ni and PGM elements.
Pb Nitrate or Pb Acetate	Used for the activation of antimony sulfide minerals such as stibnite.
NaHS	Commonly used prior to collector addition for the activation of Cu, Pb, and Zn minerals.
NaCN	Acts as a surface cleaning agent or "activator" to improve the flotation of PbS.

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6.3.4 Dispersants

Many ores contain significant quantities of clay minerals and other "primary slimes". These can have an adverse effect on flotation metallurgy. This can be due to a combination of factors such as, (a) increasing pulp viscosity which adversely affects air bubble distribution and froth drainage/mobility, (b) slimes can form a coating on the surface of valuable minerals thereby inhibiting their flotation.

The usual practice for minimizing the aforementioned effect of "slimes" is to conduct the flotation at lower percent solids to reduce the pulp viscosity. However, this also reduces the effective residence time in the flotation circuit. Consequently the use of both inorganic and organic dispersing and viscosity reducing agents is commonly practiced. These include sodium silicate, soda ash, various polyphosphates, and low molecular weight polyacrylates such as CYQUEST 3223 and CYQUEST 3270 dispersants.

Section 6.4 Flotation practice for sulfide ores

6.4.1 Copper ores

Most copper ores today are mined from porphyry deposits, though a few vein-type deposits are still being exploited. Nevertheless, the choice of reagent suite for flotation of these ores depends more on the type and amount of the various minerals present than on the origin of the ore. The major considerations include:

- The ratio of chalcopyrite to secondary copper minerals such as chalcocite, covellite, bornite etc.
- The amount and activity (tendency to float) of the iron sulfide minerals such as pyrite, marcasite, and pyrrhotite.
- To what extent, if any, the copper minerals are tarnished or oxidized.
- The presence of minerals containing penalty elements such as arsenic, antimony, and bismuth.
- Whether or not the ore contains recoverable amount of gold and silver, and how these are associated with the other minerals.
- Whether the ore contains significant amounts of primary slimes such as clays and other talcose minerals.
- The natural pH of the ore pulp after grinding.
- The degree of liberation of the various valuable and gangue minerals.

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The use of a lime circuit is practically universal in the flotation of copper ores. Lime alkalinity is generally maintained in the pH range of 9.5 to 11.5 in the rougher circuit and as high as 12.0 in the cleaner circuits. The higher pH serves to depress the iron sulfide gangue minerals which are commonly present. The pH can also influence the froth structure and flotability of the copper minerals.

These characteristics are adversely affected below some minimum pH value which varies from ore to ore, especially when xanthates and dithiophosphates are used. Some of the new chemistries such as the 5000 Series collectors developed by Cytec may allow for operation at considerably lower pH values (pH 8-10, for example). If free metallic gold is present, the use of lime should be carefully controlled since excessive lime concentrations have been reported to have a depressing effect on the gold. If lime depression of gold becomes a problem, soda ash can be used in place of lime. In a limited number of operations, flotation is carried out at natural pH without any pH regulating agents, or in acid circuit.

The choice of collectors can be made on the basis of the mineralogy of the ore, metallurgical objectives, and the operating conditions. In existing plants, the choice of collectors is influenced by the pH of the operating circuit and whether or not the pH can be changed. For new orebodies, a thorough investigation of representative chemical families, selected on the basis of ore characteristics, will be required. Statistical methods can be used to optimize operating conditions (see Section 12). Best metallurgy is usually obtained by taking advantage of the unique chemistries of the Cytec proprietary products. Plant experience in the past 10 years has established that Cytec's 5000 Series collectors, and formulations containing these, can offer a wide range of benefits such as:

- Very high selectivity against pyrite, pyrrhotite, unactivated sphalerite, and galena in mildly alkaline circuits.
- Selectivity against arsenic and antimony minerals.
- Significant reduction in lime usage.
- Rapid flotation kinetics especially of coarse middlings resulting in improved metals recovery.
- Better copper/moly separation compared to xanthate.
- Less sensitive to pulp potential changes than xanthate.

The 5000 Series collectors can sometimes be used with xanthate to meet a specific metallurgical objective.

In the case of slightly oxidized or easily tarnished copper ores, AERO 404, 407, and 412 promoters are in commercial use in conjunction with the 5000 Series collectors and xanthate. Best

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metallurgy is usually obtained when the former collectors are added to the grinding mill or a lengthy conditioning stage, in amounts from 5 g/t to 50 g/t.

In acid circuits, excellent performance has been observed with AERO 6697 promoter, AERO 5688 promoter, and the 400 Series promoters. All these have been used commercially for many years.

Copper sulfides in massive iron sulfide host are usually finely disseminated with pyrite and pyrrhotite. The intimate mineral associations may require very fine grinding for adequate liberation of the copper minerals. Preference should be given to selective flotation rather than bulk flotation of the sulfides; the rougher concentrate may still require regrinding to achieve satisfactory liberation and concentrate grades. The choice of collectors is similar to that for porphyry copper ores, except that the most selective collectors are utilized. These include AEROPHINE 3418A collector, the 5000/7000 series such as AERO 5415, 5460, 5500, 5540, 5560, 7518, and 7380 collectors. All these collectors can be used alone or in conjunction with dithiophosphates such as sodium AEROFLOAT, AEROFLOAT 211 and AEROFLOAT 238 promotors. The optimum collector chemistry should be established by a systematic laboratory study. If necessary, small amounts of ethyl or isopropyl xanthate can be used as an auxiliary collector. Stage-addition of collectors may be desirable to enhance selectivity.

For ores with high pyrite and/or pyrrhotite content, increased selectivity is sometimes achieved by the use of sulfur dioxide or alkaline sulfites. Recently, several synthetic polymeric depressants have been developed. These have many advantages over the traditionally-used depressants in terms of performance, safety, ease of handling, and environmental aspects. Examples of synthetic polymeric depressants are Reagents S-7260, S-7261, S-7262, and related products. (see Section 6.3)

For copper ores that contain precious metal values, the collector selection should include AERO 6697, 5688, and 7249 and 3418A promoters, in addition to the 5000 Series prompters mentioned above. AEROFLOAT 208 promoter is also well recognized as a good promoter for native gold and silver. A small amount of xanthate may sometimes be necessary, especially in the scavengers, to maximize recovery. If some of the gold is associated with copper oxide minerals, or tarnished iron and copper sulfides, the use of AERO 6493 promoter, in conjunction with the Cu-Au collectors mentioned above, can improve gold recovery.

In any of the copper flotation circuits discussed above, if "slimes" pose a problem by reducing recovery or grade, the use of a slimes dispersant or depressant is highly recommended. Examples include

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the S-7260 series and CYQUEST 3223 dispersant, either alone or in combination with sodium silicate or soda ash. (see Section 6.3)

Oxide and metallic copper ores

"Oxide" copper is a general term used to describe non-sulfide copper minerals found in oxidized zones of copper deposits. These non-sulfide copper minerals include malachite $Cu_2CO_3(OH)_2$, pseudomalachite $Cu_5(PO_4)_2(OH)_4$, azurite $Cu_3(CO_3)_2(OH)_2$, chrysocolla $(Cu, Al)_2H_2Si_2O_5(OH_4).nH_2O$, cuprite Cu_2O, atacamite $Cu_2Cl(OH)_3$, paratacamite $Cu_2(OH)_3Cl$, tenorite CuO, and native Cu. All of these minerals are referred to in this paper as "well-defined oxide copper minerals".

"Acid Soluble copper" (or AS Cu), "Non-Sulfide copper (NS Cu)", and "oxidized" copper (ores or minerals) are terms used in the industry to describe "Oxide Copper" minerals. All of the terms are rather vague and none of them clearly defines the various copper species present in the ore. These terms are often used interchangeably, but preference is given to AS Cu because the chemical assays obtained for "oxide" copper are based on dilute acid digestion of the ore.

Oxide copper minerals generally do not respond well to traditional methods of concentration using known sulfide copper collectors. Their recovery in a froth flotation circuit requires special treatment. The traditional method involves sulfidization (at -500 to -600 mV vs. a combination Sulfide Ion Electrode) using sodium sulfide (Na_2S), sodium hydrosulfide (NaSH), or ammonium sulfide ($(NH_4)_2S$) followed by flotation using xanthate or other sulfide collectors (Jones et al, 1986; Nagaraj and Gorken, 1989). Sulfidizing agents are usually stage-added for both efficacy and control. The use of NaSH will reduce excessive alkalinity which Na_2S can cause. A pH greater than 10.5 can adversely affect copper oxide mineral recovery. Sulfidization is best conducted using a sulfide ion electrode or a noble metal electrode; the former is strongly recommended. Oxide copper minerals will float within certain limits of pulp redox potentials. These limits may be broad or narrow and slightly different for each oxide mineral. For an ore containing several oxide copper minerals, it is common to have varying froth mineralization in different sections of the flotation circuit as the pulp potential changes. Chrysocolla is generally found to respond poorly to sulfidization-flotation. Many of the collectors used for copper sulfide flotation are also applicable for the flotation of sulfidized copper oxide minerals. Some collectors have been found to be particularly effective for sulfidized oxides. Examples of these include AERO 3302, AERO

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5100, and AERO 407 or 412 promoters, often in combination with a small amount of xanthate.

In principle, the sulfidization-flotation method is quite attractive, but in practice it suffers from two major disadvantages: (a) it is difficult to control the dosage of the sulfidizing agent; an excess causes depression of both sulfide and oxide minerals, and an insufficient amount produces poor recoveries, and (b) the different oxide minerals respond differently to sulfidization (Nagaraj and Gorken, 1989; Soto and Laskowski, 1973; Castro et al, 1974; Deng and Chen, 1991), and frequently sulfidization simply fails to provide acceptable oxide copper recovery.

The decision to recover oxide copper minerals from an ore depends on whether the ore contains sufficient oxide copper to be economically viable and whether such oxide copper is in a form that is amenable to flotation. It is often assumed that sulfidization-flotation is the preferred method for oxide copper recovery, but this is not necessarily valid until other options have been evaluated.

A wide variety of collectors has been tested in the laboratory for oxide copper flotation without sulfidization. These include a large number of organic complexing agents, fatty acids, fatty amines, and petroleum sulfonates (Nagaraj, 1979; Nagaraj, 1987; Deng and Chen, 1991). Except for a very limited use of fatty acids (which are quite non-selective), none of the proposed reagents has been used in an operating plant because of high cost, consumption, and inadequate performance. Alkyl hydroxamates, however, are among the very few collectors that have shown significant promise.

Alkyl hydroxamates are marketed under trade name AERO 6493 promoter. Extensive laboratory studies and plant experience on a wide variety of oxide and mixed sulfide-oxide ores from around the world have shown that well defined oxide copper minerals such as malachite, cuprite, tenorite, etc., are floated by AERO 6493 promoter. Certain copper occurrences in the ore, for example copper-containing goethite, are not amenable to flotation and they are not recovered by AERO 6493 promoter. This observation is generally overlooked. Even if species such as Cu-containing goethite were made to float, they would produce a very low-grade concentrate, which may not be a desired product (direct leaching is perhaps better in such cases). Experience has shown that any lack of performance with AERO 6493 promoter is usually attributed to mineralogical constraints in the ore. A microscopical examination, verified by microprobe work, is strongly recommended before embarking on any flotation testing program. Relying solely on chemical assays of AS Cu will lead to erroneous conclusions and will prevent a mean-

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ingful cost-benefit assessment of AS Cu recovery by flotation. Due to similar reflective light microscopy characteristics, goethite and Cu-bearing goethite can easily be misidentified as cuprite by the untrained eye. Cu-bearing goethite will also report as acid soluble copper in chemical analyses. Misidentification of Cu-bearing goethite as cuprite will lead to the erroneous conclusion that cuprite is not recovered by alkyl hydroxamates.

AERO 6493 promoter should be added "neat" or "as-is". At temperatures below 20°C, this collector may begin to solidify and it may be necessary to warm it slightly. For laboratory tests, AERO 6493 promoter can be added to the floatation cell either in the rougher stage along with the sulfide collector(s) and/or frother, or to the scavenger stage. The recommended conditioning time is 1-3 min. For plant evaluation, AERO 6493 promoter can be added either to the mill discharge/cyclone overflow (along with sulfide collectors, if this is necessary), or to the scavenger circuit. The appropriate addition point will have to be determined in the individual plants. The frother dosage and froth depth may need adjustment because AERO 6493 promoter may have a tendency to enhance frothing on certain ore types. Addition of AERO 6493 promoter to the grinding mill is generally not recommended in view of the fact that there is an iron-rich environment in the mill which may cause loss of hydroxamates via complexation with iron species.

Gangue species that readily generate slimes, for example clays, sericite, limonite, etc. may interfere with oxide copper flotation with hydroxamate and cause excessive frothing. One obvious solution would be to include a desliming step. If this is not feasible, then a dispersant such as sodium silicate or CYQUEST 3223 antiprecipitant may be necessary. These can be added either to the mill or to the flotation bank. They can also be stage added. Typical dosages are 200-500 g/t for sodium silicate and 25-50 g/t for CYQUEST 3223 dispersant. Dispersant dosage must be selected carefully, because an excess of dispersant may hinder or even depress oxide copper flotation. Soda ash can be used as a dispersant and pH modifier in non-lime circuits. It is important to note, however, that oxide copper minerals slime easily and, therefore, any desliming step may result in copper losses in the slimes fraction.

If the ore contains large amounts of pyrite or pyrrhotite, they may be depressed using sodium cyanide, sodium metabisulfite, SO_2 or a combination of these. These depressants should be added prior to hydroxamate addition. Typical starting dosages are 25-100 g/t for sodium cyanide, 100-400 g/t for sodium metabisulfite, and 500-1000 g/t for SO_2. Again, the dosage of these depressants must be evaluated carefully because they can hinder oxide copper flotation.

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The froth character associated with the use of AERO 6493 promoter is very important. An excessive froth is indicative of one or more of the following: (a) dosage of the hydroxamate is too high, (b) ore has problem gangue minerals, (c) ore has activated pyrite or pyrrhotite, (d) ore has large amounts of goethite (limonite), hematite, or magnetite. If the froth has a tendency to flatten and additional frother does not help, it may be indicative of a more fundamental problem related to the adsorption of hydroxamate on undesired minerals.

Optimum pH range for oxide mineral flotation with AERO 6493 promoter is 8.5-10. If a copper circuit is operating at pH values much greater than, say, 10.5, this may pose a problem for effective use of hydroxamate. In such cases, addition of hydroxamate to the scavengers would be preferable since the pH of the pulp in the scavengers would be lower than that in the rougher. Minor pH adjustment in the scavenger circuit may be possible, but pyrite flotation may be enhanced at lower pH values if xanthate is the collector. Alternatively, the entire circuit can be run at a lower pH by using a selective sulfide collector such as the 5000 series and related collectors. This will not only be beneficial to the performance of hydroxamates, but also result in savings in lime cost.

Dosages: 25-100 g/t appear to be appropriate for initial phase of testing. The optimum dosage will depend on the oxide content of the ore, the nature and extent of iron-containing gangue and silicates, and the amount of pyrite or pyrrhotite present.

An alternative to sulfidization-flotation and alkyl hydroxamate flotation for oxide mineral recovery, is the LPF process (Leach-Precipitation-Flotation). The ore is leached with sulfuric acid (which will also dissolve chrysocolla, if present) and the copper in solution is precipitated on to iron powder. The precipitated copper (and copper sulfide minerals, if present) is then floated in acid circuit. Perhaps the best collectors for this application are AERO 6697 promoter and AERO 5688 promoter which have been used in commercial operations.

Metallic copper, if present in the ore, responds readily to flotation, preferably in a low pH circuit. The most effective collector for recovery of metallic copper is Reagent S-7151 promoter. AERO 404 and 407 promoters have also been used commercially with success.

6.4.2 Copper-molybdenum ores

Where molybdenite is present in copper ores in economic quantity, it is floated with the copper sulfides to produce a bulk Cu-Mo concentrate. Subsequently, the Cu sulfides and molybdenite are separated in the Mo circuit by depressing Cu sulfides and floating

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the naturally hydrophobic molybdenite. The oily collector AERO 3302 promoter and related products have found acceptance at a number of plants in the bulk Cu-Mo circuit to enhance the recovery of molybdenite. In view of their high efficacy for molybdenite, and selectivity for copper sulfides, they should be the primary choice in collector combinations for treating these types of ores. Their use has also increased recovery of accessory gold values sometimes associated with these ores. AERO 3302 promoter and related products are added to the grinding mill in dosages of 5-25 g/t. A second collector is usually necessary for maximizing copper recovery. The choice of a secondary collector is dependent upon the amount of pyrite in the concentrate and its degree of activation. It is also common practice to add 20-50 g/t of hydrocarbon oil, such as diesel or fuel oil, to enhance the flotation of molybdenite.

Cu-Mo separation

In the Cu-Mo separation circuit, the molybdenite is floated using hydrocarbon oil while the Cu sulfides and pyrite are depressed as described below.

1. Sodium hydrosulfide, sodium sulfide or ammonium sulfide is used to depress the copper sulfides and pyrite. A recent trend in Cu-Mo separation has been toward the use of this process with sodium hydrosulfide as the preferred reagent. The use of nitrogen gas instead of air has been introduced at some plants. The nitrogen reduces the oxidation and consumption of the sodium hydrosulfide, making the separation process more efficient. In the final molybdenite cleaning stages, some operations are using cyanide to depress residual copper sulfides and pyrite. In some cases, the final molybdenite concentrate may have to be subjected to a cyanide or a ferric chloride leach treatment to remove residual copper.
2. Noke's reagents, which are thiophosphorus or thioarsenic compounds, are widely used in the separation of molybdenite from copper, causing depression of copper minerals and pyrite. The final stages of cleaning usually require the addition of sodium cyanide.
3. Cu sulfides and pyrite can also be depressed under more oxidizing conditions with the use of sodium or potassium ferrocyanide. Oxidizing agents such as hypochlorite or hydrogen peroxide were used at one time to improve the efficiency of the separation. Similarly a steaming or a roasting process was used in the past to

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strip collector coating from Cu sulfides and pyrite prior to the addition of ferrocyanide. Sodium cyanide is often used in the Mo cleaners to assist in depression of copper sulfides and pyrite.

4. Recently Cytec has introduced several experimental polymeric depressants to replace the hazardous inorganic depressants mentioned above and to improve the efficiency of the separation process (see Section 6.3.2).

6.4.3 Lead ores

Galena is the most common lead mineral. Depending on the degree of oxidation, lead ores may contain significant amounts of cerussite and anglesite. As galena is a soft, high specific gravity mineral, slimming due to overgrinding of the galena is a problem. To reduce this problem, unit cells in the grinding circuit, or stage grinding with flotation between stages, is practiced at some operations.

Galena generally floats easily and is recovered with AEROPHINE 3418A, AEROFLOAT 241 or 242 promoters, and ethyl or isopropyl xanthate. AEROPHINE 3418A, AEROFLOAT 241, and AEROFLOAT 242 promoters are more selective than xanthates in the presence of zinc and iron sulfides. Stage addition of these collectors can further enhance the selectivity. AEROPHINE 3418A and AEROFLOAT 242 are the preferred collectors for argentiferous galena.

The 400 series of AERO collectors, in particular AERO 404 promoter, may help the recovery of partially tarnished galena. The 400 series of AERO collectors may tend to collect zinc sulfides and therefore, care should be used with its application. Dosages generally range from 2 g/t to 10 g/t.

AEROPHINE 3418A promoter has given very good test results on a number of lead ores and is in plant use as the principal collector for galena. Its use should be considered for treating lead or argentiferous lead ores, particularly where selectivity against iron and zinc sulfides is desired. AEROPHINE 3418A is an exceptional collector for silver and argentiferous galena.

Galena floats readily in the presence of cyanide, and it is actually required in some cases to activate the galena, probably due to its cleaning action on galena particle surfaces. Cyanide is utilized to effect a more selective flotation of galena in the presence of zinc and iron sulfide minerals.

Best flotation conditions are obtained in natural or slightly alkaline circuits up to pH 8.5. Control of pH with soda ash, rarely with caustic soda, is preferred. However, many operations use lime without detriment to galena recovery.

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6.4.3.1 Oxidized lead ores

The degree of oxidation in lead ores may range from slight tarnishing of the galena to complete oxidation. The most common oxide lead minerals are cerussite, anglesite, and plumbojarosite.

In the case of tarnished galena, AERO 404 promoter is effective, sometimes with prior addition of small amounts of sodium sulfide or sodium hydrosulfide. Where the oxide lead minerals are present in appreciable amounts, it is the usual practice to float the lead sulfides first, as described in the foregoing paragraphs under Lead Ores. Then, if present, the zinc sulfide is floated, followed by flotation of the lead minerals. Either sodium sulfide or sodium hydrosulfide is used as a sulfidizing agent. AERO 404, 407, or 412 promoters in combination with isopropyl or amyl xanthate are the preferred collectors for the lead minerals. It is common practice to add the sulfidizing agent as well as collectors in stages throughout lead rougher flotation. The dosage of sulfidizing agent varies a great deal, but will usually be between 500 g/t to 2500 g/t. Pulp potential controlled addition of sulfidizing reagents should be considered. (see Section 6.4.1 under copper oxide ores).

Anglesite usually does not respond well to the preceding flotation process, but can be recovered by a gravity concentration process. AEROPHINE 3418A promoter has been used in plants for the flotation recovery of argentiferous plumbojarosite.

The use of soda ash as an alkalinity regulator and water-softening agent should be considered. Sodium or ammonium phosphate, used from 500 g/t to 2500 g/t, has also been found helpful in improving flotation of lead oxide minerals.

6.4.4 Zinc ores

The most common zinc sulfide minerals, sphalerite and marmatite, rarely float well without pre-activation by copper sulfate. The copper sulfate is added to a conditioning step, usually at the same point as, or after, lime addition. The optimum conditioning time will vary with different ores. Adsorption of copper ion will take place on the surfaces of the zinc minerals which will than behave as the corresponding copper minerals. Some plants have found the order of lime and copper sulfate addition will influence flotation results.

Zinc minerals generally occur in the presence of pyrite. Therefore, in order to obtain the highest and most economical concentrate grade, it is important to use:

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- a selective collector or collector combination.
- the appropriate copper sulfate dosage
- the appropriate collector dosage
- the appropriate pH level (8.5 -12.0)
- the correct order of addition of lime and copper sulfate

There is increasing evidence that there are strong interactions between each of the factors listed above. Any test program should vary all of these factors in a designed experimental program. Testing of one variable at a time will not reveal any interaction and will rarely reveal an optimum.

Pyrite activation may take place during the conditioning step with copper sulfate. If this tendency exists, it can usually be overcome with the addition of lime to further raise the pH and depress the pyrite. It is, therefore, common practice to float zinc sulfides at pH levels from about 8.5 to as high as 12.0. Cleaning of zinc concentrate is generally carried out at pH levels that are in excess of 10.0. Generally the use of an AERO or AEROFLOAT promoter as the principal collector, with possibly some xanthate as an auxiliary collector, provides maximum recovery with the desired selectivity. It is recommended that such collector combinations be added together in one or more stages as required.

The most widely used AERO and AEROFLOAT promoters in zinc flotation are Sodium AEROFLOAT, AEROFLOAT 211, AERO 4037, and AERO 3477 promoters. The 400 series of AERO promoters as well as AERO 5100 and 7279 promoters also are excellent collectors for zinc minerals. Their use in zinc circuits has resulted in savings in collector costs due to a reduction in total collector consumption. AEROPHINE 3418A and AEROFLOAT 242 promoters are each in plant use and should be included in any zinc sulfide flotation investigation.

6.4.4.1 Oxide zinc ores

The most common oxidized zinc minerals are smithsonite, hydrozincite, hemimorphite, and willemite, often in association with carbonates and siliceous gangue. Usually these oxide zinc minerals occur with the lead sulfide and oxide minerals as well as the zinc sulfide minerals. The most widely accepted technique for the flotation of oxide minerals has been in use at zinc operations in the Mediterranean area for a number of years. By the use of sodium sulfide and an amine, both carbonate and silicate zinc minerals are recovered. The amines which should be investigated are AERO 8625

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and AERO 8651 promoters. Recent studies indicate promising results with the use of AERO 6493 promoter without sulfidization.

As most oxide zinc minerals occur in mixed sulfide-oxide ores of lead and zinc, the procedure consists of floating the lead and zinc sulfides, then the lead oxides and finally the oxide zinc minerals. The feed to the oxide zinc flotation circuit requires careful desliming prior to flotation and is then floated with a relatively large amount of sulfidizing agent and a cationic collector, such as AERO 8625 and AERO 8651 promoters, with frother added as required. Investigators originally reported best results at pH levels between 10.5 and 11.0, although some ores respond well to the process at lower pH levels. Reagent consumptions are usually of 1000 g/t to 7500 g/t sodium sulfide or sodium hydrosulfide, and 50 g/t to 300 g/t cationic collector. Soda ash and sodium silicate can be used to improve flotation.

Less common is a process which utilizes large amounts of amyl xanthate, in conjunction with sodium sulfide. In this process, desliming prior to zinc oxide flotation is also necessary. Consideration should be given in this latter process to evaluating the more powerful alkyl dithiophosphates in particular AERO 3477 and 3501 promoters, as well as the series promoters.

6.4.5 Lead-zinc ores

Most lead-zinc ores can be classified as complex ores, and recovery problems will increase with the degree of dissemination of the minerals. The presence of large quantities of pyrite increases the problem of recovery and selectivity. Frequently, lead-zinc ores contain small amounts of copper minerals as well as silver and gold. When free gold is present, the use of lime as an alkalinity regulator in the lead circuit may be undesirable, as it has been reported to have a depressing effect on free gold recovery. It has also been noted that zinc minerals may become activated by lime. Therefore, the use of soda ash as the pH regulator in the lead circuit may be necessary. If the ore contains a significant amount of soluble salts, the use of polyphosphates or CYQUEST 3223 antiprecipitant may be beneficial.

General practice in the treatment of lead-zinc ores is to float the lead concentrate first, while depressing the zinc minerals. After lead flotation, the zinc minerals are reactivated with copper sulfate and floated selectively.

Depression of the zinc minerals and pyrite in the lead flotation circuit is usually achieved with cyanide, almost invariably in combination with zinc sulfate. The amount of zinc sulfate is usually three

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to five times that of cyanide. These depressants are added to the grinding circuit ahead of lead rougher flotation and, if required, to the head of lead cleaning circuit. If the lead rougher concentrate is reground before cleaning, depressant may be added to the regrinding mill. Sodium sulfite or bi-sulfite is finding increasing use as a zinc mineral depressant in combination with cyanide and zinc sulfate. In some cases, it is the only depressant used. When gold and silver are present, it is preferable to premix zinc sulfate or zinc oxide with cyanide to form the zinc cyanide complex in order to prevent dissolution of the gold and silver. A 2:3 ratio of Zn to NaCN is utilized in preparing the zinc cyanide complex. More detailed instructions for preparing this complex are given in the Complex Copper-Lead-Zinc ores section following.

In the case of unoxidized lead-zinc ores, flotation of the lead is accomplished as previously described under Lead ores, generally with AEROPHINE 3418A or AEROFLOAT 241 or 242 promoters used alone or in combination with xanthate. AEROFLOAT 25 and 31 promoters have been used in the past but these collectors have been superseded.

Where zinc sulfides tend to float because of slight pre-activation, best results may be had with AEROFLOAT 241 due to its high degree of selectivity against zinc minerals. The use of AEROPHINE 3418A promoter, as the lead collector, also should be included in any collector screening program where zinc minerals tend to float into the lead concentrate due to undesired pre-activation. Alcohol-type frothers are generally preferred for improved selectivity.

Some lead-zinc ores contain carbonaceous shale or graphitic compounds which tend to dilute the lead concentrate, retard lead flotation rate or cause an unmanageable froth condition. The use of AERO 633 depressants in amounts up to 250 g/t in the lead roughing circuit and lesser amounts in the cleaning circuit can alleviate these conditions.

After flotation of the lead minerals, the pH of the zinc circuit feed (lead circuit tailings) may require adjustment with lime, conditioned with copper sulfate and floated as described under Zinc Ores. The amount of copper sulfate required for adequate zinc mineral activation varies, but is of the order of 50 g/t for each percentage point of zinc. The most favorable sequence of addition of lime and copper sulfate should be established experimentally, although lime is usually added prior to copper sulfate addition. Additional lime may be required after copper sulfate addition in order to increase the pH to the desired level.

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The undesired presence of dolomite or magnesite fines in the zinc concentrate may be reduced by the use of lignin sulfonate, quebracho or similar tannin extract, usually added to the zinc cleaner circuit.

A number of operations recover a pyrite concentrate after flotation of the lead and zinc minerals. This is usually accomplished by adding sulfuric acid to the zinc circuit tailings to lower the pH to between 7 and 8.5. The pyrite is floated with AERO 404 or 407 promoters or isobutyl or amyl xanthate. Soda ash has been used to counteract the depressing effect of lime, by precipitating the calcium ions as their carbonates. It is also possible to float the pyrite with AERO or AEROFLOAT promoters without pH adjustment with the addition of a small amount of copper sulfate for the reactivation of pyrite.

6.4.6 Complex copper-lead-zinc ores

The treatment of these ores follows a pattern which is very similar to that for Lead-Zinc Ores. The amount of copper minerals present is considerably higher and usually justifies, from an economic point of view, the production of separate copper, lead, and zinc concentrates. Therefore, the importance of selective flotation becomes even more evident.

Standard practice in treating these complex ores is to selectively depress zinc minerals, using one of the previously described methods, and float a copper-lead bulk concentrate. The copper-lead concentrate, which may require regrinding, is then separated into a copper concentrate and a lead concentrate in a separation circuit. In the copper-lead bulk flotation step, the use of very selective collectors is of great importance. AEROPHINE 3418A, AEROFLOAT 241, or AEROFLOAT 242 promoters are the recommended principal collectors sometimes used with ethyl xanthate for maximum recovery. The use of a small amount of AERO 404 promoter is recommended to improve recovery of slow floating or tarnished copper and lead sulfides, if present. Alcohol-type frothers are recommended for maximum selectivity.

Where selectivity against pyrite is a problem, aeration conditioning ahead of flotation is sometimes beneficial. Under these circumstances, investigation of the use of AEROPHINE 3418A promoter is strongly recommended, owing to its selectivity against pyrite. The use of the AERO and AEROFLOAT dithiophosphate collectors in combination with the 5000 series of AERO collectors or AEROPHINE 3418A promoter has shown improved selectivity against sphalerite, thereby sending more recoverable zinc to the zinc flotation circuit.

For some ores, it is advantageous to selectively float a copper concentrate followed by separate selective flotation of a lead concen-

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trate followed by separate selective flotation of a zinc concentrate. Successful sequential flotation of the copper, lead, and zinc concentrates requires the use of an appropriate depressant at the correct dosage prior to copper flotation for the depression of galena, sphalerite, and pyrite. A selective copper collector such as Sodium AEROFLOAT, AEROFLOAT 211, AEROFLOAT 238, AERO 5415, or AERO 5100 promoters (or one if its formulations) is added to float the copper minerals while minimizing the recovery of galena. The pulp may then be conditioned with cyanide followed by the flotation of the lead minerals with AEROFLOAT 242, AEROPHINE 3418A, or an ethyl or isopropyl xanthate. Flotation of the zinc minerals follows lead mineral flotation. Flotation of zinc minerals is completed in the usual manner as described in the Zinc ores section.

6.4.6.1 Copper-lead separation

Separation of copper from lead in a cleaned bulk concentrate is accomplished by depressing the lead and floating the copper or vise versa, the choice depending on the response of the minerals to be separated, the type of copper minerals and the relative abundance of the copper and lead minerals. Excellent descriptions of the copper-lead separation process can be found in the literature.

Depression of lead minerals

This approach is usually preferred where the amount of lead in the bulk concentrate is more than twice the amount of copper.

For the depression of galena the use of sodium dichromate (usually about 1000 g/t bulk concentrate) is common, being added just ahead of the separation circuit or to a conditioning step, as required. A small amount of a specific copper collector such as AERO 5100 or AERO 5460 promoter may be required to improve the copper flotation. The copper concentrate produced is cleaned as required with small amount of dichromate.

A second method of galena depression is treatment of the bulk concentrate slurry with SO_2 gas in an absorption tower or added to a stainless steel conditioner to provide up to 5 minutes conditioning at a pH of about 5. Small amounts of causticized starch and/or sodium dichromate may enhance galena depression. Again, a specific copper collector such as AERO 5100 or AERO 5460 promoter may be helpful in providing maximum copper recovery.

A third, seldom-used method for galena depression is the combination of ferrous sulfate and causticized starch.

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Depression of copper minerals

Although not commonly practiced, when there is less than two parts of lead to one part of copper in bulk concentrates, it may be preferable to depress the copper minerals in order to make their separation from the lead minerals. For the depression of copper minerals, cyanide (usually 250-500 g/t of bulk concentrate) or the cyanide-zinc complex are used. In this process short conditioning with cyanide is preferred and the stage addition of cyanide can be advantageous. The lead concentrate is usually cleaned at least once with small amounts of cyanide. Control of pH in the range 7.5 to 9.0 is desirable and is determined experimentally.

When using a straight cyanide separation, losses of precious metal and secondary copper minerals may occur through dissolution. These losses are largely eliminated when using the zinc-cyanide complex. This complex can be prepared on site by mixing the following ingredients in a tank with 100% freeboard:

- 100 kg of technical grade zinc sulfate ($ZnSO_4 \bullet H_2O$) containing 36% Zn) or 45 kg pure zinc oxide.
- 55 kg sodium cyanide.
- 600-650 kg (liters) cool water.

The zinc sulfate is dissolved, or the zinc oxide is slurried, in the water. If using zinc sulfate, the pH of the solution should be raised to at least pH 8 using lime, before any further steps are taken. The cyanide is then added to the tank (under agitation) and mixed until dissolved. If zinc oxide has been used, the tank will require gentle agitation to keep the fine zinc oxide in suspension. During preparation of this reagent, adequate ventilation must be provided.

