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Mine Waste Disposal Technology

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BUREAU OF MINES

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PREFACE

This Information Circular summarizes recent Bureau of Mines research results concerning mine waste disposal. These papers represent only a sample of the Bureau's efforts related to the embankment stability and environmental impacts associated with mine waste disposal, but they delineate several of the principal areas of this program. The seven technical presentations reproduced herein were given by Bureau and contractor personnel at the July 16, 1981, Technology Transfer Workshop on Mine Waste Disposal Technology in Denver, Colo.

Those desiring more information concerning the Bureau's Mine Waste Management program, Minerals Environmental Technology in general, or information on other specific research should feel free to contact the Bureau of Mines, Minerals Research Directorate, 2401 E Street, N.W., Washington, D.C. 20241.

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MINE WASTE DISPOSAL TECHNOLOGY

Proceedings: Bureau of Mines Technology Transfer Workshop, Denver, Colo. July 16, 1981

COMPILED BY STAFF—MINERALS RESEARCH

ABSTRACT

This Bureau of Mines publication consists of an overview of the mine waste management research currently being conducted by the Bureau. The following papers, given at a Technology Transfer Workshop, emphasize the increasing importance of research related to the safety and environmental considerations of mine waste disposal in recent years. Current work related to legislation passed in the last 10 years and their subsequent standards includes the development of adequate monitoring systems, development of stability and seepage prediction and control techniques, control of runoff water, and control of leaching solutions. Selected topics are included here that cover coal mine waste disposal, embankment slope stability monitoring, flow failures of mine tailings impoundments, and controlled burnout of abandoned coal mine fires. The projects described provide a current documentation of problems being addressed.

BUREAU OF MINES RESEARCH IN MINE WASTE DISPOSAL TECHNOLOGY

by

Roger A. Bloomfield[1] and Richard J. Seibel[2]

INTRODUCTION

Waste generated by the mining and milling industries now total about 2.3 billion tons annually. Most of this waste is characterized as overburden and tailings or refuse. Although solid wastes are common to all mining and milling operations, the technology for handling and disposing in each case may vary somewhat depending on the specific characteristics of the waste. For example, if the waste contains a high percentage of clay minerals, such as phosphate slimes, the equilibrium water content may approach 90 pct. If the waste contains sufficient pyrite, common to coal waste, the runoff and leachate can be expected to be acidic. Combustion is a problem sometimes encountered when coal waste is disposed of in a dry condition. Lead-zinc tailings generally average about 0.3 pct lead and 0.3 pct zinc which may be soluble in a low pH environment. Where a relatively coarse waste is produced in conjunction with an underground mine, backfilling is often considered advantageous. Research described herein is aimed at alleviating the structural, environmental, and economic problems associated with the disposal of all mining and milling wastes.

The need for research on solid mining wastes related to the structural stability of impoundments can be readily demonstrated by the following examples. In 1966, a coal waste pile in Aberfan, Wales, failed and devastated a school building leaving 144 children dead. In 1970, 16 million cubic feet of saturated tailings and slimes failed and flowed into the lower levels of the Mufalire mine in Africa, causing extensive damage and the loss of 89 lives. A coal refuse embankment at Buffalo Creek in West Virginia failed in 1972 sending 21 million cubic feet of water and sludge downstream, leaving over 100 fatalities, 1,100 injuries, 1,500 houses demolished, and 4,000 people homeless. Several tailings embankments have failed in the western United States over the past decade but have received little publicity since no fatalities occurred; however, the environmental consequences have been severe in some cases. Current research related to safe mine waste disposal is being conducted in the areas of engineering property characterization, waste handling and placement systems, and stability control and analysis techniques

Research related to the safety and environmental impacts of mine waste management has become increasingly important since over 10 Federal Acts have been passed in recent years, most notably the Federal Water Pollution Control Act Amendments of 1972, the Resource Conservation and Recovery Act of 1976, the Surface Mining Control and Reclamation Act of 1977, and the Federal Mine Safety and Health Amendments Act of 1977 Current work related to the Acts and their subsequent standards include the development of adequate monitoring systems, development of seepage prediction and control techniques, control of runoff water, and control of leaching solutions

Bureau of Mines research in the mine waste disposal area is conducted in two Divisions, the Division of Minerals Health and Safety Technology and the Division of Minerals Environmental Technology. The objectives of these programs along with project descriptions are discussed in the following sections.

Background

The Bureau of Mines has been active in mine waste disposal research for the past 20 years. Experience gained on committees such as the ASCE-NSF workshop on Research Needs for Mining and Industrial Solid Waste Disposal, DOI Task Force on Coal Waste Hazards, West Virginia Ad Hoc Commission on Inquiry Into the Buffalo Creek Flood, Interagency Task Force on Reserve Mining Taconite Onland Disposal, NRC-Industry Uranium Waste Disposal Committee, Surface Mining Legislative Task Force, DOI Dam Safety Committee, Industry-Government Working Group for Research on Manganese Nodule Process Rejects, and the EPA Mining Solid Waste Coordinating Committee provides the background for a dynamic research program responsive to both Federal and industrial needs. Over the past decade, Bureau personnel have been highly active in both national and international symposia, workshops, and short courses.

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[1] Program manager
[2] Chief, Branch of Mine Waste Management Both authors are with the Division of Minerals Environmental Technology, Bureau of Mines, Washington, D C

Waste disposal practices in both the coal and metal mining industries have undergone considerable change during the past 20 years as a result of legislative pressures and changes in refuse characteristics. For a long time, lump coal was at a premium for domestic use, and power plants did not use the fines. The coarse washery waste was dumped onto large piles and the wash water was discharged into streams. In response to various state laws in the 1950's and 1960's, operators began pumping wash water up the hollows behind the waste piles where, after seeping through the coarse waste, the water was sufficiently clarified for discharge into the streams. This solved one problem but created another—saturated, unstable waste impoundments. Such a situation was responsible for the Buffalo Creek disaster. This prompted legislative action requiring proper design of waste impoundments to ensure stability. Meanwhile, the coal industry was undergoing another subtle change. Because steam plants could now handle fines, fine grinding with flotation was introduced to promote recovery, which produced a larger percentage of fine waste and resulted in more impounding structures.

The history of metal mines tailings disposal is similar. Early-day milling required only coarse crushing followed by gravity separation on tables or in jigs. Now, with flotation and low-grade feed material, many mills grind to 60 pct minus 200 mesh and finer. Like the coal industry, many mill operators, until recently, dumped the tailings into the nearest stream. This is no longer acceptable, and active tailing ponds now cover from as little as 1 acre to as many as 5,000 acres, attaining heights of 300 to 400 feet with projected heights approaching 500 feet. In general, metal mine tailings ponds have been better engineered than coal embankments, but some of these have also failed. As dams become larger, the need increases for better engineering and operating procedures.

The Bureau of Mines began investigating the properties of mill tailings hydraulically emplaced in underground mine openings as early as 1955. The primary reason at that time for returning the waste material underground was to provide backfill for ground support—elimination of surface disposal was a secondary benefit. In the 1960's, the Bureau began research on stabilizing mine-waste piles and on eliminating burning coal-refuse dumps. Since then there has been continuing emphasis on the solution of waste-disposal problems. The Bureau has over 100 reports and other publications on this subject from in-house and contract research.

Objectives

The general overall goals of the Bureau's mine waste disposal research are to (1) define and assess the major structural stability and environmental problem areas associated with the disposal of mine and mill wastes for various commodities, (2) design and develop control techniques addressing these problems and promote their incorporation in industry practice, and (3) develop alternative disposal practices promoting effective land use and waste utilization.

The following is a description of the specific objectives of the Bureau's mine waste disposal research.

- Compile a complete inventory of waste disposal sites and develop an updating system.
- Collect environmental data around several large tailing ponds and continue long-term monitoring.
- Determine engineering properties, compaction characteristics, and design criteria for waste disposal for various metal, nonmetal, and coal commodities.
- Develop and demonstrate underground disposal systems for coal and metal mines.
- Develop alternative techniques for innovative surface disposal systems and waste utilization methods.
- Develop, test, and monitor control methods to minimize seepage through waste ponds.
- Develop rapid in situ and remote monitoring techniques for determining embankment stability.
- Develop coal waste dewatering systems to improve the stability of fine coal refuse.
- Develop safety and risk analysis methods for evaluating waste embankment stability.

Implementation

As mentioned previously, the Bureau's research in this area is carried out in two Divisions. Research conducted in the Division of Minerals Health and Safety Technology emphasizes matters related to embankment stability. The program is closely coordinated with the Mine Safety and Health Administration (Department of Labor), primarily through formal joint ranking sessions where a project prioritization scheme is established. Waste management research conducted in the Division of Minerals Environmental Technology is aimed at minimizing the environmental effects of mine waste disposal. Since this is somewhat broad in nature, some stability work is also performed in this program due to the close relationship of structural and environmental problem areas. Work in this program is closely coordinated with other Federal agencies—the Office of Surface Mining (OSM), Environmental Protection Agency, and Geological Survey. Joint project ranking sessions are conducted with OSM.

Project Descriptions

The following is a description of mine waste research projects in the Divisions of Health and Safety and Environmental Technology listed by in-house and contract categories.

Minerals Environmental Technology, Solid Waste Management

IN-HOUSE RESEARCH

1. *Control of Hazardous Alkalies From Cement Kiln Dust. Objective:* Eliminate an environmental problem in cement production caused by the alkali and the alkali-free materials. Presently, 6 to 10 million tons per year of kiln dust are discarded to dumps or land fills because alkali renders the dust unsuitable for cement. The discarded

dust is a hazardous waste because the alkali can leach out and contaminate ground water

2. *Characterization of Potential Manganese Nodule Processing Rejects.*
   Objective: Verify, improve, and update the information available on manganese nodules; clarify some of the terminology used in identification and characterization of nodules and their processing reject materials; and prepare representative manganese nodule reject materials for future study. This research is conducted under an interagency agreement with the National Oceanic and Atmospheric Administration.

3. *Tailings Pond Research and Testing Facility.*
   Objective: Continue pond liner testing program on samples installed in industrial ponds. Remove and test liner samples for properties such as permeability and tear strength. Install several new liners in operating ponds. Test dewatering techniques such as sand bed drainage and capillary assisted evaporation.

4. *Controlled Burnout of Coal Waste Bank Fires at the Moss No. 1 Plant.*
   Objective: Assist in technology transfer of the concept of controlled burnout of a fire in a coal waste bank utilizing Bureau of Mines developed in situ combustion methodology.

5. *Controlled Burnout of Fire at Abandoned Coal Mines—Calamity Hollow.*
   Objective: Demonstrate the concept of controlled burnout of fire in an abandoned coal mine by utilizing Bureau of Mines methodology for in situ combustion to deplete the fuel responsible for the fire in an environmentally sound manner and utilize the heat produced.

6. *R&D Support for Burnout Control of Fires on Abandoned Coal Mine Lands.*
   Objective: Develop analytic models for controlled burnout, improve methods of high temperature borehole constructions and instrumentation for monitoring burnout control, and evaluate burnout control for coal wastes. The latter two items will involve activities in the Bureau's surface trench burn facility

7. *Abating Pollution From Lead Tailings Piles in Southeast Missouri.*
   Objective: Examine lead mine tailings ponds in the southeast Missouri lead belt to characterize the nature and magnitude of compositional and physical makeup of contaminating constituents and assess potential methods of rendering these tailings environmentally safe to regional surface and ground water systems.

8. *Pollutant Identification in Auto Shredder Offal Material.*
   Objective: Select samples of fluff and other residues that are commonly disposed of in landfills from a number of shredders throughout the country. Subject samples to chemical analysis and handpicking to determine their chemical and physical makeup. Leach samples according to proposed EPA standards and analyze the leachate to determine its composition.

9. *Physical and Chemical Characteristics of Manganese Nodule Processing Wastes*
   Objective: Obtain samples of waste from the Avondale Research Center Perform laboratory tests to determine the following materials properties: (1) Material composition-mineral and chemical, (2) permeability, (3) settling rates, (4) size distribution, (5) specific gravity, (6) densities, (7) electrical conductivity, and (8) strengths

10. *Improved Hydraulic Method for Coal Waste Disposal*
    Objective: Provide an improved method of waste disposal by providing economical slurry transport and a stable blended waste. Present methods of handling coal refuse include sludge disposal ponds for fine coal refuse, which are hazardous because of the hydraulic properties of the material, and coarse waste dumps that tend to ignite because of porous nature of the embankments Recently, some coal cleaning plants have installed dewatering devices for the fine refuse and combine the coarse and dewatered sludge prior to disposal. The procedure is extremely expensive, produces a "bulked" material for disposal that has poor long-term stability characteristics, and presents materials handling problems.

11. *Development and Evaluation of Waste Disposal Plans and Techniques.*
    Objective: Identify major waste problems through meetings with industry representatives. Maintain industry contacts and perform background and preliminary research in the development of new research projects

12. *Wastewater Monitoring Techniques for Uranium Tailings.*
    Objective: Simplify and improve data gathering techniques to improve the state of the art of waste water monitoring and provide a basis on which realistic regulations may be promulgated. In this case the benefit would apply to industry as well as to applicable regulatory agencies. Seepage from uranium tailings has been a quite controversial subject among regulating agencies and industry. Certain proposed regulations could have a drastic effect on the economics of uranium mining. Many uncertainties are due to the many unknowns in water quality, quantity, and movement in the disposal vicinity.

13. *Design of Slime-Sealed Impoundments to Prevent Ground-Water Contamination.*
    Objective: Use mill tailings or natural soil rather than fabric liners to reduce seepage in tailings impoundments. Determine potential methods for reducing the seepage beneath tailings ponds through laboratory permeability testing of mill tailings and natural soil within a tailings area in combination with additives and compaction techniques. Transfer results to field tests where the actual seepage for a given area will be measured.

14. *Survey Foundry Wastes in Southeastern United States to Identify Environmental Hazards.*
    Objective: Initiate a systematic program to identify the various types of waste materials generated by foundries in the southeastern United States, determine whether

the wastes currently or potentially pose environmental hazards, and recommend research to alleviate any problems.

15. *Assessment of Environmental Impacts Associated With Byproduct Gypsum Stacks From Florida Phosphates.*
Objective: Determine the extent and nature of radiation-related and other environmental problems associated with the disposal and subsequent leaching of byproduct gypsum, and recommend and initiate research to alleviate the problems.

16. *Borehole Mining Cavities for Disposal of Radioactive Mine Wastes.*
Objective: Investigate aspects of rock mechanics, environmental engineering, safety engineering, economics, and nuclear energy regulation relative to adaptation of the Bureau borehole mining technology to the construction of storage cavities for solid wastes generated in uranium in situ leaching operations.

Contract Research

1. *Containment Pond Liner Materials Testing.*
Objective: Determine performance of typical pond liner materials under actual industrial service. Install liners in operating ponds and test them for material properties such as permeability and tear strength at various time intervals.

2. *Demonstration and Evaluation of Underground Disposal of Coal Mining Wastes.*
Objective: Conduct a 1-year demonstration of a system of underground disposal of coal mining wastes in an active underground coal mine. Provide an assessment of technical problems and economics of implementation of a total system of underground disposal of coal mining waste in an active underground coal mine.

3. *Design and Engineering of a Burnout Control System for a Coal Waste Bank.*
Objective: Utilize the controlled acceleration of a fire by air injection so that complete fuel burnout is accomplished in a time period that is short compared to the many decades that such fires could continue if left unattended. Inject air into and remove combustion products from the waste bank by a surface exhaust ventilation system operating through boreholes extending into the bank.

4. *Developing a Slurry Fill for Modified In Situ Oil Shale Mining.*
Objective: Develop an underground slurry filling system that will minimize surface and underground environmental disturbance. Specifically, render in situ shale rubble virtually impermeable and minimize contamination of the ground water and its aquifers, bind the spent shale together in a coherent mass sufficiently strong to resist surface subsidence and eliminate the potential for differential settlements of the surface, and dispose of the maximum amount of surface retorted shale material as the major constituent of the fill, thereby minimizing or eliminating disposal of spent shale on the surface.

5. *Development of Systematic Waste Disposal Plans.*
Objective: Develop a set of guidelines for the mining industry that provides rationale for the development of systematic waste disposal plans. The resulting manual will be suitable for use by Federal and State personnel, mine operators and planners, and design professionals. Emphasis will be placed on the impact of a disposed waste on all aspects of the environment, including, but not limited to: future land use and economic factors, long-term stability, hydrology effects, potential contamination of water supplies and emanation of hazardous substances. Impact of existing and proposed State and Federal regulations on the disposal of waste will be emphasized as well.

6. *Mine Waste Location by Satellite Imagery.*
Objective: Investigate the potential for using satellite (Landsat) data for detecting coal and metal-nonmetal waste and tailings disposal sites. Determine if the band-ratio method of preprocessing Landsat multispectral data before classification can be used to detect and identify waste embankments.

7. *Engineering Property Changes and Environmental Effects on Coal Mine Waste Due to Slaking.*
Objective: Determine the changes in physical properties and chemical environmental impact potential of coal mine waste dumps and embankments as a function of age, and evaluate the effects of slaking on the stability of waste piles and recommend techniques to control the detrimental changes.

8. *Conceptual Designs for Retaining Structures for Open Pit Backfilling.*
Objective: Develop conceptual designs of retaining structures for constructing backfill areas to dispose of wastes within open pits. Milling, smelter, and overburden wastes will be considered. Develop detailed conceptual designs for two operating open pit mines.

9. *State-of-the-Art Environmental Assessment of Onshore Disposal of Manganese Nodule Rejects.*
Objective: Identify all reasonable state-of-the-art and emerging techniques for onshore disposal of various types of manganese nodule rejects and develop site selection design criteria and associated significant environmental and land use effects for those techniques, sites, and types of rejects identified.

10. *Evaluation of Lixiviation of Mine Waste.*
Objective: Determine which types of, and to what extent, coal wastes contaminate ground water through leaching of acid- or toxic-forming materials. Collect solid waste samples and evaluate, through laboratory studies, the extent to which components of these wastes are leached by rainwater.

11. *Field Test for Environmental Disposal of Mill Tailings in Surface Backfill.*
Objective: Perform a small-scale field test to dispose of mill tailings along the Coeur d'Alene River valley while preventing seepage through the waste and into the water system; conduct geotechnical, hydrologic, and vegeta-

tive experiments at the site; and evaluate feasibility of full-scale implementation.

12. *Investigations of the Flow Characteristics of Mine Tailings.*
Objective: Determine the flow characteristics of liquefied mine tailings and, when a mass of mine tailings flows and liquefies, the probable extent of travel. Analyze case studies of mass flow failures of mine embankments.

13. *Designs for an Underground Disposal System for Active Uranium Mines.*
Objective: Define environmental stability, health and safety, and regulatory constraints on backfilling. From these definitions, develop a number of initial conceptual designs based on related research and other pertinent information. As the final phase, develop a detailed design at a specific site.

14. *Disposal of Gold Dredge Tailings.*
Objective: Develop and demonstrate methods to partially level the ridges of coarse gravel left by the dredge waste conveyor and pump the finer gravel, sand, and soil onto the smoothened area. This would eliminate ridges of sterile, washed, coarse gravel that exist from present methods of operation and place the topsoil on the surface in a usable condition.

15. *Inventory of Waste Embankments, Surface and Underground Openings.*
Objective: Compile an inventory of mine waste embankments to form a data base for improving waste management and waste disposal techniques in the metal-nonmetal mining industry.

16. *Measurement of Seepage Beneath Tailings Ponds.*
Objective: Apply the results of the in-house project "Design of Slime Sealed Impoundments to Prevent Ground Water Contamination" to field conditions where seepage can be accurately measured to determine the best possible method for sealing the bottom of the tailings pond to prevent seepage. Five separate areas are being prepared to test different natural soil-tailing combinations and compaction to achieve the desired results.

17. *Removing Heavy Metal From Runoff Water Draining Lean Copper-Nickel Ore Stockpiles.*
Objective: Develop techniques for removing heavy metal pollutants from runoff draining copper-nickel lean ore and waste stock piles. The development of copper-nickel resources in northeastern Minnesota depends upon the selection of an adequate abatement program to control heavy metal leaching from exposed waste rock and lean ore piles. The low grade of the ore deposit makes it necessary to find an inexpensive control system that will allow the mining to be economically feasible.

18. *Assessment of Ground and Surface Water Effects Around Coal and Mineral Storage Areas.*
Objective: Survey the extent to which storage piles of coal and minerals release toxic ions or other significant organic pollutants to ground or surface waters and assess pollution types and levels. Develop mitigative measures for each mineral commodity for improved control of runoff.

19. *Detection of Lixiviant Excursions With Geophysical Resistant Measurements During In Situ Leaching.*
Objective: Develop, test, and demonstrate a geophysical resistance measuring system that can reliably detect the excursion of a lixiviant having a resistivity half that of the ground water it replaces when the lixiviant has migrated half way from an injection well to a monitor well at a depth of at least 500 feet.

20. *Evaluation of Best Management Practice for Mining Solid Waste.*
Objective: Determine best management practices for mine waste disposal in the areas of ore (copper, iron, lead, nickel, molybdenum, and zinc), phosphates, and uranium by utilizing the results of an extensive monitoring program for ground and surface water and air quality. (Cooperative agreement with the Environmental Protection Agency)

Minerals Health and Safety Technology, Mine Waste Stability

IN-HOUSE RESEARCH

1. *Evaluation of Filter Cloth for Stabilization of Coal Mine Wastes.*
Objective: Evaluate the criteria for selection of filter cloth to control seepage in coal mine waste dams. Conduct laboratory tests of various filter cloths under simulated mine waste dam environments. Develop preliminary guidelines for use of filter cloths in coal mine waste dams

2. *Consolidation of Coal-Clay Wastes by an Improved Flocculation Technique.*
Objective: Demonstrate the technical feasibility of using an improved flocculation technique to dewater waste coal sludge generated in coal preparation plants to produce a consolidated stable waste material containing 50 or more weight-percent solids that can be safely stored. Through laboratory investigations, optimize flocculation and consolidation parameters, and establish mass flow rates. Based on laboratory investigations, design and assemble a larger-scale field test unit. Demonstrate the feasibility of mixing dewatered coal sludge with coarse coal refuse material for long-term stabilization of both waste products.

3. *Mixing Coarse and Fine Coal Waste.*
Objective: Determine optimum mixing ratios of coarse and fine coal wastes to achieve maximum fill strengths for surface disposal, and develop a method to mix and transport the mixtures while minimizing segregation. Obtain samples of coal wastes from preparation plants that mix fine and coarse coal wastes and that impound fine coal wastes behind an embankment of coarse wastes. Determine fill strengths of coarse and fine wastes by laboratory tests. Prepare a report on the waste disposal practices at the coal preparation plants visited.

4. *Alternative Coal Waste Disposal Methods.*
Objective: Complete physical property tests of coarse anthracite coal waste; collect Shelby tube samples of anthracite fine waste and determine their physical properties; and conduct laboratory model tests stimulating injection of fine waste particles into voids created by coarse waste. Injection of fine waste particles into voids is aimed at increasing the support capability of backfilling and other waste disposal methods.

5. *Factor of Safety/Risk Analysis in Tailings Embankments Design*
Objective: Apply techniques of operations research and statistics to the design of tailing dams; establish a confidence interval or level of uncertainty about the factor of safety; investigate state-of-the-art sampling procedures for tailings embankments; construct simplified factor of safety charts for field applications; investigate industry sampling procedures used to ascertain the safety of existing tailings embankments; and develop a statistical procedure to be applied to onsite embankment maintenance and control.

Contract Research

1. *Safety and Health Problems Associated With Underground Mine Waste Disposal.*
Objective: Identify potential safety and health hazards to underground miners that could result from the underground disposal of mine waste.

2. *Disposal of Wastes Over Active Underground Mines.*
Objective: Determine if guidelines can be developed for safe and economical methods for the disposal of wastes over active underground coal mines. Identify different types of mine wastes and evaluate their hazards to underground mining; evaluate underground parameters that may affect waste pile stability; evaluate peripheral conditions that may affect the waste disposal area; evaluate probable safety features that may be incorporated either above or below ground; and compare available waste disposal techniques for their applicability.

3. *Centrifuge Model Testing of Waste Embankments.*
Objective: Determine safety criteria for tailings embankments by simulating field conditions using a centrifuge for modeling. Conduct tests on a 25-foot-radius centrifuge to investigate seepage and erosion effects, foundation differentials, and other embankment construction problems.

4. *Satellite Monitoring of Coal Waste Embankments.*
Objective: Demonstrate the use of a satellite communication link with a remote data collection station located on a coal waste embankment. Select a suitable satellite and negotiate an agreement to use it Purchase or rent a data collection platform for placement at the test site to connect with the instruments already in place from previous work. The present data collection station at the test site will remain in use to check data received from the satellite and to provide additional long-term information from the instruments. Compare and evaluate data from the two data stations to provide information as to the most reliable and cost-effective system.

5. *Critical Parameters for Tailing Embankments.*
Objective: Construct probability density functions of soil parameters for tailings embankments that are representative of the major mining commodities in the United States. Collect and categorize engineering parameters of tailings embankments for future input to slope stability models; construct probability density functions of the data; and compute the mean and variance of the data.

6. *Compaction Criteria for Metal-Nonmetal Tailings.*
Objective: Develop data to determine compaction criteria for various metal-nonmetal wastes for both surface and underground disposal. Determine the extent of compaction criteria available through an extensive literature search. Obtain samples of tailings from various types of metal-nonmetal mines and conduct laboratory tests to determine density, grain size, permeability, and static shear. Determine correlations between laboratory results and actual field compaction efforts.

Summary

The Bureau of Mines has been active in mine waste disposal research for many years. Current legislative actions and the concern for structural and environmental integrity of mine waste management systems have kept this research area an important part of the Bureau's minerals research program. The overall objectives and specific projects described in this paper provide a current documentation of problems being addressed. Industry input continues to play a major role in the formulation of the research programs and an active technology transfer program is being implemented to assist in the implementation of results obtained.

DISPOSAL OF COAL MINE WASTE IN ACTIVE UNDERGROUND COAL MINES

by

Leslie S. Rubin[1], Mackenzie Burnett[2], Al Amundson[3], Gary J. Colaizzi[4], and Ralph H. Whaite[5]

BACKGROUND

Nature and Extent

Approximately 25 pct of the raw coal extracted from underground mines in the United States is rejected as waste and deposited on the surface, usually near the preparation plants[6]. These accumulations on the surface result from screening, crushing, sizing, and washing of coal to separate impurities from the marketable product and are called waste banks, refuse banks, culm banks, or bony piles. The reject material contains a certain amount of combustible as well as pyritic and other objectionable material; it is generally separated into two classes, coarse and fine refuse. Coarse refuse accumulations generally resemble conical or ridge-shaped mounds that may extend up to 700 feet in height and over 1 mile in length. The fine refuse is often impounded in settling ponds.

The Bureau of Mines estimates that 174,000 acres in the United States used for disposal of coal processing waste remain unreclaimed. Burning material is contained in about 290 banks covering approximately 3,000 acres distributed throughout 13 of the 26 coal producing States[6]. In the Eastern Coal Province alone there are at least 3,000 to 5,000 sizable waste piles and impoundments which cumulatively contain over 3 billion tons of refuse[7]. At current coal production rates, about 60 million tons of mine refuse and processing waste are generated each year.

Waste banks and impoundments are potential health and safety hazards. Waste bank slides and impoundment failures have destroyed entire communities. In addition, such banks are environmentally degrading, create an eyesore, and are sources of acid drainage pollution. The burning banks contribute significantly to the degradation of surrounding atmospheric conditions and inhibit development and economic growth. The resulting smoke, dust, and poisonous and noxious gases can be fatal to human, animal, and plant life.

Legislation and Regulations

Recognition of the severity of the solid waste problem in the United States by the Congress resulted in passage of the Solid Waste Disposal Act, Public Law 89–272, on October 20, 1965. Under this act the Bureau's responsibility for research and development work on the disposal or utilization of mineral waste was expanded. It included economic and resource evaluation studies aimed at delineating factors causing and contributing to waste disposal problems in mineral and fossil fuel industries, and scientific and engineering research to find ways of utilizing, or otherwise disposing of, a variety of inorganic waste materials.

The first Federal regulations governing the surface disposal of coal wastes were promulgated pursuant to the Federal Coal Mine Health and Safety Act of 1969, Public Law 91–173. Under the act the Secretary of the Interior was required to develop health and safety standards for surface disposal of waste. The first regulation required that waste piles constructed after 1971 be located a safe distance from underground mine openings and preparation plants and that new refuse piles constructed over exposed coal beds be separated from the coal by clay or some other inert material. The second regulation concerned the construction of coal waste piles, requiring the waste be spread in layers and placed so as to minimize the passage of air through the piles. They were not to be constructed in a way that would impede drain-

[1] Senior engineer, Mechanical Processing and Fluid Transport Division, Foster-Miller Associates, Inc., Waltham, Mass
[2] Division manager, Mechanical Processing and Fluid Transport Division, Foster-Miller Associates, Inc., Waltham, Mass
[3] Chief engineer, Western Slope Carbon, Inc., Salt Lake City, Utah
[4] Supervisory mining engineer, Denver Research Center, Denver, Colo
[5] Mining engineer, Denver Research Center, Bureau of Mines, Denver, Colo
[6] National Academy of Sciences, National Academy of Engineering Underground Disposal of Coal Mine Wastes Government Printing Office, Washington, D.C., ISB No 0–309–02324–6, 1975, 172 pp.
[7] Johnson, W., and G. C. Miller Abandoned Coal-Mined Lands—Nature, Extent, and Cost of Reclamation BuMines Spec Pub. 6–79, 1979, 29 pp

age or impound water and were to be designed to prevent sliding and shifting of materials. The third requirement involved inspection and monitoring of all retaining dams if it was thought that failure of a water or silt-retaining dam would create a hazard to miners.

In 1966 at Aberfan, Wales, more than 150 residents (mostly children) in that small community perished, when an 800-foot-high coal refuse pile slid down the mountainside into the town. The laws enacted by Pennsylvania, West Virginia, and Kentucky regarding coal waste disposal were strengthened in 1968 primarily as a result of the Aberfan disaster. In the aftermath of the Buffalo Creek disaster in 1972, an inspection and inventory of all waste impoundments was conducted in order to determine their stability. Federal inspectors from the then Department of the Interior Mining and Enforcement and Safety Administration, who were trained in coal waste pile stability, made the examinations. In March 1978 this work was transferred to the Mine Safety and Health Administration (MSHA) in the Department of Labor. The emergency investigation of 139 impoundments, initiated after the Buffalo Creek disaster, resulted in the rating of 8 impoundments as imminent hazards, 66 as obviously deficient, 61 as not obviously deficient, and 4 as missing data. Under the authority of Public Law 92–367, The Dam Safety Act, enacted August 8, 1972, the Army Corps of Engineers conducted a survey of all waste impoundments in the Appalachian region and reported the results to the State governors. Because of the potential hazards to miners and the public from improperly constructed coal waste piles and impoundments, the Department of the Interior, in 1974, published proposed new and expanded regulations controlling waste piles in the Federal Register.

The Surface Mining Control and Reclamation Act of 1977, Public Law 95–87, signed into law on August 3, 1977, created within the Department of the Interior an Office of Surface Mining Reclamation and Enforcement (OSM), and regulations under the provisions of the Act were promulgated in December 1977. Under this authority the regulations initiated by the act of 1969 were upgraded and greatly expanded and included requirements for underground disposal, wherever practiced, as well as for surface disposal of coal processing waste.

Under the act of 1977, the general requirements for the construction and maintenance of coal processing waste banks included that the work be done within a permit area and in accordance with rules established for disposal of underground development waste and excess spoil and to prevent combustion. The disposal area should not adversely affect water quality, water flow, or vegetation, should not create public health hazards or cause instability in the disposal area, and should not extend to within 8 feet of any coal outcrop. Moreover, all coal processing waste banks should be inspected, on behalf of the person conducting underground mining activities, by a qualified registered engineer or other person approved by the regulatory authority. Furthermore, coal processing waste should not be used in the construction of dams and embankments unless it has been demonstrated to the regulatory authority that the stability of such a structure is in accord with the regulations.

The requirements for returning coal processing waste to underground workings are in five parts: (1) Each plan shall describe the design, operation and maintenance of any proposed coal processing waste disposal facility for approval of the regulatory authority and the Mine Safety and Health Administration; (2) Each plan shall describe the source and quality of waste to be stowed, area to be backfilled, percent of mine void to be filled, method of constructing underground retaining walls, influence of the backfilling operation on active underground mine operations, surface area to be supported by the backfill, and anticipated occurrence of surface effects after backfilling; (3) the applicant shall describe the source of the hydraulic transport mediums, methods of dewatering the placed backfill, retainment of water underground, and treatment of water if released to surface streams, including the effect on the hydrologic regime; (4) the plan shall describe each permanent monitoring well to be located in the backfilled area, the stratum underlying the mined coal, and gradient from the backfilled area; and (5) where applicable the preceding requirements shall also apply to pneumatic backfilling operations.

Costs of Waste Bank Reclamation

Costs of coal waste disposal in the United States vary within a narrow range and depend primarily on the choice of materials handling systems, topography of the mining operation, and the distance between the source of waste and the disposal site. Almost exclusively, waste disposal for underground mining operations is currently accomplished by surface disposal on waste piles near the preparation plant.

Current practices of surface disposal consist of dual waste disposal systems, one for the coarse waste and one for the fine waste. Coarse waste is usually handled by truck, belt conveyor or aerial tram or a combination thereof, and deposited directly on the waste pile. Fine waste is normally transported as a slurry by pipeline to an impoundment. If the fine waste is completely drained, it can be removed from the impoundment and deposited on the waste pile.

Typical costs as of 1975 for operating total waste systems (belt, truck, dozer) range from $0.15 to $0.40 per ton of waste handled[6]. Engineering estimates for installing a new system based on costs of new equipment are considerably greater Due to inflation and the upgraded requirements promulgated under Public Law 95–87, costs of surface reclamation by 1980 have risen sharply. While some information on the costs of disposal of coal processing waste in abandoned underground mines is available; similar data on costs of disposal in active underground mines are lacking.

Recommendation of National Academy of Sciences

The inclusion of regulations pertaining to the underground disposal of coal processing waste that were developed after passage of the 1977 act may be attributed to a 1975 study conducted by the National Academy of Sciences (NAS) for the National Science Foundation[6]. This study addressed the state-of-the-art technology for mine backfilling concurrent with coal extraction, and provided an overview of the technical and economic constraints on underground disposal of coal mining wastes. The NAS study recommended that underground waste disposal be demonstrated at sites representative of U.S. coal mining practice in order to establish the feasibility of this method of disposal of coal mining waste in the United States.

BUREAU OF MINES PROJECT FOR STUDY OF TECHNICAL AND ECONOMIC EVALUATION OF UNDERGROUND DISPOSAL OF COAL MINE WASTE

In response to the recommendation of NAS and the promulgation of rules governing the underground disposal of coal preparation plant waste subsequent to Public Law 95–87, the Bureau of Mines initiated a detailed study of the technical, economic, and environmental constraints on underground waste disposal. A request for proposals to conduct the study was advertised April 14, 1977, and responses were received from seven engineering firms from various parts of the country.

The objective of the proposed study was to determine the specific circumstances under which the disposal of coal mine waste underground concurrently with extraction is technically and economically feasible. The work was to be conducted in two distinct phases.

The contract to perform the work was awarded to HRB-Singer, Inc., of State College, Pa.[8]. A final report on the two-phased investigation that assessed the technical and economic feasibility of disposing of coal refuse underground in active mines was submitted to the Bureau of Mines in January 1980.

