Chapter 8

SITE MINING AND FOUNDATION ISSUES

The construction of embankments and impoundments for coal refuse disposal and process water supply is an integral part of most coal mining operations. However, in many coal mining areas, it is difficult to locate a site for construction that has not been or will not be undermined to some extent, especially near the coal preparation plants that the impoundments serve. When mining is planned or is occurring in the vicinity of an impoundment, measures must be taken to evaluate the potential impact of mining-induced movement on both the safety of the embankment and the integrity of the impoundment. The potential for water or slurry from the impoundment to affect the safety of underground mine workings must also be considered. Even for non-impounding embankments, there may be performance concerns related to drainage structures that could be affected by surface movement induced by underground mining. Additionally, environmental considerations with respect to the effects of mining on the hydrogeologic regime may be important.

It can be difficult to identify and evaluate the potential subsidence mechanisms that may induce ground movement and affect the integrity of a refuse embankment or impoundment area because of uncertainties or inaccuracies related to the location of mining and the characterization of overburden and protective features such as barriers and mine seals. Accordingly, designers need to take steps to accurately assess these features and to provide appropriate protective measures.

An assessment of the potential for mining to affect the safety of an impoundment should be addressed in plans submitted to MSHA for approval as per 30 CFR § 77.216-2. Applicable MSHA permit requirements related to the potential for an impoundment to affect the safety of underground mine workings can be found in 30 CFR § 75.1716, which identifies requirements for obtaining permits when mining is planned under a body of water of sufficient size to present a hazard to miners. Because mining in the vicinity of an impoundment can present a hazard due to fracturing of the overburden beyond the horizontal limits of mining, MSHA may require a permit even in situations where the mine workings do not extend directly under the impoundment.

This chapter focuses on coal refuse impoundment design issues caused by the presence of underground mine workings, mine spoil materials, and mine highwalls. Based on past experience, these design issues include:

- Mine subsidence where the load imposed by overburden and/or an impoundment causes ground movement from the caving of overburden strata and/or crushing or bearing capacity failure of pillars. This response can impact the hydrogeology (i.e., seepage rates and paths), cause differential movements in the dam or impoundment reservoir area, result in loss of impoundment freeboard, and disrupt internal drainage systems. Mine subsidence can also aggravate pre-existing conditions and create a weakened or worse state that increases susceptibility to other failure mechanisms.

- Mine breakthrough where the load imposed by the overburden and impoundment or deterioration due to weathering, degradation or seepage forces causes collapse of overburden strata and uncontrolled release of fluid fine coal refuse and water into mine workings and beyond. Breakthroughs have also occurred where an impoundment intersects a coal seam or overburden barrier left in place to protect active or abandoned mine workings and the hydraulic load imposed by the impoundment exceeds the resistance provided by the barrier.
- Mine blowouts where water impounded in an underground mine breaks through a sealed mine opening and/or a section of the coal outcrop barrier. A blowout can be a secondary consequence of impoundment breakthrough into an underground mine or a result of natural accumulation of water or of underground slurry disposal.
- Seepage, internal erosion and stability concerns when an embankment is to be constructed over an area where there may be mine openings related to highwall mining and auger holes.
- Seepage and internal erosion concerns where an embankment is to be constructed on top of spoil and the spoil was placed under uncontrolled conditions.
- Seepage and internal erosion concerns where an embankment abuts a steep rock slope or highwall.

For non-impounding embankments, mine subsidence can cause disturbance to internal drainage systems, surface water control structures, slopes and liner systems. Additionally, surface cracking can lead to increased infiltration of water. Thus, potential underground mining impacts can be important design considerations for non-impounding structures.

Descriptions related to the circumstances behind and lessons learned from 11 mine breakthrough and four mine blowout events are presented in Appendix D of an MSHA (2003) report to congress titled, Guidance for Evaluating the Potential for Breakthroughs from Impoundments into Underground Mine Workings and Breakthrough Prevention Measures. This chapter draws extensively from the 2003 MSHA report for guidance related to underground mine issues.

8.1 GENERAL CONSIDERATIONS

In addition to other geotechnical considerations associated with coal refuse disposal facilities, there are siting factors and structural analysis and design issues that need to be considered when a facility is sited above or adjacent to abandoned, active or planned future underground mine workings. These factors and issues include:

- Availability of information resources including mine maps, coal contour and outcrop maps and the interpretation and accuracy of such maps (i.e., certified as accurate or uncertified, includes the latest mining activity, etc.).
- Availability of on-site reconnaissance and surface exploration data related to the identification and mapping of geologic features, soil and rock overburden and natural outcrop barriers, and underground mine voids. These data may originate from surficial mapping, geophysical methods, long-hole directional drilling, and conventional drilling and sampling.

- Evaluation of the potential for mine subsidence and breakthrough and mine seal barrier failure considering possible failure mechanisms and methods of evaluation and analysis.
- Available methods and design measures for mitigating or controlling mine subsidence and breakthrough and mine seal barrier failure.
- Impoundment construction and operations monitoring guidelines.
- Surface mine spoil properties, if such material is present in foundation zones or used in constructing embankments.
- Surface mine bench/highwall issues.

8.2 AVAILABLE SOURCES OF SITE INFORMATION

MSHA (2003) developed guidelines for evaluation of sites with potential for breakthrough into underground mines, covering preliminary investigation, on-site reconnaissance, and direct/detailed investigation. Table 8.1 presents sources for information related to mine maps, conditions, and records. Further discussion is presented in the following paragraphs.

A general discussion regarding the location, content and accuracy of mine maps with respect to the planning of site reconnaissance and exploration programs and the geotechnical analysis and design of coal refuse disposal facilities is presented in Section 6.4.1.5 of this Manual. The discussion presented herein relates to the general usefulness of mine maps with respect to locating coal contours and outcrops and to mine map interpretation and accuracy based on information from MSHA (2003) and NRC (2002).

8.2.1 Mine Maps

Underground mine maps can be used to locate a mine with respect to the surface or other underground mines and to determine the dimensions of pillars and mine openings. Information provided on most underground mine maps includes:

- Pillared, worked out, and abandoned areas; pillar locations; extraction heights; sealed areas; future mining projections; adjacent mine workings within 1,000 feet; locations of surface or auger mining; mined areas of the coal bed; and the extent of pooled water at the time the map was prepared.
- Dates of mining, coal seam sections, coal seam elevations, and survey data and markers.
- Surface features, coal outcrop, and 100-foot overburden contour or other prescribed mining limit; mineral lease boundaries; surface property or mine boundary lines, and identification of coal ownership.
- Areas used for underground injection of coal refuse slurry, acid mine drainage sludge, or other sediments or wastewater.

Sources for mine maps include local mining and mineral rights holding companies, local engineering and surveying firms, as well as the OSM National Mine Map Repository and state regulatory authorities. Other sources could include museums, mining schools, and mineral resource organizations. Mine maps should be assessed to determine their reliability (e.g., determining if mine production continued after the date of the map). Table 8.1 cites potential information sources or methods for corroborating mine maps, and Table 8.2 lists federal and state agencies that may have data bases and can provide assistance in locating mine maps. Exploration and reconnaissance methods for evaluating underground workings when there is no mapping or when available mapping is suspect are presented in Section 8.3.

TABLE 8.1 SOURCES OF INFORMATION FOR EVALUATION OF BREAKTHROUGH POTENTIAL

1. Search available mine maps for the area. The local mining companies should be contacted. Other potential sources of mine maps include OSM's National Mine Map Repository, museums, mining schools, and state agencies that deal with mine safety and reclamation.
2. Where there are multiple coal seams, the mine maps for each seam should be obtained.
3. Available aerial photographs of the area should be checked for indications of mine openings, auger openings, other types of surface disturbance, or the presence of mining facilities.
4. Topographic maps should be checked for indications of mine openings or mine facilities.
5. Geologic maps and cross-sections should be checked for the presence of coal seams in the area.
6. Available information on the condition of, and surface disturbance to, the outcrop barrier and overburden should be collected and reviewed. This would include geologic maps, aerial photographs, publications available from government agencies, and previous engineering reports, plans, or permits pertaining to the area.
7. Review boring logs from drill holes in the area, potentially for coal or gas exploration as well as geotechnical purposes.
8. Collect and review available information on the mining conditions and practices for workings under or near the site. Include information on: roof falls; roof support measures, especially supplemental support used near the outcrop or under shallow cover; pillar stability, particularly information on second mining of pillars; and floor conditions, especially information on punching or heaving.
9. Interview miners, especially older or retired miners who worked in the area, concerning the mining conditions, including roof, pillar, floor, and water conditions; the practices used when mining near the outcrop; and whether the available mine maps are consistent with their knowledge of the mining that took place.
10. Interview mine surveyors who may have worked in the area about their knowledge of mining in the surrounding area.
11. Search available information on auger mining. Often the depth of auger holes are not shown on the mine map, or only approximate depths are indicated.
12. Compare production records with the data on mine maps to help validate that additional mining did not occur after the mine map was produced.

(MSHA, 2003)

8.2.2 Coal Contour Maps and Outcrops

A coal outcrop is generally regarded as the location (horizontal or at a slope equivalent to the dip of the seam) where the bottom of the coal seam intersects the surface, and it is commonly represented as the coal outcrop line on contour maps. Because the actual coal seam is generally not visible at the surface, the outcrop line is usually shown as a contour corresponding to the bottom elevation of the coal seam in that area. Subcrop locations, which are interfaces with other geologic media, are sometimes shown as lines on mine maps where the coal crops out below glacial, alluvial or other sediment deposits or volcanic material.

At some sites, coal outcrops are visible because of surface mining activities, abandoned portals, outcrop sample areas, auger mining, highway cuts, house and building excavations, and house-coal openings. However, locating coal outcrops in steeply sloped areas may be impractical because of difficult access to the outcrop areas and the need for surface excavation. Field location of outcrops can also be complicated by the presence of unmined coal horizons; multiple coal seams in the same vicinity; and spoil, refuse, backfill, or other fill material placement.

The steepness of the ground surface, stress relief fracturing, weathering, hillside creep, landslides, and man-made disturbance can all affect the coal at a projected outcrop location. These effects, cou-

TABLE 8.2 SOURCES FOR MINE MAPS

OSM National Mine Map Repository	Indiana Geological Survey
Virginia Department of Mines, Minerals, and Energy	Ohio Division of Geological Survey
West Virginia Geological Survey	Ohio Division of Natural Resources
Kentucky Map Information Center	Maryland Coal Mine Mapping Project
Pennsylvania Department of Environmental Protection	Utah Geological Survey
Illinois Department of Natural Resources	

pled with the possibility that the projected coal outcrop location may be inaccurate because of local changes in coal structure, may complicate coal barrier analyses.

The surface topography maps submitted to MSHA, OSM, and state regulatory authorities delineating coal outcrop locations are usually scaled at 500 feet or less to the inch. The maps are typically based on aerial photography or USGS mapping. Coal outcrop locations are usually projected based on the structure of the coal and its inferred (and sometimes measured) intersection with the ground surface. It is recommended that the coal outcrop location be measured and the distance from the ground surface to intact coal determined at impoundment sites where breakthrough potential is a concern.

Most mid- and large-size coal companies currently employ precise surveying techniques and mine planning software to generate mine maps. This has been prompted by the need to accurately locate ventilation shafts, to prevent mine pool blowouts, and to have accurate locations should a mine rescue be necessary. These software codes utilize data from borehole geotechnical and geologic logs, horizontal drilling, surface geophysical exploration, outcrop exploration, land and in-mine surveys and topographic mapping to generate an outcrop line on maps. The results are generally more accurate than previous methods; some mining companies make these maps available.

One of the key steps in delineating coal outcrop locations on surface topographic maps is the coordination of survey control points. Common control points should be used for both the underground mine map and the surface topography map. The accuracy of each map generated from these common control points is then a function of the equipment or method used to create the elevations and contours. Where practical, and at critical locations, surface topographic surveys should be conducted along the coal outcrop as part of permit submittals. These surveys should fully document existing locations where the coal seam is already exposed due to natural or man-made activities. The surveys should be tied to either the U.S. Geological Survey or the U.S. Coast and Geodetic Survey benchmark system used for the underground mine surveys. By using common control points for the surface and underground surveys, accurate vertical and horizontal control can be achieved.

8.2.3 Interpretation and Accuracy of Mine Maps

Mine mapping is a key element in evaluating breakthrough potential. As described in Section 6.4.1.5, the accuracy and completeness of the information on mine maps can vary widely. In times past some mines were not mapped at all, and the accuracy of older mine maps was questionable. Often there are significant differences in the accuracy and completeness of underground mine maps (especially those prepared before 1969) because of the varying requirements associated with their development. Sometimes the horizontal and vertical (overburden) distances between mined barriers and an impoundment are not accurately shown.

Accidents have occurred where an active mine has broken into old/abandoned mine workings that either were not shown on a map or were shown as being hundreds of feet from their actual location. These incidents may have occurred because the complete workings were not surveyed, the area was inaccurately surveyed, the data were not properly recorded, or coal was "robbed" by others after the mine closed. It is important to determine if the mine map being relied upon is current and not an earlier (interim) map. This can be accomplished by cross-referencing dates on the mine map with available production records. Problems can also occur when maps for adjacent mines are not referenced to a common coordinate system or when data from a map are inaccurately transposed from one coordinate system to another. Many potential problems can be avoided by referencing all mine maps to the state plane coordinate system. Unfortunately, some maps and records for older mines have been lost or destroyed.

When mining occurs near an outcrop, subsurface conditions may exacerbate problems related to inaccurate mine maps. It is not uncommon for roof conditions to deteriorate as mining approaches an outcrop. This occurs because the mining encounters less cover, more weathered roof strata, more frequent jointing, and possibly highly weathered joints near an outcrop (sometimes referred to as hillseams, as discussed in Section 6.4.1.2). Furthermore, the last cut made toward an outcrop typically is not provided with roof support. Because surveyors will not have direct access to the unsupported section of the entry, distances may be estimated rather than directly surveyed. Also, if a roof fall occurs or an area is restricted from access after mining, it may not be accurately surveyed. Dashed lines on a mine map indicate that the area was not surveyed and that locations are determined based on the best available information, which may be estimates from the section foreman's map.

Areas of secondary recovery mining or partial mining of pillars and barriers are typically not accurately located, as these areas cannot be surveyed. The extent of secondary mining indicated on mine maps is generally based on estimates from foreman reports. Reconnaissance and exploration methods for evaluation of such conditions are discussed in Section 8.3.

8.3 ON-SITE RECONNAISSANCE AND EXPLORATION

The accuracy of available mapping for coal contours and outcrops can vary depending on the age, location and source of the mapping. Therefore, when earthen dams and refuse impoundments are to be constructed over or in proximity to active and abandoned underground mine workings, the accuracy of this mapping information must be verified as part of site reconnaissance studies. The following sections provide a description of surficial reconnaissance and geophysical and geotechnical exploration methods that can be used to confirm the location and configuration of underground mine workings.

8.3.1 Surficial Reconnaissance

Surficial reconnaissance consists of walking the disposal facility site and vicinity (both dam and impoundment areas to be developed) and observing topography, rock and coal outcrops, surface cracks and subsidence features, soil types, vegetative cover, spring discharges, perennial and intermittent watercourse locations, and any other information that may contribute to more complete knowledge of the site. This type of surficial reconnaissance and geologic mapping generally requires an experienced geologist or engineer with knowledge of refuse disposal and embankment design. If possible, the reconnaissance should be conducted during times when vegetation is dormant so that site features are more visible. Additional details regarding the types of features that should be documented during a surficial reconnaissance are discussed in Section 6.4.2.

Test pits can greatly enhance surficial reconnaissance, as areas can be excavated to: (1) determine the relationship of seeps to the coal seam(s), (2) verify coal seam outcrop(s), and (3) establish the depth of weathering, presence of fill, etc. Test pits can be cost effective particularly when active mining operations are nearby and on-site equipment can be employed.

8.3.2 Geophysical Methods

Where there is uncertainty regarding the presence and extent of mining or the accuracy of available mine maps, some geophysical exploration methods can be employed as a logical first step for site evaluation and reconnaissance. Geophysical methods, through detection of anomalies, can indicate whether mine workings are present and if detected mine workings generally correspond to available mine maps. Geophysical methods that have been used for locating mine workings include seismic, electromagnetic and electrical resistivity techniques. A detailed description of these and other geophysical methods is provided in Section 6.4.4. The advantage of geophysical methods is that a large area can be quickly explored, as compared to drilling programs where information is obtained one borehole at a time. Geophysical methods entail the use of sophisticated equipment and data processing techniques. Therefore, these studies should be planned, implemented, and the data interpreted only by persons experienced in this specialty field. Results of geophysical site exploration should be confirmed by subsurface drilling or other independent technique.

Geophysical methods have limitations related to void and mine depths, overburden type and condition, and whether voids contain water. No single method has proven successful in a majority of cases, even where depth and mine void filling is ideal for the method being employed. Thus, geophysical methods should be viewed as a tool to identify potential anomalies that can be further defined by other methods such as drilling.

8.3.3 Long-Hole Directional Drilling

Directional or long-hole drilling generally refers to in-mine drilling operations used for identifying and understanding geological and mining conditions in advance of mining. Directional holes have been drilled for horizontal distances of more than 5000 feet (Kravits and Schwoebel, 1994). Long-hole drilling can also be performed from the surface, independent of mining operations, and thus can be employed near abandoned mines. Advances in directional, long-hole drilling include a technique that can be used for locating or verifying the absence of mine workings. The position of the end of the hole is determined using a surveying tool that is an integral part of the drilling system. The borehole can be guided or steered from underground mine workings or from the surface to determine whether there are mine workings in a particular area. This technique can also be used to advance a hole within the coal seam and roughly parallel to a coal-outcrop line to establish the thickness of solid outcrop barrier around an impoundment site. Reportedly, long-hole drilling can achieve accuracies of better than $\pm 1°$ of azimuth and $\pm 0.25°$ of inclination. Long-hole drilling is also discussed in Section 6.4.3.9. Boreholes associated with such drilling should be properly sealed, as they can become seepage paths or increase breakthrough potential.

8.3.4 Conventional Drilling and Sampling

Drilling boreholes is probably the most effective method for determining the location, extent and variability of subsurface conditions at specific points. The difficulty with drilling boreholes is determining how many are needed to adequately characterize a site and where the boreholes should be located. Section 6.4.3 describes subsurface exploration and in-situ test planning for earthen dams and refuse impoundments. Geotechnical exploration for an impoundment site is much more complicated when there are underground mine workings at or in the vicinity of the site. For this situation, a substantial number of borings is often needed for characterizing conditions near the embankment and impoundment, and sometimes angle or horizontal drilling may be useful. Not only does the accuracy of available mine maps need to be determined, but the nature and variability of the overburden also need to be defined. An advantage of using geophysical methods as a preliminary step in site investigation is that the results of the geophysical work may allow the designer to optimize the number and location of boreholes and minimize associated costs. In general, the scope of the drilling needs to be such that the accuracy of the mapping information can be verified and the geologic, soil, and water conditions can be determined sufficiently to enable evaluation of: (1) the adequacy of barriers and overburden with respect to breakthrough and (2) assessment of the potential for subsidence to affect the dam. Table 8.3 presents exploration considerations for assessing breakthrough potential at impoundment sites (MSHA, 2003) and also provides a discussion of the use of test pits to verify coal seam outcrops. Table 8.4 presents exploration considerations for outcrop barriers developed by OSM (Kohli and Block, 2007).

Exploration borings that are advanced from an impoundment site into underground mine workings could be a potential breakthrough path if not properly plugged. Guidance related to sealing of boreholes is presented in Section 6.4.3.13.

TABLE 8.3 EXPLORATION CONSIDERATIONS FOR ASSESSING IMPOUNDMENT BREAKTHROUGH POTENTIAL TO UNDERGROUND MINES

1. Identify locations under the reservoir, and around the perimeter of the reservoir, where the conditions appear to be most critical. Verification should be based on the locations where mapping or geophysics indicates that the total cover or the outcrop barrier widths are minimums, and other locations where the competent rock portion of the overburden is reduced.
2. These critical locations should be explored by drilling or test pits. The number of locations explored will depend on: (a) how far the mining is or is expected to be from the impoundment, (b) the level of uncertainty, (c) the results that are found, and (d) the degree of conservatism associated with the design or remedial measures being considered.
   - In an outcrop barrier, the main concerns are to: (1) verify how close to the surface mining has occurred, (2) determine whether the pillars have been first-mined only, and (3) establish whether auger or highwall mining has reduced the size of the barrier. Exploration should be conducted at several of the more critical areas to verify the extent of mining and determine the cover conditions, in particular, the thickness and condition of the rock strata. If the mapping is found to be inaccurate in any of those areas, then additional areas will need to be checked.
   - Where mining has occurred below the level of the bottom of the reservoir (below drainage), the main concern will be to confirm the depth to mining, the mining method, the extent of mining, the characteristics of the overburden, and the size and condition of the pillars.
   - During exploration, evidence should be collected which helps to corroborate available information on the type of mining and characteristics of gob materials, age of mining, presence of water and pools, and past injection of slurry or other materials in the mine workings.
3. Where practical, selected borings that are drilled for other purposes, such as foundation investigation, should be extended to the coal seam level(s) to provide additional points to confirm whether or not mining has occurred. Where longwall or retreat mining has occurred, borings can provide information on mine convergence and gob materials (backfill classification and penetration resistance.)
4. For each area investigated, sufficient information on the geologic conditions (e.g., rock quality, joints, weathering), and the engineering properties of the materials, should be obtained.
5. A sufficient number of vertical drill holes should be drilled at critical locations to bracket the farthest extent of the mining. Where practical, horizontal and angle holes may be helpful in locating and mapping conditions.
6. In determining where holes should be drilled, the lateral extent should consider possible zones of subsidence. The lateral extent of the area impacted by subsidence is larger than the mined area and is generally defined by the angle of draw from mine workings that are in proximity to the impoundment.
7. If the designer determines that multiple seam mining has occurred in the area, information should be collected on the mining in the other seams including mine map overlays and the nature of the interburden between the seams.
8. The extent of augering and highwall mining is usually not accurately mapped and should be evaluated to assess the extent of penetration of the coal barrier.
9. Borehole cameras and imaging systems such as borehole sonar (flooded mine) or laser scanners (dry mine) can be used to obtain additional information about the conditions at mine level, such as mine void dimensions and orientation and rib characteristics, as well as the amount of subsidence or collapse that has occurred, and the conditions at discontinuities intersected by the borehole.

(ADAPTED FROM MSHA, 2003)

TABLE 8.4 OUTCROP BARRIER EXPLORATION CONSIDERATIONS

1. Determine the minimum overburden thickness above the mine workings closest to the mine side of the barrier. Both the rock thickness and the thickness of unconsolidated material should be considered.
2. Inspect the surface area close to the proposed outcrop barrier for natural benches and other surface features (e.g., road cuts) that could reduce the overburden thickness.
3. Verify the correct location of the outcrop on the mine maps using land or GPS surveys. The outcrop may have been plotted by the mine survey or from the topographic maps.
4. Look for evidence of any water seepage above the barrier and from adjacent mine workings. Determine the quantity of discharge and its locations.
5. Determine if there are adits or auger holes in the mine barrier that are not shown on the map.
6. Identify other features that may impact the integrity of the barrier.
7. Look for evidence of subsidence or sinkhole cracks or other zones of weakness in the overburden above the mine workings adjacent to the barrier.

(KOHLI AND BLOCK, 2007)

8.4 EVALUATION OF MINE SUBSIDENCE AND BREAKTHROUGH

Mine subsidence can impact coal refuse disposal facilities directly by decreasing the stability of an embankment or impoundment leading to movement or release of refuse materials and water, or indirectly by affecting the grade or structural integrity of a drainage component. Ways in which mine subsidence can have an adverse effect on a refuse embankment include causing cracking, providing paths for seepage, disrupting internal drains, damaging decant pipes, and reducing freeboard. Sinkholes beneath a dam can cause internal erosion that can lead to dam failure. When mine subsidence leads to fracturing or sinkhole development extending to a surface impoundment, the possibility of a breakthrough into underground mine workings is an important consideration. A breakthrough is a sudden, uncontrolled release of water and/or fine coal refuse from an impoundment into an underground coal mine. Even when a slurry impoundment is no longer in use, there may still be potential for a breakthrough if the buried fine refuse remains in a loose, saturated condition.

The ability of overburden strata, outcrop barriers, and mine bulkheads to prevent breakthrough depends on many factors. These factors include:

- Thickness of strata above the mine workings
- Engineering properties of strata above the mine workings
- Width and integrity of the outcrop barrier
- Thickness and integrity of the material above and below the coal barrier
- Hydraulic conductivity of the coal and surrounding strata
- Piping potential of any natural soil, mine spoil, or refuse surrounding the outcrop barrier
- Size, depth and location of mine voids
- Hydrostatic and earth pressures
- Water and fine coal refuse levels
- Flowability of the fine coal refuse
- Presence and orientation of stress relief fractures, rock discontinuities, and weathered joints

- Impacts of any physical disturbance (e.g., landslides, road cuts, auger or highwall mining) on the material in the outcrop area
- Presence of mine pools or slurry within the workings and the effect of slurry injection on the mine floor and barrier

Differential movements associated with mine subsidence, even if they do not represent an impoundment breakthrough concern, need to be evaluated to assess potential effects on surface and subsurface drainage structures. Subsidence and potential breakthrough failure mechanisms should be evaluated based on site-specific data. Collection and evaluation of information and field data on the mine, overburden and disposal embankment may be necessary for evaluating the potential for subsidence and breakthrough and for assessing possible mitigation measures. The need for collecting these data is dependent upon site conditions; however, the data review will minimize assumptions associated with the identification of failure mechanisms.

Table 8.5 presents a summary of guidelines and associated references for evaluating potential impacts from subsidence. Additionally, MSHA (2003) presents guidance for evaluating the potential of breakthrough from impoundments to mine workings and breakthrough prevention measures. The following sections present and discuss information from this guidance document.