From the foregoing description of accepted separation methods, it is obvious that no standard practice can be recommended. For each application, a thorough evaluation of mineralogy, and the effectiveness and economics of various separation methods will have to be made based on carefully conducted laboratory studies. This should undoubtedly involve careful selection of reagents. While other methods and variations of the above-described methods are in use, these will at least serve as a guide.

6.4.7 Copper-zinc ores

The separation of copper sulfides from sphalerite or marmatite, particularly in the presence of iron sulfides, requires careful selection of collectors, pH regulators and depressants. The following general

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procedures and reagents have been found to give good separations on many copper-zinc ores.

To minimize activation of the zinc minerals by any dissolved salts in the grinding circuit, alkalinity is maintained at pH 8 to 10 by the addition of lime and/or soda ash. If the flotation feed contains liberated precious metal values, soda ash is preferred as the principal alkalinity regulator. To further aid selectivity against iron and zinc sulfides in the copper flotation step, sodium sulfite or bi-sulfite, or zinc sulfate and cyanide, are added to the grinding circuit or the conditioner ahead of copper flotation. Sulfur dioxide may also be used, added to the conditioner ahead of copper flotation.

During the copper flotation step dithiophosphates such as AEROFLOAT 208 or 238 promoters, and AERO 3477 or 3501 promoters have traditionally been used. However, for increased copper-zinc selectivity, collectors such as AEROPHINE 3418A, AERO 5100, or AERO 5460 promoters are now recommended. The use of an alcohol-type frother is preferred to assist selectivity.

After flotation of the copper minerals, the zinc minerals are activated and floated as previously described under Zinc Ores.

6.4.8 Gold and silver ores

Gold ores

Treatment methods for the recovery of gold from gold-bearing ores depend on various factors, such as: (a) the mode of occurrence of the gold and associated minerals and (b) the grade of gold in the ore.

Ores in which the gold is associated with mostly non-sulfide gangue minerals, and is readily recoverable by gravity methods, flotation or cyanidation, are generally referred to as "free-milling" ores. The choice of treatment method for such ores depends upon (a) the grade of the gold in the ore, (b) the recoveries obtained by each method, (c) possible environmental constrains, and (d) overall process economics. If flotation is used to upgrade such ores prior to cyanidation, the common collectors used are xanthate, such as AERO 343 or 317. The use of a secondary collector such as AEROFLOAT 208, AERO 3477 or AERO 3418A promoter can often improve recoveries. If the gold is tarnished and slow-floating, the use of a 400 Series collector such as AERO 407 or 412 promoter is often helpful. By carefully designed flotation test work, Cytec has the ability to design a custom collector formulation for specific ores and process conditions.

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It is interesting to note that Nagaraj et al (1989, 1992) have reported that 100% pure metallic gold does not readily adsorb any known sulfide collectors. However, if the gold is alloyed with even a small amount of silver or copper, adsorption is significantly enhanced. Fortunately, almost all naturally-occurring gold does contain silver, usually in the range of 2 to 12 percent; this is sufficient for good collector adsorption and flotation (unless the gold surface is heavily tarnished). Other elements such as copper and tellurium are also frequently found in native gold.

Gold is commonly found in deposits which contain significant amounts of sulfide minerals, particularly the iron sulfides pyrite-marcasite, pyrrhotite, and arsenopyrite. The treatment method for these so called "refractory" gold ores depends upon whether or not significant amounts of the gold are associated by intimate physical locking with, or in solid-solution in, the iron sulfide minerals.

- Ores in which little of the gold is associated with sulfide minerals can often be treated by direct cyanidation of the whole ore. In many cases, however, results are unsatisfactory due to the adverse effect of the sulfide minerals on both cyanide consumption and gold recovery. In this case, the gold is separated from the sulfide minerals by flotation and the concentrate treated by cyanidation. The gold collectors mentioned above are suitable, but addition of lime to pH 11.0 or higher is often necessary to prevent the sulfide minerals from floating. An alternative method for these ore types is the use of AERO 6697 promoter at pH 8 to 9 to float the free gold away from the sulfides. AERO 6697 promoter is an excellent collector for gold over a wide pH range but has little tendency to float iron sulfide minerals at moderately alkaline pH levels. Thus, the consumption of lime is reduced and gold recovery is often enhanced, since lime has a tendency to depress free gold.
- For ores in which a significant amount of the gold is intimately locked with, or in solid solution in, the iron sulfide minerals, these sulfides must be floated together with any free gold, prior to further treatment of the flotation concentrate. The flotation is usually conducted at natural pH with a combination of a strong sulfide collector such as AERO 317 or 350 xanthate. In many cases, the use of a secondary collector for the free gold is beneficial. Such collectors would include AEROFLOAT 208, AERO 407, 6697, 7518 and 3418A promoters. For tarnished ores and for ores containing significant quantities of arsenopyrite, the use of copper sulfate (50 to 500 g/t) to activate the sulfides should be investigated. The flotation concentrate is then generally subjected to oxidation

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(e.g. roasting, bio-oxidation or autoclaving) prior to cyanidation to recover the gold. In some cases, the flotation tailings contain sufficient gold for them to be also treated by cyanidation.

Finally, it should be noted that much of the current global production of gold comes from ores which contain their major value as minerals of base metals, particularly copper. These ores are usually referred to as base-metal ores, but may contain sufficient amounts of gold to influence the selection of the optimum flotation reagent. The treatment of these ore types is discussed in Section 6.4.

Silver ores

Most of the silver recovered commercially is associated with the base metal sulfide ores of copper, lead, lead-zinc and copper-lead-zinc ores. Silver occurrence ranges from a minor to a major constituent in these ore types. Of major importance in the flotation of these silver bearing ores is the choice of collector, regulating agents and depressants.

In general, the silver tends to concentrate with the copper and lead sulfides in these types of ores. AEROFLOAT 242 promoter and AEROPHINE 3418A promoter are strongly preferred as collectors. AERO 7518 and AERO 7640 promoters have demonstrated good recovery of silver associated with copper sulfides. They may also be used as auxiliary collectors for silver in the flotation of argentiferous galena. Silver also occurs in association with sphalerite, arsenopyrite, and even with pyrite. In the latter case, depending on the silver content of the pyrite, a pyrite concentrate may be produced from the base metal circuit tailings, which can be treated by roasting and cyanidation for silver recovery. Silver sulfides and silver-antimony-arsenic sulfides such as argentite, polybasite, proustite, pyrargyrite, stephanite, and tetrahedrite respond best to flotation in a natural circuit. Regulating agents, such as sodium sulfide, lime, caustic soda and starch tend to depress the silver minerals. If cyanide must be used in the base metal flotation circuits, it is recommended that the zinc cyanide complex be used to reduce the dissolution of the silver.

When the silver ore contains only minor amounts of base metal sulfides, bulk flotation of all sulfides is usually the best practice for maximum silver recovery. If silver-bearing zinc sulfides, arsenopyrite, pyrrhotite and pyrite are present, copper sulfate will usually be required to activate these minerals prior to collector addition. If, on the other hand, these sulfide minerals do not contain silver, then careful use of lime may be required to prevent concentrate dilution. The use of the dithiophosphates, AEROFLOAT 242 and AERO 3477

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promoters with small amounts of a lower xanthate, such as isopropyl xanthate (usually 20-50 g/t of total collector), are recommended for these ore types. AEROPHINE 3418A, used alone or in combination with xanthate, is also recommended. AERO 7518 and AERO 7640 promoters are particularly useful when part of the silver minerals occurs as attachments to the gangue.

With partially oxidized silver-bearing ores, cyanidation of flotation tailings for silver and gold recovery may be economically justified. In addition, sulfidization prior to flotation is commonly practiced when the silver values are associated with oxide minerals such as cerussite, malachite, cuprite and cerargyrite. When sulfidization is practiced the use of AERO 407 or AERO 7151 promoters are recommended.

6.4.9 Nickel and cobalt ores

Copper-cobalt ores are treated by selective flotation, floating in order the copper and cobalt minerals, or by bulk flotation, followed by separation of the copper and cobalt minerals.

In general the preferred treatment method is selective flotation for optimum recovery of copper and cobalt in their respective concentrates. In this process, lime is added to the grinding circuit to maintain a pH of 10 to 11 in the copper circuit. The ground pulp is conditioned for 10 to 15 minutes with small amounts of sodium cyanide, about 25 g/t. Higher quantities of cyanide will tend to depress the copper. An alcohol frother, such as AEROFROTH 70 or OREPREP 501 frother, and a dithiophosphate collector, such as AEROFLOAT 208 or 238 and AERO 3477 or 3501 promoters preferred, are then added to selectively float the copper sulfides. AEROPHINE 3418A promoter also has demonstrated excellent selectivity against cobalt minerals, particularly cobaltiferous pyrite. AERO 7151 promoter also exhibits excellent selectivity and should be included in any test program. After copper flotation, the pulp is conditioned for up to 15 minutes with sulfuric acid to reach pH 8 to 9, and small amounts of copper sulfate, isopropyl xanthate and a suitable frother are added for cobalt flotation. Investigation of the use of one of the aqueous 400 series of AERO promoters or the 5000 series of AERO promoters, used neat and in combination with xanthate, is recommended for this flotation step. Rougher concentrates from both circuits are cleaned as required.

In the bulk flotation of copper and cobalt minerals, AERO 3894, 5415, and 5460 promoters have been used successfully. One of the aqueous 400 series of AERO promoters or the dithiophosphates

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mentioned in the preceding paragraph, are also recommended as collectors, operating at natural pH. The bulk concentrate, after cleaning, is fed to the separation circuit where the pulp is conditioned with lime to pH 11 and a small amount of sodium cyanide, if required, to depress the cobalt minerals, and the copper sulfides are then selectively floated.

Copper-Nickel ores

The principal sulfide minerals in copper-nickel ores are chalcopyrite, pentlandite and pyrrhotite. Platinum group metals and gold can be present in economically important amounts. As pyrrhotite is usually nickel bearing, it may be necessary to activate the pyrrhotite with copper sulfate and make a bulk flotation concentrate for maximum copper and nickel recoveries. This is usually done at natural pH with a powerful xanthate, such as isobutyl or amyl xanthate (20-50 g/t), sometimes in combination with AERO 3894 promoter (10-25 g/t) and a suitable frother. Cytec has demonstrated that partial replacement of a xanthate, up to 75%, with AERO 3477, 407 or 412 promoter has resulted in increased recovery of all metals in this bulk float. AERO promoters of the 5000 series as well as AEROPHINE 3418A, should be evaluated for improved selectivity and cost benefits.

The results of test work conducted by Cytec personnel on a sample of copper-nickel ore with the objective of bulk flotation demonstrate the synergistic effect of the conjoint use of isobutyl xanthate and AERO 3477 promoter. At a collector ratio of 1:3 xanthate to dithiophosphate, higher flotation rates and recoveries were achieved than with the use of xanthate alone.

It has been Canadian practice for many years to either:

- recover the magnetic pyrrhotite by magnetic separation ahead of flotation and then float chalcopyrite, pentlandite and some nickeliferous pyrrhotite with xanthate in a natural circuit.
- float these latter minerals first, followed by magnetic recovery of the pyrrhotite from the flotation tailing, again using a strong xanthate such as amyl xanthate.

The presence of talc or talcose type minerals requires the use of dextrin, guar gum or, as practiced in some Australian nickel operations, CMC or some similar colloid for their depression. Alcohol or low molecular weight glycol frothers are preferred for improved selectivity against the talc. Cytec's polymeric depressants, AERO 8860GL and 9349 depressants have demonstrated strong talc depressing abilities and should be evaluated.

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If the copper content justifies it, the copper-nickel concentrate is separated into a copper concentrate and nickel tailing by depressing the nickel-bearing minerals with the addition of lime to a pH of 1.5 to 12.0 together with the addition of 200 to 500 grams of cyanide per ton of bulk concentrate. Starch or dextrin may be used to assist in depressing the nickel-bearing minerals.

When it is undesirable to recover the pyrrhotite with the copper and nickel sulfides, chalcopyrite and pentlandite can be floated together without the use of copper sulfate. This is accomplished by using a collector such as AEROFLOAT 208 or 238 promoter, or AERO 3477 or 3501 promoter with, if needed, a small amount of xanthate. AERO 7151 and 7016 have demonstrated improved selectivity against pyrrhotite and are worth investigation as collectors. Cytec's polymeric depressants, AERO 7261A, 7262G and 9349 depressants have recently proved beneficial in depressing pyrrhotite and other gangue minerals in nickel circuits and should considered as an alternative to cyanide. Copper-nickel separation can then be accomplished in the same manner as described in the foregoing.

Nickel ores

The principal sulfide minerals in nickel ores are pentlandite, millerite, pyrite and pyrrhotite as is the case in some of the high-grade ores of Western Australia. Pentlandite, arsenopyrite and pyrrhotite are predominant in the case of the low grade large open pit operations of the world. Platinum group metals and gold can be present in economically important amounts in both types of ore bodies. Additionally, talc or talcose type minerals may be associated with these ores. In the case where pyrrhotite is nickel bearing, it may be necessary to activate the pyrrhotite with copper sulfate and make a bulk flotation concentrate for maximum nickel recoveries.

Flotation pH can be either neutral or alkaline using soda ash or lime. In some operations, better nickel recoveries and grade are achieved using soda ash in preference to lime. The choice of collectors can vary from strong xanthates like ethyl or amyl xanthates, to Cytec's AERO 8474, 8475 and 8649 promoters which are dithiocarbamates. The 5000 and 7000 series of AERO promoters should also be considered as mentioned in the previous copper-cobalt and copper-nickel sections. Generally, an alcohol such as OREPREP 501 or a glycol blend like OREPREP OXT140 are the frothers of choice. Cytec's polymeric depressants should be considered where pyrrhotite and or arsenopyrite minerals are to be depressed.

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6.4.10 Platinum group metal ores

Most copper-nickel and some nickel ores contain platinum group metals. Cytec's research established in the 1970's that the highest recoveries of these metals are achieved with a combination of a long-chain xanthate, such as AERO 317 and 350 xanthates, and AERO 3477 or 5430 promoters. Where the frothing properties of AERO 3477 can not be tolerated, the non-frothing AERO 5430 is preferred. The best xanthate to dithiophosphate ratio is in the range 1:1 to 1:3 and total collector usage is generally from 25 to 75 g/t. Higher recoveries are obtained with considerably higher flotation rates. More recently, such collectors as AERO 5415, AERO 5100, AERO 3302, and Reagent S-6894 have been shown to further improve flotation kinetics and overall PGM recoveries. AERO 5415 and 5100 promoters should be tested as auxiliary collectors at a dosage of 5 to 15 g/ton whereas Reagent S-6894 should be tested as a total replacement for the AERO 3477 promoter on a gram-for-gram basis.

For the depression of Mg silicate minerals such as pyroxenite, the use of Reagent S-8860GL depressant as a replacement for guar gum or CMC replacement has recently been demonstrated.

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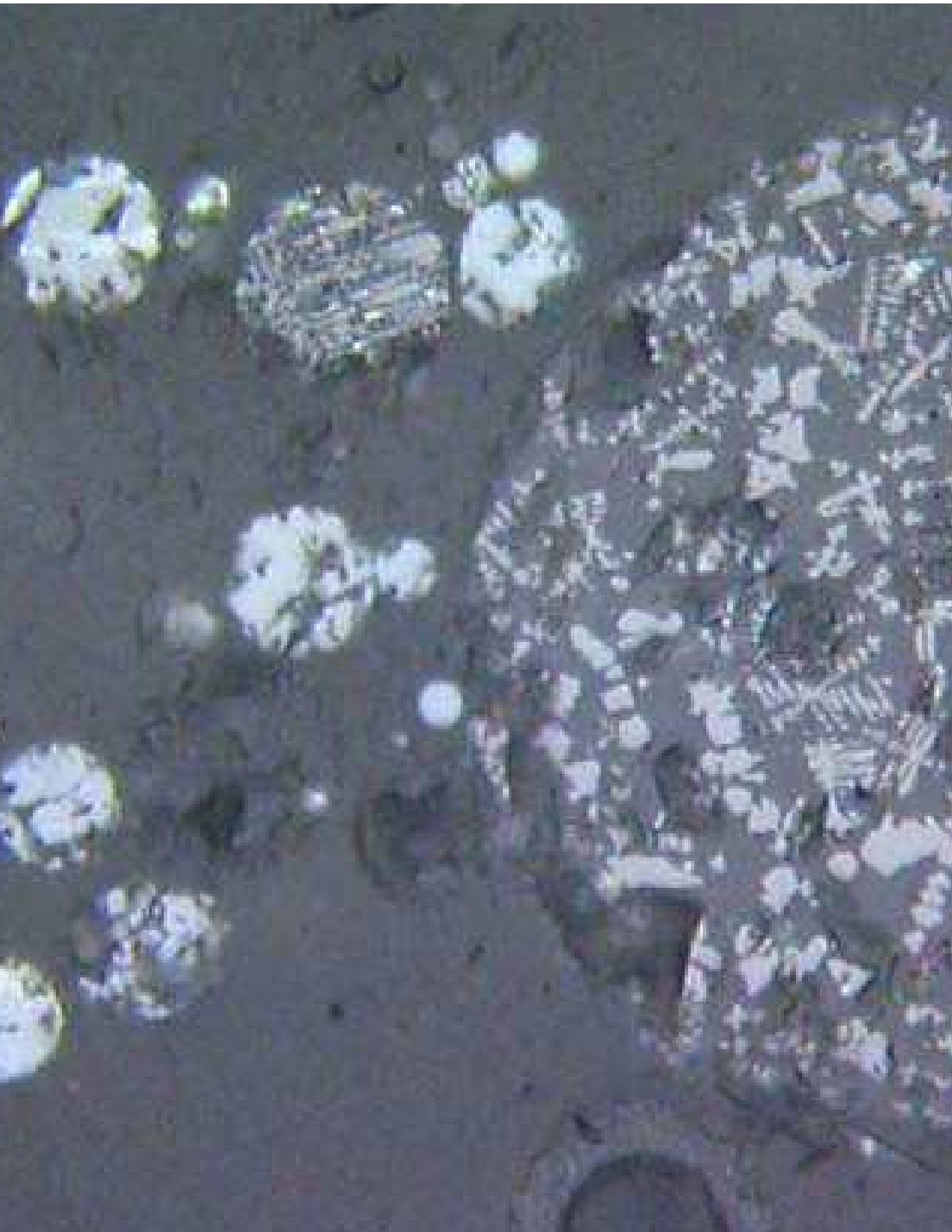
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7 Flotation of non-sulfide ores



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Section 7 Flotation of non-sulfide ores

7.1 Overview

The minerals included in this section are often referred to as "Industrial" or "Non-Metallic" minerals; their concentration by froth flotation often presents a greater challenge to the metallurgist than do metallic sulfide minerals. Nagaraj et al (1999) have discussed the major theoretical and practical differences between the flotation of sulfide and non-sulfide ores. These include:

1. Sulfide minerals have a strong affinity for S-containing ligands, and their surface chemistry is generally determined by electrochemical reactions. On the other hand, non-sulfide minerals have a strong affinity for O-containing ligands, and their surface chemistry is largely determined by ion exchange reactions. Put simply, in the case of sulfide minerals, there is strong collector adsorption by metal complexation. However, in the case of non-sulfide minerals, physical adsorption plays a significant role in addition to chemisorption. Consequently, collector adsorption on non-sulfide minerals is usually much less specific or selective than in the case of sulfide minerals.
2. In non-sulfide systems there are only small differences between the surface properties of the mineral being floated and the gangue minerals e.g. feldspar from quartz and mica, and sylvite from halite. Highly specific treatment conditions are required to make a clean separation of such mineral mixtures.
3. Many non-sulfide ores contain substantial amounts of primary slimes such as clays and iron oxides. In addition, the valuable minerals themselves are often soft and tend to form slimes during the grinding process. These slimes can cause problems in flotation such as high pulp viscosity, slime coatings of one mineral on the coarser particles of another mineral, high collector consumption caused by indiscriminate adsorption and large mineral surface areas, the reduced efficiency of attachment of ultra-fine particles to air bubbles, and dilution of the concentrate by mechanically-entrained gangue slimes in the froth. Furthermore, the physical adsorption of sparingly-soluble collectors, such as fatty acids, is much slower and less efficient for fine particles than for coarse ones.

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4. For many non-sulfide ores, the effect of water quality on flotation is significantly greater than for sulfide ores. Possible reasons for this are (a) some collectors, such as fatty-acids, can react with multivalent cations, such as calcium and magnesium, to form insoluble compounds thereby consuming collector, (b) these insoluble compounds can adsorb indiscriminately on the mineral surfaces reducing flotation selectivity, (c) soluble ions can compete with the collector for adsorption on the valuable mineral surface, and (d) some soluble species, especially iron, can adsorb on gangue minerals causing inadvertent activation.
5. The specifications for the final concentrate product are often much stricter than for sulfide concentrates. Rather than simply incurring a financial penalty, "off-spec" product may actually be unsaleable. Examples include (a) the iron content of glass-sands, (b) the carbonate content of foundry sand, (c) the CaF_2 content of acid-grade fluorspar, and (d) the specific gravity of barite for use in drilling mud.

As a result of the problems and constraints listed above, a variety of pre-treatment and processing techniques, which are relatively rare in sulfide flotation, are quite common in the flotation of non-sulfide ores. These include:

Scrubbing and desliming - This is a common pretreatment method in the processing of phosphate, feldspar, glass sand, potash, cassiterite, garnet, kyanite, and spodumene ores. The high-intensity scrubbing step is usually conducted at high solids (~ 70%) followed by thorough desliming using mechanical classifiers or hydrocyclones. The split-size varies depending on the ore, but can be as low as 10 microns for cassiterite ores to as high as 100 microns for phosphate ores. In a few cases (e.g. potash and iron ores) desliming is accomplished by selective flocculation, followed by sedimentation or flotation of the flocculated slimes.

High-solids conditioning - The flotation efficiency of many non-sulfide minerals, especially the coarser fractions thereof, is often greatly enhanced by the input of mechanical energy during the collector conditioning stage. This is accomplished by high-intensity conditioning at high solids (~ 70%). Without this step, many minerals will simply not float.

High temperature flotation - For certain ores, especially fluorspar, satisfactory separation of the value mineral from the gangue can only be accomplished by conducting the flotation at elevated temperatures e.g. 60 to 70 degrees Celsius. Fortunately, in most cases, these elevated temperatures are necessary only in the cleaning stages.

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Modifying agents – A bewildering array of reagents, both organic and inorganic, has been proposed to assist in the separation of non-sulfide minerals. A handful of them actually work in practice. The use of modifying agents is far more critical in non-sulfide flotation than it is in sulfide flotation, the main reasons being that the collectors used are generally unselective and the differences in mineral surface characteristics are usually small. Commonly used slime dispersants include sodium silicate, soda-ash, polyphosphates, and low molecular weight anionic polymers such as CYQUEST 3223 or CYQUEST 3270 antiprecipitant; these products also act as viscosity-reducing and scrubbing aids.

pH is often a critical variable in flotation of non-sulfide minerals. Sulfuric acid, soda-ash, sodium hydroxide (and occasionally ammonium hydroxide) are the usual pH modifiers.

Commonly used activators and depressants include, sodium silicate for depressing silicates and sericitic slimes, hydrofluoric acid for activating feldspar and depressing quartz, quebracho for depressing carbonate minerals and tannins, starches, lignin-sulfonates, and glues for depressing clays and iron-oxide slimes. For the future, functionalized polymers hold great promise as selective depressants. Cytec developed one such product, ACCO-PHOS 950 depressant, some years ago. It is used as a depressant for phosphate minerals in the amine flotation of silica from phosphate concentrates in Florida. Unlike natural polysaccharides, synthetic polymers provide the ability to more closely control such properties as molecular weight and degree of funtionalization. Several other experimental or semi-commercial products are available from Cytec for testing as specific gangue depressants.

Pulp density – Water is perhaps the most important modifying agent in non-sulfide flotation. Operators are often required to increase plant throughput without installation of additional flotation capacity. As a result, there is a temptation to increase pulp density in order to maintain flotation residence times; this may, or may not, be the proper thing to do. Higher pulp densities mean higher pulp viscosity, which can lead to poorer recoveries and concentrate grades, probably as a result of less efficient distribution of air bubbles in the pulp. In many cases, reducing the pulp density more than compensates for the reduction in residence time.

Finally, as with sulfide ores, thorough mineralogical studies and carefully planned and controlled investigation of all possible variables, is the only way to develop the optimum treatment conditions for any specific ore. The recommended procedures for laboratory flotation testing are not all that different from those for sulfide ores. These recommendations are covered in some detail in Section 4.

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Section 7.2 Cytec reagents

7.2.1 AERO 825, 827, 828, 850, 851, 852, 853, 854, 855, 856, 857, 858, 862, 864, 865, 866, and 869, Reagent S-9386, and Reagent S-9485 promoters

These are anionic, petroleum based sulfonate promoters most widely used for the acid circuit flotation of iron ores and iron-bearing mineral impurities from glass sands and feldspars. These promoters are also used for acid circuit flotation treatment of chromite, kyanite, and garnets. They have application for the treatment of a wide variety of complex metal-silicates, metal oxides, and tungstates.

In alkaline circuits, these petroleum sulfonate-based promoters are used for the flotation of barite. They also have application for the treatment of some carbonate and oxide ores containing copper, boron, and rare earth elements in alkaline and acid circuits.

Comments

AERO 825 and *827* promoters are the traditional petroleum sulfonate that must be dispersed in water with vigorous agitation. Hot water improves dispersion. Usually fed as a 5-20% dispersion in water. Products must be heated to 82 degrees C to reduce viscosity and improve handling characteristics.

AERO 850 promoter is a unique formulation that requires conditioning at a pH of 2.5 - 2.8 followed by flotation at a pH of 7.8 - 8.3. This product permits use of the stronger sulfonate chemistry without acid-proofing the flotation circuit. Only the conditioner requires lining to prevent acid attack of the surface.

AERO 856 promoter is formulated for the flotation of barite in an alkaline circuit. AERO 856 is a strong and yet very selective promoter yielding high recoveries of barite at high concentrate grades.

AERO 828, 851, 852, 853, 854, 855, and *857* promoters are formulated petroleum sulfonate reagents that are designed to be much more effective in circuits with high levels of heavy minerals and concentrations of ilmenite. They are much more selective than pure petroleum sulfonates and produce greater yields of silica sand and feldspar.

AERO 865 promoter is designed for circuits with high concentrations of biotite.

AERO 866 and *869* promoters are considered to be the strongest promoters for removal of iron and other heavy minerals. They are superior to other reagents in removing minerals that contain iron stains.

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Reagent S-9386 promoter - a formulated product that out performs other collectors in circuits with an excess of slimes.

Reagent S-9485 promoter - a new odorless product with a high flash point that gives improved reduction of iron on stained quartz.

7.2.2 AERO 830, 845, and Reagent S-3903 promoters

These anionic, alkyl succinamate promoters were developed to provide more selectivity than can usually be obtained with fatty acids and/or petroleum sulfonates. When used as the principal collector, AERO 830 and 845 are excellent promoters for barite, celestite, and scheelite in alkaline circuits and for cassiterite in acid circuit. AERO 3903 promoter is structurally related to 845 which was developed to provide better selectivity with some cassiterite ores which do not respond favorably to flotation with AERO 845 promoter.

AERO 830 and 845 promoters are also used as secondary collectors with fatty acids and petroleum sulfonates, usually from 5% to 20% of the total collector dosage, to provide improved metallurgy and circuit control. As such, they have found acceptance in the treatment of phosphate, fluorite, scheelite, feldspar, and glass sand ores. Particularly when used with fatty acids, the point of 830 or 845 addition has been found to have a significant influence on the resulting metallurgy. Their use should be evaluated using conditioning times ranging from the same as for the primary collector, to a very brief contact time with the pulp before rougher flotation. Generally, the short conditioning times with 830 and 845 have favored best metallurgy.

Comments

1. When used as the principal collectors, they tend to produce more froth than fatty acids and petroleum sulfonates. If this is a problem, frother addition should be reduced and stage-addition of the collector tested. Emulsification of the collector with 10 to 30% its weight of fuel-oil has been found effective in extreme cases of over-frothing.
2. Conditioning at high solids is usually not required.
3. The dosage required is often much lower than that for fatty acids and petroleum sulfonates.
4. *AERO 845* promoter is completely water-soluble. *AERO 830* and *3903* promoters are semi-liquid to soft pastes and are water-dispersible; they are usually fed as 5% to 10% dispersions.

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7.2.3 ACCO-PHOS 950 depressant

A synthetic polymeric depressant developed to reduce the loss of phosphate values floating into the silica froth product when using amine collectors. ACCO-PHOS 950 depressant is in commercial use in the second stage "reverse" flotation of silica at plants using the "double float" method of processing pebble phosphate ores. It has also shown efficacy in depressing Ca-activated silica during fatty acid flotation of phosphate.

ACCO-PHOS 950 depressant has also given excellent results for the flotation treatment of high grade phosphate ores in North Africa, where it is only necessary to float away silica gangue using amine collectors to leave behind the phosphate values.

ACCO-PHOS 950 depressant has recently demonstrated effective depression of P_2O_5 to improve fluorite concentrate grades.

Typical dosage range is 20-100 g/t in the conditioning stage prior to collector addition.

Comments

- Used to depress phosphates during amine collector flotation of silica or in fluorite flotation.
- Short contact time with pulp preferred. Add to the head of silica flotation circuit for phosphate operations or prior to the fatty acid float for fluorite flotation.
- Water-soluble liquid can be diluted to any convenient strength for feeding.

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TABLE 7-1 USAGE OF CYTEC'S 800 PROMOTERS

Reagent	Form	Usual Dosage g/ton	Usual Feeding Method	Usual Point of Addition
AERO 825 promoter	Viscous Liquid	250-150	10-30% dispersion in water	Conditioner
AERO 827	Viscous Liquid	250-1500	10-30% dispersion in water	Conditioner
AERO 828	Liquid	250-150	Undiluted	Conditioner
AERO 830	Liquid/ Paste	150-750	5-10% dispersion in water	Conditioner
AERO 845	Liquid	150-750	Undiluted	Conditioner
AERO 847	Liquid	25-100	5-15% w/Fatty Acids	Conditioner
AERO 848	Liquid	25-100	5-15% w/Fatty Acids	Conditioner
AERO 850	Viscous Liquid	250-1500	Undiluted	Conditioner
AERO 851	Viscous Liquid	250-1500	Undiluted	Conditioner
AERO 852	Viscous Liquid	250-1500	Undiluted	Conditioner
AERO 853	Viscous Liquid	250-1500	Undiluted	Conditioner
AERO 854	Viscous Liquid	250-1500	Undiluted	Conditioner
AERO 855	Viscous Liquid	250-1500	Undiluted	Conditioner
AERO 856	Viscous Liquid	250-1500	Undiluted	Conditioner
AERO 857	Viscous Liquid	250-1500	Undiluted	Conditioner
AERO 858	Viscous Liquid	250-1500	Undiluted	Conditioner
AERO 862	Viscous Liquid	250-1500	Undiluted	Conditioner
AERO 865	Viscous Liquid	250-1500	Undiluted	Conditioner
AERO 866	Viscous Liquid	250-1500	Undiluted	Conditioner
AERO 869	Viscous Liquid	250-1500	Undiluted	Conditioner
AERO 870	Viscous Liquid	25-100	10-20% dispersion in water	Conditioner
AERO S-3903	Liquid	150-750	5 -10% dispersion in water	Conditioner
S-9386	Liquid	250-1500	Undiluted	Conditioner
S-9485	Liquid	250-1500	Undiluted	Conditioner

7.2.4 AERO 702, 704, 708, 718, 722, 726, 727, 727J, 728 and 730 promoters

These are anionic, tall oil fatty acid-based promoters, most widely used for alkaline circuit flotation of iron ores and iron-bearing mineral impurities from glass sands. They are also effective reagents for the removal of carbonate minerals from foundry or molding sands. The 700 series promoters are also used for the flotation of fluorspar.

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Comments

AERO 702, 704, 708, 718, are straight tall oil fatty acid promoters with varying acid values, rosin acid content, and percent fatty acid.

AERO 722, 727, 727J, and *725* promoters are formulated tall oil fatty acids that contain surfactants and other chemical coupling agents that make them much more effective than straight tall oil fatty acids. In many applications, the use of these products has resulted in the reagent usage being reduced by as much as fifty percent. The products also reduce and/or eliminate the build-up of organic residue on the surfaces of the conditioners, flotation cells, etc. The reduction of total reagent consumption is very important in plants with closed water circuits.

AERO 727 and *727J* are very effective promoters for the flotation of phosphate.

AERO 730 is a formulated tall oil fatty acid which was developed for alkaline circuit flotation of barite.

TABLE 7-2 USAGE OF CYTEC'S 700 PROMOTERS

Reagent	Form	Usual Dosage g/ton	Usual Feeding Method	Usual Point of Addition
AERO 702 promoter	Liquid	250-1500	Undiluted	Conditioner
AERO 704	Liquid	250-1500	Undiluted	Conditioner
AERO 708	Liquid	250-1500	Undiluted	Conditioner
AERO 718	Liquid	250-1500	Undiluted	Conditioner
AERO 722	Liquid	250-1500	Undiluted	Conditioner
AERO 726	Liquid	250-1500	Undiluted	Conditioner
AERO 727	Liquid	250-1500	Undiluted	Conditioner
AERO 727J	Liquid	250-1500	Undiluted	Conditioner
AERO 728	Liquid	250-1500	Undiluted	Conditioner
AERO 730	Liquid	250-1500	Undiluted	Conditioner

7.2.5 AERO 3000C, 3030C, 3100C, and reagent S-8651 and S-9549 promoters

These are cationic promoters that are used in acid or alkaline circuits for the flotation of mica. They can also be used with the addition of hydrofluoric acid for the flotation of feldspar.

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Comments

AERO 3000C and *3030C* promoters are liquid and can be fed neat to the conditioner eliminating the difficult make-up associated with most amines. These products are very effective in the notation of mica and perform very well in both alkaline and acid circuits for this purpose.

AERO 3100C promoter is the traditional cationic amine. It is very strong and selective making it the choice reagent for optimum recovery of feldspar when used in combination with hydrofluoric acid.