During phase I, which included a literature survey, coal-producing regions of the United States were identified in which the problems of surface disposal of coal refuse were most severe. Criteria used in identifying the regions included regulatory constraints, availability of disposal sites, and other social, economic, and physiographic factors such as the amount of coal refuse produced in the region, population density, precipitation, local relief, steepness of slopes, volatility and sulfur content of the coal including the refuse, urban area density, number of coal preparation plants, and number of burning refuse banks and impoundment failures reported in the region over the past decade.

From an initial list of 20 regions, three were selected as having the most severe problems of surface refuse disposal: southwestern Pennsylvania, southern West Virginia, and southwestern West Virginia combined with eastern Kentucky. For each of these three regions, a model mine was postulated that typified the geologic conditions, mining operations, economic factors, and environmental conditions that prevail in each area. Costs of surface disposal were also estimated for each region..

The mine model that was developed for southwestern West Virginia and eastern Kentucky, the "Kanawha Region," was selected as the basis for a detailed analysis of underground disposal systems in phase II. The Kanawha Region was selected because of its past and present problems of refuse disposal (Buffalo Creek is located in this region), and because the topography, mine size, mine type, and mining operations of this region are similar to conditions existing elsewhere in West Virginia, eastern Kentucky, and Virginia.

In phase II of this 2-year study, various methods used to stow refuse underground were reviewed and evaluated, and three methods that would be most suitable for implementation at the model mine were selected. For each of these methods, a conceptual design of a stowing system was developed.

The stowing systems developed included: (1) A mechanical backfilling system in which coal refuse is transported from the preparation plant to the mine site by truck, introduced to the underground workings by gravity feed via a service shaft and conveyor belt, and emplaced in a mined out area by battery-powered scoop cars; (2) a pneumatic backfilling system in which the refuse is transported from the preparation plant to the mine entry by truck, introduced to the stowing area by rail cars, and emplaced in the mined out voids by pneumatic stowers; and (3) a direct hydraulic system whereby refuse in the form of a slurry is transported from the preparation plant to the underground stowage area hydraulically in a pipeline. The three methods were selected not only on the basis of their appropriateness for the model mine, but also on the basis of the extent to which the methods exemplified the major methods of backfilling employed currently throughout the world. Environmental factors associated with each type of backfilling, the technical feasibility of each design concept, and the costs of implementing each conceptual design were assessed.

All three systems of stowing refuse underground in active mines are more expensive than surface disposal of refuse in the Kanawha Region. Estimated costs of surface disposal (1978) ranged from $2.46 to $2.61 per ton of refuse, whereas estimated costs of mechanically stowing 1 ton of refuse were $4.14; pneumatically stowing 1 ton of refuse, $7.67; and hydraulically stowing 1 ton of refuse, $4.31. The costs of hydraulic stowing were somewhat unrealistic because the preparation plant of the model mine was located more than 3 miles from the mine site. If the preparation plant were located closer to the mine, the per-ton cost of refuse disposal would be considerably lower.

The members of the evaluation team determined that the elements of each conceptual design were acceptable and their costs reasonable. While the model mine does not completely encompass the complexity of variable conditions from mine to mine in relation to the full range of backfilling options, it does provide some useful guidelines to the mine operator in efforts to comply with regulations at an acceptable cost.

[8] Bucek, M. F., J. K. Clauser, J. A. Schad, and N. K. Chukravorti. Technical and Economic Evaluation of Underground Disposal of Coal Mining Wastes. HRB-Singer, Inc. (State College, Pa.), Final Rept. Contract J0285008, January 1980, 349 pp.; BuMines OFR 80–015, available for consultation at the Bureau of Mines facilities in Albany, Oreg., Avondale, Md., Denver, Colo., Pittsburgh, Pa., Reno, Nev., Rolla, Mo., Salt Lake City, Utah, Spokane, Wash., Tuscaloosa, Ala., Minneapolis, Minn., and Central Library, U.S. Department of the Interior, Washington, D.C.; DOE facilities in Carbondale, Ill., and Morgantown, W. Va.; and from National Technical Information Service, Springfield, Va., PB 80–154768.

DETAILED DESIGN AND DEMONSTRATION OF UNDERGROUND DISPOSAL OF COAL MINING WASTES—BUREAU OF MINES REQUEST FOR PROPOSALS

The Bureau of Mines, as part of its minerals environmental technology program and continuing search for more definitive answers to the underground disposal of coal wastes, decided to initiate a detailed mine site specific design of a system of underground disposal of coal wastes; and implement a cooperative demonstration of the system in an active underground coal mine.

The scope of work was divided into two distinct phases: Phase I—detailed design of underground coal waste disposal system; and phase II—demonstration and evaluation of the underground disposal system. It was required that the disposal system design include the handling of the solid waste (coarse and fine) generated as a result of the underground coal mining and associated coal preparation processes.

It was required also that the proposal include a description of the proposed demonstration mine site for development of the detailed design; and a discussion of the proposed conceptual disposal system including method or methods (hydraulic, mechanical, pneumatic, etc.) for the proposed mine. The mine description included its location and geologic setting, mining methods, current waste disposal methods, production, coal preparation, characteristics of coal wastes, etc.

Phase I—Design

During phase I, which was to be completed in 6 months, it was required that the contractor prepare a detailed mine site specific design including calculations, drawings, and costs for the subsequent demonstration of the complete underground disposal system. In developing the design the contractor was required to assess or incorporate the following considerations: (1) Physical and chemical characteristics of coal wastes produced by the mine to determine material flow characteristics, wear rates, etc.; (2) availability of required equipment and technology to introduce and operate the system; (3) extent to which current operating methods in the mine may be changed or disrupted in order to accomodate the disposal system; (4) any anticipated decrease in mine productivity due to the introduction of the disposal system, (5) effects on health and safety of production personnel as well as those engaged in implementing the proposed disposal system; and (6) associated benefits, if any, such as strata control or increase in recoverable coal due to the underground disposal of the coal wastes.

The overall compatibility between the current mining system and the underground disposal scheme was to be assessed. This assessment was to include an analysis of safety and conformance of all equipment and design criteria required by all applicable State and Federal laws and regulations, including the Coal Mine Health and Safety Act, Surface Mining Control and Reclamation Act, etc.

A comprehensive monitoring system was to include material flow quantities, placed material density, compressibility, mine roof support, effects on ventilation, dust, noise, and where applicable effects upon mine water handling, influx, discharge, etc.

During phase I, the contractor was required to obtain or be actively engaged in acquiring all the necessary State and Federal permits for implementing the underground disposal system demonstration.

Phase II—Onsite Demonstration

During this phase, the contractor will conduct a demonstration of the complete underground waste disposal system. The demonstration shall be full scale in order to perform an evaluation of the technical and economic success of the underground disposal system.

The work will include the construction of all facilities necessary to integrate the approved waste disposal system into the active underground mine. All equipment and installations shall comply with all Federal and State laws and regulations. All monitoring and control systems shall be thoroughly checked to accurately reflect the performance of the entire system. An accurate daily log shall be maintained and include all pertinent cost data, from which the cost per ton of waste placed underground shall be calculated. From this data, any loss or gain in productivity shall also be measured and reported. In addition to environmental effects, the final report shall summarize all findings, results, conclusions, and recommendations including sketches and design drawings as needed for clarification.

Two Contracts Awarded for Phase I

Only two responses were received and both proposals were evaluated by a committee of technically qualified Bureau, OSM, and MSHA employees assisted by Bureau contracting personnel. It was decided to fund both proposals because they represented alternative methods, which could be quite important to the mining industry. Both contracts were awarded on September 27, 1980.

One contract was awarded to GEX Colorado, Inc., Palisade, Colo., operators of underground coal mines in Palisade, working as a team with Michael Baker, Jr., Inc., a diverse civil engineering firm in Beaver, Pa. The proposal submitted by this team covering phase I was concluded with a comprehensive detailed design of an underground disposal system for the Roadside mine in Mesa County, Colo., about 12 miles northeast of Grand Junction, Colo. The plan involved hydraulic underground disposal of the 1,750 tons of refuse produced daily by the Roadside preparation plant. Complete details of the proposed demonstration work were given in a final report by the GEX Colorado—Michael Baker, Jr., team.

Work on the second contract is more advanced and is therefore the one about which this paper is primarily concerned. It involves a method of underground disposal of preparation plant waste in an active mine that utilizes both hydraulic and pneumatic systems. The contract to prepare this design was awarded to Foster Miller Associates, Inc. (FMA)/Western Slope Carbon, Inc. (WSC).

The WSC Hawk's Nest mine near Paonia in western Colorado was selected as the site for the demonstration The Federal leases that make up the Hawk's Nest mine property lie along the north fork of the Gunnison River about 1.5 miles east of Somerset, Colo. Elevation varies from 6,140 feet along the river to 8,200 feet on the northern edge of the leases. Both phases of the proposed work are to be managed by FMA with technical and labor support from WSC.

Design Submitted by Foster-Miller Associates/Western Slope Carbon, Inc., for Demonstration at the Hawk's Nest Mine, Paonia, Colo.

The underground waste disposal system being considered for the WSC Hawk's Nest mine utilizes hydraulic transport, in-mine dewatering, and pneumatic haulage. This disposal system is summarized in the generalized flowsheet of figure 1. Fine refuse (minus 35 mesh) and coarse refuse (plus 35 mesh) exit from the wash plant as two separate slurries. Both slurries are transported 4,400 feet through separate pipelines to the underground dewatering-pneumatic feeding station. Here the refuse slurries are dewatered on separate components and the solids fed to a pneumatic feeder for transport and placement in the appropriate mine location. Screen drainage and centrifuge effluent are returned to the wash plant, closing the water circuit.

The remainder of this paper describes: (1) The mine refuse, (2) the waste disposal system and equipment selection, (3) the influence of the disposal system on mine productivity, and miner health and safety, and (4) the phase II program for system installation and evaluation.

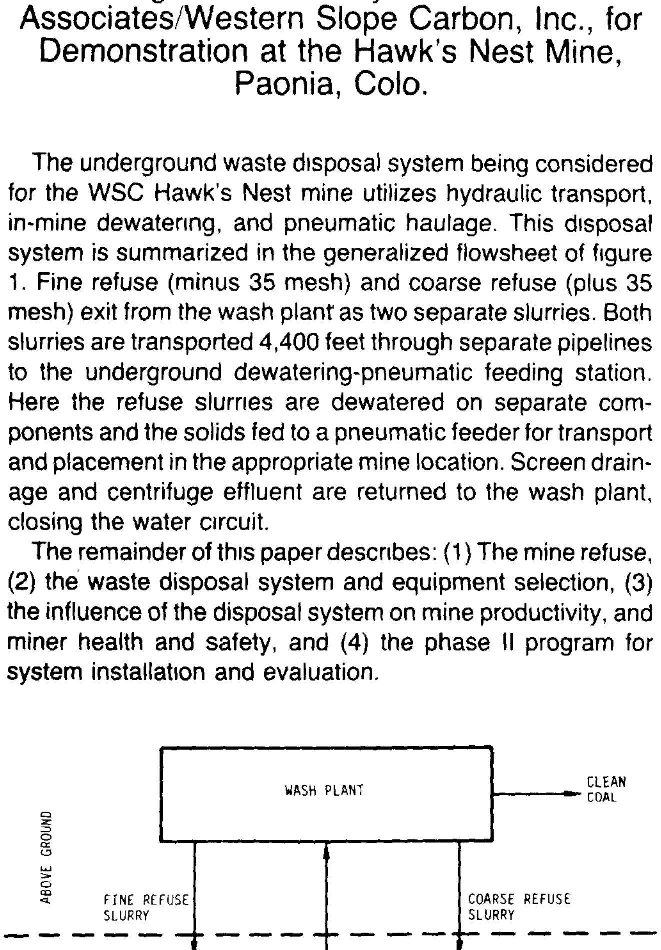


Figure 1.—Simplified flowsheet for underground disposal system.

Geology at the Hawk's Nest Mine

The WSC Hawk's Nest Mine is located at the eastern end of the Somerset District, a part of the Paonia Coal Field. Locally, the coal bearing strata is about 2,500 feet thick and is divided into four members, in ascending order. (1) Rollins sandstone member, (2) lower coal member (Bowie), (3) upper coal member (Paonia), and (4) barren member. The main coal beds within WSC's lease are found in the upper coal, or Paonia member. Of the four minable seams in this member, WSC is presently mining the E seam (Johnson's Hawk's Nest or F bed) and plans future development of rock tunnels to the underlying Wild seam and D seam.

The E seam is an attrital coal with a seam thickness ranging from 4.5 to 12 feet. Floor sediments are generally carbonaceous shale. Roof sediments vary from carbonaceous shales with interbedded siltstones to good roof-supporting sandstone. The Wild seam (Johnson's E seam) ranges from 5.5 to 10 feet in thickness above and below a minable coal. The coal invariably exhibits numerous splits and generally has poor roof and floor sediments, usually carbonaceous shales and interbedded siltstones. The D seam is an attrital coal with thicknesses ranging from 6 to 14 feet. This coal seam directly overlies the Bowie member and generally has very good floor and roof sandstones.

Physical and Chemical Characteristics of Coal Mine Wastes at Hawk's Nest Mine

The average run-of-mine feed rate for which the wash plant was originally designed is 400 tph. Twenty to 25 pct of the feed is discharged from the wash plant as waste WSC presently utilizes two independent systems for disposing its mining waste. One system handles that waste material which is 35 mesh by 0 (fine refuse), approximately 15 pct of the total. The second system handles the coarse or 6-inch by 35-mesh refuse.

Fine refuse as indicated in the flowsheet of figure 2 is a combination of cyclone overflow, sieve bin drainage, and drainage from the refuse dewatering screen. This 5- to 6-pct-solids waste is concentrated in a static thickener to a 25- to 30-pct-solids consistency, transferred to a thickener underflow sump, and pumped underground to clarifying lagoons at rates of 12 to 14 tph (solids). The fine refuse typically contains 30 pct ash forming materials and averages 10,000 Btu/lb.

Coarse refuse is the reject material from the jig which has been sized and dewatered on a 5- by 12-foot double deck screen with 1-inch and 35-mesh cloth openings. The surface wet plus 35-mesh waste discharges into a refuse storage hopper for truck loading and transport to the Delta County landfill site. The landfill site is located in an arid desert-like region, 26 miles from the mine. Present coarse refuse disposal costs are approximately $3.67 per ton for loading, transporting, spreading, and covering.

The average hourly production of coarse refuse measured during June, July, and August 1980, equaled 87 tph. The abrasiveness of the coarse refuse as measured by the silica and alumina content (42 and 10 pct respectively) was high.

Water, after being exposed to coarse refuse for 1 week, reached a pH of 8.25. Corrosion of coarse refuse slurry equipment should not be a problem.

Conceptual Design

The proposed waste disposal system will utilize hydraulic transport and pneumatic conveying for transportation and placement of coal wastes underground. The proposed combination of hydraulics and pneumatics utilized the best features, cost and safety, of each transportation system.

The coal waste disposal system has been divided into two subsystems.

1. The above ground system is responsible for material sizing, slurry formation, and slurry pumping. It must fit into the existing wash plant with minimal changes, and permit continuous material handling and storage during emergency situations.
2. The below ground system transports, dewaters, and pneumatically stows waste material. This system fits into the existing mine and mine plan, is flexible enough to permit movement of the dewatering-pneumatic feeding station, and does not interfere with the present mining methods.

The waste disposal system's conceptual design is illustrated in figure 3. Refuse, separated from clean coal in the Baum jig, is removed by draining bucket elevators and is discharged onto a double-deck dewatering screen with 2-inch and 0.5-mm cloths. The 2-inch by 0.5-mm waste discharges into the coarse refuse storage hopper. Plus 2-inch waste is fed to the coarse refuse cone crusher for reduction to a 2-inch top size.

During underground disposal system operation, coarse refuse stowed in the storage hopper is conveyed to the coarse refuse slurry tank where it mixes with water and is pumped underground by a pair of centrifugal pumps.

In a parallel fashion, the fine refuse (thickener underflow) is pumped underground through a separate pipeline. Rationale for two separate pipelines is discussed later.

Below ground, the fine refuse is clarified and dewatered by a solid bowl scroll type centrifuge. Coarse refuse is dewatered on a single deck screen (35-mesh cloth). Effluent from the centrifuge and drainage from the screen are collected in a common sump and returned to the coarse refuse slurry tank at the wash plant. Surplus water drains back to the static thickener closing the water circuit and permitting the eventual capture of the screen drainage solids (minus 35-mesh particles) as thickener underflow. The underground dewatered solids are fed to the pneumatic feeder by conveyor and pneumatically transferred approximately 1,000 feet for placement in the mine.

When required, flow from both lines can be diverted to an underground emergency storage sump to facilitate pipeline drainage.

When the underground mine disposal system is not operational, the coarse refuse belt conveyor's direction of travel is reversed and coarse refuse is discharged into 20-ton trucks for surface disposal. The fine refuse during this period of time will be pumped to existing in-mine fine refuse disposal ponds

Above Ground System

As illustrated in the conceptual design flowsheet (fig 3) by the letters E (existing equipment), M (equipment requiring modifications), and N (new equipment to be purchased) most of the major above ground system components presently exist. Major new components include (1) Flop gate actuator. (2) coarse refuse crusher, and (3) coarse refuse slurry pump

Coarse Refuse Crusher

Present state-of-the-art hydrotransport technology limits particle top size to one-third the pipeline inside diameter Since the mine's refuse has a top size of 5 to 6 inches and the coarse refuse slurry pipeline has a 6-in-ID, crushing of the coarse refuse to a 2-inch top size is required

The comminution device necessary to reduce the run-of-mine refuse at the mine in preparation for hydrotransport meets the majority of the following criteria: (1) Processes highly abrasive material, (2) generates a minimum of fines, (3) has capability to reduce 20,000-psi compressive strength material, (4) reduces minus 8-inch material to 100 pct passing 2 inches, and (5) capacity to be a minimum of 25 tph.

A cone crusher is the only reduction device that addresses the majority of the criteria. A 2-ft-head-diameter cone crusher of the secondary type has been specified for the circuit and is used in conjunction with a scalping mechanism to remove the near size, minus 2 inch, material in its feed The crusher is set at 1 inch and draws approximately 30 hp at 25 tph

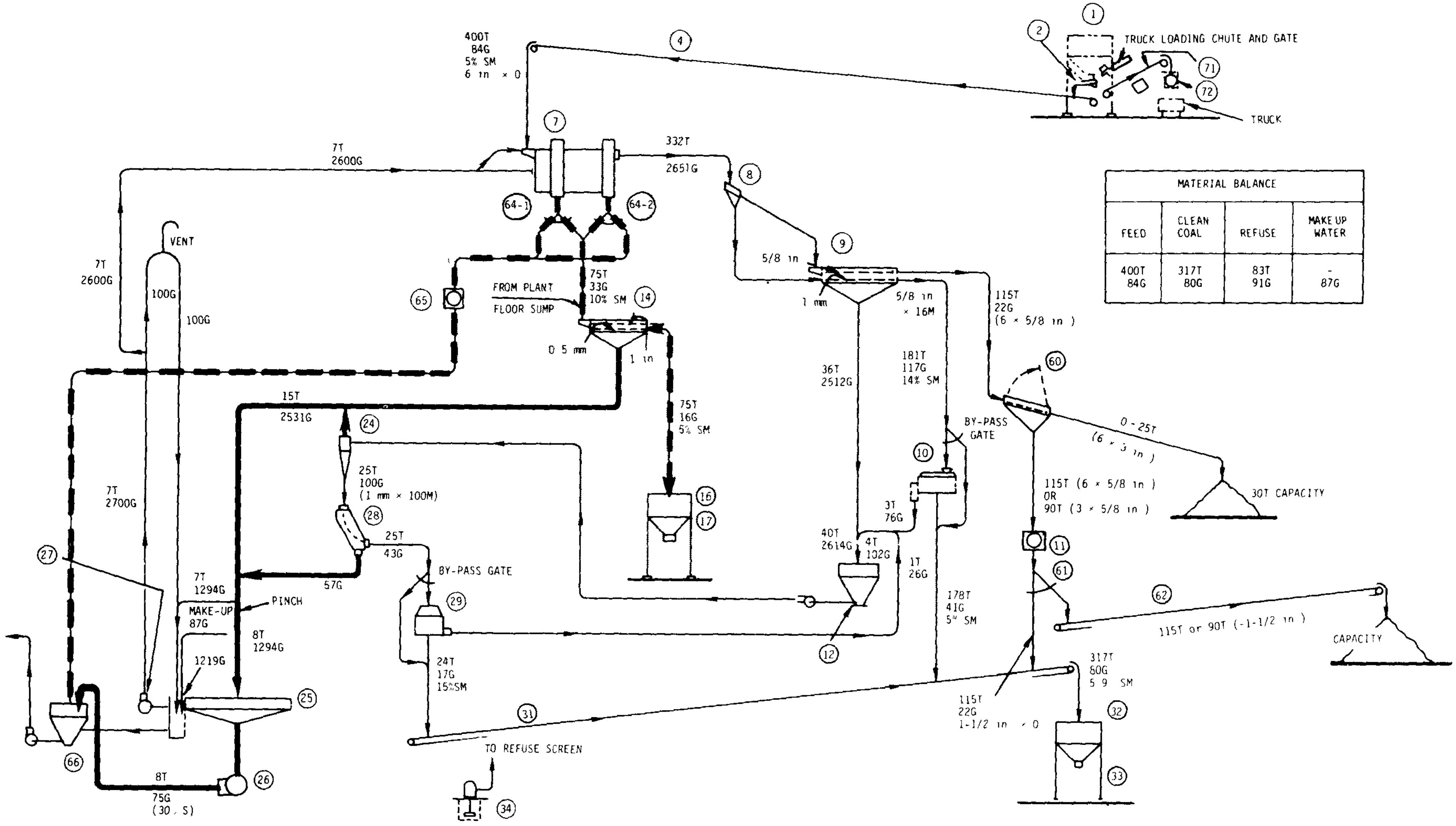
Coarse Refuse Slurry Tank

Thickener underflow at the mine was previously transferred to a thickener underflow sump (2,500 gal) before being pumped underground for disposal. Bypassing the thickener underflow sump and pumping the thickener underflow directly underground frees the thickener sump for use as a coarse refuse tank. The modifications required to complete this tank transformation included: (1) Pumping thickener underflow directly underground, bypassing the thickener sump, (2) provide sump with a jet type inlet for water return, inlet is opposite suction side of the coarse refuse solids pump, and (3) provide overflow baffles and drainage pipe for water return to the thickener. The coarse refuse mixing tank is illustrated in figure 4.

Coarse Refuse Slurry Pump

The pressure required to pump coarse slurry through a given pipeline is a function of slurry flow rate, slurry concentration, slurry specific gravity, particle size distribution, pipe size, length, and friction. Presently there is no general theory which combines these parameters in a correlatable fashion to actual data.

At the (FMA) coarse slurry transport facility, studies were conducted on a 300-ft-long, 6-in-ID pipeline to determine the pressure requirements for coarse slurry pumping. Results of these tests indicated a 4,400-ft-long, 6-in-ID pipeline with a



MATERIAL BALANCE			
FEED	CLEAN COAL	REFUSE	MAKE UP WATER
400T 84G	317T 80G	83T 91G	- 87G

Figure 2.—Preparation plant flowsheet.

KEY

T	TONS PER HOUR
G	GALLONS PER MINUTE
S	SOLIDS
SM	SURFACE MOISTURE
M	U.S. STANDARD SIEVE MESH
1	RAW COAL TRUCK BIN (EXISTING)
2	42- by 84-INCH INCLINED FEEDER
4	30-INCH PLANT FEED CONVEYOR (6 INCHES BY 0)
7	BAUM JIG
8	CLEAN COAL FIXED SIEVE
9	ONE 8- BY 20-FOOT DOUBLE DECK CLEAN COAL SCREEN
10	ONE WEMCO 1300 CENTRIFUGE
11	CLEAN COAL CRUSHER
12	FINE COAL SUMP AND PUMP
4	ONE 5- BY 12-FOOT DOUBLE-DECK REFUSE SCREEN
16	100-TON REFUSE BIN
17	REFUSE BIN GATE
24	THREE 20-INCH CLASSIFYING CYCLONES
25	19-FOOT, HIGH-CAPACITY STATIC THICKENER
26	THICKENER UNDERFLOW PUMP
27	CLARIFIED WATER PUMP
28	FINE COAL SIEVE BEND
29	ONE EBW-36 CENTRIFUGE
31	30-INCH CLEAN COAL STORAGE CONVEYOR
32	CLEAN COAL STORAGE BIN
33	CLEAN COAL BIN GATE
34	PLANT FLOOR SUMP PUMP
60	6- BY 9-FOOT HINGED STATIONARY GRIZZLY
61	DIVERTER GATE
62	24-INCH STOKER CONVEYOR
64–1	DIVERTER GATE
64–2	DIVERTER GATE
65	REFUSE CRUSHER
66	THICKENER UNDERFLOW SUMP AND MINE PUMP
71	42-INCH CRUSHER FEED CONVEYOR
72	CRUSHER (EXISTING)

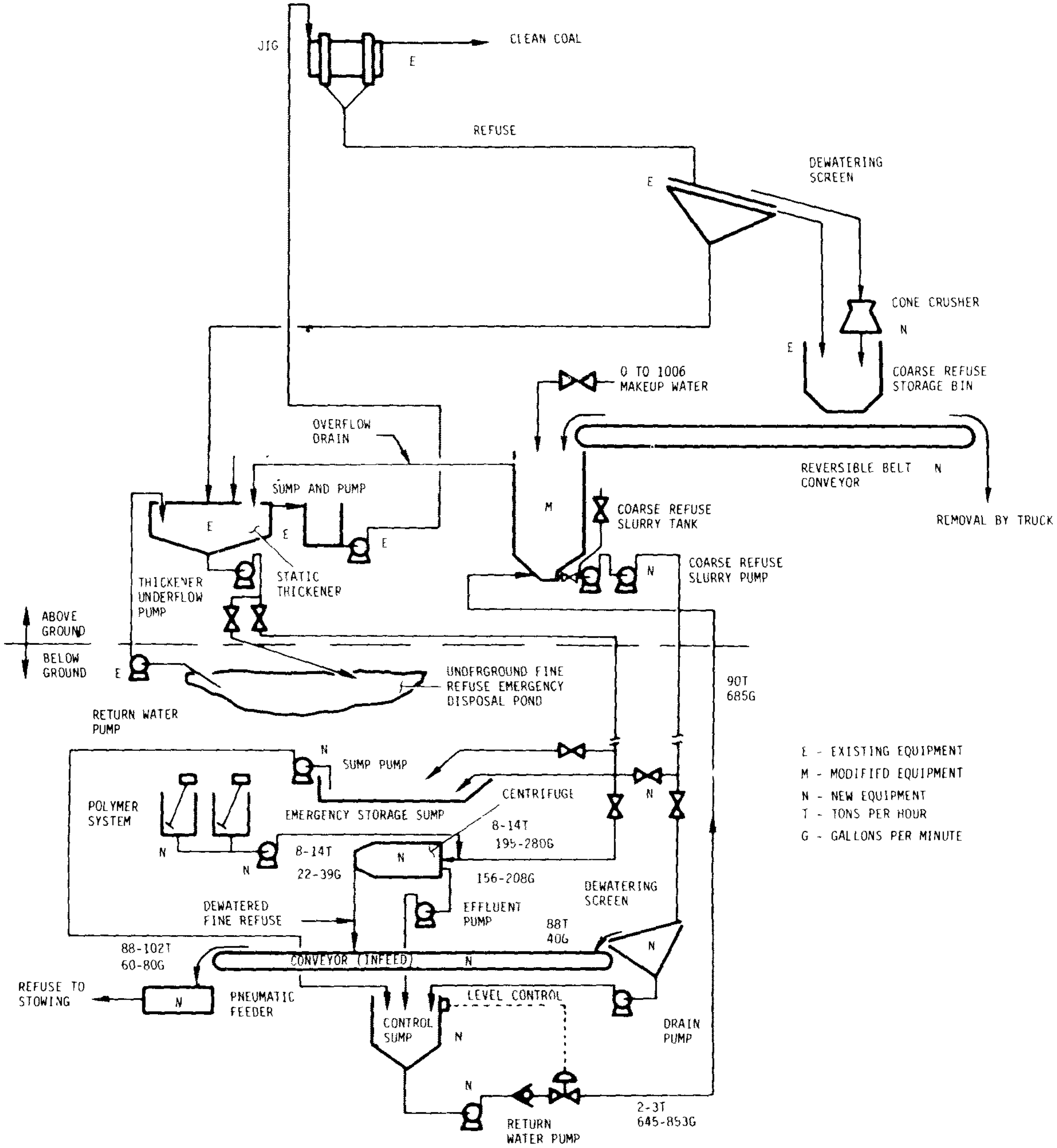


Figure 3.—Underground disposal system conceptual design.

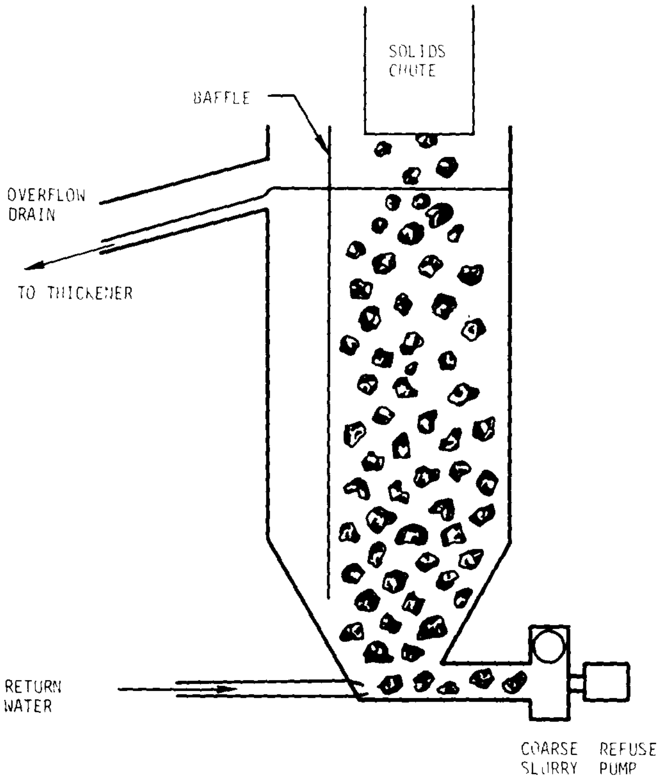


Figure 4.—Coarse refuse slurry tank.

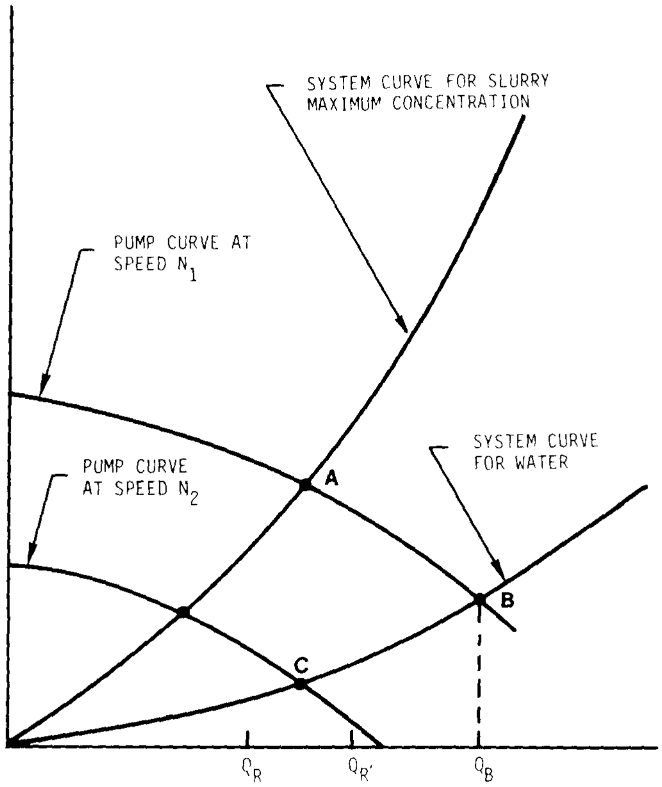


Figure 5.—Pump curve, system curve relationship.

180-foot drop in elevation will require a pump output pressure of 260 psi to hydraulically transport 90 tph of coarse solids.

The pumping system consists of three centrifugal pumps in series driven by one 350- to 400-hp variable speed motor. The pumps are all Ni-hard with wearing plates and suction head liners.

The need for variable speed drive is the result of pumping downhill. Water alone with a 180-foot head will drain from the coarse refuse slurry tank at a rate of 845 gpm. Turning on the slurry pumps at full speed with only water will drastically increase the water flow rate to a level which cannot be handled by the underground water return pump.

Figure 5 illustrates the pump curve superimposed on the system curves for water and maximum slurry concentration. Point A represents the steady-state operating condition for maximum slurry concentration transport. Point B represents the operating condition for water only at the same pump speed (N1). At this condition the water flow rate QB is significantly greater than the water return pump capabilities that are bracketed by flows QR and QR′. To drop the water only flow rate to a condition that can be met by the water return pump, the operating condition must be moved to point C. This can only be attained by reducing the pump speed to N2.

During nonsteady state conditions when the pipeline is being filled or emptied of solids, intersection of the pump and system curve must be maintained between flow rates QR and QR′. This constantly changing condition requires a control loop that monitors the slurry flow rate and adjusts the pump speed accordingly.

Pipelines

The underground waste disposal system utilizes three separate pipelines; one each for the fine slurry, coarse solids' hydraulic transport, and return water. The pipe inside diameters will be 4, 6 and 8 inches respectively. Initially, the pipelines are mild steel construction. Aluminum could be used to reduce pipe weight and simplify the labor requirements for installation. Pipes are joined with Victaulic-type couplings. Slurry pipeline bends have a radius equal to or greater than eight times the pipe diameter. During the first year of operation (phase II), sections of various specialty pipes resistant to wear will be inserted in both slurry lines. Visual and analytical examination of these pipe segments will provide pipeline life-cost ratios

The fine slurry and coarse solids will be handled separately to minimize the fine refuse dewatering equipment and total system cost. Presently, fine refuse is concentrated in the static thickener to a 25 to 30 pct (by weight) solids consistency and discharged at rates of 225 to 250 gpm (12 to 14 tph of solids). If thickener underflow were mixed with the coarse slurry to form a 20 pct (by volume) solids concentration, the anticipated 100 tph of total refuse would require 790 gpm of water. Once below ground at the dewatering station, the combined slurries would report to a horizontal screen for coarse particle removal and dewatering. Assuming little or no particle attrition, the screen drainage would contain 12 to 14 tph of solids and 700 to 750 gpm of water. One centrifuge can separate and dewater thickener underflow solids at slurry rates of 225 to 250 gpm with 12 to 14 tph solids. At solids rates of 12 to 14 tph, with a hydraulic loading of 750 gpm, three to four centrifuges are required for slurry processing The cost of a separate 4-inch pipeline for fine refuse transport, including slurry pump and installation is approximately $70,000. The alternative capital cost of two to three additional centrifuges for the single pipeline system is $500,000 to $700,000. The dual pipeline system is less costly and the space requirements for the underground dewatering station are considerably smaller.

The initial pipelines are 4,400 feet long with a vertical drop of 180 feet from beginning to end The first 200 to 250 feet of pipeline running from the wash plant to the return entry is uphill at a 5-pct grade. Once inside the return entry, the pipeline slopes downhill. In the winter months the outside temperature can reach −30° F. Pipeline freezing is prevented by venting the apex of the pipelines and draining the exposed pipelines back into the wash plant. Pipelines inside the mine need not be buried as the return air is warm enough to prevent freezing.

