METHODS OF WORKING COAL AND METAL MINES

VOLUME 3

Planning and Operations

BY

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CHAPTER 1

MINING SYSTEMS

INTRODUCTION

The general plan for extraction of a mineral deposit involves the driving of development openings, from the surface, or from a central shaft, to "block out" portions of the deposit. These development openings furnish information as to thickness, width, dip, grade, etc., of the deposit. They also serve as openings for circulation of ventilation air, as haulageways for material, supplies,and extracted mineral, and as access openings from which working faces for actual extraction operations are driven.

Deposits of coal, and most other non-metallic minerals, such as salt, borates, clay, limestone, usually occur in "beds" or "seams" which are most commonly flat-lying or only slightly inclined. Such minerals normally occur as deposits of considerable areal extent, of fairly uniform thickness, and of consistent dip. Their relatively flat inclination simplifies exploration by drilling from the surface; therefore the characteristics of such a deposit can be determined before actual mining commences. This permits detailed advance planning of an extraction system.

In contrast to the non-metallic minerals most metalliferous deposits tend to be somewhat irregular in outline and variable in thickness and in tenor of ore, and usually only the near-surface portions can be economically explored by drilling from the surface before mining commences.

A number of commercially valuable minerals commonly occur as bedded deposits. Included in this classification are coal, salt, limestone, potash, gypsum, borax, trona, phosphate, some iron ores, and some uranium ores. Of these, coal is by far the most important, both economically and in total tonnage produced.

DOMESTIC COAL PRODUCTION

During the past several years the United States annual production of bituminous coal and lignite has been slightly over 400 million tons. Demand for total energy continues to increase but petroleum and natural gas are encroaching increasingly into the fields formerly served by coal.

Table 1 shows that maximum coal production occurred in 1947 and that there has been a subsequent decline in demand which leveled off in about 1958.

Table 3 shows the steady decline in production of Pennsylvania anthracite which has been in progress since the 1920's.

TABLE 1. GROWTH OF THE BITUMINOUS COAL AND LIGNITE MINING INDUSTRY IN THE UNITED STATES(16)

Year	Production (net tons)	Value of production		Number of mines	Capacity at 280 days (million tons)	Foreign trade*	
		Total	Average per ton			Exports (net tons)	Imports (net tons)
1890	111,302,322	$110,420,801	$0.99	†	137	1,272,396	1,047,416
1891	117,901,238	117,188,400	0.99	†	148	1,651,694	1,181,677
1892	126,856,567	125,124,381	0.99	†	162	1,904,556	1,491,800
1893	128,385,231	122,751,618	0.96	†	174	1,986,383	1,234,499
1894	118,820,405	107,653,501	0.91	†	196	2,439,720	1,286,268
1895	135,118,193	115,779,771	0.86	2,555	196	2,659,987	1,411,323
1896	137,640,276	114,891,515	0.83	2,599	202	2,515,838	1,393,095
1897	147,617,519	119,595,224	0.81	2,454	213	2,670,157	1,442,534
1898	166,593,623	132,608,713	0.80	2,862	221	3,004,304	1,426,108
1899	193,323,187	167,952,104	0.87	3,245	230	3,897,994	1,409,838
1900	212,316,112	220,930,313	1.04	†	255	6,060,288	1,911,925
1901	225,828,149	236,422,049	1.05	†	281	6,455,085	2,214,507
1902	260,216,844	290,858,483	1.12	†	316	6,048,777	2,174,393
1903	282,749,348	351,687,933	1.24	†	350	5,835,561	4,043,519
1904	278,659,689	305,397,001	1.10	4,650	386	7,206,879	2,179,882
1905	315,062,785	334,658,294	1.06	5,060	417	7,512,723	1,704,810
1906	342,874,867	381,162,115	1.11	4,430	451	8,014,263	2,039,169
1907	394,759,112	451,214,842	1.14	4,550	473	9,869,812	1,892,653
1908	332,573,944	374,135,268	1.12	4,730	482	11,071,152	2,219,243
1909	379,744,257	405,486,777	1.07	5,775	510	10,101,131	1,375,201
1910	417,111,142	469,281,719	1.12	5,818	538	11,663,052	1,819,766
1911	405,907,059	451,375,819	1.11	5,887	538	13,259,791	1,972,555
1912	450,104,982	517,983,445	1.15	5,747	566	16,475,029	1,456,333
1913	478,435,297	565,234,952	1.18	5,776	577	18,013,073	1,767,656
1914	422,703,970	493,309,244	1.17	5,592	608	17,589,562	1,520,962
1915	442,624,426	502,037,688	1.13	5,502	610	18,776,640	1,703,785
1916	502,519,682	665,116,077	1.32	5,726	613	21,254,627	1,713,837
1917	551,790,563	1,249,272,837	2.26	6,939	636	23,839,558	1,448,453
1918	579,385,820	1,491,809,940	2.58	8,319	650	22,350,730	1,457,073
1919	465,860,058	1,160,616,013	2.49	8,994	669	20,113,536	1,011,550
1920	568,666,683	2,129,933,000	3.75	8,921	725	38,517,084	1,244,990
1921	415,921,950	1,199,983,600	2.89	8,038	781	23,131,166	1,257,589
1922	422,268,099	1,274,820,000	3.02	9,299	832	12,413,085	5,059,999
1923	564,564,662	1,514,621,000	2.68	9,331	885	21,453,579	1,882,306
1924	483,686,538	1,062,626,000	2.20	7,586	792	17,100,347	417,266

TABLE 1 (*cont.*)

Year	Production (net tons)	Value of production		Number of mines	Capacity at 280 days (million tons)	Foreign trade*	
		Total	Average per ton			Exports (net tons)	Imports (net tons)
1925	520,052,741	1,060,402,000	2.04	7,144	748	17,461,560	601,737
1926	573,366,985	1,183,412,000	2.06	7,177	747	35,271,937	485,666
1927	517,763,352	1,029,657,000	1.99	7,011	759	18,011,744	549,843
1928	500,744,970	933,774,000	1.86	6,450	691	16,164,485	546,526
1929	534,988,593	952,781,000	1.78	6,057	679	17,429,298	495,219
1930	467,526,299	795,483,000	1.70	5,891	700	15,877,407	240,886
1931	382,089,396	588,895,000	1.54	5,642	669	12,126,299	206,303
1932	309,709,872	406,677,000	1.31	5,427	594	8,814,047	186,909
1933	333,630,533	445,788,000	1.34	5,555	559	9,036,947	197,429
1934	359,368,022	628,383,000	1.75	6,258	565	10,868,552	179,661
1935	372,373,122	658,063,000	1.77	6,315	582	9,742,430	201,871
1936	439,087,903	770,955,000	1.76	6,875	618	10,654,959	271,798
1937	445,531,449	864,042,000	1.94	6,548	646	13,144,678	257,996
1938	348,544,764	678,653,000	1.95	5,777	602	10,490,269	241,305
1939	394,855,325	728,348,366	1.84	5,820	621	11,590,478	355,115
1940	460,771,500	879,327,227	1.91	6,324	639	16,465,928	371,571
1941	514,149,245	1,125,362,836	2.19	6,822	666	20,740,471	390,049
1942	582,692,937	1,373,990,608	2.36	6,972	663	22,943,305	498,103
1943	590,177,069	1,584,644,477	2.69	6,620	626	25,836,208	757,634
1944	619,576,240	1,810,900,542	2.92	6,928	624	26,032,348	633,689
1945	577,617,327	1,768,204,320	3.06	7,033	620	27,956,192	467,473
1946	533,922,068	1,835,539,476	3.44	7,333	699	41,197,378	434,680
1947	630,623,722	2,622,634,946	4.16	8,700	755	68,666,963	290,141
1948	599,518,229	2,993,267,021	4.99	9,079	774	45,930,133	291,337
1949	437,868,036	2,136,870,571	4.88	8,559	781	27,842,056	314,980
1950	516,311,053	2,500,373,779	4.84	9,429	790	25,468,403	346,706
1951	533,664,732	2,626,030,137	4.92	8,009	736	56,721,547	292,378
1952	466,840,782	2,289,180,401	4.90	7,275	703	47,643,150	262,268
1953	457,290,449	2,247,943,799	4.92	6,671	670	33,760,263	226,900
1954	391,706,300	1,769,619,723	4.52	6,130	603	31,040,465	198,799
1955	464,633,408	2,092,382,737	4.50	7,856	620	51,277,256	337,145
1956	500,874,077	2,412,004,151	4.82	8,520	655	68,522,629	355,701
1957	492,730,916	2,504,406,042	5.08	8,539	680	76,445,529	366,506
1958	410,445,547	1,996,281,274	4.86	8,264	625	50,293,382	306,940
1959	412,027,502	1,965,606,901	4.77	7,719	614	37,226,766	374,713
1960	415,512,347	1,950,425,049	4.69	7,865	609	36,491,424	260,495
1961	402,976,802	1,844,562,662	4.58	7,648	585	34,969,825	164,259

* Figures for 1890–1914 represent fiscal year ended June 30.
† Data not available.

TABLE 2. GROWTH OF THE BITUMINOUS COAL AND LIGNITE MINING INDUSTRY IN THE UNITED STATES[16]

Year	Men employed	Average number of days worked	Average days lost per man on strike	Net tons per man		Percentage of underground production		Percentage of total production	
				Per day	Per year	Cut by machines*	Mechanically loaded	Mechanically cleaned†	Mined by stripping
1890	192,204	226	‡	2.56	579	‡	‡	‡	‡
1891	205,803	223	‡	2.57	573	5.3	‡	‡	‡
1892	212,893	219	‡	2.72	596	‡	‡	‡	‡
1893	230,365	204	‡	2.73	557	‡	‡	‡	‡
1894	244,603	171	‡	2.84	486	‡	‡	‡	‡
1895	239,962	194	‡	2.90	563	‡	‡	‡	‡
1896	244,171	192	‡	2.94	564	11.9	‡	‡	‡
1897	247,817	196	‡	3.04	596	15.3	‡	‡	‡
1898	255,717	211	‡	3.09	651	19.5	‡	‡	‡
1899	271,027	234	46	3.05	713	22.7	‡	‡	‡
1900	304,375	234	43	2.98	697	24.9	‡	‡	‡
1901	340,235	225	35	2.94	664	25.6	‡	‡	‡
1902	370,056	230	44	3.06	703	26.8	‡	‡	‡
1903	415,777	225	28	3.02	680	27.6	‡	‡	‡
1904	437,832	202	44	3.15	637	28.2	‡	‡	‡
1905	460,629	211	23	3.24	684	32.8	‡	‡	‡
1906	478,425	213	63	3.36	717	34.7	‡	2.7	‡
1907	513,258	234	14	3.29	769	35.1	‡	2.9	‡
1908	516,264	193	38	3.34	644	37.0	‡	3.6	‡
1909	543,152	209	29	3.34	699	37.5	‡	3.8	‡
1910	555,533	217	89	3.46	751	41.7	‡	3.8	‡
1911	549,775	211	27	3.50	738	43.9	‡	‡	‡
1912	548,632	223	35	3.68	820	46.8	‡	3.9	‡
1913	571,882	232	36	3.61	837	50.7	‡	4.6	‡
1914	583,506	195	80	3.71	724	51.8	‡	4.8	0.3
1915	557,456	203	61	3.91	794	55.3	‡	4.7	0.6
1916	561,102	230	26	3.90	896	56.9	‡	4.6	0.8
1917	603,143	243	17	3.77	915	56.1	‡	4.6	1.0
1918	615,305	249	7	3.78	942	56.7	‡	3.8	1.4
1919	621,998	195	37	3.84	749	60,0	‡	3.6	1.2
1920	639,547	220	22	4.00	881	60.7	‡	3.3	1.5
1921	663,754	149	23	4.20	627	66.4	‡	3.4	1.2
1922	687,958	142	117	4.28	609	64.8	‡	‡	2.4
1923	704,793	179	20	4.47	801	68.3	0.3	3.8	2.1
1924	619,604	171	73	4.56	781	71.5	0.7	‡	2.8
1925	588,493	195	30	4.52	884	72.9	1.2	‡	3.2
1926	593,647	215	24	4.50	966	73.8	1.9	‡	3.0
1927	593,918	191	153	4.55	872	74.9	3.3	5.3	3.6
1928	522,150	203	83	4.73	959	76.9	4.5	5.7	4.0
1929	502,993	219	11	4.85	1064	78.4	7.4	6.9	3.8