Reagent S-9549 promoter - a liquid cationic collector that is odorless, has a high flash point, and is an excellent collector for feldspar, mica, and kaolin.

TABLE 7-3 USAGE OF CYTEC'S 3000 PROMOTERS (AMINES)

Reagent	Form	Usual Dosage g/ton	Usual Feeding Method	Usual Point of Addition
AERO 3000C promoter	Liquid	100-500	Undiluted	Conditioner
AERO 3030C	Liquid	100-500	Undiluted	Conditioner
AERO 3100	Paste	100-500	10-15% dispersion in water	Conditioner
Reagent S-8651	Liquid	100-500	10-15% dispersion in water	Conditioner
Reagent S-9549	Liquid	100-500	10-15% dispersion in water	Conditioner

7.2.6 AERO 6493 and 6494 promoters

These are anionic, alkyl hydroxamate-based, collectors. Their main use currently is in the flotation of colored impurities, such as Fe and Ti minerals, from kaolin clays. In this application they provide improved selectivity and ease of use, resulting in product of improved brightness. They also have made possible the treatment of kaolin clays which hitherto had been economically untreatable. (see Section 7.3). They are also used in the novel selective flocculation process developed recently to remove colored impurities from difficult-to-treat clays.

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Laboratory and plant trials have shown that they will also float various "oxide" copper minerals (malachite, cuprite, azurite, high-copper chrysocolla, and atacamite) without the need for pre-sulfidization. (see Section 6.4.1).

Comments

Both AERO 6493 and 6494 promoters are liquid at temperatures above 15ºC and can be added neat to the conditioners at room temperature. They perform well in a pH range from neutral to pH 9.0. AERO 6494 promoter results in somewhat more froth than AERO 6493 promoter and, therefore, may be preferred where this is desirable.

TABLE 7-4 USAGE OF CYTEC'S PROMOTERS
Hydroxamate Collector Line

Reagent	Form	Usual Dosage g/ton	Usual Feeding Method	Usual Point of Addition
AERO 6493 promoter	Liquid(*)	500-1000	Undiluted	Conditioner
AERO 6494	Liquid(*)	500-1000	Undiluted	Conditioner

* Liquid at temperature above 15ºC

Section 7.3 Treatment of specific ores

Barite

A large number of barite producers utilize flotation to recover and improve the grade of barite used as an additive in drilling mud, the formulation of brake shoe linings, and many other applications.

Commonly used collectors are alkyl sulfates or petroleum sulfonates. AERO 827 promoter has been used for many years in conjunction with sodium silicate to float barite concentrates. The flotation feed is conditioned at 60-65% solids at a pH of 9.5 to 10.2 which is achieved through the addition of 500 to 2000 grams per ton of sodium silicate. The normal range of AERO 827 promoter required is 500 to 1000 grams per ton. The feed is normally conditioned for a minimum of five minutes prior to introduction to the flotation cell where the pulp is diluted to 25-30% solids.

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The newest product to gain acceptance is AERO 856 promoter, a new formulated liquid product that can be fed "neat" to the conditioner. It has much greater selectivity and, on most plant feeds, has exhibited a significant increase in recovery.

Another collector that has gained wide acceptance is AERO 845 promoter, used either as the sole collector or as a replacement for 10% to 50% of the primary collector, resulting in improved grade and recovery. When used as the sole collector. AERO 845 promoter is added to the conditioner after addition of 1500 to 2500 grams per ton of sodium silicate. It is recommended that a stage addition of the AERO 845 promoter be used with a total dosage of 150 to 500 grams per ton. AERO 845 promoter is particularly recommended where selectivity against fluorite and calcite are important considerations.

A new product that was recently introduced as an improved barite collector is Reagent S-8920 promoter. It is used as a direct replacement for the other 800 promoter products. The advantages of this collector have been improved selectivity and froth control in the presence of slimes.

The combination of the 800 series promoters and sodium silicate has been widely accepted for commercial use in separating barite from such gangue minerals such as siderite, goethite, hematite, limonite, calcite, fluorite, quartz, and various silicates. De-sliming of the feed is not required.

Barite ores often are found containing fluorite. In these cases. AERO 845 promoter is the preferred collector because of the high degree of selectivity against fluorite in the presence of moderate to large amounts of sodium silicate. If the fluorite concentration is of commercial significance, the fluorite can be recovered from the barite flotation tailings by flotation with a fatty acid collector such as AERO 702 promoter. In most cases, the barite flotation tailings must be de-watered to reduce the concentration of sodium silicate prior to conditioning with AERO 702 promoter for flotation of the fluorite. Quebracho can be added in the conditioning step to depress calcite which is often present with fluorite minerals.

Cassiterite

Recovery of fine cassiterite, down to 5 µm from gravity plant tailings, by flotation is now practiced successfully at a number of operations. Typically, the tailings from gravity concentration, after removal of the plus 45µm material, are cycloned at high pressure in clusters of small diameter cyclones for removal of minus 5-7 µm slimes in preparation of flotation. If economically sufficient additional cassiterite can be liberated, the plus 45µm portion of the gravity plant tailing is reground and combined with the minus 45µm portion for cyclone treatment.

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If sulfides are present, the deslimed fines are treated in a first flotation step with xanthate, a frother and copper sulfate if required. The sulfide flotation tailing goes to the cassiterite flotation circuit for rougher flotation and usually several steps of recleaning with cleaner tails returning to the head of the rougher circuit. Concentrates produced assay in the range 10% to 30% Sn with recoveries of 50% to 70% of the tin in this circuit's flotation feed. The first successful commercial operation, utilizing the process patented by Prof. N. Arbiter, used AERO 845 promoter (200 g/t of flotation feed), AEROFROTH 65 frother and sulfuric acid to pH 2-3. This process with some modification is still in use. However, with many ores selectivity against some gangue minerals was not good and this lead to the introduction and commercial use of AERO 3903 promoter.

In more recent years the arsonic and phosphonic acids have been tested successfully on more difficult ores to improve selectivity. Of these the styrene phosphonic acid is now in commercial use.

Modifying agents and selective depressants have been evaluated and successfully introduced. Flotation is always carried out in acid circuit from pH 2 to 5 preadjusted with sulfuric acid. Where necessary, frothers such as AEROFROTH 65 or OREPREP 507, 549, 579, or 587 can be used.

Selectivity is improved by the use of sodium silicate (500-1000 g/t) and sodium fluoride (20-500 g/t) or sodium fluosilicate (20-500 g/t). Modifying and depressing agents are usually added to a 5 minute conditioning step, followed by collector to the second conditioning step, where acid and frother are also added. Automatic pH control in rougher and cleaner circuits is highly desirable in this very sensitive operation.

Coal

Flotation of fine coal in the minus 0.6 mm size range typically utilizes fuel oil as the primary collector and a frother such as Cytec's OREPREP 571 or AEROFROTH 88 frother. However, due to increased environmental concerns associated with the use of fuel oil as a collector, the industry has requested non-fuel oil collectors and Cytec has successfully introduced new, non-fuel oil ACCOAL 9628 and 9630 promoters that are approved by the West Virginia DEP. These new promoters are used in conjunction with DEP-approved OREPREP 571 or AEROFROTH 88 frothers.

Since the flotation behavior of coal plant feed varies significantly from plant to plant and often within an individual plant, optimization of ACCOAL promoters normally requires preliminary evaluation

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of the full range of promoters. This is followed by a more detailed dosage study and a plant trial with the best promoter and frother combination.

Feldspar

Feldspars are an integral part of every ceramic product produced. Potassium feldspars are used to produce high strength electric insulators, fine china, and specialty ceramic products. Sodium feldspars are used in the manufacture of glass. Finely ground feldspar is used to produce sanitary ware such as toilets and lavatories and comprises up to fifty percent of their composition. It is also used as a pigment for high traffic paints such as the traffic lane stripes on highways and it is also a key component of foam rubber.

Feldspars are found either as pegmatite (hard rock) or as highly weathered *in-situ* deposits. Both types can be concentrated via flotation but the weathered feldspars are usually more difficult as the grain surfaces are pitted and eroded creating a large increase in the surface area of the feldspar particles. The weathered feldspars are also softer and break down in processing - creating slimes which absorb greater quantities of reagents.

In either case, the feldspar minerals are usually associated with silica sand, micaceous minerals (muscovite and biotite), tourmaline, garnets, ilmenite, and other iron oxides. Feldspar can be separated from the other minerals through the use of multi-stage flotation. The following procedures are normally used:

1. Attrition scrubbing at 70% solids or greater if required.
2. Thorough desliming to remove all finely disseminated minerals.
3. To remove the mica, condition the feed at 50-60% solids with the pH adjusted to 3.0-3.5 with sulfuric acid. A tallow amine (cationic collector) such as AERO 3000C promoter is added at a dosage of 250 to 500 g/t and the feed conditioned for three minutes. The feed should be diluted to 20 - 30% solids in the flotation cell. It is often necessary to add fuel oil to the mica conditioner at a dosage of 25 to 500 g/t for optimal removal.
4. To remove the iron and other heavy minerals, the tailings from the mica float should be dewatered and placed in a conditioner where very high solids conditioning (70-75%) for five minutes with the pH adjusted to 2.5 - 3.0 is required. An anionic collector such as AERO 855 or 869 promoter is added at a dosage of 25 to 500 g/t. After conditioning, the feed enters a flotation cell where it is diluted to 20 - 30% solids.

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5. To separate the feldspar from the silica sand, the tailings from the heavy mineral float are again dewatered and placed in a conditioner where solids are adjusted to 50-60%. Sulfuric acid is added to attain a pH of 2.0 - 2.5 and hydrofluoric acid is added at a dosage of 400 to 750 g/t. A tallow amine such as AERO 3000C promoter (cationic collector) is added at a dosage of 250 to 500 g/t. A conditioning time of 3 minutes is recommended. The conditioned feed is diluted to 20 - 30% solids in a flotation cell where the feldspar is removed from the silica sand. It is often necessary to add kerosene, #2 fuel oil, or some other light oil for optimum removal of the feldspar - in particular the weathered feldspars.

Fluorite

The standard flotation reagent for fluorite is a pure oleic acid or a very high grade of tall oil fatty acid such as AERO 702 promoter, with such modifying agents as sodium carbonate, sodium silicate, starch, and quebracho, or a tannin if carbonates are present. Many operations need to heat the conditioned pulp, especially in the cleaning circuits to achieve the desired selectivity, recovery, and reagent economy.

In most standard practices, the ore is conditioned with 500 to 2500 grams per ton of sodium carbonate, (depending on the water hardness), 50 to 500 grams per ton of quebracho, followed by the addition of AERO 702 promoter at a dosage of 500 to 1000 grams per ton. In most cases, the addition of a heavy oil such as Number 5 fuel oil, is used as a froth control agent.

AERO 845 promoter has shown promise, in the laboratory and in the plant, as a partial (and occasionally total) substitute for oleic and fatty acids. One of the main advantages indicated is the possible reduction of the temperature required in the cleaning stages since AERO 845 promoter is water soluble and more selective than fatty acids.

If AERO 845 promoter is being used alone, the previously described standard practice is followed, with the exception that the AERO 845 promoter is applied by stage-addition with a recommended dosage of 100 to 500 grams per ton. In cases where AERO 845 promoter does not give satisfactory recovery when used alone, it should be tested as a 10% to 20% replacement for the fatty acid.

ACCO-PHOS 950 depressant, at dosages of 20-100 g/t with conditioning prior to conditioning with AERO 702 promoter has recently demonstrated effective depression of P_2O_5 to improve fluorite concentrate grades.

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Foundry/Molding sand

Many sands with ideal grain size and distribution for the fabrication of sand molds for metal casting contain carbonate minerals. The presence of carbonate minerals in the sand results in a reaction of the molten metal to release carbon dioxide which creates deformities in the casting.

The carbonate minerals can be removed via flotation with a tall oil fatty acid collector at a pH of 7.0 or greater.

The sand should be thoroughly washed of slimes and organic matter. The sand enters a conditioner where the pH is adjusted to be alkaline. It is very important that the percent solids in the conditioner be maintained at or near 70%. The tall oil fatty acid should be added to the conditioner at a dosage of 400 to 700 g/t of dry solids and the sand conditioned for a minimum of five minutes. The conditioned feed should be diluted to 30-35% solids in the flotation cells for optimum removal of the carbonate minerals.

If excessive sand losses are noted in flotation, the pH can normally control the losses through adjustment of one to two pH units. If the losses persist, the addition of sodium silicate at a dosage of 250 to 500 g/t in the conditioner will eliminate the losses.

Cytec's 700 series of formulated tall oil fatty acid promoters are much more selective than straight fatty acids for carbonate flotation. The dosages required are often 50% lower than for fatty acids. In addition, the heavy residue that collects on the flotation equipment with the use of a tall oil fatty acid collector is eliminated. These products are much more effective in obtaining a consistent ADV (Acid Demand Value) for foundry operators.

Glass Sand

Essentially the same procedure as described above for feldspar treatment through Step 4 or Step 5 is used to treat glass sands, depending on the minerals present in the sand deposit. If feldspars are present and to be recovered, the tailings from Step 5 are the final glass sand product. In the absence of economic feldspar values, the tailings from Step 4 would be the final silica product. Cytec's AERO 866 and AERO 869 promoters are widely utilized in such glass sand flotation operations globally and the entire 800 series of AERO promoters should be evaluated to determine the optimum collector for a particular sand deposit.

At some glass sand operations, naturally-occurring organic colloids may make a fatty acid float of the iron-bearing minerals preferable.

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After desliming, the pulp is conditioned at high solids with one of the 700 series AERO promoters such as AERO 704, 726, 727 or 730 promoters and soda ash or caustic soda to pH 8-9. Fuel oil may be added to the flotation circuit for froth control.

Iron ores

Acid circuit flotation of iron oxides was practiced for many years using the 800 series of AERO promoters in conjunction with heavy fuel oil at a pH of 3-5, adjusted with sulfuric acid following high solids conditioning. Depending on gangue minerals present, fatty acid-based 700 series of AERO promoters can be used in a neutral to acid circuit, again adjusted with sulfuric acid.

Reverse flotation of silica to produce a final iron ore concentrate is being practiced to float the quartz and other silicates using ether-amine collectors and AEROFROTH or OREPREP frothers as required.

Kaolin clay

Kaolinite, the principal mineral in china clay has the commonly accepted composition of $2H_2O.A1_2O_3.2SiO_2$. Kaolin clays are generally found as sedimentary deposits formed by the weathering of feldspathic rocks. The kaolinite is almost invariably associated with impurities such as iron oxides, rutile, silica, feldspar, mica, sulfides and organic matter. For most applications, these impurities have to be removed from the kaolin clay to produce a useful end product.

Processed kaolin clays can be divided into two broad categories:

a) Dry-processed clays of low to medium purity, for use in relatively low-cost applications such as ceramics and other structural materials.

b) Wet-processed kaolin of high purity and brightness, used mainly as filler and coatings in high-grade paper, and also in paints and plastics.

Low-grade clays are produced employing relatively low-cost dry processing methods, including air flotation, sizing, and some magnetic separation and froth flotation. On the other hand, high-grade clays are generally produced by employing advanced, state-of-the-art technologies, including mostly wet processes, from advanced high-gradient magnetic separation to froth flotation techniques.

Flotation concentration of low-grade kaolin clays is normally carried out by direct flotation of kaolin clay from colored impurities, even though the kaolin clay portion of the raw material makes up the majority of the mass. In this flotation application, fatty acids and

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their mixtures are generally used as collectors as well as frothers. Cytec offers a complete line of fatty-acid based collectors for this application (see Table 7-2).

On the other hand, flotation concentration of high-grade kaolin clays is conducted by employing reverse flotation of heavy and colored mineral impurities away from kaolin clay. The majority of US producers, mostly located in the middle-Georgia area, use this reverse flotation process.

In recent years, ever-increasing demand for high-grade, performance products with stricter product specifications, has resulted in several technologically advanced process and equipment developments in the kaolin clay industry. Flotation is normally used along with other innovative processes such as magnetic separation, selective flocculation etc. to produce high-grade clay products.

Reverse flotation of colored impurities from kaolin clay is a highly competitive and technologically advanced process application. Since fatty acids and their derivatives have, until recently, been the only collector type available for flotation, the industry innovators looked for other ways to improve the overall process. As a result, numerous, highly successful and competitive process applications were developed, based on improved modifiers and equipment during blunging, conditioning, and flotation stages.

However, with the recent introduction of hydroxamic acid collectors by Cytec, further significant improvements have been realized. Hydroxamic acid-based collectors not only simplify the overall process by eliminating activators and cumbersome collector schemes, but also make it possible to process some types of kaolin clays that are not treatable with standard fatty acids. Some of these developments have been reported by Yoon et al.

A typical process for hydroxamic acid flotation includes high solids (50% or higher) and high-intensity blunging to disperse clay minerals from impurities, followed by conditioning with hydroxamic acid collectors and flotation, preferably with the use of columns. Dosages for hydroxamate collectors vary between 0.5 to 1.0 Kg/ton of flotation feed, depending on the amount of impurity minerals and kaolin clay type. AERO 6493 promoter is also used in the novel selective flocculation process developed recently. This process is especially applicable for the fine kaolin clays. The hydroxamate collector, used in the blunging-conditioning step, adsorbs selectively on the colored impurities which then form large aggregates. These aggregates are selectively flocculated with high molecular weight flocculants, specifically Hydroxamated PAMs (These are novel flocculants developed by Cytec; See Section 9).

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Cytec's current hydroxamate product line includes AERO 6493 and 6494 promoters. These collectors are designed to possess different frothing properties to respond effectively to various kaolin clays and flotation concentration methods.

Kyanite

Kyanite is usually found with sulfide minerals such as sphalerite and pyrite. In the majority of plants, the ore is first de-slimed to remove as much of the clay minerals as possible. The ore is then ground to the desired flotation particle size and the sulfide minerals are removed by flotation using AERO 343 xanthate or another suitable sulfide collector. After removing the sulfide minerals, which in most cases are an undesirable commercial mineral, the pulp is placed in a conditioner and the pH reduced to 2.5 to 2.8 with sulfuric acid. AERO 855 promoter, a formulated petroleum sulfonate-based collector, is added at a dosage of 250 to 750 grams per ton. The pulp is conditioned at sixty eight to seventy percent solids for five minutes. The conditioned pulp is then diluted with water to 25-30% solids and the kyanite is floated.

The AERO 855 promoter is much more selective than previously-used collectors for kyanite flotation. In one plant, a flotation feed containing 45-48% kyanite is producing a kyanite concentrate grade of 92-95% with a recovery of over 92%.

Iron minerals such as hematite and magnetite will be floated with the kyanite. In most cases, these are removed after flotation by magnetic separation.

Phosphate

Collophane, the principal phosphate mineral of the Southeastern United States sedimentary deposits, floats readily with crude fatty acids and soaps, fuel oil and soda ash, caustic soda or ammonia. The process generally used in U.S. Florida plants is known as the "double float" method. After desliming, the pulp is conditioned at high solids using the above reagents, followed by pulp dilution and flotation of the phosphate from the silica in the "rougher" float after conditioning at a pH of 9.0-9.5 at 70-72% solids. The phosphate concentrate is then conditioned with sulfuric acid and washed with water to remove reagents. The washed concentrate is then subjected to the second "reverse" float using a fatty amine or ether amine collector to remove silica into the froth product at natural pH, typically 6.5-7.0. North African and Middle Eastern phosphate operations have increasingly moved to flotation, but unlike the U.S. Florida plants that utilize a "double float", they typically employ

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either a fatty acid or an amine float. Cytec's AERO 727, 727J and 728 promoters have been successfully used where only the fatty acid float approach is practiced. Cytec's AERO 8651 promoter, a fatty amine, is utilized in operations running an amine float, and Cytec has additional fatty and ether amines available.

To improve selectivity in the "reverse" float in the Florida "double float" process or for operations utilizing only an amine float, Cytec has developed and successfully introduced ACCO-PHOS 950 depressant, which minimizes phosphate losses into the silica froth product when using amine collectors. Typical dosage range for ACCO-PHOS 950 depressant is 20-100 g/t in the conditioning stage prior to amine addition and conditioning.

AERO 845 promoter has commercial application in the treatment of sedimentary pebble phosphates, added in conjunction with fatty acid at about 5-10% of the total collector dosage. One plant in Africa processing this type of phosphate ore uses 150 g/t AERO 845 promoter with about 1600 g/t fatty acid as collectors. The use of AERO 845 promoter increases phosphate recovery while at the same time reducing consumption of fatty acid, diesel oil, and caustic soda. Essential for effective use of AERO 845 promoter at this plant is a brief conditioning time with the AERO 845 promoter, one minute or less, while conditioning time for all other reagents and fatty acid remains at three minutes.

Apatite occurring in "hard rock" deposits, as distinct from sedimentary pebble deposits, is being upgraded by notation with fatty acids, petroleum sulfonates and AERO 845 promoter, in alkaline circuits. Gangue minerals tend to be more of a problem in the flotation of hard rock apatites, where calcareous and micaceous gangue predominates. The proper selection of suitable depressants and regulators, therefore, assumes more importance with hard rock apatites than for the treatment of pebble phosphates.

AERO 845 promoter has shown improved selectivity and recovery of fine phosphate, compared to other anionic collectors, for the treatment of hard rock apatites. One plant uses AERO 845 promoter as the rougher circuit collector (90-100 g/t) with glycol frother (2-4 g/t), followed by a scavenger circuit using fatty acid (70-80 g/t) as collector. The high-grade rougher concentrate is cleaned in a circuit separate from that for the scavenger concentrate. The AERO 845 promoter used in the rougher circuit recovers about 75% of the total recovered phosphate, with excellent rejection of gangue minerals.

AERO 847 promoter, mixed 5% to 10% by weight with fatty acids, has demonstrated improved selectivity in plants treating hard rock apatites.

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Potash

Flotation concentration of potash accounts for about three-quarters of the potash production worldwide. Leaching and re-crystallization or fractional crystallization processes are also used alone or in conjunction with flotation to produce the final product quality.

The most common potash minerals are sylvite (KCl), carnallite ($KMgCl_3.6H_2O$), and kainite ($KCl.MgSO_4.3H_2O$). In most cases, the potassium minerals are floated away from halite (NaCl) and other gangue minerals.

Even though the straight flotation of sylvite is the most common process employed worldwide (mainly in Saskatchewan potash fields in Canada and in U.S., Europe, Russia and South America), the reverse flotation of halite from sylvite is also employed, mainly in the Dead Sea region of Jordan and Israel.

Flotation of potash differs considerably from the standard flotation applications since the minerals to be separated are water-soluble salts and flotation is carried out in saturated brine solution. Temperature is one of the main factors that effect the flotation process. The solubility of NaCl in water, which is much higher than KCl, decreases with decreasing temperature, whereas the solubility of KCl is not affected by temperature. Other important factors are:

a) Presence of carnallite in the ore. It has been shown that Mg_2+ ions associated with carnallite depress the flotation of KCl with amines, especially in the presence of slimes.

b) Presence of clay in the ore. Clays not only compete with sylvite in adsorption of amine, reducing amine adsorption on sylvite, but also crowd the concentrate reducing grade and causing problems in the down-stream operations. Therefore, desliming is generally employed ahead of flotation.

Primary long-chain amines are the usual collectors for the flotation of sylvite. Cytec offers two primary amines with different properties. AERO 3000C promoter is a fully neutralized, formulated long-chain amine collector which is liquid at 45 °F. It is specially formulated to outperform paste amines on a weight-equivalent basis. In addition to its improved selectivity over other paste amines, it is less affected by slimes compared to other amines. AERO 3000C promoter can be prepared as a 5-10% solution. Depending on the type and concentration of KCl ore, its dosage varies from 200 to 500 grams per ton. AERO 3000C promoter can also be fed neat. In addition to AERO 3000C promoter, Cytec offers AERO 3100C promoter as a paste primary amine, which can also be used as an effective sylvite collector.

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For the flotation of coarse sylvite, hydrocarbon oils (as extender oils) are used in conjunction with amines to improve the flotation recovery.

The reverse flotation of halite from sylvite is practiced mainly in the Dead Sea region in both Israel and Jordan. Morpholine type collectors are found to be more effective in this process.

A number of sylvite ores with high clay content require additional steps to overcome the harmful effects of these clays on the overall selectivity. Various polymers or modified polymers are used to depress clays ahead of sylvite flotation. Even though common depressants such as CMC, guar, and starch are used in the industry, modified polymers (either anionic or non-ionic) are often more effective clay depressants. These depressants require conditioning ahead of flotation with the amine collector. Reagent 8860 and Reagent 8860GL depressants were specifically developed by Cytec to depress talc-like minerals in sulfide flotation and may be applicable to depressing clays in sylvite flotation.

Cytec developed a commercially successful selective flocculation/ flotation process to remove clays ahead of potash flotation. This process eliminates the need for mechanical removal of slimes, which is capital and operating-cost intensive. In this process, the ground ore is first gently conditioned with 25 to l00g/ton of a flocculant such as SUPERFLOC N-100 and then with 20 to 100 g/ton of AERO 870 promoter to float the flocculated clay slimes; the floated clay product is usually low enough in potash to be discarded, but can be refloated in a cleaning stage if necessary The flotation tailing is fed to the potash flotation stage and generally requires the use of less clay depressant than in the case of mechanical desliming. The process can also make feasible the treatment of high-clay potash ores which, heretofore, could not be treated economically.

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8 Flocculants and dewatering aids



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Section 8 Flocculants and dewatering aids

8.1 Synthetic polymeric flocculants

At various stages of mineral processing it is necessary to separate aqueous mineral suspensions into their component solid and liquid phases. Typical examples of this are thickening of flotation concentrates, recovery of pregnant leach liquors, and dewatering of tailings. In many cases, the mineral particles settle out of suspension very slowly, so that the liquid-solid separation is slow and incomplete. To improve the settling rate, high molecular weight organic polymers (flocculants) are used to aggregate the suspended particles and cause the efficient separation of the solids from the aqueous suspending medium.

8.2 Stabilization of suspensions

In a mineral suspension there is usually a wide difference in particle size. Some particles may be large enough to settle out quickly, while very fine particles may not settle at all. The rate of settling of any given particle is dependent upon its size, its density relative to that of the suspending medium, the viscosity of the medium, and the interactive forces between this and other suspended particles.

The major interactive forces between suspended solids are of two kinds - attractive and repulsive. The former arise from short-range Van der Waals' forces, the latter from overlap of the similarly charged electrical double layers of the particles. If repulsive forces dominate, particle aggregation cannot occur, whereas, if attractive forces take over, aggregation and settling of the much larger aggregates will take place. These attractive forces can operate only when the particles are very close together. The shortest distance of approach between particles is a direct function of the magnitude of the electrical double layer which is itself a direct function of the charge on the surface of the particles. This surface charge, therefore, has a profound effect on the stability of an aqueous suspension of solid particles.

In aqueous mineral suspensions, mineral particles almost invariably carry a surface charge, which is generally negative, except in a few instances where the pulp pH is very low. This surface charge is due to one or more of the following factors:

- Unequal distribution of constituent ions.
- lonization of surface groups.
- Specific adsorption of ions from solution.

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• Isomorphous substitutions in the mineral lattice.

Because of this surface charge, ions of opposite charge in solution will be attracted towards the surface. There will therefore be a higher concentration of counter-ions close to the surface than in the bulk of the liquid (see figure 8-1). This concentration falls off with increasing distance from the particle, so that there is a bound layer of counter-ions at the particle surface, succeeded by a more diffuse layer. Beyond the diffuse layer is the bulk solution, in which the ionic distribution is random. The bound layer moves with the particle as the latter travels through the medium, so that there is a plane of shear between the bound and the diffuse layers. The potential at the plane of shear and the bulk solution is the "zeta potential."

The zeta potential depends upon the surface charge of the particle, and, since it can be determined more easily than the actual surface charge, is often taken to be a convenient measure of charge.

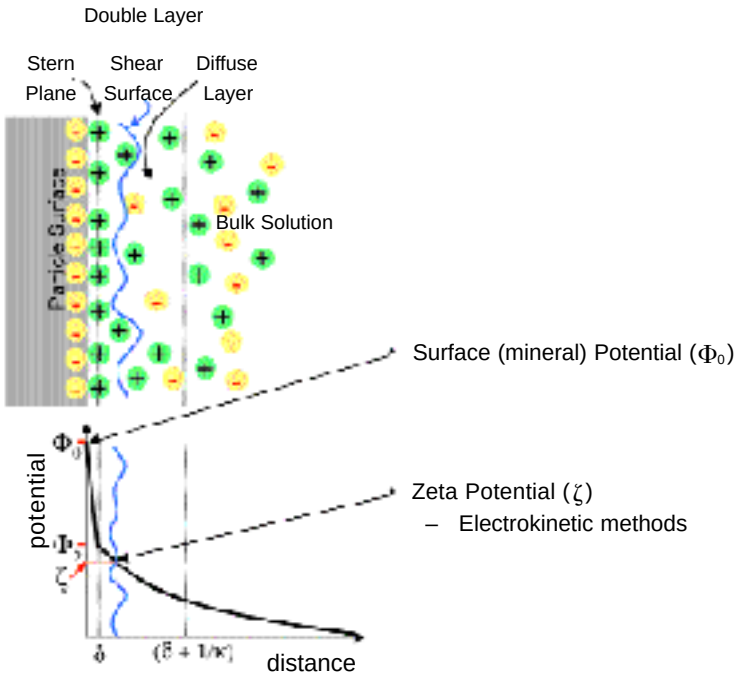


Fig. 8-1 The electrical double layer.

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Most zeta potential determinations rely on electrophoretic methods, and measure the mobility of individual charged, suspended particles under the influence of an applied potential.

8.3 Destabilization of suspensions

Destabilization of suspensions may be commonly achieved by one of three methods:

- Electrolyte addition.
- Addition of hydrolyzable metal ions.
- Polymer flocculation.

Electrolyte addition can bring about coagulation (as opposed to flocculation) by two mechanisms.

First, the addition of any electrolyte to the suspension will result in compression of the electrical double layer, and a lowering of the zeta potential. The magnitude of this effect increases with increasing charge on the counter-ion, so that for negatively-charged suspensions, trivalent cations (Fe^{3+}, Al^{3+}) are more effective than divalent cations (Ca^{2+},Mg^{2+}), which are in turn more effective than monovalent cations (Na^{+}).

Second, counter-ions may react chemically with the particle surface and be adsorbed onto it. Specific counter-ion adsorption will result in a lowering of the particle charge, and can reduce it sufficiently to enable close approach of the particles allowing coagulation of the suspension to take place.

In mining applications, coagulation by either of these methods usually results in the formation of very small, slow settling flocs. However, lime addition is often practiced, either at the flocculation stage, or earlier in the mineral treatment process, since such coagulation reduces the dosage of synthetic flocculant needed to give the required settling rate.

Hydrolyzable metal ions (such as Al^{3+}, Fe^{3+}) are usually added in the pH range and at the concentration level where the metal hydroxide is precipitated. Under the proper conditions, the bulky hydroxide precipitate "sweeps up" the suspended particles as it falls to the bottom of the vessel.

This approach usually works well only when there is a very low level of suspended solids. Because of this, and because of the restrictions of pH required to give a bulky precipitate, this mode of flocculation is rarely, if ever, practiced in mining applications.

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Charged, water-soluble organic polymers are polyelectrolytes. Therefore, if this charge is opposite in sign to that carried by the suspended particles, addition of such a polymer to the suspension will result in aggregation by specific ion adsorption, as described above. However, the flocculating action of polymer flocculants also proceeds via either "Charge Patch attractions", or "Polymer bridging".

Charge Patch attraction occurs when the particle surface is negatively charged, and the polymer is positively charged. The polymer must have a high density of charge - usually one cationic charge to every 4 or 5 carbon atoms in the polymer chain.

Initially, these polymers adsorb onto the surface of the particle by electrostatic attraction. However, if, as is often the case, the charge density on the polymer is much higher than that on the particle surface, the polymer will neutralize all the negative charge within the geometric area of the particle on which it is adsorbed, and still carry an excess of unneutralized cationic charge. The result of polymer adsorption of this type is the formation of positively charged patches, surrounded by regions of negative charge. These positive charge patches can then bring about aggregation through electrostatic attraction of negatively-charged areas on the surface of other particles (see figure 8-2).

Charge Patch Flocculation

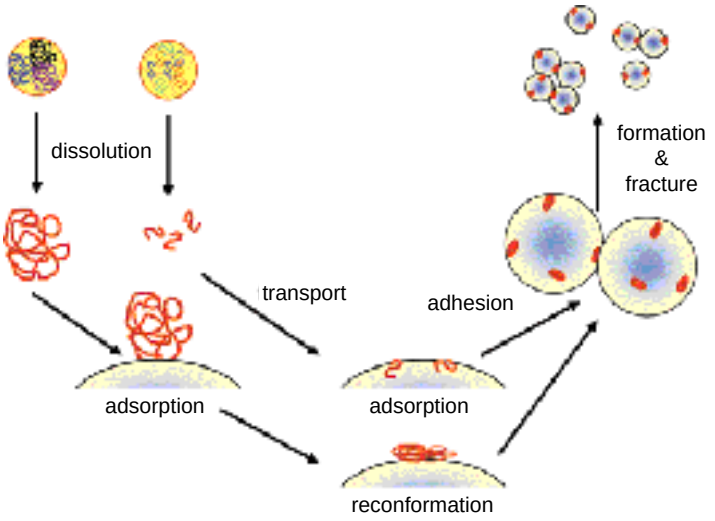


Fig. 8-2 Charge Patch Neutralization

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The most common types of polymer to operate by this mechanism are the polyamines. These are condensation polymers, and are relatively low in molecular weight, with the result that flocs formed in this way are fairly small, and slow-settling.

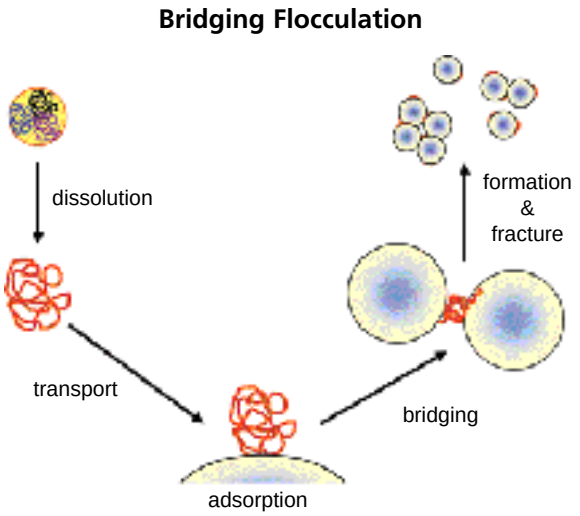


Fig. 8-3 Polymer bridging.