8.4.1 Mine Subsidence Considerations

The possible impacts of mine subsidence on an impoundment and mine workings must be evaluated whenever a refuse disposal facility or other impoundment is to be located in the vicinity of existing underground workings or planned underground mining. Subsidence generally entails both vertical and horizontal movement, strain, tilt and curvature and may manifest on the surface as: (1) cracks, fissures or fractures; (2) pits or sinkholes; and (3) troughs or sags. Surface fractures may occur where there are areas of tension or shear stresses in the ground. When the area of surface subsidence is relatively small and the workings are close to the surface (normally 100 feet or less but as deep as 150 feet), the subsidence feature may be of pit or sinkhole size. Larger areas of surface subsidence are referred to as troughs or sags and are typical of deeper and secondary mining operations. Figure 8.1 illustrates the disturbance of geologic strata over mine workings, including some of the following effects (Kendorski; 1993, 2006):

- <u>Floor heave</u> – Upward thrust of the floor in the mine working area.
- <u>Caved zone</u> – Caving of the overburden directly over a mine void and bulking of the caved material leading to support of overlying strata generally extending to a height of 3 to 10 times the extraction thickness.
- <u>Fractured zone</u> – A zone of vertical fracturing and bed separations. Overburden in this zone moves vertically in large blocks along existing joints and new vertical fractures. Typically this zone extends no more than 24 times the extraction thickness above the mine, but can reach 30 times the extraction thickness.
- <u>Main roof</u> – This zone, which is sometimes subdivided into the Dilated Zone and the Constrained Zone, is an area of no significant increase in vertical hydraulic conductivity. This zone has been characterized as extending above the Fractured Zone up to 60 times the extraction thickness.
- <u>Surface zone</u> – Surface cracks are typically present in this zone and are generally limited to areas placed in tension by subsidence. Cracks can be created in dry clayey soil and joints can open in massive sandstones. Such cracks can extend downward to a depth of 50 feet.

Figure 8.2 illustrates the influence of extraction width on subsidence at the surface. For the maximum subsidence to be observed at the surface, the coal extraction width must typically reach the critical

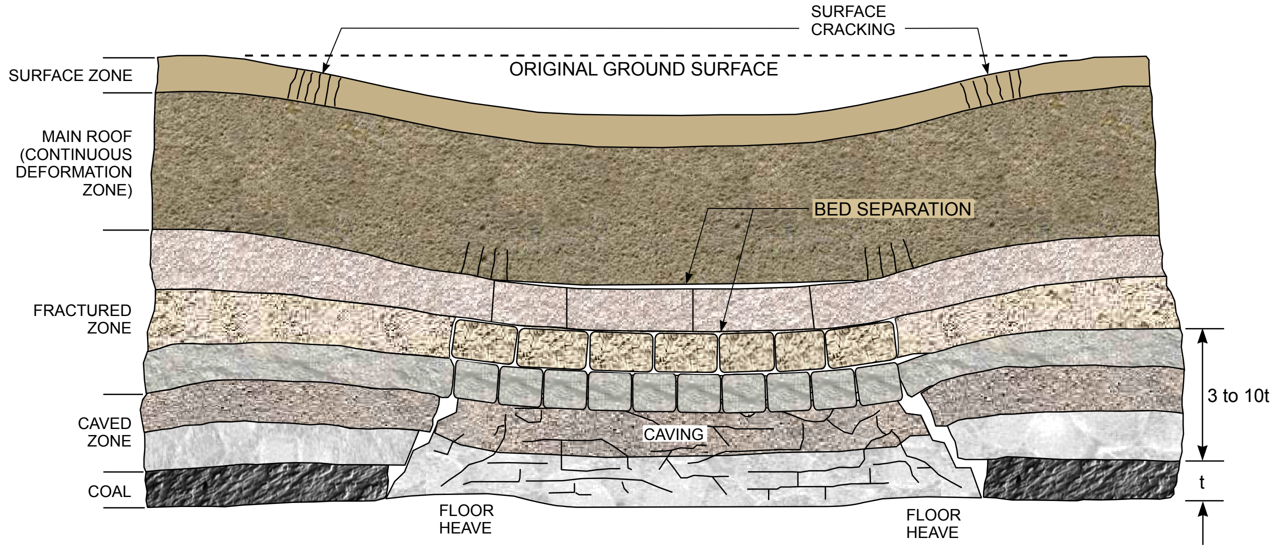
TABLE 8.5 SUMMARY OF GUIDELINES FOR MINING UNDER OR NEAR BODIES OF WATER

Reference	Description	Criterion
Room and Pillar (first mining only)		
Babcock and Hooker, 1977	Minimum Solid Overburden	$10t$ or $5s$, whichever is greater
	Minimum Solid Overburden where competent bed of sandstone or other rock is present.	$< 10t$ or $5s$, provided competent bed is $> 1.75s$
Panel and Pillar		
Babcock and Hooker, 1977	Minimum Solid Overburden	270 feet or $3p$, whichever is greater
	Maximum Width of Extraction Panel (p). For multiple seams, superimpose panels and pillars with panel widths being determined from the depth of the uppermost seam and pillar width being determined by reference to the thickest and/or deepest seam, whichever gives the greater dimension. Where panel and pillar system are employed in upper seam, apply above criterion considering upper seam or body of water.	$p \leq 1/3\ H$
Total Extraction		
Skelly and Loy, 1977	Minimum Overburden (H)	$100t$ or ≥ 700 ft, whichever is greater
	Surface Tensile Strain (ε_t)	< 0.010
Babcock and Hooker, 1977	Minimum Solid Overburden	$60t$
Kendorski et al., 1979	Minimum Overburden • Catastrophic-Size Water Body • Major-Size Water body with Limited Potential	 varies from $60t$ to $117t$ varies from $37t$ to $105t$
	Surface Tensile Strain • Catastrophic-Size Water Body • Major-Size Water Body with Limited Potential	 ≤ 0.010 ≤ 0.015

Note: These guidelines generally apply to the prevention of significant impacts to overburden and inflow into mines. Lower strains than those indicated can cause cracking of embankment dams.

H = thickness of rock overburden (ft)
t = average extraction thickness (ft)
s = entry width (ft)
ε_t = surface tensile strain (dim)

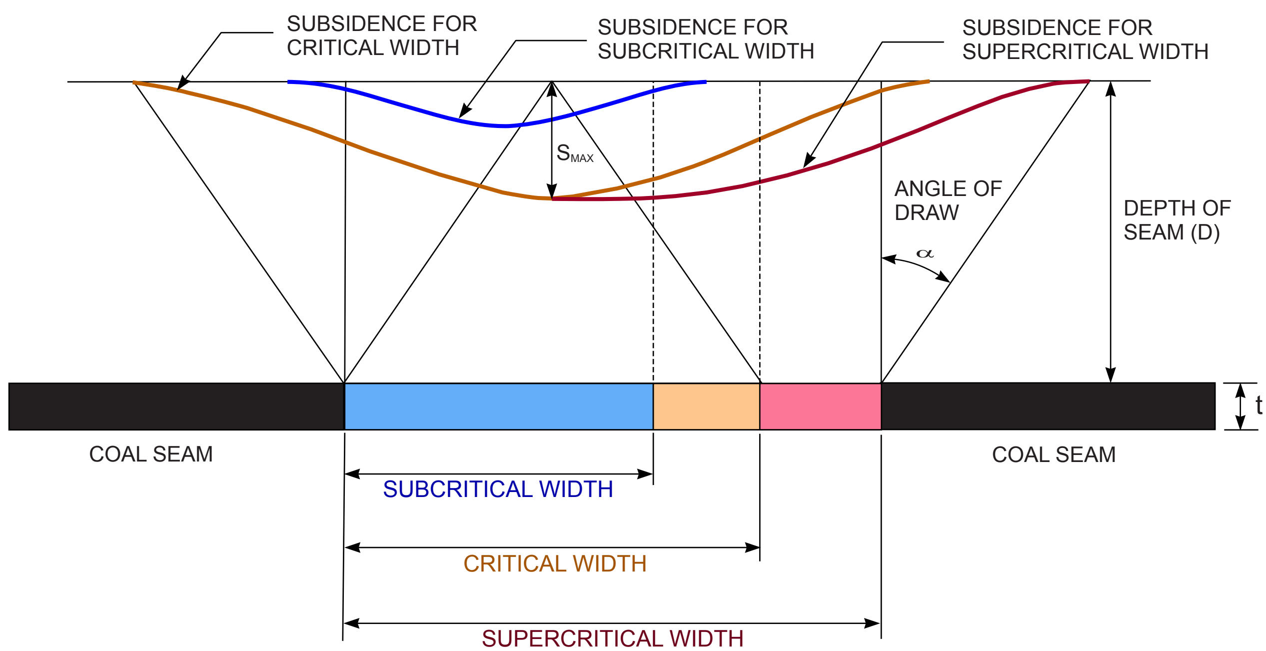
width shown. If the mined width is less than critical, it is termed subcritical, and the amount of subsidence is less than the maximum subsidence experienced with wider extraction areas. If the mined width is greater than the critical width, supercritical conditions prevail, and the subsidence trough will exhibit a flat bottom depression, as shown in Figure 8.2. The figure also shows the angle of inclination between the vertical at the edge of the workings and the point of zero vertical displacement at the edge of the trough, which is termed the limit angle or angle of draw. Depending on the prediction method used, survey standards employed, or criticality of a surface structure, determination of the angle of draw is typically based on a minimal vertical movement of between 0.01 and 0.1 feet (for predictive measures that assume an asymptotic approach to zero such as the tangent-hyperbolic function). The "coal extraction width" can be a result of longwall mining, second mining of pillars, or pillars crushing or punching into the floor.



(ADAPTED FROM SINGH AND KENDORSKI, 1981;
PENG AND CHIANG, 1984)

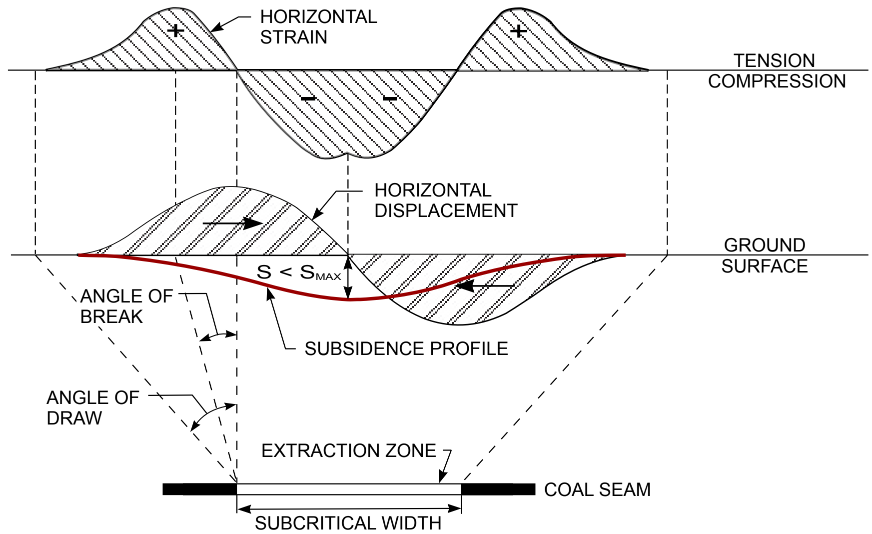
FIGURE 8.1 STRATA DISTURBANCE AND SUBSIDENCE CAUSED BY MINING

Subsidence lowers surface grades and elevations and can affect surface drainage and alter impoundment freeboard. Strains and horizontal displacements associated with subsidence can impact the structural integrity of surface structures such as dams (including internal drains and decants), embankments, and bridges and buried structures such as pipelines and large culverts. Zones of surface tension and compression develop during mining, resulting in horizontal movement profiles that are primarily a function of the extraction width, as depicted in Figure 8.3. The most severe subsidence impacts on many impoundment structures occur where the tensile stain is highest, while for other structures it is the area of greatest tilt or subsidence-induced slope. The existing condition of a structure and slope of the terrain on which the structure is situated also affect subsidence impacts.

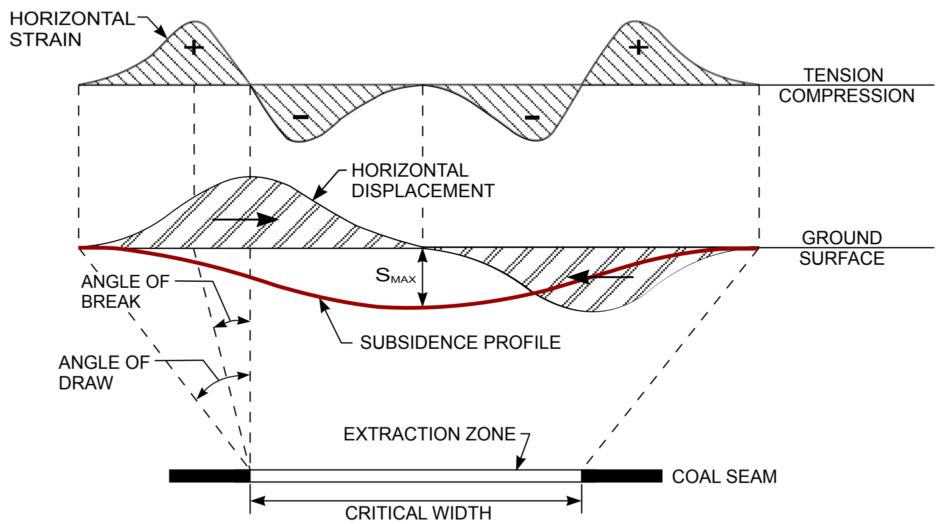


(SINGH, 1992)

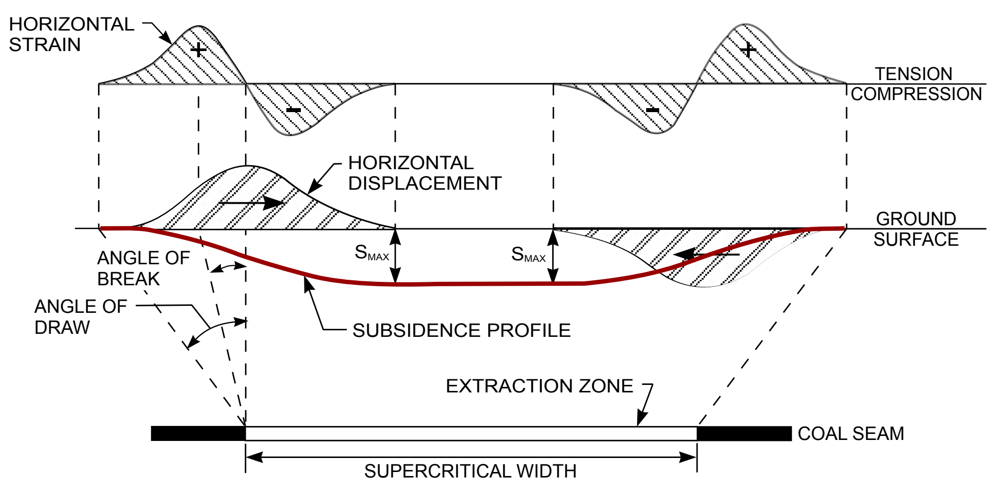
FIGURE 8.2 INFLUENCE OF EXTRACTION WIDTH ON SUBSIDENCE



8.3a SUBCRITICAL WIDTH



8.3b CRITICAL WIDTH



8.3c SUPERCRITICAL WIDTH

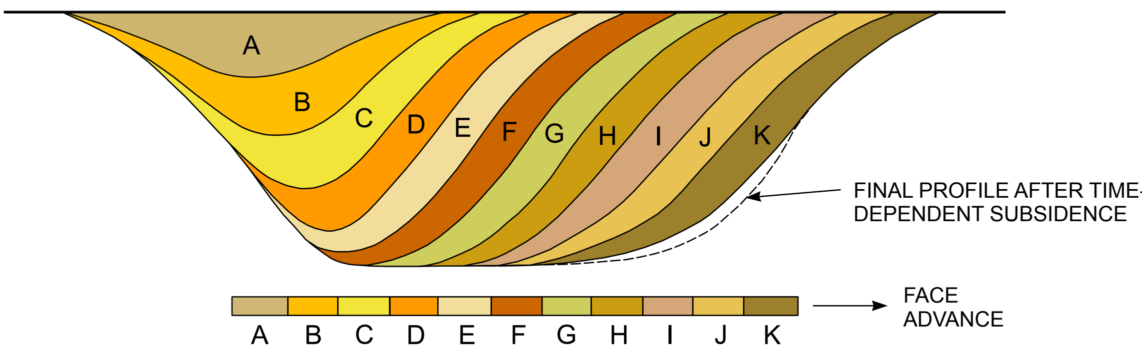
(SINGH, 1992)

FIGURE 8.3 DISPLACEMENT AND STRAIN FOR VARIOUS EXTRACTION WIDTHS

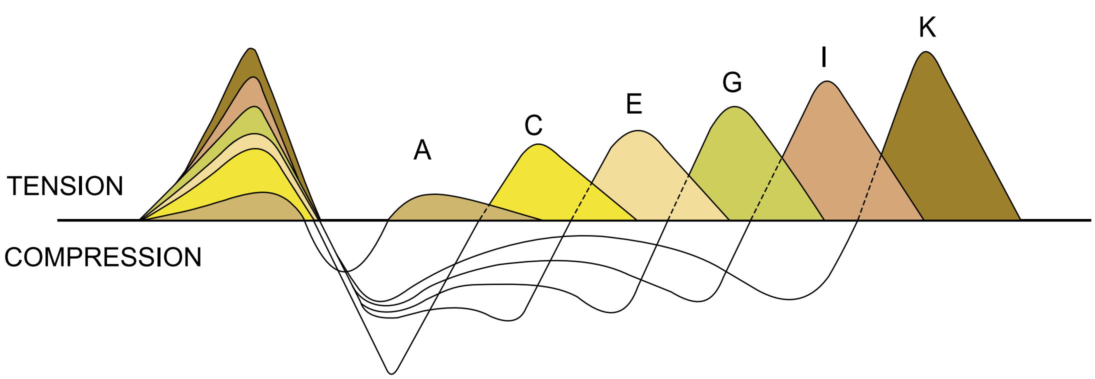
For total extraction mining, subsidence follows the advance of the mine workings, and horizontal tensile and compressive strain regions move laterally with the mining, as shown in Figure 8.4. Therefore, determination of the impacts of full extraction mining beneath embankments and impoundments must be based upon subsidence development (i.e., dynamic subsidence) and the associated traveling strain curve and not just the final subsidence profile. Areas of greatest impact generally occur near the boundaries of the extraction zone. Figure 8.5 depicts the ground movements caused by subsidence. Singh (1992) summarizes factors that affect mine subsidence, including: (1) seam thickness, depth, and dip; (2) mine floor, roof and overburden; (3) geologic discontinuities and in-situ stresses; (4) surface topography; (5) groundwater; and (6) percent of extraction, advance rate, backfilling, and elapsed time.

8.4.2 Potential Subsidence and Failure Mechanisms

There are several potential mechanisms that are associated with mine subsidence and mine breakthroughs. Subsidence can occur as a result of caving and fracturing of the mine overburden and migration of the disturbance to the surface (e.g., sinkhole development), pillar crushing, or pillar punching. Also, planned subsidence can occur as a result of: (1) total extraction mining using longwall methods or secondary mining of pillars or (2) room and pillar mining with partial extraction leaving pillar remnants designed to yield. The condition of the coal and overburden and the mine extraction method will determine the extent to which subsidence will propagate upward toward the surface and the degree of fracturing and deformation that can occur at the surface. Longwall subsid-



8.4a SUBSIDENCE DEVELOPMENT



8.4b TRAVELING STRAIN CURVE

(RELLENSMANN AND WAGNER, 1957)

FIGURE 8.4 DEVELOPMENT OF SUBSIDENCE TROUGH AND STRAINS WITH FACE ADVANCE

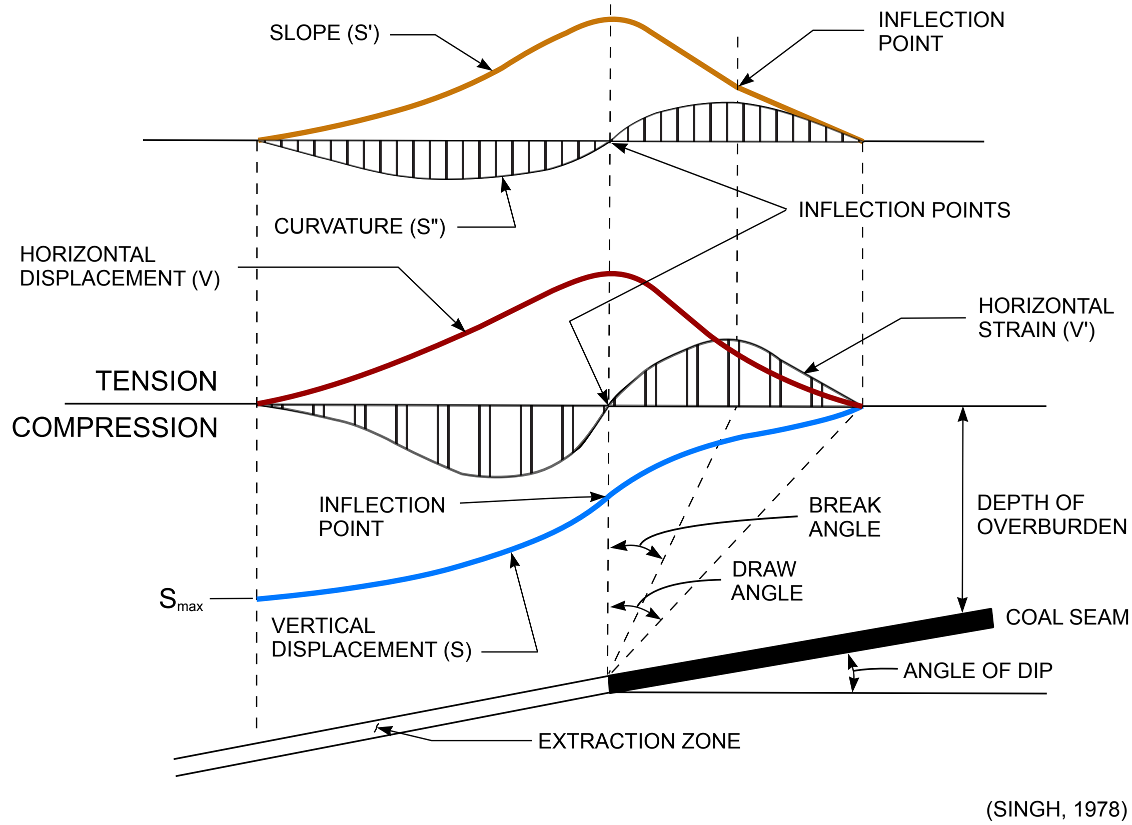


FIGURE 8.5 GROUND MOVEMENTS CAUSED BY SUBSIDENCE

ence occurs shortly after mining, but room and pillar subsidence may occur years or decades after mining, as conditions or in-mine support deteriorate.

Subsidence of underground mine workings can lead to breakthrough if an impoundment is located directly over or near the mine. If the caved or fracture zone intersects with the surface or surface soil layer, a breakthrough can occur. The driving force for a breakthrough is the pressure gradient (earth and water pressure) between the impoundment and the underground mine workings. The added weight from the embankment and reservoir, combined with increased seepage heads, increases the stress on the underlying strata. Mechanisms that can cause a breakthrough include internal erosion or piping, outcrop barrier failure, hillside movement and disturbance, mine seal failure, or barrier (coal, soil and/or rock) decomposition. These mechanisms are frequently interrelated and should be carefully evaluated as part of the analysis of the potential for breakthrough.

8.4.2.1 Sinkholes

A sinkhole is a depression or opening in the ground surface above an underground mine void where the mine roof has fractured and fallen/caved and the disturbance has intersected the surface or soil mantel. The fracturing of the mine roof material can eventually extend high enough that an opening, or at least a weakened area, is created at the ground surface, particularly where the overburden is thin. A sinkhole can serve as a direct conduit from an impoundment to underground mine workings. Factors contributing to sinkhole development include: (1) the presence of a mine void, (2) low overburden thickness, (3) mine roof material that is not sufficiently strong or durable to span the mine opening, (4) fractures in the mine roof, (5) unconsolidated soil and weathered rock above the mine roof, and (6) pressure exerted by refuse and/or water at the surface. The action of water flowing through fractured strata can cause deterioration and can enlarge a sinkhole beyond the depth and extent that would occur in the absence of water.

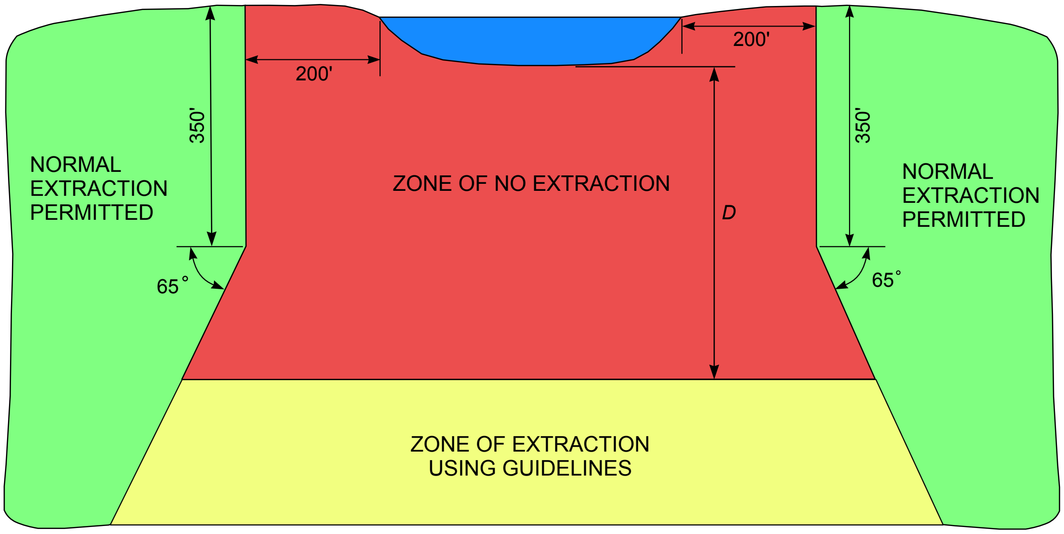
A sinkhole will not develop if the rock strata above the mine are either strong enough to span openings without collapsing, the strata above the mine are thick enough that an arch forms over openings and prevents the collapse feature from propagating to the surface, or the swell of falling rock checks the propagation of the collapse before it affects the mantle of unconsolidated material. One study in Pennsylvania (Gray et al., 1977) found that 81 percent of sinkholes occurred where there was less than 100 feet of cover, with most occurring at 50 to 60 feet of cover. However, there have been unusual cases, such as a sinkhole in Illinois (DuMontelle et al., 1981) where the cover was 165 feet, although the overburden thickness in this case included a substantial amount of glacial till. The mine depth associated with sinkholes reported from mines located primarily in Colorado, Utah and Wyoming (Dunrud and Osterwald, 1980) was generally 10 to 15 times the seam thickness.