TABLE 2 (*cont.*)

Year	Men employed	Average number of days worked	Average days lost per man on strike	Net tons per man		Percentage of underground production		Percentage of total production	
				Per day	Per year	Cut by machines*	Mechanically loaded	Mechanically cleaned†	Mined by stripping
1930	493,202	187	43	5.06	948	81.0	10.5	8.3	4.3
1931	450,213	160	35	5.30	849	83.2	13.1	9.5	5.0
1932	406,380	146	120	5.22	762	84.1	12.3	9.8	6.3
1933	418,703	167	30	4.78	797	84.7	12.0	10.4	5.5
1934	458,011	178	15	4.40	785	84.1	12.2	11.1	5.8
1935	462,403	179	§ 7	4.50	805	84.2	13.5	12.2	6.4
1936	477,204	199	21	4.62	920	84.8	16.3	13.9	6.4
1937	491,864	193	§19	4.69	906	‡	20.2	14.6	7.1
1938	441,333	162	13	4.89	790	87.5	26.7	18.2	8.7
1939	421,788	178	36	5.25	936	87.9	31.0	20.1	9.6
1940	439,075	202	8	5.19	1049	88.4	35.4	22.2	9.2
1941	456,981	216	27	5.20	1125	89.0	40.7	22.9	10.7
1942	461,991	246	7	5.12	1261	89.7	45.2	24.4	11.5
1943	416,007	264	§15	5.38	1419	90.3	48.9	24.7	13.5
1944	393,347	278	§ 5	5.67	1575	90.5	52.9	25.6	16.3
1945	383,100	261	9	5.78	1508	90.8	56.1	25.6	19.0
1946	**396,434	214	§23	6.30	1347	90.8	58.4	26.0	21.1
1947	**419,182	234	§ 5	6.42	1504	90.0	60.7	27.7	22.1
1948	**441,631	217	§16	6.26	1358	90.7	64.3	30.2	23.3
1949	**433,698	157	§15	6.43	1010	91.4	67.0	35.1	24.2
1950	**415,582	183	§56	6.77	1239	91.8	69.4	38.5	23.9
1951	**372,897	203	§ 4	7.04	1429	93.4	73.1	45.0	22.0
1952	**335,217	186	§ 6	7.47	1889	92.8	75.6	48.7	23.3
1953	**293,106	191	§ 3	8.17	1560	92.3	79.6	52.9	23.1
1954	**227,397	182	§ 4	9.47	1724	88.8	84.0	59.4	25.1
1955	**225,093	210	§ 4	9.84	2064	88.1	84.6	58.7	24.8
1956	**228,163	214	§ 4	10.28	2195	84.6	84.0	58.4	25.4
1957	**228,635	203	§ 3	10.59	2155	80.9	84.8	61.7	25.2
1958	**197,402	184	§ 3	11.33	2079	75.3	84.9	63.1	28.3
1959	**179,636	188	§24	12.22	2294	72.1	86.0	65.5	29.4
1960	**169,400	191	§ 4	12.83	2453	67.8	68.3	65.7	29.5
1961	**150,474	193	§ 4	13.87	2678	64.7	86.3	65.7	30.3

* Percentages for 1890–1913 are of total production, as a separation of underground and strip production is not available for these years.

† Percentages for 1906–26 are exclusive of coal cleaned at central washeries operated by consumers.

‡ Data not available. § Bureau of Labor Statistics, U.S. Department of Labor.

** Average number of men working daily.

TABLE 3. STATISTICAL TRENDS IN THE

Year	Production (net tons)	Value of production	Average value per net ton	Exports* (net tons)	Imports* (net tons)	Apparent consumption (net tons)
1925	61,817,149	$327,664,512	$5.30	3,179,006	382,894	64,061,000
1926	84,437,452	474,164,252	5.62	4,029,683	813,956	77,221,000
1927	80,095,564	420,941,726	5.26	3,325,507	119,030	74,672,000
1928	75,348,069	393,637,690	5.22	3,336,272	384,707	73,650,000
1929	73,828,195	385,642,751	5.22	3,406,369	487,172	71,457,000
1930	69,384,837	354,574,191	5.11	2,551,659	674,812	67,628,000
1931	59,645,652	296,354,586	4.97	1,778,308	637,951	58,408,000
1932	49,855,221	222,375,129	4.46	1,303,355	607,097	50,500,000
1933	49,541,344	206,718,405	4.17	1,034,562	456,252	49,600,000
1934	57,168,291	244,152,245	4.27	1,297,610	478,118	55,500,000
1935	52,158,783	210,130,565	4.03	1,608,549	571,439	51,100,000
1936	54,579,535	227,003,538	4.16	1,678,024	614,639	53,200,000
1937	51,856,433	197,598,849	3.81	1,914,173	395,737	50,400,000
1938	46,099,027	180,600,167	3.92	1,908,911	362,895	45,200,000
1939	51,487,377	187,175,324	3.64	2,590,000	298,153	49,700,000
1940	51,484,640	205,489,814	3.99	2,667,632	135,436	49,000,000
1941	§56,368,267	240,275,126	4.26	3,380,189	74,669	52,700,000
1942	§60,327,729	271,673,380	4.50	4,438,588	140,115	56,500,000
1943	§60,643,620	306,816,018	5.06	4,138,680	166,020	57,100,000
1944	§63,701,363	354,582,884	5.57	4,185,933	11,847	59,400,000
1945	§54,933,909	323,944,435	5.90	3,691,247	149	51,600,000
1946	§60,506,873	413,417,070	6.83	6,497,245	9556	53,900,000
1947	§57,190,009	413,019,486	7.22	8,509,995	10,350	48,200,000
1948	§57,139,948	467,051,800	8.17	6,675,914	945	50,200,000
1949	§42,701,724	358,008,451	8.38	4,942,670		37,700,000
1950	§44,076,703	392,398,006	8.90	3,891,569	18,289	39,900,000
1951††	42,669,997	405,817,963	9.51	5,955,535	26,812	37,000,000
1952	40,582,558	379,714,076	9.36	4,592,060	29,370	35,300,000
1953	30,949,152	299,139,687	9.67	2,724,270	31,443	28,000,000
1954	29,083,477	247,870,023	8.52	2,851,239	5831	26,900,000
1955	26,204,554	206,096,662	7.86	3,152,313	170	23,600,000
1956	28,900,220	236,785,062	8.19	5,244,349	46	24,000,000
1957	25,338,321	227,753,802	8.99	4,331,785	1138	20,800,000
1958	21,171,142	187,898,316	8.88	2,279,859	4363	19,000,000
1959	20,649,286	172,319,913	8.35	1,787,558	2633	18,800,000
1960	18,817,441	147,116,250	7.82	§§1,440,400	1476	17,600,000
1961	17,446,439	140,337,541	8.04	1,546,488	792	15,900,000

* U.S. Department of Commerce.

† Data first collected in 1929.

‡ As reported by the Commonwealth of Pennsylvania, Department of Mines.

§ Includes some "bootleg" coal purchased by authorized operators and prepared at their breakers.

** Output per man calculated on authorized tonnages only; bootleg purchases excluded.

PENNSYLVANIA ANTHRACITE INDUSTRY[17]

Average number of employees	Average number of days worked	Average tons per man per day	Average tons per man per year	Quantity cut by machines (net tons)	Quantity produced by stripping (net tons)	Quantity loaded mechanically underground† (net tons)
160,312	182	2.12	386	941,189	1,578,478	—
165,386	244	2.09	511	931,650	2,401,356	—
165,259	225	2.15	485	1,171,888	2,153,156	‡2,223,281
160,681	217	2.17	469	1,289,809	2,422,924	‡2,351,074
151,501	225	2.16	487	1,159,910	1,911,766	3,470,158
150,804	208	2.21	460	1,410,123	2,536,288	4,467,750
139,431	181	2.37	428	1,587,265	3,813,237	4,384,780
121,243	162	2.54	411	1,674,223	3,980,973	5,433,340
104,633	182	2.60	473	1,648,249	4,932,069	6,557,267
109,050	207	2.53	524	1,981,088	5,798,138	9,284,486
103,269	189	2.68	505	1,848,095	5,187,072	9,279,057
102,081	192	2.79	535	2,162,744	6,203,267	10,827,946
99,085	189	2.77	523	1,984,512	5,696,018	10,683,837
96,417	171	2.79	478	1,588,407	5,095,341	10,151,669
93,138	183	3.02	553	1,881,884	5,486,479	11,773,833
91,313	186	3.02	562	1,816,483	6,352,700	12,326,000
88,054	203	**3.04	617	1,855,422	7,316,574	13,441,987
82,121	239	**2.95	705	2,285,640	9,070,933	14,741,459
79,153	270	**2.78	751	1,624,883	8,989,387	14,745,793
77,591	292	**2.79	815	1,336,082	10,953,030	14,975,146
72,842	269	**2.79	751	1,210,171	10,056,325	13,927,955
78,145	271	**2.84	770	1,232,828	12,858,930	15,619,162
78,600	259	**2.78	720	1,209,983	12,603,545	16,054,011
76,215	265	**2.81	745	1,016,757	13,352,874	15,742,368
75,377	195	**2.87	560	557,599	10,376,808	11,858,088
72,624	211	**2.83	597	611,734	11,833,934	12,335,650
68,995	208	2.97	618	496,085	11,135,990	10,847,787
65,923	201	3.06	615	386,128	10,696,705	10,034,464
57,862	163	3.28	535	318,699	8,606,482	6,838,769
43,996	164	4.02	659	381,424	7,939,680	6,978,035
‡‡33,523	‡‡197	‡‡3.96	‡‡780	393,932	7,703,907	6,660,939
31,516	216	4.25	918	400,402	8,354,230	7,308,110
30,825	196	4.18	819	292,307	7,543,157	6,657,479
26,540	183	4.36	798	184,028	7,877,761	5,332,043
23,294	173	5.12	886	260,502	7,096,343	4,700,542
19,051	176	5.60	986	225,520	7,112,228	4,044,392
15,792	196	5.63	103	236,166	7,246,646	3,377,778

†† Figures for 1951 and subsequent years are not strictly comparable with previous years. See Production and Employment sections, Coal—*Pennsylvania Anthracite*, *Minerals Yearbook*, 1951.