Polymer bridging is shown schematically in figure 8-3. The process probably takes place in two stages, the first of which involves adsorption of polymer molecules onto individual, suspended particles. The size of the polymer molecule is such that considerable portions of the polymer chain are unattached to the particle. This results in either the ends of the chain being left dangling, or loops of the unadsorbed segments sticking out from the particle surface into the medium. In the second stage of the process, the free ends, or loops of the polymer chains contact and adsorb onto other suspended particles, forming particle aggregates, or flocs. If the polymer chains are long enough, this bridging can readily take place without charge neutralization between particles occurring.

Clearly, bridging can only take place with polymers of very high molecular weight, which need not carry a charge opposite in sign to that of the suspended particles. The majority of synthetic polymers of this type are based on acrylamide and its derivatives as the monomers. This includes acrylamide-quaternized aminoalkyl acrylate co-polymers (cationic); polyacrylamide (non-ionic) and acrylamide-

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acrylic acid co-polymers (anionic). The mode of initial adsorption of such polymers onto a suspended particle varies according to the respective charges of both polymer and particle. It may be purely electrostatic if these charges are opposite in sign. If not, then other physico-chemical reactions may take place. In the case of nonionic polyacrylamides, the most likely mechanism of adsorption is through hydrogen bonding between the oxygen atoms associated with hydrated metal ions at the particle surface, and amido-hydrogen atoms on the polymer. In the case of anionic flocculants and negatively-charged suspensions, adsorption may also take place via hydrogen-bonding. In pulps to which lime has been added, polymer adsorption often also occurs through cation bridging. In this mode, the divalent calcium ions can form an electrostatic "bridge" between the negatively-charged particle-surface, and the negatively-charged carboxyl groups of one acrylamide-acrylic acid copolymer.

Both non-ionic and anionic polyacrylamides are widely used in mining applications. They can be manufactured with very high molecular weights (5-20+ x 10^6), and thus are capable of forming large, rapid-settling, good-compacting flocs. Cationic polyacrylamides are rarely used in the mining area. They are usually much less cost-effective than their non-ionic and anionic counterparts, because of higher cost and lower molecular weight (2-8 x 10^6).

8.4 Flocculant testing

It is impossible to predict from theoretical knowledge which synthetic flocculant is most suited to a particular suspension. Flocculation can occur by all of the above mechanisms, and suspensions produced from mineral ores are inherently variable in character. Flocculant selection is generally done on an empirical basis, with some pre-selection based on experience. All types of Cytec's flocculants should be evaluated for their relative performance in the suspension under investigation.

Performance criteria include those of cost, required settling rate, supernatant clarity, and compaction requirements. These criteria should be clearly established before any testwork is carried out, since they are very dependent on equipment and throughput requirements of individual plants.

Initial testing should be carried out in the laboratory. The main aim of such testing is to screen the range of Cytec's SUPERFLOC flocculants in order to determine which individual product is most cost-effective for that particular substrate. However, the tests can also yield additional information as to the approximate dosage rates required to achieve the desired plant performance, approximate

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supernatant clarities and mud solids contents which can be attained, and will enable estimation of required thickener areas to be made.

It is important for good laboratory results that the flocculant solutions be made fresh each day. Solutions of dry polymers are generally made at 0.1%. A mixer must be used that will create a vortex that goes to the bottom of the beaker. With vigorous mixing, the powder is sprinkled into the shoulder of the vortex at a rate which produces uniform dispersal with no lumps. Stirring is continued at a slower rate until all of the flocculant is dissolved, usually 1-2 hr. Solutions of emulsion polymers are generally made up at 0.5-1%. Either a tilted Braun hand blender or Waring blender (with transformer) should be used for breaking. With the mixer running, the emulsion is quickly squirted with a syringe into the vortex. After initial mixing of not more than 6-10 seconds with the Braun or Waring blender, transfer the polymer solution to a jar tester equipped with three inch paddles and continue stirring for 30-60 minutes at 100-200 rpm. Further dilution of these polymer solutions to about 0.05% or lower for the actual testing is best.

For settling applications, the standard cylinder test is generally used. The substrate slurry is placed in a graduated cylinder (500-1000 ml) and the desired polymer dose is added as a dilute solution. For good mixing, use a plunger, applying 6-10 moderate up-and-down strokes. Mix for approximately 15-20 seconds to insure thorough dispersion between the bottom and the top of the suspension. For dual polymer applications, the first polymer is added and mixed vigorously into the substrate, followed by the addition of the second polymer with more gentle mixing with the plunger. In the case of slimes which form fragile flocs, the procedure should be modified to give more gentle mixing. It is most important that mixing techniques be uniform throughout the entire test procedure. Variation in mixing methods can be a major source of uncertain results and poor reproducibility of settling tests. After the polymer is mixed into the substrate, the plunger is removed and the time measured for the interface line to fall a specified distance. After a suitable time for settling, a sample of the supernatant liquid can be removed with a pipette or syringe in order to measure clarity. Variables that can affect polymer dosage and settling rates include mineralogical composition, particle size of the mineral constituents, pH, temperature, solids content, and water chemistry.

Subsequent testing with the selected flocculant should be carried out in the plant. During this, it must be borne in mind that synthetic flocculants can often be used most efficiently as very dilute (0.01-0.05%) solutions, and, in many cases, perform best when

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added simultaneously at various points along the feed launder or pipe. The flocs formed by anionic flocculants and negatively-charged suspended particles are fragile, and will rupture if mixing is too vigorous. Since adequate mixing is vital to effective use of the flocculant, varying the point(s) of addition to obtain optimum results forms an essential part of plant testing.

8.5 Cytec's flocculants

Cytec manufactures a complete line of flocculants in plants located around the world. (See Tables 8-1 to 8-3 for a representative listing of Cytec's flocculants.)

Cytec's polyacrylamides and acrylamide-acrylic acid co-polymers range from non-ionic up to 100% anionic charge. These are very high in molecular weight (5-20+ x 10^6), and are manufactured and sold as both dry powders, and in emulsion form.

Cytec's cationic polymers cover a wide range of chemical types, molecular weights, and charge densities. The lower molecular weight (10 x 10^3 - 0.5 x 10^6) polymers, typified by the polyamines, are very highly charged. These are sold as concentrated (up to 50% active) solutions. Cationic acrylamide co-polymers are available at several levels of cationic charge, and at much higher molecular weights (2-8 x 10^6). They are produced as dry powders, or as emulsions.

The listing of flocculants in Tables 8-1 to 8-3 is not intended to be exhaustive, but is given to illustrate the general range of flocculants available. Through research and development and the inherent flexibility of its several manufacturing processes, Cytec has the capability to tailor-make flocculants for optimum performance in many types of applications. Typical of these developments is the perfection of a line of anionic polymer emulsions with very high molecular weight (20+ x 10^6, the 1260 series of SUPERFLOC flocculants) which can provide improved performance in many applications. Please contact your Cytec representative for further information and to find out what Cytec can do for your application.

8.5.1 Anionic flocculants

Anionic flocculants have very wide application in the mining industry. They are principally used for thickening ore pulps and concentrates, such as coal tailings, copper, lead, and zinc concentrates and tailings, diamond and phosphate slimes, and bauxite red muds. Normal dosage rates for these applications are in the range 2.5-50 g/t.

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Anionic flocculants are also used as filtration aids for vacuum or pressure filtration of coal refuse and mineral concentrates. Dosage rates are usually between 50-500 g/t.

Anionic flocculants are used as dewatering aids in the centrifugation of mineral slurries and tailings, usually at dosage rates of 5-250 g/t.

8.5.2 Nonionic flocculants

Nonionic flocculants are principally used in the thickening of ore pulps and concentrates, especially iron ore slimes, and gold flotation tailings. They are particularly effective in acidic media such as pregnant uranium leach liquors. Typical dosage rates are 1-50 g/t.

Nonionic flocculants are also used as dewatering aids in vacuum and pressure filtration, and centrifugation, usually at dosage rates of 5-250 g/t.

8.5.3 Cationic flocculants

Cationic flocculants are chiefly used for thickening of coal refuse, iron ore slimes, and mineral concentrates. Dosage rates in these applications usually range from 25-250 g/t. Cationic flocculants are efficient clarification agents for surface mine run-off water. In this case, typical doses are 5-50 g/t.

Local requirements dictate that not all of the products referred to above are available at a given location. Contact the Cytec subsidiary nearest you for information as to the flocculants available in your area. Cytec has a highly-trained technical field staff, covering every country in the world. They are fully qualified to assist in the evaluation and introduction of Cytec's flocculants for any mining application.

8.5.4 Other flocculants

In addition to the products listed in the tables below, specific flocculants have been developed for use in red mud and alumina substrates in the Bayer process. These products are described in more detail in Section 9.

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Table 8-1 Cytec's anionic flocculants

Emulsions	Type	Charge	Molecular Weight
SUPERFLOC A-1849RS	Anionic Polyacrylamide	Low	High
SUPERFLOC AF 122	Anionic Polyacrylamide	Low	Very High
SUPERFLOC AF 124	Anionic Polyacrylamide	Moderate	Very High
SUPERFLOC A-1820	Anionic Polyacrylamide	Moderate	High
SUPERFLOC A-1883RS	Anionic Polyacrylamide	Moderate	High
SUPERFLOC 1204	Anionic Polyacrylamide	Moderate	Moderate
SUPERFLOC A-1885RS	Anionic Polyacrylamide	Moderate	High
SUPERFLOC AF 126	Anionic Polyacrylamide	Moderate	Very High
SUPERFLOC AF 128	Anionic Polyacrylamide	Moderate	Very High
SUPERFLOC 1240	Anionic Polyacrylamide	High	High
SUPERFLOC 1238	Anionic Polyacrylamide	High	High
SUPERFLOC 1236	Anionic Polyacrylamide	High	High
SUPERFLOC 1232	Anionic Polyacrylamide	High	High
SUPERFLOC 1230	Anionic Polyacrylamide	High	High
SUPERFLOC 1229	Anionic Polyacrylamide	High	High
SUPERFLOC 1227	Polyacrylate	High	High
ACCO-PHOS 1250	AMPS/Acrylamide Copolymer	Low	Moderate
Dry			
SUPERFLOC A-100	Anionic Polyacrylamide	Low	High
SUPERFLOC A-110	Anionic Polyacrylamide	Low	High
SUPERFLOC A-120	Anionic Polyacrylamide	Moderate	High
SUPERFLOC A-130	Anionic Polyacrylamide	Moderate	High
SUPERFLOC A-130HMW	Anionic Polyacrylamide	Moderate	High
SUPERFLOC A-150	Anionic Polyacrylamide	High	High
SUPERFLOC A-185HMW	Anionic Polyacrylamide	High	High
SUPERFLOC A-190K	Polyacrylate	High	Moderate
Solutions			
SUPERFLOC 550	Anionic Polyacrylamide	High	Low

Table 8-2 Cytec's nonionic flocculants

Emulsions		Molecular Weight
SUPERFLOC 1128		High
Dry		
SUPERFLOC N-100		High
SUPERFLOC N-300		High
SUPERFLOC N-300LMW		Moderate

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Table 8-3 Cytec's cationic flocculants

Emulsions	Type	Charge	Molecular Weight
SUPERFLOC C-1591	Cationic Polyacrylamide	Low	Moderate
SUPERFLOC MX10	Cationic Polyacrylamide	Low	High
SUPERFLOC C-1592	Cationic Polyacrylamide	Low	Moderate
SUPERFLOC MX20	Cationic Polyacrylamide	Low	High
SUPERFLOC C-1594	Cationic Polyacrylamide	Moderate	Moderate
SUPERFLOC MX40	Cationic Polyacrylamide	Moderate	High
SUPERFLOC C-1596	Cationic Polyacrylamide	Moderate	Moderate
SUPERFLOC MX60	Cationic Polyacrylamide	Moderate	High
SUPERFLOC 1598	Cationic Polyacrylamide	High	Moderate
SUPERFLOC MX80	Cationic Polyacrylamide	High	High
Dry			
SUPERFLOC C-491	Cationic Polyacrylamide	Low	Moderate
SUPERFLOC C-492	Cationic Polyacrylamide	Low	Moderate
SUPERFLOC C-492HMW	Cationic Polyacrylamide	Low	High
SUPERFLOC C-494	Cationic Polyacrylamide	Moderate	Moderate
SUPERFLOC C-494HMW	Cationic Polyacrylamide	Moderate	High
SUPERFLOC C-496	Cationic Polyacrylamide	Moderate	Moderate
SUPERFLOC C-496HMW	Cationic Polyacrylamide	Moderate	High
SUPERFLOC C-498	Cationic Polyacrylamide	High	Moderate
SUPERFLOC C-498HMW	Cationic Polyacrylamide	High	High
Solutions			
SUPERFLOC C-577	Polyquaternary Amine	High	Low
SUPERFLOC C-581	Polyquaternary Amine	High	Low
SUPERFLOC C-587	Polyquaternary Amine	High	Low
SUPERFLOC C-591	Polyquaternary Amine	High	Low
SUPERFLOC C-595	Polyquaternary Amine	High	Low

8.6 AERODRI dewatering aids

Dewatering is the removal of water from the void spaces in a filter cake. The filter cake is a porous system in which the channel structure can be approximated as an assembly of capillaries. The residual saturation in the cake can then be related to the capillary rise phenomenon. The capillary rise equation is

$$h = \frac{2\,\gamma \cos\theta}{g\,\rho\,R}$$

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where h is the capillary rise, γ is the liquid/air surface tension, θ is the liquid/solid contact angle, R is the capillary radius, g is the acceleration due to gravity (vacuum or pressure in the case of filtration), and ρ is the liquid density. Surfactants are used to improve the removal of water from a filter cake by both lowering the surface tension and increasing the contact angle (increasing particle surface hydrophobicity) by adsorbing onto the particle surfaces. Although lowering surface tension can play a role in moisture reduction (typically lowering surface tension from 72 dynes/cm to about 30 dynes/cm, which effectively reduces the capillary rise by a factor of about 2), the increase in contact angle is the more important factor. The use of the proper surfactant can increase the contact angle from near zero for thoroughly wetted particles ($\cos\theta$ of about 1) to 70-80° ($\cos\theta$ of about 0.2-0.3) for a reduction in capillary rise by a factor of 3-5.

AERODRI dewatering aids are surface-active agents that have been specially formulated to maximize the contact angle as well as reduce the surface tension of the water. They have found wide use in the mining industry for reducing filter cake moisture, increasing filtration rates, improving filter cake handling qualities, and reducing filter cloth blinding. They have application for filtration of sulfide and non-sulfide mineral concentrates, clean coal, and alumina hydrate precipitated from Bayer process liquors. Dosages required to obtain benefits vary greatly, and may range from as little as 25 g to as much as 500 g AERODRI dewatering aid per ton of solids. It has usually been observed that upon reaching an effective dosage, the filter cake characteristics change abruptly.

AERODRI dewatering aids may be applied full strength or diluted to the filter feed, or as a dilute solution in spray water in operations where greater filter cake washing efficiency is needed.

AERODRI 100 dewatering aid

At room temperature AERODRI 100 dewatering aid forms clear aqueous solutions in concentrations up to about 1.7%, and viscous dispersions at higher concentrations up to about 10%. It is readily soluble in polar and non-polar organic solvents at room temperature. AERODRI 100 dewatering aid is effective in mild acid solutions and in the presence of small concentrations of electrolytes.

AERODRI 100 dewatering aid is biodegradable and exhibits low alkali tolerance. Thus, residual quantities, occasionally present in filtrates, may be eliminated by adjusting filtrate pH with lime addition if such is not deleterious to subsequent plant operation stages.

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Circuit pH	Decomposition Time
8.3	6 days
9.9	4 days
11.1	4 hours
11.8	2 hours
12.5	15 minutes

AERODRI 100 dewatering aid, when fed full strength, should be preconditioned with the pulp for periods of up to 10 minutes to optimize filter cake moisture reduction.

AERODRI 104 dewatering aid

AERODRI 104 has a lower viscosity, and is more readily dispersible, than AERODRI 100 dewatering aid. It is preferred where preconditioning with the pulp is limited and dilute feed solutions are not practical. It may be applied full strength, as an aqueous solution up to about 3% concentration, or as an aqueous dispersion at higher concentrations up to about 17%. AERODRI 104 dewatering aid is biodegradable and exhibits the same alkali tolerance as for AERODRI 100 dewatering aid.

AERODRI 200R dewatering aid

AERODRI 200R dewatering aid was developed for applications where recirculation of residual product in the water supply system is undesirable. AERODRI 200R dewatering aid is at least 95% retained on the mineral solids, thereby minimizing any build-up in a closed-circuit water system. It may be applied full strength in a well-agitated system for adequate preconditioning with the pulp, or as an aqueous dispersion of up to about 10% concentration to the filter boot or further upstream from the filter.

Physical characteristics of AERODRI dewatering aids

	100	104	200R
Appearance	Clear to Slightly Hazy colorless to light yellow liquid		
Solubility in Water, 20°C	1.7%	3.0%	Dispersible
Specific Gravity, 20°C	1.08	1.03	0.96
Viscosity @20°C (cps)	250	26	30
Flash Point °C (closed cup)	32	46	45
Freezing Point °C	4	-4	4

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AERODRI 1000 dewatering aid

AERODRI 1000 dewatering aid was developed for use in the centrifugal dewatering of coarse clean coal (>0.5 mm), without attendant foaming problems which could aversely affect subsequent processing stages, such as the recovery of heavy media. Its use can result in increased calorific value of the final coal product. This allows increased recovery of coal without adversely affecting overall calorific value of the final product. This also enables the processing of raw coal feed which previously had too high a moisture content in the final product. Use of AERODRI 1000 at one coal processing operation enabled the elimination of thermal drying, previously required, with substantial cost savings.

AERODRI 1000 dewatering aid should be applied by spray nozzles to the oversize coal product discharging from sieve bends or vibrating screens, which feed the centrifuge dewatering unit. It should be diluted at least 100:1 before spray application. This can be accomplished by feeding AERODRI 1000 dewatering aid through an eductor into the water line feeding the spray nozzles, with sufficient water flow to achieve the necessary dilution ratio.

Physical characteristics

AERODRI 1000 dewatering aid

Appearance	Clear, pale yellow liquid
Solubility in Water	Dispersible with vigorous agitation, 100-1 dilution preferred.
Specific Gravity	0.93 @ 20°C
Flash Point (closed cup)	52°C

Other dewatering aids

In addition to the products listed above, specific dewatering aids have been developed for use in the dewatering of alumina trihydate in the Bayer process. These products are described in more detail in Section 9.

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9 Bayer process reagents



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Section 9 Bayer process reagents

The Bayer Process, developed and patented by Karl Joseph Bayer in 1888, is used for the production of alumina from bauxite. The process is based on the fact that hydrated aluminium oxides are soluble in caustic at elevated temperatures and pressures. The solubility of aluminium oxide varies widely according to the form in which it is present. Alumina occurs in bauxite in the trihydrate form (gibbsite) and as the monohydrate (boehmite and diaspore). The trihydrate is more soluble than the monohydrate.

The process may briefly be described, as follows-

Bauxite is digested in caustic soda solution at elevated temperatures and usually under pressure. After digestion, the solution containing the dissolved aluminium oxide in the form of sodium aluminate has suspended in it the residue from the bauxite. This insoluble residue, called 'red mud,' consists predominantly of iron oxide, titania and silica. The red mud is separated from the aluminium oxide rich solution with the aid of synthetic flocculants in vessels referred to as Thickeners, Decanters or Settlers. The terminology used is dependent on the operating company. The clarified liquor is further polished (mud particles removed) via filtration. Alumina trihydrate is then precipitated from the liquor, filtered and washed before it is calcined at extremely high temperatures. The product derived is anhydrous Alumina, Al_2O_3.

The underflow (mud) from the Thickeners, in addition to the mud removed at filtration, still has entrained in it a significant amount of liquor containing caustic and alumina. Most of this is recovered by washing the mud in a Counter Current Decantation Circuit (CCD circuit). Synthetic flocculants are also used here to aid in the mud/liquor separation process.

The entire process may be represented by the equations:

Extraction

$$Al_2O_3.3H_2O + 2NaOH = 2NaA1O_2 + 4H_2O \quad (1)$$

Precipitation

$$2NaAlO_2 + 4H_2O = Al_2O_3.3H_2O + 2NaOH \quad (2)$$

Calcination

$$Al_2O_3.3H_2O = Al_2O_3 + 3H_2O \quad (3)$$

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A simplified flowsheet of the Bayer Process is shown in Figure 9-1 below. The dissolution and mud separation stages are generally referred to as the "Red Side" of the circuit while the precipitation, alumina filtration, and calcination are referred to as the "White Side."

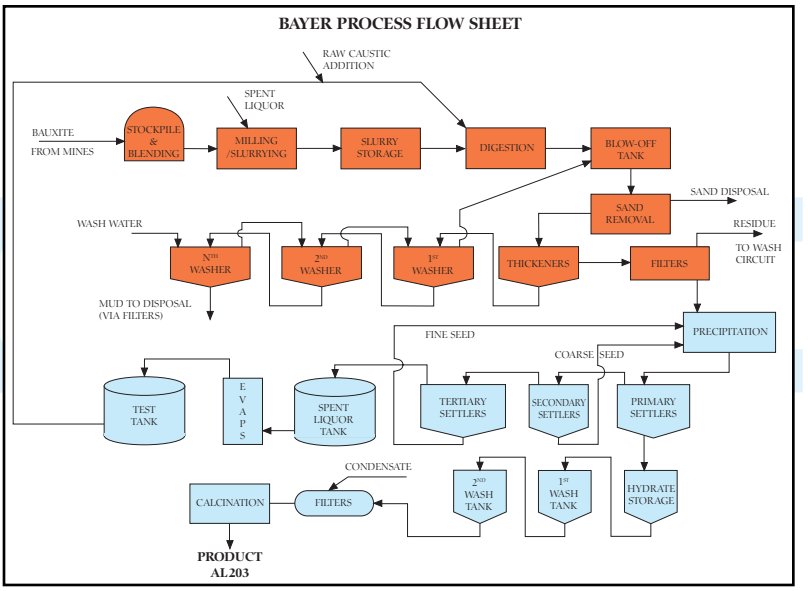


Figure 9-1

A wide variety of chemical reagents is used in the various stages of the process and these are described below. Because of the unique conditions (liquor temperatures, high electrolyte levels etc.) in the process streams, specialized techniques are generally required for testing and using the various reagents in both the laboratory and plant. Also, optimum reagent dosages vary widely owing to the widely-varying nature of different bauxites and the red muds they produce. We recommend that you consult your Cytec representative for detailed information before testing our products.

9.1 Red mud flocculants

Up to the mid-1970s, starch was the most common flocculant used in the separation of red mud from the pregnant liquor. The introduction of high molecular weight, synthetic polyacrylate flocculants at that time provided several advantages compared to starch.

- Higher thickener and washer underflow densities.
- Higher vessel throughputs.

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- Higher washing efficiency resulting in reduced alumina and soda losses.
- Improved pumpability of the underflow muds.
- Elimination of rodent problems and bacterial growth.
- Much lower dosages, thereby reducing handling and storage costs.

Cytec is a major supplier of these flocculants in both dry-powder and emulsion forms. These flocculants are available in a range of anionic charges and the optimum flocculant for any particular stage of the red mud circuit is dependent on the soda content of the liquor. In the thickener stage, where the soda level is very high, the more highly anionic flocculants are the most effective. As the soda level decreases down the washer train, flocculants of lower anionic charge can be used. Cytec pioneered and patented the use of medium anionic copolymer flocculants in the washer stages. For logistical reasons, the number of different flocculants used in the red mud circuit is generally limited to two or three products.

9.1.1 Cytec's standard dry red mud polyacrylate flocculants

SUPERFLOC A-190 A-185 A-170 A-150 flocculants

----------> Decreasing anionicity

9.1.2 Cytec's emulsion red mud polyacrylate flocculants

SUPERFLOC 1227 1229 1230 1232 1236 1238 1240 flocculants

----------> Decreasing anionicity

9.1.3 Cytec's hydroxamated polyacrylamide red mud flocculants

In the late 1980s, Cytec introduced a range of proprietary emulsion products incorporating new chemistry based on hydroxamated polyacrylamide (HXPAM). These unique flocculants have since replaced polyacrylates in the thickener (and, in some cases, first washer) stages in many alumina plants around the world. Copolymer flocculants continue to be used in the washer train where overflow clarity is not a major requirement.

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The advantages of Cytec's HXPAM flocculants include:

- Greatly improved thickener overflow clarities resulting in higher liquor filtration rates, easier cake release, and reduced costs. Even in cases where suspended solids content is not significantly reduced, the liquor is still easier to filter since the fine mud particles therein are present as small flocs (pin flocs) rather than as dispersed individual particles.
- Faster mud settling rates without sacrificing overflow clarities, thereby increasing plant throughputs and/or reducing the number of thickeners on-line.
- Some muds which can not be adequately settled using polyacrylate flocculants can be handled using HXPAM flocculants.
- Higher thickener underflow densities, thereby reducing soda and aluminate losses.
- Improved rheological properties of underflow muds, thereby reducing the torque on thickener rakes, improving mud pumpability, and permitting higher underflow densities.
- Reduction in the amount of lime needed in digestion. This is due to the high affinity of the hydroxamate group for the Fe ions which are present on the red mud particles, rather than relying on Ca ion activation which is needed for flocculation with polyacrylate flocculants. The reduced lime consumption not only reduces costs but can lead to higher quality alumina with reduced calcium content.
- One plant has found that the use of HXPAM in the red mud circuit enabled the elimination of the need for crystal growth modifiers in the alumina precipitation stage.
- It has generally been found that the use of HXPAM reduces the amount of scaling in thickeners and related equipment. This extends the thickener on-line time and reduces descaling costs.

Cytec's standard hydroxamated red mud flocculants are:

SUPERFLOC HX-200 HX-300 HX-400 flocculants

----------> increasing degree of hydroxamation.

The optimum flocculant for any particular mud can be determined only by experimentation.

Higher solids versions of HX-200, HX-300, and HX-400 are also available as SUPERFLOC HX-2000, HX-3000, and HX-4000 flocculants respectively. These products provide lower shipping and handling

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costs. In a very few cases, plants have experienced mud-handling problems when using HXPAMs alone. These problems have been solved by the introduction of polymers containing both hydroxamate and carboxylate groups. This product group consists of SUPERFLOC HX-925, HX-927, HX-929, HX-945, HX-947, and HX-949 flocculants.

9.2 Humate removal reagents

Most bauxites contain significant quantities of organic matter. During the digestion stage, this breaks down into various species, one of which is humates. The humates are responsible for the dark color of the liquor and also for reducing the brightness of the final hydrate product. This latter effect is a problem when the hydrate is to be sold to the chemical industry.

In turn, these humates in the liquor are believed to break down into smaller organic molecules such as acetates, formates, and oxalates. These small organic molecules (especially oxalates) can have detrimental effects on the various stages of the Bayer process such as:

- "Poisoning" of the hydrate seed surface, thereby preventing agglomeration. This leads to a very fine hydrate particle size which makes the hydrate difficult to settle. The unsettled hydrate ends up in the spent liquor and is recirculated to the digesters via the evaporators.
- The recirculated fine hydrate causes scaling of the evaporator tubes, reducing heat transfer and throughput. This, in turn, results in lower evaporation rates and higher soda losses.
- The above effects lead to reduced alumina production since, to maintain the optimum blow-off ratio, less bauxite can be processed.

Removal of the humates at an early stage can lead to reduced concentrations of organic species in the liquor, thereby eliminating or reducing these problems. Cytec's humate removal reagents are low-to-medium molecular weight, liquid cationic polymers. These polymers form complexes with both the soluble and colloidal humates to form relatively insoluble precipitates. When the humate molecular weight is high, the complexes formed are very insoluble. On the other hand, the lower molecular weight organic species may also form complexes with the polymer but may not precipitate. Consequently, not all the color associated with humates may be removed but, generally, sufficient color is removed to solve the problems listed above.

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9.2.1 Cytec humate removal reagents

The current Cytec product in commercial use is CYQUEST 365 humate removal reagent.

CYQUEST 365 reagent can be used as supplied or diluted to any convenient strength with spent liquor. Dilution may improve the efficiency of humate removal by ensuring more complete dispersion in the slurry or liquor. The product is best added as soon after digestion of the bauxite as possible, before the humates have had much time to decompose to lower molecular weight species. In plant practice, this usually means addition to the digester blow-off slurry (feed to the thickener/settler). If more convenient, it can instead be added to the thickener overflow, but this may lead to liquor filtration problems caused by the precipitated complexes. In laboratory testing this is not a problem and addition to the overflow liquor is usually the most convenient.

In both laboratory and plant practice, the % reduction of humate content of the liquor is usually determined by color reduction, as determined by use of a spectrophotometer to measure absorbance, usually at either 575 or 691 nanometers. Typical plant dosages of CYQUEST 365 reagent range from 10 to 100 ppm; since humates in plant liquors have accumulated over a long period of time, it may take a period of weeks or months to reduce humate content to a satisfactory level unless very high dosages are used initially.

9.3 Iron removal reagents

Bayer liquors contain significant amounts of iron in solution. This results from the iron minerals in bauxite. This iron co-precipitates with the alumina trihydrate and ends up contaminating the product alumina.

To overcome this problem, Cytec developed CYQUEST 700 (powder) and CYQUEST 637 (liquid) iron removal reagents. CYQUEST 700 reagent is best added to the overflow, whereas CYQUEST 637 reagent is best added to thickener feeds. Both products work to insolubilize the iron so that it is removed with the red mud or filter cake. Typical dosages range from 20 to 50 ppm. Titanium in liquor is also reduced by the use of CYQUEST 700 reagent.

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9.4 Dewatering/filtration reagents

The precipitated hydrate is filtered before being calcined. Dewatering aids are used in the filtration stage to reduce both the moisture and soda contents of the calciner feed. The benefits of this are:

- To maintain stack gas temperatures and reduce corrosion of the calciner flue stack.
- To reduce the quantity of wash water used in the filtration stage. This allows more wash water to be used in the mud washing circuit, thereby reducing soda and alumina losses.
- To reduce the soda content of the final alumina product.

9.4.1 Cytec's dewatering aids

The Cytec products available are:

AERODRI 100 dewatering aid
AERODRI 104 dewatering aid
AERODRI 200R dewatering aid
AERODRI 413 dewatering aid
AERODRI 419 dewatering aid

The optimum product is determined by laboratory and plant tests with the choice being based on product dosage versus moisture and soda reduction of the filter cake.

9.5 Hydrate flocculants

Polymeric flocculants are used in the tertiary hydrate thickener to:

- Reduce suspended hydrate in the tertiary thickener overflow. This increases plant productivity by reducing the amount of hydrate which is inadvertently recirculated.
- Increase the settling rate of the fine hydrate to increase thickener throughput and/or reduce the number of thickeners in service.
- Improve the rheological properties of the settled hydrate to reduce torque on the rakes and to improve pumpability of the hydrate slurry.

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9.5.1 Cytec's hydrate flocculants

The HXPAM-based products offered by Cytec are:

SUPERFLOC HF-100 HF-40 HF-80 flocculants

----------> increasing degree of hydroxamation

Cytec also offers SUPERFLOC HX-A flocculant which is a natural polymeric flocculant.

9.6 Defoamer/antifoam reagents

Excessive foaming can be a problem in several stages of the "White Side". The major problem areas are the liquor entering the precipitators and in the hydrate classification circuit. Defoamer reagents are used to help "collapse" any foam that has formed, while antifoam reagents are used to minimize the formation of foam in the first place. The major benefits of reducing foaming are:

- To reduce heat losses and thereby increase productivity in the precipitation circuit.
- To reduce scaling at the top of the precipitators. This scale can eventually fall and block the airlifts or draft tubes.
- To prevent short-circuiting of hydrate in a continuous circuit, thereby improving agglomeration and hydrate yield.
- To improve housekeeping (reduce spillage) and prevent safety hazards.

9.6.1 Cytec's defoamers/antifoams

The Cytec products available are:

CYBREAK 601 antifoam/defoamer
CYBREAK 626 antifoam/defoamer
CYBREAK 627 antifoam/defoamer
CYBREAK 631 antifoam
CYBREAK 639 antifoam
CYBREAK 640 antifoam

The optimum products for a particular application are determined by laboratory screening tests. However, since it is impossible to duplicate plant conditions exactly in the laboratory, plant tests are essential in selecting the most cost-effective product.

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10 Solvent extraction



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Section 10 Solvent extraction

10.1 Solvent extraction of metals from aqueous media

Solvent extraction (SX) is a hydrometallurgical process for the separation, purification and concentration of metal ions in solution. In its simplest form the process consists of two stages:

- *Extraction* – The metal is selectively transferred from the aqueous phase to the solvent.
- *Stripping* – The metal is transferred from the loaded solvent to the aqueous phase.

Phase contact and disengagement are commonly carried out in contactors called mixer-settlers, although other types of equipment, e.g. pulsed columns, sieve-plate columns, etc. are both available and used.

In the mixer, one phase is intimately dispersed within the other by some form of agitation. The dispersion then flows to the settler where phase disengagement occurs under quiescent conditions. Several contactors connected in series are usually needed to obtain the most efficient operation. For similar reasons, it is also common practice to contact the aqueous and solvent phases counter-currently rather than co-currently.

10.2 CYANEX extractants

All of Cytec's solvent extraction reagents are organophosphines derived from phosphine. Phosphinic and thiophosphinic acids are compound formers which extract cations, whereas phosphine oxides and sulfides are solvating agents.