Analysis of roof strata to determine if they will indefinitely span mine entries is difficult because: (1) the strength of the rock is not easily defined, (2) the effect of joints on the integrity of the roof (especially at shallow depths) adds considerable uncertainty, and (3) long-term integrity of coal pillars and roof support system can be an issue. For these reasons, the adequacy of the mine overburden to prevent sinkhole development is commonly assessed by applying certain "rule-of-thumb" type guidelines that have been developed based on experience. Guidelines developed by Babcock and Hooker (1977) are illustrated in Figure 8.6. These guidelines are generally considered to be conservative, and they should be carefully reviewed before detailed analyses to assess the potential for subsidence and related differential movement and strain are performed. Mine development work has been performed and production areas have been successfully implemented following the guidelines shown in Figure 8.6. However, where subsidence has the potential to affect a high-hazard-potential impounding embankment or to allow a breakthrough that could affect the safety of miners or the public, these guidelines should be used only in conjunction with more rigorous site-specific analysis.

With respect to sinkholes, the guidelines recommend for first mining only that the thickness of solid strata should be equal to at least 5 times the entry width or 10 times the extraction height, whichever is greater. These guidelines are consistent with findings that compression arches in overburden are normally stable if the mining width is limited to one-fourth to one-half of the overburden height. In other words, the compression arch is typically stable if the thickness of the rock strata above the mine is from 2 times (strong strata) to 4 times (weaker strata) the entry width. Adding a margin of safety, the "rule-of-thumb" is that the overburden thickness should be 5 times the entry width. Similarly, the criterion related to the extraction height is based on adding a margin of safety to the observation that the height of collapse above a mine entry generally does not exceed 3 to 5 times the extraction height, likely because of the swell of the collapsed material.

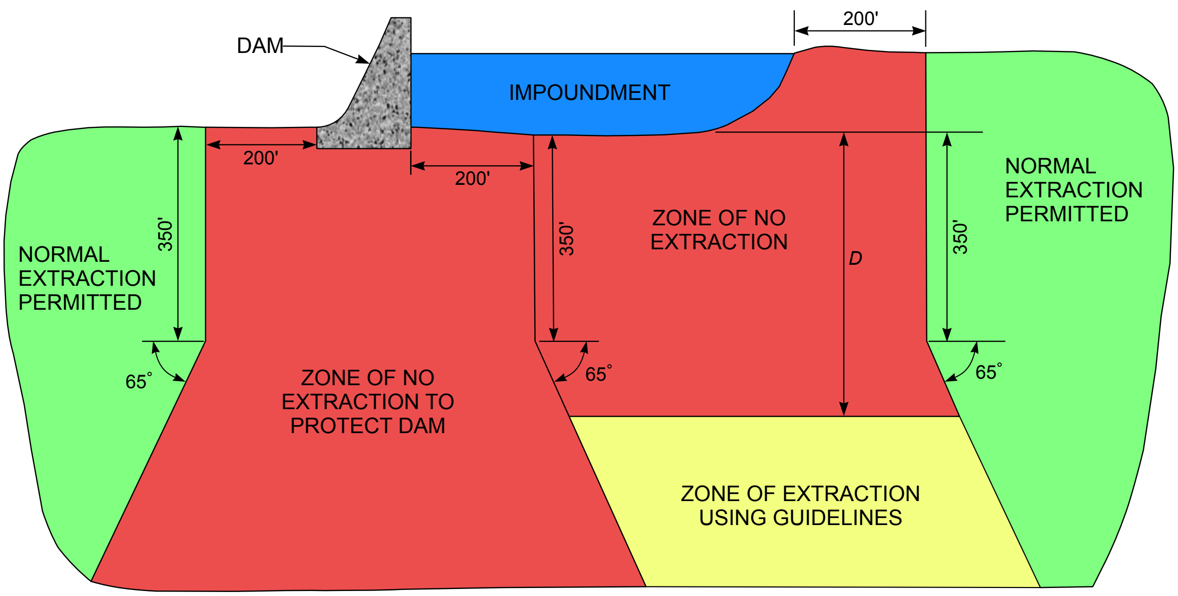
A key point in the use of these guidelines is that the strata thickness refers to "solid overburden strata." Soil, weathered rock, or weak rock should not be included in the solid strata thickness since they may not provide the strength needed to resist sinkhole development (MSHA, 2003).

The guidelines in Babcock and Hooker (1977) suggest that a lesser strata thickness may be acceptable if the overlying strata consist of competent rock (e.g., competent bed of sandstone or similar material) with a thickness of at least 1.75 times the entry width. Competent rock is normally taken to mean a homogeneous, massive layer of sandstone or limestone (MSHA, 2003). This guideline is based on analyses of the overlying strata as a beam with minimum tensile strength (20 psi). While any intact piece of rock will likely exceed this tensile strength, joints or discontinuities will reduce the effective strength of the beam. Because the potential impact of joints and weathering, especially near an outcrop, cannot be modeled with confidence, this approach should normally not be used for a critical subsidence or breakthrough situation. Furthermore, there is a possibility that the strength of the overlying strata will deteriorate over time due to seepage and weathering, especially as the mine pool level rises.



ROOM AND PILLAR	D = 5s or 10t, whichever is larger	s = maximum entry width (ft)
PANEL	D = 3p or 270 feet, whichever is larger	p = panel width (ft)
TOTAL EXTRACTION	D = 60t	t = extraction height (ft)

8.6a SAFETY ZONE BENEATH BODY OF SURFACE WATER



ROOM AND PILLAR	D = 5s or 10t, whichever is larger	s = maximum entry width (ft)
PANEL	D = 3p or 270 feet, whichever is larger	p = panel width (ft)
TOTAL EXTRACTION	D = 60t	t = extraction height (ft)

8.6b SAFETY ZONE BENEATH DAM AND IMPOUNDED BODY OF SURFACE WATER

NOTE: THE ZONE OF NO EXTRACTION TO DEPTH "*D*" IS PRESUMED TO COMPRISE SOLID ROCK STRATA (IF MATERIAL OTHER THAN SOLID ROCK COVER IS INCLUDED, IT IS NECESSARY TO DEMONSTRATE THE NATURE AND PERMEABILITY OF SUCH MATERIAL). THE ZONE OF NO EXTRACTION MAY BE INCREASED OR DECREASED, IF JUSTIFIED BY LOCAL OBSERVATION AND/OR EXPERIENCE.

FIGURE 8.6 SAFETY ZONE GUIDELINES FOR MINING IN THE VICINITY OF DAMS AND IMPOUNDMENTS

In guidance for evaluating breakthrough potential, MSHA (2003) indicates that any location where the cover over a mine entry is less than 100 feet of solid strata (especially at locations near the outcrop where additional weathering and stress relief has occurred) is a concern for sinkhole development. Accordingly, if sinkhole development cannot be reliably ruled out, preventive measures should be considered.

8.4.2.2 Pillar Failure

Loss of support due to coal pillar failure causes a mine roof to sag or collapse. This can create or open fractures in the overburden. These fractures may cause a roof fall and consequent sinkholes, or the fractures may create zones where internal erosion can occur. Furthermore, failure of one pillar transfers the load to surrounding pillars and may lead to progressive pillar failure (sudden or gradual) or excessive displacements over a relatively large area.

Pillar crushing occurs when the load on a pillar exceeds its strength. This can be caused by existing loads, additional loading from impounded slurry and/or water, loss of strength in the coal from chemical decomposition or from slow oxidation, mine fire, or loss of buoyant pressure resulting from a lowered mine pool. In addition to pillar strength, the pillar width to height ratio (w/h) is also important (Mark, 2006). For "slender" pillars ($w/h < 4$), failure often results in nearly complete loss of load-bearing capacity, sometimes with sudden and total collapse. Pillars with w/h between about 4 and 10 are largely elastic with a possible plastic core, and failures tend to occur gradually with post-failure residual strength essentially constant. The pillars deform until they have shed enough load to stop the process. Pillars with w/h greater than 10 (referred to as "squat") have a plastic core and may strain harden once the loss of initial strength due to crushing or yielding of the outer elastic portion of the pillar occurs. After this initial crushing, the pillars gain strength as they deform. The implications for surface structures of the failure of slender pillars with shallow cover are much more significant than those associated with yielding of squat pillars at great depth.

A number of formulas for analyzing the strength of a pillar have been developed, and computer programs for performing pillar analyses are available. One example is the program ARMPS (Analysis of Retreat Mining Pillar Stability) developed by National Institute for Occupational Safety and Health (NIOSH). This program uses the Mark-Bieniawski formula to determine pillar strength, and it has the capability to account for loadings on barriers and abutment pressures.

Pillar stability formulas can be divided into two categories – analytical and empirical. Analytical formulas require extensive material testing, an understanding of loading under varying conditions, and a safety factor (typically about 2) based on knowledge and understanding of all variables. These relationships are best applied by engineers who are experienced in mining rock mechanics. One such relationship, Wilson's equation, which is one of the first analytical models developed for estimating pillar strength, is directly calculated, thus making it more flexible and adaptable to actual conditions than any empirical equation. It can be used to estimate the stress distribution from the edge of a pillar to the center based on the confined core theory (Wilson and Ashwin, 1972). Wilson's equation uses the Mohr-Coulomb failure criterion for modeling the coal and surrounding rock; however, at high confinement (high w/h ratio) coal strength is not linear with the result that it overestimates pillar strengths. Scovazzo (1995) modified Wilson's equation to incorporate more appropriate coal and rock failure criteria, specifically those presented by Kalamaras and Bieniawski (1993).

The most commonly used pillar stability formulas are empirical equations where few parameters need to be defined. Since empirical equations are based on statistical analysis of failures and successful designs, a stability factor (not to be confused with safety factor) is employed. One commonly used empirical equation is the Mark-Bieniawski formula. For the Mark-Bieniawski formula, the recommended stability factor is 1.5 for mines less than 750 feet deep (Mark, 2006). A smaller stability factor is generally used for mines greater than 750 feet deep; for example, a stability factor of 0.9 is used for pillars with over 1,250 feet of cover. These recommended stability factors are based on the assumption of equal area loading. Empirical equations are applicable to specific mining regions and coal seams and should not be applied outside the region for which they were developed unless statistical analyses are performed for the region in which the equation is intended to be used. While some long-term pillar instability can be tolerated in certain mining situations, impoundment designers should consider a higher margin of safety for overburden support and control of detrimental differential movements within the dam and foundation.

Conservative coal strengths based on statistical analysis of failure should be used for pillar failure analysis. Laboratory tests for verifying the strength of the coal can be conducted, although this rarely occurs and is not encouraged. The limited use of testing is in part due to the scatter typically encountered in uniaxial rock tests, a problem that is exacerbated by coal cleating and softness, which make test samples difficult to prepare. A laboratory testing program should include sufficient samples and statistical analysis to verify that data are reliable. As a result, the mass uniaxial compressive strength of coal is often assumed to be 900 psi.

The uniaxial compressive strength of coal determined in the laboratory is many times the in-situ strength of coal in a pillar. This size effect is caused by flaws in the coal that are present in the larger mass of the pillar but not in sample sizes tested in the laboratory. It would take test samples with the impractical minimum dimension in the range of 3 feet (Pariseau, 1975) to 5 feet (Bieniawski, 1968) for accurate determination of the mass uniaxial compressive strength of coal. The most accepted method for reducing laboratory compressive strengths for coal to reflect in-situ strength of the coal was developed by Hustrulid (1976) and is recommended by Bieniawski (1992) for use with his pillar formula.

In any analysis of pillar stability, it is best if the method being used is calibrated to conditions at the subject mine. Thus, the model should be applied to a number of locations in the mine to determine how well it reflects actual conditions or the actual performance of mine pillars. Pillar analysis is more complicated when multiple seams are mined. In such cases, the loading conditions are much more complex due to load transfer and the potential for stress concentrations. Simplified software models, such as LAMODEL, are available for performing this type of analysis. Higher pillar stability factors should be used for multiple seam analyses to account for the additional uncertainty. Both LAMODEL and ARMPS are available from the NIOSH web site.

8.4.2.3 Pillar Punching (Floor Failure)

Pillar punching (pillar foundation failure) occurs when a pillar pushes into the mine floor allowing the roof to sag. This sagging can create fractures and/or open existing joints in the overburden. If the punching occurs over a large area, the surface will be affected in a similar manner (normally without bed separation) to the situation where total extraction of a thinner seam has occurred.

Pillar punching occurs when the load on a pillar exceeds the bearing capacity of the mine floor beneath it. Pillar punching can be caused by existing loads, saturation of the floor material causing softening and loss of strength, additional loading from impounded slurry and/or water, and loss of buoyant pressure from a lowered mine pool.

A key factor to consider in evaluating the potential for pillar punching is experience elsewhere in the mine. A review of floor and pillar performance data, particularly for wet conditions, should be made as part of foundation bearing capacity analyses for evaluation of pillar punching. It may be appropriate to reduce the floor strength when both slender pillars and wet conditions are present or are anticipated as the result of construction of an impoundment. Additional information on pillar punching and associated floor heave can be found in Ganow (1975) and in Adler and Sun (1968).

8.4.2.4 Outcrop Barrier Failure by Shear or Punching

In an outcrop barrier situation, a potential failure mode that must be considered is whether the pressure from water and/or slurry in the impoundment may become high enough to overcome the shear strength that is holding in place the plug of material separating the impoundment from the mine works. Failure can occur through the coal seam itself, through the strata above the coal seam, through a weakened floor layer, or along an interface between strata. Figure 8.7 illustrates a typical outcrop barrier cross section, including the coal barrier, jointing and stress relief fractures. Surface features such as slope instability and regrading for access roads are also shown on the figure.

Analysis of this failure mode for a postulated plug separating the impoundment from the mine involves comparison of the cumulative force tending to push the plug into the mine to the available resistance from shear forces around the perimeter of the plug. The pressure driving the plug is normally taken as the hydrostatic head from the impoundment plus applicable lateral earth pressure from settled fines. A major issue with this type of analysis is judging what to use for the shear strength along the top, bottom and sides of the plug. Factors such as the presence of weathered joints, the presence of cleats in the coal, the softening of the floor from saturation, and the difficulty of obtaining test data can result in uncertainties with respect to shear strength. Furthermore, as seepage into the mine occurs, the strength along potential shearing surfaces may degrade over time, and, if sloughing or a roof fall occurs, the thickness of the plug may be reduced. Additionally, the potential hydraulic gradient across the plug must be carefully considered, as water pressures may be elevated on planes of discontinuity within the plug and present a more severe loading condition than at the plug boundary. Given these uncertainties, any shear-type failure analysis should be based upon conservative values for shear strength and a conservative factor of safety of at least 2.0 (MSHA, 2003).

8.4.2.5 Internal Erosion

Internal erosion, or piping, is the movement of material (typically soil particles) under seepage forces, and it can occur where the gradation is such that particles are dislodged and carried along by seepage and drag forces. When soil overlies rock, movement of material can occur along or into joints, stress relief fractures, or subsidence fractures. Figure 8.7 illustrates typical conditions encountered near a coal seam outcrop where weathered joints may represent seepage pathways from a future impoundment. The presence of stress relief fractures or subsidence fractures would exacerbate the concern for seepage and internal erosion. Seeping water creates a drag force on the material that it is seeping through or around. If this drag force is greater than the frictional or cohesive forces holding material particles in place, they will be transported by the seeping water. As smaller particles are carried away, additional flow occurs, increasing the drag force and dislodging even larger particles. The process can continue until a channel (also referred to as a "pipe") is formed. Piping into foundation discontinuities can lead to failure of an embankment. In a breakthrough situation, where the source of the seepage is an impoundment and internal erosion occurs through the material between the pool and the mine workings, the pipe can gradually enlarge and lengthen to the point where there is a significant discharge into the mine. Piping is dependent on the type, gradation, consistency, and cementation of the intervening material; relative hydraulic conductivities of the materials along the critical seepage front; and the hydraulic gradient.

One technique for preventing internal erosion is to incorporate filter layers (material with grain-size distributions that prevent particles from moving along a seepage path) at critical locations. Granular filter material or various geosynthetic materials can be used for this purpose. Filter criteria are discussed in Section 6.6.2.3. Also, the gradient can be reduced by using lower-hydraulic-conductivity materials along the upstream portion of the seepage path to increase head loss before the water reaches a critical zone. Excess water pressure can also be controlled by providing drainage to reduce pressures upstream of more erodible zones, such as by installing an internal drain protected by filters.

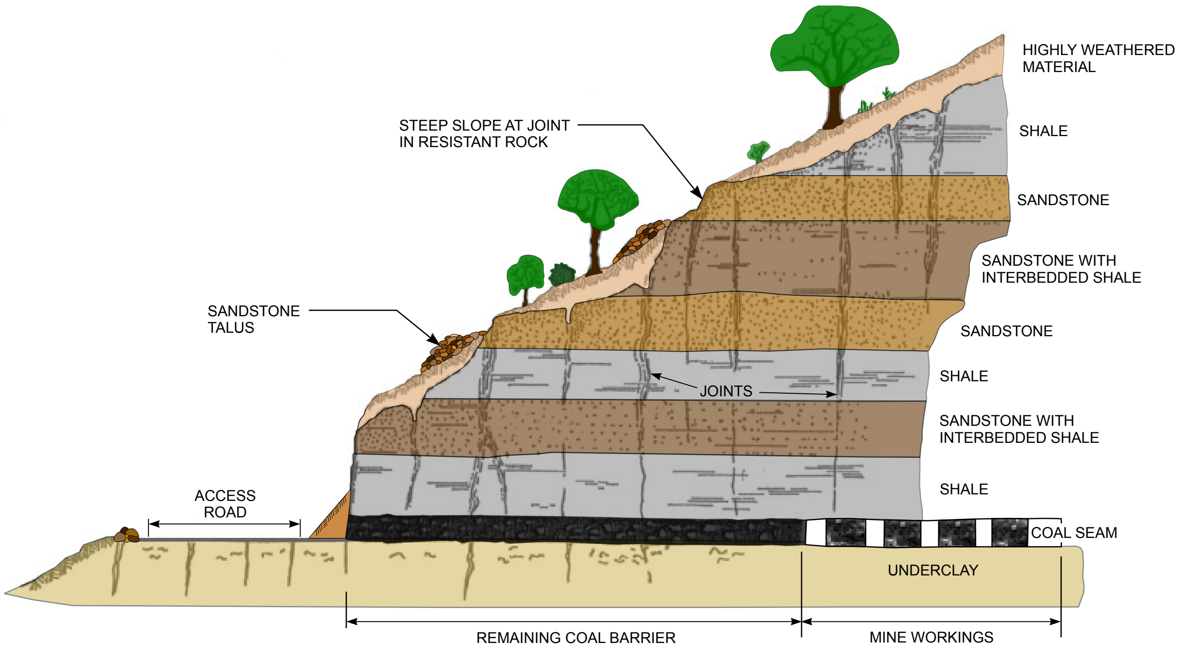


FIGURE 8.7 HILLSIDE DISTURBANCE AT COAL OUTCROP

In breakthrough situations, seepage into an underground mine can occur through the roof, floor, or coal seam. If the material at the location(s) where the seepage enters the mine is weathered, fine-grained, or loosened as coal may be along a face or rib, then the force of the seeping water may carry particles away and/or cause the in-situ material to slake or unravel. Over time, this can cause the area to weaken, slough, or progressively deteriorate. This is especially a concern near an outcrop barrier, where the internal erosion could cause the ground to give way or a pipe to form through the affected zone. Either situation could lead to an uncontrolled hydraulic connection between an impoundment and the mine workings.

Whenever seepage is flowing outward and exiting at an unconfined surface, as may be the case near the toe of a dam or impounding embankment, the value of the critical gradient with respect to piping is approximately one. However, when seepage is generally horizontal, such as through a rib, or vertically downward, such as through the mine roof, internal erosion can develop under smaller gradients.

Determining the hydraulic gradient between an impoundment and a mine requires accurate characterization of the permeabilities of the intervening materials. Near an outcrop, the seepage may be governed by flow along weathered joints and thus may be difficult to characterize and model. Gradients can be estimated by drawing flow nets or by using a finite element seepage program. Once the gradient has been estimated, the effect of the seepage force on the seepage medium can be determined. The seepage force, per unit volume of seepage medium, is equal to the product of the gradient and the unit weight of water. Whether the medium material will be dislodged by a combination of seepage and gravity forces depends on the frictional and cohesive strength of the material.

Cohesionless soils, particularly silts and fine sands, are most susceptible to piping. Clays are more resistant to piping because the cohesive strength of these soils helps to prevent particles from being carried away. However, clayey soils are not immune to internal erosion, especially if they abut open-graded materials, open joints or fractures. Also, soft rocks, such as weakly cemented sandstones, have been associated with piping failures. Even shales, which are usually considered resistant to piping, have developed piping voids under conditions of very high gradients (Sowers and Sowers, 1970). In these unusual cases involving rock strata, weaker characteristics of the rock mass presumably led to pathways of increased seepage that intersected and then led to piping of more erodible materials.

If the materials between the mine works and an impoundment are prone to internal erosion or are sensitive to progressive deterioration, then the impoundment should be designed: (1) with suitable filters and drains so that seepage can be collected and released in a controlled fashion, (2) with seepage barriers so that pressures are minimized, or (3) with some combination of these two approaches.

8.4.2.6 Bulkhead Failure

If there is a mine opening in a potential breakthrough area, then the potential for failure of the bulkhead used to block the mine opening must be evaluated. Depending on their thickness and shape, bulkheads can fail when the pressure acting on them causes the bending or shear strength of the bulkhead material to be exceeded. Bulkheads can also fail if the material that they are anchored or keyed into is not strong enough to resist the applied pressures and the pressures from water seeping around the bulkhead, or if the bulkhead is not adequately anchored to the surrounding rock strata. Hydraulic fracturing within floor strata can aggravate seepage and transmission of pressures resulting in the typical recommendation to remove underclay/claystone at bulkhead locations. Analysis of bulkheads is addressed in Section 8.5.2.

8.4.2.7 Trough Subsidence and Subsidence Cracks

If room and pillar workings are located near the footprint of an impoundment (with inadequate stability factors on the remaining pillars), or if either longwall mining or secondary mining of pillars has occurred, an analysis of the potential impact of trough subsidence should be performed. In this situation, zones of tension and compression stress or deformation extend from the mine workings to the ground surface. The area at the ground surface affected by total extraction mining is typically larger than the mined area and is related to the draw angle. The assumed draw angle should be consistent with local experience, with past subsidence associated with mining in the coal seam, site topography, and the method of mining.

The effect that subsidence has on steep natural slopes is a matter of debate (Luo et al., 1996). It appears that the effect is greater than the generally accepted regional angle of draw and is dependent upon local geologic conditions and the direction of mining. The approach of mining from the downslope and towards the upslope appears to result in the greatest horizontal displacement and accompanying stress.

If mining is approaching an impoundment, operations should be terminated at a point where ground deformations will not adversely affect the impoundment. It is rare for the angle of draw to exceed 25° in the U.S., and typical values range from 10 to 25°. In the Northern Appalachian Coalfield, the maximum angle of draw is normally 25°, while in the Black Warrior Basin and Southern Appalachian Coalfield, it is typically around 12°. Work by Scovazzo (2008) at three mines in the Arkoma Basin places this value at 19°. For the Illinois Basin, the angle of draw typically is in the range of 20 to 25° in the southern portion, increasing to 30° in the northern portion as glacial till thickness increases. The angle of draw varies with the dip of the coal seam and the slope of the ground surface and decreases as the percentage of hard rock (sandstone and limestone) in the overburden increases. No pattern for the western U.S. has been identified.

For total extraction mining (longwall mining or secondary mining of pillars) directly below an impoundment, the U.S. Bureau of Mines (USBM) guidelines provided in Babcock and Hooker (1977) indicate that the amount of cover should be at least 60 times the mining height, as illustrated in Figure 8.6. This guideline was based on studies that looked at disturbance of the rock strata, and the change in the hydraulic conductivity of these strata, above mined areas. Data indicate that it is unlikely that the rock strata above an area of total extraction mining will be disturbed for more than 25 to 35 times the mining height. This thickness guideline (Babcock and Hooker, 1977) is based on the concept of a constrained zone. When total extraction occurs, cracks and joints open at the surface and in the mine roof, but if the overburden is thick enough, the induced stress is absorbed or resisted without fracturing. It should be noted that the Babcock and Hooker (1977) strata-thickness criterion is for evaluation of the potential for developing a hydraulic connection between the surface and the mine; it does not address the potential for other adverse effects such as tensile strains at the surface and loss of freeboard.

Another USBM research report titled, *Criteria for Determining When a Body of Surface Water Constitutes a Hazard to Mining* (Kendorski et al., 1979), recommends cover thicknesses greater than 60 times the mining height when the mining height is less than 7.5 feet. For example, this report recommends that the overburden thickness should be 71, 80, 95, and 117 times the mining height for mining heights of 6, 5, 4, and 3 feet, respectively. The report also indicates that, where inflow can be tolerated, the thickness of cover can be reduced if certain types of strata (e.g., claystone or shales that are less prone to cracking and have low hydraulic conductivity) are present.

The preceding guidelines refer to bodies of surface water and do not specifically consider mitigating factors or measures such as fine refuse deposits that may be associated with a slurry impoundment. These guidelines are important historical literature and should be reviewed as part of initial evaluations; however, it is recommended that subsidence analyses and prediction models be used for evaluating potential deformations and strains on embankments and impoundments. On the basis of these analyses, smaller overburden buffers than identified in the aforementioned references may be sufficient to prevent a breakthrough to a reservoir or slurry impoundment and may satisfactorily limit impoundment leakage/seepage.

A number of subsidence prediction models are available. Two commonly used models include SDPS (Surface Deformation Prediction System) developed at Virginia Polytechnic Institute and State University and CISPM (Comprehensive and Integrated Subsidence Prediction Model) developed at West Virginia University. These programs can be used to estimate surface subsidence and ground strain caused by mining. Two cautions are offered concerning the use of these types of programs: (1) they should only be used for the type of topography and mining conditions for which they were developed and (2) the strain computations by these programs typically do not account for strain concentration along existing discontinuities. Concerning the second point, studies have shown that site topography may have a substantial effect on the development and concentration of horizontal strain.