‡‡ Estimated.

§§ Revised.

Figure 1 illustrates the distribution of coal consumption from 1952 to 1961. It will be noted that change from coal burning locomitives to diesel locomotives has practically eliminated this market for coal. The retail delivery market for coal also shows a steady decrease which might be expected to continue.

The electric utilities show a continuous increase in the amount of coal consumed and this increase may be expected to continue in spite of the increases in efficiencies which have reduced coal consumption to less than one pound per kilowatt-hour of electricity produced. This is illustrated in Fig. 2.

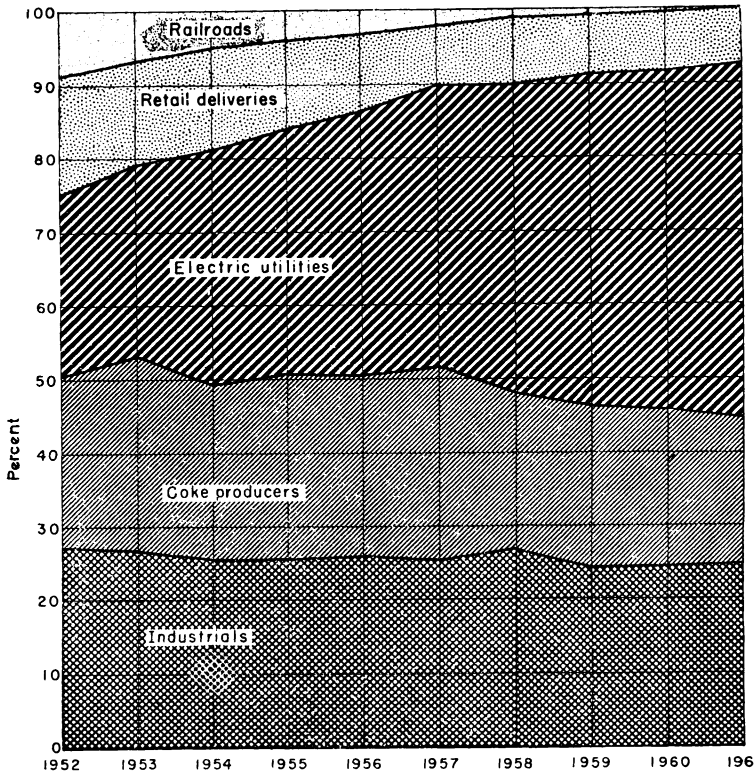

FIG. 1. Percentage of total consumption of bituminous coal and lignite, by consumer class, and retail deliveries in the United States, 1952–1961.[16]

It is expected that the prices of both fuel oil and natural gas will be subject to a steady slow rise as reserves decrease and that the unit prices for coal delivered to electrical utility plants can be held at or near their present levels by improved mining techniques. Thus it is expected that coal's competitive position in its battle for markets will improve over the long term and that it will regain some of the markets now being served by petroleum and natural gas. The average value of a ton of coal at the mine during the past 45 years is shown in Fig. 3.

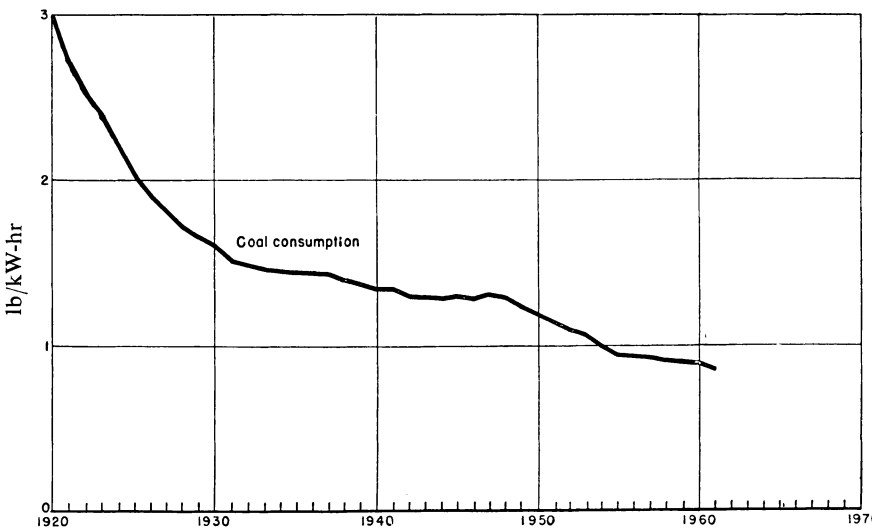

FIG. 2. Trend in fuel economy at electric-utility power plants in the United States, 1920–1961.(16)

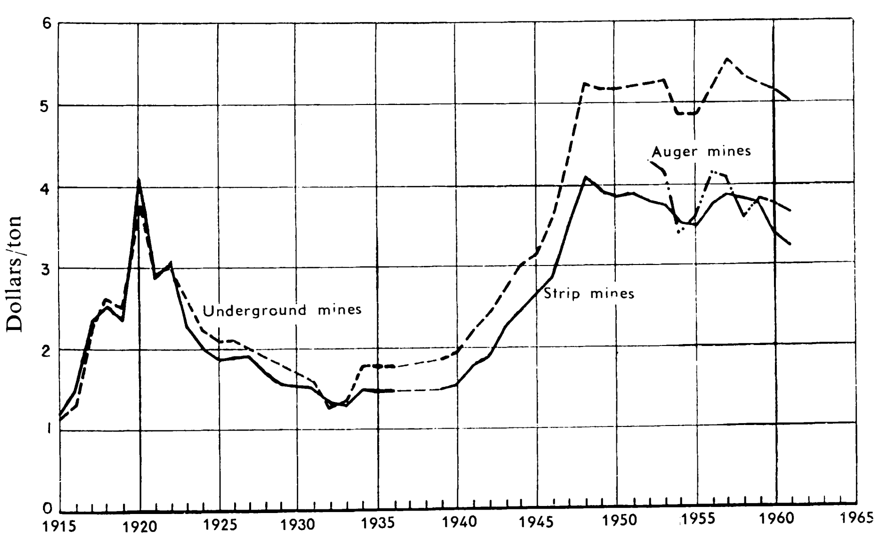

FIG. 3. Average value per ton, f.o.b. mines, of bituminous coal and lignite produced in the United States, 1915–1961, by underground, strip, and auger mines.(16)

WORLD COAL PRODUCTION

The importance of the coal mining industry is indicated by the following tables which list production by countries for the years 1957–1961. These tables indicate that total production is increasing annually and that production in the near future will attain a rate of three thousand million tons annually.

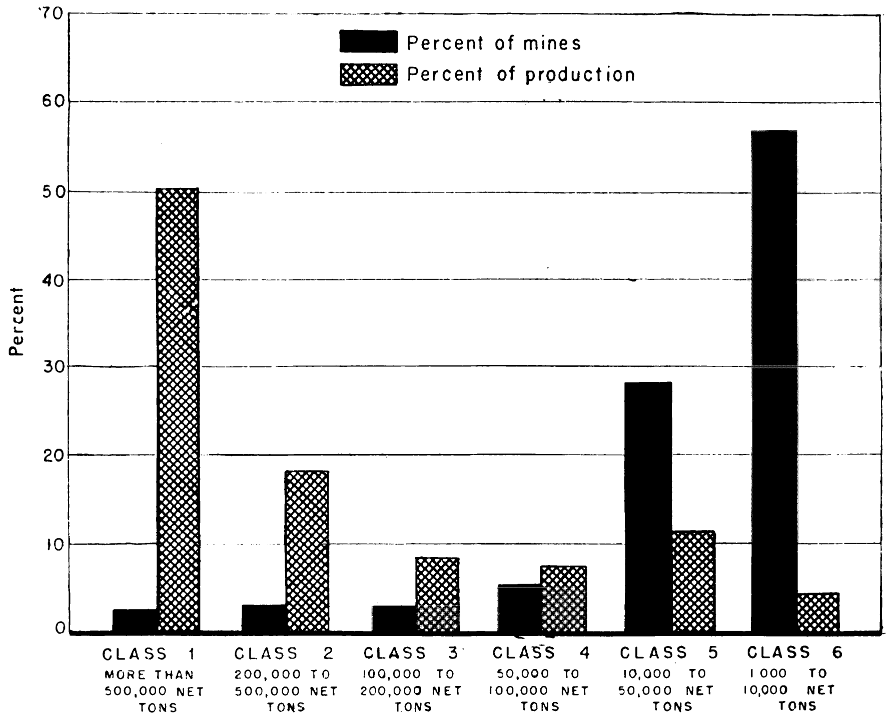

FIG. 4. Percentage of number of mines and of production of bituminous coal and lignite mines in the United States, 1961, by size of output.[16]

Total world energy requirements are increasing rapidly and annual coal production would increase much more rapidly except that petroleum and natural gas, as in the United States, are filling an increasing proportion of the demand for fuels.

The tonnage figures given include coal produced by surface stripping. In the United States about 30 per cent of annual production is obtained by stripping. In Russia, Germany, and England, lesser but important quantities of coal are mined by stripping.