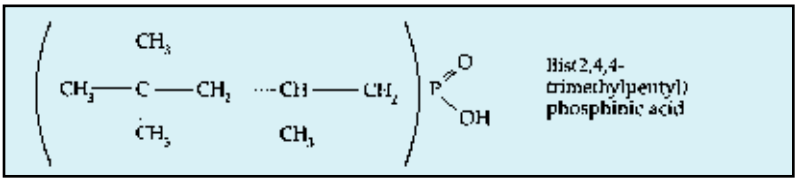
In general, the phosphine oxides, CYANEX 921 and 923 extractants have high extraction coefficients for many metals and organic solutes but very low selectivity.

CYANEX 272, a dialkylphosphinic acid and CYANEX 302, a monothiophosphinic acid, have high extraction coefficients and selectivity for many base and ferrous metals at specific pH's, but also reject calcium and magnesium.

CYANEX 301, a dialkyldithiophosphinic acid, also has a high extraction coefficient for many metals. Extraction occurs at a low pH where e.g. cobalt and nickel can be co-extracted and calcium, magnesium and manganese effectively rejected.

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CYANEX 272 extractant



This product is well established commercially and has been used in SX plants around the globe for over a decade. It has become the extractant of choice for separating cobalt and nickel from sulphate media. CYANEX 272 extractant possesses all the desired features of a good extractant including high selectivity, low aqueous solubility and high chemical stability. Notable features also include good selectivity for cobalt over calcium. Besides cobalt/nickel purification, other applications (practiced commercially) include iron and zinc extraction and the purification and separation of the heavy lanthanides. Other metals may be selectivity extracted depending on pH.

CYANEX 921 extractant

$[CH_3(CH_2)_7]_3P{=}O$ Trioctylphosphine oxide

Commonly known as TOPO, this product has been used for many years with DEHPA (di-2-ethylhexylphosphoric acid) to recover uranium from wet process phosphoric acid. It is also used to extract acetic acid from effluents from industrial processing plants. CYANEX 921 extractant possesses a high extraction coefficient for many other metals and organics such as phenol and ethanol.

CYANEX 923 extractant

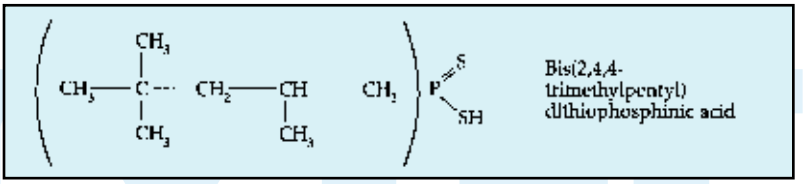
$R_3P{=}O$ $R_2R'P{=}O$ (Mixed trialkyl phosphine oxides)
$R'_3P{=}O$ $R'_2RP{=}O$
R = hexyl
R′ = octyl

A phosphine oxide which exhibits extraction properties similar to those of TOPO. It may be particularly useful in any application currently using TOPO (i.e. CYANEX 921 extractant) with the advantages associated with handling a liquid versus a solid extractant. Being completely miscible with all common diluents, a further advantage is that it can be used at higher concentrations than would be possible with CYANEX 921 extractant. It is particularly useful for the recovery of carboxylic acids, phenol and ethanol from effluent streams. It will

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also extract sulphuric, hydrochloric, nitric, perchloric and phosphoric acids. Other applications include arsenic removal from copper electrolytes. Commercial uses include the recovery of acetic acid from chemical processing plants, cadmium removal from hydrochloric/phosphoric acid mixtures and the bulk extraction of rare earths from phosphoric acid.

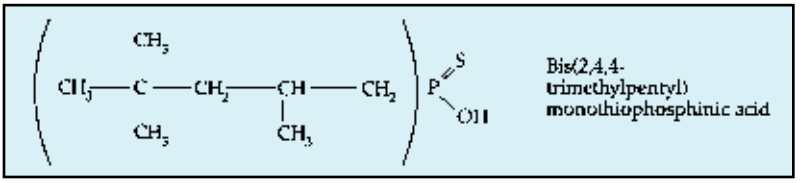
CYANEX 301 extractant



This sulphur-containing compound is a much stronger acid than its analogous oxy-acid, CYANEX 272 extractant. As such, it is capable of extracting many metals at low pH (<2). Although it does not discriminate among heavy metals in this pH range, it does exhibit a high degree of selectivity for heavy metals vs alkaline earths and alkali metals.

Applications include the co-extraction of cobalt and nickel from low pH acid leach solutions and zinc removal from acidic process effluents.

CYANEX 302 extractant



This thio acid is potentially useful for separating cobalt from nickel while rejecting manganese. It can also be used to recover zinc from sulphate media at low pH, cadmium from sulphate, chloride or mixed sulphate/chloride media and for the removal of cadmium from wet process phosphoric acid.

Detailed product brochures are available for each of these CYANEX extractants. Each brochure provides specific details on the chemical and physical properties of the extractant, recommended analytical procedures to determine chemical composition and many application details and specific examples far too numerous to present here. Please contact your local Cytec representative to request product brochures of interest.

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11 METALLURGICAL COMPUTATIONS



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Section 11 Metallurgical computations

Useful formulas and computations

With few exceptions, modern ore dressing plants are continuous operations from the moment crushed run of mine ore enters the process until the barren tailings are impounded and the extracted mineral values are ready for shipment or subsequent processing. Almost invariably, some form of wet grinding is employed as an initial treatment to liberate the mineral values from the gangue, with subsequent transport of the finely divided ore solids through the separation or extraction process as aqueous slurries or pulps.

More than ever, the successful performance of today's large, complex mineral processing plants is entirely dependent upon precise measurement and control of many process variables. These variables are measured by frequent sampling and analysis of various process pulp streams.

The following formulas and computational methods will provide the mineral engineer with a rational basis for calculating what is occurring in the plant. The material shown has been widely used by the industry in one form or another and is included here as a convenient reference for the reader.

11.1 Ore-specific gravity and pulp density relations

The inherent specific gravity of the incoming run of mine ore and the subsequent pulp densities generated in various parts of the milling circuit are important factors in many of the formulas and computations used to control plant operations and to achieve optimum process performance. Although many computer programs are now available to perform these calculations, it is important to understand the fundamental relationships involved and how they are determined.

1. The specific gravity of a solid, liquid or slurry (pulp) is defined as the ratio of the weight of a given volume of the substance to the weight of an equal volume of water at standard conditions (sp. gr. 1.000 at 4°C). For convenience, in plant practice it is usually assumed that the specific gravity of mill water is unity when making specific gravity (or density) determinations. For practical purposes, this assumption does not affect the accuracy of subsequent computations, however a correction will be necessary if precise values are required.

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a. Ore specific gravity can be readily determined by placing a known weight of dried ore into a graduated cylinder containing a known volume of water. Care should be taken to insure that the ore particles have been completely wetted and that any entrained air has been allowed to escape. The volumetric increase represents the volume of the ore sample, as follows:

Let: S = specific gravity of the ore.
w = ore weight, grams.
V = volume increase, ml.

Then: $\frac{w}{V} = S$

2. Pulp density is defined as any weight per unit volume relationship, including specific gravities. As employed in ore beneficiation, the term pulp density is often used to refer to the weight percentage of solids contained in the ore-water slurry. It is a measure of the water-to-solids ratio of the ore pulp which can be of critical importance to certain unit processes in the flowsheet. This necessitates that suitable pulp density levels be established and maintained for optimum results. Pulp density measurements are also valuable for estimating important plant tonnages and flows where other means are not available.

a. Definition and notation

Let: P = Decimal fraction of solids by weight.
S = Specific gravity of ore solids.
s = Specific gravity of pulp.
W = Weight (grams) of 1 liter of pulp.
w = Weight (grams) of dry ore in 1 liter of pulp.
D = Dilution ratio - wt. of water: wt. of dry ore in pulp
L = Weight (grams) or volume (ml) of water in 1liter of pulp.
K = The solids constant.

Assume: The specific gravity of mill water as unity:
(1000 grams per unit volume of 1 liter).

b. Formulas

From 2a, P x W = w, or $\frac{w}{W} = P$ *(1)*

then, W – (P x W) = W(1 – P) = L , the weight and volume of water. *(2)*

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also, $\frac{W}{1000} = s$, or $W = 1000s$

Hence, $\frac{P \times W}{1000 - W(1 - P)} = \frac{P \times s}{1 - (s)(1 - P)} = S$, specific gravity of the ore. *(3)*

therefore, $\frac{S(s - 1)}{s(S - 1)} = P$, decimal fraction of solids by weight. *(4)*

and, $\frac{W(1 - P)}{P \times W} = \frac{1 - P}{P} = D$, the dilution ratio. *(5)*

Also, $\frac{1 - P}{D} = \frac{1}{D + 1} = P$, the decimal fraction of solids by weight. *(6)*

c. Pulp relationships using constant, K

From the foregoing relationships a solids factor, K, is derived which ordinarily is constant for a particular ore. The following expressions are, in general, used to calculate the K value for any ore or its fraction:

$K = \frac{S}{S - 1}$ or $K = P \times \frac{s}{s - 1}$ *(7)*

hence, $S = \frac{K}{K - 1}$ *(8)*

Employing these formulas, the apparent ore specific gravity, S, and constant, K, are readily determined for any unknown ore by the simple procedure of weighing a liter (1000 ml) of pulp to obtain (s), drying the sample and weighing the remaining ore solids in order to calculate a percentage solids by weight. K is obtained by substituting this data in formula (7) and converting to S using formula (8). Once an ore's constant, K, is known, it can then be used to determine the pulp relationships of other slurries of the same ore. As follows:

$P = \frac{K(s - 1)}{s}$ or $P = \frac{K(W - 1000)}{W}$ *(9)*

$w = K(W - 1000)$ (10)

$W = 1000 + \frac{w}{K}$ or $W = \frac{1000K}{K - P}$ *(11)*

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Pulp density tables

A set of tables covering the ranges of ore specific gravities and pulp densities most commonly useful in milling will be found in Section 14.2. These tables were constructed employing the formulas given above and their use greatly simplifies the solution of many plant problems dealing with pulp flow and circulating load tonnages, as well as the sizing of pumps, conditioners, flotation cells and other process equipment.

For each given weight percent solids at a given dry ore specific gravity, the table columns show the values for:

- The weight ratio of solids to liquid. (The reciprocal of this value is the dilution ratio, D.)
- The pulp specific gravity (s).

The tables can also be used to solve for:

V = Decimal volume fraction of solids in the pulp.

$$V = \frac{P \times s}{S} \qquad (12)$$

V_p = Volume, (m^3) of 1 metric ton of pulp.

$$V_p = \frac{1}{s} = \frac{1000}{W} \qquad (13a)$$

V_s = Volume of pulp, m^3, containing 1 metric ton of dry solids

$$V_s = \frac{1}{P \times s} = \frac{V_p}{P} \qquad (13b)$$

Note: To convert to $\frac{ft^3}{\text{short ton}}$ multiply

$$\frac{m^3}{\text{metric ton}} \times 32.04$$

11.2 Flotation cell and conditioner capacities

To achieve the desired results, the volumetric capacity of the conditioners and flotation cells needed for a given feed tonnage is directly dependent upon the pulp densities and residence times required for each step. When daily ore tonnage and treatment times have been

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established, the total volumetric capacities and number of equipment units required can be estimated using the following formula:

$$N = \frac{F \times T \times V_s}{C \times 1440} \qquad (14)$$

where: N = Number of equipment units.
C = Volume per unit of equipment.
F = Dry tons ore feed per 24 hours.
T = Residence time, minutes.
V_s = Pulp volume per dry ton of ore.

Once the total volumetric requirement is known, N x C, the number of equipment units of the desired size can then be determined. In *(14)* above, no allowance is made for an increase in the required volume for flotation pulp aeration. Usually 10 to 20% additional volume is added to N x C to cover this factor.

Example: Estimate the volume of conditioners and flotation cells required to handle 9100 dry tons of ore per 24 hours at 30% pulp solids by weight, with an ore specific gravity of 3.1. Five minutes conditioning time and 15 minutes flotation time are desired.

From the tables, V_S Can be calculated:

$$V_s = \frac{1}{P \times s} = \frac{1}{(0.3 \times 1.255)} = 2.66 m^3$$

From equation *(14)*, for flotation time:

$$N = \frac{(9100)(15)(2.66)}{(1440)(C)} = \frac{252 m^3}{C}$$

Adding 15% as a volume factor for aeration, the estimated flotation cell volume needed will be $290 m^3$. If cells of $29 m^3$ volume are chosen, N will be 10.

Similarly calculating for the 5-minute conditioning time at the same pulp density gives:

$$N = \frac{(9100)(5)(2.66)}{(1440)(C)} = \frac{84 m^3}{C}$$

Therefore, the total conditioner volume required is $84 m^3$ which can be achieved with as many units of a given size as is desired.

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11.3 Determination of closed circuit mill tonnages

Circulating loads in grinding circuits

Classifiers operating in closed grinding circuits may receive feed from one or more mills as shown in Figures 6-1 and 6-2 to produce a finished size product which proceeds to the next operation, and the oversize (sands which are returned for further grinding). The Circulating Load, (CL), is the tonnage of oversize, and the Circulating Load Ratio, (Rcl) is the ratio of the circulating load to the tonnage of new ore entering the grinding circuit.

Estimates of the circulating load ratio and tonnage can be calculated on the basis of differences in the dilution ratios and screen size analyses of mill discharge(s) or classifier feed, the finished classifier product (overflow) and the classifier sands (underflow) returning to the grind. Preferably, estimates should be based on data from several sets of pulp samples taken over a period of time to assure greater accuracy of results.

Figure 6-1

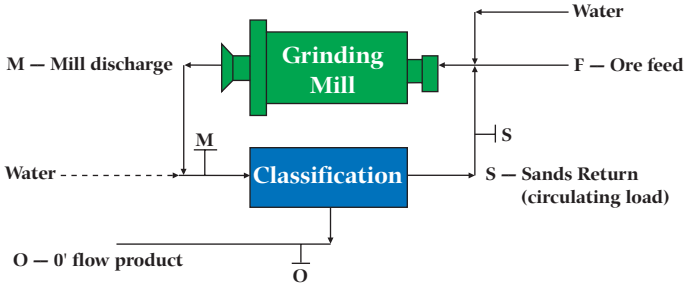
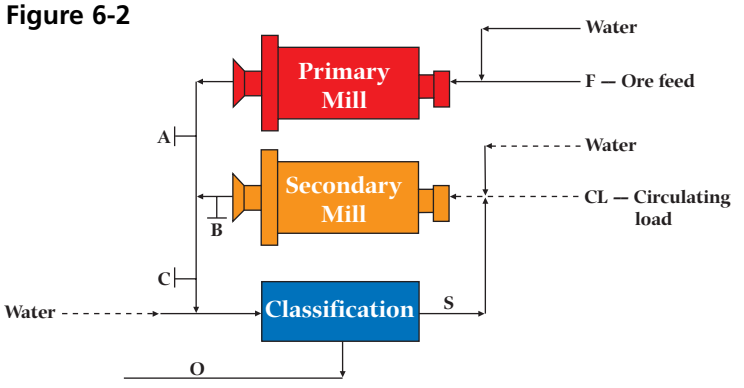


Figure 6-2



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11.3.1 Circulating load using pulp densities

Two typical grinding-classification circuits are illustrated in Figures 6-1 and 6-2, indicating nomenclature and pulp sampling points. Methods for estimating the circulating loads are given below.

a. Circuit Figure 6-1

Where, (in dry tons ore per 24 hours)

F = New ore feed to grinding.
M = Ore solids in mill discharge, or classifier feed.
S = Coarse sands returned to mill.
O = Classifier overflow product.

And, liquid-to-solid dilution ratios of pulp samples

D_m = Mill discharge, or classifier feed if dilution water is added.
D_S = Classifier sands.
D_O = Classifier overflow.

then, $\frac{CL}{F} = \frac{D_o - D_m}{D_m - D_s} = R_{cl}$, the circulating load ratio *(15)*

and, $F \times R_{cl} = CL$, circulating load (tons/24 hours)

Or, if (F) is unknown:

$R_{cl} \times 100$ = percent circulating load.

It will be seen from formula *(15)* that the capacity and separating efficiency of the classifier unit are critical factors governing the size of the circulating load, since CL becomes infinity where D_m equals D_S.

Example: A ball mill in closed circuit with a set of cyclones receives 1000 dry tons/day of crushed ore feed. The pulp densities for 0, M and S averaged 30, 55 and 72% respectively for an 8-hour shift, corresponding to D ratios of 2.33, 0.81 and 0.39. The circulating load ratio equals:

$$\frac{2.33 - 0.81}{0.81 - 0.39} = 3.62 \text{ or } 362\%$$

and the circulating load tonnage is 3.62 x 1000 = 3620 tons/day

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b. Circuit Figure 6-2

In this configuration another mill has been added to the previous circuit to increase grinding capacity. The new unit functions as the primary mill receiving only new ore feed (F), and operating in open circuit with the original mill which remains in closed circuit with the classifiers. The secondary mill now receives all of the circulating load, which can be estimated either by the previous method given, or by taking pulp samples A, B, and C to determine the respective dilution ratios, D_a, D_b and D_c.

then, $$\frac{D_a - D_c}{D_c - D_b} = R_{cl} \quad (16)$$

Example: The product from a primary rod mill receiving 1500 tons/day of new ore feed joins the product of a secondary ball mill flowing to a sump feeding a set of cyclones in closed circuit with the ball mill. The pulp densities of samples taken at points A, B and C averaged 60, 71, and 67% solids respectively, equivalent to D ratios of 0.67, 0.41 and 0.49.

then, $$R_{cl} = \frac{0.67 - 0.49}{0.49 - 0.41} = 2.25 \text{ (or } 225\%)$$

and, CL = 2.25 x 1500 = 3375 tons/day

11.3.2 Circulating loads based on screen analysis

A more precise method of determining grinding circuit tonnages employs the screen size distributions of the pulps instead of the dilution ratios. Pulp samples are screened and the cumulative weight percentage retained is calculated for several mesh sizes. The percentage through the smallest mesh can also be used to determine R_{cl}, as follows:

Circuit Figure 6-1

Where,

m = Cum. wt. % on any mesh in the mill discharge, or classifier feed.

s = Cum. wt. % on the same mesh in the classifier sands.

o = Cum. wt. % on the same mesh in the classifier overflow.

then, $$\frac{m - o}{s - m} = R_{cl} \quad (17)$$

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Example: The same as circulating load using pulp densities where the screen analyses of the three samples are as follows.

Screen analysis

Mesh Size	M %	M Cum.% (m)	S %	S Cum.% (s)	O %	O Cum.% (o)
+35	12.2	-	16.6	-	-	-
+48	27.1	39.3	34.7	51.3	0.8	-
+65	15.8	55.1	19.6	70.9	4.1	4.9
+100	10.3	65.4	9.6	80.5	12.8	17.7
+200	12.1	77.5	10.9	91.4	15.0	32.7
-200	22.5	-	8.6	-	67.3	-

Applying formula *(17)*:

The +65 mesh ratio = $\frac{55.1 - 4.9}{70.9 - 55.1} = 3.18$

The +100 mesh ratio = $\frac{65.4 - 17.7}{80.5 - 65.4} = 3.16$

The -200 mesh ratio = $\frac{22.5 - 67.3}{8.6 - 22.5} = 3.18$

From the above the average, R_{cl} is 3.19. At a 1000 tons/day mill feed rate, the circulating load is 3190 tons per 24 hours.

b. Circuit Figure 6-2

Where a, b, and c are the respective cumulative weight percentages for any given mesh size of samples A, B, and C,

and F = New feed tonnage.
CL = Circulating load tonnage.

then, $(F \times a) + (CL \times b) = (CL + F)c$ *(18)*

and, $\frac{CL}{F} \quad \frac{a - c}{c - b} = R_{cl}$

The calculations are then carried out in the same manner as for the previous example. It should be noted that errors in sampling and/or screen analyses may show widely divergent results on the different screen sizes. Any obvious anomalies should be discarded when averaging results.

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11.4 Measuring an unknown tonnage by pulp dilution

If other procedures are not practical for determining the tonnage rate of solids flowing in a certain pulp stream, an approximate measurement may be obtainable using the pulp dilution method.

This procedure is based on adding a known amount of mill water to the pulp flow for which the tonnage estimate is needed, then determining the specific gravities and dilution ratios of the pulp before and after the water addition. Ore tonnage (F) is then estimated from:

$$F = \frac{L}{D_2 - D_1} \quad (19)$$

where, F = Tons per day dry ore in pulp.

L = Tons per day mill water added.

1 short ton of water = 240 U.S. gallons

D_1, and D_2, are the dilution ratios in tons of water per ton of ore, before and after the water addition, respectively.

Note: Chemical methods have also been suggested for determining unknown mill tonnage rates but such procedures are generally impractical for all but exceptional circumstances. If of interest, reference (4) listed at the end of this section covers the subject in detail.

11.5 Classifier and screen performance formula

Classification efficiency is generally defined as the weight ratio of classified material in the sized overflow product to the total amount of classifiable material in the classifier feed, expressed as a percentage. For two-product separations, the general form used is:

$$\frac{O}{F} \times \frac{o - f}{f(100) - f)} \times 10{,}000 = \%\ \text{efficiency, E} \quad (20)$$

Where, F = Feed to Classifier, dry tons/day ore.

O = Classifier overflow, dry tons/day ore.

f = Wt. % of ore in feed finer than the mesh of separation (m.o.s.).

o = Wt. % of ore in the sized product finer than the m.o.s.

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Example: Using the calculated tonnages and the screen analysis data from previous example, determine the classification efficiency of the cyclones at a m.o.s. of 65 mesh, where 0 = 1000, F = 4190, f = 44.9 and o = 95.1:

$$E = \frac{1000}{4190} \times \frac{95.1 - 44.9}{(44.9)(100 - 44.9)} \times 10{,}000$$

= 48.4% efficiency

Screening formula

Where, a = Feed, wt.% coarser than m.o.s.
b = Feed, wt.% finer than m.o.s.
c = Oversize, wt.% coarser than m.o.s.
d = Oversize, wt.% finer than m.o.s.
f = Undersize, wt.% finer than m.o.s.
m.o.s. = Designated mesh of separation.

a. Recovery of undersize through the screen

$$\frac{(c - a)}{(c + f) - 100} \times 100 = R \text{ , wt.\% recovery of fines.} \quad (21)$$

b. Efficiency where undersize is desired product

$$\frac{Rxf}{b} = E \text{ , \% screen efficiency} \quad (22)$$

and for a quick estimate, E = 100 - d.

c. Efficiency where oversize is desired product

100% - R = 0, wt.% oversize (23)

$$\frac{O \times c}{a} = E \text{ , \% screen efficiency}$$

d. Overall efficiency of screening

$$E = \frac{(O \times c) + (R \times f)}{100} = \% \text{ overall efficiency} \quad (24)$$

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11.6 Concentration and recovery formula

Using these formulas, the metallurgical performance of the concentration plant or of a particular mill circuit is readily assessed. They are similarly applied for calculating the results of laboratory testing. Since the computations are entirely dependent on the assays and weights, where known, of the process feed and products of separation, the calculated results are only as accurate as the sampling, assaying, and weighing methods employed to obtain the required data. As will also be seen, any increase in the number of separations and mineral components to be accounted for, greatly increases the complexity of the computations.

11.6.1 Two product formula

Applicable to the simplest separation where only one concentrate and one tailing result from a given ore feed.

Definition and notation

Product	Weight or Wt.%	Sample assay %	Calculated
Feed	F	f	
Concentrate	C	c	
Tailing	T	t	
Ratio of concentration			$\overline{K}$
Recovery, %			R

a. Ratio of concentration can be thought of as the number of tons of feed required to produce 1 ton of concentrate. The ratio, $\overline{K}$, for a separation can be obtained directly from the product weights or from the product assays if the weights are not known:

$$\overline{K} = \frac{F}{C} = \frac{c - t}{f - t} = \text{the concentration ratio.} \qquad (25)$$

At operating plants, it is usually simpler to report the $\overline{K}$ based on assays. If more than one mineral or metal is recovered in a bulk concentrate, each will have its own $\overline{K}$ with the one regarded as most important being reported as the plant criteria. If the tonnage of concentrates produced is unknown it can be obtained using the product assays and the tons of plant feed:

$$C = \frac{F}{\overline{K}} = F\,\frac{f - t}{c - t} = \text{the weight of the concentrate.} \qquad (26)$$

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b. Recovery, %

Represents the ratio of the weight of metal or mineral value recovered in the concentrate to 100% of the same constituent in the heads or feed to the process, expressed as a percentage. It may be calculated in several different ways, depending on the data available.

By assays f, c and t only:

$$R = \frac{c(f - t)}{f(c - t)} \times 100 = \text{recovery } \% \qquad (27)$$

By $\overline{K}$ plus assays f and c

$$R = \frac{c}{\overline{K}f} \times 100 = \text{recovery } \% \qquad (28)$$

By weights F and C, plus assays c and t

$$R = \frac{Cc}{Cc + t(F - C)} \times 100 = \text{recovery, } \% \qquad (29)$$

Example: A copper concentrator is milling 15,000 tons/day of a chalcopyrite ore assaying 1.15% copper. The concentrate and tailings produced average 32.7% and 0.18% copper, respectively. Calculate:

by (25) $\overline{K} = \dfrac{32.7 - 0.18}{1.15 - 0.18} = 33.53$

by (26) $C = \dfrac{15{,}000}{33.53} = \dfrac{(15{,}000)(0.97)}{32.52} = 447.4$ tons

by (27) $R = \dfrac{(32.7)(1.15 - 0.18)}{1.15(32.7 - 0.18)} \times 100 = 84.8\%$

by (28) $R = \dfrac{32.7}{(33.53)(1.15)} \times 100 = 84.8\%$

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11.6.2 Three product (bi-metallic) formulas

Frequently, a concentrator will mill a complex ore requiring the production of two separate concentrates, each of which is enriched in a different metal or valuable mineral, plus a final tailing acceptably low in both constituents. Formulas have been developed which use the feed tonnage and assays of the two recovered values to obtain the ratios of concentration, the weights of the three products of separation, and the recoveries of the values in their respective concentrates. For illustrative purposes data from a copper-zinc separation is assumed.

Definition and notation

Product	Weight or Wt.%	% Cu Assay	% Zn Assay	Calculated
Feed	F	c_1	z_1	
Cu concentrate	C	c_2	z_2	
Zn concentrate	Z	c_3	z_3	
Tailing	T	c_4	z_4	
Ratios of concentration				$\overline{K}_{cu}$ and $\overline{K}_{zn}$
Recovery, %				R_{cu} and R_{zn}

The ratios of concentration, $\overline{K}_{cu}$ and $\overline{K}_{zn}$ are those for the copper and zinc concentrates, respectively, with R_{cu} and R_{zn} the percentage recoveries of the metals in their corresponding concentrates. As follows:

$$C = F \times \frac{(c_1 - c_4)(z_3 - z_4) - (z_1 - z_4)(c_3 - c_4)}{(c_2 - c_4)(z_3 - z_4) - (z_2 - z_4)(c_3 - c_4)} = \text{tons Cu concentrate} \quad (30)$$

$$Z = F \times \frac{(c_2 - c_4)(z_1 - z_4) - (c_1 - c_4)(z_2 - z_4)}{(c_2 - c_4)(z_3 - z_4) - (z_2 - z_4)(c_3 - c_4)} = \text{tons Zn concentrate} \quad (31)$$

$$R_{cu} = \frac{C \times c_2}{F \times c_1} \times 100 \text{ copper recovery, \%} \quad (32)$$

$$R_{zn} = \frac{Z \times z_3}{F \times z_1} \times 100 \text{ zinc recovery, \%} \quad (33)$$

$$\overline{K}_{cu} = \frac{F}{C} \text{ and } \overline{K}_{zn} = \frac{F}{Z} = \text{ratio of concentration} \quad (34, 35)$$

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Example:

Product	Tons	Assay % Copper	Zinc
Feed	1000	2.7	19.3
Cu concentrate	C	25.3	5.1
Zn concentrate	Z	1.2	52.7
Tailing	T	0.15	0.95

Then,

$$C = 1000 \times \frac{(2.7 - 0.15)(52.7 - 0.95) - (19.3 - 0.95)(1.2 - 0.15)}{(25.3 - 0.15)(52.7 - 0.95)(5.1 - 0.95)(1.2 - 0.15)}$$

$$= 1000 \times \frac{131.96 - 19.27}{1301.51 - 4.36} = \frac{112{,}690}{1297.15} = 86.9 \text{ tons Cu concentrate}$$

$$Z = 1000 \times \frac{(25.3 - 0.15)(19.3 - 0.95) - (2.7 - 0.15)(5.1 - 0.95)}{(25.3 - 0.15)(52.7 - 0.95) - (5.1 - 0.95)(1.2 - 0.15)}$$

$$= 1000 \times \frac{461.50 - 10.58}{1301.51 - 4.36} = \frac{450{,}920}{1297.15} = 347.6 \text{ tons Zn concentrate}$$

$$R_{cu} = \frac{(89.9)(25.3)}{(1000)(2.7)} \times 100 = \frac{2198.6}{2700} \times 100 = 81.4\%$$

$$R_{zn} = \frac{(347.6)(52.7)}{(1000)(19.3)} \times 100 = \frac{18{,}318.5}{19{,}300} \times 100 = 94.9\%$$

$$\overline{K}_{cu} = \frac{(1000)}{(86.9)} = 11.51, \quad \overline{K}_{zn} = \frac{(1000)}{347.6} = 2.88$$

The three product solution illustrated above can be somewhat simplified by taking an intermediate tailings sample between the two stages of concentration; i.e., a copper tail (zinc feed) sample in the previous example. Then, adding the notations:

Copper tail (zinc feed) = CT
with copper and zinc assays = c_5 and z_5

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Assume mill feed, F, as Unity 1

Then,
$$C + CT = 1 \quad (a)$$
$$(C \times c_2) + (CT \times c_5) = c_1 \quad (b)$$
$$(C \times c_5) + (CT \times c_5) = c_5 \quad (c)$$

Subtracting (c) from (b),
$$C(c_2 - c_5) = (c_1 - c_5)$$

Then, $C = F \dfrac{(c_1 - c_5)}{(c_2 - c_5)}$ = tons copper concentrate *(36)*

and similarly, $Z = (F - C) \dfrac{(z_5 - z_4)}{(z_3 - z_4)}$ = tons zinc concentrate *(37)*

Example: It is decided to take a copper tail (zinc feed) sample in order to provide a check on the results calculated in the previous example. The sample (CT) assayed 0.55% Cu (c_5) and 20.9% Zn (z_5), respectively. The check weights of the copper and zinc concentrates are computed as follows:

Copper concentrate,

$$C = 1000 \times \frac{(2.7 - 0.55)}{(25.3 - 0.55)} = 1000 \times \frac{(2.15)}{(24.75)} = 86.9 \text{ tons}$$

Zinc concentrate,

$$Z = (1000 - 86.9) \times \frac{(20.9 - 0.95)}{52.7 - 0.95} = 913.1 \times \frac{(19.95)}{(51.75)} = 352.0 \text{ tons}$$

As can be seen, the calculated weights of the copper concentrate check exactly, while the zinc concentrate checks within 1.3%. An average of the zinc concentrate weights, obtained using both methods, could be used if desired.

It should be understood that there are certain limitations to the use of three-product formulas, since it is required by definition that two of the three products involved must be concentrates of essentially different metals or mineral components. The formulas will only give reliable results when the assays indicate that a differential concentration of the two components into separate concentrates has occurred.

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11.7 Flotation reagent usage formula

The consumption or usage rate of the chemicals employed in flotation is generally expressed in terms of grams per metric ton of ore treated. Depending upon the particular reagent, it may be fed as a dry solid, as a water solution or dispersion, or in the undiluted "as-is" liquid form. The normal procedure when checking or setting reagent feed rates is to measure the amount being fed to the circuit per unit time, usually per minute. Liquid or reagents in solution or dispersion are measured in ml and dry solids in grams. When feeding liquids, the specific gravity and weight percent strength of the reagent must also be known. With this information, along with the known ore tonnage being treated per unit time, the reagent measurements can then he translated into grams/metric ton consumption rates, as follows:

11.7.1 For dry reagents

$$\frac{(\text{g reagent / min.})(1440 \text{ min. / day})}{\text{tons ore / day}} = \frac{\text{g reagent}}{\text{ton ore}} \quad (38)$$

11.7.2 For liquid reagents

$$\frac{(\text{ml reagent / min.})(\text{reagent sp. gravity})(1440 \text{ min. / day})}{\text{tons ore / day}} = \frac{\text{g reagent}}{\text{ton ore}} \quad (39)$$

11.7.3 For reagents in solution

$$\frac{(\text{ml solution / min.})(\text{g reagent / liter solution})(1440 \text{ min. / day})}{\text{tons ore / day x 1000}} = \frac{\text{g reagent}}{\text{ton ore}} \quad (40)$$

Note: $\frac{1\text{g}}{\text{metric ton}} = \frac{0.0020\text{lb}}{\text{per short ton}}$

Example: At a 10,000 tons/day milling rate, a plant is using 590 ml/min. of a 200 g/L xanthate solution. Calculate the dosage rate.

$$\frac{(590)(200)(1440)}{10{,}000 \text{ x } 1000} = 17\text{g/t}$$

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11.8 Material balance software

In the past few years several software programs have been introduced to perform aforementioned computations as well as to provide material balances in operating circuits using several sophisticated statistical tools. Examples of commercially available software packages include MATBAL* and JKSimMet**. Excel Solver can also be used.

* MATBAL is a proprietary program of Algosys Inc.