Furthermore, where the confinement and continuity in the overburden is diminished, such as on hillsides and in highly weathered and fractured material, the tensile strain that is induced by mining may accumulate along one or more of the joints rather than being more evenly distributed. This can be significant in subsidence and breakthrough potential evaluations where open joints can provide seepage pathways. For total extraction mining, the strata and ground surface above the mine are affected regardless of the mining depth, and the effect of tensile strains and strata disturbance on dam stability and breakthrough potential should be evaluated.

8.4.2.8 Failures Related to Auger and Highwall Mining

Auger or highwall mining openings in an abutment can have adverse effects on an impounding embankment. Deformation of the abutment and embankment can occur if the webs between the holes deteriorate or fail. The holes can also provide seepage paths. Breakthroughs can occur through auger or highwall mining holes, through the coal remaining at the ends of auger holes, or as a result of the collapse of these holes. The importance of identifying the locations of these holes during site exploration cannot be overemphasized. During highwall reclamation these holes are typically covered without backfilling. If they are not discovered and accounted for in the impoundment design, then seepage pressures under high impoundment head can cause movement of backfill and water into auger holes. If the auger holes are accidentally connected to the mine, then there is a direct path for a breakthrough or piping. If the auger holes were terminated close to the mine workings, the remaining coal barrier can potentially fail in a shear or punch-type mode. Also, the narrow pillars or webs of coal between these holes may be marginally stable and may subsequently deteriorate and collapse, resulting in highwall instability and ground deformations that could adversely affect the embankment or lead to a breakthrough. NIOSH has a computer program (ARMPS-HWM) that performs an empirical analysis of highwall mine pillars.

8.4.2.9 Hillside Movement or Disturbance

Hillside movement and disturbance (landslide, creep, or human impacts) can occur at and above a coal outcrop. Weathering tends to reduce the strength of surface soils and rocks while gravity provides a driving force to move soil and rock down the hillside. Surface mining (contour mining) and site development activities, such as road and channel construction, can accelerate the process through removal of soil and rock from the hillside or by aggravating or causing landslides. Any movement or removal of soil and rock from the hillside can reduce and/or disturb the outcrop and overburden barriers between the underground mine and an overlying impoundment, thus contributing to the occurrence of a sinkhole, internal erosion, or shear-type failure. Figure 8.7 illustrates the effect of hillside disturbance on the coal outcrop barrier.

8.4.2.10 Mine Blowout

A mine blowout occurs when water impounded in an underground mine breaks out through either a sealed mine opening or a thin section of the coal outcrop barrier. Such a blowout may be a secondary consequence of an impoundment breaking into an underground mine. Additionally, a mine blowout in an impoundment watershed may represent an inflow source that should be considered in impoundment design. Such an event could cause a significant amount of water to flow into the impoundment, damage impoundment structures, and potentially affect impoundment safety. A guidance manual for design published by the Office of Surface Mining (OSM) is, Outcrop Barrier Design for Above Drainage Coal Mines, by Kohli and Block (2007). This document presents compiled information on case histories as well as design studies by Pearson et al. (1981). If seepage from an underground mine is occurring at an embankment or impoundment site, the conditions and potential effects should be evaluated. As a minimum, the facility design should account for collecting seepage and discharging it in a controlled manner that will not adversely affect the embankment or other facility structures.

8.4.2.11 Seismic and Blasting Events

Typically, earthquakes and surface blasting do not affect underground mines. The exception is when a fault crosses through a mine or when surface waves can affect the stability of the portals at or near their contact with the surface. The presence of an active fault passing through a U.S. coal mine is not a situation that normally occurs. If such a fault is identified, it would be necessary to conduct comprehensive stability analyses not just for the mine but also for the associated surface facilities including the refuse embankment and impoundment. Bhabdaru and Arora (1987), Nicholls et al. (1971), Rupert and Clark (1977), Fourie and Green (1993), and Fernandez and Van Der Heever (1996) describe blasting and earthquake impacts on mines.

When blasting is planned within 500 feet of an active underground mine, the Surface Mining Control and Reclamation Act of 1977 requires that the Operator's blasting plan be approved by OSM (or appropriate state agency) and MSHA. Blasting regulations are provided in 30 CFR §

780.13 and 30 CFR § 816.61 through 816.68 and also 816.79. State criteria may also be applicable. The blasting plan should be prepared by a professional licensed in the state where the blasting is to be performed. Potential concerns when an impoundment is present include fracturing of abutments, impacts to pipes and other rigid structures, or possibly upstream construction. Where an impoundment is present, the potential impacts should be evaluated, and monitoring of particle velocity with a seismograph may be appropriate. Section 6.6.7 addresses potential blasting impacts on impoundment structures.

8.4.3 Mine Subsidence Potential and Analysis

8.4.3.1 Pillar Evaluation and Analysis

For room and pillar mining, subsidence potential can be evaluated through analysis of the overburden stress imposed on coal pillars and comparison of this stress to the pillar strength. To determine the average stress on a coal pillar, the tributary area approach may be used with the assumption that the overburden pressure increases at a rate of 1.1 psi per foot of depth:

$$S_p = 1.1\,H\left[(w+B)/w\right]\left[(L+B)/L\right] \tag{8-1}$$

where:

S_p = average pillar stress (psi)
H = depth below ground surface (ft)
w = pillar width (ft)
L = pillar length (ft)
B = opening width (ft)

Other assumptions inherent in this formula include:

- The coal seam is subject only to vertical pressure.
- Each pillar supports the column of rock and other overburden overlying the pillar and a proportionate share of surrounding openings.
- The load is distributed uniformly over the pillar cross section.

Most empirical pillar equations have been developed and the results statistically analyzed based on these assumptions, particularly with respect to tributary area loading. Therefore, empirical equations should only be used for uniformly distributed tributary loading. Numerical models such as LAMODEL can be used to analyze non-uniform pillar loading (transfer of load to flanking barrier pillars). However, if pillars are to be analyzed without considering tributary area loading, then the pillar loads should be determined through a detailed rock mechanics analysis using numerical modeling software. Typically, a higher safety factor is used for such a pillar analysis because of uncertainty as to the loading and the variability in the numerical methods typically employed.

Pillar strength formulas used for design of ground control measures are discussed in Bieniawski (1992). For subsidence evaluations for coal refuse disposal facilities, pillar strength formulas consistent with conditions in the underlying mine should be adopted; otherwise formulas applicable to local mines should be employed. Several empirical formulas that have been used to determine pillar strength are presented in Figure 8.8. The Pittsburgh Coal seam with a pillar height of 10 feet is used as an example of the influence of the width to height ratio on the overall pillar strength. The potential for subsidence can be determined by dividing the pillar strength by the pillar load. An important consideration when comparing the pillar strength formulas is the value of the recommended stabil-

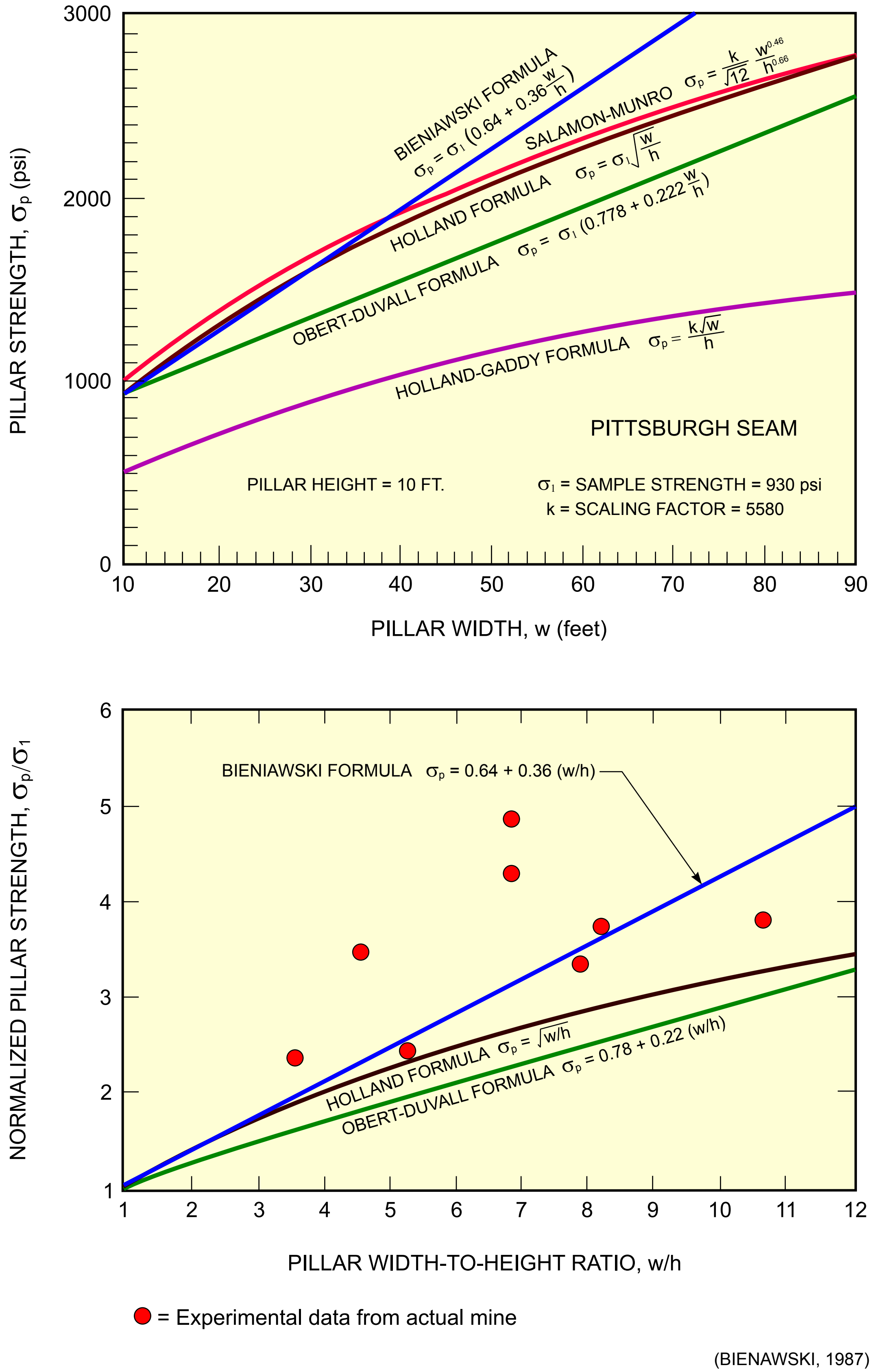


FIGURE 8.8 COMPARISON OF PILLAR STRENGTH FORMULAS WITH RESPECT TO WIDTH-TO-HEIGHT RATIO

ity factor, which varies depending on the empirical formula used (Bieniawski, 1992). Mark (2006), in addressing the stability factor for the Mark-Bieniawski Formula, recommends a stability factor of 1.5 for pillars with under 750 feet of cover and 0.9 for pillars with over 1,250 feet of cover. As previously noted, the term "stability factor" should not be confused with "safety factor." While some long-term pillar stability can be tolerated in certain mining situations, impoundment designers should consider a higher margin of safety for overburden support and control of detrimental differential movement within the dam and foundation.

While the occurrence of subsidence over mine workings in some areas is a function of time, the general assumption for pillar analyses is that coal does not deteriorate over time. The reason is that coal is highly jointed due to cleating, and additional cracking of the coal does not reduce the confinement of the roof and floor, nor does it markedly reduce the coal's mass strength. Initial pillar designs typically take into account spalling for coal seams susceptible to spalling. This is done by reducing the effective dimensions of the pillars by the anticipated spalled width and widening the opening width by this spalled dimension. Oxidation of the coal can occur for seams above drainage, but this effect tends to be limited to the yielded outer portions of the pillar. Once pumping and ventilation is discontinued, mines below drainage will flood, thus mitigating further deterioration of the coal.

The weathering of soft (high-clay-content) partings (non-coal layers in the pillars) reduces pillar strength over the long term, although the impact appears minor and may not occur after the mine is flooded (Biswas et al., 1995). Biswas et al. (1999) presented an approach for estimating time-dependent deterioration of partings, as well as coal. The authors indicated that much work remains before time effects on coal pillar strength are fully understood.

Computer programs for pillar stability analysis include the ARMPS and LAMODEL models. These programs can be used to evaluate stress distribution and factor of safety. Analyses can be performed considering the pillar sizes and arrangement within the mine, recognizing that some smaller pillars may shed load to larger pillars.

In addition to pillar stability, floor stability is also related to subsidence potential. Bearing capacity analyses have been applied to evaluation of pillar punching into soft floor materials, using the following formulas (Bieniawski, 1992; Brady and Brown, 2004):

For long ($L/B > 10$) pillars:

$$Q_u = 0.5\,(\gamma B N_1) + (c N_c) \tag{8-2}$$

For pillars with $L/B < 10$:

$$Q_u = 0.5\,(\gamma B N_1 S_1) + (c \cot\phi\, N_q\, S_q - c \cot\phi) \tag{8-3}$$

where:

N_c = $(N_q - 1)\cot\phi$

N_1 = $1.5\,(N_q + 1)\tan\phi$

N_q = $\exp\left[\pi \tan\phi\right]\left[\tan^2(\pi/4 + \phi/2)\right]$

S_1 = $1.0 - 0.4\,(B/L)$ (dimensionless shape factor)

S_q = $1.0 - \sin\phi\,(B/L)$ (dimensionless shape factor)

γ = density of floor strata (pcf)

B = width of pillar (ft)

L = length of pillar (ft)

c = cohesion of the floor strata (psf)

ϕ = friction angle of the floor strata (degrees)

Bowles (1996) presents several other bearing capacity equations that are applicable to conditions/situations not reflected in the above equations, including: (1) evaluation of strength parameters for rock for use in bearing capacity analysis, (2) the presence of adjacent pillars, (3) the presence of layered foundations, and (4) the effects of horizontal movement of very soft materials below the pillars. Equations developed by Terzaghi, Vesic, and others include parameters to account for these situations. Bowles (1996) presents bearing capacity factors for rock that are recommended for use in the Terzaghi equation. Bowles also presents bearing capacity failure modes for evaluating heave considering the effect of adjacent pillars (using an excavation trench model) for layered strata and for the case when the opening widths are less than 0.7 times the pillar width. Bieniawski (1987) presents equations based on the work of Vesic for analyzing multilayered conditions. Vesic's equation is commonly used for the Illinois Basin because of work by Y. P. Chugh from Southern Illinois University. Similar to pillar partings, floors of mines that are high in clay content tend to deteriorate over time, but this effect is limited to the unconfined floor in openings and near pillar edges (Chugh et al., 1987).

The strength parameters (c and ϕ) of the mine floor strata can be affected by moisture. Just as the evaluation of pillar strength should reflect pillar performance at local mines, available records of floor performance should be reviewed when selecting strength data for analysis. For evaluating pillar punching potential, Bieniawski (1992) recommends a factor of safety against bearing capacity failure (q_u/S_p) of 2.0 based upon the assumptions associated with computing the average pillar stress, although there are situations where a lower factor of safety (generally at least 1.5) is acceptable to provide long-term stability.

It is also possible that very soft material can be squeezed from beneath a pillar. This type of foundation failure is appropriately called a squeeze failure and can be analyzed as described in Bowles (1996).

8.4.3.2 Subsidence Evaluation and Analysis

Empirical approaches for subsidence analysis include: (1) the graphical method, (2) profile functions and (3) influence functions. The graphical method involves use of charts or nomographs and is generally applied in only a few geologically-similar regions. The profile functions approach involves the derivation of mathematical functions that describe the profile of the subsidence trough at the surface based on observed data. This method can be applied to geologically dissimilar conditions by modifying the profile constants, but the method is limited to predicting subsidence at specific points. The use of influence functions involves application of the principal of superposition to evaluate surface subsidence at any point influenced by the underground mine, based upon measured profile data. This method is based upon the assumption of homogeneous, isotropic overburden material, but it has been found to be suitable for subsidence prediction over underground workings with irregular or complex geometries. Singh (1992) provides selected profile and influence functions and examples. Continuum mechanics utilizing various elastic-plastic models has also been employed to analyze subsidence, although the complexity involved in depicting overburden response to mining has limited its application.

Computer programs such as SDPS and CISPM can be used for evaluation of stress at mine areas with varying overburden conditions, pillar and barrier arrangements, and subsidence parameters (settlement, horizontal displacement, curvature and tilt). Based on empirical or site-specific regional parameters, SDPS calculates the ground deformation indices using both the profile function method and the influence function method. The profile function method requires the following minimum input:

panel width, overburden depth, seam thickness, and percent of hard rock within the overburden. The influence function method requires that the mine plan and measured subsidence survey information applicable to the area be input, although average parameters applicable for eastern U.S. coal fields can be selected. The results can be plotted in relation to mine or surface structure geometry.

Procedures for the evaluation of subsidence at coal refuse impoundments overlying mine workings were discussed by Newman (2003) and Karmis and Agiotantis (2004) and include the following steps:

- Digitizing the mine layout plan and ground surface topography.
- Inputting the mine layout plan and topography into software such as SDPS.
- Evaluation of the geologic and overburden properties for classifying and establishing the dimensions of hard rock strata.
- Calibration of the model using regional parameters (e.g., influence angle, strain coefficient, edge effect).
- Establishment of extraction characteristics and the mining height of each panel.
- Adjustment of the extraction geometries based on mine characteristics.
- Calculation of the pertinent deformation indices.
- Contouring and superposition of the results on mine and topographic maps.
- Evaluation of the results.

8.4.3.3 Subsidence Damage Criteria

The strain criteria presented in Table 8.5 have been used in assessing the potential for subsidence impacts on earthen embankment dams and water bodies at the ground surface. Subsidence settlement can affect drainage features and freeboard, as well as impounding structure stability, thus site-specific subsidence parameters need to be established. Impacts on the internal integrity of structures are not clearly related to any specific magnitude of subsidence, but rather to differential movements. For undermined impoundments, the National Coal Board (1975), Babcock and Hooker (1977), and Whittaker and Reddish (1989) have published guidelines based on case studies for limiting surface tensile strain in the overburden to a range of 0.5 to 1.5 percent with the objective of minimizing the inundation hazard to mines. These guidelines, which are empirical, are based upon data from reportedly successful mine operations beneath bodies of water. When sufficient engineering data and mining experience are available, these conservative guidelines should be updated based upon the new data. While strain levels are a rational basis for assessing structural impacts, the difficulty in accurate prediction and measurement of strains suggests that prudence is warranted with respect to establishing tolerable strain and assessing safety zone offsets based on a strain criterion.

Additionally, a refuse embankment or impounding structure may also be subject to damage and failure due to surface tensile strain. Tensile strain has been reported as a gauge of impending cracking in earth embankments by Sherard (1973), who reported that initial embankment cracks generally occur in the range of 0.1 and 0.3 percent tensile strain, as discussed in Section 6.6.3.1. A study of the effects of subsidence on spoil heaps in England found that cracking occurred where observed tensile ground strains exceeded 0.3 percent (Forrester and Whitaker, 1976). Cracking can endanger a dam by reducing the strength of an embankment slope and by creating paths for seepage and material movement resulting in internal erosion. This criterion (as opposed to loss of strength) is particularly relevant to embankment features that control seepage such as liners and cohesive soil cores.

Other structures that may be part of refuse embankments include conduits and pipes, which may be sensitive to low levels of strain depending on their constituent materials and the direction of straining relative to their orientation. Peng and Luo (1993) report that the critical tensile strain for strain-sensitive structures is approximately 0.2 percent, which is the same as the strain associated with the appearance of visible cracks in masonry walls.

Nieto (1979) published a case history for central Illinois illustrating an evaluation of underground mining and subsidence at a depth of 620 feet. The study involved assessment of the potential for damage to an earth dam associated with adoption of a limiting tension strain of 0.25 percent, consistent with the references cited above and in Section 6.6.3.1.

8.4.4 Mine Breakthrough Potential and Analysis

In this section, mine breakthrough potential and analysis are discussed for: (1) a coal seam with mine workings outcropping into the impoundment area (outcrop barriers) and (2) a coal seam with mine workings extending beneath all or a portion of the impoundment pool (overburden barriers).

8.4.4.1 Outcrop Barriers

Outcrop barriers between mine workings and an impoundment, where the coal seam outcrops into the impoundment, may include:

- Natural soil, coal, and rock overburden between the mine workings and the ground surface
- Coal barriers separating auger holes and house-coal adits from underground mine workings
- Bulkheads and seals consisting of shot rock, aggregate, coarse refuse, grout and/or concrete used to backfill "punch-outs," auger holes, portals, ventilation boreholes, and rock dust or utility boreholes.

The primary modes of outcrop barrier failure include: (1) uplift failure, (2) shear plug failure, and (3) fractures initiated through discontinuities or subsidence that become enlarged as a result of seepage and internal erosion. Uplift failure occurs when buoyant forces reduce the overburden weight to the point where the differential pressure on the outcrop barrier causes failure. It can be analyzed separately or in conjunction with shear plug failure. Shear plug failure analysis is generally applied to outcrop barriers along coal-floor and coal-roof planes, assuming that the overburden is completely saturated. Because of uncertainties about the limits of mine workings, as well as overburden material properties, conservative assumptions are generally adopted. Figure 8.9 illustrates a sliding wedge analysis at an outcrop barrier and the associated free body diagram for computation of forces using the approach proposed by Newman (2003). Following this approach, which considers only the resistance along the coal-floor surface, calculation of the factor of safety *FS* for a unit width along the sliding wedge can be determined from:

$$FS = \frac{\left[F_{slurry} + F_{rock} + F_{coal}\right] \tan \phi}{F_{water} + F_{K_o}} \tag{8-4}$$

where:

$$F_{slurry} = \left[(w\, h_{slurry}) - \frac{w^2}{\underline{2S}}\right](\gamma_{slurry} - \gamma_w)$$

$$F_{rock} = \left[\frac{w^2}{\underline{2S}}\right](\gamma_{rock} - \gamma_w)$$

$$F_{coal} = \left[w\, h_{seam}\right]\left(\gamma_{coal} - \gamma_w\right)$$

$$F_{water} = h_w\, h_{seam}\, \gamma_w$$

and:

F_{slurry}	=	effective weight of slurrry (lb/ft)
F_{rock}	=	effective weight of overburden rock (lb/ft)
F_{coal}	=	effective weight of coal (lb/ft)
F_{water}	=	effective force of water (lb/ft)
F_{K_o}	=	lateral force on barrier from overburden (lb/ft)
h_w	=	net hydrostatic head on coal seam (ft)
h_{seam}	=	coal seam height (ft)
γ_w	=	unit weight of water (pcf)
γ_{slurry}	=	unit weight of slurry (pcf)
γ_{rock}	=	unit weight of overburden rock (pcf)
γ_{coal}	=	unit weight of coal (pcf)
ϕ	=	shear strength for barrier contact (degrees)
w	=	outcrop barrier width (ft)
S	=	ground slope at outcrop (dimensionless)

The effective lateral force on the barrier from overburden F_{K_o} is given by the following relationship, based on the properties of the slurry and conservatively treating the soil/weathered coal as slurry:

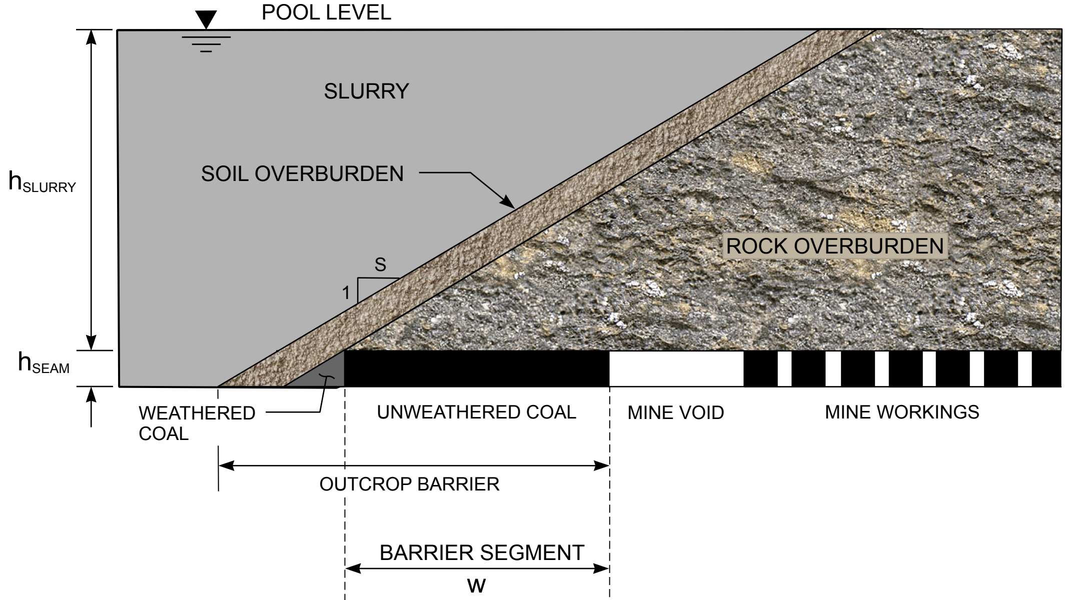
$$F_{K_o} = \left[\left(\gamma_{slurry} - \gamma_w\right) h_{slurry}\, h_{seam}\right]\left(1 - \sin \phi_{slurry}\right) \tag{8-5}$$

where:

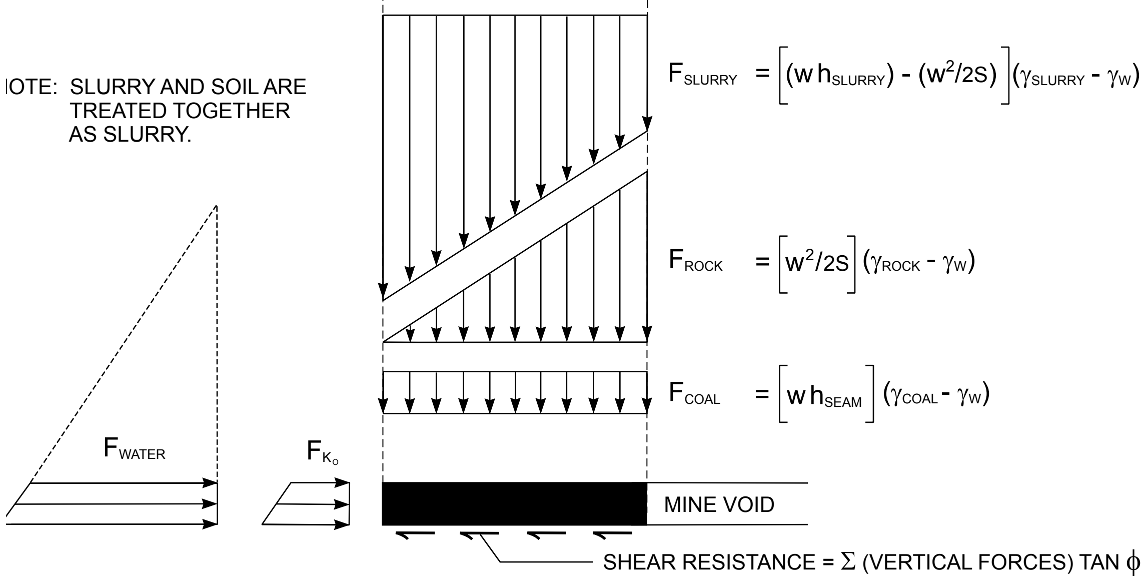
h_{slurry}	=	depth of slurry to barrier (ft)
h_{seam}	=	coal seam height (ft)
ϕ_{slurry}	=	shear strength of slurry (degrees)

The preceding equations can be modified to include resistance along the roof-coal interface and to incorporate protective embankments at the outcrop. In applying the analysis, the weight of overburden rock and coal may be taken as the total weight only if the mine workings are not and will not be flooded. The effective (or buoyant) weight is applicable if the coal and overburden are affected by seepage and the mine workings are flooded. Conservative assumptions associated with this analysis include:

- Hydrostatic impoundment head is assumed to be acting at the barrier.
- There is no cohesion along the coal-floor interface, and the resistance along the roof-coal interface is ignored consistent with Figure 8.9.
- Three-dimensional effects are not considered (side resistance of the wedge of coal should generally be ignored because of the potential for discontinuities).