Underground System

Figure 6 conceptually illustrates the position of the underground system equipment within a crosscut of a room-and-pillar mine. Fine and coarse refuse materials are respectively dewatered by a solid bowl centrifuge and a single deck screen. Centrifuge effluent and screen drainage are transferred to a control sump by low suction head pumps (pumps insensitive to air ingestion). The control sump maintains a sufficient leve of water to prevent pump cavitation. Control of the water level is achieved with a level sensor, controller, and anticavitating control valve on the discharge side of the water control pump The dewatered solids are discharged onto a common conveyor and fed to the pneumatic feeder.

Coarse Refuse

Coarse refuse dewatering is accomplished on a single-deck, horizontal screen. The necessary screening area is a function of particle size distribution, water flow rate, solids flow rate, and cloth opening. For a water and solids flow rate of 1,000 gpm and 100 tph respectively, and a 35-mesh cloth, a 5- by 16-foot screen is sufficient.

The screen has an overall height of 4 feet, 8 inches, the lowest profile screen of all major screen manufacturers. The cloth is fabricated from stainless steel profile wire panels of 1/8-inch grizzly rod construction.

Fine Refuse

Dewatering of fine refuse presents a more formidable problem. Many types of equipment (belt filters, vacuum filters, centrifuges, and filter presses) have been evaluated and utilized both on a laboratory and commercial basis for fine refuse dewatering. Since fine refuse dewatering is conducted underground, equipment height and required floor space are important design criteria. Of the present state-of-the-art dewatering components, the solid bowl scroll type centrifuge is considered most beneficial from a height, floor space, and performance basis.

The solid bowl centrifuge being considered for fine refuse processing has a 36-in-diameter bowl measuring 96 inches in length. Its success in achieving effluent clarity (1,000 ppm or less solids) and its production of a 35- to 50-pct moisture cake has been continually demonstrated

Pneumatic System

The pneumatic system has been designed to transport 100 tph of mining waste with 10 to 15 pct moisture for a distance of 1,000 feet. Components that comprise this system include: (1) Infeed conveyor, (2) airlock feeder, (3) hydraulic power pack, (4) blower, (5) pipeline, and (6) deflector

The infeed conveyor is a chain flight conveyor driven by a variable speed hydraulic motor The conveyor speed is varied automatically in response to the pneumatic system back pressure. The feeder is an eight-pocket, rotary airlock feeder drive by a slow-speed, high-torque hydraulic motor Hydraulic power for both these components is provided by an electrically driven hydraulic power pack. This power pack is equipped with relief valves, controls, and pressure sensing switches for measuring and correcting equipment overload.

The blower is electrically driven by a 750-hp motor and is provided with acoustical protection such as filters, air chamber baffles, and an acoustically clad housing. The pipeline is 10-inch Scd. 40 mild steel pipe capable of conveying 200,000 to 300,000 tons of semiabrasive shale Use of lightweight pipes at the end of the pneumatic pipeline will facilitate pipe movement

Evaluation of Disposal System

Monitoring Material Flow

During the first year of operation the waste disposal system will be equipped with instrumentation for monitoring material flows and system performance. All online continuous measurements will be documented on a multichannel chart recorder. Those variables to be monitored and the type of instrumentation which will be utilized are summarized in table 1.

Periodic sampling around individual components will provide further equipment and system evaluations Those components and associated streams which will be periodically analyzed include: (1) Solid bowl centrifuge-feed, cake, and effluent, (2) dewatering screen—solids and drainage, and (3) crusher-feed and product.

Sampled streams will be primarily analyzed for moisture content and size distribution. As content, British thermal unit value and sulfur content will be included in the analysis on a less regular basis.

Effects on Mine Operation

In general, it appears that installation of the proposed disposal system will not materially affect the overall operation of the mine. System components located in the wash plant and on the surface will be operated by the wash plant crew as part of the normal operating shift. Underground equipment will be operated by two additional miners. Productivity on an overall mine basis will, however, be virtually unaffected as

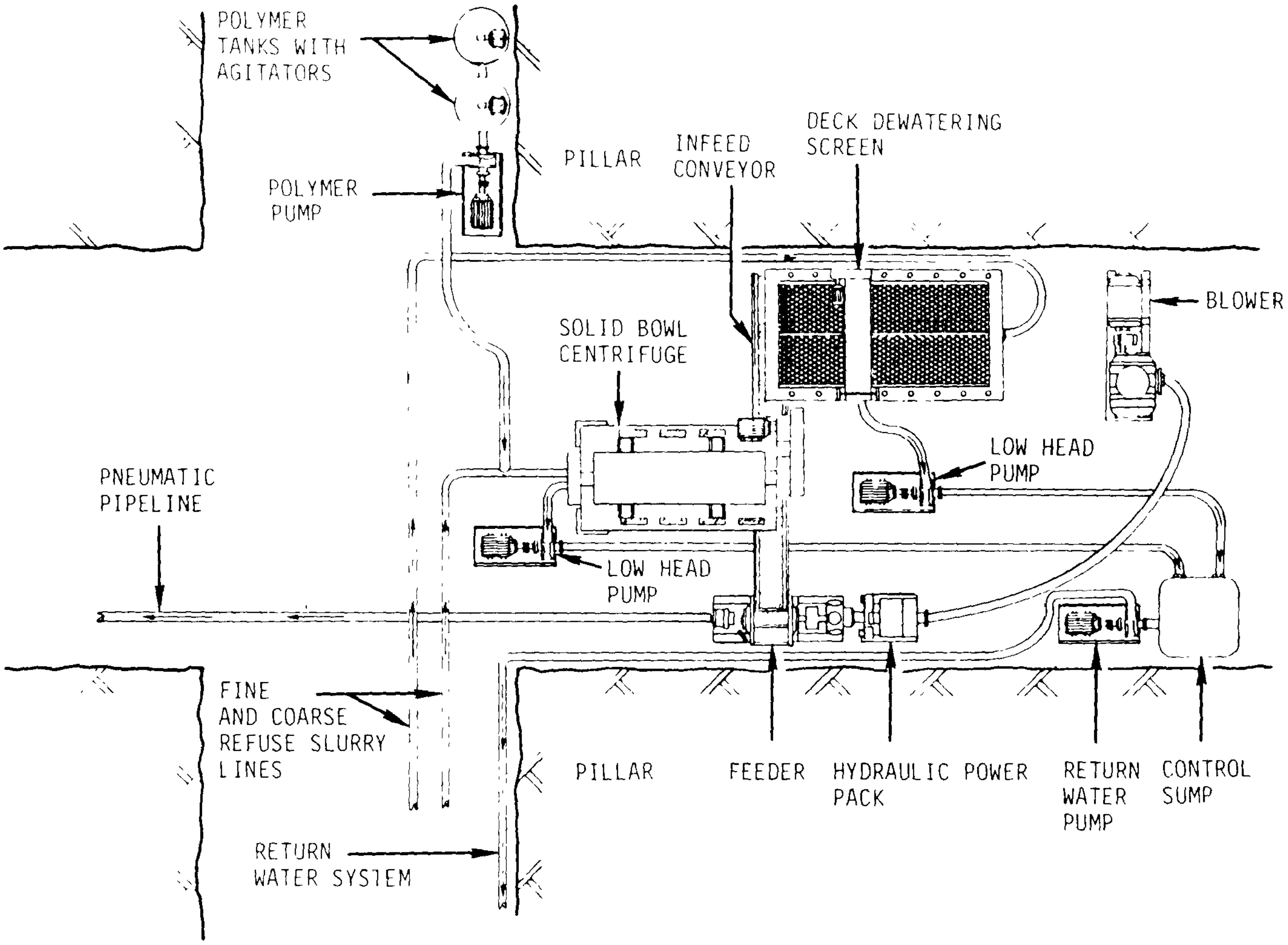


Figure 6.—In-mine dewatering-pneumatic feeding system.

the two men operating the dewatering-pneumatic system will be replacing mine-waste truck drivers.

The pipelines are small and will not interfere with coal haulage. The dewatering-pneumatic feeding station will be in a vacant crosscut; thus, it will not interfere with ventilation or material transport. Within the disposal section the pneumatic pipeline will be handled by the additional pneumatic equipment operator and a part-time swing man from the mining crew.

In-mine waste disposal at the HNM may improve coal recovery as present coal recovery in room and pillar operations averages 50 to 75 pct. WSC anticipates future efforts with longwalls will increase coal recovery to 85 or 90 pct.

Effects on Mine Personnel

Compared with shuttle cars, battery scoops, and conveyors, hydraulic haulage and pneumatic conveying are believed to offer the safest method for transporting refuse. Waste is

TABLE 1.—System instrumentation

Variable	Instrument
Coarse refuse slurry flow rate Fine refuse slurry flow rate Return water flow rate	Magnetic flow meter and ultrasonic flow meter
Coarse refuse slurry pump discharge pressure Fine refuse slurry pump discharge pressure	Online strain gage type pressure transducer
Coarse refuse slurry concentration Fine refuse slurry concentration	Nuclear density gage or weight activated strain gage density meter.
Coarse refuse slurry pump power consumption Pneumatic system power consumption	Watts transducers

TABLE 2.—System operation measurements

Measurement	Method
Physical:	
Density of material as placed	In-place sampling
Roof support	Preloaded strain gage roof bolts
Subsidence	Surface measurements
Compressibility	In-place sampling
Environmental	
Dust	GCA-RAM 1 dust monitor
Noise	Sound-level meters
Ventilation	Anemometer
Hydrology Water quality and flow	pH meter and laboratory analysis only required if water discharge is significant

contained within a pipeline and no moving parts can interfere with mining personnel.

Backfilling with pneumatics can generate dust in two ways. The rock being conveyed is broken by impact on pipe transitions and in-line collisions, and high velocity air ejected from the nozzle at the end of the pipeline can raise dust from the roof, floor, and ribs. By combining pneumatics with hydraulic haulage and controlling the dewatering before pneumatic conveying (combined refuse moisture of 10 to 12 pct), refuse generated dust can be substantially reduced. Dust created by high-velocity air and solids impacting dry surfaces will have to be controlled Dual split ventilation will most likely be used to control the dust if necessary.

The pneumatic blower is a source of noise It is acoustically clad and located around a corner from the dewatering-pneumatic feed control station. At the discharge nozzle of the pipeline, the airstream is directed away from the operator and when the system is fully loaded, the discharge is relatively quiet. In any event, it is not necessary for the operator to be in constant attendance at the discharge nozzle. Noise generated by the remaining underground equipment will be monitored during Phase II. If noise levels are excessive, steps will be taken to correct this situation

Effects on Mine Conditions

The mine may be affected by the backfill operation in three separate categories: (1) Physical stability, (2) environmentally, and (3) hydrologically.

During Phase II, specific measurements will be taken periodically before and during actual system operation These measurements and methods of obtaining them are listed in table 2.

Outlook for the Foster-Miller Associates/Western Slope Carbon, Inc., Design

The information to be gained from this joint Bureau of Mines and WSC program will benefit all underground coal producers. Operation of the combined system of hydraulics, pneumatics, and underground dewatering will provide operating experience in most transport and stowing mechanisms which can be utilized for underground disposal of mining waste. The benefits of such a system are numerous. Environmental improvements will result in the elimination of:

1. Unsightly surface disposal sites
2. Smoke and fumes from burning waste heaps.
3. Dust brought into the air by wind erosion.
4. Muddy acid runoff water

In addition to these improvements, there are numerous potential mining benefits·

1. Reduction in disposal costs.
2. Improved coal recovery and mine productivity
3. Improved roof control.
4. Reduction in subsidence.

The economics and benefits of underground disposal of mining waste are site specific and will vary from mine to mine The potential gains are encouraging and all underground coal producers owe it not only to the general public but to themselves to evaluate the potential for disposing mining wastes underground.

FACTOR-OF-SAFETY CHARTS FOR ESTIMATING THE STABILITY OF SATURATED AND UNSATURATED TAILINGS POND EMBANKMENTS

by

D. R. Tesarik[1] and P. C. McWilliams[2]

ABSTRACT

The factor of safety, the traditional measure of stability for earth embankments, is presented graphically in factor-of-safety charts. Factor-of-safety contours are drawn for homogeneous earth embankments for two contrasting situations: no phreatic surface and a phreatic surface that assumes 10 percent freeboard. The factor of safety can be read directly from the charts if the physical properties of the soil and the geometry of the embankment are known. The curves provide a quick first approximation of the "most stable" and "least stable" condition of the embankment. The factor-of-safety calculations were done by the Simplified Bishop Method of Slices and a series of least-squares curve fitting steps were used to present the factor of safety in final chart form.

Introduction

The exact determination of the factor of safety for a soil or tailings embankment generally requires an extensive physical property sampling program and the use of a computer model. Precalculated slope stability charts assuming homogeneous materials and uniform conditions provide a quick and easy method of approximating the factor of safety.

The Bureau of Mines has developed factor-of-safety charts for estimating the stability of tailings pond embankments that are easy to read, contain a wide range of slope angles, consider pore water pressure, and cover a wide range of internal friction angles.

This paper discusses slope stability charts in general, the computer model that was used to calculate the factor of safety from the Simplified Bishop equation, and the procedure used to construct the charts. Several examples of how to use the charts and the assumptions accompanying their use are included. The charts (examples appear in the appendixes[3]) provide estimates of stability for personnel responsible for judging the stability of soil and tailings embankments in the mineral industry.

Slope Stability Charts

Slope stability charts reduce a multidimensional problem that includes the following listed parameters into a two-dimensional graphic display for quick and easy reference:

F = factor of safety.
γ = unit weight of the soil in lb/ft^3.
H = height of the embankment in feet.
ϕ' or ϕ = internal friction angle in degrees[4].
c' or c = cohesion in lb/in^2 or lb/ft^2.
$r_u = \frac{u}{\gamma h}$ = pore pressure ratio at a point, where u is the pore pressure at that point and h is the depth of the point below the soil surface.
β = slope of the embankment in degrees, sometimes expressed as increments in the x and y directions respectively; for example, 2:1 = 26.57°.
$D = \frac{L}{H}$ = depth factor where L is the distance from the top of the embankment to the stiff base (fig. 1).[5]

[1] Mathematician
[2] Mathematical statistician
Both authors are with the Spokane Research Center, Bureau of Mines, Spokane, Wash.

[3] The complete set of charts are in BuMines RI 8564, "Factor of Safety Charts for Estimating the Stability of Saturated and Unsaturated Tailings Pond Embankments," (in press)
[4] The prime (') symbol indicates that the parameter is in terms of effective stress
[5] Values of F for D = 1 25 were calculated but have been eliminated from the charts Investgation of 2,409 values of F shows that values of F for D = 1 50 are more critical (than for D = 1.25) 81 percent of the time When values of F for D = 1 25 are more critical, the difference is usually in the third decimal place

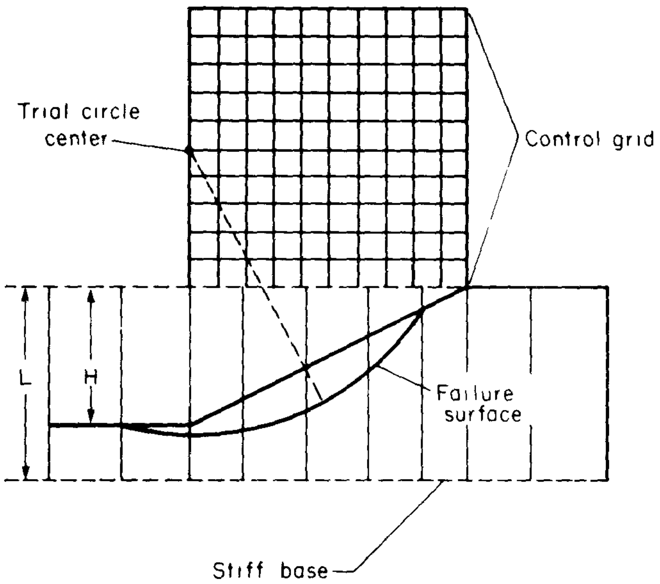


Figure 1.—A typical embankment cross section and failure surface.

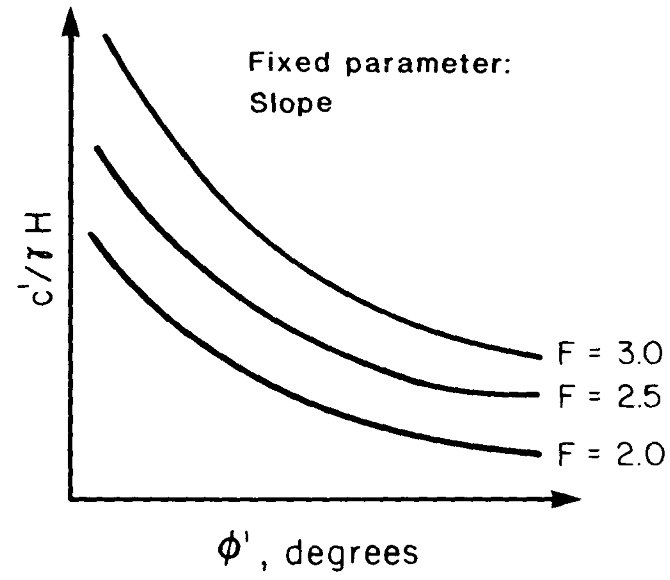


Figure 2.—Factor of safety, F, contours, Singh format.

Previous authors have combined several parameters into different forms such as $c'F\gamma H$ (11),[6] $c/\gamma H\tan\phi$ (2), or $c/\gamma H$ (6–10). From the available literature reviewed, the chart format presented by Singh (10) for dry embankments is probably the easiest format to use if the factor of safety is desired, given that all other parameters are known (fig. 2). The charts in this publication employ a "Singh-like" format for both dry and saturated embankments.

Data Processing

The computer program (1) used to determine the minimum factor of safety for given slope conditions is based on Bishop's Simplified Method of Slices. The program divides the slope cross section into slices. The number of slices is approximately equal to the number input by the user. This number may differ slightly from the final value used because of the boundary conditions used by the program. Eighty slices were input for the analysis used in this paper. The failure surface is assumed to be an arc of a circle with a minimum penetration depth equal to 20 percent of the embankment height (for example, 20 feet for a 100-foot-high embankment).

The slice boundaries are determined before any trial circles are analyzed, so the number of slices in each sliding mass varies with the radius of the trial circle. A sample failure surface and slices are shown in figure 1.

For each set of parameters input, slope geometry, ϕ', c', and γ, 1,331 trial circles were examined to determine the critical circle. This was accomplished by using the control grid option of the program (fig. 1). At each point on the control grid, the program computes a minimum radius; that is, a radius that will just touch the slope and a maximum radius based on the geometry and boundary conditions of the slope profile. A series of factors of safety is computed starting with a radius slightly smaller than the maximum radius. Each successive radius is reduced by approximately one-eleventh of the difference between the minimum and maximum radii (1).

[6] Underlined numbers in parentheses refer to items in the list of references preceding the appendixes

The Simplified Bishop Equation

The equation which is solved to determine F is the simplified Bishop equation (equation 1). Slice geometry and the forces on a typical slice are shown in figure 3

$$F = \frac{1}{\Sigma\, W_i \sin \alpha_i} \Sigma\{c'b_i + W_i\,(1 - r_{u_i})\tan\phi'\} \frac{\sec \alpha_i}{1 + \dfrac{\tan\phi' \tan\alpha_i}{F}} \qquad (1)$$

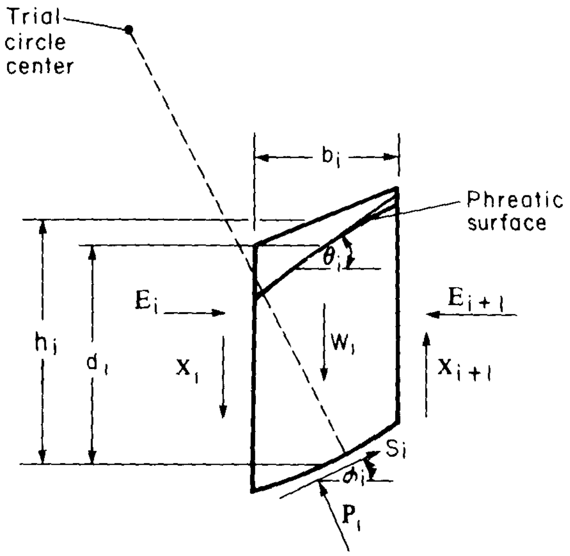
where

c' = cohesion of soil,
b_i = breadth, i^{th} slice,
ϕ' = angle of internal friction, effective stress,
W_i = weight, i^{th} slice,
r_{u_i} = pore pressure ratio, i^{th} slice,

$$r_{u_i} = \frac{u_i}{\gamma h_i},$$

where u_i = pore pressure, i^{th} slice,

$= 64.4 \cdot d_i \cdot \cos^2 \theta_i$ for steady-state seepage,



b_i = Breadth
E_i, E_{i+1} = Horizontal forces
X_i, X_{i+1} = Resultant vertical shear forces
W_i = Weight
S_i = Shear force
P_i = Total normal force
α_i = Angle of tangent to arc
θ_i = Angle of tangent to phreatic surface
d_i = Distance from phreatic surface to slice base
h_i = Height of slice

Figure 3.—Forces on a typical slice.

where

d_i = vertical distance from the slice base to the phreatic surface at the midpoint, i^{th} slice,
θ_i = angle of the tangent to the phreatic surface at the midpoint, i^{th} slice,
γ = density of soil,
h_i = height, i^{th} slice,

and

α_i = angle of tangent to arc, i^{th} slice.

The equation is derived by summing moments about the trial circle center. The derivation is given in a paper presented by Bishop (3) and will not be shown here.

The equation is called "simplified" because the respective summations of the horizontal side forces (E_i) and the vertical shear forces (X_i) are assumed to be 0 (3), thus:

$$\Sigma(X_i - X_{i+1}) = 0, \tag{2}$$

$$\Sigma(E_i - E_{i+1}) = 0. \tag{3}$$

Construction of the Factor-of-Safety Contours With No Phreatic Surface

Bishop and Morgenstern have shown that if r_u, $c'/\gamma H$, and ϕ' are held constant, then the factor of safety is a function

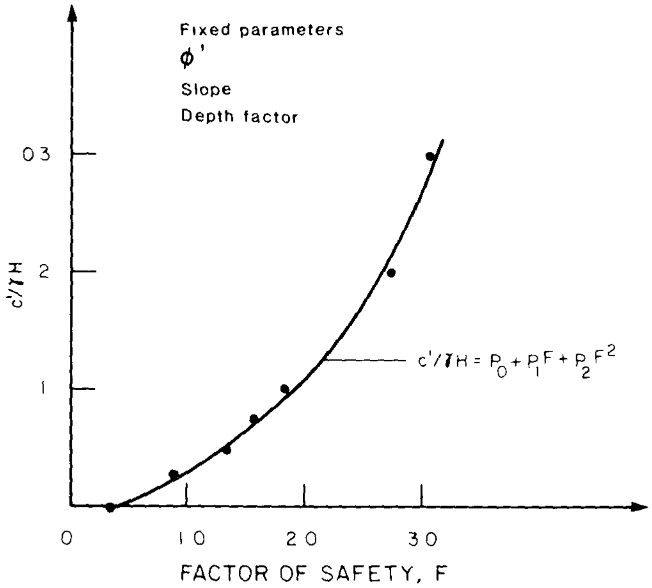


Figure 4.—$c'/\gamma H$ expressed as a function of factor of safety, F, for ϕ, slope and depth factor, D, fixed.

of the geometry of the embankment and sliding mass (4). This is seen by multiplying equation 1 by (H·H) (H·H) and expressing W_i as $\gamma \cdot b_i \cdot h_i$. The resulting equation is

$$F = \frac{1}{\sum \frac{b_i}{H} \cdot \frac{h_i}{H} \cdot \sin \alpha_i} \sum \left\{ \frac{\left[\frac{c'}{\gamma H} \cdot \frac{b_i}{H} + \frac{b_i}{H} \cdot \frac{h_i}{H} \cdot (1 - r_u) \tan \phi'\right] \sec\alpha_i}{1 + \frac{\tan \phi' \tan \alpha_i}{F}} \right\} \tag{4}$$

Note that $r_u = 0$ for embankments containing no water. By inspecting equation 4, it is obvious that the critical terms F, $c'/\gamma H$, and ϕ' are rather entwined, and some manipulation is required to "free" these quantities. In order to express $c'/\gamma H$ as a function of ϕ' as desired in the final charts, the first step entails fitting a second-order polynominal to F versus $c'/\gamma H$—$c'/\gamma H = P_0 + P_1F + P_2F^2$, where P_0, P_1, and P_2 are the equation's coefficients as shown in figure 4. Values of $c'/\gamma H$ were chosen as 0.0, 0.025, 0.05, 0.075, 0.1, 0.2, and 0.3. The values in the range of $0.0 \leq c'/\gamma H \leq 0.1$ were chosen to be consistent with previously published literature (4, 9). In order to attain a factor of safety of at least 3.0 (a predetermined minimum F to display on the charts), the ratio was extended to include 0.2 and 0.3. In some cases, values of 0.4 and 0.6 were used.

The information contained in the family of plots of figure 4 was then rearranged in the more convenient format of "Singh-charts." Factor-of-safety values were contoured with the dependent variable still $c'/\gamma H$, but ϕ' became the new independent variable. The rearrangement of information was achieved by:

1. Fixing F at a desired contour value (say F = 1.1).
2. Selecting ϕ' to correspond to a member of the "ϕ'-fixed"

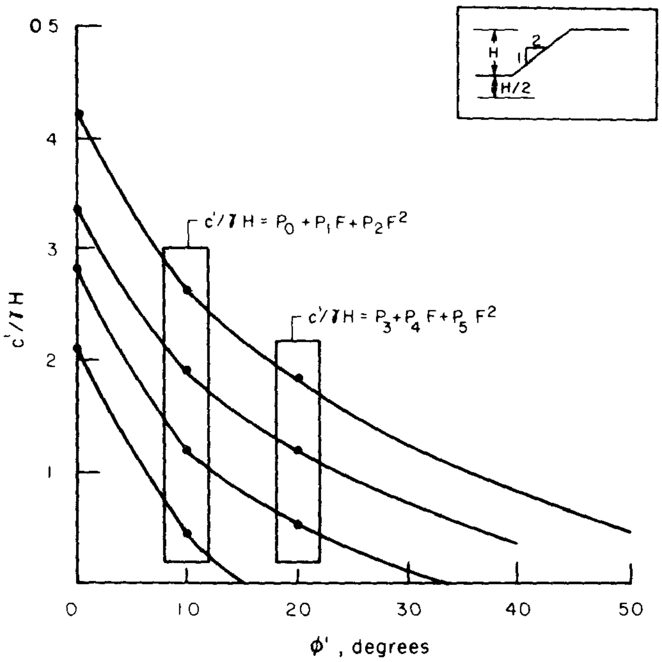


Figure 5.—Factor of safety contours with slope and depth factor as overriding parameters.

graphs of figure 4 and substituting the F-contour value into the quadratic equation, giving the appropriate c'/γH value. The F-contour value is this ϕ', c'/γH pairing.

3. Varying ϕ' over its domain of values, giving a complete curve for this F-contour.

4. Setting F = "next desired contour value" and repeating steps 1 through 3. Thus, the final product (fig. 5) is a Singh chart with fixed overriding parameters of slope and depth factor, D.

Construction of the Factor-of-Safety Contours with a Phreatic Surface

To construct charts containing pore pressure, the value r_{u_i} in equation 4 must be considerd since it no longer has a value of 0. For F to depend on geometry alone, r_{u_i} must be constant along with the ratio c'/γH. The following alternatives were considered in dealing with pore pressure.

1. Let the user calculate an average r_u for the entire embankment. Bishop and Morgenstern have devised an averaging technique that has been shown to be accurate in many problems (4).

2. Assume a fixed phreatic surface in which the headwater is a distance of H/10 feet from the top of the embankment—called 10 percent freeboard (fig. 6). Recall that $r_{u_i} = u_i/(\gamma \cdot h_i) = 62.4 \cdot d_i \cdot \cos^2\theta/(\gamma \cdot h_i)$. By holding γ constant and substituting $62.4 \cdot d_i \cdot \cos^2\theta/(\gamma \cdot h_i)$ into equation 4, it is seen that F is a function of the geometry of the embankment and sliding mass alone, provided that the ratio c'/H is also held fixed.

Regarding option 1, it was found that for the case of homogeneous embankments with a phreatic surface due to steady state seepage, the values of F obtained when using an average r_u value differed, in some cases, significantly from the value of F obtained when r_u took on its corresponding values for each slice. Further, the value of F was sometimes quite sensitive to small changes in r_u, necessitating the calculation of r_u to the nearest hundredth place for accuracy. This would require either a large number of charts with r_u in increments of 0.01 or interpolation by the user.

Because of these reasons, and in order to eliminate the calculation of an average r_u, the second alternative—to make γ a fixed parameter and limit the steady-state seepage charts to the condition of 10 percent freeboard—was chosen.

The method used for generating the curves containing pore pressure is the same as described in the preceding section, except that c'/H is now plotted on the y axis instead of c'/γH since γ is now a fixed parameter. The location of the phreatic surface for input into the Bishop computer code was determined by running a finite-element program. Figure 6 illustrates the phreatic surface for an homogeneous slope of 2.5 to 1.

A natural question is whether linear interpolation is valid for the charts—for example, if γ = 95, what does one do? Because of the discreteness of the Bishop process, one does not find a smoothness between points. Thus, interpolation should be done with caution, particularly if F varies significantly from chart-to-chart.

Using the Slope Stability Charts

FOR EMBANKMENTS WITH NO PHREATIC SURFACE

If the factor of safety is desired for an embankment with no phreatic surface, the following steps are necessary:

1. Calculate c'/γH.

2. Determine the slope for the embankment and if the embankment is constructed on a stiff base.[7] If the embankment is on a stiff base, use the charts such as shown in appendix A. If the stiff base is approximately H/2 feet below the embankment, use the charts such as shown in appendix B.

3. Locate the appropriate chart by using the information in the upper right-hand corner of the chart.

4. Find where the ordered pair ϕ', c'/γH intersects the factor-of-safety contours. This is the critical factor of safety desired.

For Embankments With 10 Percent Freeboard

1. Calculate c'/H.

2. Determine the slope for the embankment, the density of the soil, and if the embankment is constructed on a stiff base. If the embankment is on a stiff base, use the charts such as shown in appendix C. If the stiff base is approximately

[7] A stiff base is assumed when the shear strength of the material on which the embankment is constructed is higher than the shear strength of the material in the embankment

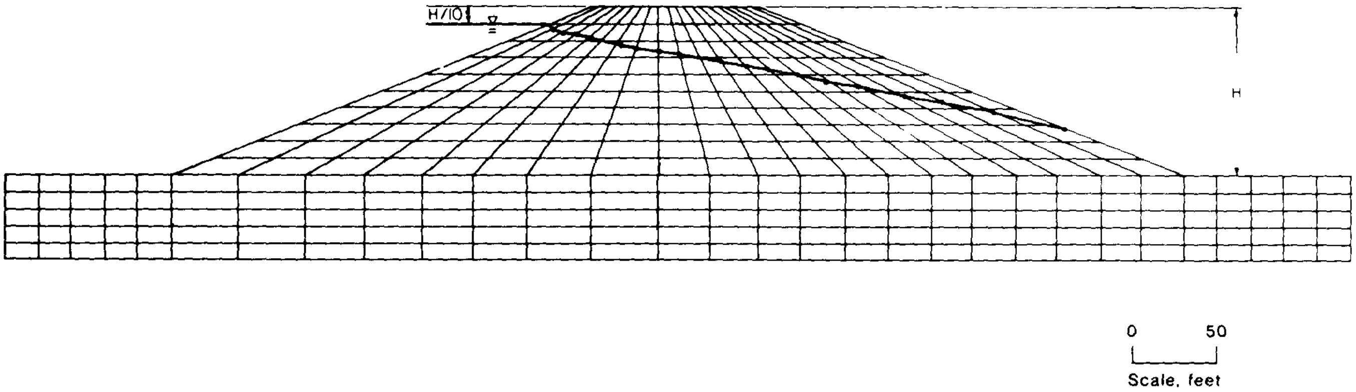


Figure 6.—Phreatic surface computed by the finite-element method.

H/2 feet below the embankment, use the charts such as shown in appendix D.

3. Locate the appropriate chart by using the information in the upper right-hand corner of the chart.

4. Find where the ordered pair ϕ', c'/H intersects the factor-of-safety contours to determine the critical safety factor.

Example 1

The following physical parameters are part of the data collected by the Bureau of Mines from West Virginia coal refuse embankments (5).

$$\gamma_{natural} = 90.74 \text{ lb/ft}^3$$
$$\gamma_{saturated} = 101.12 \text{ lb/ft}^3$$
$$c' = 3.2 \text{ lb/in}^2,$$
$$\phi' = 33.48°.$$

If an embankment 100 feet high with slope 1.5 to 1 and no phreatic surface is to be constructed on a stiff base, the factor of safety is obtained as follows:

$$1. \frac{c'}{\gamma H} = \left(3.2 \frac{\text{lb}}{\text{in}^2}\right) \cdot \left(144 \frac{\text{in}^2}{\text{ft}^2}\right) \div \left(\left(90.74 \frac{\text{lb}}{\text{ft}^3}\right) \cdot \left(100 \text{ ft}\right)\right) = 0.051.$$

2. The slope is 1.5 to 1 and the embankment is to be constructed on a stiff base.

3. The appropriate chart has 1.5 to 1 in the upper right-hand corner (fig. A–2). The ordered pair (ϕ', c'/γH) = (33.48°, 0.051) lies close to the midpoint between the contour lines of F = 1.6 and F = 1.8, yielding a factor of safety of approximately 1.7

To verify the result, program Bishop was run for this case. The result (F = 1.729) verifies the factor of safety obtained via the charts.

Example 2

Suppose an embankment with the same physical properties and of the same geometry as in example 1 is to be constructed; however, a phreatic surface at 10 percent freeboard is anticipated. The following steps are taken:

$$1. \frac{c'}{H} = \left(3.2 \frac{\text{lb}}{\text{in}^2}\right) \cdot \left(144 \frac{\text{in}^2}{\text{ft}^2}\right) \div 100 \text{ ft} = 4.61 \frac{\text{lb}}{\text{ft}^3}$$

2. The slope is 1.5 to 1 and the density is rounded to 100 $\frac{\text{lb}^8}{\text{ft}^3}$.

3. The appropriate chart has γ = 100, and 1.5 to 1 in the upper right-hand corner (fig. C–2). The factor of safety corresponding to the ordered pair (33.48°, 4.61 $\frac{\text{lb}}{\text{ft}^3}$) is approximately 1.18.

The results obtained from running the computer code for this specific case yield 1.183 as a factor of safety. The difference between the chart value for F and the value obtained from the individual computer run can be attributed to a combination of the following:

1. Variation introduced by the procedure used to construct the charts.

2. Variation introduced by using 100 $\frac{\text{lb}}{\text{ft}^3}$ instead of 101.12 $\frac{\text{lb}}{\text{ft}^3}$.

Note the impact of using the phreatic surface—the factor of safety is substantially reduced from 1.73 to 1.18.