Great advances have been made in devising and applying machinery to coal production with the objective of increasing the amount of coal produced per man-day. Methods developed in the United States have resulted in consistent

TABLE 4. WORLD PRODUCTION OF BITUMINOUS COAL, ANTHRACITE, AND LIGNITE, BY COUNTRIES*[16]
(Thousand short tons)

Country	1957	1958	1959	1960	1961†
North America:					
Canada:					
Bituminous	10,940	9434	8680	8840	8189
Lignite	2249	2253	1948	2170	2209
Greenland: Bituminous	19	35	29	31	35
Mexico: Bituminous	1566	1621	1748	1958	2004
United States:					
Anthracite (Pennsylvania)	25,338	21,171	20,649	18,817	17,446
Bituminous	490,097	408,019	409,248	412,766	399,959
Lignite	2607	2427	2780	2746	3018
Total North America	532,816	444,960	445,082	447,328	432,860
South America:					
Argentina: Bituminous	230	288	331	309	371
Brazil: Bituminous (including lignite)	2285	2469	2568	2568	2425
Chile: Bituminous (mined)	2310	2204	2083	1569	1944
Colombia: Bituminous	2205	2690	2756	2866	2756
Peru: Bituminous and anthracite	155	246	191	179	184
Venezuela: Bituminous	39	40	37	39	34
Total South America	7224	7937	7966	7530	7714
Europe:					
Albania: Lignite	259	282	317	320	‡330
Austria:					
Bituminous	168	155	148	146	117
Lignite	7581	7158	6857	6584	6240
Belgium: Bituminous and anthracite	31,968	29,831	25,085	24,763	23,742
Bulgaria:					
Bituminous and anthracite	424	419	551	628	650
Lignite	12,681	13,614	16,359	18,249	19,853
Czechoslovakia:					
Bituminous	26,655	26,380	27,694	28,896	28,867
Lignite	56,235	62,653	59,198	64,378	71,984
Denmark: Lignite	2822	2695	2540	2545	2384
France:					
Bituminous and anthracite	62,610	63,632	63,501	61,685	57,715
Lignite	2528	2555	2398	2509	3203
Germany:					
Bituminous and anthracite:					
East	3035	3201	3132	2999	2950
West (including Saar)	166,206	165,286	157,237	157,911	158,309
Lignite: East	234,346	236,962	236,776	248,461	260,586
West	106,716	103,052	102,991	105,974	107,140
Pech coal: West	2048	2013	2022	1969	1947

TABLE 4 (*cont.*)

Country	1957	1958	1959	1960	1961†
Europe—Continued					
Greece: Lignite	1100	1315	1774	2812	2778
Hungary:					
Bituminous	2510	2895	3014	3138	3387
Lignite	20,861	23,826	24,927	26,098	27,672
Ireland: Bituminous and anthracite	266	225	257	259	224
Italy:					
Bituminous and anthracite	1129	798	815	812	817
Lignite	434	916	1347	875	1661
Netherlands:					
Bituminous and anthracite	12,540	13,095	13,203	13,777	13,912
Lignite	317	281	219	4	–
Poland:					
Bituminous	103,723	104,699	109,246	115,123	117,513
Lignite	6563	8313	10,205	10,281	11,396
Portugal:					
Anthracite	550	625	581	480	500
Lignite	203	172	175	172	174
Rumania:					
Bituminous and anthracite	277	330	330	330	330
Lignite	7500	7813	8466	8667	9264
Spain:					
Bituminous and anthracite	15,356	15,922	14,926	15,193	15,185
Lignite	2777	2945	2317	1942	2302
Svalbard (Spitsbergen):					
Bituminous:					
Controlled by Norway	423	317	278	443	398
Controlled by U.S.S.R.	434	425	505	529	‡550
Sweden: Bituminous	335	352	300	272	220
Switzerland: Bituminous and anthracite (including lignite)†	11	11	11	11	11
U.S.S.R.:§					
Bituminous and anthracite	362,111	389,148	402,586	413,292	415,483
Lignite	148,777	157,721	155,851	152,406	142,727
United Kingdom: Bituminous and anthracite	250,464	241,723	230,839	216,838	213,321
Yugoslavia:					
Bituminous	1353	1332	1431	1414	1447
Lignite	18,497	19,597	21,836	23,623	25,089
Total Europe§	1,674,793	1,714,684	1,712,245	1,736,808	1,752,378
Asia:					
Afghanistan: Bituminous	30	37	40	52	58
Burma: Bituminous	1		1	**	2

TABLE 4 (*cont.*)

Country	1957	1958	1959	1960	1961†
Asia – Continued					
China: Bituminous, anthracite and lignite	144,100	297,600	383,400	463,000	‡420,000
India: Bituminous	48,727	50,788	52,722	58,070	61,872
Indonesia: Bituminous	790	665	703	724	622
Iran: Bituminous††	194	214	257	254	220
Japan:					
Bituminous and anthracite	57,025	54,756	52,093	56,292	60,054
Lignite	1823	1744	1619	1552	1443
Korea:					
North: Anthracite, bituminous and lignite	5494	7586	9760	11,707	12,996
Republic of: Anthracite	2691	2044	4559	5897	6486
Malaya: Bituminous	171	75	85	8	
Outer Mongolia: Lignite and bituminous	450	521	665	682	826
Pakistan: Bituminous and lignite	578	669	820	915	1,015
Philippines: Bituminous	211	119	154	163	168
Ryukyu Islands: Bituminous	2	1	1	1	1
Taiwan: Bituminous	3214	3508	3928	4367	4670
Thailand: Lignite	110	138	155	164	‡170
Turkey (mined):					
Bituminous	6917	7220	7191	6952	7935
Lignite	4009	4212	4038	3769	4159
Viet-Nam:					
North: Anthracite	1200	1980	2124	2682	‡2900
South: Anthracite	13	22	22	39	‡33
Total Asia§	277,759	434,799	524,640	617,272	584,730
Africa:					
Algeria: Bituminous and anthracite	260	169	134	131	86
Congo, Republic of the (formerly Belgian): Bituminous	477	324	294	195	‡80
Malagasy, Republic of: Bituminous	1				2
Marocco: Anthracite	574	562	513	454	452
Mozambique: Bituminous	298	273	283	287	354
Nigeria: Bituminous	913	1036	831	629	669
Rhodesia and Nyasaland, Federation of: Southern Rhodesia: Bituminous	4247	3897	4144	3923	3387
Swaziland: Anthracite and bituminous			1	13	1
Tanganyika: Bituminous	1	1	2	2	2
Union of South Africa: Bituminous and anthracite (marketable)	38,325	40,879	40,181	42,078	43,613
Total Africa	45,096	47,141	46,383	47,712	48,646

TABLE 4 (*cont.*)

Country	1957	1958	1959	1960	1961†
Oceania:					
Australia:					
Bituminous	22,310	22,895	22,734	25,285	27,012
Lignite	12,030	13,041	14,599	16,763	18,236
New Zealand:					
Bituminous and anthracite	931	939	941	896	844
Lignite	1994	2108	2205	2477	2418
Total Oceania	37,265	38,983	40,479	45,421	48,510
Lignite (total of items shown above) (estimate)	655,478	678,314	682,562	706,214	727,262
Bituminous and anthracite (by subtraction)	1,919,475	2,010,190	2,094,233	2,195,857	2,147,576
World total, all grades (estimate)	2,574,953	2,688,504	2,776,795	2,902,071	2,874,838

* This table incorporates some revisions.
† Preliminary.
‡ Estimate.
§ Output from U.S.S.R. in Asia (including Sakhalin) included with U.S.S.R. in Europe.
** Less than 500 tons.
†† Year ended March 20 of year following that stated.
Compiled by Pearl J. Thompson, Division of Foreign Activities.

production, under favorable natural conditions, of 50–60 tons/man-shift per man at the working face and up to 20 tons/man-shift based on total mine payroll.

Other types of bedded deposits are frequently mined by methods patterned after those devised originally for coal mining. Trona (sodium sesquicarbonate) is produced by room-and-pillar methods using conventional coal mining machinery.(2) Similar methods-and-machinery are applied to mining salt, gypsum, borax, and potash. For the metalliferous bedded deposits such as iron ores and uranium ores, extraction systems may follow similar patterns but the hardness of these ores requires the use of "hardrock" drilling and blasting methods, and the increased specific gravity of the ores requires the use of heavier types of loading and transportation machinery.

DEPTHS OF COAL MINING OPERATIONS

In the United States coal seams are generally situated at relatively shallow depths beneath the surface. Most are located at depths between 200 and 800 ft beneath the surface, while the average depth is probably between 300 and 500 ft. This situation has favored the development of pillar-mining methods. Such methods are well suited to mechanized methods and particularly to continuous mining, because they allow the face equipment to be concentrated within a relatively small area.

In British coal fields mining has progressed to greater depths and the majority of the seams now being mined are situated at depths greater than 1000 ft, with a few being worked at depths approaching 4000 ft. Such depths do not favor pillar-mining methods and 74 per cent of Britain's coal is produced by long-wall methods.

In Germany coal mining has similarly progressed to great depths, with the average depth of mining in the Ruhr District being about 2300 ft, although a few workings extend almost to 4000 ft. Under such conditions the depth pressure tends to discourage the use of pillar-mining methods. A smaller proportion of the coal

TABLE 5. NUMBER AND PRODUCTION OF BITUMINOUS COAL AND LIGNITE MINES IN THE UNITED STATES, 1960, CLASSIFIED BY THICKNESS OF SEAMS MINED[16]

Item	Less than 2 ft	2 to 3 ft	3 to 4 ft	4 to 5 ft	5 to 6 ft	6 to 7 ft	7 to 8 ft	8 ft and over	Total
Number of mines:									
Underground	35	1811	2178	990	449	266	132	128	5989
Strip	140	510	418	222	106	52	22	60	1530
Auger	3	71	129	94	40	8	......	1	346
Total	178	2392	2725	1306	595	326	154	189	7865
Percentage of mines:									
Underground	0.6	30.2	36.4	16.5	7.5	4.5	2.2	2.1	100.0
Strip	9.2	33.3	27.3	14.5	6.9	3.4	1.5	3.9	100.0
Auger	0.9	20.4	37.3	27.2	11.6	2.3	......	0.3	100.0
Total	2.3	30.4	34.6	16.6	7.6	4.1	2.0	2.4	100.0
Production (thousand tons):									
Underground	231	20,851	65,322	49,633	53,928	39,833	29,665	25,425	284,888
Strip	5660	19,503	32,934	30,456	17,692	7126	3546	5713	122,630
Auger	44	939	2781	2965	971	235	......	59	7994
Total	5935	41,293	101,037	83,054	72,591	47,194	33,211	31,197	415,512
Percentage of production:									
Underground	0.1	7.3	22.9	17.4	19.0	14.0	10.4	8.9	100.0
Strip	4.6	15.9	26.9	24.8	14.4	5.8	2.9	4.7	100.0
Auger	0.5	11.7	35.0	37.1	12.1	2.9	......	0.7	100.0
Total	1.4	9.9	24.3	20.0	17.5	11.4	8.0	7.5	100.0

is removed during the development phase, as larger pillars must be left for support. During the mining of pillars on retreat the heavy abutment pressure ahead of the retreating pillar line tends to damage or destroy the previously driven development headings. This abutment pressure produces bottom heave, crushing of pillars, and fracturing of the roof. These factors, together with the desirability for complete mining of each seam, and limitations on surface subsidence, explain the extensive use of long-wall mining methods in European coal mines.

Russia has large coal reserves situated at depths of less than 1200 ft.[3] The average depth of workings in the Kuzbass is 720 ft while in the Donbass it is 990 ft for flat seams and 1320 ft for inclined seams. Sixty per cent of the output if this latter coal field is produced from depths of less than 990 ft. The deepest level visited by the British Technical Mission was situated at a depth of 2370 ft.

Thus the Russian coal mining operations are conducted at depths greater than those of the United States but at depths much shallower than those of the other European countries. Although conditions in some Russian mines are favorable for the employment of room-and-pillar methods only 0.6 per cent of the total output was obtained by this method. With the development, in Russia, of hydraulic mining methods the proportion produced by room-and-pillar methods may increase.

In France 95 per cent of underground production is mined by long-wall methods.[4] Experiments in French coal mines to determine the depths to which pillar-mining methods are practicable have shown that the "critical depth" generally lies between 250 and 500 m (800–1600 ft), and is generally around 400 m (1280 ft). However, room-and-pillar mining methods are succesfully used in mining potash at depths of 1900–2200 ft.

Belgium has the most difficult coal mining conditions. Many seams are thin and depths of the workings range from a minimum of about 1800 to a maximum of about 4400 ft.