8.9a ILLUSTRATION OF OUTCROP BARRIER



8.9b SIMPLIFIED FREE-BODY DIAGRAM OF EFFECTIVE BARRIER SEGMENT

(ADAPTED FROM NEWMAN, 2003)

FIGURE 8.9 OUTCROP BARRIER BREAKTHROUGH ANALYSIS FOR SHEAR FAILURE

Selection of the barrier width *w* requires a thorough evaluation of mine maps and exploration information to identify the representative minimum dimension, based on the following:

- Extent of weathering of the outcrop, which can best be evaluated by exploration and sampling.
- Potential for spalling and deterioration of the barrier rib within the mine workings, which can best be judged based on pillar performance within the mine. Biswas et al. (1999) introduce a method for estimation of the decline of pillar strength with time.

- Discontinuities in coal seam or overburden (e.g., stress relief fractures to surface) that may allow hydrostatic pressures to reach the interior of the barrier, the potential for which can be evaluated based on local geologic studies and surficial reconnaissance above the outcrop.

The preceding analysis is based on the assumption of a horizontal outcrop barrier. Adjustments to account for the dip of the coal seam and barrier may be warranted as well as evaluation of possible wedge failure across overburden bedding planes where fracture systems with such orientation are present.

If a man-made barrier instead of coal is being evaluated, the preceding analyses should be performed using the properties of the barrier. The following guidance is provided:

- Earthen materials and aggregate barriers – The strength properties of earthen materials or aggregate should be established based upon how the material was placed to form the barrier and accounting for resistance at the barrier-floor interface only (i.e., resistance between the barrier and the coal seam or roof is ignored).
- Concrete, concrete block, and grouted aggregate barriers – Construction details associated with the barrier keys should be included in the analysis of resistance to failure, along with the interface friction between the barrier and floor/roof, as subsequently discussed in Section 8.5.

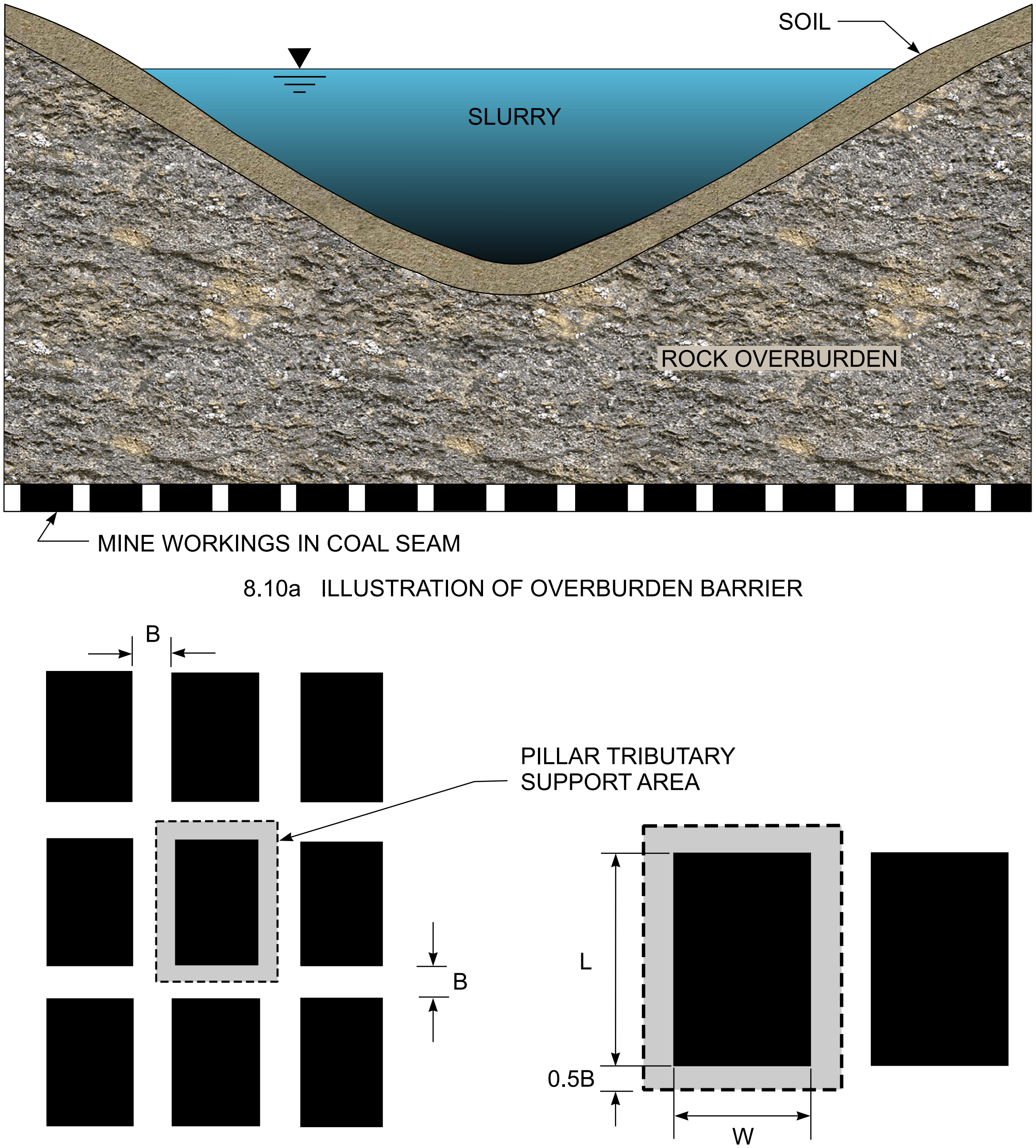
Fractures in overburden above an outcrop may have the potential for seepage and internal erosion that could propagate to an impoundment and lead to breakthrough. Natural stress relief fractures, if close to mine workings or subsidence cracks can lead to the potential for breakthrough from internal erosion. Breakthrough potential for this mode of failure can be determined by evaluating: (1) the presence of or potential for fractures and their influence on mine stability, (2) the occurrence and magnitude of subsidence strains and deformations, and (3) the type and effectiveness of any mitigation measures employed. Mitigation measures may include features to control susceptibility to internal erosion such as self-healing granular soils within the overburden and design of fills and filters to prevent piping.

8.4.4.2 Overburden Barriers

When mine workings extend below drainage and beneath the bottom of an impoundment, attention should be focused on the overburden, roof, pillar, and floor stability, as illustrated in Figure 8.10. The potential for breakthrough could be associated with the following scenarios:

- Sinkhole subsidence under relatively thin overburden strata where a roof fall propagates into the overburden and eventually daylights into the impoundment.
- Fracturing of the overburden due to pillar collapse or floor failure, leading to internal erosion propagating to the impoundment.

Analysis of roof stability and sinkhole development is addressed in Section 8.4.2. The potential for these phenomena can be determined empirically using the Rock Mass Rating (RMR) (Bieniawski, 1992) or the Coal Mine Roof Rating (CMRR) (Molinda et al., 2001) and analytically by calculating beam or arch safety factors for each stratum. The analysis of pillar and floor stability, which was addressed in Section 8.4.3, is based on the stability factors of pillars that may lie within a reasonable angle of draw around the periphery of the impoundment, considering room and pillar dimensions, mining height, overburden, and coal strength. Newman (2003) presents an analysis methodology and associated case history for evaluation of breakthrough potential for an overburden barrier.



8.10b SCHEMATIC PLAN OF PILLARS

8.10c TYPICAL PILLAR DETAIL

FIGURE 8.10 ILLUSTRATION OF OVERBURDEN BARRIER

8.5 SUBSIDENCE AND BREAKTHROUGH MITIGATION METHODS

Mitigation measures to address the potential impacts of subsidence or breakthrough may include some combination of:

- Providing a safety zone around the embankment and impoundment, limiting mining to only entry development beneath the dam.
- Providing support by backfilling portions of the mine.

- Improving the in-situ materials by grouting.
- Constructing an engineered barrier.
- Isolating the structure from the area of influence of the mining or altering the mining sequence or plan.
- Constructing secondary defense measures such as bulkheads to contain a breakthrough within the mine.
- Other engineered measures or impoundment operating procedures to control seepage and reduce pressures in the areas of potential breakthrough.

Subsidence mitigation measures are dependent on the structure that must be protected. Mine workings beneath a dam must be stabilized such that support for the impounding embankment and abutments is achieved. Overburden or outcrop barriers beneath an impoundment must be sufficient to prevent breakthrough. Mitigation measures for breakthroughs are discussed in MSHA (2003) and NRC (2002) and are summarized in the following sections, which describe potential mitigation alternatives and methods for the design of bulkheads to seal mine entries.

8.5.1 Mine Subsidence and Breakthrough Mitigation

8.5.1.1 Use of Safety Zones

For new impoundments, the most effective method for preventing damage to the dam or a breakthrough is to leave an unmined safety zone between the mine workings and the impoundment so that any mining-induced ground disturbance cannot cause a breakthrough or other significant adverse effects. At new facilities, siting of the impoundment at locations that are not or will not be undermined is preferred. If mine workings cannot be avoided, other mitigation measures may be feasible provided that support for the impounding embankment and appurtenant structures is achieved.

For existing impoundments where the mine workings are already close enough to potentially cause a problem, a safety zone can be created (if necessary) by backfilling the mine workings. Guidelines for sizing safety zones around impoundments are provided in Babcock and Hooker (1977). Figure 8.6 provides an illustration of these guidelines. Kendorski et al. (1979) provide criteria for determining when a surface water body represents a hazard to mining. Peng and Luo (1993) provide guidance for establishing safety zones for sensitive structures.

8.5.1.2 Mine Backfilling

If the thickness and natural characteristics of the overburden barrier cannot be relied on to prevent a sinkhole, subsidence cracks, or other subsidence-related failure mechanisms, filling previously-mined areas with grout or other material (commonly referred to as "stowing") may be a necessary remedy. The lateral extent of backfilling must include critical areas within the angle of draw based on the results of subsidence analyses described in Section 8.4.2.

The mine backfill material can have minimal strength, and even a partial backfill will offer confinement to mine pillars, thereby reducing spalling and dramatically increasing pillar strength. However, it is good practice to backfill to the roof of the mine, using material with sufficient strength (typically above 100 psi) to reduce consolidation and prevent erosion. This can be difficult to accomplish, as full contact with the mine roof is often not possible because of roof irregularity and the rolling nature of coal seams. However, partial roof contact will reduce roof falls and will dramatically limit roof fall propagation into the overburden.

Pozzolan slurry makes a good backfill, because it readily flows into irregular spaces. Because of the high sulfate environment in a coal mine, any fill containing cement should employ sulfate-resistant cement, preferably Type V, but in some mine environments Type II cement will suffice. Depending on cement availability, Type II or Type I cement in combination with pozzolans (e.g., fly ash, slag cement) may be required in order to provide adequate sulfate resistance. The extent of the backfilled areas must be sufficient to support the overburden and/or hillside that could potentially be inundated by the impoundment and to protect existing or planned embankment dams and mine seepage barriers from adverse subsidence effects. Boreholes should be advanced to determine if the backfilling program has achieved performance criteria. These boreholes can also be used to locate remaining voids, to perform secondary grouting, and to obtain samples from within the backfilled mine workings for strength testing.

To verify that the backfill strength and coverage is adequate, subsidence and/or pillar analyses should be conducted as part of the mine backfilling design. As a minimum, the following should be specified as part of the backfill design:

- Strength of the backfill material
- Area to be backfilled
- Methods for verification that the design strength and area of backfilling are acceptable

When used with cement, most types of fly ash improve flowability, increase sulfate resistance, and sometimes increase the compressive strength of backfill grout. Fly ash improves the flowability of grout because the spherical particles act like ball bearings that allow the grout to move more freely and the small particle size promotes better filling of voids. In addition, the pozzolanic properties of most types of fly ash increase compressive strength. Use of fly ash also reduces shrinkage and slows setup time, which is important if grout pumping must be interrupted for a few hours. The properties of fly ash will vary dependent upon the coal source used at the originating power plant. Fly ash properties are generally determined by the power companies that generated it and these properties can often be obtained from them.

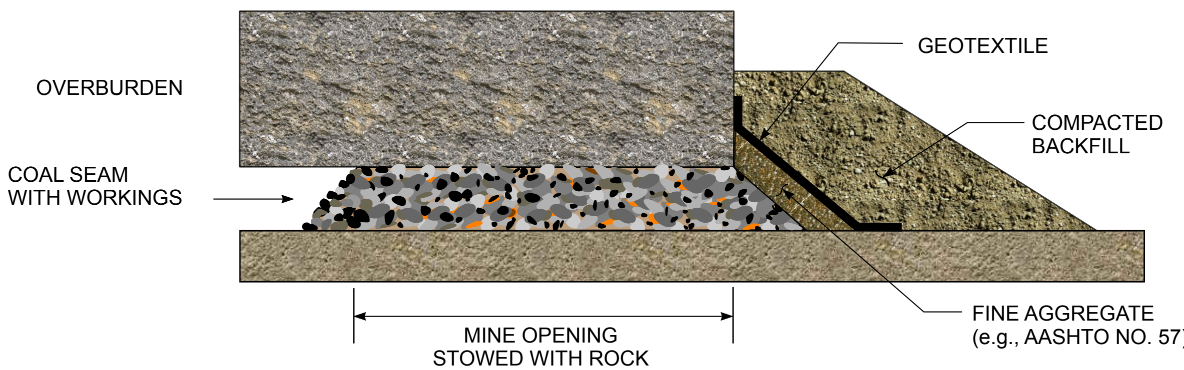
Fly ash can pose potential environmental and health risks because it contains trace amounts of toxic metals such as boron, molybdenum, selenium, and arsenic. Portland cement also contains these elements, and they can occur naturally in soil and water. (USEPA, 2000) discusses environmental impact considerations associated with the use of fly ash in mine environments.

8.5.1.3 Stowing of Mine Openings and Associated Barrier Construction

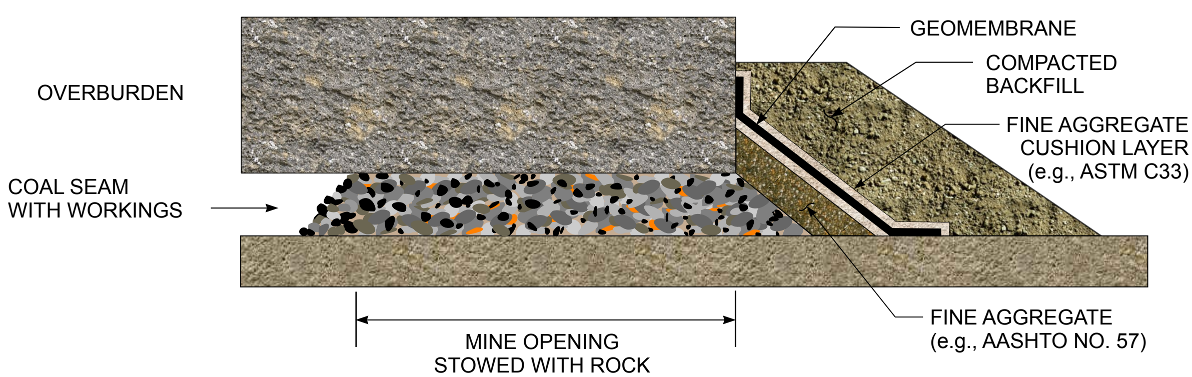
It has been common practice to seal mine entries by stowing them with competent rock or other fill in conjunction with constructing a compacted earth or coarse coal refuse embankment on the outside against the openings. This approach can also be used with auger holes. The openings should be filled far enough back into the mine to prevent adverse subsidence or sinkholes that could extend to the ground surface under the dam or impoundment. Typically, aggregate or other rock materials are pushed or rammed into openings to the extent practical with construction equipment. If possible, stowing should extend into the opening for a distance sufficient to mitigate subsidence. In some applications, including areas beneath a dam or abutment, grouting is performed in order to extend the backfilling of the opening, support the roof, and mitigate seepage. In an impoundment area, when it is impractical or infeasible to mitigate subsidence by stowing, other options can be considered such as: (1) earthen barriers designed to provide protection against potential sinkholes or cracks and (2) overexcavation to remove shallow overburden subject to sinkholes and establish a highwall cut. Monitoring systems for detecting movement and/or hydraulic pressure should be considered.

Earth or coarse coal refuse barriers should be protected by appropriate geotextile or graded filters to prevent piping into the mine, open joints, and stowing material. Drains should be installed as needed to control the hydrostatic pressure. The system of the stowed and external barrier materials must be designed to have sufficient strength and dimensions to resist shear or punching failure into the mine resulting from hydraulic and earth-load forces. The stowed material must also have sufficient bulk and material gradation to provide adequate seepage resistance and thus prevent failure of the embankment or adjoining strata due to piping or hydraulic fracturing into the mine. Figure 8.11a illustrates a stowed opening protected with geotextile and backfill. The fine-aggregate filter shown in the figure should cover the area where fractures or open joints are present or are likely to be a concern.

If a coarse coal refuse embankment is to be constructed over a coal seam outcrop with workings, the exposed coal seam must be covered and sealed with soil or other inert material to provide a fire barrier and to minimize the potential for spontaneous combustion. If water is draining out of the mine from an opening that is to be covered, a drain to prevent water from building up in the mine may be needed. For these cases, the sealing cover should include a drainage system that will release the mine water and prevent hydrostatic pressures from building up and causing problems with respect to saturation, piping, or structural instability. A discussion of internal drain and filter design is provided in Section 6.6.2. If the mine discharge is acidic, drain materials capable of resisting degradation will be required (Section 11.7), and downstream water collection and treatment could also be required (Section 10.3.3).



8.11a BACKFILL BARRIER WITH GEOTEXTILE



8.11b BACKFILL BARRIER WITH GEOMEMBRANE

FIGURE 8.11 STOWED MINE OPENINGS AND BARRIER CONSTRUCTION

Impervious membranes have been used in conjunction with mine opening seals. A membrane should extend a sufficient distance past the perimeter of an opening to provide an effective barrier encompassing mining-induced fractures and open joints. To prevent seepage from flowing around the membrane, the edges of the membrane should be anchored or embedded. The membrane should be surrounded by a layer of finer-grained, cushioning material to prevent puncture by sharp rocks that may be present in the embankment or highwall. Figure 8.11b illustrates the use of a geomembrane as part of a mine opening seal. Liner systems are discussed Section 10.4.

If mine openings are safely accessible to workers or can be rehabilitated so that they are safely accessible and plans for entry are submitted and approved by MSHA (30 CFR Subchapters G, H I and O), form-work or bulkheads can be constructed to restrict the extent of stowed material to a desired depth into the mine. In this situation, workers could enter the mine to install grout pipes or to do other needed work. This approach has been used successfully for both the pneumatic stowing and grouting of gravel and for installation of a grouted rock plug. Pneumatic stowing and grouting methods are described in the U.S. Bureau of Mines Information Circular 9359 (Walker, 1993). The following considerations apply: (1) some material types or uncontrolled (uncompacted) placement can be subject to erosion under water flow and (2) the operation exposes personnel to dust. These concerns can be overcome by grouting, although the use of cement adds cost, and environmental impacts can be an issue. If the mine opening can not be made safely accessible to workers, rams mounted on heavy construction equipment can be used to pack the stowing material into the opening, but the depth to which the opening can be backfilled is limited.

In addition to stowing the mine opening, measures to isolate the sealed opening from the impoundment and to control hydraulic pressure may also be needed. A compacted embankment fill with an internal drain protected by a filter discharging beyond the limits of the impoundment provides additional protection and redundancy for mitigating breakthrough. Figure 8.12 shows an example of a compacted fill (prior to refuse placement) placed against an outcrop and with a drain wrapped in geotextile and spoil. Drain and filter design requirements are discussed in Section 6.6.2.

8.5.1.4 Bulkhead Construction

Construction of bulkheads inside a mine or at mine portals can be a feature of a breakthrough prevention program. Bulkheads are designed to withstand fluid and/or earth pressures. They may be relatively thin reinforced-concrete structures or relatively thick structures consisting of concrete, grouted rock or polyurethane foam.

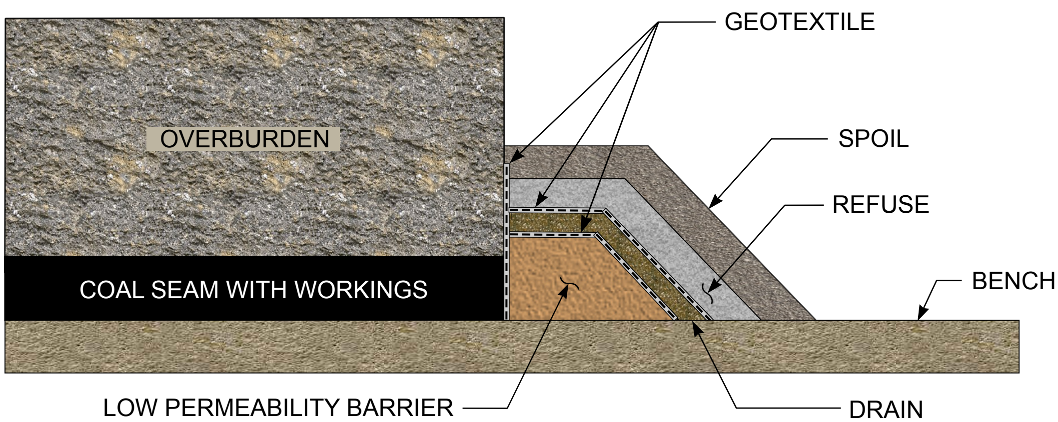


FIGURE 8.12 BARRIER AGAINST OUTCROP WITH LOW-PERMEABILITY FILL AND DRAIN

In situations where there is uncertainty about the level of breakthrough protection provided by other preventative measures, remote bulkheads can be used as a secondary defense against the discharge of water or slurry through and from a mine. These bulkheads are constructed in mine openings where the water/slurry would flow to (and flow out of) in the event of a breakthrough. For such an application, designers need to consider the rapid build up of air pressure and subsequent impact of water and debris against the secondary structure. Furthermore, the impounding water or slurry in a mine creates the potential for a blowout of an outcrop barrier or bulkhead that should be evaluated. The design of bulkhead seals is presented in Section 8.5.2.

8.5.1.5 Construction of Compacted Earthen Barriers on the Surface

Construction of a compacted earth-fill barrier around the bottom or perimeter of the impoundment area is a design measure that has been used to mitigate potential breakthrough conditions. A compacted earth-fill barrier provides additional bulk between the impoundment and the mine workings and, in combination with properly designed internal drainage, can lower the water pressure against an outcrop barrier. The water pressure can be reduced if internal drains are used to draw down the hydrostatic level in the fill and also to provide an outlet so that seepage discharges in a controlled manner. Compacted earth-fill barriers can be placed and raised as the impoundment level rises. The design of a compacted earth-fill barrier should take into account the potential effect of subsidence, including sinkholes resulting from underlying mine workings. Measures that have been incorporated into barriers to resist potential subsidence movement, cracking and internal erosion include geogrid reinforcement and graded filters.

For designing a compacted earth-fill barrier as a breakthrough prevention measure, one approach is to conduct reconnaissance and excavation of the coal seam in the vicinity of the reservoir that is known or suspected to be constructed over mine voids. The concept is to identify and intercept any mine voids, and thereby expose known or suspected workings near the outcrop. Weathered or fractured strata near the coal seam outcrop should be removed as part of the excavation, thus allowing the barrier to be founded on competent rock. Surface mining of suspect areas is a comprehensive approach for addressing large areas with unknown conditions. The stability of highwalls should also be evaluated, particularly when workings are encountered during construction.

Another approach is to construct the earth-fill barrier over the existing coal seam outcrop without excavation of the seam. The support and internal stability of the earth-fill barrier due to the presence of mine voids should be evaluated. For this approach, there may be exposed highwall if the coal seam has previously been surface-mined, augered or highwall-mined, or there may be an undisturbed outcrop that has not been affected by surface mining.

The construction of compacted earth-fill and coarse coal refuse barriers should be based upon the same concepts regarding material selection and engineering properties, foundation preparation, provisions for internal drainage, slope stability and development of construction specifications used for the design and construction of earth-fill dams and refuse embankments. Where feasible, fine coal refuse slurry should be deposited around these barriers, as the development of a delta of fine coal refuse above the normal pool level will provide additional resistance to breakthrough.

8.5.1.6 Conversion to Slurry Cells

As described in Section 3.4, slurry cells can be used to dispose of fine coal refuse. One approach that can be taken to reduce the breakthrough hazard is to convert from a full slurry impoundment into a slurry cell configuration using compacted coarse refuse to construct small, individual cells or a series of cells over previously placed fines. Some cell designs have reached depths of 12 feet (8 feet of fine refuse covered by 4 feet of compacted coarse refuse). The depth and number of active cells (i.e., cells that are not capped with backfill) at any given time is usually determined by the volume of storage and cell configuration that will result in a low-hazard potential classification for the facility, as described in Section 3.4. The benefit is that the coarse refuse dikes and covering layers combined with the thin layers of fines allow the fines to dewater and consolidate and make the total mass less flowable. Additionally, with the fines compartmentalized, a problem at one cell location is less likely to affect the entire facility or result in a catastrophic event.