Summary

The factor of safety representing critical circles for homogeneous earth slopes is displayed in an easy-to-use plot form for dry embankments and saturated embankments having 10 percent freeboard. The factor of safety can be read directly from the charts if the physical properties of the soil and geometry of the embankment are known. This form was achieved by organizing the factor-of-safety values using a

[8] A comparative computer run was made using 90 74 lb/ft³ for density above the phreatic line and 101.12 lb/ft³ for density below the phreatic line The factor of safety was 1.205.

series of least-squares curve fit routines. The soil parameters and slope boundaries include:

Slope	1:1, 1.5:1, 2:1, 2.5:1, . . ., 5:1
Effective angle of internal friction, ϕ'	0° to 50°
$c'/\gamma H$ (dry embankments)	0.0 to 0.6 (dimensionless)
c'/H (wet embankments with 10 percent freeboard)	0.0 to 80.0 (lb/ft^3)
Depth factor, D	1.00, 1.50

The charts should be a useful tool in initial embankment design or field work where detailed stability analysis is not imperative. However, the charts are not meant to replace the use of slope stability models when time and resources are available.

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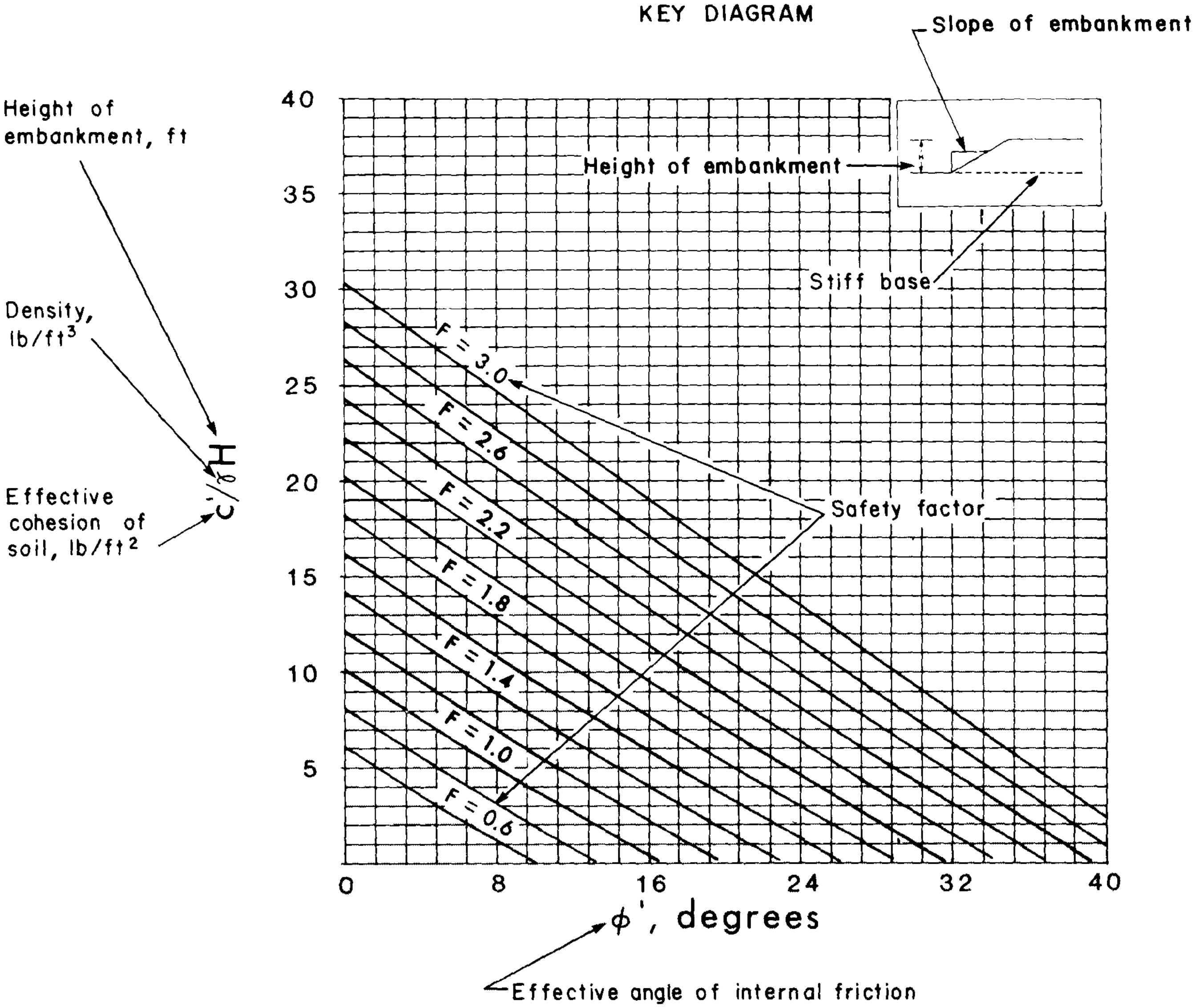

Figure A–1.—Example stability charts for embankments with no phreatic surface and depth factor = 1.0.

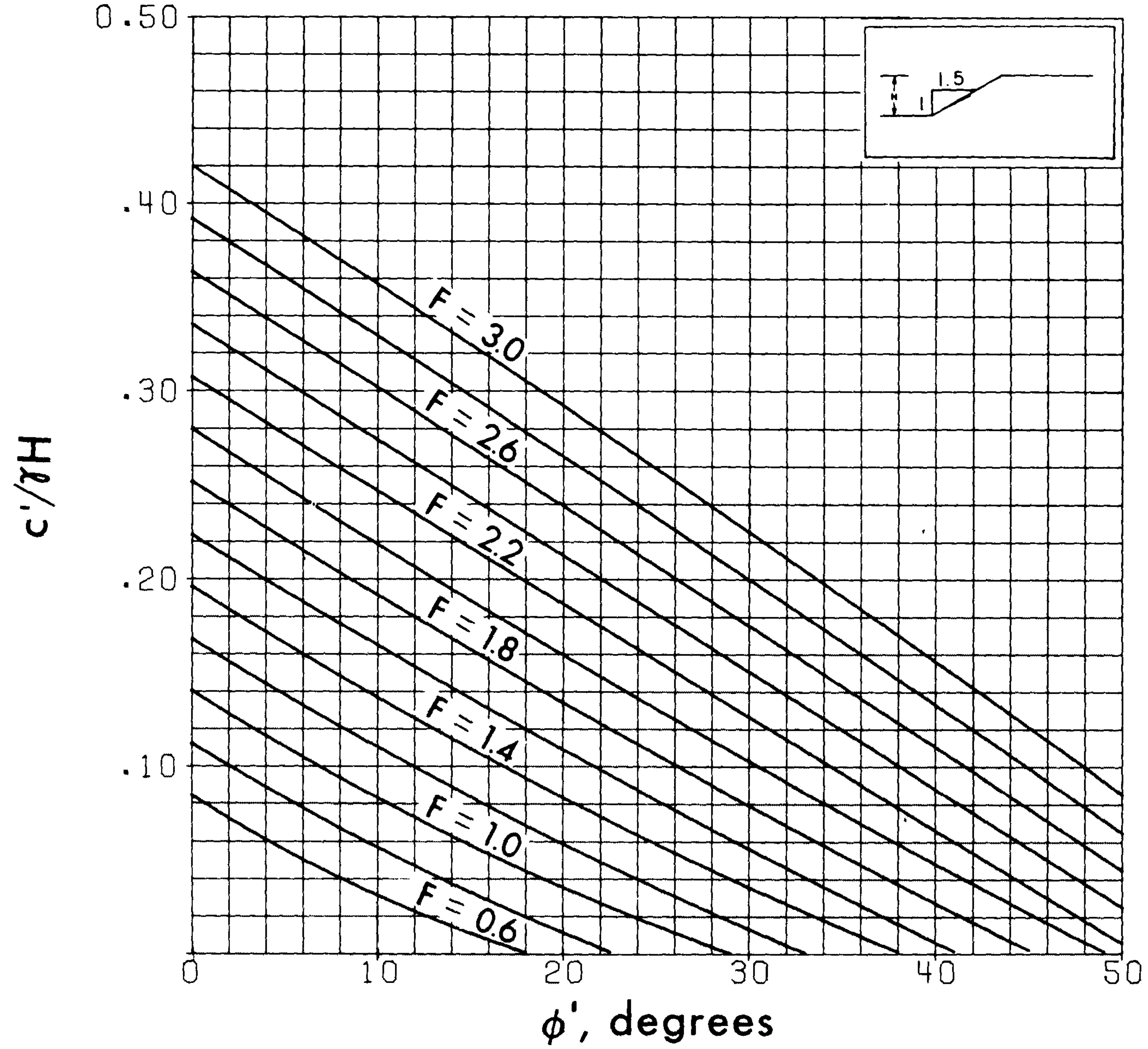

Figure A-2.—Factor of safety—1.5:1 slope, no phreatic surface, D = 1.00.

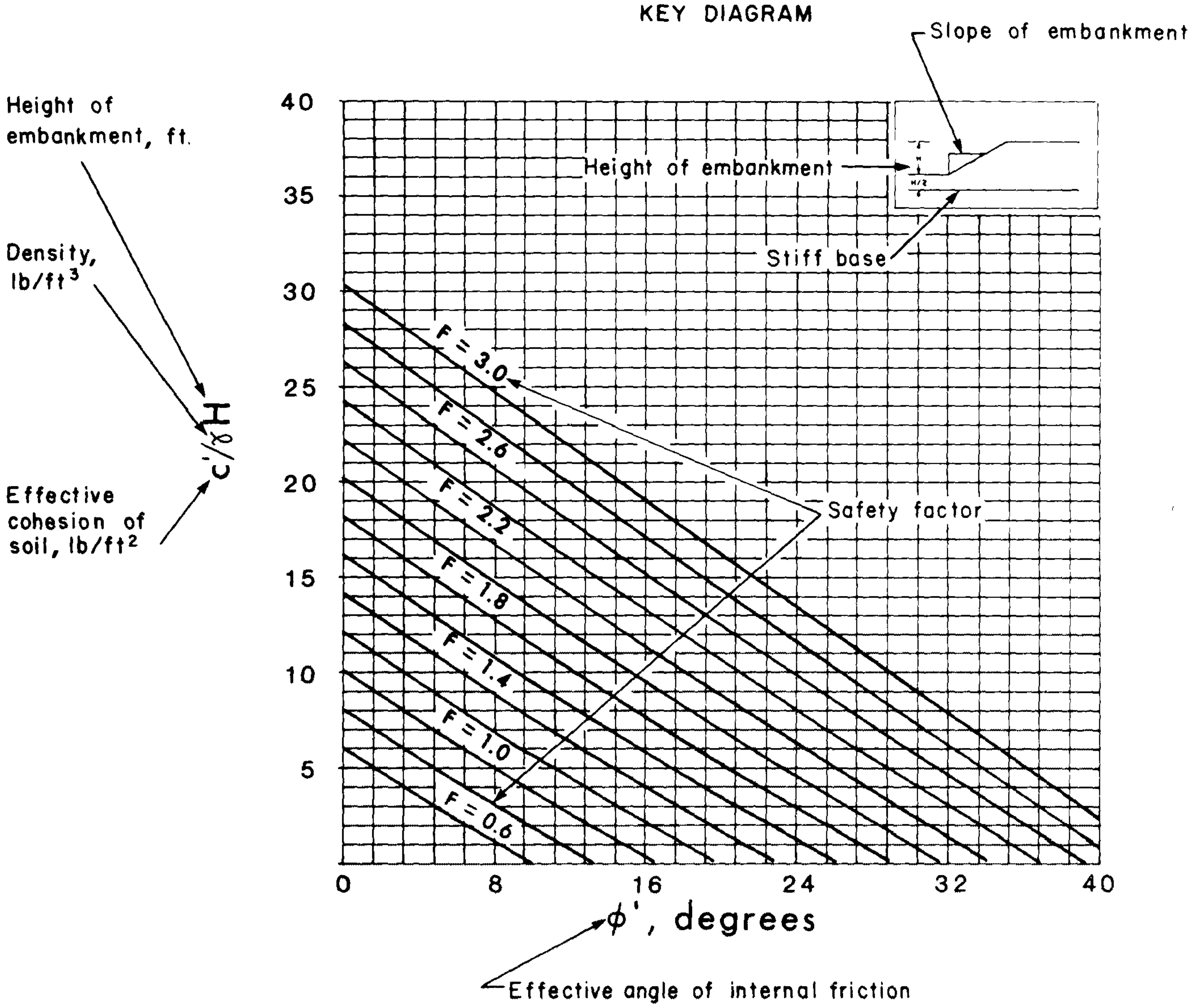

Figure B–1.—Example stability charts for embankments with no phreatic surface and depth factor = 1.50.

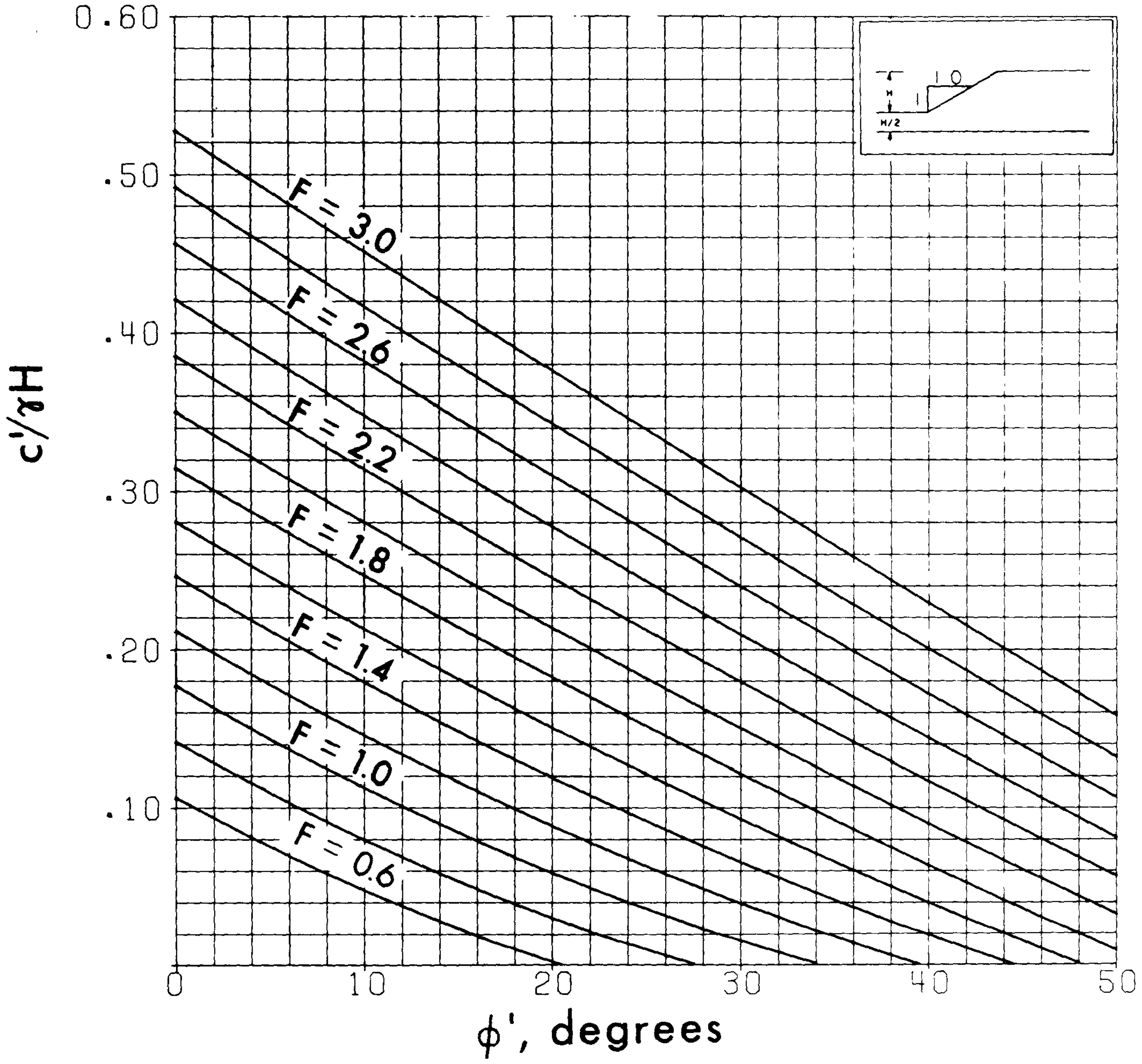

Figure B–2.—Factor of safety—1.0:1 slope, no phreatic surface, D = 1.50.

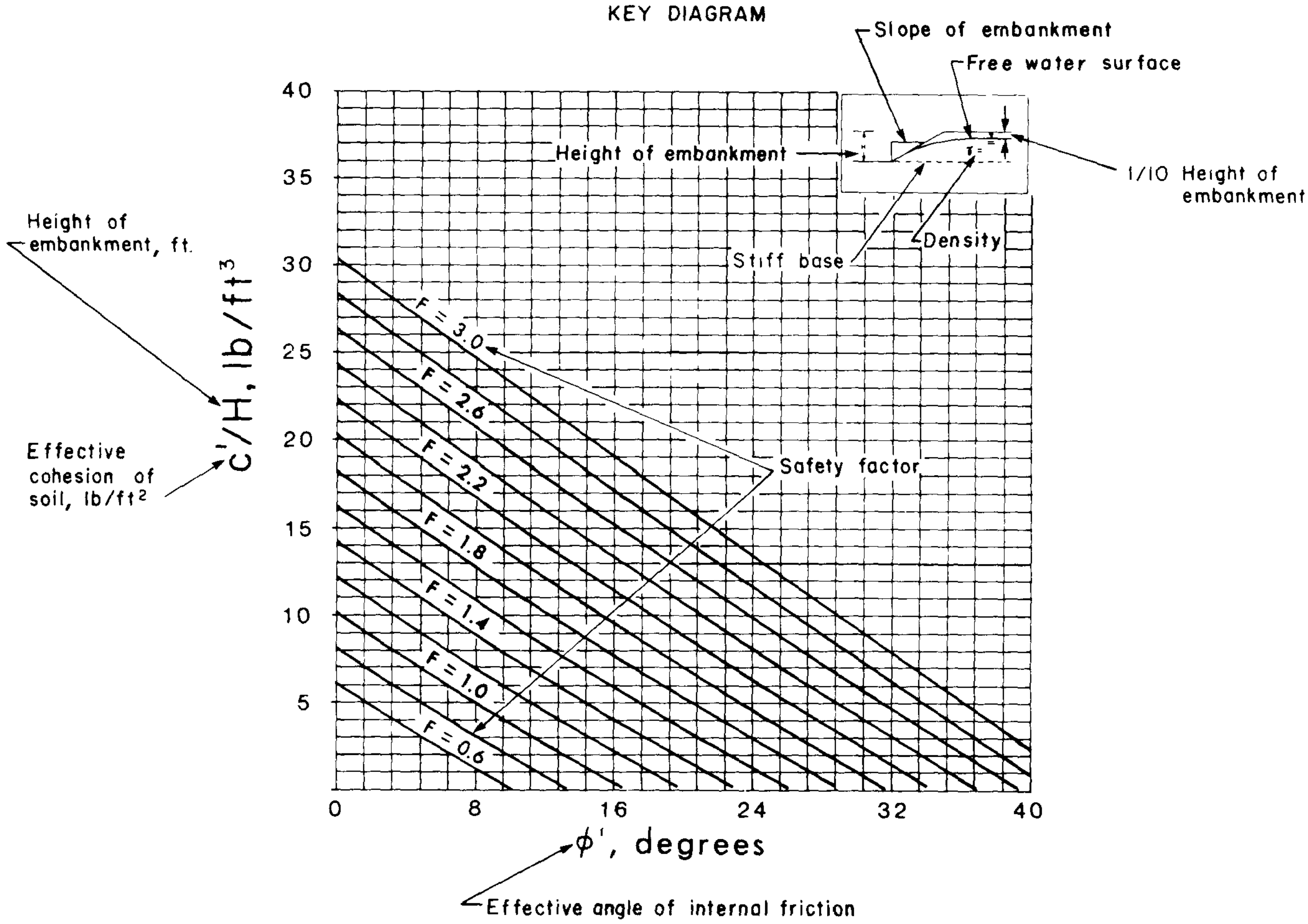

Figure C–1.—Example stability charts for embankments with a phreatic surface associated with 10 percent freeboard and depth factor = 1.00.

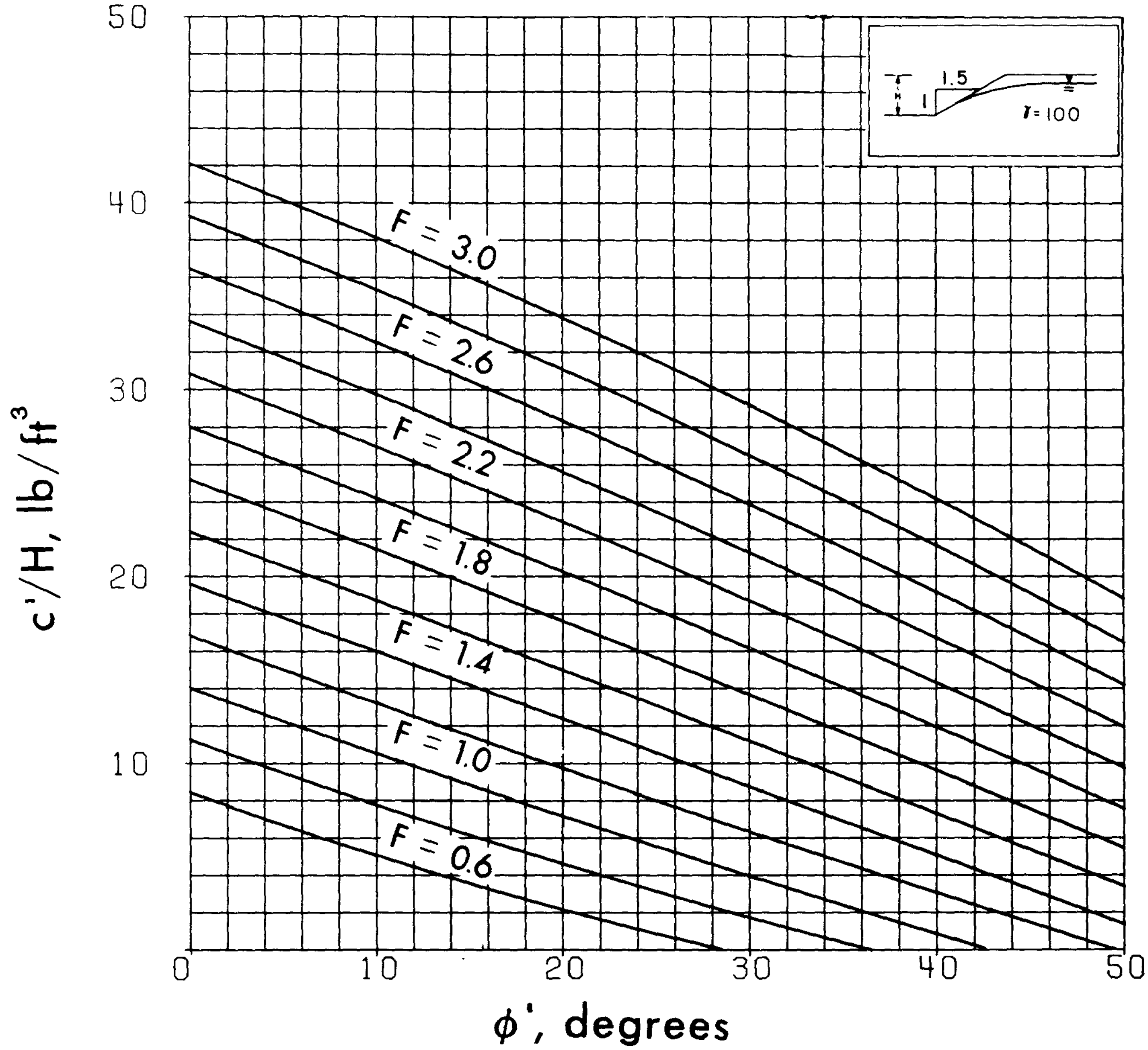

Figure C–2.—Factor of safety—1.5:1 slope, phreatic surface with 10 percent freeboard, density = 100 lb/ft³, D = 1.00.

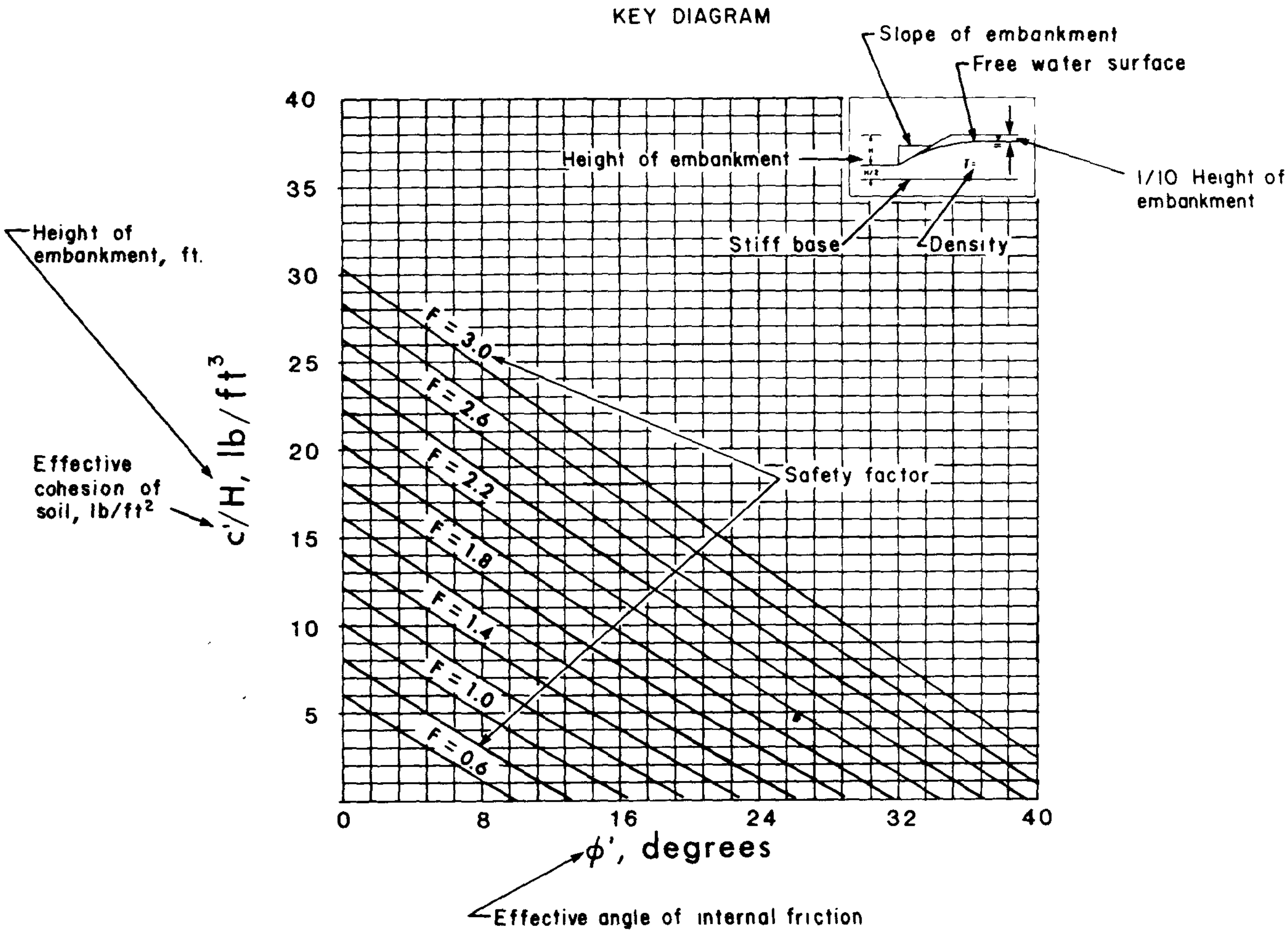

Figure D–1.—Stability charts for embankments with a phreatic surface associated with 10 percent freeboard and depth factor = 1.50.

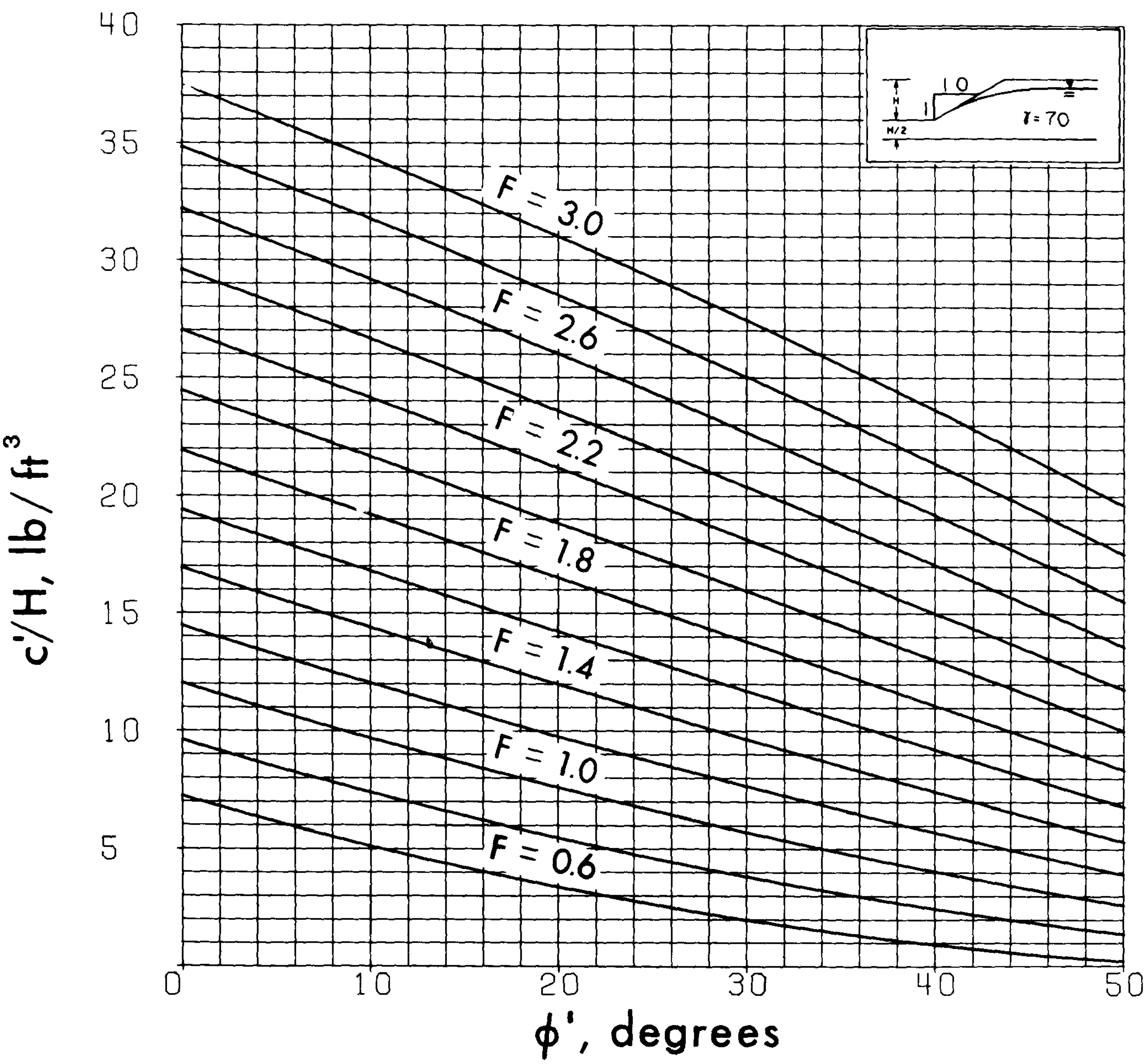

Figure D–2.—Factor of safety—1.0:1 slope, phreatic surface with 10 percent freeboard, density = 70 lb/ft³, D = 1.50.

PROBABILISTIC APPROACH TO THE FACTOR OF SAFETY FOR EMBANKMENT SLOPE STABILITY

by

P. C. McWilliams[1] and D. R. Tesarik[2]

INTRODUCTION

The Bureau of Mines is investigating the risk analysis, probabilistic approach to computing the factor of safety for the following reasons:

1. The Federal Mine Safety and Health Administration (MSHA) has imposed slope stability analysis as a requirement for the construction of coal tailings embankments (9).[3]
2. The President's Federal Coordinating Council for Science Engineering and Technology reported on Improving Federal Dam Safety in 1977. A prime research goal was to pursue a risk analysis approach to dam design
3. To resolve inconsistencies between the calculated factor of safety and the corresponding success or failure of the embankment.

In fact, theoreticians in soil mechanics have been pursuing a probabilistic approach to the factor of safety of an embankment or dam for the past 10 years. The motivation for this work is in contrast to the current practice of deterministically computing a factor of safety and treating it as an absolute with no regard of its inherent statistical variability Stated simply, the applied soils engineer works with one parameter—factor of safety—while the theoretician would impose an additional parameter; the dispersion of the factor of safety. Given these two parameters, one may then compute the probability of failure as a measure of slope stability Further, the two-dimensional modeling of Fellenius, Bishop, et al. (5) is challenged by a three-dimensional approach which postulates a cylindrical slip surface (10). Not surprisingly, three-dimensional modeling increases costs, for extensive soil sampling and laboratory testing are requisite to the required analytic techniques. A standard deviation of the factor of safety can be computed for both the two-dimensional model (via propagation of error) and the three-dimensional model. Requisite to these computations is some knowledge of the variability of the soil parameters—internal angle of friction, cohesion, and soil density

The question remains as to whether implementing these techniques is justifiable from both a scientific and economic view. A final point, the practitioners currently using deterministic two-dimensional analysis indirectly make allowances for the variability of the factor of safety by using conservative, rather than average, estimates of the soil parameters.

Background

For at least the past 10 years, geotechnical journals and books have been advocating an increased usage of probability and statistical techniques (1, 5, 10–11). In particular, the factor of safety, the current measure of the stability of an earth slope, has been challenged as to its adequacy. Slopes with a seemingly "safe" factor of safety have failed, while others whose factor of safety was at best minimal have not failed; thus, there is some validity in reinvestigating the problem. The real question is—does a better technique now exist that should replace the current factor-of-safety criteria for embankment design acceptance?

The factor of safety, F, is defined as:

$$F = \frac{\text{Resisting Moment}}{\text{Overturning Moment}} \text{ --- of a circular failure mass (fig. 1).}$$

Note that the two most popular techniques for computing the factor of safety both utilize the two-dimensional slip circle. Fellenius (or the Swedish method) first introduced this concept, most practitioners today use the modified Bishop-Morganstern algorithm (5–6).

After considerable deliberation, this work was restricted to the probability aspect only; leaving the area of risk analysis for others to pursue. It was felt that introducing the economic arguments is premature at this time. Further, the probability work provides the necessary basis for continuation into risk analysis, if so desired.

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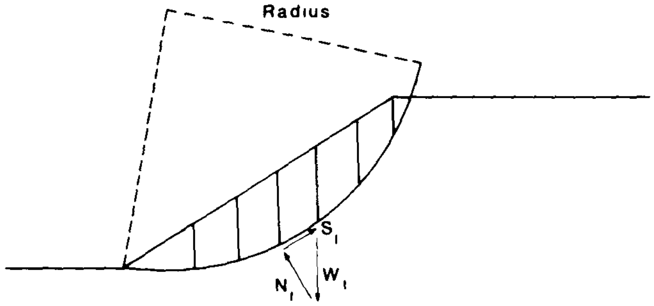
[1] Mathematical statistician
[2] Mathematician
Both authors are with the Spokane Research Center Bureau of Mines. Spokane, Wash
[3] Underlined numbers in parentheses refer to items in the list of references at the end of this report



W_i = weight of i^{th} slice
N_i = normal force, i^{th} slice
S_i = resisting shear force

Figure 1.—Circular failure mass.