MINING SYSTEMS

Practically all coal produced by underground mining is mined by one of the following systems:

(1) Pillar-mining systems. These include the room-and-pillar system and the block system as well as the bord-and-pillar system which is employed in Great Britain.

(2) Long-wall systems. These include the long-wall advancing and the long-wall retreating systems.

Pillar-mining Systems

Entries, cross entries, panel entries, and cross cuts, or rooms are driven through the coal bed to divide it into pillars or blocks which may then be extracted on retreat. Figure 5 shows the general layout for a room-and-pillar system. Figure 6 is an isometric cut-away view showing layout of a room-and-pillar system using rail face haulage and hand loading of coal. In the United States such hand loading methods have now been almost entirely replaced by mechanical loaders and 90 per cent of the coal produced by underground mines is loaded mechanically.

Long-wall Systems

"Main roads" or "mother gates" are driven through the coal bed and are semi-permanent in nature and "gate roads" or "gates" leading to the long-wall face are maintained.

A long face or "long-wall" is usually several hundred feet long and is served by two or more "gate roads"; the long dimension of the working face being at right angles to the direction of the gate roads. Some long-wall faces are several thousand

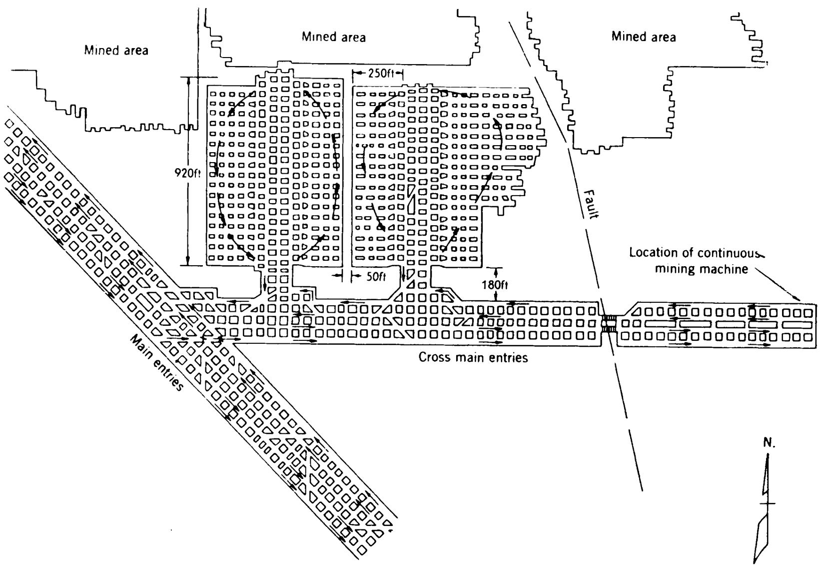

FIG. 5. Mining method at Mine 3 (room-and-pillar system).[5]

feet long while others are operated as a series of stepped faces, each several hundred feet long.

The long-wall is advanced continuously by extracting slices of coal from the face and transporting the broken coal to the gates from which it is transported to the main roads or mother gates and thence to the main shaft.

"Cross gates" are maintained as angular cross-connections between gates in order to shorten haulage and ventilation distances.

Figure 7 shows a layout for a typical long-wall mining system as employed in European mining practice. Figure 8 shows the layout for a long-wall mining system used in conjunction with a continuous mining machine at a mine in Canada.

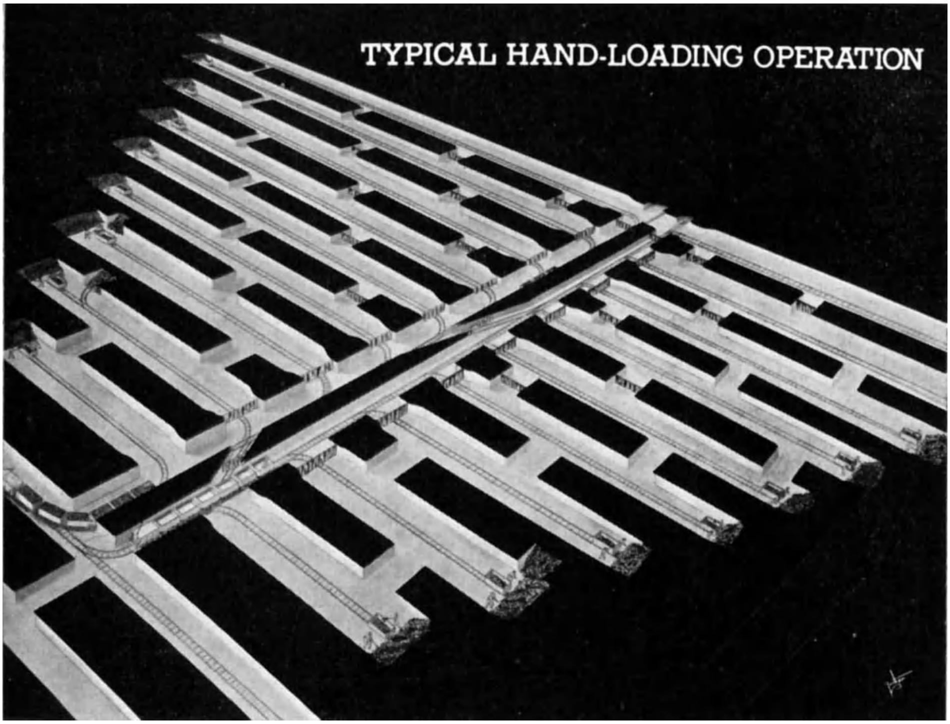

FIG. 6. Typical hand loading operation (room-and-pillar system). (Joy Manufacturing Company.)

PILLAR-MINING SYSTEMS

These include the "room-and-pillar" system, the "block" system, and (in Britain) the "bord-and-pillar" system.

With the room-and-pillar system entries, cross-entries, and panel entries are driven to "block out" large panels of coal and rooms are turned off (usually at right angles) from the entries. (The rooms correspond to the "stopes" of metal mines.) In the United States entries are ordinarily driven with widths between 15 and 25 ft. Rooms are driven as wide, or wider, than the entries. Pillars left between rooms may, or may not, be extracted. Different mines employ different sizes of pillars. Room pillars may commonly be from 20 to 40 ft wide and from 40 to 90 ft long. When room pillars are not to be recovered they are made as narrow as is feasible while still leaving enough coal to support the roof.

Figure 11 shows an idealized room-and-pillar system.

With the block system a series of entries, panel entries, rooms, and cross cuts is driven to divide the coal into a series of blocks of approximately equal size which are then extracted on retreat. Development openings are most commonly driven between 15 and 20 ft wide. Pillars are most commonly from 40 to 60 ft wide and from 60 to 100 ft long.

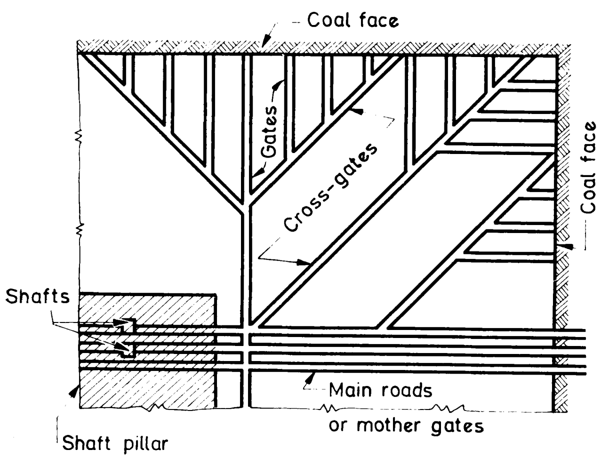

FIG. 7. Typical layout for a long-wall operation.

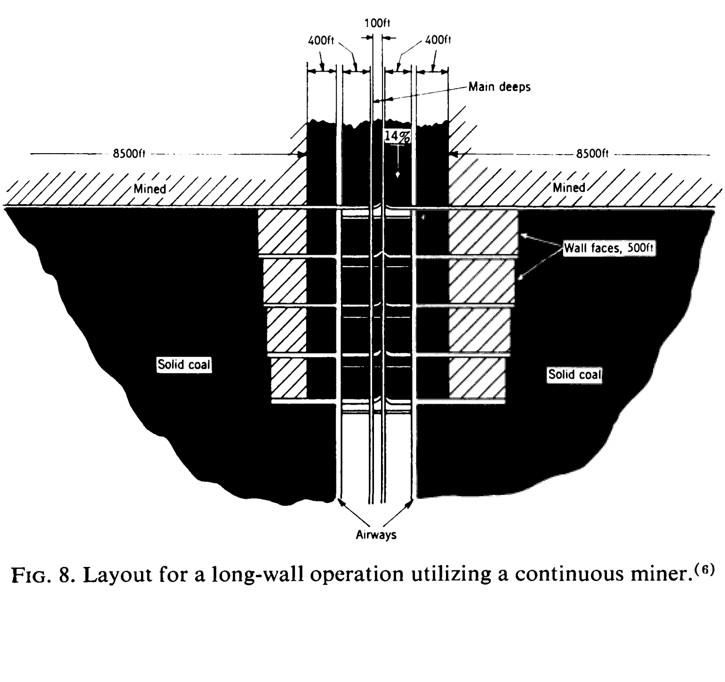

FIG. 8. Layout for a long-wall operation utilizing a continuous miner.[6]