The downstream containment structure (i.e., structural zone) for slurry cells is designed in the same manner as a dam embankment with appropriate width, slopes, benches, internal drainage system, and embankment-material strengths needed to achieve suitable safety factors for slope stability. Slurry cells are most efficient when the depth of fines in the cells is kept relatively shallow, thus promoting drainage from the fines before and during covering.

Some disadvantages of slurry cells include:

- Frequent construction of diversion ditches, new cells, and cell spillways is required as the site elevation increases.
- There is limited flexibility in that a relatively large ratio of coarse refuse to fine refuse is required to keep cell construction ahead of slurry placement, and the fine refuse must settle quickly with limited clarified water retention.
- Close planning and supervision of the site is required so that the construction, filling, and backfilling of cells is accomplished in the proper sequence to make the system function as intended.
- Slurry cell operations are not generally compatible with high production rates at some coal preparation plants.
- Slurry cell operations are not generally compatible with sites that employ conveyor belt/dozer push disposal techniques.
- If a slurry cell site is large enough to be classified as having high-hazard potential, the facility spillways must be designed to handle the runoff from the associated design storm (e.g., PMF).

8.5.1.7 Sealing Sources of Leakage

Discontinuities such as open joints or cracks in a refuse embankment foundation or impoundment area may be treated by grouting or other measures to prevent them from transmitting high hydrostatic pressures and to eliminate potential paths for internal erosion. As a secondary protection measure against leakage, fine coal waste (slurry) can be deposited to form a delta that provides an additional layer of material between the impoundment and potential seepage problem areas. This technique has been used successfully to reduce leakage through embankment and abutment areas and around the perimeter of the impoundment. The long-term benefit of this secondary measure is limited to situations where the bedding planes or joints are relatively small and the sand-sized portion of the slurry is sufficient to allow the gradual formation of a natural filter in joint openings and the eventual sealing of the openings with the clay- and silt-sized particles. If a seepage problem develops after an impoundment is constructed and the technique of distributing the slurry upstream of the seepage area does not correct the problem, then grouting, construction of an impervious barrier, or other measures may need to be taken.

8.5.1.8 Stabilization of Fines

The potential for breakthrough of the contents of a slurry impoundment into underground mine workings can be significantly lessened through stabilization of the fine refuse. Stabilization alternatives include providing drainage measures or treating the fines with additives to increase their strength and/or reduce their water content.

Stabilization of coal-waste fines has sometimes been accomplished by the addition of portland cement or lime-based products. Since the late 1970s and continuing more recently (Fiscor, 2002), studies for development and evaluation of the performance of stabilizer additives have been undertaken. Laboratory studies performed in the late 1970s and early 1980s showed that mixing fine coal refuse with lime-based products may cause the material to appear to have increased strength and stiffness in comparison to untreated material. After samples were mixed and cured, the treated refuse exhibited a relatively stiff load-deformation behavior during initial loading, but then when loading exceeded the apparent maximum past consolidation pressure, the "stabilized" refuse collapsed and the load-deformation characteristics returned to the previous "unstabilized" behavior. In addition, these laboratory studies showed that 5 percent or more by weight of the stabilizers were needed, which made the additives prohibitively expensive except for small treatment volumes. While not all stabilizer additives result in similar behavior, they should always be carefully investigated and used with caution.

Shallow or deep soil mixing methodology using fly ash and cement grout are also methods for stabilizing fine deposits that would be effective for mitigating breakthrough potential. Although not for the purpose of breakthrough mitigation, a portion of a slurry impoundment in Pennsylvania was stabilized in place by shallow and deep mixing with fly ash and cement grout (Bazan-Arias et al., 2002), as also discussed in Section 6.6.3.3. The stabilized fines were then used as the foundation for a highway embankment that crossed the upstream end of the impoundment.

For an existing impoundment, an approach that might be taken is to show that the settled fines have consolidated and gained sufficient strength that they will not flow. Whether settled fines will flow depends on factors such as their degree of consolidation and cohesive strength, pore pressures, the potential for excess pore-water pressures to be induced, and the size of opening associated with a potential breakthrough feature. One qualitative measure of flowability is the moisture content of the fines as compared to their liquid limit (LL) and plastic limit (PL). If the moisture content of fines near the bottom of an impoundment is below the LL and close to the PL, they are less likely to flow and progress to a breakthrough unless the openings in the overburden above or adjacent to the mine are sufficiently large to affect more flowable fines at shallower depths in the impoundment.

A change in conditions in an impoundment, such as inflow of runoff from a large storm, could result in the liquefaction of some portions of the fines leading to an unplanned release of slurry and water through underlying, connected mine workings. Additionally, if subsidence occurs beneath saturated, hydraulically placed fines, the sudden increase in shear stress in the fines may increase pore-water pressure, triggering static liquefaction and causing the fines to flow (Davies et al., 2002). While there may be mitigating conditions in such a situation, measures to drain the fines can be helpful in reducing or eliminating the consequences of a breakthrough because partially saturated fine refuse is likely to densify under load and thus would be more resistant to liquefaction.

8.5.1.9 Overexcavation and Induced Subsidence

Overexcavation of the overburden and mine workings around impoundment perimeters and beneath embankment structural zones may be feasible, particularly in areas of thin overburden such as along outcrops. When applied, this approach includes removal of sufficient overburden to mitigate concern for sinkhole subsidence over the remaining workings by creating a highwall. Barriers must then be designed for any exposed entries found at the base of the highwall. Additionally, highwall stability will need to be addressed as part of the overexcavation plan.

Inducing subsidence through controlled blasting may also be a feasible approach. Guidelines for evaluating the technical and cost feasibility of using controlled blasting to induce subsidence are provided by Workman and Satchwell (1987). In areas where old, unmapped, or poorly mapped mine workings make evaluation of mine breakthrough potential difficult, collapsing the workings by controlled blasting may be an option. Controlled blasting can prevent future mine subsidence from inducing cracks or sinkholes or from compromising planned mine seepage barriers. However, not all rock overburden strata are suitable for this approach; implementation requires: (1) a geologic investigation for determining the character of the strata, (2) a test area for initial demonstration, and (3) monitoring programs for evaluating performance and addressing safety issues associated with blasting and impacts to adjacent areas. Obviously, this approach should be evaluated with considerable diligence and should only be implemented under the guidance of persons with expertise and experience in dam safety, explosives, and the blasting characteristics of local rock strata.

8.5.1.10 Monitoring Provisions

Whenever there is potential for subsidence to affect a dam or for a breakthrough, critical parameters should be identified and a monitoring program should be implemented to determine whether the dam and overburden is performing/behaving as anticipated. Typical monitoring might include instrumentation for measuring reservoir and piezometric levels, discharge rates from mine workings and drains, seepage quantity and quality, water levels in the mine, ground movement, and weather conditions. Use of such instrumentation is discussed in Sections 13.2.1 and 13.2.2. Acceptable ranges and warning or action levels should be established for all monitored values. Monitoring programs should include requirements for plotting and evaluating data in a timely manner and review by an engineer familiar with the design of the facility. Such practice permits detection of trends and correlations to other data.

8.5.2 Mine Entry Bulkhead Seal Design

Bulkheads, also known as hydraulic seals, may be installed across underground mine entries or in mine openings at coal seam outcrops. Normally, bulkheads are installed across underground mine entries to control groundwater in abandoned workings, to prevent rapid inundation of active mine areas in the event of a breakthrough, or to serve as a retention dam in an underground disposal system for fine coal waste. At coal seam outcrops, bulkheads have typically been used to prevent access and to control acid mine drainage. Bulkheads have also been installed at outcrop openings within the footprint of a refuse embankment or within the refuse impoundment to prevent stored water or slurry from flowing into active or inactive underground mine workings. Also, when coal barriers are found to be inadequate, bulkheads may be added on the mine side of the barrier to enhance safety. Design considerations include: site preparation, seal type selection, design load assessment, structural resistance of the bulkhead and surrounding strata, and safety monitoring. Additional guidance is presented in *Guidelines for Permitting, Construction and Monitoring of Retention Bulkheads in Underground Mines* by Harteis, et al. (2008)

8.5.2.1 Site Preparation

As problems with bulkheads are often associated with seepage along the bulkhead and rock interface or through the surrounding strata, the nature of the rock strata at a bulkhead location is important. To the extent practical, bulkheads should be located in the most competent and least fractured area available (normally away from pillar corners), so that problems are minimized. The information gathered as part of the evaluation of a potential bulkhead location should include the type and strength of the rock in the roof and floor and the strength of the coal at the location under consideration. Data related to roof falls, pillar punching, floor heave, or other unusual conditions should be reviewed and evaluated. Coal pillars adjacent to the bulkheads should have a high factor of safety against failure, taking into account all loading factors including transfer stress due to overlying or underlying mining and stresses imposed by the dam and impoundment. Another consideration is that once a bulkhead is constructed, active mining in the vicinity, whether in the same seam or in other seams, can affect the surrounding strata and the load on the bulkhead itself.

As part of the evaluation, fractures or joints in the surrounding rock should be located and characterized. Geophysical techniques can be employed to supplement visual observation for locating rock discontinuities. The potential for subsidence and sinkhole formation over entries should also be assessed. If joints, fractures, or subsidence cracks are present, then chemical or cement-based grouting measures may be necessary in order to minimize seepage pathways that could lead to piping problems or deterioration of the rock strata. To maintain the integrity of a bulkhead, it may be necessary to construct a grout curtain around the perimeter of the planned location. In addition to grouting, loose, cracked, and weak floor, roof, and rib material should be removed. Regardless of the bulkhead type, if the surrounding material consists of weathered or soft rock, then piping (internal erosion) and hydro-fracturing of this material should be considered as potential failure mechanisms.

In-mine bulkheads have failed because of softening of the floor material. These failures occur with a "rooted" claystone referred to as fireclay. When subjected to water, this material breaks down from a rock-like material to a soft soil. The presence of water causes the electrostatic charge on the component clay particles to break down. This process results in dispersion of the clay particles, loss of cohesive strength, and potential erosion from seeping water. For this reason, claystone and fireclay floors should be removed from beneath the planned footprint of a bulkhead by trenching to a depth sufficient to expose hard, competent rock. Trenching should minimize the potential for a floor failure directly beneath a bulkhead. Floor trenches should be backfilled with a seepage-resistant material such as a cement-based or polyurethane-foam product. The surface of fireclay running under the pillars should be sealed and the bulkhead should be constructed as soon as possible after the floor is exposed in order to limit the tendency for the fireclay to swell or absorb moisture from the air.

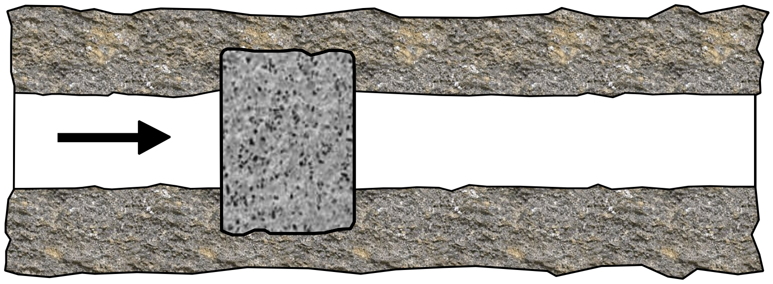
Shales in mine roofs, floors, and partings tend to be pyritic and thus prone to swelling. Bulkheads in shale should be designed to accommodate additional load due to swelling and should also accommodate deterioration of the shale over time due to weathering and moisture effects. High-clay-content shales can break down in the same manner as fireclays, and similar measures to those discussed above can be used to mitigate potential problems.

The integrity of a bulkhead system can be affected by hydraulic fracturing. This type of fracturing can occur when pressure from seeping water is sufficiently great to cause cracks in rock strata to widen and grow. When coal is mined to create an entry, the associated stress relief can result in stress fractures or the opening of natural discontinuities in the mine roof, ribs, and floor. Specifically, the floor may heave upward in some mines. This opening of the rock strata can lead to additional hydraulic fracturing if seepage pressures elevate. Strata grouting and notching of bulkheads into competent material are methods for mitigating potential bulkhead damage due to hydraulic fracturing.

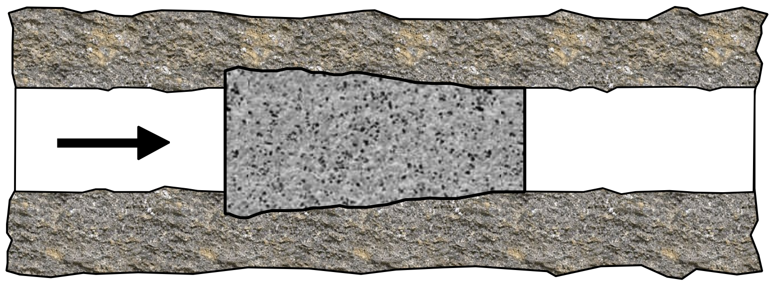
Prior to construction, supplemental roof support should be provided near the interior (inby) and exterior (outby) sides of the bulkhead location. Roof support alternatives include roof jacks and cribbing on both sides of the bulkhead. Unlike typical mine supports, these supplemental roof supports are considered permanent installations. Therefore, corrosion protection should be incorporated into all steel supports, and concrete or other durable cribbing material should be used. Soft floor materials should be removed before installing supplemental roof supports. Notching of the bulkhead into the surrounding rock strata is also recommended. A notch will create a longer flow path for seepage and will increase the resistance to a contact failure between the bulkhead and the rock strata. In most cases, notching will place the bulkhead in contact with more competent material as loose material is removed to construct the notch. When roof notching is not feasible, a steel structural-angle member (with corrosion protection) can be bolted to the roof on the outby side of the bulkhead to provide lateral support. Angles with dimensions of 6 inches by 6 inches by ½ inch have been used for this purpose, but the angle should be sized based on the design loads. Seepage resistance can also be increased by treating the surrounding rock strata with a low-hydraulic-conductivity coating such as shotcrete.

8.5.2.2 Bulkhead Types

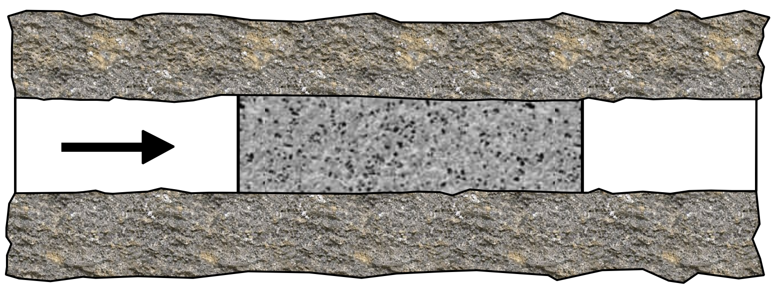
Several types of materials and physical arrangements have been used for bulkhead construction. Figures 8.13 and 8.14 illustrate entry and drift opening bulkhead concepts, including bulkheads



8.13a WALL KEYED INTO RIBS, ROOF AND FLOOR



8.13b TAPERED PLUG



8.13c PARALLEL PLUG

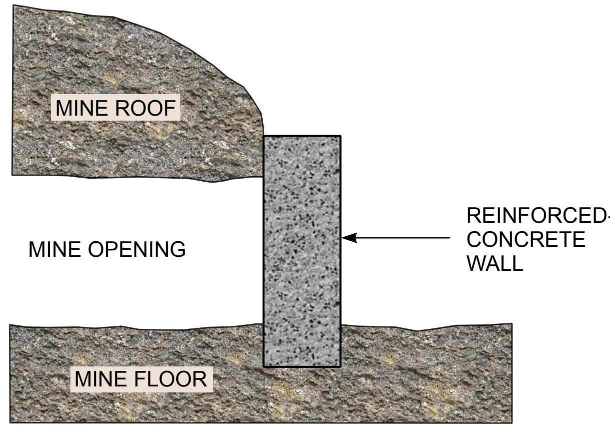
= DIRECTION OF HYDROSTATIC PRESSURE

(HARTEIS AND DOLINAR, 2006)

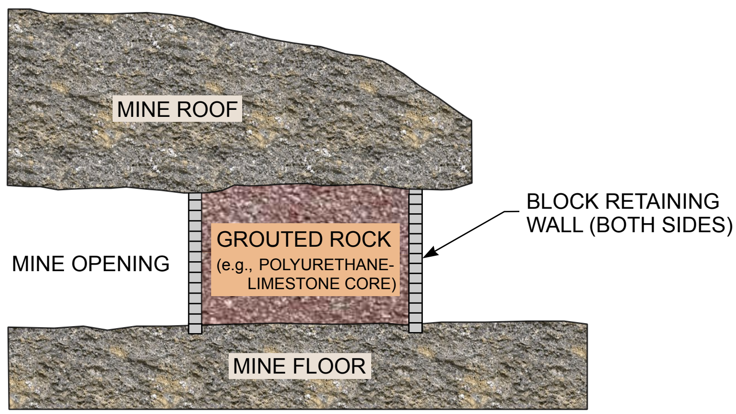
FIGURE 8.13 BASIC BULKHEAD DESIGN CONCEPTS

installed at mine entrances to prevent water from entering. Bulkhead arrangements typically are either thick plugs with a straight or tapered length or thin walls with a straight or arched shape. Each type of construction has advantages and disadvantages. In general, the thicker the bulkhead, the more resistance there will be to seepage and piping around the perimeter.

Kirkwood and Wu (1995) present technical considerations for mine seals to withstand hydraulic heads in underground mines. Solid concrete block walls can be designed to withstand these loads and have been tested under hydraulic pressure of 40 psi (92 feet of head), but it is recommended that the maximum hydraulic pressure be limited to 2.6 psi (6 feet of head) because of long-term effects of strata movement and deterioration surrounding the seal (Chekan, 1985). Reinforced-concrete walls can be designed for significant hydraulic loads, provided it is recognized that the maximum safe head is controlled by the capacity of the surrounding rock. While notching a seal will help increase the total shearing resistance provided by the surrounding rock and increase the seepage path, the bearing



8.14a REINFORCED CONCRETE "CAP" BULKHEAD



8.14b GROUTED BULKHEAD

(HARTEIS AND DOLINAR, 2006)

FIGURE 8.14 DRIFT OPENING BULKHEAD DESIGN CONCEPTS

capacity of the floor and adjacent ribs must be addressed. The difficulty of achieving adequate contact with the roof and dealing with the limited access for working the concrete around the reinforcing can best be handled by using experienced work crews. When constructed as a drift opening bulkhead, a reinforced-concrete wall (Figure 8.14a) can be used to cap off an entry, but contact with the mine roof may require notching, dowels, or other measures to improve the seal and structural integrity.

Thick concrete-plug seals overcome these limitations through development of thicker barriers with significantly more contact with the mine ribs, floor and roof. Typically concrete-block walls serve as forms at either ends of the seal, and concrete or cement-foam materials are used to construct the plug, as shown in Figure 8.15. Grouted-rock seals are similarly constructed, but include rock (gravel to large hand-placed rock) that is grouted in place, as shown in Figure 8.14b. Grout materials vary from cement to rigid polyurethane foam. Grouted-rock seals are generally as thick as or thicker than concrete-plug seals.

When cement-based products are used to construct thick plugs, consideration should be given to the possibility of heat buildup within the fresh concrete mass during curing (Figure 8.15). This buildup

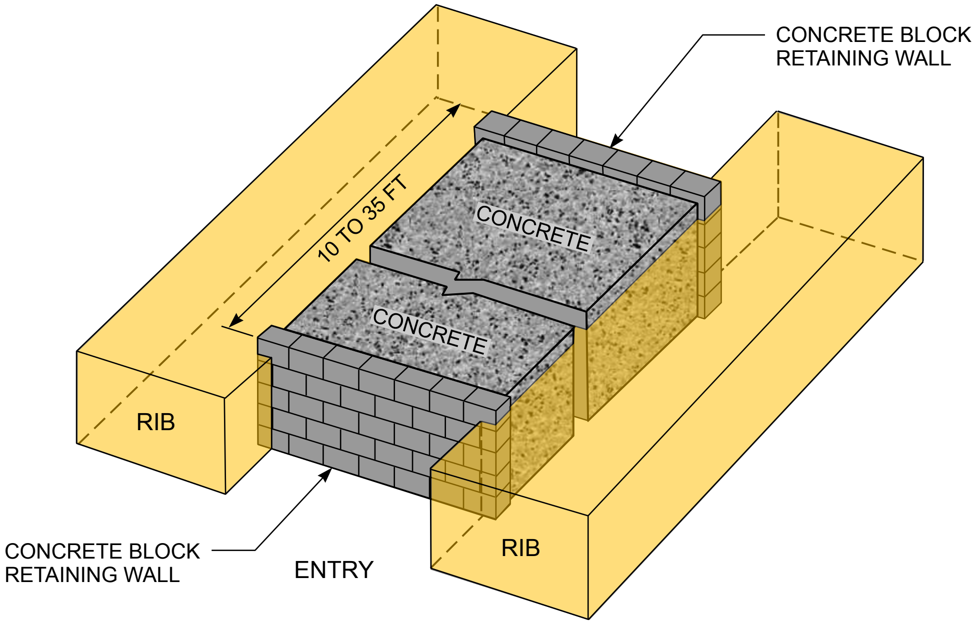


FIGURE 8.15 BULKHEAD CONFIGURATION SHOWING CONCRETE-BLOCK RETAINING WALLS AND CONCRETE CENTER

of temperature is termed the heat of hydration and, if not accounted for, can lead to internal cracking within the plug and to property changes in the surrounding rock strata during the curing period. Measures to minimize heat buildup include using low-heat cements, replacing a portion of the cement with certain pozzolans, using larger aggregate, and cooling the mix water. Guidance can be found in ACI 207.1, *Guide to Mass Concrete* (ACI, 2005a). Polyurethane foam will also generate heat as it cures. The potential for this heating to cause a fire due to contact with combustible material must be taken into account.

Also, polyurethane foam is highly sensitive to moisture that may be present at a bulkhead construction site. Moisture can cause the foam to expand more than intended, resulting in a lower density and lower strength material. The product manufacturer should be consulted regarding measures to prevent moisture from adversely affecting the foam.

Regardless of type of material selected for bulkhead construction, resistance to deterioration from acid mine water should be evaluated. For example, sulfates in groundwater, coal and the surrounding rock strata can cause deterioration or spalling of certain types of cements that are not inherently resistant to sulfate attack. Because of the high-sulfate environment in a coal mine, any cement used in fill should be sulfate resistant. Type V cement is preferable, but in some mine environments, Type II cement will suffice. Depending on cement availability, Type II or Type I cement in combination with pozzolans (e.g., fly ash, slag cement) may be used in order to provide adequate sulfate resistance.

When formwork is used for bulkhead construction, it should be adequately braced for structural support, and vents should be installed at the top of the formwork to release entrapped air and to prevent the formation of voids. Because concrete can shrink during curing, remaining voids between the surrounding rock strata and the bulkhead should be filled by contact grouting.

In selecting the type of bulkhead, consideration should be given to the potential for roof convergence. Some types of materials (e.g., low-density cementitious foams, lightweight concretes, and polyurethane foams) can accommodate this compression by deforming without cracking.

8.5.2.3 Design Loads

Bulkheads should be designed to resist the estimated maximum hydraulic and geologic loads. The hydraulic pressure is affected by the projected head of water or slurry behind the bulkhead. This level may be influenced by such factors as:

- Other mine drift openings
- Shaft or borehole openings
- Flooded overlying mines
- Flooded adjacent mines with inadequate barrier pillars
- Partial height seals located up-dip that act as a weir spillway and divert the water elsewhere in the mine
- Changes in the mine floor slope that divert water into other parts of the mine
- Maximum seasonal groundwater levels or effects of slurry injection
- Maximum design storm water level in an overlying impoundment

The potential water level in overlying flooded mines is a concern if cracks in the interburden provide a hydraulic pathway between mines. There have also been instances where flooded mines located beneath the subject seam dip such that the flood water heads in the seam below are greater than the elevation of the bulkhead in the overlying mine opening or entry. In this case, if cracks are present in the interburden strata, the water level in the underlying mine could control the bulkhead design head. The inlet elevation of drainage pipes extending through up-dip ventilation seals is normally not considered the limiting design water pool level because these pipes can clog. Because of these considerations, an accurate, up-to-date contoured mine map is essential for predicting the design pool level.

In addition to static hydraulic loads, seismic loads may need to be considered if a site is located in a seismically active area. In this event, bulkhead design should account for both the inertial and the hydrodynamic forces that could result from an earthquake. The hydrodynamic forces would result from an increase in static pressure caused by acceleration of the water mass behind the bulkhead.

The roof in a mine entry can exert a compressive load on the top of a bulkhead seal. This compressive force should be determined based on experience with convergence or by estimating the stress that could occur in the zone of material forming the pressure arch over the entry.

A bulkhead should be designed to have the structural capacity to resist the forces acting on it with a factor of safety consistent with the degree of uncertainty associated with the type and magnitude of load and the consequences of failure. A bulkhead must be able to resist the shear and bending stresses caused by the pressures acting on its face. The bending stresses between the roof and floor and between the adjacent ribs can be calculated based on the width and length dimensions and the type of edge restraint (Timoshenko and Woinowsky-Krieger, 1959; Young and Budynas, 2001). For bulkheads that are thick relative to their span, Young and Budynas (2001) provide guidance on stress multipliers. In addition to resisting lateral loads, the bulkhead should have the capacity to resist vertical bearing loads caused by the mine roof convergence and stress transfer. With relatively thin bulkheads, grouting may be necessary in order to prevent excessive seepage from adversely affecting the anchorage of the bulkhead or the surrounding strata.

8.5.2.4 Design of Solid Concrete Block Bulkheads

The analysis of a solid concrete block bulkhead requires estimation of the shear strength around the perimeter of the wall and evaluation of the structural capacity of the wall itself. The structural capacity of the wall is dependent on the bending stresses near the center of the entry. Because a concrete block wall has no steel reinforcement, it has relatively poor flexural strength, and, except in the case of very narrow entry widths, the lack of flexural strength limits the maximum hydraulic head more than the strength of the surrounding strata. Kendorski et al. (1979) provide guidance for block wall design and the estimation of flexural stress. The structural capacity of the concrete block wall will also depend on the strength of the block, the strength of the mortar, joint thickness, and the quality of construction. Masonry design and construction guidance such as ACI 530, *Building Code Requirements for Masonry Structures* (ACI, 2008), should be followed.