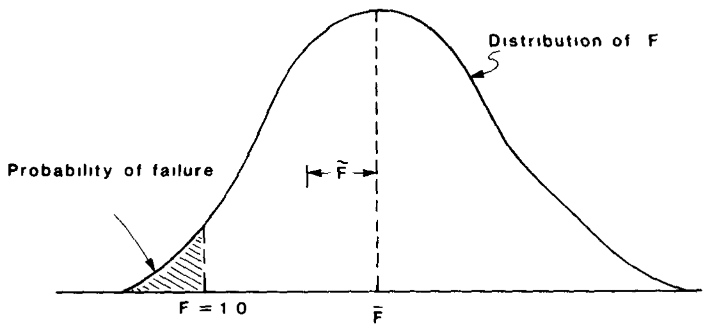


Figure 2.—Concept of probability of failure.

Population Distributions and the Probability of Failure

The Bureau has done considerable research relative to the factor of safety. The problems are by no means resolved at this time, but much interesting work has been done and is worthy of consideration. The fundamental point is that practitioners have relied on a single quantity—factor of safety—as the measure of the stability of an embankment. The factor of safety obtained from field sampling is, however, an estimate of the true factor of safety, and one should consider the concept of a population of factor-of-safety values, with a mean, $\bar{F}$, and a standard deviation, $\tilde{F}$.[4] If one is skeptical regarding a population of safety factors, reference is invited to papers by Lee and Singh (7) who sent the same soil samples to 28 different soils laboratories in the Los Angeles area. The coefficient of variation[5] for cohesion, c was 20 to 40 pct; for the internal angle of friction, ϕ, from 10 to 20 pct, and for soil density, γ, from 0 to 5 pct. Besides laboratory variability, there is the natural variability inherent in the soil itself. Thus, many authors are proponent of a probability of failure approach.

Since the embankment should fail if the factor of safety $\leqq 1.0$, the probability of failure equals the area of the population distribution to the left of F = 1.0 (fig. 2):

It is worth noting that current practitioners indirectly account for the variability in F by using conservative estimates for input, giving a conservative ($\hat{F}$) value fo the factor of safety. Thus, in meeting a requirement such as "factor of safety must exceed 1.5," the industry uses $\hat{F}$ rather than $\bar{F}$; thus effectively sliding the population of the right, and thus decreasing the probability of failure as illustrated in figure 3.

All of the preceding is fine, but the theory is only of practical value if the population assumptions are valid. Next, consider the practicality of obtaining $\bar{F}$ and determining the population of F.

[4] To avoid notation confusion, statisticians use "s" for standard deviation and soil engineers use "s" for strength—deference is made to soils engineers Thus, s = soil strength and $\tilde{F}$ = standard deviation of the factor of safety

[5] Coefficient of variation = $\frac{\text{standard deviation}}{\text{mean}} \cdot 100$.

Propagation of Error for Fellenius and Bishop Equations

Now $\bar{F}$ can be computed from either Bishop's equation or from Fellenius' equation. Since Fellenius' equation has F in a closed rather than iterative form, it is chosen for illustration:

$$F = \frac{\bar{c}L + \tan\phi \sum^{i} (\gamma A_i \cos\theta_i - u_i \Delta l_i)}{\gamma \sum^{i} A_i \sin\theta_i}$$

and computation of the variance of F gives:

$$\tilde{F}^2 \sim F_c^2 \tilde{c}^2 + F_\phi^2 \tilde{\phi}^2 + F_\gamma^2 \tilde{\gamma}^2 + \sum^{i} F_{u_i}^2 \tilde{u}_i^2$$
$$+ \text{ covariance terms.}^6$$

Newly defined terms are:

variance	=	square of standard deviation,
γ	=	soil density,
A_i	=	area of i^{th} slice,
u_i	=	pore pressure for the i^{th} slice,
$\tilde{c}^2, \tilde{\phi}^2, \tilde{\gamma}^2, \tilde{u}_i^2$	=	variance of soil parameters, and
F_j	=	partial derivatives with respect to soil parameters.

Due to independence of most of the parameters or to the relative small magnitudes of the resulting terms, the covariance terms can be ignored, leaving the preceding first-order approximation for $\tilde{F}^2$. It is important to note that to find $\tilde{F}$, one must input the variances of the soil parameters. Good estimates for $\tilde{c}$, $\tilde{\phi}$, and $\tilde{\gamma}$ are usually available. Harder to estimate are the $\tilde{u}_i$ values. The Bureau has incorporated the preceding formulation in the modified Bishop computer program.

[6] $\tilde{F}^2$ was computed by statistical propagation of error techniques (3).

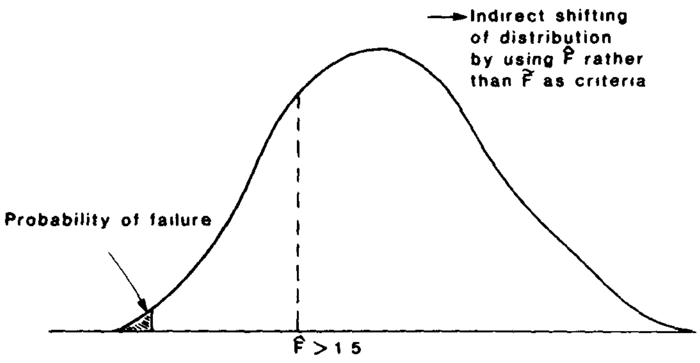


Figure 3.—Current practice of establishing factor of safety.

Possible Reduction of Factor-of-Safety Requirements

Knowing $\bar{F}$ may relax the requirements for the minimum acceptable factor of safety. Consider two alternative designs with the following statistics:

	Design one	*Design two*
F-value	1.60	1.40
$\bar{F}$	.30	.15
Pr (failure)[7]	2.2 pct	.4 pct

Without knowledge of $\bar{F}$, by using today's deterministic approach and accepting F greater than 1.5, design one would have to be chosen. But using probability of failure as the criteria, design two is deemed best. For, from the perspective of probability of failure, design two is five-times less likely to fail as is design one. Thus, acknowledging that variability in the factor of safety exists and using that quantity accordingly could result in a change of criteria that many practitioners would favor.

Sensitivity Analysis

An ongoing effort is to measure the dependency of the standard deviation of the factor of safety, S_F, on the respective standard deviations of ϕ, c, γ, and u. Although the sensitivity analysis is not complete, changes in soil density, i.e., through compaction, seem to be prominent in reducing the factor of safety. Final documentation of this project will, of course, include a complete sensitivity analysis and corresponding recommendations.

Population Estimates from Field Data

A direct approach to finding the population distribution of F was to sample a tailings embankment and actually generate a family of F-values. Due to funding consideration, this experiment was conducted in a simulated sense; that is, tailings material contained by a very stable earthen embankment was sampled. The conjecture was what if the contained tailings material was used as an embankment wall; what would the stability picture be? The stability of the pond tested was not addressed, the important point was to determine the shape and dispersion of the population F-values; for example, answer the question as to whether one can assume a normal distribution, a log-normal distribution, or what, if any conventional distribution does fit the situation?

In order to generate the necessary population distributions, some 50 Shelby tubes were used to gather the tailings material. Direct shear tests were run to obtain a family of ϕ, c, and γ values. For each data input, a factor of safety was computed via the modified Bishop program. In all, 74 factor-of-safety values were derived for the simulated embankment. No prior work of this kind, carrying the work to the concluding distribution of the factor of safety, was found.

Figure 4 and summary statistics represent the data. Summary statistics are:

Sample mean = 1.53	Maximum F = 2.35
Sample standard deviation = 0.40	Maximum F = 0.56
Sample size = 74	Coefficient of variation = 26.1 pct
Probability of failure = 8/74 ~ 11 pct	

The following conclusions would seem in order:

1. The distribution does show central tendency; thus, normal-distribution theory may be appropriate.

2. The variability is large, as seen by the large probability of failure (again, this is a simulated case).

Note that it is not suggested that the industry will ever be required to draw large populations of F-values; rather this is background work requisite to using $\bar{F}$ properly. As mentioned, the preceding is an artificial case; the Bureau intends to repeat the experiment in a more realistic environment

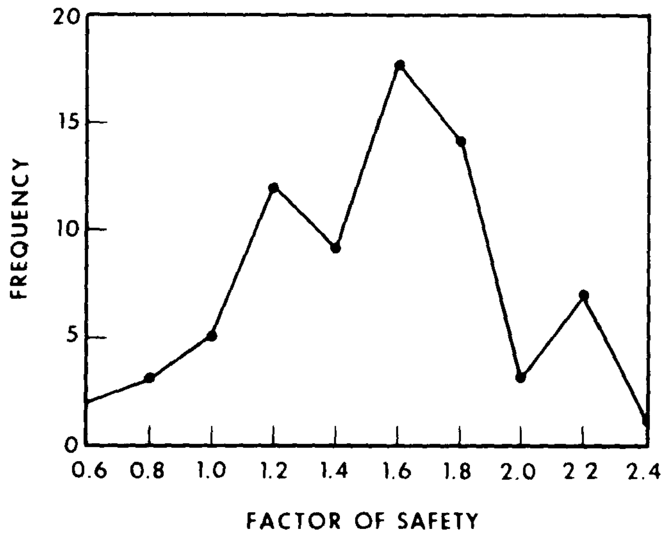


Figure 4.—Histogram of F-values for a simulated embankment.

[7] Probability of failure assumes a normal distribution for F.

Three-Dimensional Modeling

Another major thrust has been to change the factor of safety to a three-dimensional cylindrical surface model. Ang, Cornell, and Vanmarcke (2, 10) have written a series of articles over the past 10 years on this subject. A very brief sketch of this rather complex technique follows:

1. The basic premise that failures occur in three dimensions not two, thus a cylindrical surface is a better model to postulate.

2. The factor of safety is still defined as the ratio of the resisting to the overturning moment (fig. 5).

3. The ensuing formula for the factor of safety is:

$$F_b = \frac{M_r}{M_O} = \frac{\bar{s}_u Lrb + R_e,}{Wab}$$

where:

M_r = resisting moment,
M_O = overturning moment,
$\bar{s}_u$ = average design value of undrained shear strength,[8] and
R_e = contribution of end sections of failure mass to the resisting moment.

[8] The analysis can also be performed in terms of effective stress

4. By propagation of error the preceding formula for F_b produces:

$$\tilde{F}_b = \frac{\tilde{s}_{ub} Lr}{Wa}$$

where:

$\tilde{F}_b$ = standard deviation of the three-dimensional factor of safety, and
$\tilde{s}_{ub}$ = standard deviation of the shear strength in the embankment.

5. In order to find $\bar{s}_u$ and $\tilde{s}_{ub}$, one must measure strengths in three dimensions along the embankment Further, Vanmarcke (10) specifies that the usual variance (and thus standard deviation) computations for strengths must be reduced to account for the high intercorrelation between adjacent strength readings.

6. The variance reduction procedure requires an iterative search for "best" spacing of borings on the embankment surface. Average values of strength and end resistance, plus autocorrelation values for variance reduction work, are input for computation of F_b and its standard deviation.

7. Finally, the modeling finds a critical length, b_c, which is the most likely failure length for the hypothesized cylindrical model.

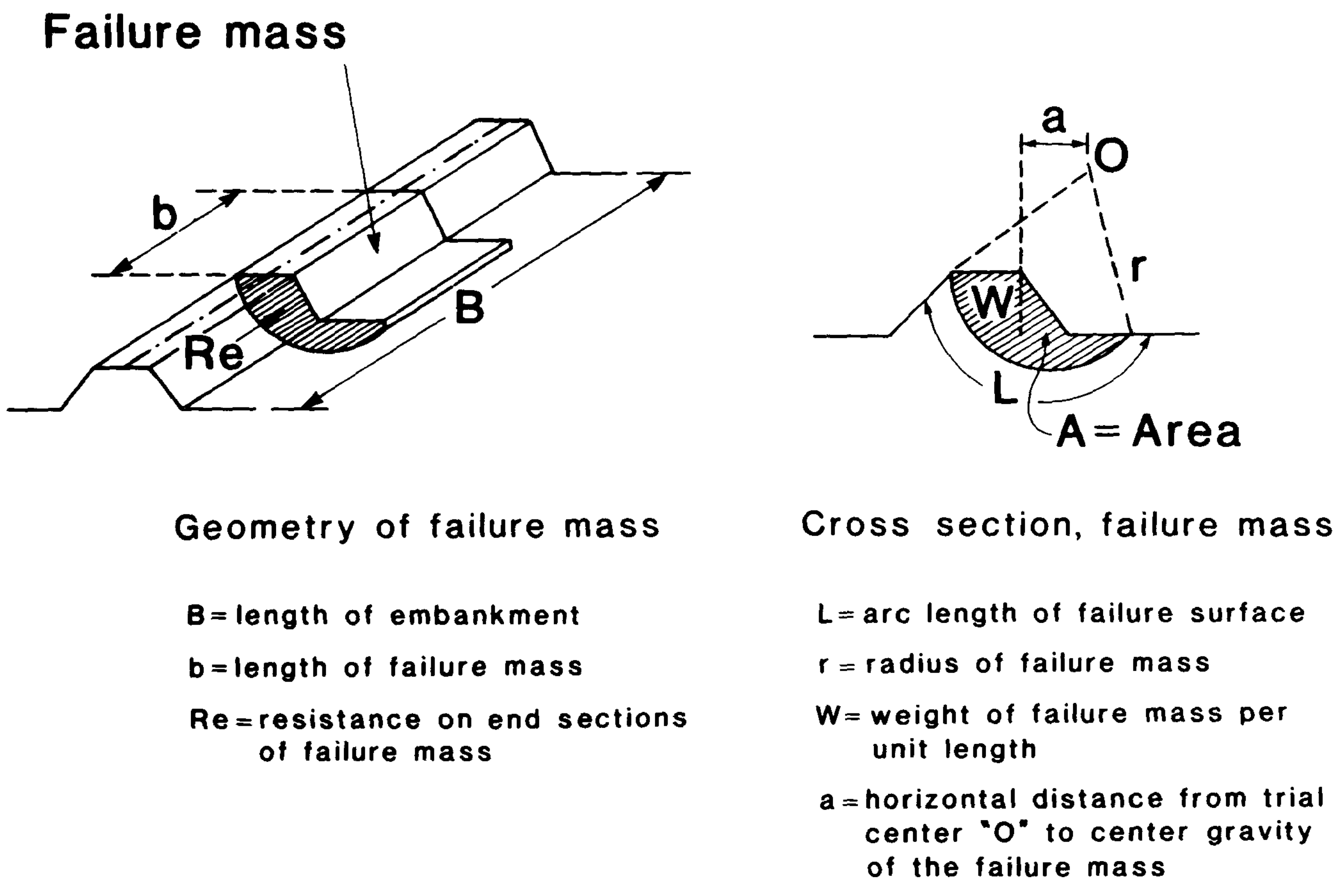


Figure 5.—Three-dimensional failure mass showing geometry of failure mass and cross section of failure mass.

The preceding outlines the procedure being field tested by contractors for the Bureau of Mines. Besides usual borehole work, a Dutch cone penetrometer was used to find the necessary supplementary borings. Preliminary results are encouraging; a final report will be forthcoming at the project's termination in the fall of 1981. It is not fair to make rigid cost comparisons at this time, for the initial field work is experimental and not production efficient. However, it will cost more to obtain field data of this detail—the question is, of course, is the product worth the expenditure?—a question for the future.

In deference to the preceding work, the cylindrical model may be better than the slip circle model, but the cylindrical model has shortcomings too; which one would be amiss to ignore. The "real-world" failure mechanism need not be a rotating cylinder either; it has been hypothesized that a model of a "sliding trough" with probability assigned on a particle basis would be better modeling than either the slip circle or the cylindrical model (a three-dimensional random-walk concept (4)). So the future will undoubtedly bring more evolution of ideas regarding "best" modeling for embankment slope stability.

Summary

The following represent current state of the art of the factor-of-safety work as perceived by the authors:

1. A strong conviction that some measure of the variability of the factor of safety should be used—ranging from something relatively simple, such as the coefficient of variation, to the probability of failure approach.
2. It is not yet established whether three-dimensional modeling should replace the current two-dimensional slip-circle modeling.
    a. The theoretical work will and should continue—it's neither complete at this time nor has it been honed as a practical field tool.
    b. At this time there is no justification for recommending a change in MSHA's current criteria. Only when it can be proven that failed embankment would have been properly predicted by three-dimensional modeling will there be universal acceptance for a model change.
    c. However, a designer with real doubts about an embankment's safety would be advised to use the three-dimensional modeling now.
3. The Bureau will continue its pursuit of both major thrusts; inclusion of the variability of the factor of safety and investigation of three-dimensional modeling.
4. The Bureau of Mines has, available for distribution, a computer program which outputs the usual factor of safety (Fellenius, Bishop) plus the standard deviation of the factor of safety using first-order approximations. To use this added feature, the user must input estimates of the standard deviations of the soil parameters. The program can be obtained by contacting the Spokane Research Center.

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APPLICATION OF REMOTE SENSING FOR COAL WASTE EMBANKMENT MONITORING

by

C. M. K. Boldt[1] and B. J. Scheibner[2]

INTRODUCTION

Remote sensing, as related to mine waste embankment monitoring, is defined as the gathering of information without direct human contact. In this report two forms of remote sensing were studied; instrument data gathering by satellite and aerial photogrammetry.

The speed of technological advance is incredible. Over 140 years ago the use of photography was first documented; 19 years later in 1858, a ballon floating several hundred meters over Paris took the first aerial photograph. It was not until early World War II that aerial photography was considered more than flights of fancy, when the Germans were believed to have relied heavily on information gathered from interpreting air photos taken of the entire Western Front every 2 weeks (1).[3]

The first U.S. satellite, Explorer I, was launched on January 31, 1958. As early as 1960, satellites were being used to transmit weather data, and in 1962, the first telecommunications satellite was put into orbit. Since these early days, satellites have achieved an ever-increasing role in our lives, and have become more and more diversified in their uses. The launch of the first Earth Resources Technology Satellite (ERTS-1), now the Landsat series of satellites, ushered in a new type of technology—that of managing and inventorying earth resources by merging space and remote-sensing technologies. Not only is the satellite capable of mapping large areas through photography, but it can also relay information from point to point on the earth's surface. In May 1974, the Synchronous Meteorological Satellite/Geostationary Operational Environmental Satellite (SMS/GOES) was launched, the first of a series. This is a no-charge operational system allowing any user to request access from the National Oceanic and Atmospheric Administration (3).

Since the launch of these satellites, they have been used for various studies including monitoring weather conditions in remote, high mountain areas (2), automatic data collection from hydrologic stations (4), and monitoring global volcanic activity (6).

[1] Civil engineer.
[2] Geologist. Both authors are with the Spokane Research Center, Bureau of Mines, Spokane, Wash.
[3] Underlined numbers in parentheses refer to items in the list of references at the end of this report.

General

The mining of earth resources is a geographically inflexible industry in that ore is where you find it, not where it is convenient, environmentally favorable or profitable to mine. A hundred miles of winding mountain roads can separate one mine from another or they can be so close together their waste piles touch. There are over 6,000 coal mines throughout the United States and the Department of Labor, Mine Safety and Health Administration (MSHA), is one agency required by law to inspect them. Improving the inspection and procedures and decreasing the workload by using remote sensing techniques is the objective of this research.

The Bureau of Mines has two projects in the remote sensing field as it pertains to mining. One of these is a recently completed contract that studied the use of aircraft mounted cameras and the results of photogrammetry, or obtaining measurements by use of photography. The second project is an ongoing contract to study the effectiveness of using an instrumentation system wired to remote data collection station to record embankment data.

Aerial Monitoring

The first project, an aerial monitoring contract (J0188027), was completed by Chicago Aerial Survey (CAS) of Des Plaines, Ill., titled "Improving Surface Coal Disposal Site Inspections." The objective was to determine and expand the data collection capabilities of aerial photogrammetry on actual coal waste embankments. A previous Bureau of Mines project titled "Rapid Monitoring of Coal Refuse Embankments" by CH2M Hill, which studied an active landslide and two coal waste embankments had indicated aerial reconnaissance could be used to monitor embankment movements and changes (5). Table 1 lists major differences between the CAS and CH2M Hill projects.

TABLE 1.—Differences between CAS and CH2M Hill projects

CAS	CH2M
Used vertical photography.	Used convergent and vertical photography combination.
One flyby necessary per site photographed.	Three flybys necessary per site photographed
Accuracy of ±0.3 foot with modification of elevation reading network.	Accuracy of ±0.15 foot.
Targets were set adjacent to the embankment as bench marks with elevations read anywhere on the embankment.	Targets were set on the embankments and read directly for elevations only at that point
Monitored 15 coal waste sites in West Virginia and Kentucky between March and December 1979.	Monitored two coal waste sites in West Virginia monthly between July and December 1979

Chicago Aerial Survey and its subcontractor, Dames & Moore, monitored 15 coal waste sites in West Virginia and Kentucky each month for 10 months (fig. 1).

Black-and-white and color-infrared (CIR) photographs were taken and analyzed for each site by CAS while Dames & Moore verified the conditions on the ground and solicited industry and MSHA inspector comments. Distortion in the photographs was corrected to produce a scalable photograph or orthophoto. From these, computer assisted stereoplotters were able to pick off elevations at predetermined points forming a 100-foot-grid pattern over the embankment face, crest, and impoundment. CIR photographs were taken seasonally to compare resolution and clarity with those of black-and-white photographs.

Major conclusions at the end of the project showed aerial photogrammetry (fig. 2) is useful as a supplement to existing MSHA inspection procedures. The interpretation of the orthophotos can offer cross sectional profiles, volume estimations, vertical movement, and objective documentation over the life of a mine. Seepage areas, erosion gullies, diversion systems, and overall views of the site can be examined first on the photographs then in an organized pattern over the waste site. It is believed this type of monitoring can be useful on high-hazard embankments when the inspector and mine personnel need more quantitative and qualitative information readily available.

Accuracy of the system is dependent on altitude, terrain, sun angle, atmospheric conditions, and human error. With the interpreter picking only optimum points from which elevations can be read instead of reading elevations every 100 feet whether the ground is visible or not, the accuracy approaches the ±0.3-foot range. This compares with a ±0.6-foot accuracy achieved with the 100-foot-grid pattern. It was estimated that present MSHA inspection procedures cost $ 470 per site. Using aerial photogrammetry would increase the cost to a little over $500 to fly, develop, and interpret the photographs per site with a one time survey fee of $2,000 to establish ground targets. Many more embankments could be inspected per day by using aerial photogrammetry allowing inspection personnel more time to interpret photographs and to concentrate on problem areas rather than make an individual inspection of each site.

Satellite Monitoring

The purpose of this project is to develop a system that can monitor the stability of one or more coal waste embankments using various sensors and data acquisition equipment. It consists of two phases, phase I being the installation and initial monitoring of selected instruments at a test site. The raw data are then sent to the user's receiving station via commercial telephone lines. Phase II, using the same test site and embankment instrumentation, will test the principle of relaying data to the user via a satellite and a central receiving station.

During phase I, a test site was selected at a West Virginia coal mine and 17 instruments were installed on a 200-foot-high embankment. Three multiposition borehole extensometers (fig. 3) measured vertical movement; three biaxial tiltmeters, one inplace inclinometer, and one traversing probe inclinometer measured horizontal deformation; and seven vibrating wire and two resistance piezometers measured water levels within the embankment. A pond level sensor, a V-notched weir with a recorder for monitoring seepage on the face of the embankment, and meteorological instruments were also installed. Fourteen instruments are located along the crest of the embankment, and three vibrating wire piezometers at various levels down the face of the embankment.

Each instrument is located in a drill hole and protected above ground by a covered standpipe. Two covered cable junction boxes with lightning protection circuits connect the sensors via cable to the onsite data station housing a signal conditioner, power supply unit, data logger, printer, modem, and telephone. Data were scheduled to be received at the user's receiving station at 2-day intervals. Incoming raw data are routed directly through a minicomputer for reduction to usable information. A paper tape of the raw data is also made at the test site to maintain a check on the accuracy of the transmitted data.

Several problems were encountered which included main power shutdowns, telephone lines knocked down, and instrument failures in the trailer. This resulted in only three sets of data being received for a total of 61 days of data collection. However, even though a relatively stable embankment was selected for the test site, and with the limited amount of information available, trends could be noted. These trends included correlations between piezometer readings and time of rainfall, while extensometer data suggested a slight surface settlement of the crest.

Phase II, which will be completed by 1984, will demonstrate the use of a satellite data collection system to monitor the stability of a coal waste embankment. This self-contained system will consist of a data collection platform wired to the existing instruments with an antenna and a battery-solar power source (fig. 4). This method has been used since 1974 by Landsat users to monitor global volcanic activity, and since 1976 by the U.S. Geological Survey to monitor stream flow and snow pack data in remote regions (3). Data would be relayed directly from the embankment to the GOES satellite, eliminating onsite power and telephone lines that are costly to install in remote or difficult access areas. Another purpose for using the satellite is the speed with which an inspector can obtain data during the rainy season. From this viewpoint,

Figure 1.—Typical camera mount aircraft over a coal waste embankment.

Figure 2.—Aerial photograph of a monitored coal waste embankment. (Approximate scale: 1 inch = 250 feet.)

several dams in potentially hazardous or populated areas could be monitored from a central district office If one dam out of several shows indications of problems, that dam can be checked first.

The data collection platform is simple to use The embankment sensors are connected to the data collection platform which is powered by a battery-solar array combination Data from the sensors are transmitted via an antenna to the satellite, and then to the central data acquisition center. From here they can be sent to the user via mail or telephone link, depending on the urgency of the data.

Use of the GOES and the associated data collection station facilities is provided free of charge by the National Oceanic and Atmospheric Administration (3). This was one of the reasons for selecting this satellite, the other being that one of its main functions is data collection.

Cost of the data collection platform is about $5,000. This includes the platform, cables, battery-solar array and antenna, though one antenna could serve several platforms depending on their locations Uninstalled instrument costs for this project range from $1,000 for a resistance piezometer, $4,000 for an extensometer, to $12,000 for an inplace inclinometer.

The basis of this test was to define the most suitable instruments to use in an embankment that could adequately monitor horizontal and vertical movements, and water levels If these water levels and movements could be ascertained, then in an actual monitoring situation only two to five instruments would be needed, depending on the size of the embankment, its condition, and its location in relation to populated areas, transportation facilities, or utility centers. Though the initial cost of the instruments may appear high, their use as a tool for an early warning of movement within an embankment, the capability of providing more frequent monitoring,

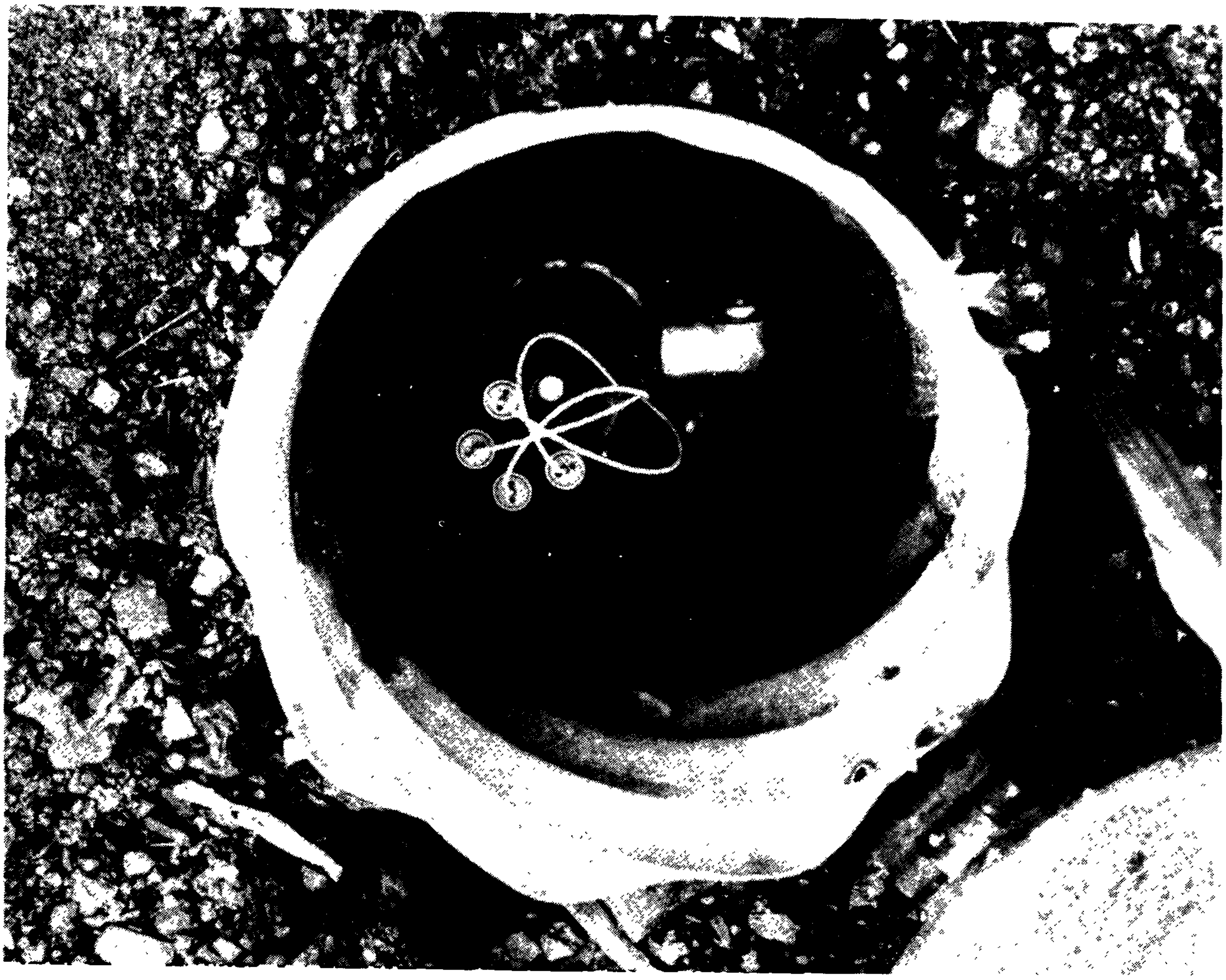

Figure 3.—Multiposition borehole extensometer in standpipe with cover.

and the opportunity for maintaining permanent records for review and study would seem to override that disadvantage.

Conclusions

In using the remote sensing techniques, the following situation is possible.

A high-hazard rated embankment in a remote area of the Appalachian region has been outfitted with instruments relaying data through the GOES satellite. The instruments are a piezometer, inclinometer, and a pond level sensor. This same embankment is also being monitored by use of aerial photogrammetry. Flights over the site have been scheduled by the inspector every 3 months, making sure a flight occurs right before and after the wet season, probably December and March. Using the information from these two techniques the inspector feels confident the embankment, though showing creeping movement on the order of 6 inches every 12 months is in no imminent danger of failure. The pond level is slowly rising since mining activity has increased due to a rise in the spot market price.

An unseasonal rainfall dumps an inch of rain in the area over a 3-hour interval; the inspector, knowing the pond level is already rising, requests updated data through the satellite system. The pond level sensor relays information that indicates the pond is now within 18 inches of the crest and the inclinometer has detected an increase in deformation indicating a possible mass movement along the dam face. The inspector requests data on the embankment every hour. Twenty-four hours after the initial rainstorm, the piezometer notes a rise of water within the embankment but the pond level sensor records dropping water levels, while the inclinometer indicates a slight settling of the embankment.

At this point the inspector ascertains that the slope stability is adequate and there is no immediate danger of dam failure. Information is now requested every 6 hours until the inspector feels the embankment has stabilized. However, due to some changes having taken place and the dropping water levels, the inspector schedules an aerial flight the following day. The

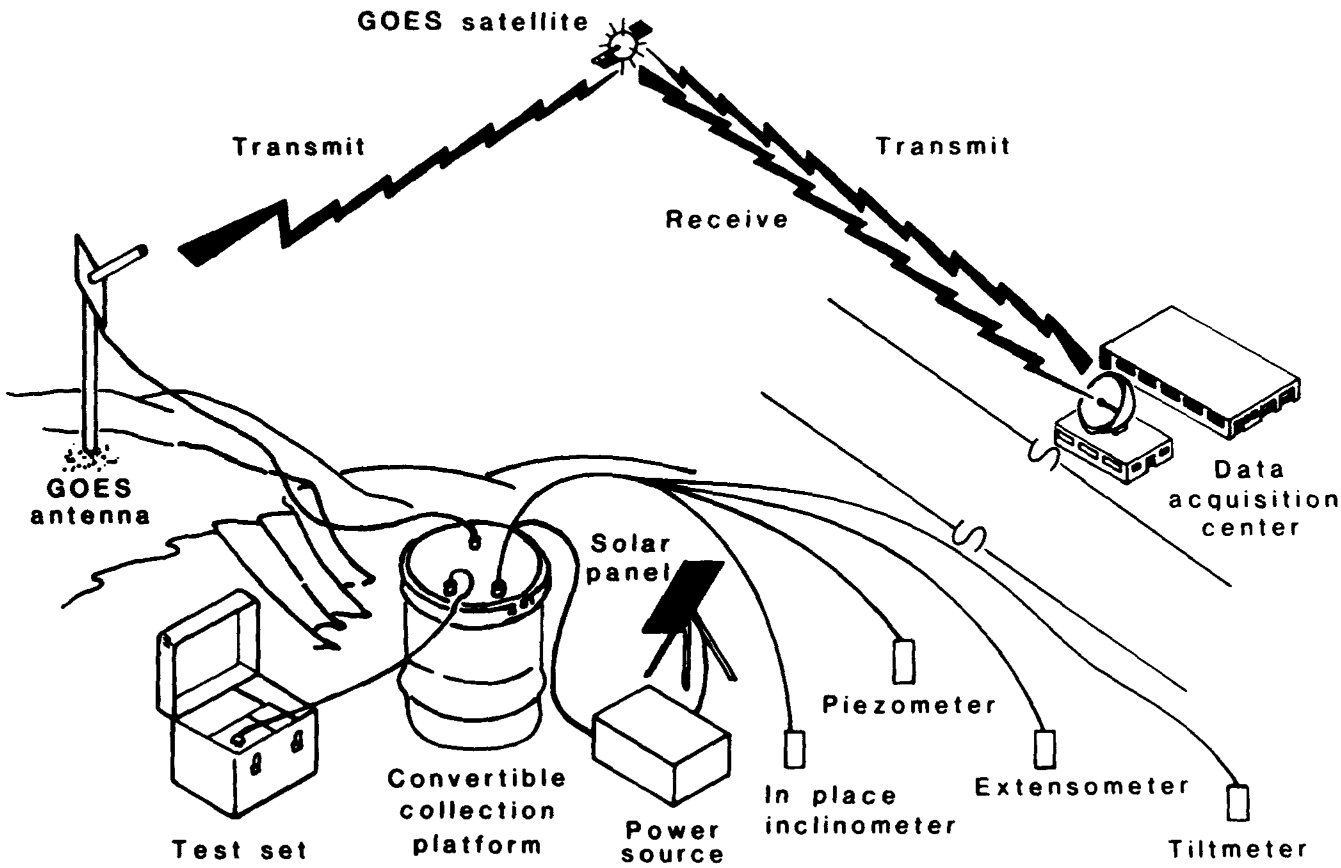

Figure 4.—Typical GOES convertible data collection platform installation.

data gleaned from the photographs show a well-formed erosion gully at one of the abutments, the diversion ditch clogged in spots with rain washed debris, and the embankment elevation unchanged. With this information in hand the inspector decides to visually inspect the embankment, going over repair requirements with the operator.