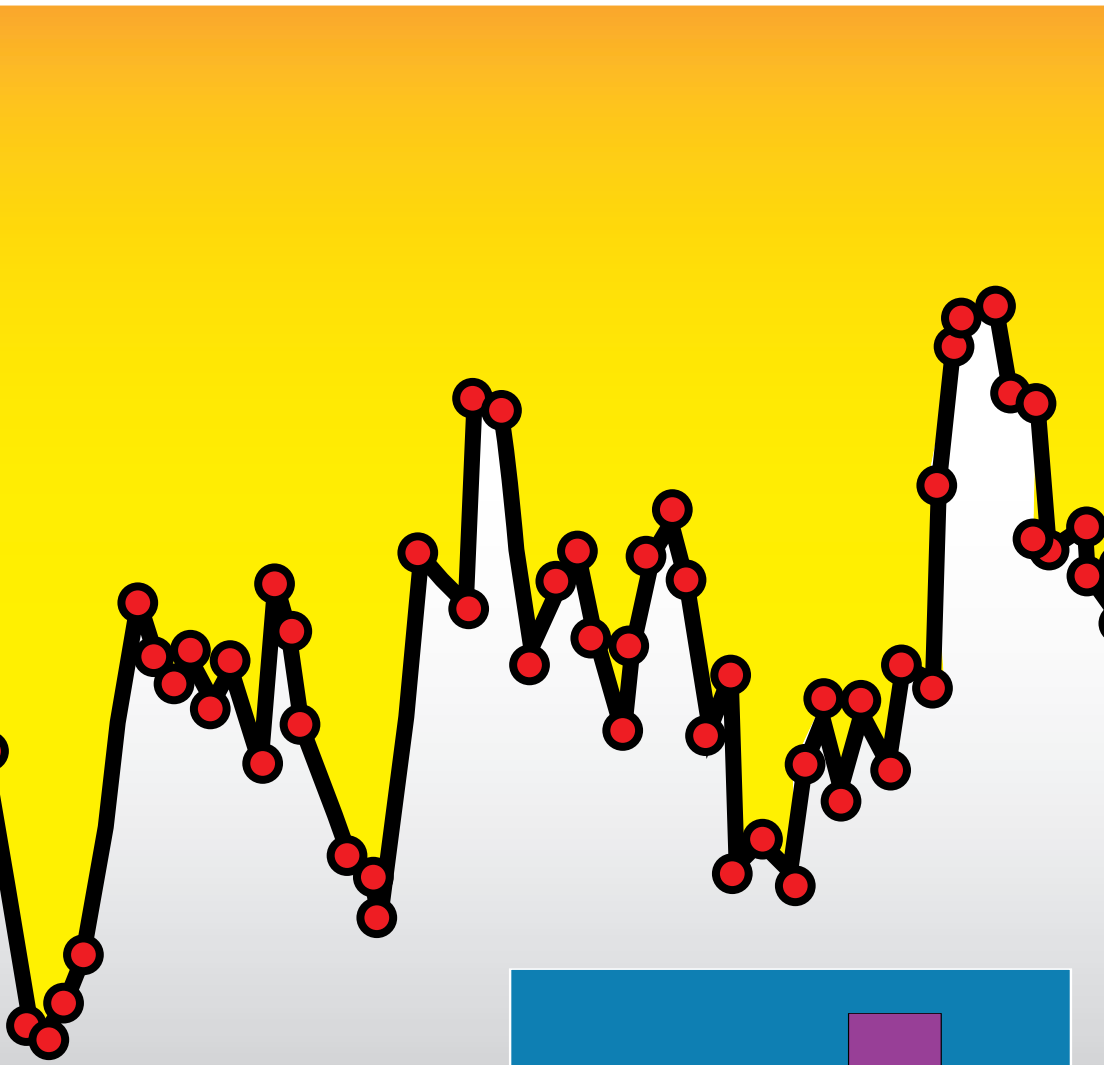
** JKSimMet is a proprietary program of JK Tech/Contract Support Services

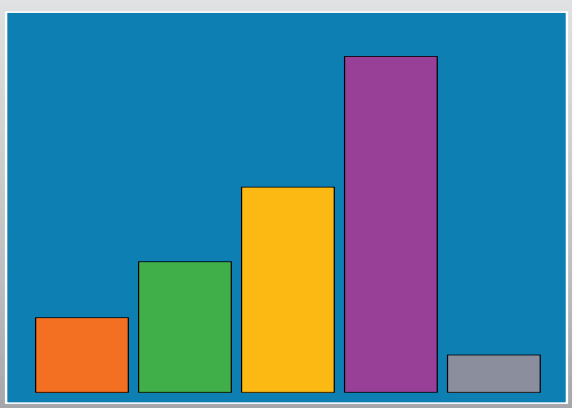
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12 Statistical methods in mineral processing





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Section 12
Statistical methods in mineral processing

12.1 Statistics in laboratory work

The purpose of laboratory work is to screen potential products for the customer's application, to identify potential improvements attainable using them, and to generate information to justify testing at larger (plant) scale.

To be a sound basis for operating decisions, the data generated in a program of tests, and the conclusions drawn from that data, should meet accepted scientific standards. Statistical procedures are accepted standard methods for drawing conclusions from data, and using them will add credibility to conclusions. In addition, from laboratory work it is often required that we characterize the performance of a new proposed system as a function of several factors. This will be the case when realizing improvements from a new reagent requires, in addition, other changes in process conditions or plant operations. Also part of the statistical approach, are methods for modeling complex, multivariable systems over a range of operations with a reasonable amount of effort.

12.1.1 Statistical distributions and summary statistics

How large a change in performance can be detected in a test program depends on the magnitude of errors due to the test procedure and to analyses, and on steps taken to minimize the impact of the systematic sources of error. Error, as used in the statistical sense referring to the numerical result obtained from an experiment, is the difference between the actual result and its ideal or "true" value. Just how large this is depends on (generally unperceived) variations

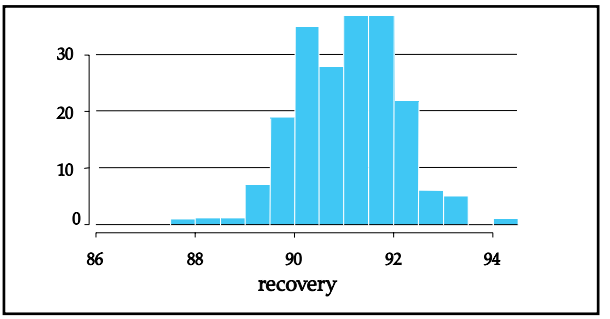


Figure 1.
Distribution of 200 observations from a theoretical normal distribution with mean 91.0, standard deviation 1.0

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in materials and technique, and how sensitive the final result is to those variations. A distribution function represents the variability of a test result. Recovery of Cu for many tests with a standard reagent might, for example, form a distribution like that illustrated in Figure 1. The horizontal axis gives values of the recovery; the vertical axis, numbers or the fraction of observations in each category of recovery value. If the total number of observations increased, the form of this distribution would approach a limiting curve. It is unlikely you will see so many observations in laboratory work; however, it is important to understand that any particular experimental result is just one observation from an ensemble of potential results described by such a distribution curve.

Average (or mean) and standard deviation are the most common statistics for summarizing either a distribution or a set of results. The same terms (mean, standard deviation) are used to denote the mean and to estimate its confidence limits from a small sample of observations from the distribution. Standard formulae for calculating the mean and standard deviation for a small set of data are below. Hand calculators with statistical functions can calculate these directly, and spreadsheet software provides these as built-in functions.

$$\bar{x} = \frac{1}{n} \sum_{i=1}^{n} x_i \qquad s = \sqrt{\frac{1}{n-1} \sum (x_i - \bar{x})^2}$$

Confidence limits for the calculated mean are:

$$\bar{x} \pm \frac{ts}{\sqrt{n}}$$

where t is the value obtained from Student's t table with n-1 degrees of freedom and the chosen confidence level. 95% is the most common confidence level for reporting.

Example: A flotation test is repeated 5 times on a substrate. The recovery results are: 90.2, 90.5, 89.3, 90.0, 90.2. The average and standard deviation are:

$$\bar{x} = \frac{90.2 + 90.5 + 89.3 + 90.0 + 90.2}{5} = 90.04$$

$$s = \sqrt{\frac{(90.2 - 90.04)^2 + (90.5 - 90.04)^2 + \ldots}{5 - 1}} = 0.45$$

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When reporting an average and standard deviation, a useful general rule is to round the standard deviation to two significant figures and the average to the same place of decimals as the standard deviation.

The confidence limits for the calculated mean are 90.04 ± 0.56 (95% confidence). The 0.56 figure comes from $ts/\sqrt{n}$ where t=2.776 from the Student's t table, with s=0.45 and n=5.

12.1.2 Statistical considerations in comparative testing

In testing reagents where incremental improvements in performance are sought, it is common for the magnitude of improvements, the precision of analyses, the systematic error of results, and the effects of deliberate variations in laboratory technique or treatment of the data, all to be comparable in magnitude.

Techniques to cope with these random and non-random sources of error in testing, so that valid conclusions can be drawn despite several sources of error in experimentation, include: use of controls, replication, randomization, and blocking.

Controls are the principal guard against effects of ore variation and most systematic sources of testing error. A control is a standard test condition, often representing current practice in the plant, against which other results are compared. One or more runs of the control are run beside, or in the same experiment series with, test reagents, and the results of tests compared with these controls. The difference between test and control run using the same ore is likely to be more accurate than a comparison of a test result with a fixed number.

Replication of experiment runs accomplishes several goals. Agreement among repeat runs of a given experiment provides a quality control check on their results. Second, replication of control and experimental runs enables an estimate of error to be derived from a body of experimental work. This is necessary for application of most formal statistical methods. Third, the average of replicated runs is more precise, due to the "law of averages", than single determinations.

Randomization guards against some more sources of systematic error in testing. Results for samples being tested in a given session may change systematically from beginning to end, due to aging of the samples or to improvement (or degeneration) of the experimenter's technique in the course of testing.

Blocking is a way to improve testing accuracy when replicated tests are used. It consists of dividing the tests into subsets (blocks) which can be conducted over a relatively short time and with relatively

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uniform material, each block containing one or more replicates of each treatment. In the statistical analysis, the standard deviation within blocks is estimated and determines the precision of treatment comparisons.

12.1.3 Comparison of two treatments with the unpaired t test

The simplest comparative experiment is to compare two or more treatments using a given test. Consider the comparison of a candidate reagent against that currently in use. A procedure for carrying out the comparison using replication to enable statistical procedures to be employed is:

1. Choose a number of replicate tests to be run for both.
2. Use a randomization procedure to generate an order to run tests and controls.
3. Carry out the runs.
4. Compute and report a confidence interval for the difference in response between the candidate and control reagents.

The randomization of the order of runs is the key feature of the procedure. It protects the results against distortions due to time effects and ensures that the variability of samples reflects the full variability of the test procedure. Variability of test results interspersed with tests at other conditions is larger than that of back-to-back repeats of the same test; the larger variability is the one that actually reflects the error in comparisons between the different reagents.

A confidence interval for the difference in mean recovery between treatment A and control is a useful standard way to report the comparison. The confidence interval for a difference between two means is calculated by the unpaired t confidence interval formula.

$$\bar{x}_A - \bar{x}_B \pm ts_P = \sqrt{\frac{1}{n_A} + \frac{1}{n_B}}$$

n_A and n_B are the numbers of observations for the two treatments. The pooled standard deviation is calculated from standard deviations of the two groups as

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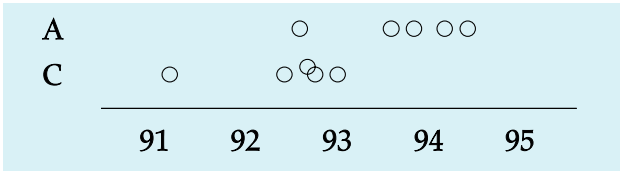
$$s_P = \sqrt{\frac{(n_A - 1)s_A^2 + (n_B - 1)s_B^2}{n_A + n_B - 2}}$$

The factor t depends on the degrees of freedom ($n_A + n_B - 2$ for this two sample test) and the confidence level (95% is customary for most purposes). It must be looked up in a table of Student's t, contained in most collections of mathematical tables such as those in the CRC Handbook of Chemistry and Physics. For 95% confidence, the tabulated values of t are approximately 2.

Example: To compare a treatment A with control C, using ten runs in all, we generate a random sequence of 5 A and 5 C, and carry out the ten runs in that order. We suppose the results, recoveries for each of the ten tests, are:

Test	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**
Treatment	C	A	C	C	A	A	A	C	C	A
Recovery	91.2	93.6	92.4	92.7	92.6	93.8	94.4	93.0	92.7	94.1

A diagram such as the dot plot below gives the clear impression that treatment A gives higher recovery than C; however, from the statistical analysis it will turn out that the difference is near the edge of statistical significance.



Given these data, average and standard deviations are:

Treatment	**Average**	**Std deviation**
A	93.7	0.69
C	92.4	0.70

The confidence interval for the difference in mean recovery is

$s_P = \sqrt{}\ (\ (n_A\text{-}1)^* s_A^{\,2} + (n_B\text{-}1)^* s_B^{\,2}\ )/\ (n_A + n_B\text{-}2) = 0.695$ [pooled std deviation]

$93.7 - 92.4 \ \pm\ 2.206 \times 0.695 \times \sqrt{}(1/5 + 1/5) \ =\ 1.3 \pm 1.2$

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12.1.4 Comparison of two treatments using the paired t test

When comparing two treatments, somewhat better accuracy for comparison might be obtained if tests are conducted, not in a random order, but alternating between the two treatments. The idea of randomization suggests, in this case, the modification where pairs consisting of one test for each of the two treatments are run in random order.

With paired observations, an alternative form of the t confidence interval is used.

$$x_A - x_B \pm t s_d \sqrt{\frac{1}{n}}, \text{ where } s_d = \sqrt{\frac{\sum (x_{Ai} - x_{Bi})^2}{n - 1}}$$

s_d is the estimated standard deviation of differences of pairs of tests. The Student's t factor is for n-1 degrees of freedom and the desired confidence level.

Example: Performance of two reagents is tested on a pulp which varies over time. The work is carried out by taking a pulp sample and running it in the laboratory, using both the standard control reagent, and a test reagent. Results for five pulp samples are:

Test	Control	Difference
91.1	90.2	0.9
87.4	86.8	0.6
89.2	89.2	0.0
91.0	90.5	0.5
93.0	92.8	0.2

The average and standard deviation of the differences are:

$\overline{d} = 0.44$

$s_d = 0.35$

The 95% confidence interval for the difference is then:

$0.44 + 2.76\ (0.35/\sqrt{5}) = 0.44 \pm 0.43$

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12.1.5 Response surface analysis

In response surface analysis, we characterize performance of a system as a function of one or more continuously variable factors. A response that we are interested in is regarded as a function of these variables or factors. For example, the filtration rate of a flocculated suspension of mineral may depend on a reagent dosage, mixing rate, pH, and other variables connected with the test system. There are two parts to the methodology. First, the design of experiments is concerned with the arrangement of observations needed to generate information from which the unknown function can be inferred. Second, response surface methods provide tools to derive response functions from the data and to work with and visualize the functions.

For example, we may be interested in the maximum of the dose response curve generated by varying dose of a given reagent. The curve is a response surface with one factor. The experimental design to estimate it will consist of tests at a number of doses (three or more) in a range of interest. Statistical analysis will consist of fitting the function using linear or nonlinear regression methods.

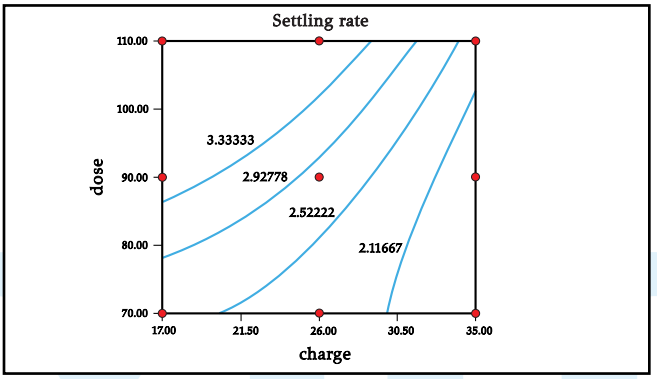
For two or more factors, empirical response functions are linear and polynomial function forms, quadratic and cubic. Tools to lay out the experimental designs and to fit empirical response functions are provided in statistical software such as Echip and Design Expert. Semi-empirical equations have an algebraic form derived from simplified theoretical analysis of the system, and parameters to be determined by fit to the data. Generally, the same response surface designs intended for empirical model fitting will also be good for estimating the parameters of such custom equations.

Example: A nine-point experiment was carried out to determine settling rate of flocculated mineral as function of the dose of a flocculating reagent and its percent charge, a function of composition. Results of the tests are:

	Charge on reagent	17	26	35
		settling rate, m/hr		
Dose, g/t	70	2.6	2.5	1.6
	90	3.3	2.9	2.0
	110	5.2	3.5	2.4

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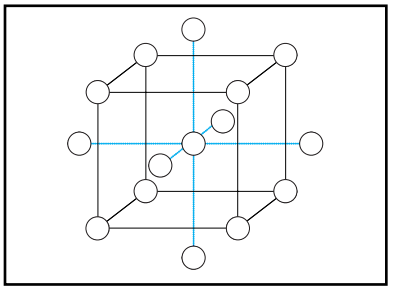
The following quadratic response surface was fitted and represented as a contour plot using software for response surface analysis.

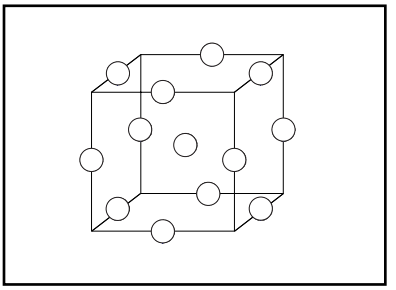


Response surface designs for 3 factors

For the study of the effects of three or four factors, specialized response surface designs, intended for fitting quadratic functions to data, are recommended. The possible experiment conditions, choices of levels of the three or four factors, can be thought of as defining points in three or four-dimensional space. The experiments to carry out can be represented as a geometric figure in this space. For more than 4 factors, response surface designs have a large number of points due to the many parameters of the general quadratic function and are therefore not commonly used.

For three factors, the Box-Wilson or face-centered cubic design pictured below (left) consists of 15 or more points, eight at the corners of a cube, two each on each of the three axes, and one or more at the center. The Box-Behnken design (right) for three factors consists of 13 or more points. Twelve are at midpoints of the edges of a cube and correspond to experiments where one of the three factors is at its midpoint value, the other two at high or low levels. One or more midpoints complete the design.





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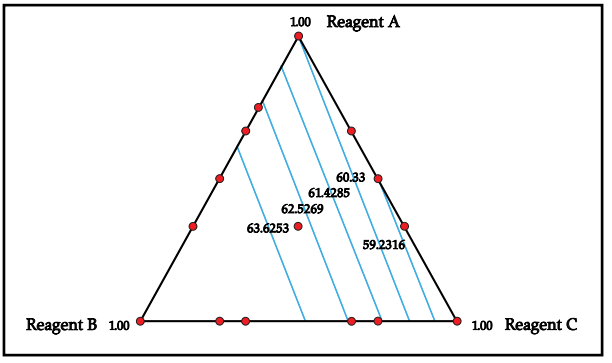
12.1.6 Mixture experiments

Mixture designs are a special type of factorial experimental design. They are used to optimize a reagent system which is a formulation with two or more components. The amounts of each component are factors in the sense of response surface designs, and the objective of laboratory work is to model (i.e., to derive an equation for) a response, say recovery of a mineral as a function of the proportions of the components. The difference between mixture and response surface designs is that, in mixture designs, the proportions of several constituents are constrained to add to one. The range of the factors is not a general region but a line segment in one dimension (for two constituents), a triangle in two dimensions (three constituents), or generally a simplex.

Mixture experimental designs are most often used to optimize formulations when a synergistic pair or trio of reagents has been found. A synergistic mixture is one where the response, for example recovery, is higher for the mixture than the average of responses for the constituents. Two reagents which are selective to different minerals, are likely to be synergistic.

Example: Three Cytec flotation reagents and mixtures of them were tested on a copper ore. The mixture experimental design includes runs of each reagent alone, of mixtures of the two, and of mixtures of all three, the constraint being that the total of doses for the three reagents is the same. A quadratic function was fit to Cu recoveries from the tests. The figure shows the arrangement of 15 reagent mixtures which were tested; they are represented as red dots on the triangular plot. Contours for a quadratic function fit to the results are also shown.

The overall conclusion from this set of tests is that effectiveness of the reagents are B > C > A ; the highest recoveries are in the B corner. Cost of the reagents may, however, make a point along the BA or BC axis the optimum for the application.



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Section 12.2 Planning and analyzing plant trials

An evaluation of a new reagent or a new set of operating conditions in a mineral processing plant generally involves changing from the standard or control reagent or set of operating conditions to a test reagent or set of operating conditions. Data are collected during one or more periods (e.g. shifts, days, weeks) of operation under the test regime and are compared to data collected during a like number of periods of operation under the control regime. Control periods may precede, follow, or be interspersed among the test periods. For a given measure of performance (e.g. percent recovery), the comparison is the difference in average performance between test and control periods. The main planning variable is the length and number of periods to run under the test and control regimes.

The most important variable affecting the overall metallurgical performance in most flotation plants is the "quality" (i.e. flotation characteristics) of the ore entering the plant. Unfortunately, this is usually the variable over which the plant operator has the least control. Two principles should be applied to improve the precision of "test versus control" comparisons in view of the importance of this source of variability. The first is to intersperse test and control periods, which achieves the same effect as replication in laboratory experiments. The second is, where possible, to use multiple lines where test and control regimes are run side by side to improve comparisons.

12.2.1 Sequential or "switchover" trials

The first thing to know about planning plant trials is that interspersing test and control periods is a key to better precision of reagent or operating condition comparisons. A common trial plan is simply to run a single line for a single unbroken period under the test regime and attempt to compare performance with previous data. A misconception about this one period trial is that longer is better, as far as power to detect small differences is concerned. In fact, it is often the case that, beyond a certain point, lengthening the trial actually decreases its power to detect small differences by exposing the trial to the effects of variability from sources operating on longer time scales. For example, when a month of test operation is compared to the preceding month of control operation, day-to-day variation is effectively averaged out, but month-to-month variation becomes important.

Instead of a trial comparing a month of test operation to the preceding month of control operation, a trial comparing four weeks of test operation interspersed with four weeks of control operation could be run. Such a design still averages out the week-to-week variation and also distributes test and control periods within each month, thus canceling out month-to-month variation.

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From the standpoint of maximizing the power of the trial to detect small differences by dealing with variation on more than one time-scale, doing more switchovers tends to be better than doing fewer. But frequent switching over does increase the logistical complexity of the trial, and can require operating in a way that is no longer representative of actual long-term operation.

The form of the on-off trial with a single line is illustrated as the prototype for the slightly more elaborate designs involving two lines. (See Section 12.2.2) Operation of the line is cycled between the test and control reagent. Each test period is paired off with the control period (either before or after, in this case after). An estimate of the effect of the test reagent, or difference in response between the test and control, is available for each such pair. An approximate confidence interval for the difference is derivable from the t test. The degrees of freedom for t are n-1, where n=3 in the example.

Single line on-off trial design

	Line 1		
1	test	y_1	$d_1 = y_1 - x_1$
2	control	x_1	
3	test	y_2	$d_2 = y_2 - x_2$
4	control	x_2	
5	test	y_3	$d_3 = y_3 - x_3$
6	control	x_3	

Confidence interval for the (test-control) comparison

$$\bar{d} \pm t s_d \sqrt{\frac{1}{n}}, \text{ where } s_d = \sqrt{\frac{\sum (d_i - \bar{d})^2}{n - 1}}, n = \text{number of test periods}$$

For a discussion on the use of the REFDIST approach to planning and analyzing sequential plant trials, please see Section 12.2.3

12.2.2 Parallel line trials

If the plant has two or more similar sections or lines, it is an effective strategy to run simultaneous "parallel" or "side-by-side" trials. Test and control regimes are run at the same time on different lines and the results compared at each point in time. With this arrangement, the period-to-period variation is subtracted out of the

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comparison of test and control regimes, resulting in greater power to detect small differences. Usually, some provision is made for switching regimes between lines, so that consistent line-to-line differences can also be eliminated from the comparison of regimes.

Ideally, the sections should be completely separate through all the stages, including regrinding and cleaner flotation. If the sections are separate only through the rougher stage, the operator should bear in mind the effects which any recycle streams (both mineral and reagent-containing water) may have. Rougher grade/recovery data can be a useful indication of how the two reagent regimes might be expected to perform on a total-plant basis. However, we recommend that promising rougher circuit performance be confirmed by full-plant testing, to ensure that the predicted benefits extend through the regrind and cleaning circuits.

Two lines with alternation between test and control reagent on one of them

A test plan for a trial carried out in a plant with parallel lines, but with provision for feeding the test reagent on Line 1 only, is shown below. The response, e.g., recovery, is indicated as y_i for the test reagent, x_i for Line 1 running the control, and w_i for line 2. The analysis of the experiment starts with calculation of test minus control comparisons, d_i, which are designed so that consistent line, and some time differences, will cancel out.

	Line 1	Line 2			
1	test	control	y_1	w_1	$d_1 = y_1-x_1-w_1+w_2$
2	control	control	x_1	w_2	
3	test	control	y_2	w_3	$d_2 = y_2-x_2-w_3+w_4$
4	control	control	x_2	w_4	
5	test	control	y_3	w_5	$d_3 = y_3-x_3-w_5+w_6$
6	control	control	x_3	w_6	

Two-line crossover design

In the two-line crossover design, reagent regimes for the two lines are swapped, or crossed-over, between test periods. This type of trial does depend on being able to use the test reagent on either line. The form of comparison corrects for the same sources of variation common to the lines as the previous design. An advantage is that test reagent feed is not stopped altogether at any time during the trial.

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	Line 1	Line 2			
1	test	control	y_1	x_1	$d_1 = (y_1+y_2-x_1-x_2)/2$
2	control	test	x_2	y_2	
3	test	control	y_3	x_3	$d_2 = (y_3+y_4-x_3-x_4)/2$
4	control	test	x_4	y_4	
5	test	control	y_5	x_5	$d_3 = (y_5+y_6-x_5-x_6)/2$
6	control	test	x_6	y_6	

Confidence interval for the "test-control" comparison

The following equation is used to calculate a confidence interval for the mean difference between test and control results in either of the two designs described above. The equation is formally equivalent to the paired t test in Section 12.1. The effective sample size "n", is the number of switchovers or crossovers per line to the test reagent. (n=3 in both the examples above).

$$\bar{d} \pm t s_d \sqrt{\frac{1}{n}}, \text{ where } s_d = \sqrt{\frac{\sum (d_i - \bar{d})^2}{n-1}}, n = \text{number of test periods}$$

12.2.3 The REFDIST approach to planning and analysis of sequential plant trials

If performance data are available from a period of routine operation under the control regime for some length of time before the trial was conducted, they can be used to calculate statistical criteria for planning and for judging the outcome of the trial.

The REFDIST (for "reference distribution") approach to analyzing accumulated data on plant operations was pioneered by Cytec. It provides a basis not only for calculating an objective criterion for trial success, but also for identifying a trial design that is most powerful for substantiating treatment effects in the presence of routine variation. It takes correct account of the fact that, in continuous operations, data take the form of a "time series" of values that often fail to conform to the assumptions required for simpler statistical analyses.

The basic idea of the approach is to calculate "test-minus-control" differences in sets of consecutive measurements drawn from the accumulated data, where the labels "test" and "control" are assigned to the measurements in the same pattern as test and control conditions would be implemented in the actual trial. Assuming that no deliberate changes in operating conditions were being made when these measurements were taken, the calculated differences reflect

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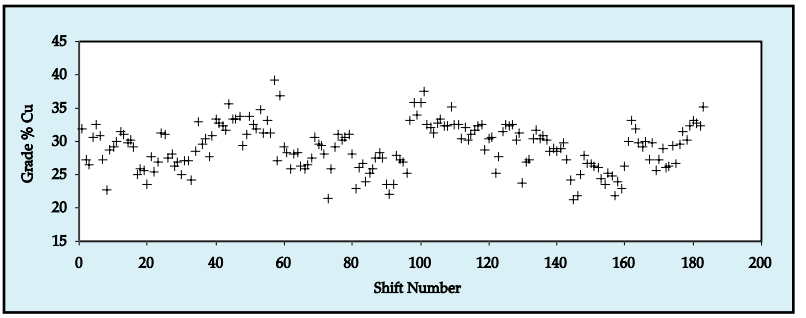
routine variation, expressed in a form that is directly comparable to the actual trial result. If significant changes to plant operating conditions were made during the "base-line" period, it may be possible to modify the REFDIST analysis to take these into account.

The set of calculated differences, or reference distribution, can validly be used to assess the outcome of the actual trial. When the difference observed in the actual trial exceeds in magnitude most or all of the differences tabulated in the reference distribution, the conclusion may reasonably be drawn that the change in operating conditions has a real effect on the performance of the process. This use of the reference distribution for trial evaluation is an alternative to the Student's t confidence interval. The reference distribution is also valuable for planning purposes. A percentile of the reference distribution for a given trial design measures the size of difference between test and control reagents required to be reliably detected with the proposed trial.

These criteria will be valid regardless of whether or not the variation conforms to the assumptions of standard statistical tests. In particular, the assumption that each data point represents an independent random sample of process performance is often violated in the plant trial situation. Their validity does depend, however, on the amount and form of the routine variation that occurred when the data were accumulated being representative of the routine variation that occurs during the actual trial.

An example using plant data to plan a trial

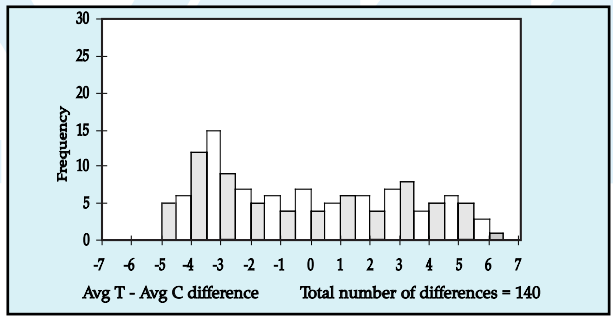
The following figure illustrates copper grade recorded for each 12-hour shift over a three-month period. The data were extracted from the plant database to help in planning a trial to compare a new collector to the standard (control) collector.
The REFDIST approach can be used with these data to calculate "critical values" that a Test-minus-Control difference in average grade recorded in the trial must exceed in order to "stand out" from



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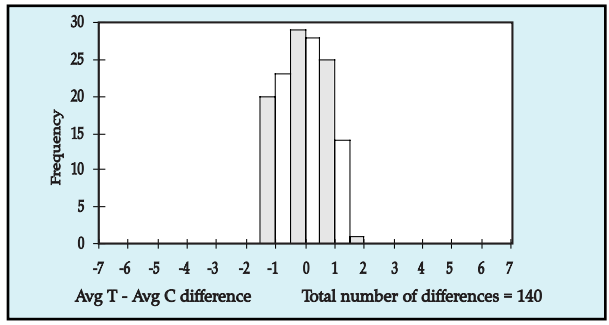
the routine variation. The calculations can be done for each of several possible trial designs and the results compared to see which design gives the smallest critical values.

The following figure illustrates the reference distribution for one particular trial design, a single switchover design comparing average grade during 22 consecutive shifts of operation with the test collector with average grade during the preceding 22 consecutive shifts of operation with the control collector. The conclusion of the REFDIST analysis is that the Test-minus-Control difference in average grade must be at least about 5.2% before it is larger than most (95%) of the values in the reference distribution.



If instead of a single-switchover design a multiple-switchover design is used, the Test-minus-Control difference needed to stand out from routine variation will generally be smaller. The following figure illustrates the reference distribution for an alternative trial design of the same length (44 shifts) where switching from test to control or vice versa is done every shift. For this design, the Test-minus-Control difference in average grade need be only about 1.2% before it is larger than most (95%) of the values in the reference distribution.

For more information about the Cytec REFDIST P/C software program and how to use it, please consult your local Cytec representative.



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13 Safe handling, storage and use of Cytec reagents



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Section 13 Safe handling, storage, and use of Cytec's reagents

Introduction

Cytec has established a reputation as a safety and environmentally conscious manufacturer of mining chemicals. The number one priority is that our customers have and use all the information provided in this section regarding the recommended safe procedures for handling, storage and feeding of Cytec's products.

In this section you will find information on the following:

1. Material Safety Data Sheets (MSDS)
   – where to obtain a copy
   – how to read and interpret.
2. Contact information for your local Cytec representative.
3. Cytec's safety consultants.
4. Materials of Construction for safe handling, storage and use of Cytec's reagents.
5. Emergency Response and Incident Management (ERIM) Policy.
6. Product Stewardship.
7. Safety Aspects of Product Packaging and Delivery.
8. Handling and use of experimental products (TSCA statement).

Section 13.1 Material safety data sheets

The objective of the MSDS is to concisely inform you about the hazards of the materials you work with, so that you can protect yourself and respond to emergency situations. The purpose of an MSDS is to tell you:

- The material's physical properties and health effects that may make it hazardous to handle.
- The type of protective clothing you need.
- The first-aid treatment to be provided when you are exposed to a hazard.
- The pre-planning needed for safely handling spills, fires, and day-to-day operations.
- How to respond to accidents.
- How to safely store the product.

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Cytec provides an MSDS for all of its products. You may obtain an updated copy by contacting your local representative, Cytec office, or by accessing the Cytec website at www.cytec.com on the Internet.

For an explanation of what an MSDS can tell you about a material you may obtain a copy of "The MSDS Pocket Dictionary" from Genium Publishing Corporation , One Genium Plaza, Schenectady, NY 12304-4690 – tel: 518-377-8854 / e-mail: sales@genium.com

Section 13.2 Contact information

Please refer to the end of the Handbook for locations of Cytec offices worldwide.

Section 13.3 Cytec safety consultants

Cytec has experts in the safety aspects of our chemicals and they are available for consultation. Contact your local representative or a Cytec office.

Section 13.4 Materials of construction compatibility

Most of Cytec's products are compatible with stainless steel, mild steel, cast iron, high-density polyethylene, high-density polypropylene, PT FE materials and phenolic or epoxy thermosetting materials. Do not use copper, brass, aluminum, rubber, PVC or Tygon tubing in feed or storage system. For more details of a specific product, consult the product data sheet.

Section 13.5 Emergency response & incident management (ERIM) policy

We at Cytec are committed to protecting the public safety and environment. In the event our products or materials are involved in an incident, a timely and effective response will be made.