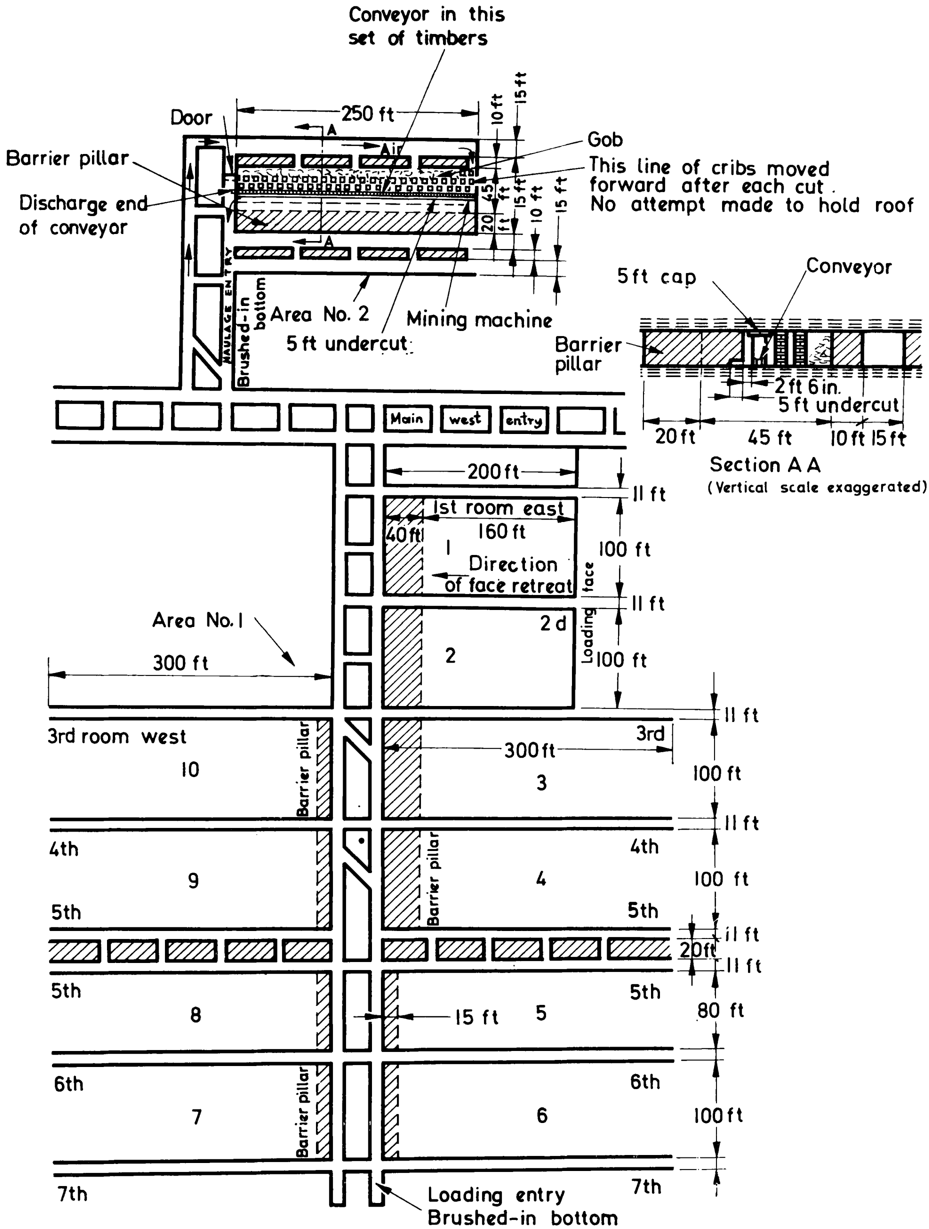

FIG. 9. Layout for a long-wall operation utilizing shaking conveyors – as used in the northern Illinois area during the 1930's.[18]

Figure 12 shows an idealized block mining system.

Generally, the room-and-pillar system is favored for the thinner coal beds (that is, those less than 3–4 ft thick) while the block system is used more often in the

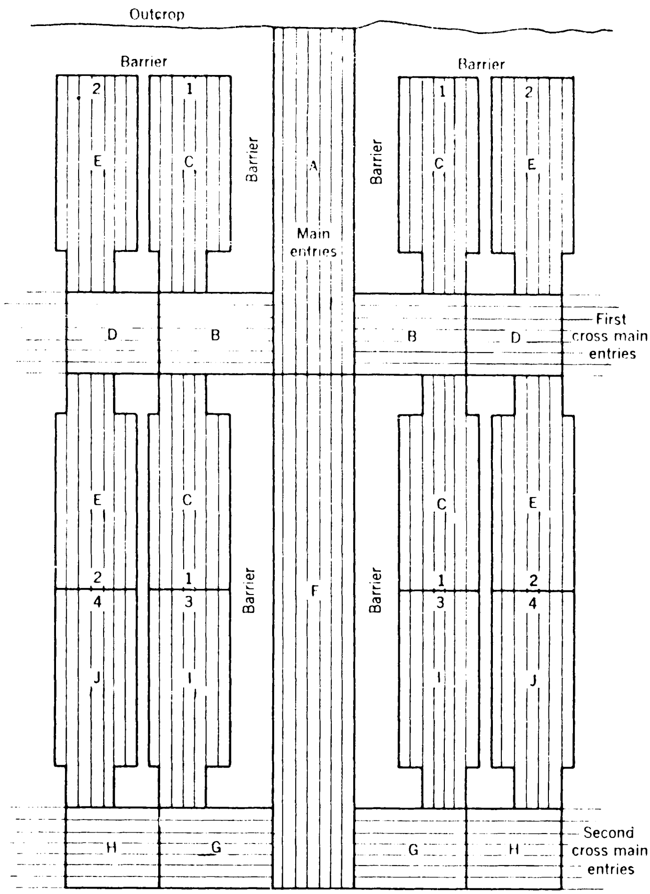

Fig. 10. Schematic plan of development of a pillar-mining system from the outcrop.[7]

thicker coal beds where equipment can move about freely without the necessity of "brushing" roadways to obtain headroom.

The bord-and-pillar system, such as used in some British mines, divides the coal into very large pillars or blocks by means of very narrow openings. "Bords" approximately 12–15 ft wide are driven at approximately right angles to the main coal cleat, and narrow openings ("walls") about 6 ft wide are driven parallel to the main cleat. With the bord-and-pillar system pillars as large as 130–200 ft square are blocked out and extracted on retreat. This system may be considered a compromise between the block system and the retreating long-wall.

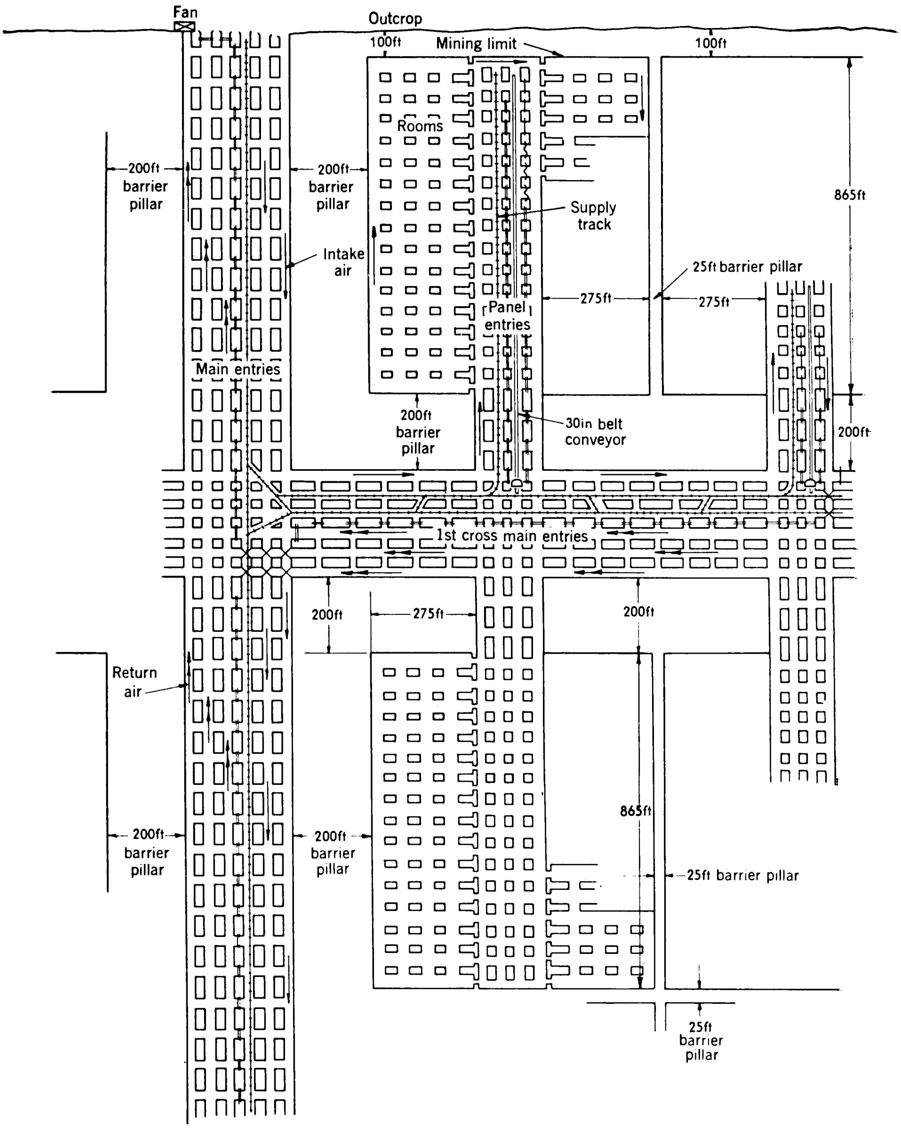

FIG. 11. Development by room-and-pillar system from the outcrop.[7]

The widths of development openings such as entries, rooms, cross cuts, are determined by several factors which include: strength of roof, depth of workings, widths required for efficient use of machinery, and thickness of seam.

In thinner seams the tendency is to make the development opening wider in order to increase the amount of coal obtained per linear foot of advance of the mining unit. Also greater widths of opening will be required in thin seams to obtain equivalent cross-section of airway. However, the maximum width of an opening

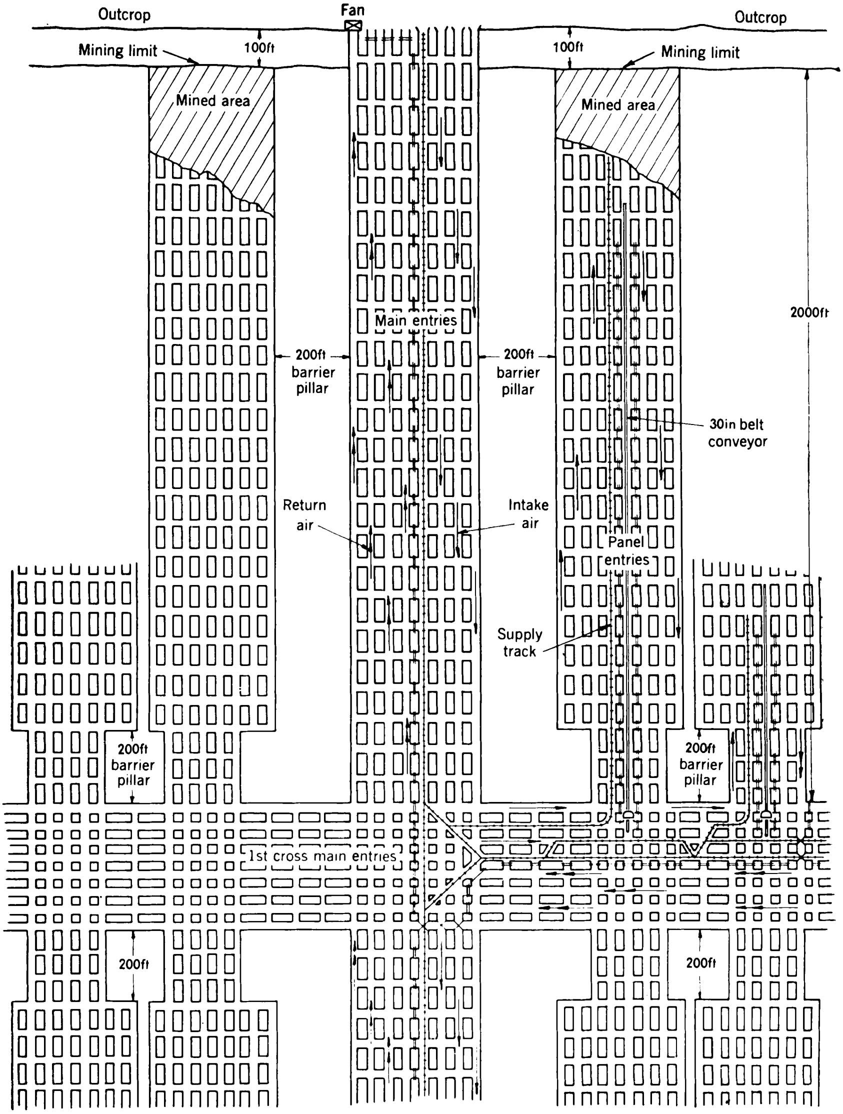

FIG. 12. Development of block system from the outcrop.(7)

will be limited by the strength of roof, and coal seam, as related to the depth pressure.