From this scenario it can be seen that remote sensing would be a powerful tool for mine inspection and mine company personnel. By monitoring internal changes through instrumentation and transmitting the data via satellite, and observing external surface movement by use of aerial photogrammetry the user can more readily and accurately evaluate waste embankments even in the most remote regions of the country. It is not recommended that these two remote sensing techniques be used on every embankment. Rather, they should be seriously considered for high hazard embankments which the inspector and mine personnel feel with added information could provide an early warning system. This will increase chances of correcting a hazardous situation before it advances to failure.

Though this paper has discussed remote sensing systems only in connection with coal waste embankments, the systems could be used for other types of embankments such as metal-nonmetal mine waste embankments and tailings ponds.

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SUMMARY OF RESEARCH ON CASE HISTORIES OF FLOW FAILURES OF MINE TAILINGS IMPOUNDMENTS

by

P. C. Lucia,[1] J. M. Duncan,[2] H. B. Seed[2]

INTRODUCTION

In the past decade there has been a dramatic increase in the size of tailings dams and mine waste impoundments. Tailings dams are now included among the largest dams in the world, and a 700-foot-high tailings dam is now in the planning stages (7).[3] Studies conducted after the failure of the coal waste impoundment at Buffalo Creek, W. Va., in 1972, indicated that a great many of the mine waste impoundments constructed at that time received no engineering design consideration (13). Recent history indicates that the consequences of failure of a large mine waste deposit can be disastrous from the standpoints of both loss of life and environmental damage.

Liquefaction and flow of liquefied tailings or other types of waste deposits have resulted in many deaths. Seismically induced failures in Chile at the Barahona and El Cobre tailings dams resulted in over 200 deaths. In Africa, the failure of the Mulfulira and the Bafokeng tailings dams resulted in a total of about 100 deaths. The failure of the coal waste dam at Aberfan, Wales, resulted in over 140 deaths, many of them school-age children between the ages of 7 and 10. The failure of the coal waste dam at Buffalo Creek and the subsequent deaths of over 100 people brought the problem of mine waste disposal to the public's attention in the United States.

Severe environmental pollution can develop depending on the nature of the tailings and its proximity to rivers and streams. The failure of a phosphate tailings pond in Florida polluted the Peace River for a distance of about 120 km. In Japan, the Mochikoshi tailings dam failed during a 1978 earthquake. The flow of cyanide-laden tailings polluted a river for about 30 km and destroyed the marine life in a bay.

It is clear that the consequences of failure can be disastrous both from physical and environmental loss in some cases. There are many other cases where a failure results in only a temporary loss of storage to a mining company. A careful review of many failures has resulted in empirical and theoretical methods by which the consequences of a failure can be evaluated. This paper will summarize the case histories studied and an empirical approach to the assessment of consequences based on those case histories.

Soil Behavior at Liquefaction

Any mine waste material, regardless of origin, can be classified according to the principles of soil mechanics. Tailings are frequently angular, bulky grained sand, and silt-size particles. It has been known for some time that sand and silt particles are susceptible to rapid and large reduction in strength due to very minor disturbances if they are deposited in a loose condition and they are saturated.

Tailings are commonly deposited using hydraulic methods, where the particles separate by size due to gravity in a peripheral discharge system or by cycloning. As they settle from the water in which they were transported, tailings often accumulate in loose deposits, and silt sizes may form a metastable honeycombed structure, as shown in figure 1.

Rapid loading of this type of soil structure, either by seismic or by static means, results in a rapid buildup in pore pressures. The induced shear stresses are resisted at the points of contact between soil particles because the water in the voids has no shear strength. The soil particles move under the shear stresses and tend to densify, which results in compression in the pore fluid. This transferring of compressive stress to the pore fluid results in reduced compressive forces at interparticle contacts, and a consequent weakening of the soil structure. After a small amount of strain (less than 1 pct) particle-to-particle contact may be lost, and complete breakdown of the structure may occur. This phenomenon, which results in nearly complete loss of strength, is called liquefaction. The stress-strain curve in figure 2 illustrates the loss of strength resulting from liquefaction.

The breakdown in structure and loss of point-to-point contact results in the induced shear stresses being transferred to the fluid, which has no shear strength, only viscous shearing resistance. The soil mass is then driven by forces for which it does not have sufficient resistance, resulting in the

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[1] Presently employed by Converse-Ward-Davis-Dixon, San Francisco, Calif.
[2] University of California at Berkeley.
[3] Underlined numbers in parentheses refer to items in the list of references at the end of this report.

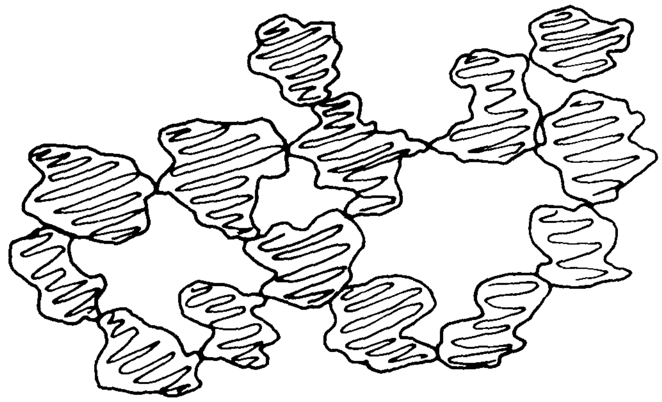

Figure 1.—Honeycomb structure of loosely deposited angular bulky grained soils.

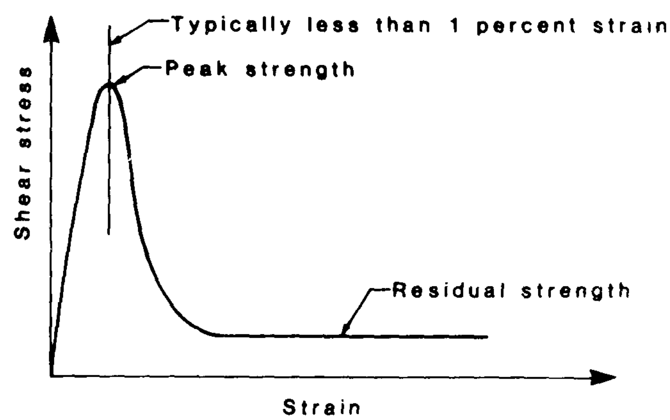

Figure 2.—Typical stress-strain curve for loose bulky grained soil.

onset of flow of the mass. While the soil particles are in motion, it may be imagined that they form a "minimum resistance" structure. During flow, at any given time, there is always some soil-to-soil contact, and this results in some small shear resistance for the flowing soil mass. This shear strength is termed the residual strength; the magnitude of the residual strength is a function of soil type, initial density, and rate of flow. The flowing soil mass will come to rest when the shearing resistance in the soil due to its residual strength is equal to the shear stress.

Case Histories

In order to develop a rational approach to predicting how far tailings will flow if failure occurs, case histories of flow failures were collected. Due to the reluctance of many mining companies to release information on previous failures, the case history data were supplemented with information on flow failures involving similar types of soils. Table 1 summarizes information on the failure of the dams or waste impoundments used in this study.

Typical of the extremes of behavior noted in the study are the cases shown in figures 3 and 4. Figure 3 shows the failure of a gypsum tailings pond (9) in Texas where the liquefied tailings flowed over an essentially flat slope and came to rest with a slope of about 1 pct on the surface of the tailings. The other extreme is the liquefaction and flow of the coal waste dump at Aberfan, Wales (8). In this case, the liquefied material flowed down a 12° slope, partially covering a school at the bottom of the slope. The flowing material covered the hillside at a relatively uniform thickness over the distance of

TABLE 1.—Data on case histories

Dam	Location	Probable cause of failure	Height, m	Total quantity of material stored, tons	Material involved in failure, tons	Travel distance, km
Barahona	Chile	Seismic	65	NA	NA	NA
El Cobre (Old)	Chile	Seismic	35	7 6 × 10^6	1 9 × 10^5	12
El Cobre (New)	Chile	Seismic	15	5 0 × 10^5	5 0 × 10^5	12
Hieno Viejo	Chile	Seismic	5	NA	1 2 × 10^3	1
Los Maquis	Chile	Seismic	15	6 0 × 10^4	3 0 × 10^4	5
La Patagua	Chile	Seismic	15	NA	5 0 × 10^4	5
Cerro Negro	Chile	Seismic	20	7 9 × 10^5	1 2 × 10^5	5
Bellavista	Chile	Seismic	20	7 0 × 10^5	1 0 × 10^5	2 5
Ramayana	Chile	Seismic	5	NA	2 0 × 10^2	NA
Tailings Dam	Southwest United States	Seepage	44	NA	2.0 × 10^2	24
Bafokeng	South Africa	Seepage	20	2 2 × 10^7	5.2 × 10^6	45
Gypsum	Texas	Seepage	11	7 0 × 10^6	2 0 × 10^5	.3
Mochikoshi	Japan	Seismic	32	8 2 × 10^3	1 4 × 10^3	30
Phosphate	Florida	Seepage	4	NA	8 0 × 10^6	120
Tip No 7	Aberfan, Wales	Static	37	4 3 × 10^5	1 9 × 10^5	6
Tip No 4	Aberfan, Wales	Static	46	1 7 × 10^7	NA	.7
Abercynon	Abercynon, Wales	Static	37	NA	1 8 × 10^5	6
Blackpool	England	Static	40	NA	1.5 × 10^4	1
Cholwich	England	Static	46	NA	2 5 × 10^4	2
Louisville	Kentucky	Seepage	31	1 0 × 10^6	1 0 × 10^6	.1
Jupille	Belgium	Static	46	6 0 × 10^5	1.5 × 10^5	6
Fort Peck	United States	Static	69	NA	5.0 × 10^6	4
East Chicago	United States	Static	2	NA	NA	02
Koda Numa	Japan	Seismic	3	NA	NA	02
Uetsu	Japan	Seismic	10	NA	NA	11

NA Not available

flow. In one case, essentially no damage resulted from the failure; in the other case over 100 lives were lost.

The collection and understanding of the behavior of liquefaction failure can serve as a guide to the engineer in evaluating the potential consequences of failure of tailings impoundments. The case histories collected during this study are discussed briefly in the following sections, considering first the tailings dams, and then waste impoundments and other earth structures

Chilean Tailings Dam

The Barahona tailings dam in Chile failed in 1928 (1) during an earthquake. The material flowed down a 9° slope, eventually getting into a river. A total of 54 lives were lost.

Dobry and Alvarez (6) discussed the failure of 10 tailings dams in Chile due to an earthquake in 1965. The tailings flowed distances ranging from 1 to 12 km. The downstream geometries and final slopes on which the tailings came to rest are unknown.

Tailings Dam (Southwest United States)

A copper tailings dam in the southwestern part of the United States failed due to excessive seepage and subsequent erosion. The loss of confinement resulted in the liquefaction and flow of the tailings. The tailings flowed in rivers and streams about 24 km from the disposal site. The tailings that remained in the pond stopped flowing and became stable when the slope angle of the tailings was about 1.5°.

Bafokeng, South Africa

The Bafokeng platinum tailings pond failed in 1974 (3) due to excessive seepage. The liquefied tailings flowed down a mine shaft downstream from the dam, resulting in the deaths of 11 men. The tailings flowed over a 1° slope to the Kwa-Leragane River about 600 m from the dam. Most of the tailings that reached the river were carried to the Vaalkop reservoir about 45 km away. When the tailings stopped, the slope of their surface was about 1.3°, measured from the river back into the pond.

Figure 3.—Failure of a gypsum tailings pond.

Figure 4.—Failure of a coal waste dump at Aberfan, Wales.

Gypsum Tailings, Texas

The failure of a gypsum tailings pond in 1966 (9) is shown in figure 3. The dike around the pond failed due to excessive seepage. The liquefied tailings flowed a distance of about 300 m from the embankment and the failure extended about 100 m back into the pond (fig. 3). The tailings came to rest at a slope of about 1°.

Mochikoshi Tailings Dam, Japan

The Mochikoshi gold tailings pond failed in 1978 (10) during a major earthquake. The tailings pond was formed by three small dams. Two of the dams failed during the earthquake, resulting in the flow of tailings down the hillside which was somewhat steeper than 20°. The tailings polluted a river and a bay about 30 km from the site. The flow of tailings from the pond stopped when the tailings in the pond became stable at a slope of about 4° to 5°.

Phosphate Tailings Pond, Florida

A phosphate tailings dike failed in 1971 (4), polluting the Peace River in Florida over a distance of about 120 km. Phosphate tailings differ from most other tailings in that they are clay-size rather than bulky silt-size particles. They frequently have water contents of several hundred percent and little if any residual shear strength. Therefore, they flow much the same as would water.

Aberfan, Wales

In 1966, a coal waste dump, tip no. 7, failed (8). The subsequent liquefaction and flow of the coal waste resulted in 144 deaths in the village of Aberfan. This failure was one of three that had occurred in the area over a period of years. In December 1939, a similar tip had failed at Abercynon, about 8 km from Aberfan, under very similar conditions. The material flowed down a 12° slope, traveling about 610 m and depositing material about 6 m thick over the slope. At Aberfan, in 1944, tip no. 4 failed just upslope from the one that failed in 1966. In 1944, the liquefied material flowed down a 12° slope about 610 m and blanketed the slope to a depth of about 4.5 m. The failure in 1966 was similar in all respects to the two previous failures, only in this case the village was within 610 m of the tip that failed, and the failure thus resulted in a tremendous loss of life.

Blackpool and Cholwich, England

In the china clay industry in England, the clay-size particles are removed for use in making china, leaving sand and silt-size particles as a waste product. In Blackpool, a waste deposit failed in October 1967 (2), resulting in a flowslide down a 7° slope. When it came to rest, the material blanketed the slope to a depth of 3.0 to 3.6 m over a distance of about 120 m. In October 1968, a similar waste deposit at Cholwich failed. The flowslide traveled about 180 m down a 6° slope and came to rest with a 7° slope in the waste.

Carbide Lime Tailings Pond, Louisville

In 1963, a carbide lime tailings pond in Louisville, Ky., failed, resulting in a flowslide. Carbide lime differs from tailings in that it is not a waste product of a mining operation but rather a byproduct of the production of acetylene gas. The failure in the pond was due to excessive seepage and the subsequent erosion of the embankment. The flow occurred over flat land and involved the entire pond. The liquefied tailings became stable when the slope of the tailings was about 1.5°.

Jupille, Belgium

A fly ash deposit failed in Jupille, Belgium, in 1961 resulting in 11 deaths (2). The waste deposit was formed by dumping the ash and allowing it to fall at its angle of repose. The liquefied material flowed down a slope as steep at 18° and traveled a distance of about 610 m. Almost all the material in the waste deposit flowed to the bottom of the slope, and was believed to have traveled at speeds of 110 to 160 km/hr.

Fort Peck Dam

Fort Peck Dam is a hydraulic fill dam constructed in a manner similar to a tailings pond. The method of deposition results in coarser particles near the edge of the embankment and finer particles in the core. More importantly, the resulting structure was composed of bulky grained particles in a very loose state, similar to tailings. The dam failed in 1938 (5). The resulting flowslide traveled about 480 m from the toe of the dam over nearly level ground. The liquefied material came to rest at an average slope of about 2.5°.

East Chicago

Peck and Kaun (12) described a flowslide that occurred in uniform fine sand during construction of a dock wall in 1946. The sand was initially placed under water behind a dock wall. Removal of the wall and the subsequent loss of support initiated the flowslide. The flowslide stabilized when the slope in the liquefied material was about 4°, and the area over which it flowed was essentially level.

Uetsu Railway Embankment

A sand fill placed to serve as a railway embankment failed during the 1964 Niigata earthquake (14). The embankment was constructed through a rice field and the bottom portions of the embankment were saturated. The liquefied material flowed about 120 m over ground which sloped at 2°, and came to rest at a slope angle of about 4°.

Koda Numa, Japan

A small railway embankment at Koda Numa, Japan failed during the 1968 Tokachioki earthquake (11). The soil was a fine to medium sand. The embankment liquefied and flowed in both directions from the centerline, over level ground. The liquefied material flowed about 18 m, coming to rest at slope angle of about 4°.

Summary of Case Histories

A summary of the final stable slopes and downstream ground slopes for the case histories discussed is presented in table 2, for those cases where this information could be determined. With the exception of the coal and china clay waste deposits, none of the liquefied soils were able to sustain slopes greater than about 5°. At Barahona, the downstream slope was about 9°, and the liquefied tailings did not have sufficient strength to stop on this slope. At Mochikoshi, the downstream slope was in excess of 20°; the material that remained in the pond was only able to sustain a slope of 4° to 5°, indicating that the 20° slope was far in excess of a slope that the liquefied tailings could sustain.

While the data are limited, they indicate a consistent pattern of behavior. Liquefied soils have a low shear strength that enables them to come to rest on some small slope. In cases where the downstream ground slopes were steeper than 4° only the unsaturated coal and china wastes came to rest. All of the liquefied saturated tailings materials continued to flow until they reached inclinations of 1° to 4°

Simplified Procedure for Predicting the Distance of Flow

A review of the case histories indicates that liquefied soils can sustain themselves on mild slopes in a stable condition. An idealized cross section is shown in figure 5. The actual shape at the toe of the slope is shown in a dashed line with the idealized shape shown by solid lines. Available data on the postfailure conditions usually included α, β, L, and material index properties such as water content. To quantify the behavior observed in the case histories, the shear strength required to give a factor of safety of 1.0 was calculated. The assumption was made that at the moment the flow stopped, the shear strength of the liquefied mass equaled the shear stresses induced by the low slope angle. This assumption neglects inertia forces; however, they may be negligible at low velocities just before flow stops.

In calculating the shear strength of the soil, the assumption was made that the behavior was undrained, and that the strength could be represented as an equivalent shear strength, S_u. This back-calculated strength represents the residual strength of the previously discussed "minimum resistance" structure. The residual strength of the following mass may vary from place to place through the mass due to different degrees of drainage, and variations in particle size or void ratio, among other factors. The shearing resistance also varies with rate of flow due to viscosity. Calculating the shear

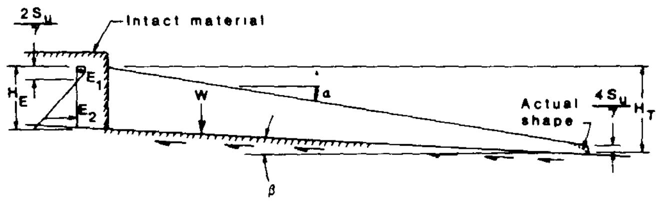

Figure 5.—Idealized cross section.

TABLE 2.—Postfailure conditions for case histories

Dam	Location	Material type	Downstream ground slope, degrees	Final slope of liquefied materials, degrees	Calculated residual shear strength, psf
Barahona	Chile	Copper tailings	9	(1)	NA
Tailings Dam	Southwest United States	Copper tailings	0 (1)	1 5 (2)	50
Bafokeng	South Africa	Platinum tailings	1	1 3	15
Gypsum	Texas	Gypsum tailings	0	1	20
Mochikoshi	Japan	Gold tailings	0 (2)	4 to 5 (2)	210
Phosphate	Florida	Phosphate tailings	NA	(1)	NA
Tip No. 7	Aberfan	Coal waste	12	12	375
Tip No. 4	Aberfan	Coal waste	12	12	330
Abercynon	Abercynon	Coal waste	12	12	450
Blackpool	England	China clay	7	7	140
Cholwich	England	China clay	6	7	340
Louisville	Kentucky	Carbide fine tailings	0	1 5	53
Jupille	Belgium	Fly ash	>18	(1)	NA
Fort Peck	United States	Clay to fine sand	0	2 5	250
East Chicago	United States	Fine sand	0	4	20
Koda Numa	Japan	Fine sand	0	4	25
Uetsu	Japan	Fine sand	0	4	35

NA Not available.
[1] Material traveled in rivers or other waterways
[2] Measured slopes on materials remaining in tailings pond.

strength based on the geometry when the mass comes to rest makes it unnecessary to consider viscous and inertial effects.

Various modes of failure were considered to find the most critical. It was found that considering shear along the base and active pressure at the back of the liquefied wedge required greater shear strength than the most critical circular or noncircular slip surfaces, and this mechanism was therefore used to develop dimensionless stability charts. The condition analyzed is shown in figure 5. For this condition the equation of stability can be written in dimensionless form if it is assumed that the factor of safety is unity:

$$\frac{W}{\gamma H_T^2}\sin\beta - \frac{S_u \cdot L}{\gamma H_T^2 \cos\beta} + \frac{E_2}{\gamma H_T^2}\cos\beta - \frac{E_1}{\gamma H_T^2}\cos\beta = 0 \qquad (1)$$

and this equation can be solved in terms of the dimensionless parameter, N_o, where

$$N_o = \frac{\gamma H_T}{S_u}$$

where γ = total unit weight of tailings,
H_T = as shown on figure 5,
S_u = as residual shear strength,

in which N_o varies with α and β as shown in figure 6. Given values of α and β, the stability number, N_o, can be determined directly from figure 6. The shear strength, S_u, can be calculated using equation 2 if γ and H_T are known. The shear strength values determined for the case histories studied are shown in table 2.

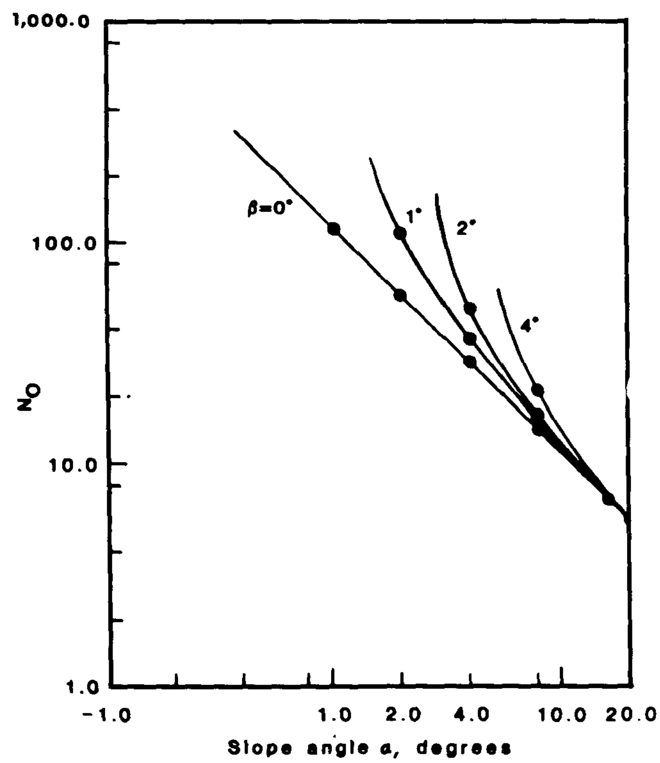

Figure 6.—Slope stability chart.

The distance through which liquefied tailings may flow before coming to rest (or "freezing") can be estimated based on these charts. This requires knowledge of S_u, the residual strength after liquefaction. At the present time, there are no laboratory procedures for measuring this important property of tailings materials. Studies are now underway to develop suitable procedures. At the present time, the most appropriate procedure appears to be to estimate the residual strength of the tailings during flow using back-calculated values for similar materials.

The value of β the downstream slope angle, also must be known in order to estimate the flow distance. Where downstream conditions are not uniform, it is necessary to use judgment to determine an average value of β for the area downstream where the liquefied tailings would flow. When the value of L has been calculated, using the steps described in the following, the value of β should be reviewed to confirm that it is representative of the ground slope within the flow area.

When values of S_u and β have been established, the flow distance, L, can be estimated using the steps outlined. Because the flow distance is affected by the volume of material which liquefies, it is necessary to use a number of trials to calculate L, and it is convenient to use a chart of the type shown in figure 7 to determine the distance for L for which the requirements of shear strength, bed slope, and flow volume are all satisfied simultaneously.

The steps for plotting the curves shown in figure 7 and determining L are as follows:

1. Using figure 6, determine the value of N_o for a number of assumed values for α. For each of these values of N_o, calculate H_T using this formula:

$$H_T = \frac{N_o S_u}{\gamma} \qquad (3)$$

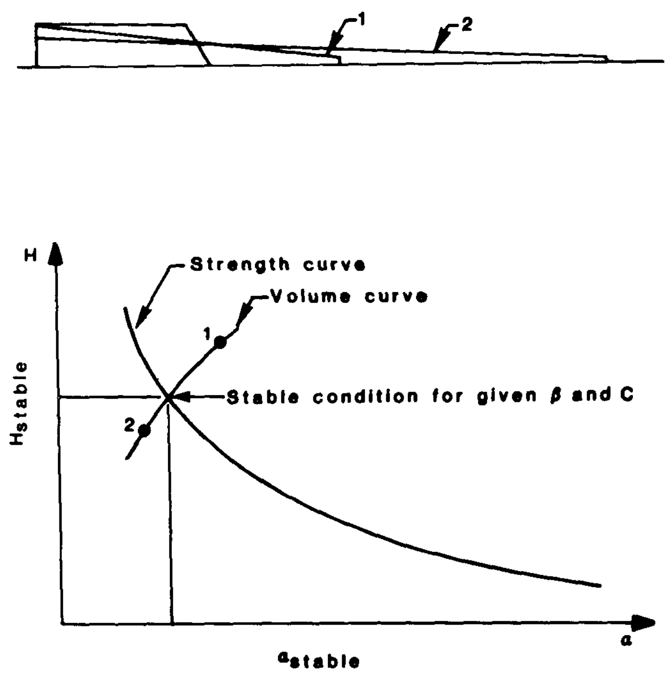

Figure 7.—Prediction of distance of flow.

2. Plot the values of H_T versus the corresponding values of α on a diagram like that shown in figure 7. These values of H_T and α will define a "strength curve" which slopes down to the right.

3. Estimate the volume of tailings that would be involved in the flow, V_f. As shown in table 1, in many cases, the volume of tailings which flow is considerably less than the total volume of the pond. However, if the objective is to determine the greatest possible flow distance, the maximum possible volume of tailings should be considered. The data in table 1 indicate that in some cases, especially those where the tailings were extremely fluid, the entire volume of tailings in the pond did flow. Therefore, in the absence of evidence to the contrary, it appears that the most appropriate assumption will often be that 100 pct of the tailings will flow.

4. For a number of assumed values of α, calculate H_T using this formula:

$$H_T = \sqrt{A_1^2 H_c^2 + A_2 V_f} - A_3 H_c \tag{4}$$

where
$$A_1 = \left(\frac{\tan\alpha}{\tan\alpha - \tan\beta}\right)^2 \tag{5}$$

$$A_2 = \frac{2\tan^2\alpha}{\tan\alpha - \tan\beta} \tag{6}$$

$$A_3 = \frac{\tan\beta}{\tan\alpha - \tan\beta} \tag{7}$$

$$H_c = \frac{4S_u}{\gamma} \tag{8}$$

and V_f = volume of material which flows.

The values of H_T and the corresponding values of α will define a "volume curve" which slopes up to the right, as shown in figure 7.

5. Where the strength curve and the volume curve intersect, all conditions with regard to strength and geometry are satisfied simultaneously, with a factor of safety equal to one. The values of H_T and α at the point of intersection are those corresponding to the limiting stability conditions. The value of the flow distance L (fig. 5) can be calculated using the expression

$$L = \frac{H_T - H_c}{\tan\alpha} \tag{9}$$

This procedure is based on case histories of failures where the liquefied tailings became stable after flowing over slopes of less than 3° or 4°. Steeper slopes may require different analyses where viscosity, dynamic effects, drainage, and other factors are considered. It appears unlikely that tailings materials will come to rest on slopes steeper than about 9°. In such cases, it should be expected that flow will continue until a flatter area or a body of water is reached.

Conclusions

A review of case histories of failure shows that liquefied mine tailings composed of sand and silt sizes have some small residual strength after liquefaction, and they will come to rest at slopes of 1° to 4°. Mine tailings such as phosphates, which consist of clay-size particles and have water contents of several hundred percent, flow in much the same way as water when loss of impoundment occurs.

A simplified procedure has been developed for predicting how far tailings will flow in case of failure when the slope of the surface on which the material flows is less than about 4°. The procedure requires that the residual shear strength of the liquefied tailings be known. At present, the residual strength after liquefaction may be estimated based on values back-calculated from field experience. Further research is required to develop laboratory procedures for measuring this important property of mine tailing materials.

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SUMMARY OF RESEARCH ON ANALYSES OF FLOW FAILURES OF MINE TAILINGS IMPOUNDMENTS

by

J. K. Jeyapalan,[1] J. M. Duncan,[2] and H. B. Seed[2]

INTRODUCTION

While earth dam engineering has evolved with the development of a large body of published theory and engineering practice, tailings embankment design and construction has received relatively little geotechnical engineering input until recently. Thousands of large and small mine waste piles and impoundments that have received no engineering attention are scattered throughout the United States and other parts of the world. Because of the poor quality of construction and maintenance, many of these dams have failed, and the existence of these potentially hazardous impoundments is of considerable concern to the public and to the mining industries. A characteristic common to most tailings dam failures is that the mine tailings tend to liquefy and flow over substantial distances, with the potential for extensive damage to property and life. Failures of El Cobre Dam, Chile (1965); Aberfan, Wales (1966); Buffalo Creek, W. Va. (1972); and Mochikoshi Dam, Japan (1978) are examples of such catastrophic dam incidents. In those four incidents, 450 lives were lost, and the loss of property was approximately $200 million. Table 1 summarizes some of the master failures that have been described in the literature.

In order to be able to assess the potential for damage in case of such a failure, it is necessary to be able to predict the characteristics of the flow and the possible extent of flood movement. Developing procedures for such analyses was the purpose of the research summarized in this paper. The study involved the following series of investigations:

1. Review of the available literature on the behavior of liquefied tailings and similar earth materials.
2. Review of the available methods of analyzing the flow of fluids from behind a breached dam.
3. Development of new analysis procedures to study the characteristics of the flow of mine tailings after loss of impoundment.
4. Experimental study of the flow of highly viscous fluids released suddenly from impoundment to check the theory previously developed.
5. Application of the analysis procedures to field cases for which sufficient details were available to permit comparison between calculated and observed behavior.

This paper deals briefly with the development of an analytical procedure for predicting the extent of flow of liquefied tailings if a failure should occur and comparison of the theoretical results obtained with field observations in three cases where such flows developed.

Rheological Properties of Liquefied Tailings

In order to analyze the flow phenomena associated with events of this type, it is necessary to use a suitable rheological model to represent the behavior of tailings during flow. Several rheological models were reviewed and an appropriate model for the liquefied tailings was chosen.

The subject of behavior of loose sands under undrained loading has been studied thoroughly and is well understood. Typical behavior of a loose sand under undrained loading conditions is shown in figure 1. Because the volume of such a saturated sample of loose sand can not change during the undrained shear test, the contractive volume change tendency causes a large positive change in pore pressure, as shown in the lower part of figure 1. The peak strength of the sample is reached at a small value of axial strain (typically less than 1 percent), and the shearing resistance then decreases rapidly to a small residual value. At strains larger than about 3 percent, the shear strength remains constant at the residual value.

Mine tailings consist predominantly of sand and cohesionless silt sizes, and it would be expected that their behavior under undrained loading conditions would be similar to that of loose sand.

The data described indicate that neither loose sands nor tailings lose all strength when they fail. Instead, their shearing resistance decreases to a finite low value and at a given

[1] Presently at Department of Civil Engineering, Texas A&M University, College Station, Tex.
[2] University of California at Berkeley

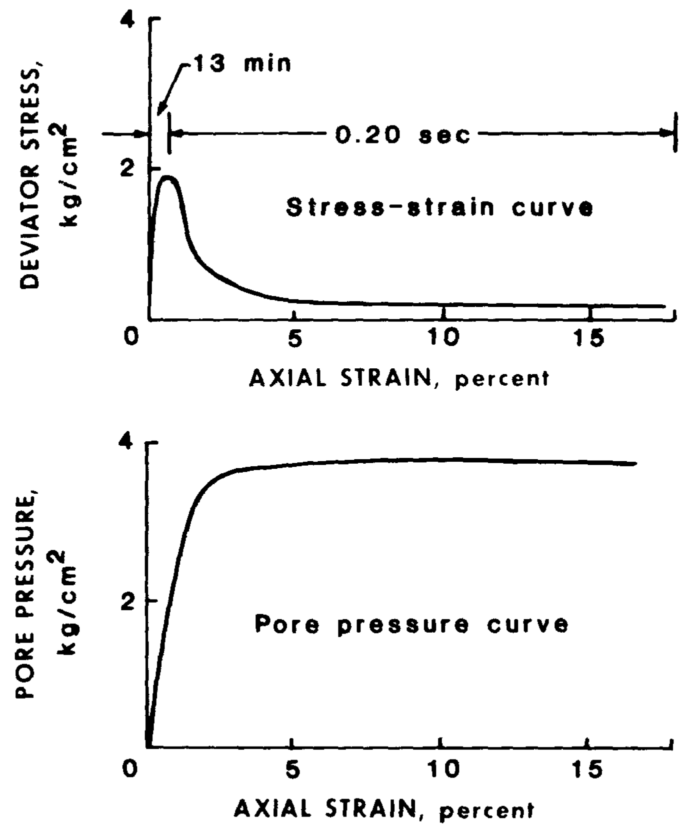

Figure 1.—Behavior of loose sand under undrained loading (after Castro, (3)).

strain rate remains constant thereafter. However, the residual shear resistance increases with strain rate. Therefore, a Bingham[3] plastic model as shown in figure 2 was chosen to represent the behavior of tailings materials during flow. The mathematical representation of this type of behavior is expressed by the equation:

$$\tau = \tau_y + \eta_p \gamma \qquad \text{for values of} \qquad \tau > \tau_y \qquad (1)$$

where τ_y and η_p are referred to as the Bingham yield strength and plastic viscosity. This model can also be written as an apparent viscosity model as shown in figure 2. The ranges of values expected for the Bingham parameters for tailings can be estimated by comparing the characteristics of these deposits with the properties of similar materials. Based on such comparisons, the probable ranges of yield strength and plastic viscosity of types of tailings other than phosphate tailings are shown in the spectra in figure 3 (11).[4]

Dimensionless Numbers for Typical Tailings and Associated Flow Regimes

Whether the flow of liquefied tailings will be laminar or turbulent can be determined using the procedure suggested by Hanks and Pratt (8) which is based on determination of the Reynolds and Hedstrom numbers for the liquefied material. Typical ranges of various parameters, flow velocities, depths of flow, total unit weights, yield strengths, and plastic viscosities for liquefied mine tailings are given in table 2. The probable values for minimum and maximum dimensionless parameters, Reynolds number, and Hedstrom number are also listed in this table. The probable ranges of Reynolds number for flows of phosphate tailings and other tailings are plotted with the probable ranges of Hedstrom number in figure 4. It is apparent from this plot that, based on the Hanks and Pratt criterion, the flow of phosphate tailings would be expected to be turbulent, whereas flows of other types of tailings would be expected to be laminar. Therefore, different types

[3] Reference to specific trade names is made for identification only and does not imply endorsement by the Bureau of Mines

[4] Underlined numbers in parentheses refer to items in the list of references at the end of this report

TABLE 1.—List of failures of tailing dams

Name and location of dam	Year of failure	Consequences of failure	Type of tailings	Method of construction	Cause of failure	Height of Dam, m	Volume flowed, million tons	Distance inundated, km	Source
Barahona, Chile	1928	54 killed	Copper	Upstream	Seismic	61	4.0	NA	Dobry and Alvarez (5)
Old El Cobre, Chile	1965	210 killed	Copper	Upstream	Seismic	35	2 0	12	do.
New El Cobre, Chile	1965		Copper	Upstream	Seismic	15	5	12	do.
Hierro Viejo, Chile	1965	Pollution	Copper	Upstream	Seismic	5	001	1	do.
Los Maquis, Chile	1965	Pollution	Copper	Upstream	Seismic	15	03	5	do.
La Patagua, Chile	1965	Pollution	Copper	Upstream	Seismic	15	.05	5	do
Cerro Negro, Chile	1965	Pollution	Copper	Upstream	Seismic	20	12	5	do.
Bellavista, Chile	1965	Pollution	Copper	Upstream	Seismic	20	10	2 5	do.
Ramayana, Chile	1965	Pollution	Copper	Upstream	Seismic	5	.0002	NA	do.
Bafokeng, South Africa	1974	12 killed	Platinum	Upstream	Seepage	20	5 2	45	Midgley (14), Blight (2)
Buffalo Creek, West Virginia	1972	118 killed	Coal	Tip	Erosion	18	55	64	Seals, Marr, Lambs (16)
Aberfan Tip 7, Wales	1966	144 killed	Coal	Tip	Seepage	37	.2	6	Aberfan Tribunal (10)
Gypsum, Texas	1966	Pollution	Gypsum	Upstream	Static	11	.2	3	Kleiner (12)
Mochhoshi, Japan	1978	Pollution	Gold	Upstream	Seismic	32	14	30	Marcuson (13)
Jupille, Belgium	1961	11 killed	Fly ash	Tip	Static	45	.3	6	Bishop (1)
Phosphate, Florida	1971	Pollution	Phosphate	Centerline	Seepage	4	.8	120	Wahler and Schlick (18)

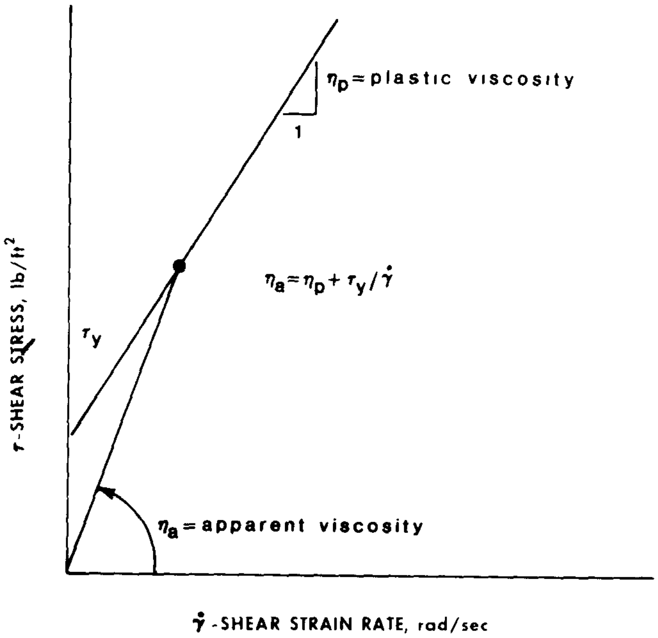

Figure 2.—Bingham plastic model for liquefied tailings.