Our objectives are:

- First, and foremost, to help protect the public safety and environment by prevention of transportation incidents.
- To provide an appropriate response in the event of an incident involving one of our products or materials.
- To comply with all appropriate government regulations.
- To work to improve the safe practices and procedures of shippers, transporters, and receivers as they relate to the handling of Cytec's products and materials

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- To address public concerns about chemical transportation hazards by continuing education programs and communication with the public and designated public emergency response agencies.

For an updated brochure please contact your local representative or a Cytec office and refer to brochure # CGL-146

Section 13.6 Product stewardship

Cytec Industries is concerned about the health and well being of our customers, employees, and the community. Cytec is committed to reviewing and improving upon its manufacturing processes and products to minimize any adverse safety, health and environmental impacts. In accordance with this commitment, Cytec will strive to:

- Design safe, energy-efficient, and environmentally sound products and processes.
- Transport products safely in packaging which conserves resources and meets customers' needs.
- Bring value to its customers and shareholders by continually improving its products and processes.
- Enhance partnerships with its customers, suppliers, and the community to fulfill these responsibilities.

Product Stewardship is the responsible and ethical management of the health, safety and environmental aspects of a product from its inception through production to its ultimate use and disposition. Product Stewardship is part of the Responsible Care® Initiative of the American Chemistry Council (ACC) of which Cytec is a charter member. For our brochure on PRODUCT STEWARDSHIP, please contact your local representative or a Cytec office and request brochure # CGL-188.

Section 13.7 Safety aspects of product packaging and delivery

Products from Cytec are available in steel or plastic drums, totes, and in bulk tank trucks or tank cars. Contact your local Cytec representative or a Cytec office on advice for a suitable package for your application.

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Section 13.8 Safe handling of research samples

Cytec is constantly investigating and developing new products for the mining industry. Such materials are available free of charge in 50ml to 1L quantities for investigative purposes only. Since these products are at various stages of development, and are not commercially available, MSDSs may or may not exist.

Cytec's policy is to provide to the researcher or testing lab requesting such a sample, sufficient information to handle, use, and store the material safely. Typically, literature will accompany the sample indicating pertinent hazard information about the product such as flammability, skin contact, and the correct storage conditions, along with other helpful physical properties. At various times, an MSDS of a commercial product similar to the experimental sample will be sent, delineating the most likely hazard and storage information. In either case, all research samples will be labeled as shown below to indicate they are for investigative use only and must be handled safely by technically qualified personnel.

RESEARCH SAMPLE – FOR INVESTIGATIONAL USE ONLY

Important! The chemical and toxicological properties of this material have not been fully investigated. Its handling or use may be hazardous. Exercise due care. Since this material may contain chemicals not included in the Toxic Substance Control Act Inventory, it must be used under the supervision of technically qualified individuals. Materials not included in the Toxic Substances Control Act must not be used for commercial purposes.

Please contact your local Cytec representative for sample requests.

References

1. Bretherick, L, 1999. *Bretherick's Handbook of Reactive Chemical Hazards:* An Indexed Guide to Published Data, 6th. ed., Butterworth-Heineman, Oxford; Boston
2. Lewis, R. J. Sr., *2000. Sax's Dangerous Properties of Industrial Materials,* 10th. ed., Wiley, New York.

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14 Tables

Specific Gravities of pulps containing solids of the following different specific grades

Weight percent solids	Weight ratio of solids to solution	2.50 sp gr	2.70 sp gr	2.90 sp gr	3.10 sp gr	3.30 sp gr	3.50 sp gr	3.80 sp gr	4.20 sp gr	4.60 sp gr	5.00 sp gr
38	1: 1.6				1.546	1.360	1.373	1.389	1.408	1.423	1.437
39	1: 1.5		1.326	1.343	1.358	1.373	1.386	1.403	1.423	1.439	1.453
40	1: 1.5		1.336	1.355	1.371	1.3					
41	1: 1.459	1.326	1.348	1.367	1.384	1.4					
42	1: 1.381	1.337	1.359	1.380	1.396	1.4					
43	1: 1.3		1.371	1.392	1.411	1.4					
44	1: 1.273	1.359	1.383	1.405	1.425	1.4					
45											
46											
47											
48											
49											
50											
51											
52											
53	1: 0.887	1.466	1.501	1.532	1.560	1.585	1.609	1.641	1.677	1.709	1.736
54	1: 0.852	1.479	1.515	1.548	1.577	1.603	1.628	1.661	1.699	1.732	1.761
55	1: 0.818	1.493	1.530	1.564	1.595	1.621	1.647	1.681	1.721	1.756	1.786
56	1: 0.786	1.506	1.545	1.580	1.611	1.640	1.667	1.703	1.744	1.780	1.812
57	1: 0.754	1.520	1.560	1.596	1.628	1.659	1.687	1.724	1.767	1.804	1.838
58	1: 0.724	1.534	1.574	1.613	1.646	1.678	1.707	1.746	1.791	1.831	1.866
59	1: 0.695	1.548	1.591	1.629	1.665	1.697	1.728	1.769	1.815	1.858	1.894
60	1: 0.667	1.563	1.607	1.645	1.683	1.718	1.750	1.792	1.840	1.885	1.923
61	1: 0.639	1.577	1.623	1.662	1.702	1.740	1.772	1.816	1.866	1.913	1.953
62	1: 0.613	1.595	1.641	1.682	1.724	1.764	1.795	1.841	1.894	1.943	1.985
63	1: 0.587	1.613	1.660	1.703	1.747	1.789	1.818	1.867	1.923	1.974	2.018
64	1: 0.563	1.622	1.675	1.724	1.771	1.815	1.842	1.894	1.953	2.006	2.052
65	1: 0.538	1.639	1.692	1.743	1.796	1.842	1.867	1.921	1.981	2.035	2.083
66		1.656	1.710	1.762	1.815	1.868	1.893	1.948	2.011	2.068	2.119
67	1: 0.493	1.672	1.730	1.783	1.831	1.876	1.918	1.975	2.043	2.102	2.155
68	1: 0.471	1.689	1.749	1.803	1.854	1.901	1.944	2.004	2.075	2.138	2.193
69	1: 0.449	1.706	1.768	1.825	1.878	1.927	1.972	2.034	2.108	2.174	2.232

2 ← atomic number

He

4.00260 ← atomic weight

ELEMENTS

					3A	4A	5A	6A	7A
					5 B 10.81	6 C 12.011	7 N 14.0067	8 O 15.9994	9 F 18.998403
					13 Al 26.98154	14 Si 28.0855	15 P 30.97376	16 S 32.06	17 Cl 35.453
8B			1B	2B					
26 Fe 55.847	27 Co 58.9332	28 Ni 58.69	29 Cu 63.546	30 Zn 65.38	31 Ga 69.72	32 Ge 72.59	33 As 74.9216	34 Se 78.96	35 Br 79.904
44 Ru 101.07	45 Rh 102.9055	46 Pd 106.42	47 Ag 107.868	48 Cd 112.41	49 In 114.82	50 Sn 118.69	51 Sb 121.75	52 Te 127.60	53 I 126.9045
76 Os 190.2	77 Ir 192.22	78 Pt 195.08	79 Au 196.9665	80 Hg 200.59	81 Tl 204.383	82 Pb 207.2	83 Bi 208.9804	84 Po (209)	85 At (210)

Gas constants (R)

R = 0.0821	(atm.) (liter)/(g-mole) (°K)
R = 1.987	cal./(g-mole) (°K)
R = 1.987	Btu/(lb.-mole) (°R)
R = 1.987	chu/(lb.-mole) (°K)
R = 8.314	joules/(g-mole) (°K)
R = 1.546	(ft.-lb. force)/(lb.-mole) (°R)
R = 10.73	(lb.-force/sq. in.) (cu. ft.)/(lb.-
R = 18510	(lb.-force/sq. in.) (cu. in.)/(lb.-
R = 0.7302	(atm.) (cu. ft.)/(lb.-mole) (°R)
R = 848 x 105	(Kg./m2) (cu. cm.)/(lb.-mole) (°

Acceleration of gravity (standard)

g = 32.17 ft./sec.2 = 980.6 cm./sec.2

Table 14-1 Comparison of U.S., Tyler, Canadian, British, French, and German standard sieve series

U.S. (1)		Tyler (2)	Canadian (3)	
Standard	Alternate	Mesh designation	Standard	Alternate
107.6 mm	4.24"			
101.6 mm	4"			
90.5 mm	3-1/2"			
76.1 mm	3"			
64.0 mm	2-1/2"			
53.8 mm	2.12"			
50.8 mm	2"			
45.3 mm	1-3/4"			
38.1 mm	1-1/2"			
32.0 mm	1-1/4"			
26.9 mm	1.06"	1.05"	26.9 mm	1.06"
25.4 mm	1"			
*22.6 mm	7/8"	.883"	22.6 mm	7/8"
19.0 mm	3/4"	.742"	19.0 mm	3/4"
*16.0 mm	5/8"	.624"	16.0 mm	5/8″
13.5 mm	.530"	.525"	13.5 mm	.530″
12.7 mm	1/2"			
*11.2 mm	7/16"	.441"	11.2 mm	7/16″
9.51 mm	3/8"	.371"	9.51 mm	3/8″
*8.00 mm	5/16"	2-1/2	8.00 mm	5/16″
6.73 mm	.265"	3	6.73 mm	.265″
6.35 mm	1/4			
*5.66 mm	No. 3-1/2	3-1/2	5.66 mm	No. 3-1/2
4.76 mm	4	4	4.76 mm	4
*4.00 mm	5	5	4.00 mm	5
3.36 mm	6	6	3.36 mm	6
*2.83 mm	7	7	2.83 mm	7
2.38 mm	8	8	2.38 mm	8
*2.00 mm	10	9	2.00 mm	10
1.68 mm	12	10	1.68 mm	12

(1) U.S. Sieve Series – ASTM Specification E-11-61.
(2) Tyler Standard Screen Scale Sieve Series.
(3) Canadian Standard Sieve Series 8-GP-1b.
(4) British Standards Institution, London BS-410-62.
(5) French Standard Specifications, AFNOR X-11-501.
(6) German Standard Specification DIN 4188.

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British (4)		French (5)		German (6)
Nominal aperture	Nominal mesh number	Opening (mm)	Number	Opening
				25.0 mm 20.0 mm
				18.0 mm 16.0 mm 12.5 mm
				10.0 mm 8.0 mm 6.3 mm
3.35 mm	5	5.000 4.000	38 37	5.0 mm 4.0 mm
2.80 mm 2.40 mm 2.00 mm 1.68 mm	6 7 8 10	3.150 2.500 2.000 1.600	36 35 34 33	3.15 mm 2.5 mm 2.0 mm 1.6 mm

*These sieves correspond to those proposed as an International (ISO) Standard. It is recommended that wherever possible these sieves be included in all sieve analysis data or reports intended for international publication.

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Table 14-1 Comparison of U.S., Tyler, Canadian, British, French, and German standard sieve series (continued)

U.S. (1)		Tyler (2)	Canadian (3)	
Standard	Alternate	Mesh designation	Standard	Alternate
*1.41 mm	14	12	1.41 mm	14
1.19 mm	16	14	1.19 mm	16
*1.00 mm	18	16	1.00 mm	18
841 micron	20	20	841 micron	20
*707 micron	25	24	707 micron	25
595 micron	30	28	595 micron	30
*500 micron	35	32	500 micron	35
420 micron	40	35	420 micron	40
*354 micron	45	42	354 micron	45
297 micron	50	48	297 micron	50
*250 micron	60	60	250 micron	60
210 micron	70	65	210 micron	70
*177 micron	80	80	177 micron	80
149 micron	100	100	149 micron	100
*125 micron	120	115	125 micron	120
105 micron	140	150	105 micron	140
*88 micron	170	170	88 micron	170
74 micron	200	200	74 micron	200
*63 micron	230	250	63 micron	230
53 micron	270	270	53 micron	270
*44 micron	325	325	44 micron	325
37 micron	400	400	37 micron	400

(1) U.S. Sieve Series – ASTM Specification E-11-61.
(2) Tyler Standard Screen Scale Sieve Series.
(3) Canadian Standard Sieve Series 8-GP-1b.
(4) British Standards Institution, London BS-410-62.
(5) French Standard Specifications, AFNOR X-11-501.
(6) German Standard Specification DIN 4188.

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British (4)		French (5)		German (6)
Nominal aperture	Nominal mesh number	Opening (mm)	Number	Opening
1.40 mm	12			
		1.250	32	1.25 mm
1.20 mm	14			
1.00 mm	16	1.000	31	1.0 mm
850 micron	18			
		.800	30	800 micron
710 micron	22			
		.630	29	630 micron
600 micron	25			
500 micron	30	.500	28	500 micron
420 micron	36			
		.400	27	400 micron
355 micron	44			
		.315	26	315 micron
300 micron	52			
250 micron	60	.250	25	250 micron
210 micron	72		24	
		.200		200 micron
180 micron	85		23	
		.160		160 micron
150 micron	100			
125 micron	120	.125	22	125 micron
105 micron	150			
		.100	21	100 micron
90 micron	170			90 micron
		.080	20	80 micron
75 micron	200			
				71 micron
63 micron	240	.063	19	63 micron
				56 micron
53 micron	300			
		.050	18	50 micron
45 micron	350			45 micron
		.040	17	40 micron

*These sieves correspond to those proposed as an International (ISO) Standard. It is recommended that wherever possible these sieves be included in all sieve analysis data or reports intended for international publication.

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Table 14-2 Pulp Density Relations

Weight percent solids	Weight ratio of solids to solution	Specific Gravities of pulps containing solids of the following different specific grades									
		2.50 sp gr	2.70 sp gr	2.90 sp gr	3.10 sp gr	3.30 sp gr	3.50 sp gr	3.80 sp gr	4.20 sp gr	4.60 sp gr	5.00 sp gr
5	1:19.000	1.031	1.032	1.034	1.035	1.036	1.037	1.038	1.040	1.041	1.042
6	1:15.667	1.037	1.039	1.041	1.042	1.043	1.045	1.046	1.048	1.049	1.050
7	1:13.286	1.044	1.046	1.048	1.049	1.051	1.053	1.054	1.056	1.058	1.059
8	1:11.500	1.050	1.053	1.055	1.057	1.059	1.061	1.063	1.065	1.067	1.068
9	1:10.111	1.057	1.060	1.063	1.065	1.067	1.069	1.071	1.074	1.076	1.078
10	1: 9.000	1.064	1.067	1.070	1.072	1.075	1.077	1.080	1.082	1.085	1.087
11	1: 8.091	1.071	1.074	1.078	1.080	1.083	1.085	1.088	1.091	1.094	1.096
12	1: 7.333	1.078	1.082	1.085	1.088	1.091	1.094	1.097	1.101	1.104	1.106
13	1: 6.692	1.085	1.089	1.093	1.096	1.099	1.102	1.106	1.110	1.113	1.116
14	1: 6.144	1.092	1.097	1.101	1.105	1.108	1.111	1.115	1.119	1.123	1.126
15	1: 5.667	1.099	1.104	1.109	1.113	1.117	1.120	1.124	1.129	1.133	1.136
16	1: 5.250	1.106	1.112	1.117	1.122	1.125	1.129	1.134	1.139	1.143	1.147
17	1: 4.882	1.114	1.119	1.125	1.130	1.134	1.138	1.143	1.149	1.153	1.157
18	1: 4.556	1.121	1.128	1.134	1.139	1.143	1.148	1.153	1.159	1.164	1.168
19	1: 4.263	1.129	1.136	1.142	1.148	1.153	1.157	1.163	1.169	1.175	1.179
20	1: 4.000	1.136	1.144	1.151	1.157	1.162	1.167	1.173	1.180	1.186	1.190
21	1: 3.762	1.144	1.152	1.159	1.166	1.171	1.176	1.183	1.190	1.197	1.202
22	1: 3.545	1.152	1.161	1.168	1.175	1.181	1.186	1.193	1.201	1.208	1.214
23	1: 3.348	1.160	1.169	1.177	1.184	1.191	1.197	1.204	1.212	1.220	1.225
24	1: 3.167	1.168	1.178	1.186	1.194	1.201	1.207	1.215	1.224	1.231	1.238
25	1: 3.000	1.176	1.187	1.195	1.204	1.211	1.217	1.226	1.235	1.243	1.250
26	1: 2.846	1.185	1.195	1.205	1.214	1.222	1.228	1.237	1.247	1.255	1.263
27	1: 2.704	1.193	1.205	1.215	1.224	1.232	1.239	1.248	1.259	1.268	1.279
28	1: 2.571	1.202	1.214	1.224	1.234	1.242	1.250	1.260	1.271	1.281	1.289
29	1; 2.448	1.211	1.223	1.234	1.244	1.253	1.261	1.272	1.284	1.294	1.302
30	1: 2.333	1.220	1.233	1.244	1.255	1.264	1.273	1.284	1.296	1.307	1.316
31	1: 2.226	1.229	1.242	1.255	1.266	1.275	1.284	1.296	1.309	1.320	1.330
32	1: 2.125	1.238	1.252	1.265	1.277	1.287	1.296	1.309	1.322	1.334	1.344
33	1: 2.030	1.247	1.262	1.276	1.288	1.299	1.308	1.321	1.336	1.348	1.359
34	1: 1.941	1.256	1.272	1.287	1.299	1.311	1.321	1.334	1.350	1.363	1.374
35	1: 1.857	1.266	1.283	1.298	1.310	1.323	1.333	1.348	1.364	1.377	1.389
36	1: 1.778	1.276	1.293	1.309	1.322	1.335	1.346	1.361	1.378	1.392	1.404
37	1: 1.703	1.285	1.304	1.320	1.334	1.347	1.359	1.375	1.393	1.408	1.420

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Weight percent solids	Weight ratio of solids to solution	Specific Gravities of pulps containing solids of the following different specific grades									
		2.50 sp gr	2.70 sp gr	2.90 sp gr	3.10 sp gr	3.30 sp gr	3.50 sp gr	3.80 sp gr	4.20 sp gr	4.60 sp gr	5.00 sp gr
38	1: 1.632	1.295	1.314	1.332	1.346	1.360	1.373	1.389	1.408	1.423	1.437
39	1: 1.564	1.305	1.326	1.343	1.358	1.373	1.386	1.403	1.423	1.439	1.453
40	1: 1.500	1.316	1.336	1.355	1.371	1.387	1.400	1.418	1.438	1.456	1.471
41	1: 1.439	1.326	1.348	1.367	1.384	1.400	1.414	1.433	1.454	1.472	1.488
42	1: 1.381	1.337	1.359	1.380	1.396	1.414	1.429	1.448	1.471	1.490	1.506
43	1: 1.326	1.348	1.371	1.392	1.411	1.428	1.443	1.464	1.487	1.507	1.524
44	1: 1.273	1.359	1.383	1.405	1.425	1.442	1.458	1.480	1.504	1.525	1.543
45	1: 1.222	1.370	1.395	1.418	1.438	1.456	1.474	1.496	1.522	1.544	1.563
46	1: 1.174	1.381	1.408	1.432	1.452	1.471	1.489	1.513	1.540	1.563	1.582
47	1: 1.128	1.393	1.420	1.445	1.467	1.487	1.505	1.530	1.558	1.582	1.603
48	1: 1.083	1.404	1.433	1.458	1.483	1.503	1.522	1.547	1.577	1.602	1.623
49	1: 1.041	1.416	1.446	1.473	1.497	1.519	1.538	1.565	1.596	1.622	1.645
50	1: 1.000	1.429	1.460	1.487	1.512	1.535	1.556	1.583	1.615	1.643	1.667
51	1: 0.961	1.441	1.473	1.502	1.528	1.551	1.573	1.602	1.636	1.664	1.689
52	1: 0.923	1.453	1.487	1.517	1.544	1.568	1.591	1.621	1.656	1.686	1.712
53	1: 0.887	1.466	1.501	1.532	1.560	1.585	1.609	1.641	1.677	1.709	1.736
54	1: 0.852	1.479	1.515	1.548	1.577	1.603	1.628	1.661	1.699	1.732	1.761
55	1: 0.818	1.493	1.530	1.564	1.594	1.621	1.647	1.681	1.721	1.756	1.786
56	1: 0.786	1.506	1.545	1.580	1.611	1.640	1.667	1.703	1.744	1.780	1.812
57	1: 0.754	1.520	1.560	1.596	1.628	1.659	1.687	1.724	1.768	1.805	1.838
58	1: 0.724	1.534	1.574	1.613	1.646	1.678	1.707	1.746	1.792	1.831	1.866
59	1: 0.695	1.548	1.591	1.629	1.665	1.697	1.728	1.769	1.817	1.858	1.894
60	1: 0.667	1.563	1.607	1.645	1.684	1.718	1.750	1.792	1.842	1.885	1.923
61	1: 0.639	1.577	1.623	1.664	1.704	1.739	1.772	1.816	1.868	1.913	1.953
62	1: 0.613	1.592	1.641	1.683	1.724	1.761	1.795	1.841	1.895	1.943	1.984
63	1: 0.587	1.608	1.657	1.703	1.745	1.783	1.818	1.866	1.923	1.973	2.016
64	1: 0.563	1.623	1.675	1.723	1.765	1.805	1.842	1.892	1.952	2.003	2.049
65	1: 0.538	1.639	1.692	1.742	1.786	1.828	1.867	1.919	1.981	2.035	2.083
66	1: 0.515	1.656	1.711	1.762	1.808	1.852	1.892	1.947	2.011	2.068	2.119
67	1: 0.493	1.672	1.730	1.783	1.831	1.876	1.918	1.975	2.043	2.102	2.155
68	1: 0.471	1.689	1.749	1.803	1.854	1.901	1.944	2.004	2.075	2.138	2.193
69	1: 0.449	1.706	1.768	1.825	1.878	1.927	1.972	2.034	2.108	2.174	2.232
70	1: 0.429	1.724	1.786	1.847	1.902	1.954	2.000	2.065	2.143	2.212	2.273

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Conversion factors

	To convert	Multiply by	To obtain
A	acres	1.60×10^{-2}	rods
	acres	$1. \times 10^{5}$	sq. links
	acres	4.047×10^{-1}	hectares or sq. hectometers
	acres	4.35×10^{4}	sq. ft.
	acres	4.047×10^{3}	sq. meters
	acres	1.562×10^{-3}	sq. miles
	acres	4.840×10^{3}	sq. yards
	acre-feet	4.356×10^{4}	cu. feet
	acre-feet	3.259×10^{5}	gallons
	angstrom unit	3.937×10^{-9}	inches
	angstrom unit	$1. \times 10^{-10}$	meters
	angstrom unit	$1. \times 10^{-4}$	microns or (mu)
	atmospheres	7.348×10^{-3}	tons/sq. in.
	atmospheres	1.058	tons/sq. foot
	atmospheres	7.6×10^{1}	cms. of mercury (at 0° C.)
	atmospheres	3.39×10^{1}	ft. of water (at 4° C.)
	atmospheres	2.992×10^{1}	in. of mercury (at 0° C.)
	atmospheres	7.6×10^{-1}	meters of mercury (at 0° C.)
	atmospheres	7.6×10^{2}	millimeters of mercury (at 0° C.)
	atmospheres	1.0333	kgs./sq. cm.
	atmospheres	1.0333×10^{4}	kgs./sq. meter
	atmospheres	1.47×10^{1}	pounds/sq. in.
B	barrels (u.s., dry)	3.281	bushels
	barrels (u.s., dry)	7.056×10^{3}	cu. inches
	barrels (u.s., dry)	1.05×10^{2}	quarts (dry)
	barrels (u.s., liquid)	3.15×10^{1}	gallons
	barrels (oil)	4.2×10^{1}	gallons (oil)
	btu	1.0409×10^{1}	liter-atmospheres
	btu	7.7816×10^{2}	foot-pounds
	btu	2.52×10^{2}	gram-calories
	btu	3.927×10^{-4}	horsepower-hours
	btu	1.055×10^{3}	joules
	btu	2.52×10^{-1}	kilogram-calories
	btu	1.0758×10^{2}	kilogram-meters
	btu	2.928×10^{-4}	kilowatt-hours
	btu/hr.	2.162×10^{-1}	ft.-pounds/sec.
	btu/hr.	7.0×10^{-2}	gram-cal./sec.
	btu/hr.	3.929×10^{-4}	horsepower
	btu/hr.	2.931×10^{-1}	watts
	btu/min.	1.296×10^{1}	ft.-pounds/sec.
	btu/min.	2.356×10^{-2}	horsepower
	btu/min.	1.757×10^{-2}	kilowatts
	btu/min.	1.757×10^{1}	watts
	btu/sq. ft./min.	1.22×10^{-1}	watts/sq. in.
	bucket (br. dry)	1.8184×10^{4}	cubic cm.
	bushels	1.2445	cubic ft.

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To convert	Multiply by	To obtain
bushels	2.1504×10^{3}	cubic in.
bushels	3.524×10^{-2}	cubic meters
bushels	3.524×10^{1}	liters
bushels	4.0	pecks
bushels	6.4×10^{1}	pints (dry)
bushels	3.2×10^{1}	quarts (dry)
calories, gram (mean)	3.9685×10^{-3}	btu (mean)
centigrade (degrees)	(°C. x 9/5) + 32	fahrenheit (degrees)
centigrade (degrees)	°C. + 273.18	kelvin (degrees)
centiliters	3.382×10^{-1}	ounce (fluid) u.s.
centiliters	6.103×10^{-1}	cubic in.
centiliters	$1. \times 10^{-2}$	liters
centimeters	3.281×10^{-2}	feet
centimeters	3.937×10^{-1}	inches
centimeters	$1. \times 10^{-5}$	kilometers
centimeters	$1. \times 10^{-2}$	meters
centimeters	6.214×10^{-6}	miles
centimeters	$1. \times 10^{1}$	millimeters
centimeters	1.094×10^{-2}	yards
centimeters	$1. \times 10^{4}$	microns
centimeters	$1. \times 10^{8}$	angstrom units
centimeters of mercury	1.316×10^{-2}	atmospheres
centimeters of mercury	4.461×10^{-1}	ft. of water
centimeters of mercury	1.36×10^{2}	kgs./sq. meter
centimeters of mercury	2.785×10^{1}	pounds/sq. ft.
centimeters of mercury	1.934×10^{-1}	pounds/sq. in.
centimeters/sec.	1.969	feet/min.
centimeters/sec.	3.281×10^{-2}	feet/sec.
centimeters/sec.	3.6×10^{-2}	kilometers/hr.
centimeters/sec.	1.943×10^{-2}	knots
centimeters/sec.	6.0×10^{-1}	meters/min.
centimeters/sec.	2.237×10^{-2}	miles/hr.
centimeters/sec.	3.728×10^{-4}	miles/min.
centimeters/sec./sec.	3.281×10^{-2}	ft./sec./sec.
centimeters/sec./sec.	3.6×10^{-2}	kms./hr./sec.
centimeters/sec./sec.	1.0×10^{-2}	meters/sec./sec.
centimeters/sec./sec.	2.237×10^{-2}	miles/hr./sec.
centipoise	1.0×10^{-2}	gr./cm.-sec.
centipoise	6.72×10^{-4}	pound/ft.-sec.
centipoise	2.4	pound/ft.-hr.
circumference	6.283	radians
cubic centimeters	3.531×10^{-5}	cubic ft.
cubic centimeters	6.102×10^{-2}	cubic in.
cubic centimeters	1.0×10^{-6}	cubic meters
cubic centimeters	1.308×10^{-6}	cubic yards
cubic centimeters	2.642×10^{-4}	gallons (u.s. liquid)
cubic centimeters	$1. \times 10^{-3}$	liters

(continued on next page)

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Conversion factors (continued)

To convert	Multiply by	To obtain
cubic centimeters	2.113×10^{-3}	pints (u.s. liquid)
cubic centimeters	1.057×10^{-3}	quarts (u.s. liquid)
cubic feet	8.036×10^{-1}	bushels (dry)
cubic feet	2.8320×10^{4}	cu. cms.
cubic feet	1.728×10^{3}	cu. inches
cubic feet	2.832×10^{-2}	cu. meters
cubic feet	3.704×10^{-2}	cu. yards
cubic feet	7.48052	gallons (u.s. liquid)
cubic feet	2.832×10^{1}	liters
cubic feet	5.984×10^{1}	pints (u.s. liquid)
cubic feet	2.992×10^{1}	quarts (u.s. liquid)
cubic feet/min.	4.72×10^{2}	cu. cms./sec.
cubic feet/min.	1.247×10^{-1}	gallons/sec.
cubic feet/min.	4.720×10^{-1}	liters/sec.
cubic feet/min.	6.243×10^{1}	pounds water/min.
cubic feet/sec.	6.46317×10^{-1}	million gals./day
cubic feet/sec.	4.48861×10^{2}	gallons/min.
cubic inches	1.639×10^{1}	cu. cms.
cubic inches	5.787×10^{-4}	cu. ft.
cubic inches	1.639×10^{-5}	cu. meters
cubic inches	2.143×10^{-5}	cu. yards
cubic inches	4.329×10^{-3}	gallons
cubic inches	1.639×10^{-2}	liters
cubic inches	3.463×10^{-2}	pints (u.s. liquid)
cubic inches	1.732×10^{-2}	quarts (u.s. liquid)
cubic meters	2.838×10^{1}	bushels (dry)
cubic meters	1.0×10^{6}	cu. cms.
cubic meters	3.531×10^{1}	cu. ft.
cubic meters	6.1023×10^{4}	cu. inches
cubic meters	1.308	cu. yards
cubic meters	2.642×10^{2}	gallons (u.s. liquid)
cubic meters	1.0×10^{3}	liters
cubic meters	2.113×10^{3}	pints (u.s. liquid)
cubic meters	1.057×10^{3}	quarts (u.s. liquid)
cubic yards	7.646×10^{5}	cu. cms.
cubic yards	2.7×10^{1}	cu. ft.
cubic yards	4.6656×10^{4}	cu. inches
cubic yards	7.646×10^{-1}	cu. meters
cubic yards	2.02×10^{2}	gallons (u.s. liquid)
cubic yards	7.646×10^{2}	liters
cubic yards	1.6159×10^{3}	pints (u.s. liquid)
cubic yards	8.079×10^{2}	quarts (u.s. liquid)
cubic yards/min.	4.5×10^{-1}	cubic ft./sec.
cubic yards/min.	3.367	gallons/sec.
cubic yards/min.	1.274×10^{1}	liters/sec.
days	8.64×10^{4}	seconds
days	1.44×10^{3}	minutes

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To convert	Multiply by	To obtain
days	2.4×10^{1}	hours
decigrams	1.0×10^{-1}	grams
deciliters	1.0×10^{-1}	liters
decimeters	1.0×10^{-1}	meters
degrees (angle)	1.111×10^{-2}	quadrants
degrees (angle)	1.745×10^{-2}	radians
degrees (angle)	3.6×10^{3}	seconds
degrees/sec.	1.745×10^{-2}	radians/sec.
degrees/sec.	1.667×10^{-1}	revolutions/min.
degrees/sec.	2.778×10^{-3}	revolutions/sec.
F		
fathoms	1.8288	meters
fathoms	6.0	feet
feet	3.048×10^{1}	centimeters
feet	3.048×10^{-4}	kilometers
feet	3.048×10^{-1}	meters
feet	1.645×10^{-4}	miles (naut.)
feet	1.894×10^{-4}	miles (stat.)
feet	3.048×10^{2}	millimeters
feet of water	2.95×10^{-2}	atmospheres
feet of water	8.826×10^{-1}	in. of mercury
feet of water	3.048×10^{-2}	kgs./sq. cm.
feet of water	3.048×10^{2}	kgs./sq. meter
feet of water	6.243×10^{1}	pounds/sq. ft.
feet of water	4.335×10^{-1}	pounds/sq. in.
feet/min.	5.080×10^{-1}	cms./sec.
feet/min.	1.667×10^{-2}	feet./sec.
feet/min.	1.829×10^{-2}	kms./hr.
feet/min.	3.048×10^{-1}	meters/min.
feet/min.	1.136×10^{-2}	miles/hr.
feet/sec.	3.048×10^{2}	cms./sec.
feet/sec.	1.097	kms./hr.
feet/sec.	5.921×10^{-1}	knots
feet/sec.	1.829×10^{1}	meters/min.
feet/sec.	6.818×10^{-1}	miles/hr.
feet/sec.	1.136×10^{-2}	miles/min.
feet/sec./sec.	3.048×10^{1}	cms./sec./sec.
feet/sec./sec.	1.097	kms./hr./ sec.
feet/sec./sec.	3.048×10^{-1}	meters/sec./sec.
feet/sec./sec.	6.818×10^{-1}	miles/hr./sec.
feet/100 feet	1.0	per cent grade
foot-pounds	1.286×10^{-3}	btu
foot-pounds	3.241×10^{-1}	gram-calories
foot-pounds	5.050×10^{-7}	horsepower-hrs.
foot-pounds	1.356	joules
foot-pounds	3.241×10^{-4}	kg.-calories
foot-pounds	1.383×10^{-1}	kg.-meters
foot-pounds	3.766×10^{-7}	kilowatt-hrs

(continued on next page)

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Conversion factors (continued)

To convert	Multiply by	To obtain
foot-pounds/min.	1.286×10^{-3}	btu/min.
foot-pounds/min.	1.667×10^{-2}	foot-pounds/sec.
foot-pounds/min.	3.030×10^{-5}	horsepower
foot-pounds/min.	3.241×10^{-4}	kg.-calories/min.
foot-pounds/min.	2.260×10^{-5}	kilowatts
foot-pounds/sec.	4.6263	btu/hr.
foot-pounds/sec.	7.717×10^{-2}	btu/min.
foot-pounds/sec.	1.818×10^{-3}	horsepower
foot-pounds/sec.	1.945×10^{-2}	kg.-calories/min.
foot-pounds/sec.	1.356×10^{-3}	kilowatts
furlongs	1.25×10^{-1}	miles (u.s.)
furlongs	4.0×10^{1}	rods
furlongs	6.6×10^{2}	feet
furlongs	2.0117×10^{2}	meters
gallons	3.785×10^{3}	cu. cms.
gallons	1.337×10^{-1}	cu. feet
gallons	2.31×10^{2}	cu. inches
gallons	3.785×10^{-3}	cu. meters
gallons	4.951×10^{-3}	cu. yards
gallons	3.785	liters
gallons (liq. br. imp.)	1.20095	gallons (u.s. liquid)
gallons (u.s.)	8.3267×10^{-1}	gallons (imp.)
gallons of water	8.337	pounds of water
gallons/min.	2.228×10^{-3}	cu. feet/sec.
gallons/min.	6.308×10^{-2}	liters/sec.
gallons/min.	8.0208	cu. feet/hr.
grade	1.571×10^{-2}	radian
grains	3.657×10^{-2}	drams (avdp.)
grains (troy)	1.0	grains (avdp.)
grains (troy)	6.48×10^{-2}	grams
grains (troy)	2.0833×10^{-3}	ounces (avdp.)
grains (troy)	4.167×10^{-2}	pennyweight (troy)
grains/u.s. gallons	1.7118×10^{1}	parts/million
grains/u.s. gallons	1.4286×10^{2}	pounds/million gallons
grains/imp. gallons	1.4286×10^{1}	parts/million
grams	1.543×10^{1}	grains (troy)
grams	9.807×10^{-5}	joules/cm.
grams	1.0×10^{-3}	kilograms
grams	1.0×10^{3}	milligrams
grams	3.527×10^{-2}	ounces (avdp.)
grams	3.215×10^{-2}	ounces (troy)
grams	2.205×10^{-3}	pounds
grams/cm.	5.6×10^{-3}	pounds/in.
grams/cu. cm.	6.243×10^{1}	pounds/cu. ft.
grams/cu. cm.	3.613×10^{-2}	pounds/cu. in.
grams/liter	5.8417×10^{1}	grains/gal.
grams/liter	8.345	pounds/ 1,000 gal.