The bord-and-pillar system evolved from what was originally a room-and-pillar system. As depths of mining increased the widths of development openings decreased in order to secure stability of the openings. Widths of pillars were increased to avoid excessive crushing of pillars under the abutment pressures which were induced during mining of the pillars on retreat.

Mine Development

In coal mining there are two broad classes of underground openings:[7] (1) those made for mine development and (2) those made for production.

Development work comprises driving entries, entry cross cuts, room necks, or other openings necessary to open up access to production places from which coal will be produced later.

Development openings formerly were driven narrower than production places and only a few men could work as a group in a development heading. Because the working places were so narrow the rate of coal production was low and the cost of development coal was high.

As mines became more mechanized and yielded larger quantities of coal at a faster rate per shift it became necessary to increase the quantity of ventilating air. In order to provide adequate airways it was necessary to increase the number of development headings in a group and/or to make the development headings wider. It is common practice now to drive development headings ten or twelve in a group and, where roof conditions will allow it, the headings are made wider. As the width of the development heading approaches that of the production place the productivity from development approaches that from production places.

Production work in the room-and-pillar system involves the driving of rooms and the subsequent extraction of room pillars. With the block system the production work consists of the extraction of the blocks.

Ratio of Development to Production

For successful operation of a mine a certain ratio between development and production must be maintained. If development does not exceed production, mining in a production section could be completed before the next production section is ready for operation. In mines where the cost of coal mined in development is more than that from production sections such an imbalance between development and production would increase the overall cost of the mined coal.

Factors Determining the Amount of Development Needed[7]

Desired Output of Coal Per Day

One of the first decisions is to determine the desired daily output of coal from the mine. This factor would probably be influenced by seasonal demand.

If the cost of development coal is comparable with that from production, a light demand could mean that all of the yield might be from developing new panels in preparation for heavier demand when more production places would be needed.

Type of Mine Opening

In the case of new mines the rate of development may be limited by the type of the main access openings. If the main access is by a shaft, development can proceed in two or more directions, depending upon the plan of the mine bottom. If the mine is opened by drifts it may be necessary to advance the drifts in one direction for a long distance before lateral entries can be started.

Presence of Gas

An amount of ventilating air adequate to dilute and disperse mine gas will be required. If large amounts of gas are released when coal is mined then large quantities of ventilating air will be required, and in order to provide for passage of this air additional development entries will have to be driven, or entries will have to be made wider. Thus more development openings will be required in a gassy mine than in a non-gassy mine.

System of Mining

The system of mining used will probably be the most important factor in determining the required ratio of development to production. To demonstrate the possible effect of the mining system on the development–production ratio, the amount of development required for mining an area by the block system will be compared with the amount of development required for mining the same area by the room-and-pillar system. Figure 10 shows the general mining plan schemetically. Figure 12 shows the layout for the block mining plan and Fig. 11 shows the layout for the room-and-pillar mining plan. Each mining plan begins at the bed outcrop and each develops working panels by multiple main and cross entries, panels being turned off on both sides of the cross mains.

The development is indicated by letters *A* to *J*, and the production units resulting from the development are indicated by numerals. For example, it will be seen that, by driving entries in section *A* and *B*, the first cross mains will be advanced far enough to start development *C* in four places which, when completed, will make four production panels (four panels marked 1 in Fig. 10). Four additional panels (shown as 2 in Fig. 10) off the first cross mains, can be developed by driving development *D* and *E*. To get production panels off the second cross mains, it would be necessary to develop *F*, *G*, amd *I*, following the same pattern as in developing panels off the first cross mains.

Development Required for Block Mining[7]

Figure 12 is the plan of a mine to be mined by the block system with the blocks to be recovered on retreat. In this plan main entries and cross main entries 16 ft wide are driven in groups of eight on 45-ft centers. All cross cuts are 16 ft wide and on 80-ft centers. It is assumed that the coal bed mined is 84 in. thick.

Crew Required

A development crew will be assumed to comprise the following workmen:

Classification	Number of men	Classification	Number of men
Foreman	1	Shot firer	1
Loading-machine operators	2	Shuttle-car operators	3
Loading-machine operators' helpers	2	Roof bolters	4
Cutting-machine operators	2	Mechanic	1
Cutting-machine operators' helpers	2	Boom man	1
		Total	19

This group will work as a team, and individual workers will be used interchangeably for any work to be done.

Mining Equipment Required

Mining equipment of the same type and amount as used at a mine studied is proposed for this plan as follows:

3 mobile-loading machines (1 to be kept in reserve as a spare).
2 universal cutting machines.
2 roof-bolting machines.
2 hydraulic coal drills mounted on roof-bolting or cutting machines.
3 shuttle cars.
1 rock-dusting machine.
1 30-in belt conveyor
1 supply track.
1 car hoist.

Sequence of Development

Before any of the blocks (production places) are ready for retreat mining, the entries and cross cuts in development sections *A*, *B*, and *C* must be completed. The total development work necessary to complete four production panels (marked 1 in Fig. 10) therefore is:

Development section	Feet in unit			Number of units	Grand total (ft)
	Entry	Cross cut	Total		
A	19,280	6786	26,066	1	26,066
B	3920	1653	5573	2	11,146
C	15,400	4814	20,214	4	80,856
Total	38,600	13,253	51,853		118,068

While these four production panels are being retreated, additional development work should be in progress, so that when the four panels (shown as 1 in Fig. 10) are completed, four additional panels (shown as 2 in Fig. 10) will be completely developed. To accomplish this, the development in *D* and *E* must be completed as follows:

Development section	Feet in unit			Number of units	Grand total (ft)
	Entry	Cross cut	Total		
D	2880	1450	4330	2	8660
E	15,400	4814	20,214	4	80,856
Total	18,280	6,264	24,544		89,516

The plan of mining, the number of men in the crew, the type of equipment, and the amount of equipment are the same as those used in one of the mines studied[7]. At that mine the average advance in the group of entries (and the required cross cuts) was 171 ft/shift or 21.4 ft of development per entry per shift. At this rate of mining, it would require 691 work shifts of the unit crew to complete the 118,068 ft of development necessary for preparing four production panels for operation. To develop the next four production panels and each succeeding group on the same cross mains would require driving 89,516 ft of development work; at the rate of 171 ft/unit shift, this would require 524 unit shifts.

Development required to complete the first group of four production panels off the second cross mains would be *F*, *G*, and *I*, or a total of 138,140 ft. At the rate of 171 ft advance per shift, the development would require 808 unit shifts. For each succeeding group of four production panels on the second cross mains, the development would require 89,516 ft of development or 524 unit shifts of work.

Total Development Required

Thus, if a new mine is developed in accordance with the plan shown in Fig. 12 the following development would be required:

	Feet	Unit shifts
First set of 4 production places off first cross mains	118,068	691
First set of 4 production places off second cross mains	138,140	808
Each succeeding set of 4 places off either first or second cross mains	89,516	524

Development Required for Room-and-pillar Mining

For comparison, development of a room-and-pillar method of mining in thin coal beds has been calculated, using the same schematic plan. This plan is shown in Fig. 11.

Main and cross entries 25 ft wide are driven in groups of six on 50 ft centers. All entry cross cuts are 20 ft wide on 80-ft centers in the mains.

Panel entries 25 ft wide and on 50-ft centers are driven off the cross mains in groups of four. Leaving barrier pillars 200 ft wide to protect the main and cross main entries, rooms 40 ft wide on 55-ft centers are turned off the outside panel entries and driven 275 ft deep, with cross cuts 30 ft wide on 69-ft centers. It is assumed that the coal bed is 36 in. thick and that the entries are advanced in pairs a predetermined length, after which the succeeding pair in the panel or each succeeding pair in the mains and cross mains is advanced. Production mining will be done by driving rooms in pairs advancing on one side of the panel and retreating on the opposite side.

Crew Required

A development crew similar to that used at one of the mines studied will be assumed and comprises the following men:

Classification	Number of men	Classification	Number of men
Section foreman	1	Conveyor man	1
Loading-machine operator	1	Timberman	1
Loading-machine operator's helper	1	Supply man	1
Cutting-machine operator	1	Mechanic	1
Cutting-machine operator's helper	1	Boom man	1
		Total	10

This group of men will work as a team, and individual workers will be used interchangeably for any work to be done.

Equipment Required

The following mining equipment would be needed for this plan:

1 loading machine.
2 piggyback conveyors.
2 short-wall cutting machines.
2 chain conveyors.
1 mother chain conveyor.
1 elevating conveyor.
2 hand-held electric drills.
1 roof-bolting machine.
1 supply track.
1 car spotter.

Total Development Required

In this plan of development the following work is necessary to complete the first four production panels (shown as 1 in Fig. 10).

Development section	Feet in unit				Number of units	Grand total (ft)
	Entry	Cross cut	Room necks	Total		
A	8730	2500	—	11,230	1	11,230
B	3900	1125	—	5025	2	10,050
C	4350	1425	1076	6851	4	27,404
Total	16,980	5050	1076	23,106		48,684

While these four panels are being mined, additional development work should be in progress so that when the four panels (shown as 1 in Fig. 10) are mined out, the next four (shown as 2 in Fig. 10) should be completely developed and ready for mining. To complete the next four panels and each succeeding four off the same cross mains, the following development in *D* and *E* must be completed:

Development section	Feet in unit				Number of units	Grand total (ft)
	Entry	Cross cut	Room necks	Total		
D	4500	1350	—	5850	2	11,700
E	4350	1425	1076	6851	4	27,404
Total	8850	2775	1076	12,701		39,104

The plan of mining described, together with the number of men in the unit crew and using the same type and amount of equipment, is similar to that observed at one of the mines studied. The total advance in entries and cross cuts per unit shift using the described plan would be about 56 ft. At this rate of development, it would require 870 shifts for the 48,684 ft of development necessary to prepare four panels for production. To develop the next four panels on the same cross mains would require 39,104 ft of development or 700 unit shifts of work.

Development required to complete the first group of four production panels off the second cross mains would be *F*, *G*, and *H* or a total of 55,939 ft. At the rate of 56 ft advance per unit shift, the development would require 1000 shifts. Each succeeding group of panels off the second cross mains would require 39,104 ft of development or 700 shifts of work.

Summary and Comparison

Thus, if a new mine is developed in accordance with the plan shown in Fig. 11 the following development is required:

	Feet	Unit shifts
First set of four production places off first cross mains	48,684	870
First set of four production places off second cross mains	55,939	1000
Each succeeding set of four places off either first or second cross mains	39,104	700

Comparing this development with that for plan 1 (Fig. 12) it will be seen that many more feet of development is required under plan 1 than under plan 2. However, the total number of unit shifts required to develop four production panels is much less under plan 1. Not all of the difference between the feet of development necessary for each plan is caused by the different systems of mining. There is a vast difference in the thickness of the coal bed and, therefore, in the types of equipment used.