TABLE 2.—Summary of flow parameters of typical liquefied tailings

Parameter	Probable minimum value	Probable maximum value
PHOSPHATE TAILINGS		
Total unit weight pounds per cubic foot	80	100
Yield strength . pounds per square foot. . . .	4 0 × 10^{4}	4 0 × 10^{-2}
Plastic viscosity . pound-second per square foot . .	2 0 × 10^{-4}	2.0 × 10^{-4}
Flow depth . feet . . .	2	5
Flow velocity . feet per second	5	50
Reynolds number	4 0 × 10^{4}	2 0 × 10^{5}
Hedstrom number	8.0 × 10^{3}	1 0 × 10^{5}
OTHER TAILINGS		
Total unit weight . pounds per cubic foot.	90	110
Yield strength . pounds per square foot	20	150
Plastic viscosity . pound-second per square foot	2	100
Flow depth . feet	5	50
Flow velocity feet per second .	5	20
Reynolds number . . .	10	300
Hedstrom number . .	100	350

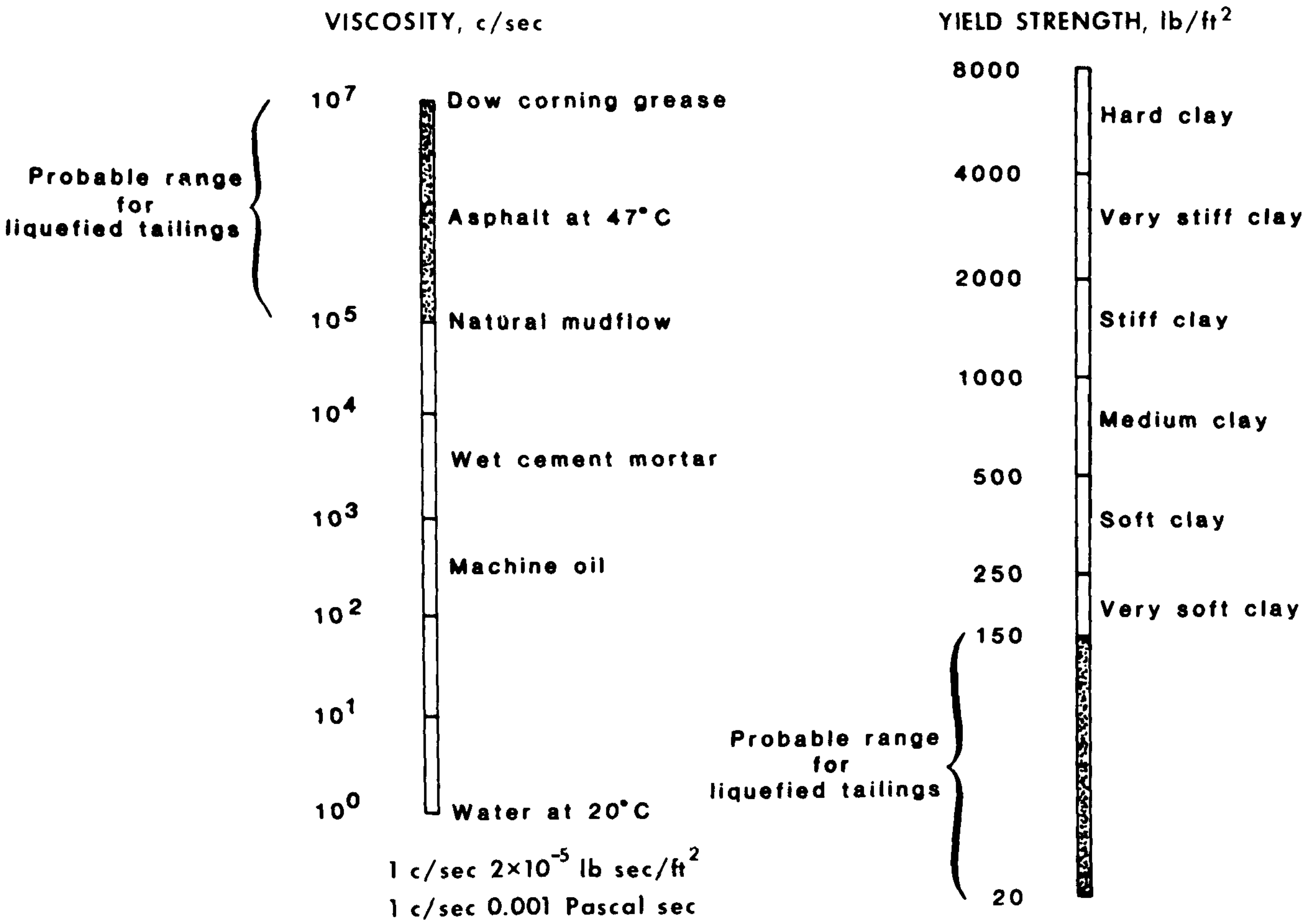

Figure 3.—Viscosity spectrum and yield shear strength spectrum.

of analyses would be applicable to phosphate tailings on the one hand, and all remaining types of tailings on the other hand.

Analysis Procedures Developed

Hydraulic wave theory was extended to analyze the laminar flow of tailings from behind a breached tailings impoundment. Various analysis procedures were developed and the applicability of these for different conditions is summarized in table 3. The analytical results can be presented in the form of prediction charts based on dimensionless resistance parameters R and S in the case of flow on planes:

$$\text{where} \quad R = \frac{2\eta_p}{\gamma H_o}\sqrt{g/H_o} \tag{2}$$

$$\text{and} \quad S = \frac{\tau_y}{\gamma H_o} \tag{3}$$

in which g = acceleration of gravity, H_o = initial height of impoundment, and γ = total unit weight of material. Typical predictive charts are shown in figures 5 and 6. For the more complex case of flow in prismatic valleys, a computer program (TFLOW) is required.

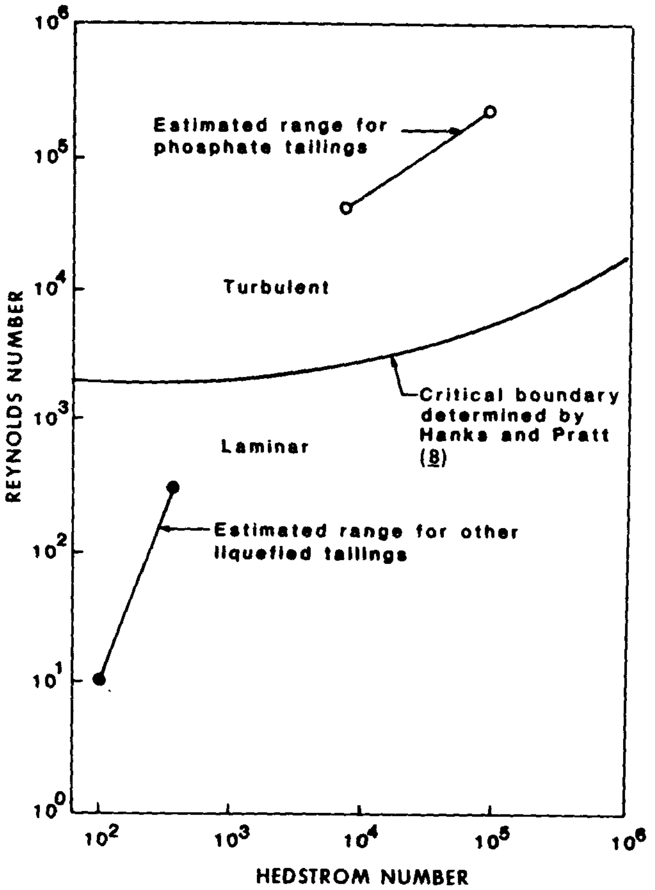

Figure 4.—Variation of critical Reynolds number with Hedstrom number.

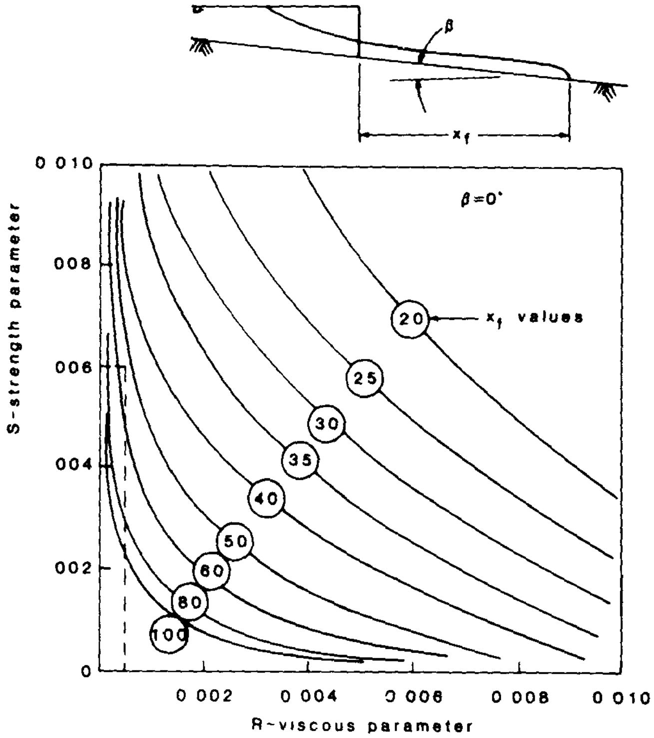

Figure 5.—Variation of inundation distance with resistance.

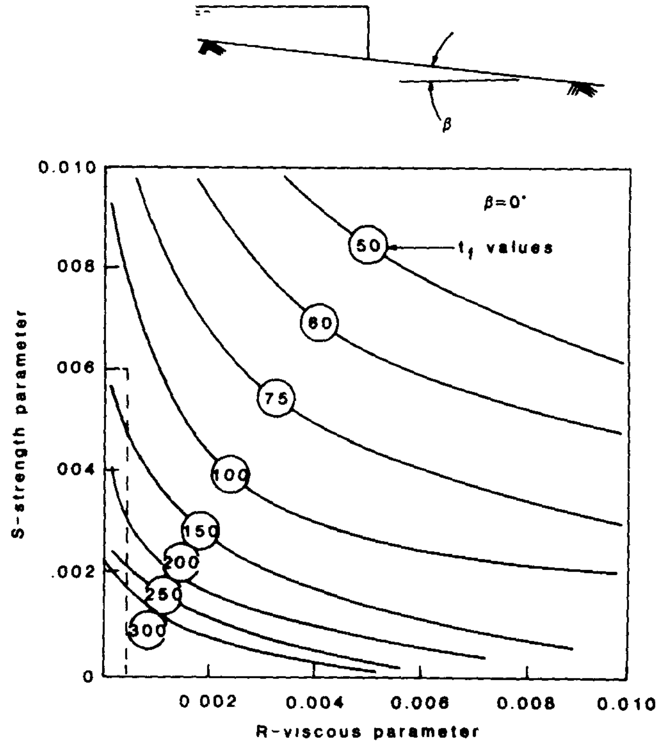

Figure 6.—Variation of freezing time with resistance parameters.

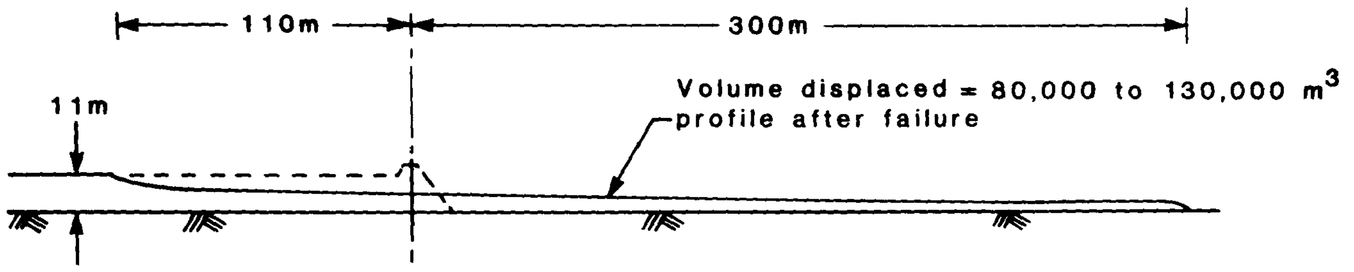

Figure 7.—Case 1—gypsum tailings dam failure.

Flume Studies

In order to assess the validity of the procedures developed for the analysis of flow failures of mine tailings impoundments, a series of flume studies was performed. The details of the experiments and comparisons of the experimental results with the results of calculations are given by Jeyapalan (11). There was close agreement between the analytical results and the laboratory test data.

Case Studies

In this section, the applicability of the charts and the computer program TFLOW for analyses of flow failures of mine tailings impoundments is illustrated through examples. Calculated results are compared with field observations for two cases where the flow was laminar, and the suitability of available hydraulic flood routing computer programs for analysis of turbulent flows is illustrated by comparison of calculations with observations for a case where the flow was turbulent.

Laminar Flow

The flow of most liquefied tailings would be expected to be laminar. Two cases of this type are discussed in the following paragraphs.

TABLE 3.—Summary of various solutions and their applicability

Solution	Applicable analysis
Ritter (15)	Flow of inviscid fluids in wide horizontal channels
Dressler (6)	Turbulent flow of water and phosphate tailings in wide horizontal channels
Tip theory (this study)	Laminar flow of viscous fluids in wide horizontal channels
Perturbed solutions (this study)	Laminar flow of viscous fluids in wide horizontal channels
Do.	Laminar flow of most tailings in wide horizontal channels
Do.	Laminar flow of most tailings in wide sloping channels
Do	Laminar flow of most tailings in prismatic sloping channels

Case 1

Kleiner (12) described the failure of a gypsum tailings impoundment in east Texas in 1966 The geometry of the site is shown in figure 7. The tailings were nonplastic silt with an average field water content of about 30 percent. Using a total unit weight of 90 lb/ft³, a simple slope stability calculation was done to determine the probable yield shear strength of the material. The value calculated by this means was 20 lb/ft². Using the viscosity-water content correlation developed as part of the research study, the probable viscosity of the material was estimated as 50,000 centipoise (1 lb sec/ft²). Using these values it was found that for the east Texas tailing impoundment, R = 0.0005 and S = 0.0006. For these values for R and S, the dimensionless inundation distance, x_f = 54 and the freezing time, t_f = 130, were obtained from the charts shown in figures 5 and 6. These dimensionless values were converted to dimensional values in Jeyapalan (11) and the results obtained from this analysis are compared with field observations in table 4.

This case was also analyzed using the computer program TFLOW for the case of a finite impoundment volume to better simulate the field conditions. This analysis gave an inundation distance of 470 m and a freezing time of 85 sec. It may be seen in table 4 that both of these values are smaller than those calculated using the dimensionless charts, because the charts do not consider the finite volume of tailings.

Since there was considerable lateral spreading in the direction perpendicular to the main direction of the flow, the field observations indicate a lower value for the inundation distance in comparison with the analysis. However, the overall agreement between the calculations and the field obser-

TABLE 4.—Comparison of theoretical results and actual observations for gypsum pond failure, east Texas

Flow characteristics	Observed values	Theoretical results from charts	Results using TFLOW for finite impoundment
Inundation distance, x_f^*, meters	300	550	470
Freezing time, t_f^*, seconds	60 to 120	132	85
Mean velocity, u_m^*, meters per second	2 5 to 5 0	4 2	5 5

NOTE—The notations x_f^*, t_f^*, and u_m^* are used here for consistency with other publications on this subject, where x_f, t_f and u_f refer to dimensionless parameters.

vations is reasonable. The degree of agreement obtained clearly depends to a great extent on the estimated values of n_p and τ_y.

Case 2

The Aberfan Tribunal (10) reported a detailed investigation of the Aberfan disaster that occurred in Wales. Similar calculations were performed using the charts and the computer program TFLOW. A triangular channel with slide slopes of one vertical on two horizontal was used to represent the gully in which flow occurred.

The results from the chart analysis are compared in table 5 with the field observations and the results obtained by the use of the computer program TFLOW. The inundation distance and freezing time from the chart analysis are considerably longer than the actual values, but the results calculated using the program TFLOW are in good agreement with observations. The larger inundation distance and freezing time from the chart analysis may be attributed to the implicit assumption of no channel side frictional resistance to flow. This case illustrates the importance of considering the cross sectional shape of the channel in the analyses.

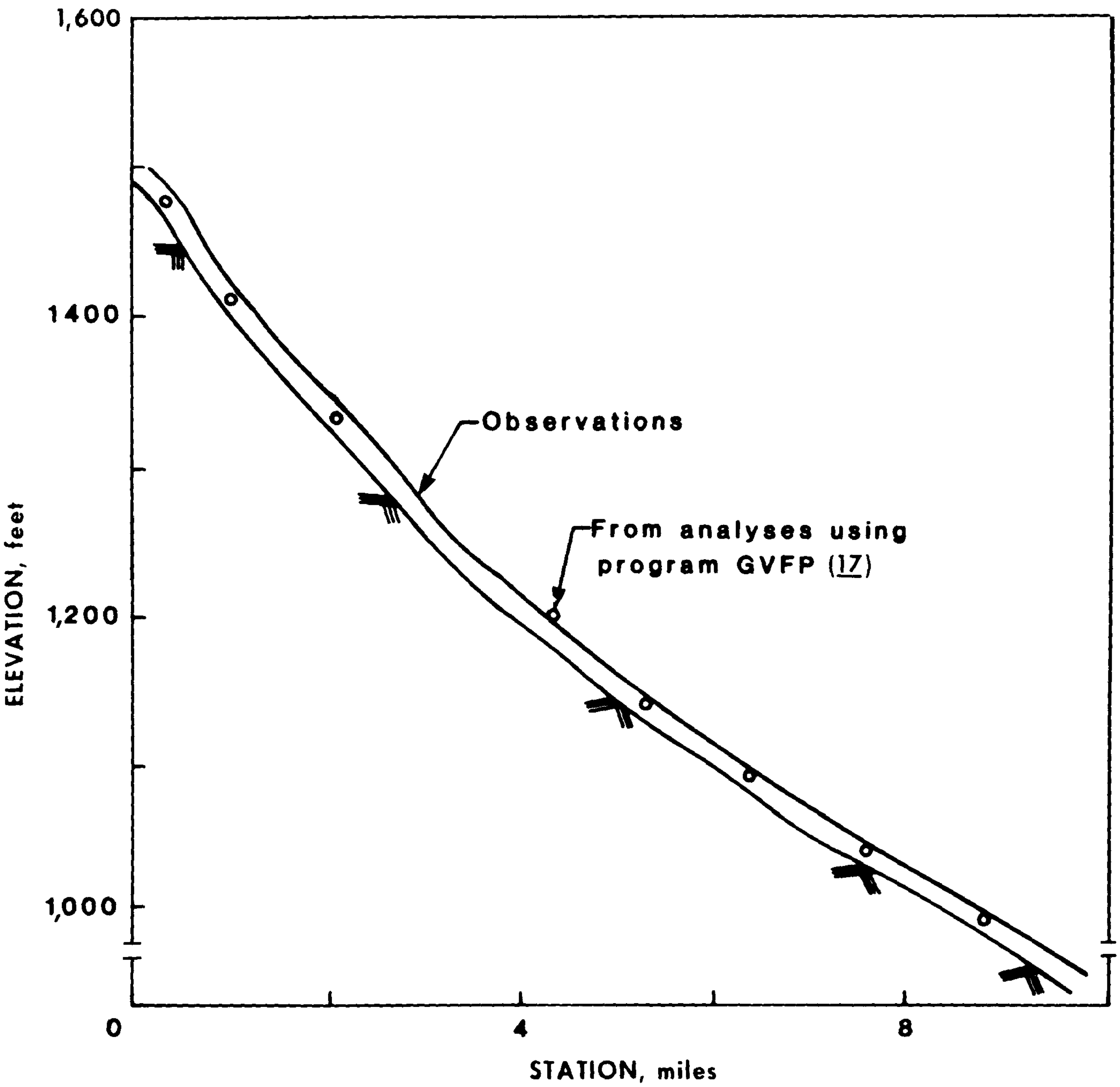

Figure 8.—Buffalo Creek flood—maximum water elevations.

TABLE 5.—Comparison of theoretical results and actual observations for Aberfan slide, Wales

Flow characteristics	Observed value	Theoretical results using charts	Results using TFLOW for a triangular channel
Inundation distance, x_f^*, meters	600	1,700	670
Freezing time, t_f^*, seconds	120	260	116
Mean velocity, u_m^*, meters per second	5 0	7 3	6.0

Turbulent Flow

If the flow will be turbulent, as is likely with very fluid tailings materials such as phosphate tailings, existing flood routing computer programs (e.g., GVFP (17) and Fread (7)) can be used for the analyses. These programs for turbulent flow analysis incorporate resistance to flow through the use of the empirical Manning's n relationship rather than friction factors or fluid viscosity. Values of Manning's n (9) have been determined by laboratory tests using water, and it is not clear whether these same values are applicable when the fluid involved in the turbulent flow is not water. At the present time, it appears that the best approach may be to use slightly higher values for Manning's n than those applicable for water, for purposes of analyzing flows of fluids such as phosphate tailings.

The use of hydraulic flood routing computer programs (GVFP (17) and Fread (7)) for analyses of turbulent flows is illustrated in this section by application to a flood consisting of water and coal waste (Buffalo Creek).

Case 3

The Buffalo Creek coal waste embankment failure released a mixture of coal waste in a large amount of water which apparently flowed much like water. The flood produced by this failure was turbulent, and the program GVFP (17) was used to analyze the characteristics of this flood as a part of this research study. The maximum calculated water elevations during the flood was compared with field observations in figure 8. Also, computed travel times for the flood peak are compared with the observations in figure 9. It may be seen that the agreement between analyses and observations shown in these figures is good.

This case illustrates the applicability of available flood routing programs for analyses of turbulent flood flows to flow of highly fluid tailings deposits. These programs can be con-

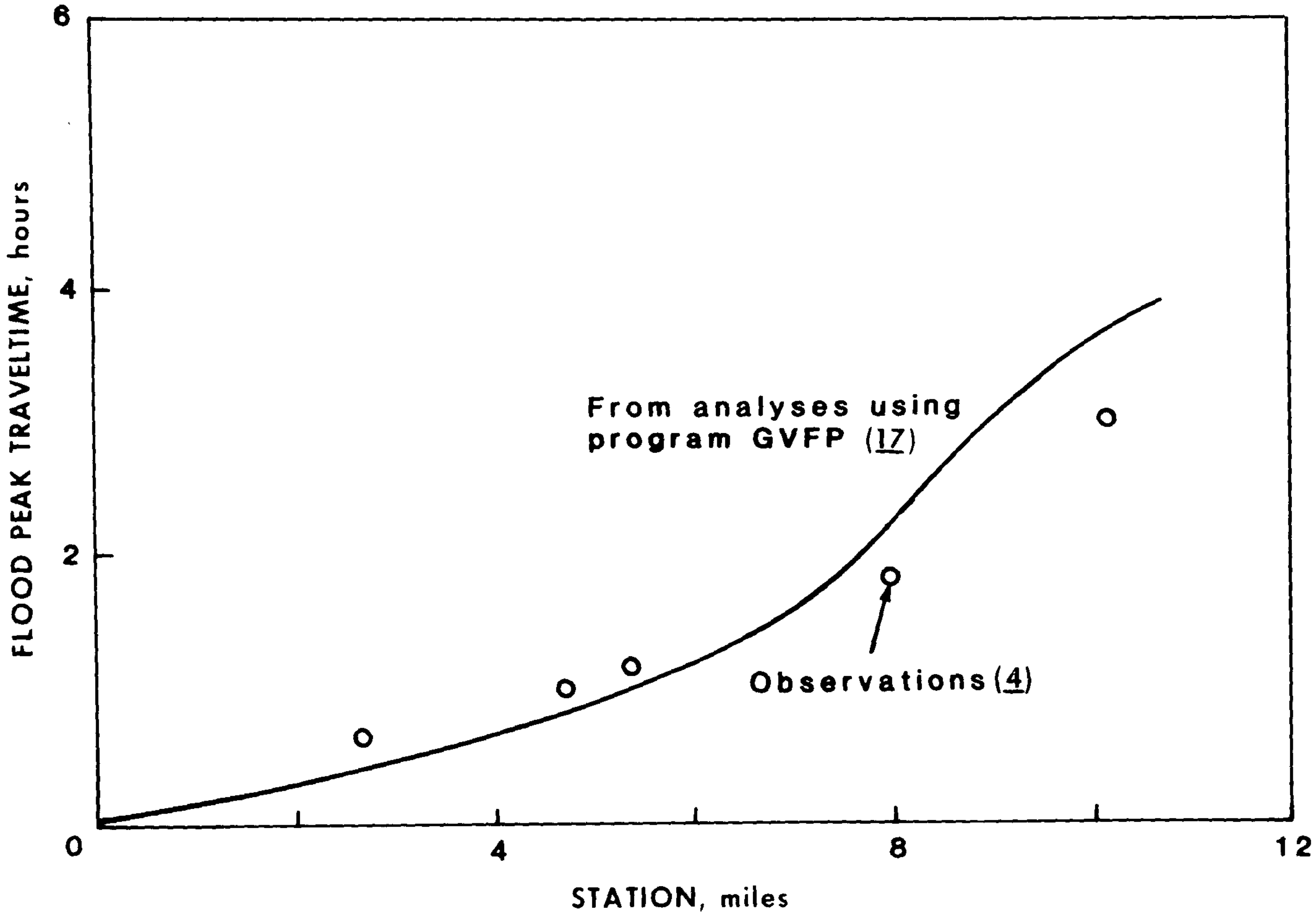

Figure 9.—Comparison of flood travel time from analyses and observation for Buffalo Creek flood.

veniently used with no modifications for the analysis of turbulent flows that are likely in the case of very fluid tailings such as phosphate tailings or the fluid mixture of water and coal waste which flowed down Buffalo Creek.

Conclusions

The main conclusions drawn from the research study are as follows:

1. Instances of flow failures of mine tailings impoundments indicate that the failure of these structures has considerable potential for damage to life and property in many cases.
2. The behavior of tailings materials during flow can be represented with reasonable accuracy by the Bingham plastic rheological model.
3. The currently available computer programs (GVFP (17) and Fread (7)) can be used without modification for analyses of potential inundation zones likely to result from turbulent flows of very fluid tailings such as phosphate tailings and the mixture of water and coal waste which flowed at Buffalo Creek.
4. The analysis procedures developed during this research study can be used for analyses of flow failures in more highly viscous tailings which undergo laminar flow. These procedures are applicable for flow of tailings on horizontal and sloping planes and in prismatic valleys. The analyses can be performed using charts in the case of flow on planes, and by means of a computer program (TFLOW) in the case of flow in prismatic valleys.
5. The flume experiments conducted as part of this research study indicate that the analysis procedures developed are reliable. Application of these analysis procedures to field cases also showed good agreement between calculations and field observations. Thus, the analysis procedures developed appear to provide a useful means for assessing the potential inundation regions downstream of mine tailings impoundments.

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CONTROLLED BURNOUT OF FIRES IN ABANDONED COAL MINES AND WASTE BANKS BY IN SITU COMBUSTION[1]

by

ROBERT F. CHAIKEN[2]

ABSTRACT

A novel approach to eliminating environmental and public safety hazards that are associated with fires in abandoned coal mines and waste banks involves the use of in situ combustion technology developed by the Federal Bureau of Mines to accelerate the burning of the wasted coals in place. This technology would be used under exhaust ventilation control conditions that would allow for total management of the hot gases produced. Combustion stoichiometries would be optimized to minimize unburnt combustibles and to maximize the heat content of the gas products, which will be exhausted at one or more fan locations. When necessary, scrubber systems would be employed to remove air pollutants, such as sulfur dioxide; heat utilization systems (process heat, steam, and electricity) would also be employed to offset operational costs. Ultimately, complete burnout would solve the fire and acid water formation problems of the abandoned coal mine or waste bank.

Pertinent technical data from the burning of tonnage quantities of coal and coal refuse under simulated in situ conditions are discussed in terms of burnout control. Based on these data a field trial of the concept is being prepared at an existing abandoned coal mine fire site (approximately 1½ acres in extent) located 6 miles from the Bureau's research facilities at Bruceton (Calamity Hollow mine fire project). The field burnout ventilation system has been designed to handle exhaust gas temperatures of approximately 1,400° C (approximately 2,600° F) and thermal power output levels of approximately 5 MW. In the present configuration, the heat will be wasted through the fan exhaust stack.

Introduction

A significant aspect of all past and current coal mining is wasted coal. Wasted coal occurs in abandoned mines, which, because of the exigencies of underground mining operations, often contain as much coal as was extracted; and coal refuse piles, which are accumulations on the surface of reject material from coal preparation plants and from underground mining operations. Individual refuse piles (or waste banks) may contain millions of cubic yards of solid waste, with a combustible content ranging from 15 to 50 pct by weight.

Fires continue to occur in this wasted coal, introducing significant hazards to public health and safety, such as emissions of toxic and obnoxious fumes to the atmosphere and destruction of residential and commercial buildings. Once established, these fires can smolder for decades, and extinguishing them by conventional methods of sealing, dig-out, and quench is costly and hazardous (6–11).[3]

The magnitude of the problem can be appreciated by examining the cost estimates for extinguishing the wasted coal fires on abandoned lands (7). In a 1968 survey by the Bureau, 292 waste banks were found to be on fire and involved about 270 million tons of refuse and 3,200 acres of land. In a 1977 survey, 261 coal deposits were classified as burning. It has been estimated that the cost of extinguishing the 292 burning waste banks would be $468 million, and the cost of controlling the 261 fires in active coal deposits would be $75.6 million. These cost estimates probably should be reexamined and brought up to date as they undoubtedly do not represent current pricing structures. These estimates are for fix-up only—they do not remove the fire potential of the wasted coal, nor do they include the effective value of the wasted coal if it could be utilized during the extinguishment and reclamation process.

Although the intrinsic public health and safety benefits of controlling fires on abandoned coal mined lands cannot be denied, neither can the high costs of conventional control techniques be ignored. Such costs present a serious constraint on how much and how fast corrective action can be accomplished.

[1] This paper is a summary and update of BuMines RI 8478, "Controlled Burnout of Wasted Coal on Abandoned Coal Mine Lands," by Robert F Chaiken, 1980.

[2] Supervisory research chemist, Pittsburgh Research Center, Pittsburgh, Pa

[3] Underlined numbers in parentheses refer to items in the list of references at the end of this report.

Burnout control of fires in wasted coal is a novel concept that offers a possibly more cost effective alternative approach to conventional fire control methodology. The concept involves acceleration of the wasted coal fire to burn the fuel completely, but to do so in an environmentally and economically sound manner. The in situ combustion control techniques currently being developed by the Bureau are expected to accomplish this goal, and at the same time allow for possible utilization of the thermal energy produced during burnout. This paper describes the burnout control concept and the status of some of the Bureau's efforts at applying it to coal mine and refuse bank fires.

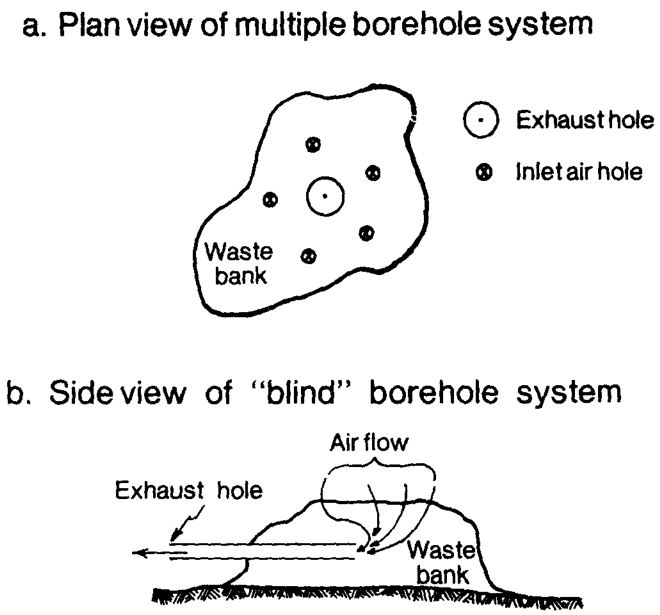

Figure 1.—Two possible borehole arrangements for burnout of a coal waste bank fire.