8.5.2.5 Design of Concrete Bulkheads

For thin bulkheads installed with an entry width-to-height ratio greater than 2, a reinforced-concrete bulkhead can be designed as a one-way slab spanning between the roof and the floor if adequate edge connections are provided at the mine roof and floor. To account for temperature and shrinkage stresses, reinforcement is required in the rib-to-rib direction if the bulkhead is designed for one-way behavior in the roof-to-floor direction. For an entry width-to-height ratio less than 2, the bulkhead can be designed for two-way behavior provided there is adequate anchorage on all four sides. However, reinforcement should be sized to separately carry the full bending moment if there is concern that either the roof or floor or ribs may not provide adequate resistance. Regardless of the width-to-height ratio, diagonal reinforcement steel should be placed in the bulkhead corners to control cracking from torsional moments. Regardless of the load path direction assumed, if the mine roof, floor, and ribs are notched into the surrounding rock strata or if the steel bar reinforcing mats near the inby and outby faces are doweled into the surrounding strata, then it is possible to develop negative moment bending stresses at the edges of the bulkhead slab. Negative steel (i.e., the reinforcement near the inby "wet side" face) should be sized to resist negative moment bending stresses.

Reinforced-concrete structures should be designed in accordance with the most recent version of ACI 318, *Building Code Requirements for Structural Concrete* (ACI, 2005b) and ACI 350 (where applicable). The codes are based on ultimate strength design in which factors are applied to the loads and strength to account for the uncertainty of these parameters in structural design. Using this approach, a safe design is achieved when the factored design strength of a structure component exceeds the factored loads. For fluid pressure loading, the load factor is 1.4 when the maximum height of the water or slurry is controllable or conservatively estimated. Thick concrete bulkheads with a thickness-to-height ratio greater than 1 should be designed in accordance with Section 8.5.2.5.2.

8.5.2.5.1 Design of Reinforced-Concrete Bulkheads for Flexure

Where the span-to-thickness ratio of the bulkhead is greater than 2, the design flexural strength of a reinforced-concrete member (ACI, 2005b) is determined using the relationship:

$$M_d = \Phi A_s f_y \left[d - \frac{a}{2} \right] \tag{8-6}$$

where:

$$a = \frac{A_s f_y}{0.85\, f'_c\, b} \tag{8-7}$$

M_d = flexural design strength (lb-in)

Φ = strength reduction factor = 0.90 (dimensionless)

A_s = area of tension reinforcement (in^2/ft)

f_y = yield strength of reinforcing steel (psi)

d = distance from extreme compression fiber to centroid of tension reinforcement (in)

a = depth of rectangular stress block at failure (in)

f'_c = specified minimum compressive strength of concrete (psi)

b = width of compression face of member, normally taken as 12 in for slabs (in)

For thick bulkheads with a span-to-thickness ratio less than or equal to two, the flexural design strength from Equation 8-6 should be modified to reflect the behavior of a thick flexural member. Because thicker members have low span to thickness ratios, a linear stress distribution is no longer valid. According to Park and Paulay (1975), for simply supported members with span-to-depth (thickness) ratios less than or equal to two, the internal lever arm can be calculated as:

$$z = 0.2(\ell + 2\text{h}) \quad 1 \leq \ell/h \leq 2 \tag{8-8}$$

$$z = 0.6\,\ell \quad \ell/h \leq 1 \tag{8-9}$$

where:

ℓ = centerline-to-centerline span distance of two bearing points or 1.15 times the clear span, whichever is smaller (in)

h = thickness of bulkhead (in)

z = internal lever arm (in)

Applying the revised lever arm value to the standard flexural equation (Eq. 8-6), the capacity can be estimated using:

$$M_d = \Phi A_s f_y z \tag{8-10}$$

The value of z should not exceed $d - a/2$. If the end supports are assumed to be fixed, rather than simply supported, the value of z should be further adjusted (Park and Paulay, 1975). Nilson et al. (2002) recommend that tension steel in a deep flexural member be distributed over the bottom third of the member depth.

8.5.2.5.2 Design of Unreinforced-Concrete Plug Bulkheads for Shear

Unreinforced-concrete bulkheads typically have a thickness-to-height ratio greater than 1. The shear resistance of these bulkheads may be governed by the:

- Shear strength of the seal material
- Shear strength of the surrounding strata
- Contact interface shear strength between the seal and the strata

In cases where there is no notching of the bulkhead into the surrounding strata, the interface resistance will be governed by adhesion or friction. The South African plug formulas, which are based on the shear strength and bearing capacity of the bulkhead material and surrounding strata, are often used to evaluate the required length of thick bulkheads (Garrett and Campbell Pitt, 1961):

$$\ell = \frac{p\,a\,b}{2(a+b)f_s} \tag{8-11}$$

$$\ell = \frac{p\,a\,b}{(a+b)f_c} \tag{8-12}$$

where:

ℓ	=	length of bulkhead (ft)
p	=	hydraulic pressure on bulkhead (psi)
a	=	width of entry (ft)
b	=	height of entry (ft)
f_s	=	minimum allowable shear strength of strata or plug material, whichever is less (psi)
f_c	=	minimum allowable compressive strength of strata rock or plug material, whichever is less (psi)

The values of f_s and f_c obtained from sampling may not conservatively reflect the strength of the de-stressed edges of the coal pillars. The designer should select the bulkhead length based on the larger of the values obtained from Equations 8-11 and 8-12. These equations are most applicable to high-head situations where the bulkhead acts as a massive plug. If the bulkhead span-to-thickness ratio is greater than one, then flexural reinforcement should be provided, as discussed in Section 8.5.2.5.1.

Selecting shear strengths for coal is problematic, as such values are typically not determined and research in this area is limited. Coal is generally crushed or yielded at ribs and extending some distance into pillars. Some designers use the cohesive strength of the coal if the ribs are scaled and the bulkhead is constructed before substantial additional yielding takes place. If this is not possible, the residual cohesive strength of coal is taken as its shear strength. A common estimate for the cohesive strength for coal is 145 psi, which, depending on the practitioner asked, may have resulted from the calculation of 16 percent of the typical compressive strength (0.16 x 900 psi = 144 psi) or from an assumed minimum value of 1 MPa (145 psi). Sixteen percent is a rule-of-thumb sometimes used to estimate the cohesive strength of rock from the compressive strength. It can be difficult to obtain representative samples of coal for testing, but a method of estimating the shear strength of coal is to conduct direct shear tests with coal samples oriented to model in-mine conditions.

Methods for increasing bulkhead resistance include notching the bulkheads into the surrounding rock strata, tapering the plug, and/or installing corrosion resistant, epoxy-coated dowel rods into the surrounding rock strata and allowing the rods to protrude into the bulkhead. The dowel rods should have an embedded length into the surrounding strata and the bulkhead sufficient to develop the strength of the dowel rod without a bond failure. The minimum required development length in concrete should be determined in accordance with the most recent version of ACI 318, Section 12.2.

Thick concrete bulkheads with a thickness-to-height ratio greater than 1 should be designed in accordance with the most recent version of ACI 207.1. Reinforcement should be provided to control temperature and shrinkage stresses at the surface and should be designed in accordance with the most recent version of ACI 318. Reinforcement is not necessary when the bulkhead thickness is greater than 6 feet, provided that steps are taken to control the effects of temperature and shrinkage. Heat of hydration can be controlled by using low-strength concrete, low-heat cement, or replacing a portion of the cement with pozzolans. It can also be controlled by reducing the placement temperature using cooled mix water or aggregates. Accelerating admixtures should not be used.

8.5.2.5.3 Design of Reinforced-Concrete Bulkheads for Shear

While Equations 8-11 and 8-12 can be used to determine the required length of thick bulkhead plugs in order to prevent failure in shear, Equation 8-13 can be used to calculate the design shear strength of concrete for thinner, reinforced-concrete bulkheads:

$$V_c = 2\,\Phi \sqrt{f'_c}\, b_w\, d \tag{8-13}$$

where:

V_c	=	shear strength of the concrete bulkhead per unit width (lbs)
Φ	=	shear strength reduction factor = 0.75 (dimensionless)
b_w	=	unit width of bulkhead (12 in)
d	=	distance from inby side of bulkhead to centroid of tensile reinforcement (in)

If Equation 8-13 indicates inadequate concrete shear strength, a more rigorous formulation should be obtained from the most recent version of ACI 318, Section 11.3. In addition, the contribution of steel shear reinforcement, as discussed in ACI 318, Section 11.5, can be added to the value obtained for the concrete to obtain a combined shear strength for the bulkhead.

For thick, reinforced-concrete bulkheads where the ratio of the clear span distance ℓ_n to the depth d from the inby side of the bulkhead to the centroid of the tensile steel reinforcement is less than four, the most recent version of ACI 318, Section 11.8 should be applied. It should be noted that the span-to-depth ratios are different for shear design than they are for flexural design.

If lightweight concrete with a density of 100 to 110 pcf is used, values of V_c obtained using $\sqrt{f'_c}$ in the Equation 8-13 should be multiplied by 0.75 for "all-lightweight" concrete and by 0.85 for "sand-lightweight" concrete, or should be in accordance with the most recent version of ACI 318, Section 11.2.

8.5.2.6 Monitoring of Bulkheads

The water pressure behind a bulkhead should be monitored so that it can be compared to the design pressure. Warning levels that would warrant drawing down the mine pool or initiating an emergency action plan should be established. To lower the water pressure, a corrosion-resistant pipe should be installed through the bulkhead with a "U-trap" and a pressure relief valve on the downstream end and with provisions to prevent clogging (such as a riser and trash rack) on the upstream end. Pipes extending through a bulkhead should be equipped with external collars to cut off seepage and minimize the potential for a pipe blowout.

A possible safety measure for regulating the maximum hydraulic pressure is to install an additional pipe through the bulkhead with a corrosion-resistant rupture disk attached to the downstream end. If the hydraulic pressure on the bulkhead reaches the rupture strength of the disk, it will fail and thus limit the load on the bulkhead. The outlet end of the pipe should project downward to prevent injuries to anyone near the pipe if the disk should suddenly rupture. If water could collect and flood workings, inhibiting escape for miners, an evaluation of the consequences of an unexpected water release should be conducted.

Seepage can occur through a bulkhead or the surrounding strata, but the presence of seepage is not necessarily a sign of distress. Since unexpected increases in seepage could indicate deterioration of the bulkhead or adjacent strata, seepage through the bulkhead and the surrounding strata along with the corresponding head behind the bulkhead should be monitored and evaluated. To this end, seepage on the active side of the bulkhead is often channelized and monitored using a weir to facilitate measurement of changes in quantity.

A final safety measure is to establish an inspection schedule for long-term monitoring of the bulkhead and to have a contingency evacuation plan in place for situations where problems are encountered with the integrity of the bulkhead or surrounding strata. To determine the impact of a bulkhead failure on active in-mine escapeways, the potential inundation area should be identified.

8.6 FOUNDATION-RELATED CONSTRUCTION AND OPERATIONS MONITORING

Observations of foundation conditions during construction and operation of impounding facilities are critical. During construction, foundation conditions including mine-related features are exposed, allowing the facility operator and engineer to assess conditions for consistency with expectations and to evaluate specific foundation preparation design measures. The operator and engineer responsible for construction certification should involve the designer during critical tasks such as construction/implementation of bulkheads and seals, cutoff trenches, and grouting programs. During construction, survey control to confirm locations of key features and the dimensions of associated structures is essential. Documentation of conditions with photographs and as-built drawings and reports of construction activity is important for certification of the work and for subsequent evaluation, if necessary. Additional discussion of construction monitoring is presented in Chapter 12. A discussion of construction and operations procedures and monitoring for bulkheads is provided in MSHA (2003), and key guidance from the MSHA document is presented in this section.

Where there are mine workings near an impoundment, the manner in which an impoundment is operated can affect the breakthrough potential. For example, measures should be taken to design and operate the impoundment in a manner that minimizes the presence of free water. Thus, decant raising should be staged so that the water level rises incrementally. Pumping can also be employed to minimize the volume of water in the impoundment. Mine personnel who work on or around the impoundment should be cognizant of key components of the operation plan and particularly of any unusual operational requirements.

A site-specific monitoring plan oriented toward breakthrough prevention and assessment of the potential for subsidence to affect the dam should be developed. Monitoring involves collection of information from both visual inspection of the impoundment and from instrumentation. Coal company personnel who inspect the impoundment, or who routinely work on or around the impoundment, should be trained to observe potential signs of trouble that could be related to subsidence effects or a breakthrough. These personnel should be aware of where underground mining has occurred and where to look for cracks or other evidence of subsidence. Signs that they should look for include: (1) unusual sudden drops in the pool level, (2) the presence of a whirlpool or bubbles in the pool, (3) cracking or sudden displacement in embankment surfaces, (4) unusual readings in piezometers, (5) changes in seepage conditions, and (6) changes in the quantity of flow and the amount of sedimentation in discharges from mine openings, backfilled mine openings, or outcrop areas. Instruments such as staff gages, weirs, survey monuments and inclinometers are useful for monitoring these conditions and detecting changes.

In determining what instrumentation should be installed and monitored, the designer should identify the parameters that will indicate how the site is performing with respect to potential failure modes. This will facilitate taking action if the facility does not perform as expected. The following items should be considered in developing a monitoring plan:

- <u>Seepage</u> – Seepage from the impoundment should be monitored, including seepage through the embankment, through any internal drainage systems, and through underground mines that receive seepage from the impoundment. Weirs or other devices should be installed so that flow rates can be easily and consistently measured. Changes in water quantity and quality in seeps and discharges (including mine/pump discharges) that are hydraulically connected to the impoundment should be monitored. Changes in seepage quantity, particularly when not correlated to rainfall or pool water levels, may indicate deteriorating or adverse conditions warranting additional investigation.

- Water levels – The pool levels in the impoundment and in underground mines should be monitored, and unexplained changes should be investigated. If there are bulkheads in the mine, the water pressure against them should be monitored. Where conditions with respect to breakthrough potential are uncertain, instrumentation can be installed to provide an alarm in the event of a sudden drop in the impoundment water level. The alarm would alert mine personnel to check on the situation and would facilitate early warning and emergency response in the event of a breakthrough failure.
- Piezometric levels – Saturation levels and water pressures in the refuse embankment, as well as in any other earthen barriers, should be monitored to determine whether hydrostatic pressures are within design limits and whether any changes or trends are reasonable. In situations where it is critical to be able to measure rapid or sudden changes in pore water pressure, a closed system such as a vibrating-wire piezometer should be used.
- Rainfall data – It is good practice to install a rain gauge in the vicinity of an impoundment, but it is especially important in situations where there is breakthrough potential and where discharges from a mine can be traced to seepage from the impoundment. In such cases, rainfall data should be routinely collected so that it can be determined whether changes in the water flow or water level data correlate to rainfall or may be occurring for other reasons.
- Ground movement – When there is potential for subsidence in the vicinity of an impoundment, the ground surface should be monitored for movement. Both horizontal and vertical movements should be measured.

The types and suitability of instrumentation for accomplishing these monitoring objectives are discussed in Chapter 13. The timely review and interpretation of instrumentation data by someone knowledgeable in the design and performance of impoundments is critical to an effective monitoring program.

The type and frequency of monitoring required depend on impoundment conditions. Typically, monitoring is performed during regular weekly inspections. Where conditions warrant, more frequent or even continuous automated monitoring may be needed. Monitoring plans should include provisions for plotting and evaluating the observed data in a timely manner. When a potentially hazardous condition develops, more frequent monitoring is required. 30 CFR § 77.216-3(b)(4) requires, in part, that when a potentially hazardous condition develops, the mine operator shall immediately direct a qualified person to monitor all instruments and examine the structure at least once every eight hours or more often, if requested by an authorized representative of MSHA.

8.7 MINE BACKFILLING DESIGN

Backfilling of mine workings is performed in order to provide localized support for pillars and the mine roof and to reduce the volume of open space that could potentially be filled with collapsed material, thus tending to minimize the deformation of the surrounding rock mass. Support for pillars and volume reduction of open space can be achieved by a variety of backfill types. While deformation and bulking of the roof strata can provide support for mine overburden, backfilling with grout to improve contact with the roof may be a desirable option. The most common types of fill material are waste rock, mill tailings, quarried rock, sand and gravel, and fly and bottom ash. Additives such as portland cement, lime, fly ash, and pastes can also be mixed with the fill to alter the characteristics and to improve the effectiveness of the backfill. Placement alternatives include stowing by hand, gravity, mechanical, pneumatic and hydraulic methods. Hydraulic placement is generally effective for the varying conditions encountered in mine workings.

Backfilling of mines has significantly reduced surface damage from subsidence by lending lateral support to pillars and by limiting the volume of voids (National Academy of Sciences, 1975). The most important consideration in mine backfill design is the planned backfill material. The mechanics of uncemented fill can be analyzed using soil mechanics principles (Coates, 1981). The strength of most uncemented hydraulic fills is related to frictional resistance to sliding between particles and can be affected by pore water pressure and erosion, as well as dynamic loads such as blasts or sudden fluctuations in the water table. Additionally, the compressibility of the backfill material is related to its ability to provide support for the pillars and roof.

Cemented backfill has cohesion resistance gained through the addition of cement or pozzolanic admixtures that render the backfill relatively incompressible. Backfill strength usually increases linearly with cement content, which can range up to 10 percent. The grain-size distribution of the fill may also be important. Well-graded material generally has a greater strength than uniformly graded material, although the fines content (i.e., minus 200 mesh portion) can adversely affect the strength. The inclusion of pozzolanic material additives such as fly ash can reduce the volume and associated cost of portland cement in a mine backfill while significantly increasing strength and providing other beneficial characteristics such as improved fluidity during placement. The strength and placeability of candidate cement mixes should be evaluated through laboratory testing.

Typically, mine backfill provides substantial filling of mine voids, such that relatively little bulking from roof materials is needed to mitigate fracturing of the overburden and the advance of subsidence in the overburden. In critical situations, the space between the backfill and the roof can be grouted. In instances where mine backfill provides support for the coal pillars but not direct support for the overburden, the mechanical behavior of the cemented fill can be modeled as providing lateral restraint according to the relationship (Cai, 1983):

$$\sigma_h = n \frac{\gamma H a}{K_p L} \tag{8-14}$$

where:

σ_h	=	passive earth pressure (psf)
n	=	correction coefficient (dimensionless)
γ	=	unit weight of overburden (pcf)
H	=	depth of overburden (ft)
a	=	width of open space (ft)
K_{pl}	=	coefficient of passive earth pressure of pillar (dimensionless)
L	=	width of adjacent pillar (ft)

With lateral support from cemented backfill, the strength of a pillar increases according to the following formula (Guang-Xu and Mao-Yuan, 1983):

$$\sigma'_1 = \sigma_1 + \sigma_h K_{pl} \tag{8-15}$$

where:

σ'_1	=	strength of pillar supported by fill (psf)
σ_1	=	strength of pillar (psf)
σ_h	=	passive earth pressure provided by fill (psf)

The supported pillars can then be analyzed for stability based on the procedures discussed in Section 8.4.3. For instances of thin overburden, Mitchell (1983) and Wizniak and Mitchell (1987) present analytical procedures for estimating surface subsidence deformation for backfill placed to the mine roof wherein the roof is modeled as a beam on an elastic foundation.

The design of mine backfill generally depends on the availability of fill materials and fly ash and their associated costs.

8.8 SURFACE MINE SPOIL ISSUES

At some refuse disposal sites, surface mine spoil may be available in the foundation area, and some of this spoil may be suitable for use in constructing structural portions of refuse embankments such as the starter dam. Mine spoil is typically quite variable in terms of its composition of soil and rock materials and also relative to maximum particle or fragment size and distribution, durability and moisture content. Typical methods of spoil placement can also result in considerable segregation. This variability makes it difficult to characterize the engineering behavior of mine spoil, and thus the field and laboratory procedures described in Sections 6.4 and 6.5 may need to be modified to accommodate its special characteristics. This variability also can lead to difficulties in estimating engineering properties such as compressibility, shear strength and hydraulic conductivity. Piping in mine spoil materials is a concern and has been identified as the cause of sinkholes and black-water releases at slurry impoundment sites founded on mine spoil. When a coal refuse facility is to be constructed over or using surface mine spoil, designers should recognize the variable character and composition of the material and understand the potential impacts that it can have on the long-term performance of a coal refuse embankment.

8.8.1 Surface Mine Spoil Characteristics

Surface mine spoils result from excavation and placement of overburden and interburden materials. These operations typically range in size from several hundred to several thousand acres, and mine life is typically five to 30 years or more. Overburden removal is generally accomplished by continuous bucket-wheel excavators, walking draglines, hydraulic excavators, stripping shovels, scrapers, dozers, or cast blasting. Where unconsolidated materials such as glacial till or loess represent a significant portion of the overburden, bucket-wheel excavators are generally used.

Spoil placement is typically accomplished by dropping the spoil materials at the angle of repose to form a ridge of piles parallel to the active pit or by placing the materials in lifts where the lift thickness and degree of compaction depend on the equipment type, weight and number of passes.

Uncontrolled spoil placement is typically associated with contour and area mining, while placement in lifts is generally associated with haul-back mining and head-of-hollow and valley-fill construction. Before the late 1970s to early 1980s, spoil ridgetops were left as deposited or were graded with a single pass of a dozer that resulted in a ridge and trough topography. With the advent of state and federal regulatory programs to return mining operations to near their original contours, spoil ridges have been extensively graded with dozers leading to increased spoil handling and machinery traffic, greater breakdown of the spoil, higher density and improved engineering behavior of the in-place spoil. A discussion of the influence of mining method on the bulk density of spoil based on testing in the Eastern Coal Province is presented in Phelps et al. (1981).

Because of the material handling that occurs during excavation, deposition and placement, mine spoil materials experience significant changes in physical integrity. These changes are related to variations in geologic characteristics, moisture, stress regimes, mining and reclamation methods, and other environmental aspects of the materials. The physical deterioration of geologic materials caused by changes in stress conditions or strength characteristics is referred to as slaking. The most distinctive aspect of the slaking process is a relatively rapid decrease in the particle or fragment size of the material.

The decrease in particle or fragment size caused by slaking can have a wide range of effects on the behavior of the material in a spoil pile. These effects will depend on the gross characteristics of the spoil pile (e.g., physical dimensions and configuration of the pile and the degree of compaction), the durability of the pile materials, the proportion of slakable materials, and changes in the pile surficial or internal moisture regimes. Possible adverse effects of spoil slaking include: (1) decreases in material strength that can reduce the stability of slopes and (2) increases in moisture content and decreases in particle and fragment size that can increase settlement and surface erosion and affect hydrologic regimes and vegetation. In a study of the environmental effects of slaking of surface mine spoils in the eastern and central U.S., Andrews et al. (1980) observed that:

- The rate and degree of particle breakdown is directly related to the material characteristics (e.g., durability) and local environmental conditions (e.g., depth of burial).
- The most active zone of slaking occurs within about three feet of the exposed mine spoil surface.
- The major observed effect of mine spoil slaking is a decrease in particle or fragment size that results in changes to the hydrogeologic characteristics (e.g., rate of infiltration, hydraulic conductivity, rate of groundwater flow) of spoil piles
- The significance of slaking seems to be minimized by the mixing of slakable (e.g., mudstone and shale) and nonslakable (e.g., limestone) materials that usually occurs during typical spoiling operations.
- No gross environmental damages due to slaking were apparent at the sites visited.

Andrews et al. (1980) developed these observations through laboratory testing of bulk samples obtained from test pits excavated in 2-, 5-, and 10-year-old spoils from four mine sites located in the Appalachian Basin. Based on extensive qualitative and semi-quantitative data collected relative to the behavior, causes, and effect of slaking materials, the study identified three field slaking modes:

1. Slaking to a constituent grain size that typically occurs in mudstones and occasionally sandstones
2. Chip slaking to thin, platy fragments that generally occurs in shales, siltstones and occasionally thinly-bedded sandstones
3. Slab or block slaking to large, approximately equidimensional fragments that generally occurs in sandstones and limestones.

The lithology, bedding and mineralogical characteristics of the spoil materials were found to have a major effect on the mode, rate and degree of slaking. In general, mudstones, siltstones and shales were found to be the most slake-prone lithologies. Slaking of sandstone and limestone was variable, but generally minor. Bedding characteristics were the primary factor in the mode of slaking (i.e., rocks with thin bedding typically exhibited chip slaking whereas rock with a massive structure were prone to block or slab slaking, or to slake to their constituent particle size). Spoils that slake to their constituent particle size were found to be less stable, while spoils in which chip slaking or slab or block slaking dominates generally were found not to be associated with stability issues.

The engineering behavior of mine spoil may be related to mining techniques. In general, overburden removal by blasting, material transport and dumping by trucks, and placement and leveling by dozers results in mechanical breakdown that leads to reduced slaking once the material is placed.