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To convert	Multiply by	To obtain
grams/liter	6.2427×10^{-2}	pounds/cu. ft.
grams/sq. cm.	2.0481	pounds/sq. ft.
gram-calories	3.9683×10^{-3}	btu
gram-calories	3.086	foot-pounds
gram-calories	1.5596×10^{-6}	horsepower-hrs.
gram-calories	1.162×10^{-6}	kilowatt-hrs.
gram-calories	1.162×10^{-3}	watt-hrs.
gram-calories/sec.	1.4286×10^{1}	btu/hr.
gram-centimeters	9.297×10^{-8}	btu
gram-centimeters	9.807×10^{-5}	joules
gram-centimeters	2.343×10^{-8}	kg.-calories
gram-centimeters	1.0×10^{-5}	kg.-meters
H		
hectares	2.471	acres
hectares	1.076×10^{5}	sq. feet
horsepower	4.244×10^{1}	btu/min.
horsepower	3.3×10^{4}	foot-lbs./min.
horsepower	5.50×10^{2}	foot-lbs./sec.
horsepower (metric)	9.863×10^{-1}	horsepower
horsepower	1.014	horsepower (metric)
horsepower	1.068×10^{1}	kg.-calories/min.
horsepower	7.457×10^{-1}	kilowatts
horsepower	7.457×10^{2}	watts
horsepower (boiler)	3.352×10^{4}	btu/hr.
horsepower (boiler)	9.803	kilowatts
horsepower-hours	2.547×10^{3}	btu
horsepower-hours	1.98×10^{6}	foot-lbs.
horsepower-hours	6.4119×10^{5}	gram-calories
horsepower-hours	2.684×10^{6}	joules
horsepower-hours	6.417×10^{2}	kg.-calories
horsepower-hours	2.737×10^{5}	kg.-meters
horsepower-hours	7.457×10^{-1}	kilowatt-hrs.
hours	4.167×10^{-2}	days
hours	5.952×10^{-3}	weeks
hours	3.6×10^{3}	seconds
hundredwgts (long)	1.12×10^{2}	pounds
hundredwgts (long)	5.0×10^{-2}	tons (long)
hundredwgts (long)	5.08023×10^{1}	kilograms
hundredwgts (short)	4.53592×10^{-2}	tons (metric)
hundredwgts (short)	4.46429×10^{-2}	tons (long)
hundredwgts (short)	4.53592×10^{1}	kilograms
I		
inches	2.540	centimeters
inches	2.540×10^{-2}	meters
inches	1.578×10^{-5}	miles
inches	2.54×10^{1}	millimeters
inches	2.778×10^{-2}	yards
inches	2.54×10^{8}	angstrom units

(continued on next page)

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Conversion factors (continued)

To convert	Multiply by	To obtain
inches	5.0505×10^{-3}	rods
inches of mercury	3.342×10^{-2}	atmospheres
inches of mercury	1.133	feet of water
inches of mercury	3.453×10^{-2}	kgs./sq. cm.
inches of mercury	3.453×10^{2}	kgs./sq. meter
inches of mercury	7.073×10^{1}	pounds/sq. ft.
inches of mercury	4.912×10^{-1}	pounds/sq. in.
in. of water (at 4° C)	2.458×10^{-3}	atmospheres
in. of water (at 4° C)	7.355×10^{-2}	inches of mercury
in. of water (at 4° C)	2.54×10^{-3}	kgs./sq. cm.
in. of water (at 4° C)	5.781×10^{-1}	ounces/sq. in.
in. of water (at 4° C)	5.204	pounds/sq. ft.
in. of water (at 4° C)	3.613×10^{-2}	pounds/sq. in.
J		
joules	9.486×10^{-4}	btu
K		
kilograms	1.0×10^{3}	grams
kilograms	9.807×10^{-2}	joules/cm.
kilograms	9.807	joules/meter (newtons)
kilograms	2.2046	pounds
kilograms	9.842×10^{-4}	tons (long)
kilograms	1.102×10^{-3}	tons (short)
kilograms	3.5274×10^{1}	ounces (avdp.)
kilograms/ cu. meter	1.0×10^{-3}	grams/cu. cm.
kilograms/cu. meter	6.243×10^{-2}	pounds/cu. ft.
kilograms/cu. meter	3.613×10^{-5}	pounds/cu. in.
kilograms/meter	6.72×10^{-1}	pounds/ft.
kilograms/sq. cm.	9.678×10^{-1}	atmospheres
kilograms/sq. cm.	3.281×10^{1}	feet of water
kilograms/sq. cm.	2.896×10^{1}	inches of mercury
kilograms/sq. cm.	2.048×10^{3}	pounds/sq. ft.
kilograms/sq. cm.	1.422×10^{1}	pounds/sq. in.
kilograms/sq. meter	9.678×10^{-5}	atmospheres
kilograms/sq. meter	3.281×10^{-3}	feet of water
kilograms/sq. meter	2.896×10^{-3}	inches of mercury
kilograms/sq. meter	2.048×10^{-1}	pounds/sq. ft.
kilograms/sq. meter	1.422×10^{-3}	pounds/sq. in.
kilograms/sq. mm.	1.0×10^{6}	kgs./sq. meter
kilogram-calories	3.968	btu
kilogram-calories	3.086×10^{3}	foot-pounds
kilogram-calories	1.558×10^{-3}	horsepower-hrs.
kilogram-calories	4.183×10^{3}	joules
kilogram-calories	4.269×10^{2}	kg.-meters
kilogram-calories	1.163×10^{-3}	kilowatt-hrs.
kilogram-calories/min.	5.143×10^{1}	ft.-lbs./sec.
kilogram-calories/min.	9.351×10^{-2}	horsepower
kilogram-calories/min.	6.972×10^{-2}	kilowatts
kilogram-meters	9.296×10^{-3}	btu

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To convert	Multiply by	To obtain
kilogram-meters	7.233	foot-pounds
kilogram-meters	9.807	joules
kilogram-meters	2.342×10^{-3}	kg.-calories
kilogram-meters	2.723×10^{-6}	kilowatt-hrs.
kilometers	1.0×10^{5}	centimeters
kilometers	3.281×10^{3}	feet
kilometers	3.937×10^{4}	inches
kilometers	1.0×10^{3}	meters
kilometers	6.214×10^{-1}	miles (statute)
kilometers	5.396×10^{-1}	miles (nautical)
kilometers	1.0×10^{6}	millimeters
kilometers	1.0936×10^{3}	yards
kilometers/hr.	2.778×10^{1}	cms./sec.
kilometers/hr.	5.468×10^{1}	feet/min.
kilometers/hr.	9.113×10^{-1}	feet/sec.
kilometers/hr.	5.396×10^{-1}	knots
kilometers/hr.	1.667×10^{1}	meters/min.
kilometers/hr.	6.214×10^{-1}	miles/hr.
kilometers/hr./sec.	2.778×10^{1}	cms./sec./sec.
kilometers/hr./sec.	9.113×10^{-1}	ft./sec./sec.
kilometers/hr./sec.	2.778×10^{-1}	meters/sec./sec.
kilometers/hr./sec.	6.214×10^{-1}	miles/hr./sec.
kilowatts	5.692×10^{1}	btu/min.
kilowatts	4.426×10^{4}	foot-lbs./min.
kilowatts	7.376×10^{2}	foot-lbs./sec.
kilowatts	1.341	horsepower
kilowatts	1.434×10^{1}	kg.-calories/min.
kilowatts	1.0×10^{3}	watts
kilowatt-hrs.	3.413×10^{3}	btu
kilowatt-hrs.	2.655×10^{6}	foot-lbs.
kilowatt-hrs.	8.5985×10^{5}	gram calories
kilowatt-hrs.	1.341	horsepower-hours
kilowatt-hrs.	3.6×10^{6}	joules
kilowatt-hrs.	8.605×10^{2}	kg.-calories
kilowatt-hrs.	3.671×10^{5}	kg.-meters
kilowatt-hrs.	3.53	pounds of water evaporated from and at 212° F.
kilowatt-hrs.	2.275×10^{1}	pounds of water raised from 62° to 212° F.
L		
liters	2.838×10^{-2}	bushels (u.s. dry)
liters	1.0×10^{3}	cu. cm.
liters	3.531×10^{-2}	cu. ft.
liters	6.102×10^{1}	cu. inches
liters	1.0×10^{-3}	cu. meters
liters	1.308×10^{-3}	cu. yards
liters	2.642×10^{-1}	gallons (u.s. liquid)
liters	2.113	pints (u.s. liquid)

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Conversion factors (continued)

To convert	Multiply by	To obtain
liters	1.057	quarts (u.s. liquid)
liters/min.	5.886×10^{-4}	cu. ft./sec.
liters/min.	4.403×10^{-3}	gals./sec.
$\log_{10}n$	2.303	In n
In n	4.343×10^{-1}	$\log_{10}n$
M		
meters	1.0×10^{10}	angstrom units
meters	1.0×10^{2}	centimeters
meters	5.4681×10^{-1}	fathoms
meters	3.281	feet
meters	3.937×10^{1}	inches
meters	1.0×10^{-3}	kilometers
meters	6.214×10^{-4}	miles (statute)
meters	1.0×10^{3}	millimeters
meters	1.094	yards
meters/min.	1.667	cms./sec.
meters/min.	3.281	feet/min.
meters/min	5.468×10^{-2}	feet/sec.
meters/min.	6.0×10^{-2}	kms./hr.
meters/min.	3.728×10^{-2}	miles/hr.
meters/sec.	1.968×10^{2}	feet/min.
meters/sec.	3.281	feet/sec.
meters/sec.	3.6	kilometers/hr.
meters/sec.	6.0×10^{-2}	kilometers/min.
meters/sec.	2.237	miles/hr.
meters/sec.	3.728×10^{-2}	miles/ min.
meters/sec./sec.	1.0×10^{2}	cms./sec./sec.
meters/sec./sec.	3.281	ft./sec./sec.
meters/sec./sec.	3.6	kms./hr./sec.
meters/sec./sec	2.237	miles/hr./sec.
meter-kilograms	7.233	pound-feet
microliters	1.0×10^{-6}	liters
micromicrons	1.0×10^{-12}	meters
microns	1.0×10^{-6}	meters
miles (statute)	1.609×10^{5}	centimeters
miles (statute)	5.280×10^{3}	feet
miles (statute)	6.336×10^{4}	inches
miles (statute)	1.609	kilometers
miles (statute)	1.609×10^{3}	meters
miles (statute)	8.684×10^{-1}	miles (nautical)
miles (statute)	1.760×10^{3}	yards
miles/hr.	4.470×10^{1}	cms./sec.
miles/hr.	8.8×10^{1}	ft./min.
miles/hr.	1.467	ft./sec.
miles/hr.	1.6093	kms./hr.
miles/hr.	2.682×10^{-2}	kms./min.
miles/hr.	2.682×10^{1}	meters/min.
miles/hr.	1.667×10^{-2}	miles/min.

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To convert	Multiply by	To obtain
miles/hr./sec.	4.47×10^{1}	cms./sec./sec.
miles/hr./sec.	1.467	ft./sec./sec.
miles/hr./sec.	1.6093	kms./hr./sec.
miles/hr./sec.	4.47×10^{-1}	meters/sec./sec.
miles/min.	2.682×10^{3}	cms./sec.
miles/min.	8.8×10^{1}	feet/sec.
miles/min.	1.6093	kms./min.
miles/min.	6.0×10^{1}	miles/hr.
milliers	1.0×10^{3}	kilograms
millimicrons	1.0×10^{-9}	meters
milligrams	1.5432×10^{-2}	grains
milligrams mil-	1.0×10^{-3}	grams
ligrams/liter	1.0	parts/million
milliliters	1.0×10^{-3}	liters
millimeters	1.0×10^{-1}	centimeters
millimeters	3.281×10^{-3}	feet
millimeters	3.937×10^{-2}	inches
millimeters	1.0×10^{-6}	kilometers
millimeters	1.0×10^{-3}	meters
millimeters	6.214×10^{-7}	miles
millimeters	1.094×10^{-3}	yards
million gals./day	1.54723	cu. ft./sec.
miner's inches	1.5	cu ft./min.
minutes (angles)	1.667×10^{-2}	degrees
minutes (angles)	1.852×10^{-4}	quadrants
minutes (angles)	2.909×10^{-4}	radians
minutes (angles)	6.0×10^{1}	seconds
minutes (time)	9.9206×10^{-5}	weeks
minutes (time)	6.944×10^{-4}	days
minutes (time)	1.667×10^{-2}	hours
minutes (time)	6.0×10^{1}	seconds
O		
ounces	8.0	drams
ounces	4.375×10^{2}	grains
ounces	2.8349×10^{1}	grams
ounces	6.25×10^{-2}	pounds
ounces	9.115×10^{-1}	ounces (troy)
ounces	2.790×10^{-5}	tons (long)
ounces	3.125×10^{-5}	tons (short)
ounces (fluid)	1.805	cu. inches
ounces (fluid)	2.957×10^{-2}	liters
ounces (troy)	4.80×10^{2}	grains
ounces (troy)	3.1103×10^{1}	grams
ounces (troy)	1.097	ounces (avdp.)
ounces (troy)	2.0×10^{1}	pennyweights (troy)
ounces (troy)	8.333×10^{-2}	pounds (troy)
ounce/sq. in.	6.25×10^{-2}	pounds/ sq. in.

(continued on next page)

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Conversion factors (continued)

P

To convert	Multiply by	To obtain
parts/million	5.84×10^{-2}	grains/u.s. gal.
parts/million	7.016×10^{-2}	grains/imp. gal.
parts/million	8.345	pounds/million gal.
pecks (british)	5.546×10^{2}	cubic inches
pecks (british)	9.0919	liters
pecks (u.s.)	2.5×10^{-1}	bushels
pecks (u.s.)	5.376×10^{2}	cubic inches
pecks (u.s.)	8.8096	liters
pecks (u.s.)	8	quarts (dry)
pennyweights (troy)	2.4×10^{1}	grains
pennyweights (troy)	5.0×10^{-2}	ounces (troy)
pennyweights (troy)	1.555	grams
pennyweights (troy)	4.1667×10^{-3}	pounds (troy)
pints (dry)	3.36×10^{1}	cubic inches
pints (dry)	1.5625×10^{-2}	bushels
pints (dry)	5.0×10^{-1}	quarts
pints (dry)	5.5059×10^{-1}	liters
pints (liquid)	4.732×10^{2}	cubic cms.
pints (liquid)	1.671×10^{-2}	cubic ft.
pints (liquid)	2.887×10^{1}	cubic inches
pints (liquid)	4.732×10^{-4}	cubic meters
pints (liquid)	6.189×10^{-4}	cubic yards
pints (liquid)	1.25×10^{-1}	gallons
pints (liquid)	4.732×10^{-1}	liters
pints (liquid)	5.0×10^{-1}	quarts (liquid)
poise	1.0	gram/cm.-sec.
pounds (avdp.)	1.4583×10^{1}	ounces (troy)
pounds	2.56×10^{2}	drams
pounds	7.0×10^{3}	grains
pounds	4.5359×10^{2}	grams
pounds	4.536×10^{-1}	kilograms
pounds	1.6×10^{1}	ounces
pounds	1.458×10^{1}	ounces (troy)
pounds	1.21528	pounds (troy)
pounds	5.0×10^{-4}	tons (short)
pounds (troy)	5.760×10^{3}	grains
pounds (troy)	3.7324×10^{2}	grams
pounds (troy)	1.3166×10^{1}	ounces (avdp.)
pounds (troy)	1.2×10^{1}	ounces (troy)
pounds (troy)	2.4×10^{2}	pennyweights (troy)
pounds (troy)	8.2286×10^{-1}	pounds (avdp.)
pounds (troy)	3.6735×10^{-4}	tons (long)
pounds (troy)	3.7324×10^{-4}	tons (metric)
pounds (troy)	4.1143×10^{-4}	tons (short)
pounds of water	1.602×10^{-2}	cu. ft.
pounds of water	2.768×10^{1}	cu. inches
pounds of water	1.198×10^{-1}	gallons
pounds of water/min.	2.670×10^{-4}	cu. ft./sec.

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	To convert	Multiply by	To obtain
	pound-feet	1.383×10^{-1}	meter-kgs.
	pounds/cu. ft.	1.602×10^{-2}	grams/cu. cm.
	pounds/cu. ft.	1.602×10^{1}	kgs./cu. meter
	pounds/ cu. ft.	5.787×10^{-4}	pounds/cu. inches
	pounds/cu. in.	2.768×10^{1}	grams/cu. cm.
	pounds/cu. in.	2.768×10^{4}	kgs./cu. meter
	pounds/ cu. in.	1.728×10^{3}	pounds/ cu. ft.
	pounds/ft.	1.488	kgs./meter
	pounds/in.	1.768×10^{2}	grams/cm.
	pounds/mil-foot	2.306×10^{6}	grams/cu. cm.
	pounds/sq. ft.	4.725×10^{-4}	atmospheres
	pounds/sq. ft.	1.602×10^{-2}	feet of water
	pounds/sq. ft.	1.414×10^{-2}	inches of mercury
	pounds/sq. ft.	4.882	kgs./sq. meter
	pounds/sq. ft.	6.944×10^{-3}	pounds/sq. inch
	pounds/sq. in.	6.804×10^{-2}	atmospheres
	pounds/sq. in.	2.307	feet of water
	pounds/sq. in.	2.036	inches of mercury
	pounds/sq. in.	7.031×10^{2}	kgs./sq. meter
	pounds/sq. in.	1.44×10^{2}	pounds/sq. ft.
	pounds/sq.in.	7.2×10^{-2}	short tons/sq. ft.
	pounds/sq. in.	7.03×10^{-2}	kgs./sq. cm.
Q	quadrants (angle)	9.0×10^{1}	degrees
	quadrants (angle)	5.4×10^{3}	minutes
	quadrants (angle)	1.571	radians
	quadrants (angle)	3.24×10^{5}	seconds
	quarts (dry)	6.72×10^{1}	cu. inches
	quarts (liquid)	9.464×10^{2}	cu. cms.
	quarts (liquid)	3.342×10^{-2}	cu. ft.
	quarts (liquid)	5.775×10^{1}	cu. inches
	quarts (liquid)	9.464×10^{-4}	cu. meters
	quarts (liquid)	1.238×10^{-3}	cu. yards
	quarts (liquid)	2.5×10^{-1}	gallons
	quarts (liquid)	9.463×10^{-1}	liters
R	radians	5.7296×10^{1}	degrees
	radians	3.438×10^{3}	minutes
	radians	6.366×10^{-1}	quadrants
	radians	2.063×10^{5}	seconds
	radians/sec.	5.7296×10^{1}	degrees/sec.
	radians/sec.	9.549	revolutions/min.
	radians/sec.	1.592×10^{-1}	revolutions/sec.
	radians/sec./sec.	5.7296×10^{2}	revs./min./min.
	radians/sec./sec.	9.549	revs./min./sec.
	radians/sec./sec.	1.592×10^{-1}	revs./sec./sec.
	revolutions	3.60×10^{2}	degrees
	revolutions	4.0	quadrants

(continued on next page)

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Conversion factors (continued)

To convert	Multiply by	To obtain
revolutions	6.283	radians
revolutions/min.	6.0	degrees/sec.
revolutions/min.	1.047×10^{-1}	radians/sec.
revolutions/min.	1.667×10^{-2}	revs./sec.
revs./min./min.	1.745×10^{-3}	radians/sec./sec.
revs./min./min.	1.667×10^{-2}	revs./min./sec.
revs./min./min.	2.778×10^{-4}	revs./sec./sec.
revolutions/sec.	3.6×10^{2}	degrees/sec.
revolutions/ sec.	6.283	radians/ sec.
revolutions/sec.	6.0×10^{1}	revs./min.
revs./sec./sec.	6.283	radians/sec./sec.
revs./sec./sec.	3.6×10^{3}	revs./min./min.
revs./sec./sec.	6.0×10^{1}	revs./min./sec.
rods	2.5×10^{-1}	chains (gunters)
rods	5.029	meters
rods (surveyors' meas.)	5.5	yards
rods	1.65×10^{1}	feet
rods	1.98×10^{2}	inches
rods	3.125×10^{-3}	miles
S		
seconds (angle)	2.778×10^{-4}	degrees
seconds (angle)	1.667×10^{-2}	minutes
seconds (angle)	3.087×10^{-6}	quadrants
seconds (angle)	4.848×10^{-6}	radians
square centimeters	1.076×10^{-3}	sq. feet
square centimeters	1.550×10^{-1}	sq. inches
square centimeters	1.0×10^{-4}	sq. meters
square centimeters	3.861×10^{-11}	sq. miles
square centimeters	1.0×10^{2}	sq. millimeters
square centimeters	1.196×10^{-4}	sq. yards
square feet	2.296×10^{-5}	acres
square feet	9.29×10^{2}	sq. cms.
square feet	1.44×10^{2}	sq. inches
square feet	9.29×10^{-2}	sq. meters
square feet	3.587×10^{-8}	sq. miles
square feet	9.29×10^{4}	sq. millimeters
square feet	1.111×10^{-1}	sq. yards
square inches	6.452	sq. cms.
square inches	6.944×10^{-3}	sq. ft.
square inches	6.452×10^{2}	sq. millimeters
square inches	7.716×10^{-4}	sq. yards
square kilometers	2.471×10^{2}	acres
square kilometers	1.0×10^{10}	sq. cms.
square kilometers	1.076×10^{7}	sq. ft.
square kilometers	1.550×10^{9}	sq. inches
square kilometers	1.0×10^{6}	sq. meters
square kilometers	3.861×10^{-1}	sq. miles
square kilometers	1.196×10^{6}	sq. yards

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	To convert	Multiply by	To obtain
	square meters	2.471×10^{-4}	acres
	square meters	1.0×10^{4}	sq. cms.
	square meters	1.076×10^{1}	sq. ft.
	square meters	1.55×10^{3}	sq. inches
	square meters	3.861×10^{-7}	sq. miles
	square meters	1.0×10^{6}	sq. millimeters
	square meters	1.196	sq. yards
	square miles	6.40×10^{2}	acres
	square miles	2.788×10^{7}	sq. ft.
	square miles	2.590	sq. kms.
	square miles	2.590×10^{6}	sq. meters
	square miles	3.098×10^{6}	sq. yards
	square millimeters	1.0×10^{-2}	sq. cms.
	square millimeters	1.076×10^{-5}	sq. ft.
	square millimeters	1.55×10^{-3}	sq. inches
	square yards	2.066×10^{-4}	acres
	square yards	8.361×10^{3}	sq. cms.
	square yards	9.0	sq. ft.
	square yards	1.296×10^{3}	sq. inches
	square yards	8.361×10^{-1}	sq. meters
	square yards	3.228×10^{-7}	sq. miles
	square yards	8.361×10^{5}	sq. millimeters
T	temperature (°C.) + 273	1.0	absolute temperature (°K.)
	temperature (°C.) + 17.78	1.8	temperature (°F.)
	temperature (°F.) + 460	1.0	absolute temperature (°R.)
	temperature (°F.) –32	5/9	temperature (°C.)
	tons (long)	1.016×10^{3}	kilograms
	tons (long)	2.24×10^{3}	pounds
	tons (long)	1.12	tons (short)
	tons (metric)	1.0×10^{3}	kilograms
	tons (metric)	2.205×10^{3}	pounds
	tons (short)	9.0718×10^{2}	kilograms
	tons (short)	3.2×10^{4}	ounces
	tons (short)	2.9166×10^{4}	ounces (troy)
	tons (short)	2.0×10^{3}	pounds
	tons (short)	2.43×10^{3}	pounds (troy)
	tons (short)	8.9287×10^{-1}	tons (long)
	tons (short)	9.078×10^{-1}	tons (metric)
	tons (short)/sq. it.	9.765×10^{3}	kgs./sq. meter
	tons (short)/sq. ft.	1.389×10^{1}	pounds/sq. in.
	tons (short)/sq. in.	1.406×10^{6}	kgs./sq. meter
	tons (short)/sq. in.	2.0×10^{3}	pounds/sq. in.
	tons of water/24 hrs.	8.333×10^{1}	pounds of water/hr.
	tons of water/24 hrs.	1.6643×10^{-1}	gallons/min.
	tons of water/24 hrs.	1.3349	cu. ft./hr.
W	watts	3.4129	btu/hr.

(continued on next page)

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Conversion factors (continued)

To convert	Multiply by	To obtain
watts	5.688×10^{-2}	btu/min.
watts	4.427×10^{1}	ft.-lbs./min.
watts	7.378×10^{-1}	ft.-lbs./sec.
watts	1.341×10^{-3}	horsepower
watts	1.36×10^{-3}	horsepower (metric)
watts	1.433×10^{-2}	kg.-calories/min.
watts	1.0×10^{-3}	kilowatts
watts (abs.)	1.0	joules/sec.
watt-hours	3.413	btu
watt-hours	2.656×10^{3}	foot-lbs.
watt-hours	8.605×10^{2}	gram-calories
watt-hours	1.341×10^{-3}	horsepower-hours
watt-hours	8.605×10^{-1}	kilogram-calories
watt-hours	3.672×10^{2}	kilogram-meters
watt-hours	1.0×10^{-3}	kilowatt-hours
weeks	1.68×10^{2}	hours
weeks	1.008×10^{4}	minutes
weeks	6.048×10^{5}	seconds
Y		
yards	9.144×10^{1}	centimeters
yards	9.144×10^{-4}	kilometers
yards	9.144×10^{-1}	meters
yards	4.934×10^{-4}	miles (nautical)
yards	5.682×10^{-4}	miles (statute)
yards	9.144×10^{2}	millimeters
years	3.65256×10^{2}	days (mean solar)
years	8.7661×10^{3}	hours (mean solar)

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Gas constants (R)

R = 0.0821	(atm.) (liter)/(g-mole) (°K)
R = 1.987	cal./(g-mole) (°K)
R = 1.987	Btu/(lb.-mole) (°R)
R = 1.987	chu/(lb.-mole) (°K)
R = 8.314	joules/(g-mole) (°K)
R = 1.546	(ft.-lb. force)/(lb.-mole) (°R)
R = 10.73	(lb.-force/sq. in.) (cu. ft.)/(lb.-mole) (°R)
R = 18510	(lb.-force/sq. in.) (cu. in.)/(lb.-mole) (°R)
R = 0.7302	(atm.) (cu. ft.)/(lb.-mole) (°R)
R = 8.48 x 105	(Kg./m2) (cu. cm.)/(lb.-mole) (°K)

Acceleration of gravity (standard)

g = 32.17 ft./sec.2 = 980.6 cm./sec.2

Velocity of sound in dry air @ 0°C and 1 atm.

33,136 cm./sec. = 1,089 ft./sec.

Heat of fusion of water

79.7 cal./g = 144 Btu/lb.

Heat of vaporization of water @ 1.0 atm.

540 cal./g = 970 Btu/lb

Specific heat of air

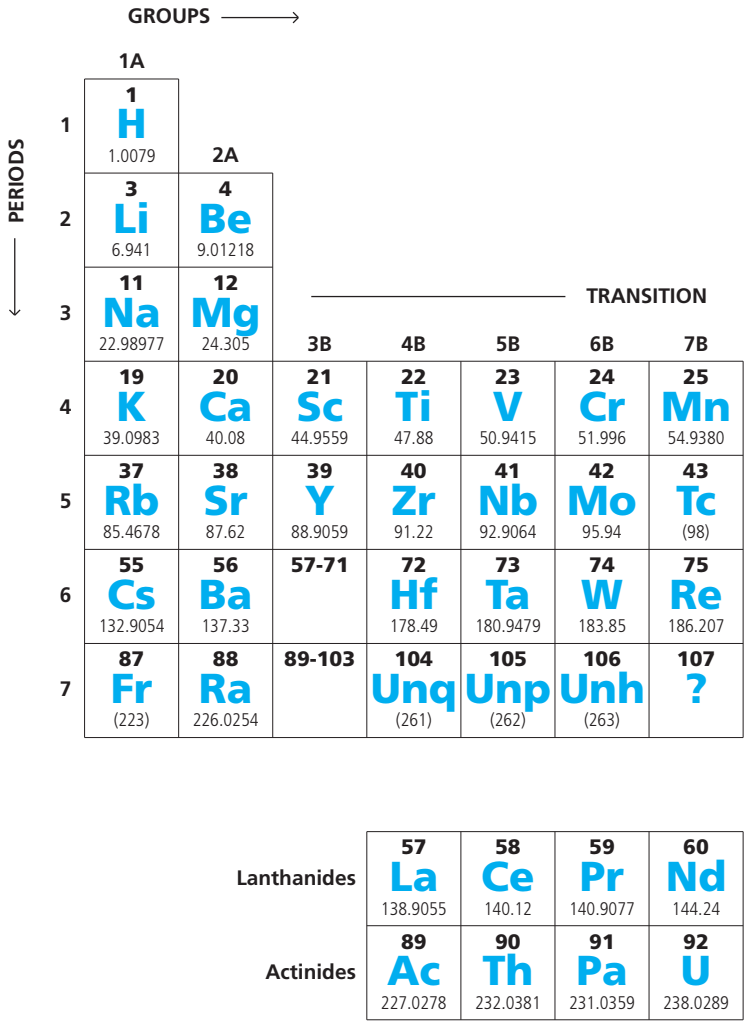
Cp = 0.238 cal./(g) (°C)

Density of dry air @ 0°C and 760 mm.

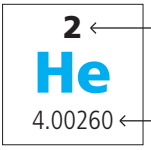
0.001293 g/cu. cm.

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Periodic Table of the elements



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ELEMENTS

					3A	4A	5A	6A	7A	8A
					5 B 10.81	6 C 12.011	7 N 14.0067	8 O 15.9994	9 F 18.998403	10 Ne 20.179
					13 Al 26.98154	14 Si 28.0855	15 P 30.97376	16 S 32.06	17 Cl 35.453	18 Ar 39.948
26 Fe 55.847	27 Co 58.9332	28 Ni 58.69	29 Cu 63.546	30 Zn 65.38	31 Ga 69.72	32 Ge 72.59	33 As 74.9216	34 Se 78.96	35 Br 79.904	36 Kr 83.80
44 Ru 101.07	45 Rh 102.9055	46 Pd 106.42	47 Ag 107.868	48 Cd 112.41	49 In 114.82	50 Sn 118.69	51 Sb 121.75	52 Te 127.60	53 I 126.9045	54 Xe 131.29
76 Os 190.2	77 Ir 192.22	78 Pt 195.08	79 Au 196.9665	80 Hg 200.59	81 Tl 204.383	82 Pb 207.2	83 Bi 208.9804	84 Po (209)	85 At (210)	86 Rn (222)

8B 1B 2B

The heavy line approximately separates the metallic elements (left of the line) from the non-metallic elements.

61 Pm (145)	62 Sm 150.36	63 Eu 151.96	64 Gd 157.25	65 Tb 158.9254	66 Dy 162.50	67 Ho 164.9304	68 Er 167.26	69 Tm 168.9342	70 Yb 173.14	71 Lu 174.967
93 Np 237.0482	94 Pu 244	95 Am (243)	96 Cm (247)	97 Bk (247)	98 Cf (251)	99 Es (252)	100 Fm (257)	101 Md (258)	102 No (259)	103 Lr (260)

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NOTES

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Technology ahead of its time™

Corporate Headquarters
Cytec Industries Inc.
Five Garret Mountain Plaza
West Paterson, NJ 07424 USA
Tel: (973) 357-3100
Product Referral: (973) 357-3193
Fax: (973) 357-3117

North American Regional Office
Cytec Industries Inc.
3259 E. Harbour Dr., Suite 100
Phoenix, AZ 85034 USA
Tel: (602) 470-1446
Fax: (602) 470-5030

South American Regional Office
Cytec Chile Ltda.
Los Orfebres 347, Parque Industrial, La Reina
Santiago – CHILE 7880032
Tel: (56) 2-275 0748
Fax: (56) 2-273 6423

Regional Office for Europe, Africa and the Middle East
Cytec Industries BV
Bowling Park Drive
Bradford BD4 7TT
West Yorkshire, ENGLAND
Tel: (44) 1274-733891
Fax: (44) 1274-724693

Australia, Baulkham Hills
Cytec Australia Holdings Pty Ltd.
Suite 1, Level 1, Norwest Quay
21 Solent Circuit
Baulkham Hills, NSW 2153
AUSTRALIA
Tel: (61) 2-9846-6200
Fax: (61) 2-9659-9776

Updates to this edition are ongoing. Contact us at Cytec.Mining@cytec.com

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