Burnout Control of Wasted Coal

The burnout control concept involves the controlled acceleration of the fire by air injection so that complete fuel removal is accomplished in a relatively short time compared to a "normal" waste coal burn time (i.e., many decades). By developing burn channels or zones in the burning wasted coal (i.e., waste bank or underground mine), it should be possible through exhaust ventilation techniques to effect a controlled complete burnout of the coal and all other nearby combustibles, such as carbonaceous rock materials and pyrites. The process can perhaps be best envisioned from figure 1 which refers to a waste bank on fire. Figure 1b depicts an exhaust borehole driven into a waste bank. By sucking on this borehole a negative pressure is maintained within the waste bank which causes ambient air to permeate into the bank. The air flow will accelerate the burning while all the combustion gases will be exhausted through the borehole. In this case the air flow will be controlled by the amount of suction and the natural permeability of the waste pile. Alternatively, as shown in figure 1a, air inlet pipes can be inserted into the waste bank to enhance the effective permeability of the refuse.

Figure 1a would also be applicable to underground mine fires, where the boreholes now extend into the burning entries. The nature of mine fires is such that propagation of the fire always proceeds in the direction of the air flow; hence through proper positioning of the air inlet holes, it is possible to direct the fire underground.

A simple burnout exhaust ventilation scheme is shown in figure 2. Here, a basic exhaust blower/duct system incor-

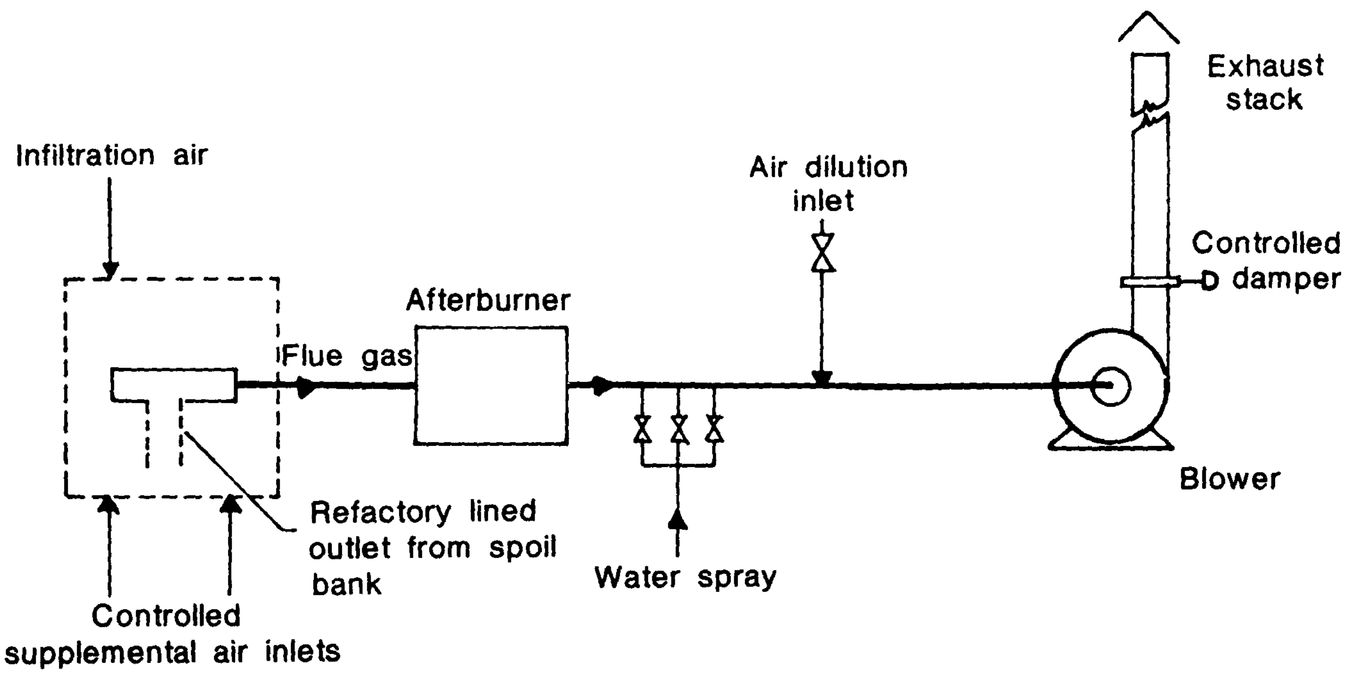

Figure 2.—Schematic of burnout exhaust ventilation system.

porates an afterburner to insure complete combustion of the gases leaving the underground fire, and a water spray/air dilution system to cool the hot gases before they enter the blower. With the ventilation scheme shown in figure 2 the heat would be wasted through the stack. Alternatively, one or more heat extraction devices, such as a steam boiler or air/air-heat exchanger, could replace the air and spray cooling method shown so that the heat generated during burnout could be utilized.

The burnout control concept has a number of distinct advantages:

1. The affected coal mine workings or refuse bank will be at negative pressure, relative to ambient; hence little or no fumes will be emitted to the atmosphere except at the fan exhaust points.
2. Accumulation of all the fumes at fan exhaust points will enable postburn incineration of the exhaust to insure complete combustion of carbon monoxide and unburnt soot and hydrocarbons to carbon dioxide and water. If required, scrubber treatment can also be applied to remove air pollutants, such as sulfur dioxide and particulates.
3. Controlled air injection might enable the burn to be carried out under oxidation conditions favorable for sulfur dioxide to react in situ to form solid sulfates, for example, calcium sulfate which would remain in the ground.
4. The heat of combustion of the burning fuel will appear as sensible heat in the exhaust products—perhaps at a temperature as high as 1,000° C (1,832° F). This heat can be recovered onsite for local use such as production of steam, hot water, process heat, and electricity.
5. The complete burnout of combustible material (carbonaceous material and pyrites) in a mine or waste bank will finally solve the environmental problems of an active fire. In contrast, fires extinguished by wetting and sealing leave wasted coal with its potential for reignition and acid water formation.
6. The solid residue from the complete burnout of a coal refuse bank is red dog, a gravel substitute with commercial value.

The described advantages of burnout control of wasted coal would suggest that the technique could be a panacea for local environmental and energy problems. Although as yet there has been no complete demonstration of burnout control in an actual abandoned mine or waste bank, several successful large-scale experiments have been carried out with coal and coal waste under simulated in situ burning conditions. Also, a field trial of the burnout of an abandoned coal mine fire is currently being prepared.

Simulated In Situ Combustion of Coal

The Bureau background for burnout control of wasted coal stems from its studies of the burning of coal underground (2–4). A novel sealed surface trench burn facility was developed that enabled tonnage quantities of coal (or coal waste) to be burned under simulated in situ combustion conditions utilizing a ventilation scheme similar to that depicted in figure 2. A photograph of the facility, which is located at the Pittsburgh Research Center, is shown in figure 3. Figure 4 is a detailed diagram of the trench as set up for a simulated coal mine entry fire.[4]

In this case the trench contains approximately 45 tons of rubblized coal (Pittsburgh seam) with a 1-ft², 35-foot-long axial channel formed through the center of the coal to represent the mine entry. The exhaust fan (15,000 scfm, 18-inch H_2O head capacity) maintains a negative pressure in the trench which induces air flow to the channel through the air inlet pipes to sustain burning along the walls of the channel. At the same time the fan exhausts the gaseous combustion products from the end of the channel. While the fire studies that have been carried out in the surface trench facility cannot be reviewed here in their entirety, some results pertinent to burnout of a mine fire and a coal refuse pile are discussed.

During one simulated mine entry fire experiment of 33-hour duration, the thermal output of the in situ combustion process was readily controlled between 1.0 and 1.7 MW, and the exiting combustion products were at very high temperature—about 1,600° C (2,900° F). Except for sulfur dioxide, the exhaust gas was exceptionally clean in terms of low concentrations of carbon monoxide, nitrogen oxides, and particulates. Figure 5 shows data time plots obtained in this experiment from measurements of the flue gas at a position about 20 feet downstream of the channel exit (that is, following some dilution with air). The exhaust temperatures at this point are approximately 1,100° C (2,000° F). It is noteworthy that while the observed sulfur dioxide emission (1,100 ppm) is commensurate with the sulfur content of the coal (2 wt-pct), the observed nitrogen oxides emission (110 ppm) is considerably lower than what might be expected on the basis of the fuel-nitrogen content (1.5 wt-pct). The nitrogen oxides emission is actually about a factor of 5 less than values reported for pulverized fuel combustors. Similarly, the particulate emissions appeared to be quite small.[5]

During the entire experiment (ignition, steady burning, and cool down), the exhaust ventilation control system provided total management of the combustion gas flow. Despite air leakage into the trench and total burnout and cave-in of a portion of the trench, no combustion products escaped to the atmosphere except through the exhaust system.

Another in situ burning experiment of approximately 75-hour duration was also carried out with 90 tons of bituminous refuse obtained from a waste bank that is currently on fire. Proximate analysis indicated the refuse was comparable to a 75 pct ash coal with a heating value of 2,800 Btu/lb. In the simulated refuse fire experiment, the 1-ft² ventilated axial channel served only as a zone for ignition. After the refuse was ignited the inlet air lines were closed, and combustion air was permeated into the trench through the top exposed surface of the coal waste under the negative pressure induced by the fan.

Figure 6 shows data time plots for this controlled burnout experiment. They are somewhat similar to the coal channel

[4] All dimensions shown in figure 4 are in metric units.

[5] In a second sealed trench burn tests involving a channeled solid block of coal, results similar to the rubblized coal burn were obtained, and in this case the measured particulate emissions at the fan exhaust stack averaged only 0.008 lb/10⁶ Btu over a 45-hour burning period.

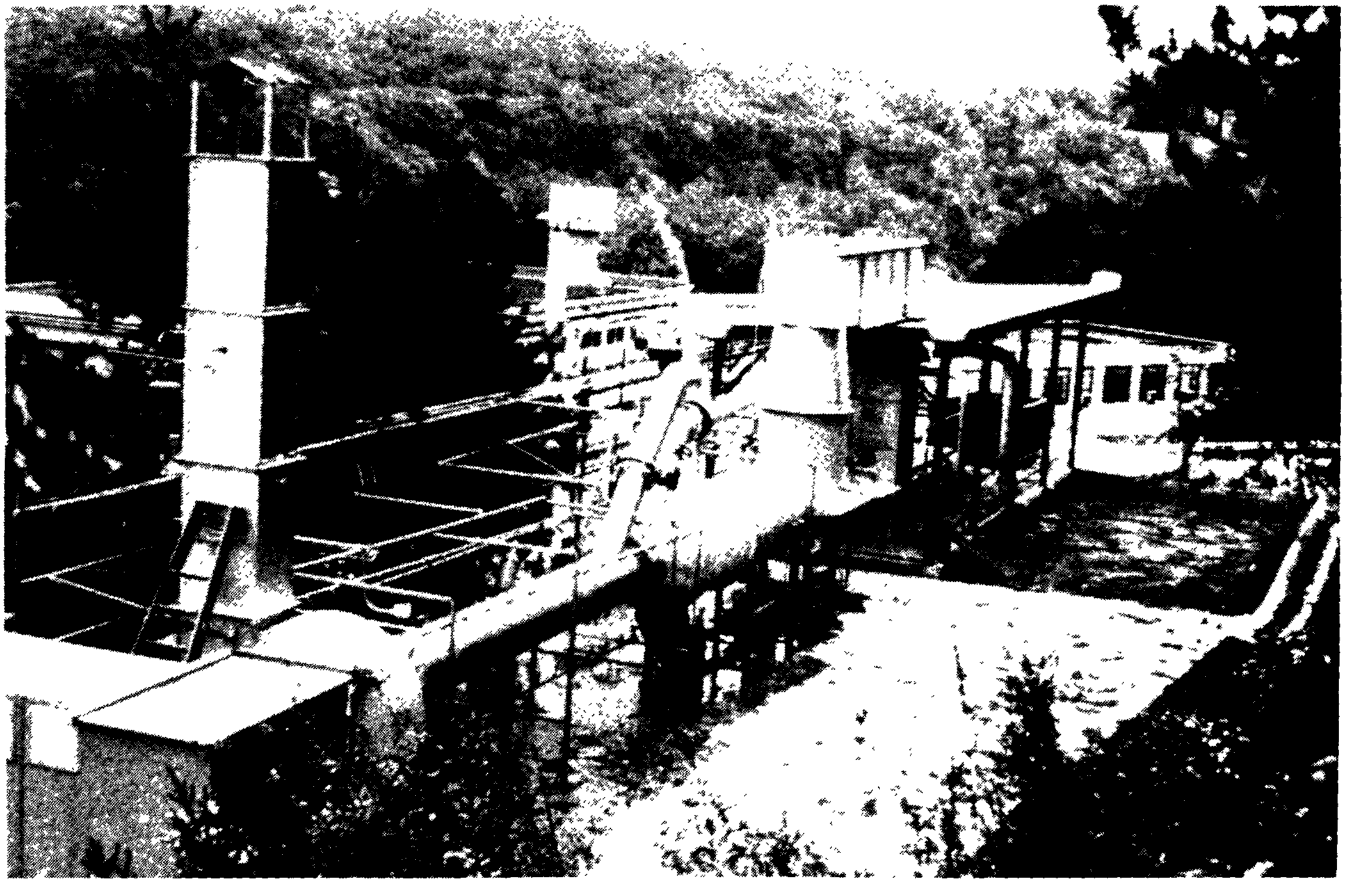

Figure 3.—Surface trench burn facility at Bruceton.

burning data previously described. The thermal power level of the exhaust is in the range of 0.5 to 0.8 MW. The 600° C (1,112° F) exhaust temperature shown in figure 6, which was measured 20 feet downstream of the channel exit, is about one-half the values recorded directly at the channel exit. This indicates the occurrence of considerable dilution of the combustion products by air leaking directly into the thermal breech line The observed elevated oxygen concentrations in the exhaust (approximately 15 pct at the 20-foot-downstream station) also reflects this air leakage. Without dilution by this air leakage the exhaust temperature and oxygen concentration would be approximately 1,000° C (1,832° F) and approximately 8 pct, respectively, in keeping with values expected from complete combustion of the fuel (1). It should be noted that air dilution that decreases the exhaust temperature also increases the mass flow rates; hence the thermal power level of the exhaust is relatively invariant to the air dilution. The low CO levels observed in the exhaust is further evidence that complete combustion was achieved.

It is also seen from figure 6 that the pollutant emissions, NO_x and SO_2, are quite low indicating a very clean burning system. Both the NO_x and SO_2 are less than what would be anticipated based upon the fuel-sulfur and fuel-nitrogen (2.2 pct and 1.6 pct, respectively on an ash-free, moisture-free basis). In this case of the NO_x it is speculated that the in situ burning geometry promotes the following redox reaction to occur:

$$CO + NO \text{////}\rightarrow CO_2 + N_2$$

In the case of SO_2 it is speculated that the high effective ash content of the coal waste acts as a solid scrubber system for the SO_2 produced by burning; e g.,

$$Ca\,O(s) + \frac{1}{2}O_2 + SO_2 \text{////}\rightarrow CaSO_4(s)$$

$$Mg\,O(s) + \frac{1}{2}O_2 + SO_2 \text{////}\rightarrow MgSO_4(s).$$

In this case the solid sulfates formed would remain underground as part of the fused ash (or red dog). Chemical analyses of the residue to obtain a mass balance on the sulfur have not been carried out as yet

As in the case of the simulated coal mine entry burn, the particulate emissions from the in situ burning of coal waste were quite low. In fact they were essentially negligible based upon the simple sampling system employed at the fan exhaust stack.

One set of conclusions that might be drawn from the surface trench burn experiments is that the controlled in situ burning of coal and coal waste can be carried out efficiently, cleanly, and with total management of the heat and gases produced. However, surface subsidence cannot be truly simulated in the surface trench experiments. Questions still to be answered on the long-term control of an actual burnout system are (1) what are the effects of crevices and channels

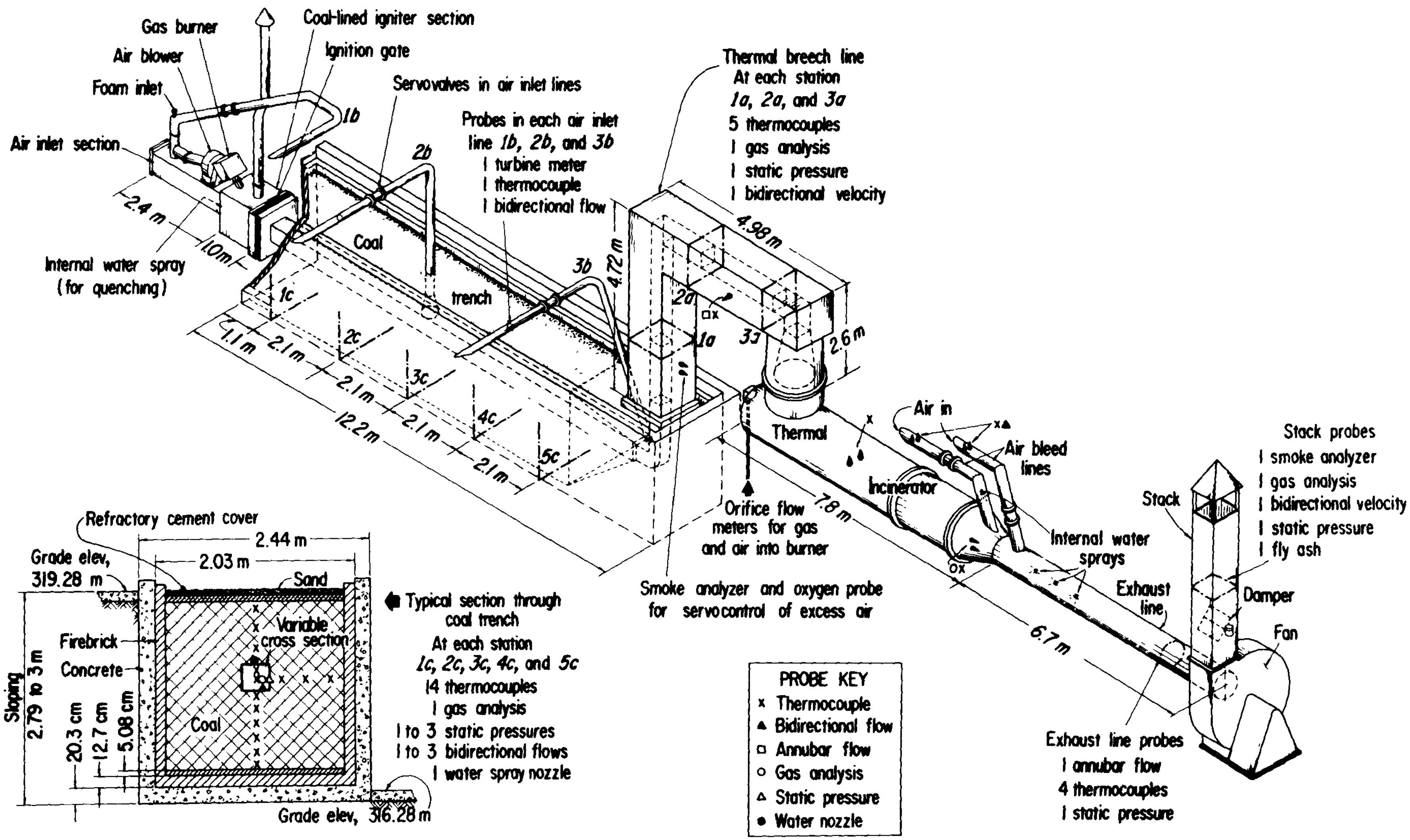

Figure 4.—Simulated coal mine entry fire: diagrammatic setup of experiment (all dimensions in metric units).

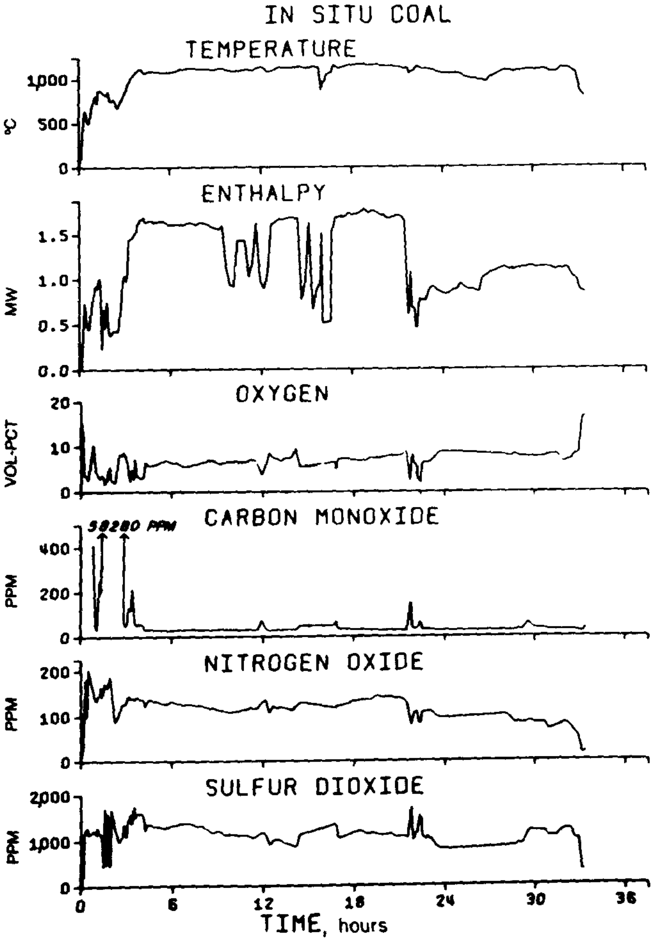

Figure 5.—Data-time plots for burnout of a simulated coal mine entry fire.

on the waste bank permeability and the effective burn volume; (2) what are the effects of roof falls on the pressure drop requirements for burning mined out entries, and would this lead to deterioration of the combustion efficiency; and (3) what surface subsidence effects will be experienced from complete burnout?

These questions are important, but they are site-selective and can only be answered through field trials in actual abandoned coal mines and actual coal waste banks.

Field Trial of Burnout Control

To answer the question of long-term control of a burnout system, a full-scale field trial of the concept is currently being developed at the site of an abandoned coal mine fire located at the Calamity Hollow section of Jefferson Borough, Pa., about 6 miles from the Pittsburgh Research Center. The work efforts to date have involved mostly site preparations, designs, procurements, and constructions. Startup of the burnout operations is scheduled for June 1981. This section of the paper is essentially a brief report on the status of the Calamity Hollow fire project.

The abandoned coal mine fire is in a shallow turn-of-the century drift mine in the Pittsburgh seam, and has been smoldering for about two and one-half decades (fig. 7). In 1964, the Bureau attempted to extinguish the fire by digging an isolation trench (outcrop to outcrop) and surface sealing, but these efforts were not totally satisfactory. Isolation of the fire by the trench barrier to about 1½ acres of coal seam was successful, but surface sealing failed to extinguish the fire; primarily due to inadequate upkeep of the surface seal.

In carrying out the field trial, the site was first surveyed (December 1979) to provide a base map on which lease lines were drawn. Site preparations involved cutting access roads for both light- and heavy-duty truck traffic, grubbing and cleaning the 3-acre work area, and leveling and slagging the ground for the proposed burnout system. A cyclone fence was installed around the site to provide for public safety and to prevent vandalism.

Concurrent with the site preparations, an exploratory drilling program was begun to accomplish several objectives: (1) Boreholes were required to determine the exact location of the isolation trench, the original highwall, and the underground high-temperature zones; (2) the boreholes would help

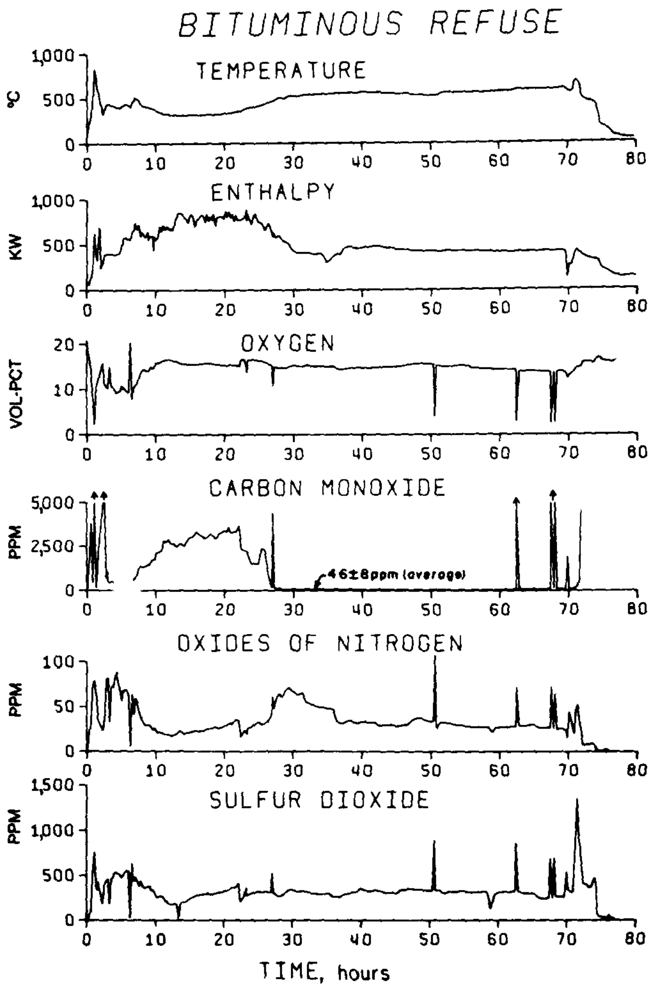

Figure 6.—Data-time plots for burnout of a simulated coal waste bank fire.

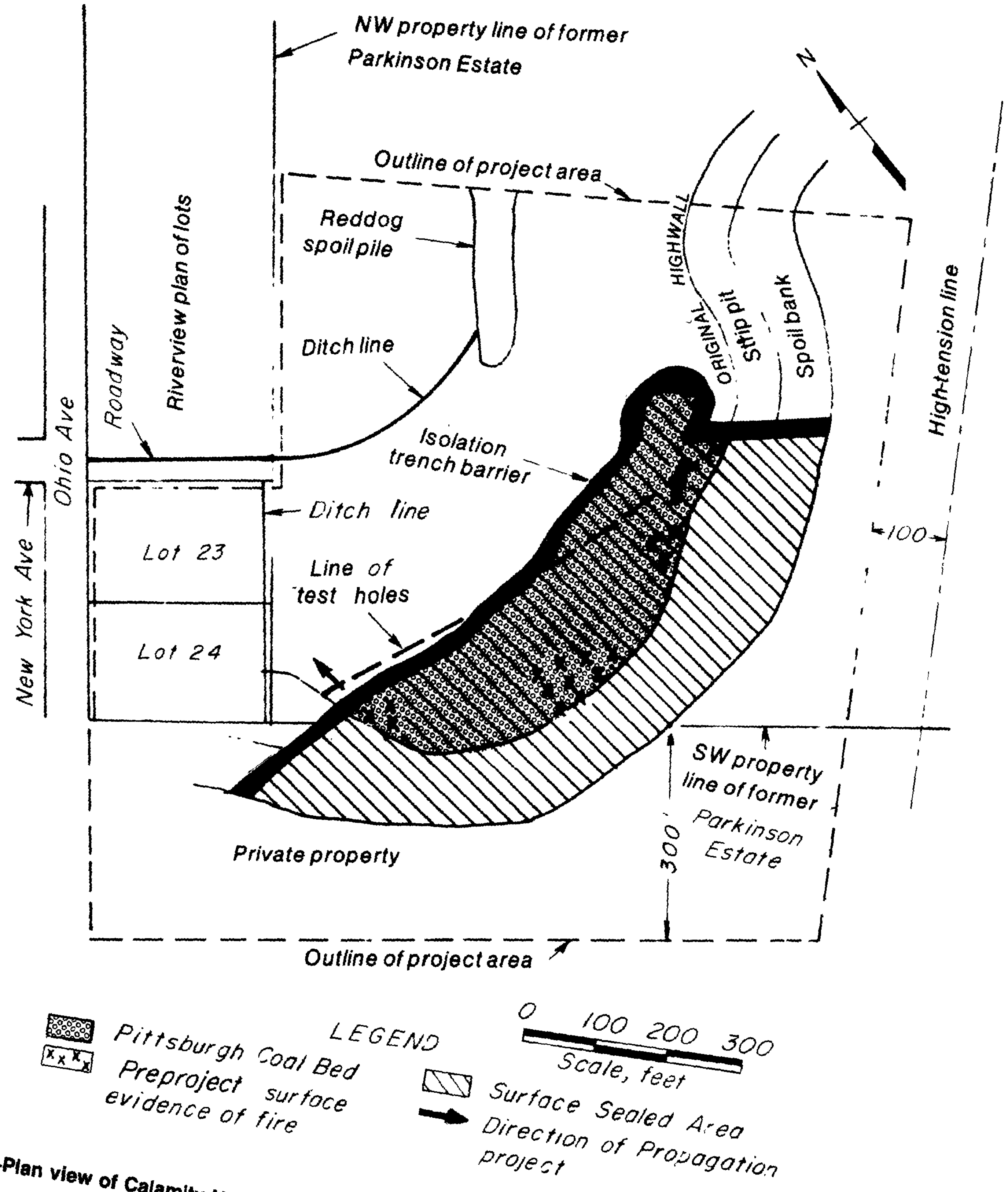

Figure 7.—Plan view of Calamity Hollow mine fire project site.

in establishing the mining pattern (note: mine maps are unavailable for the site) and the air communication between various underground zones; (3) sensors in the boreholes will be used to monitor the progress of the fire; and (4) boreholes will serve as inlets to supply combustion air to the fire, and to control the path of the fire during burnout.

With exploratory drilling completed (about 100 4-inch holes), temperature measurements and air communication tests were carried out to check the condition of the old mine workings. A small mobile fan setup was built (400 scfm at 60-inch H_2O head capacity) and used to draw air from the mine while flows were measured at surrounding boreholes. With these data a location was selected for a large 48-inch-OD, 25-ft-deep exhaust hole. This hole was augered into a warm zone (approximately 100° F), about 75 feet from a communicating high temperature zone (approximately 800° F). A large diameter (44-inch-OD, 36-inch-ID) water-jacketed manifold pipe, which was previously fabricated and lined with insulation brick, was installed in the hole. This pipe will serve as the main exhaust hole for the hot gases produced during burnout.

The ventilation system designed for Calamity Hollow is shown in figure 8. Heat generated during burnout will be wasted through the stack. To date, fabrication of the high-temperature elbow and horizontal ductwork has been completed and the components installed. The first section of ductwork leading off the elbow connection is refractory lined and will serve as an afterburner to complete combustion of the exhaust gases as required. The ductwork design includes instrument-activated automatic valves for controlling both the flow of hot exhaust gases and the cooling of these gases with ambient air. The instrumentation and control systems are currently being assembled.

The diesel generator (350-kW capacity) and the exhaust fan (25,000 scfm, 40-inch H_2O head capacity) are mounted on a 50-ton-capacity flatbed trailer. The entire mobile unit is now positioned on temporary foundations at the site to receive the horizontal ductwork. The fan-motor-generator assembly is located on the cold side of the isolation trench barrier to protect it from potential subsidence. Not shown in figure 8 are two additional items that are planned for incorporation into the burnout system: (1) An accordian type expansion joint which will allow for relative displacement of the ductwork in all directions; and (2) a drop-out pot interposed between the fan and the horizontal ductwork. The drop-out pot will serve to catch large particles that may be sucked from the mine and, if required, as a scrubber device to control air pollutant emissions.

It is anticipated that major component fabrication and installation of the ventilation system will be completed in April 1981; instrumentation and controls will be completed during May 1981; and start up of the actual burnout will be sometime in June 1981. It is anticipated that the system will be capable of burning as much as 1,500 lb of coal per hour, developing 5 MW of thermal power in the exhaust. At this rate, the estimated 4,000 tons of coal remaining on the "fire" side of the isolation trench at Calamity Hollow would be consumed in about 235 days. However, it is most likely that the burnout will be stopped before that time in order to uncover the coal and examine in detail the underground residues. In this way detailed information will be obtained on the underground zones; for example: (1) The size and geometry of burning from a single exhaust hole; (2) the effect of high temperatures on the remaining coal pillars and surrounding rock strata; and (3) the physical and chemical nature of the residues that remain underground.

Summary

A novel approach to controlling the environmental and public safety hazards associated with fires in abandoned coal mines and refuse piles has been described. This approach

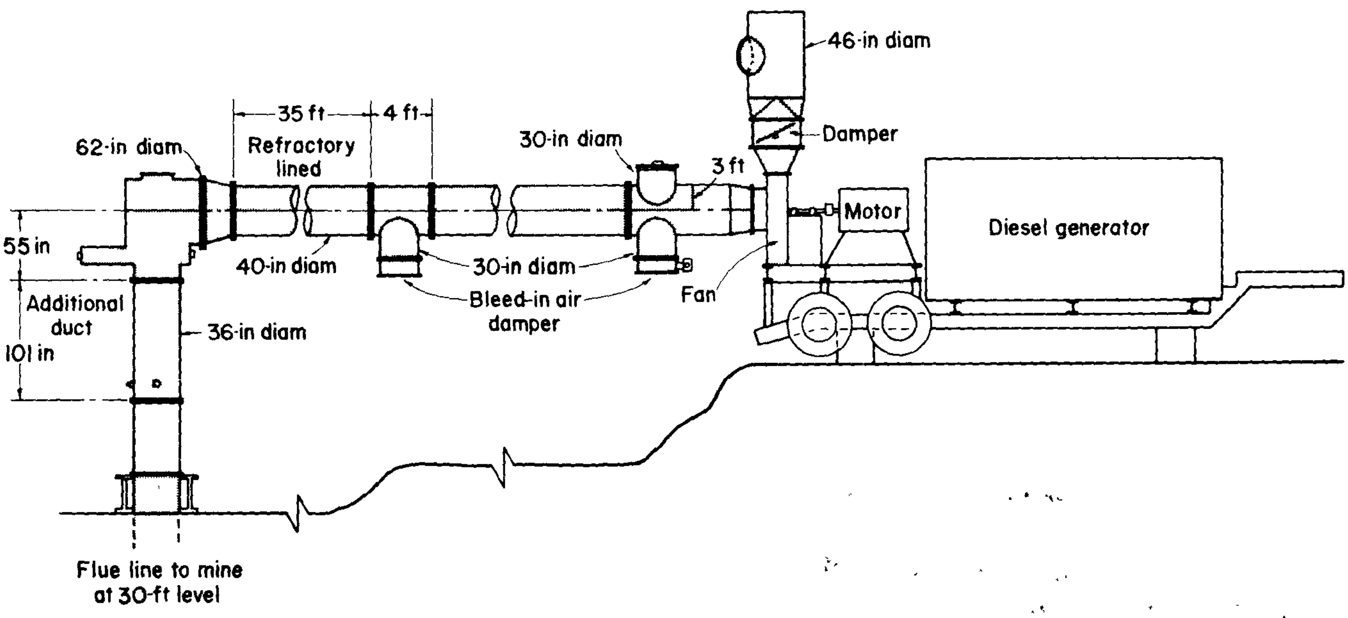

Figure 8.—Calamity Hollow burnout ventilation system.

involves complete burnout of the wasted coal in place under conditions that allow for total management of the hot gases produced. Ultimately, complete burnout would solve once and for all the fire and acid water formation problems of these fires on abandoned mine lands. In addition, utilization of the heat produced during burnout (e.g., to produce steam, electricity, or process heat) could more than offset the costs of constructing and operating a burnout exhaust ventilation system.

Data from large-scale simulated in situ combustion experiments strongly support the feasibility of the burnout concept; however such experiments do not adequately address potential problems from subsidence. A field trial of burnout control of an actual abandoned coal mine fire is currently being prepared to answer many if not all the questions raised in this report. This field trial (Calamity Hollow mine fire project) is expected to begin burnout operations in June 1981.

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