8.8.2 Design and Construction Considerations

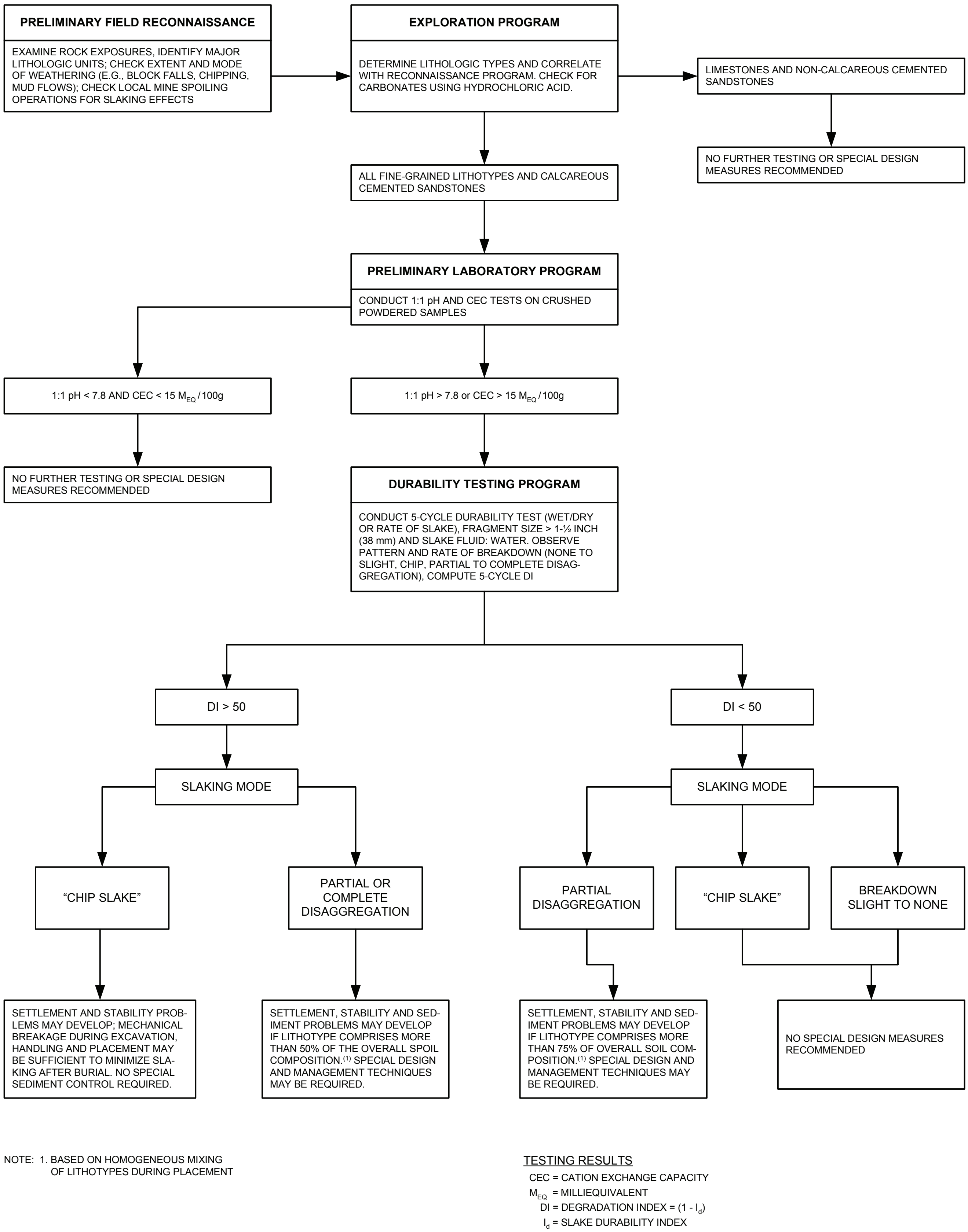
The compressibility, shear strength and hydraulic conductivity of mine spoil can vary considerably depending on the proportion of slakable and nonslakable materials and the methods used to place and reclaim the spoil. Test methods that can be used to assess the slaking potential of mine spoil are presented in Section 6.5.9.4. The application of these test methods as part of an overall management control process are described in Andrews et al. (1980). Figure 8.16 presents a system of classification and interpretation of spoil durability based on index and slake durability testing using the degradation index (DI), where $DI = 1 - I_d$ and I_d is the slake durability index. Based on conclusions and recommendations from the study, designers should consider the following measures in planning and conducting site reconnaissance, site exploration and laboratory testing programs and in developing designs and preparing construction documents:

- Review of available surface mining records in order to document the mining and reclamation practices used, the stratigraphy of overburden and interburden materials excavated, and the time frame for mining operations.
- Use of test pits to observe and document the general proportion of slakable and nonslakable materials, the type and amount of slakable material breakdown, the grain-size distribution and moisture content of bulk samples, and the existence of a permanent or perched water level in the spoil mass.
- Provisions to control seepage and prevent internal erosion through mine spoil zones under refuse facility structures, including cutoff trenches through spoil deposits or impervious blankets with filter layers.
- Measures to monitor the settlement and lateral displacement of foundation spoil materials as the refuse embankment is raised. A discussion of instrumentation is provided in Chapter 13.

If mine spoil is to be used for embankment fill, the variability of the spoil should be determined so that the embankment can be constructed in a manner that will allow the desired performance in terms of strength and hydraulic conductivity to be achieved. The strength of mine spoil is typically estimated by: (1) evaluating its angle of repose, (2) correlating with published values for rock fill if the material is durable, or (3) by testing the finer portion of the material, which typically results in a conservative estimation. Internal stability of non-uniform material can be a concern for mine spoil, particularly if it is gap-graded. The hydraulic conductivity of mine spoil is typically estimated based on grain size, and measures such as zoning may be required in order to address the variability of the material. Designers should consider the following measures, if mine spoil is to be used to construct some or all of a coal refuse embankment:

- The composition and variability of the mine spoil used for borrow should be determined using test pits to observe and document the general proportion of slakable and nonslakable materials, the type and amount of slakable material breakdown, the grain-size distribution and moisture content of bulk samples, and the chemical composition (i.e., pH and content of sulfide minerals in fine-grained constituents such as shale) of the spoil mass. Placement of slakable material in upstream zones (particularly when fine-grained) and nonslakable material in downstream zones in an embankment can provide a means of isolating the materials and taking advantage of their properties.
- If oversize material (e.g., greater than 12 inches in maximum dimension) is present, provisions to crush, mechanically degrade or isolate oversize materials should be incorporated into the construction specifications.
- If the chemical composition of any portion of the spoil mass is not acceptable, these materials should be isolated or excluded from the embankment.

The presence of mine spoil in the foundation zone of an impounding embankment may require measures to address settlement and the potential for internal erosion. The significance of such effects should be evaluated based on exploration, testing and analysis of the foundation and embankment



(ANDREWS ET AL., 1980)

FIGURE 8.16 CLASSIFICATION AND INTERPRETATION OF SPOIL DURABILITY materials. Typical measures that are employed include: (1) compensating for settlements through staged construction by loading foundation zones using broad embankment widths and (2) providing sufficient gradient on drainage structures. The potential for internal erosion can be addressed by a variety of site preparation measures including the use of cutoff trenches with seepage barriers and filters.

8.9 SURFACE MINE HIGHWALL ISSUES

Surface mine highwalls represent foundation concerns for coal refuse disposal facilities, in that the change in rock surface elevation may represent an abrupt transition in an embankment abutment with the potential for: (1) embankment cracking due to differential settlement or (2) a zone of concentrated seepage due to the difficulty of placement and compaction of fill materials close to a near-vertical highwall. Additionally, surface mine benches and highwalls may contain fractured materials, particularly if they have been subjected to auger or highwall mining activities. Subsidence associated with such mine openings and remedial measures are discussed in Sections 8.4.2.8 and 8.5.1.3. The discussion in this section focuses on abrupt rock transitions and associated seepage cutoffs.

Whenever a coal refuse embankment or earthen dam abuts a surface mine bench or highwall, there is a potential for performance problems due to the abrupt transition where the rock bench/highwall adjoins the earthen embankment. These potential performance problems include:

- Inadequate compaction of embankment materials against the rock surface sufficient to provide a low-hydraulic-conductivity contact between the earthfill and rock surfaces.
- Incomplete filling of surface cavities and depressions in rock surfaces with compacted embankment materials that can be sources of differential settlement and seepage.
- Overly steep rock surfaces and overhangs that limit the opportunity for maintaining positive compressive pressure over the full contact between earthfill and rock surfaces.
- Earthfill placed over rock surfaces with steep or abrupt slope transitions that can lead to differential settlement, cracking and elevated seepage in the earthfill.
- Highly fractured highwalls or benches resulting from mining or subsidence that represent potential seepage pathways that could impact downstream embankment and abutment areas.

To minimize these potential performance problems, rock benches and highwalls should be inspected during construction and should be treated or modified so that unacceptable conditions and performance problems do not occur. For earth embankment dams with an impervious core, these problems are of concern mostly where the impervious core abuts the rock surface. For coal refuse or other embankments that are constructed as homogeneous dams, the concern may be more significant because these structures do not have an impervious zone to serve as a barrier to internal seepage or seepage along abutment contacts.

If the control measures described in the following list cannot be fully implemented in a constructed embankment, special drainage features for intercepting and conveying seepage flows from the embankment should be considered. Fell et al. (2005), USBR (1984) and Sherard et al. (1963) describe a number of treatment and modification options for such performance problems, which may be encountered at water retention dams. Additionally, alternate measures are described that may be appropriate for coal refuse impoundments, but any such measure should be evaluated with respect to site-specific conditions. Guidance for addressing a number of conditions is provided in the following:

- <u>Loose and weathered materials</u> – The existing overburden and weathered rock should be excavated to competent rock, and the resulting rock slope should be trimmed to a regular surface to eliminate depressions, overhangs, pinnacles or sharp transitions. This treatment should focus on the abutment cutoff zone, as illustrated in Figure 8.17, and should extend as warranted by site conditions and the embankment cross section. Sherard et al. (1963) noted that the most effective way to obtain a tight bond between earthfill and a rock surface is to slope the rock surface sufficiently to permit each embankment layer to be compacted directly against the rock using heavy compactors. Equipment operation above or below a highwall should be preceded by an evaluation of the highwall stability and development of appropriate procedures, if necessary, to address the threat of rockfalls or instability during grading and excavation.

- Overly steep rock slope – If the slope of a rock surface that will abut an earthfill is steeper than 0.5H:1V, the rock surface should be flattened by excavation or backfilling with concrete, or other measures should be taken to effectively place and compact the embankment materials. Alternate measures could include incorporation of impervious zones, incorporation of internal drainage features, or broadening of the embankment. The same cautions discussed above relative to working near a highwall apply.
- Hydraulic cutoff – If flattening is not practical due to the height of the rock slope, a cutoff keyway can be excavated into rock, or other measures to control seepage along the rock interface may be implemented. The depth of a cutoff keyway should extend a sufficient depth (e.g., six feet as per Fell et al., 2005) into competent rock, and the width of the keyway should be sufficient to permit compaction of earthfill in the cutoff in accordance with embankment criteria. Alternate measures to mitigate seepage could include incorporation of internal drainage features or broadening of the embankment-abutment contact.

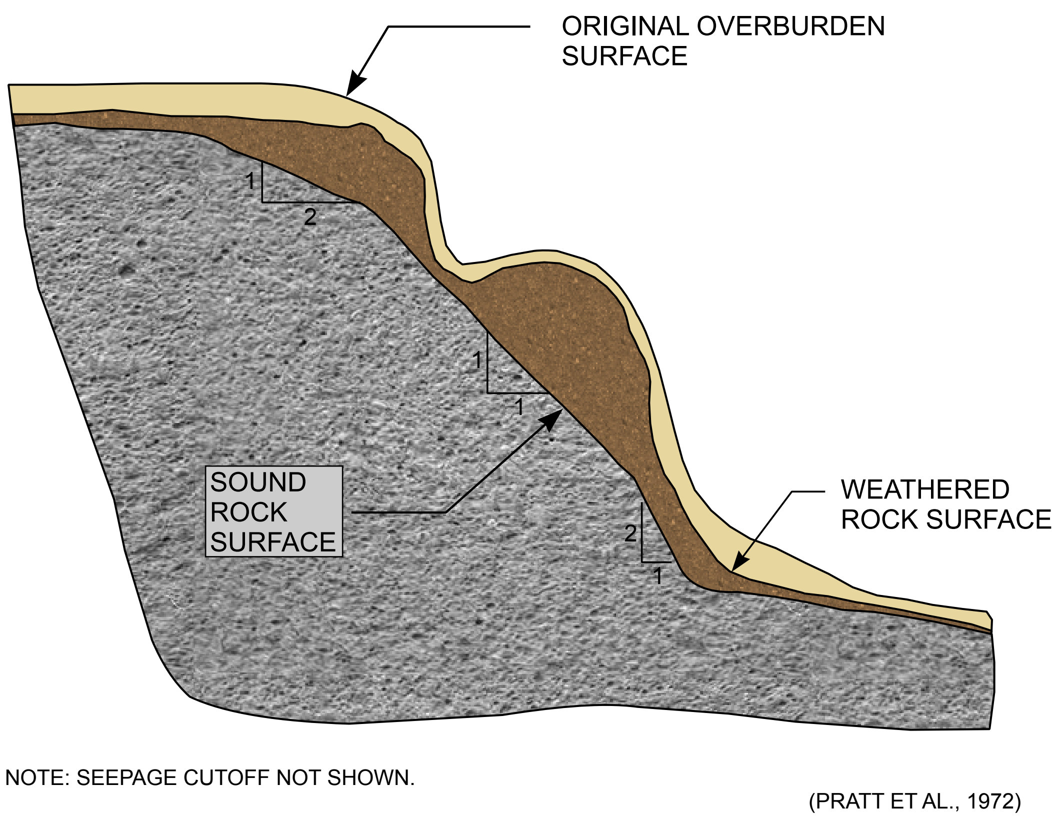
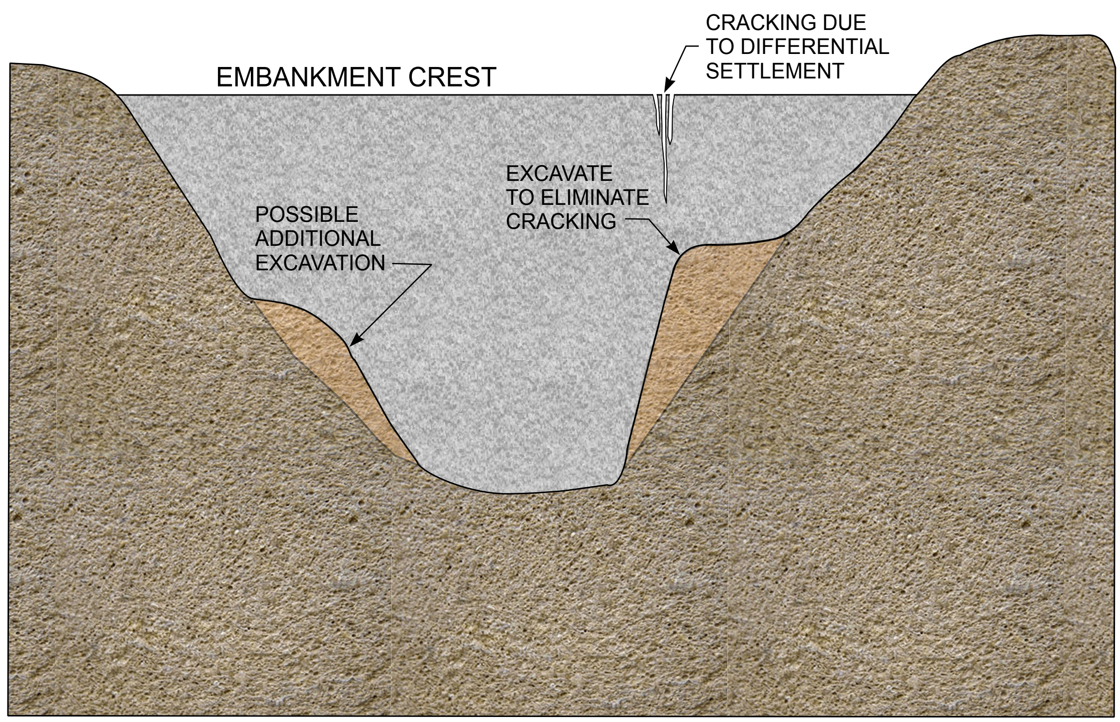


FIGURE 8.17 EXCAVATION SLOPES FOR PREPARATION OF DAM ABUTMENTS IN ROCK

- Steep, abrupt transitions in rock grade – Steep, abrupt transitions in rock grade should be trimmed (Figure 8.18) to minimize the potential for impacts of differential settlement in compacted earthfill susceptible to cracking (e.g., cohesive fill) above the transition. The same cautions discussed above relative to working near a highwall apply.
- Clay-filled seams or very weathered rock – Clay-filled seams or very weathered rock should be excavated and filled with concrete (or grouted) to prevent erosion of the seams. Treatment should focus on the abutment cutoff zone and should extend upstream and downstream to the extent warranted by the site conditions and the embankment cross section. The U.S. Bureau of Reclamation (USBR, 1984) recommends that seams narrower than 2 inches be cleaned to a depth of three times the seam width and that seams between 2 inches and 5 feet wide be cleaned to a depth of three times the seam width or to a depth where the seam is ½ inch wide or less. To avoid unnecessary excavation of stable seam material that would be held in place by concrete and subsequent earthfill, engineering judgment should be used to determine the reasonableness of these guidelines in light of site-specific conditions. Perin (2000) presents a case history that addresses treatment of stress relief fractures at a coal refuse disposal facility.
- Irregularities on slopes not steeper than 0.5H:1V – These irregularities may be treated using dental concrete, pneumatically-applied mortar or slush concrete grout, as illustrated in Figure 8.19. The extent of treatment can be limited to the abutment cutoff zone, as warranted, or may extend throughout the abutment area depending on site-specific conditions and the breadth of and material types in the embankment cross section. Generally the following treatments are considered for structures that



(FELL ET AL., 2005)

FIGURE 8.18 SLOPE MODIFICATION TO REDUCE DIFFERENTIAL SETTLEMENT AND POTENTIAL FOR CRACKING may impose significant hydraulic head in the vicinity of the irregularity (e.g., water retention dams):

> *Dental concrete* should be used to fill joints, bedding, sheared zones, overhangs or excavated surfaces, particularly in the abutment cutoff zone. Fell et al. (2005) recommend that dental concrete slabs have a minimum thickness of 6 inches, a minimum 28-day compressive strength of 3,000 psi, and a maximum aggregate size not more than one-third the depth of the slab or one-fifth the narrowest dimension between the rock surface and the edge of the form. Feathering at the ends of slabs should not be permitted, and slab edges should be sloped no flatter than 45 degrees. To achieve good bond between the rock surface and the concrete, the rock surface should be thoroughly cleaned and moistened prior to concrete placement. The finished concrete surface should have a roughened, broomed surface to facilitate earthfill placement. Sulfate-resistant cement should be used in the concrete.
>
> *Pneumatically-applied mortar (shotcrete)* can be used as an alternate to dental concrete provided that care is taken that it is applied in a manner consistent with the recommendations for dental concrete.
>
> *Slush concrete grout* is a neat cement grout or sand-cement slurry used to fill narrow surface cracks or to serve as a temporary cover over slakable materials that degrade rapidly upon exposure to air and water. Slush grout may be applied by brooming, troweling, pouring, rodding, or funneling into individual cracks (Fell et al., 2005). To facilitate adequate bond, the rock surface and cracks should be cleaned and moistened before the slush concrete grout is applied.

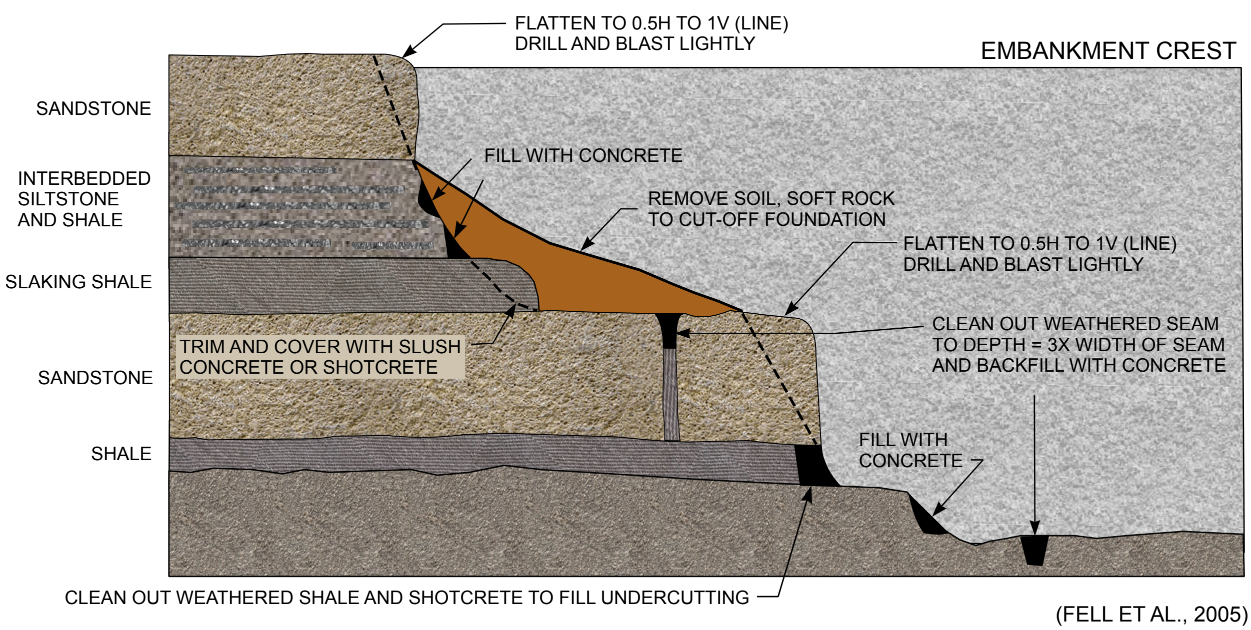


FIGURE 8.19 SLOPE MODIFICATION AND SEAM TREATMENT FOR SEDIMENTARY ROCK STRATA

In some situations, grouting of extensively fractured surface mine benches and highwalls may be more appropriate than other treatments discussed above because of the depth or extent of disturbance. Table 8.6 presents general guidance for cement grout programs (Fell et al., 2005) that may be useful for highly fractured rock foundations. USACE (1984a) and Fell et al. (2005) provide further guidance for evaluation, design and implementation of grouting programs, including the use of chemical grouts for limited applications.

TABLE 8.6 GUIDELINES FOR CEMENT GROUT PROGRAMS

Staging of Grout Program	Downstage	Top section of hole is drilled and washed, pressure tested, grouted and allowed to set for 24 hours. Top section of grout is then washed out then second stage is drilled and washed, pressure tested, grouted and allowed to set for 24 hours. Upper sections of grout are then washed out then third stage is drilled and washed, pressure tested and grouted. Use of packer allows increased pressures.
	Upstage	Hole is drilled to full depth and washed, packer is seated at the top of the bottom stage, pressure tested, grouted, and allowed to set for 6 hours. Set packer at top of second bottom stage, pressure test, grout, and allow to set for 6 hours. Continue remaining stages.
	Full Depth	Hole is drilled to full depth and washed, pressure tested, grouted. Only recommended for consolidation grouting.

Closure Criteria

No further grouting is needed when:

Erodibility of Foundation(1)	Pressure Test Value Before Grouting (Lugeon)(3)	Reduction in Lugeon Value or Grout Take from Previous Stage(2) (Lugeon)(3)	All Grout Takes (kg cement/ m)	Grout Hole Spacing (m)
Low/Non	< 10	< 20	< 25	< 1.5
High	< 7	< 15	< 25	< 1.5

Note: 1. Erodible foundations would include extremely or highly weathered rock and rock with clay-filled joints that might erode under seepage flows.
2. For rock with joints closer than 0.5m.
3. Tabulated values are for Type F portland cement; for Type A portland cement, adopt Lugeon values 20 percent greater. One Lugeon is a flow of 1 liter/minute/ meter of borehole under a pressure of 1000 kPa. In a 75-mm borehole, one Lugeon equivalent to approximately 1.3×10^{-7} m/sec hydraulic conductivity.

Depth and Lateral Extent

So far as practical, grout holes should be taken to the depth and extent necessary to meet closure criteria. Rules of thumb are not recommended. In nearly horizontally layered rock, geologic interpretations may provide a guide where testing is unavailable in the valley floor, but may be at different depths and orientations around the abutments due to the influence of stress relief, weathering, and rock types.

Grouting Effectiveness

Cement Grout Particle Size: Cement particles are mostly silt size, but include some fine sand particles in conventional cement. With plasticizers, Type A and C Portland cements have a maximum particle size of about 0.05 to 0.08 mm, while microfine cements may be about 0.02 mm.

Fracture Size: Minimum Lugeon values are indicative of rock that will accept cement grout

Cement	1 Fracture/m	2 Fractures/m	4 Fractures/m
Type A	8	16	32
Type C	5	10	20
MC-500 (microfine)	3	5	10
Type A with dispersant	8	16	32
Type C with dispersant	5	10	20
MC-500 dispersant	1	2	4

Note: Fractures are assumed to be rough and uniform width and grout is assumed to have been treated with plasticizer.

TABLE 8.6 GUIDELINES FOR CEMENT GROUT PROGRAMS
(Continued)

		Approximate Penetration from Borehole of Grout (m)		
		Fracture Spacing		
Grout Penetration	Lugeons	1m	0.5m	0.25m
	100	20	12	4
	50	12	3	2
	20	3	1.5	1
	10	2	1	NP
	5	1	NP	NP
	1	NP	NP	NP
	Note: NP indicates that grout will not penetrate the fractures.			

Practical Aspects	Grout Holes	30 to 60 mm percussion drilling and washing of borehole.
	Standpipes	Threaded galvanized pipe just larger than drill size, grouted into borehole to enable near surface grouting.
	Grout caps	Necessary when grouting closely fractured or low strength rock where standpipe cannot be sealed into rock.
	Grout Mixers	High speed, high shear, colloidal mixers.
	Agitators	Slow speed designed to prevent cement particle settling.
	Grout Pumps	Helical screw pumps or ram type pumps.
	Packers	Mechanical or inflatable.
	Water Cement Ratio	Starting mix: most sites – 2:1, for rock < 5 Lugeons – 3:1, for rock > 30 Lugeons – 1:1; for very high losses – 0.8:1; for heavily fractured, dry rock – 4:1, and for above water table where excess water is absorbed by dry rock – 5:1.
		Thicken mix: (1) to deal with severe leaks, (2) after 1½ hours with continued take, or (3) if hole is rapidly taking grout (e.g., > 500 liters in 15 minutes).
	Grout Pressure	Recommend avoiding rock fracturing. Start at 100 kPa or less for 5 minutes, then steadily increase over next 25 minutes. Occurrence of fracturing can be detected by sudden loss of grout pressure at top of hole due to increased take. Recommend grouting to refusal or minimum take.
	Monitoring	Parameters: (1) hole location, orientation, and depth, (2) stage depths, (3) water pressure test value for each stage, (4) grout mixes, (5) grout pressures (e.g., 15-minute intervals), (6) grouting times, (7) leaks, uplift, (8) total grout take for each stage, (9) amount of cement in these takes, and (10) cement takes/unit length of hole.
	Water Pressure Test	Before grouting, apply water pressure and monitor for 15 minutes.
	Stage Lengths	Based on geologic conditions, minimum drill run, allowable pressures in upper part of hole, rock fracture conditions and hole stability, water flows into the hole, and large water pressure tests or grout takes.

(ADAPTED FROM FELL ET AL., 2005)

< PREVIOUS VIEW