Results of Study

The methods and equipment used for development were studied at fourteen mines in West Virginia, Pennsylvania, Kentucky, and Ohio (*U.S. Bur. Mines Information Circular* 7813). These mines were selected to obtain operational data on various types of equipment and the mining methods used.

TABLE 6 SUMMARY OF BED THICKNESS, THICKNESS MINED, DEVELOPMENT DATA, AND COMPARATIVE YIELD FROM NONDEVELOPMENT UNITS[7]

Mine	Method of loading *	Thick-ness of bed (in.)	Thickness of bed mined (in.)	Average width of development entries (ft)	Number of entries in a set	Number of employees in development unit crew	Average yield from development per unit crew man per shift (tons)	Average yield from production per unit crew man per shift (tons)	Average advance of set of development entries per shift, includ-ing cross cuts (ft)†	Average advance of set of development entries per shift (ft)†
1	M	35	35	30	4	9	16.7	26.1	14.0	10.8
2	H	36	36	14 and 20	6	9	10.9	15.1	7.3	6.1
3	H	40	40	24	6	$12\frac{1}{2}$	12.5	14.5	8.2	5.4
4	M	42	42	20	3	$11\frac{1}{4}$	10.4	9.8	14.4	10.2
5	H	75	43	15	3	$8\frac{1}{2}$	13.5	13.5	6.4	5.0
6	M	46	46	15 and 17	4	12	25.3	30.9	28.4	22.0
7	C	46	46	27	9	7	42.8	42.3	8.7	6.1
8	M	73	55	20	6	12	42.5	43.5	23.3	18.2
9	M	59	59	18 and 20	5	$10\frac{1}{4}$ and 5	14.2	—	8.6	6.1
10	M	65	65 and 41	25 and 27	6	12	39.0	36.7	12.4	8.9
11	M	85	73	14	5	13	26.4	38.6	20.8	13.2
12	M	80	80	12 and 14	7	13	20.7	26.5	10.4	7.0
13	M	96	84	16	8	19	39.4	—	21.3	16.2
14	M	96	84	16	10	21	42.4	—	17.1	13.3

* M = mobile-loading equipment; H = hand loading; and C = continuous mining.
† Total feet of advance is obtained by multiplying by number of entries in a set.

Table 6 summarizes for the fourteen mines the data giving bed thickness, mining thickness, development data, and comparative yield from production places in some mines.

Conventional (Cyclic) Operation

"Conventional mining" is a cyclic process which includes the following operations.

(1) Cutting coal. A slot, usually horizontal, is cut for the length of the coal face. Usually this slot is cut at the bottom of the seam although coal is sometimes cut at an intermediate height or even at the top of the seam, and transverse (shear) cuts are sometimes used. This slot in the coal provides a free face to which the coal can break when it is shot.

(2) Drilling. Drills for boring shot holes may be hand held, post mounted, or mounted on a rubber-tired drill jumbo. Coal drills are usually of the auger type rather than the percussion type as used in hard rock mining operations and they may be powered by electricity, compressed air, or may be operated from the hydraulic system of a mining machine.

(3) Charging and shooting. Holes may be charged with an explosive or with a mechanical device which releases compressed air or carbon dioxide to break the coal.

(4) Loading coal. Formerly all broken coal was loaded by hand into rail-mounted cars. The efficiency of loading was improved when conveyors were introduced. However, most coal produced in the United States is now handled by mech-

TABLE 7. BITUMINOUS COAL AND LIGNITE MECHANICALLY LOADED UNDERGROUND IN THE UNITED STATES, BY TYPE OF LOADING EQUIPMENT[16]

Type of loading equipment	1960		1961	
	Net tons	Percentage of total	Net tons	Percentage of total
Mobile machines:				
Direct into mine cars	8,137,606	3.3	5,931,074	2.5
Onto conveyors	11,195,270	4.6	6,755,764	2.9
Into shuttle cars	142,775,484	58.1	132,446,554	56.3
Continuous-mining machines:				
Onto conveyors	10,474,509	4.3	11,031,679	4.7
Into shuttle cars	67,453,771	27.4	73,289,572	31.1
Scrapers and conveyors equipped with duckbills or other self-loading heads	1,232,019	0.5	1,032,009	0.4
Hand-loaded conveyors	4,517,166	1.8	4,863,270	2.1
Total mechanically loaded	245,785,775	100.0	235,349,922	100.0

anical loaders which load the broken coal onto conveyors or into shuttle cars which carry the coal from the working faces to the main transportation system for haulage to the surface (see Table 7).

(5) Installing roof support. The roof may be supported by bolts, or by wooden props, or by steel props, or posts and beams.

The cycle of operations described above, that is cutting, drilling, shooting, loading and installing support, is termed "cyclic" or "conventional" mining as opposed to the newer "continuous" or "non-cyclic" mining in which coal is ripped or cut from the seam mechanically and the drilling and shooting operations are eliminated.

When conventional mining methods are used the various work phases cannot be performed simultaneously in the same working place and it is necessary to operate from three to seven working places so that at least one place is available for coal loading at all times. Even if the schedule of the phases in a cycle is maintained, coal production ceases during the time required to move the loading equipment from one loading place to another.

Continuous (Non-cyclic) Operation

The continuous mining machine cuts or rips the coal from the face and loads it onto conveyors or into shuttle cars in a continuous operation. Thus the drilling and shooting cycles are eliminated, along with the necessity for working several headings in order to have available a heading in which loading can be in progress at all times.

Types of Continuous Mining Machines

Two basic types of continuous mining machines are available in U.S. practice. These may be classified according to the type of action which each uses in cutting and removing coal from the seam, as follows: (1) boring; (2) ripping.

The boring type machine employs cylinders, or arms, rotating on horizontal axes, on which are fixed cutting tools. These machines advance by cutting circular grooves in the face and breaking out the coal between these grooves. The coal drops to the floor where it is swept onto conveyors and conducted to the rear of the machine.

A boring-type machine may be equipped with two or more rotating heads, depending upon the ratio of the thickness of the seam to the width of the cut desired. Machines designed for use in thin seams must be equipped with a larger number of rotating elements in order to obtain a given width of cut.

Ripper type machines employ cutting teeth or picks which are either fastened to a series of endless chains travelling around rotating drums or which are fixed around the perimeters of disks or drums. In either case the teeth dig into the coal when the cutting head is advanced and rip coal from the seam. The broken coal may

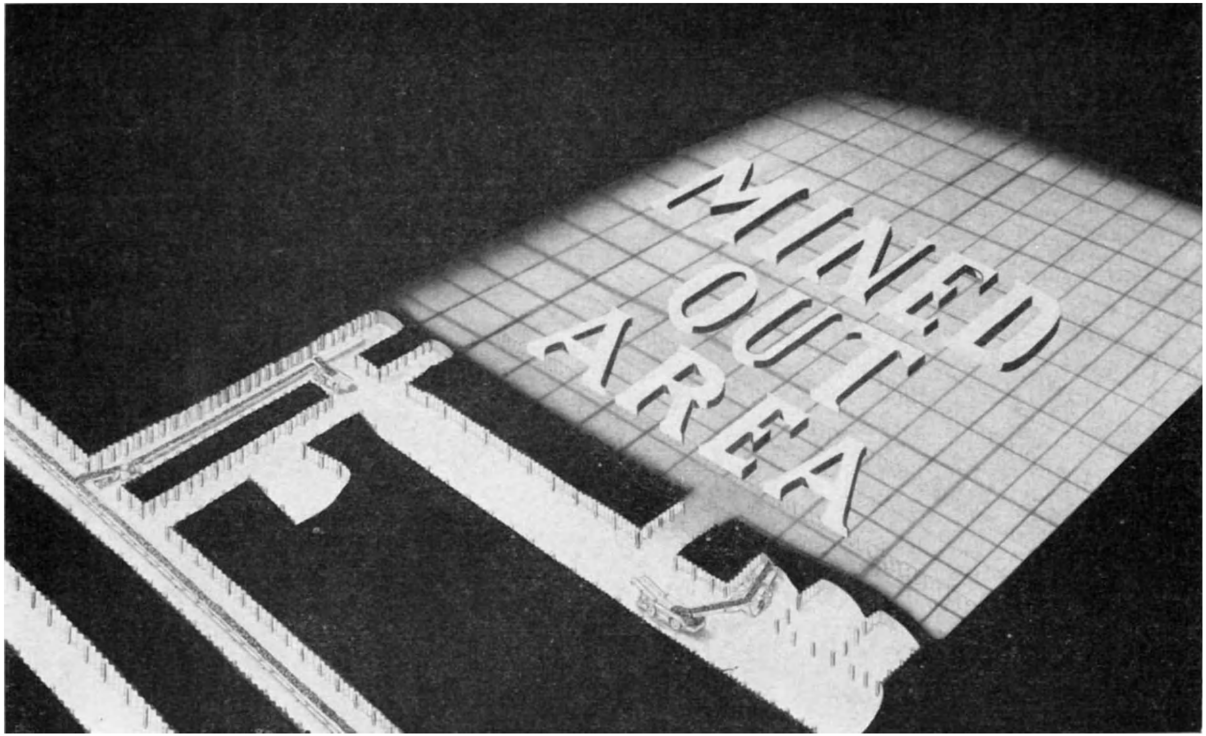

FIG. 13. Continuous miner loading into shuttle cars. (Joy Manufacturing Company.)

be transported away from the continuous miner by means of conveyors or by shuttle cars. In some instances the continuous miner deposits the coal on the floor and a following loader loads it into shuttle cars. This practice provides temporary storage, better clean up, and allows the mining machine to operate with a minimum of stoppage.

LONG-WALL MINING SYSTEMS

In the long-wall system coal is won from a continuously advancing face which may be from a few hundred feet to several thousand feet in length. The long-wall system has been little used in American coal mines because coal seams are generally mined at fairly shallow depths and favorable roof conditions allow the use of the more flexible pillar mining systems. However, several recent installations of fully mechanized long-wall systems have given very high production rates (see pp. 187, 239).

Handworked Long-wall

Originally all long-walls were handworked with the coal being holed (cut) and drilled by hand labor, shot, and then hand loaded into cars running on tracks located just behind the face for transport to the loader gate.

In firm coal it is necessary that a slot be cut back underneath the seam in order that the coal will settle and fracture. Such a slot may be cut by a miner with a hand pick. If the seam contains dirt bands the cutting may be done in these bands.

In exceptionally soft or friable coal seams, such as many of those in Germany, the coal may be won by breaking it from the face by means of hand-held pneumatic picks. The coal, after it is broken down, is then loaded onto conveyors which are located parallel to and just behind the long-wall face.

Mechanized Cyclic Mining

Long-wall mining operations in which coal is first cut and broken down from the face and then loaded onto conveyors are cyclic in nature. Usually only one shift in each 24 hr can be devoted to loading of coal. The remainder of the time is spent in cutting and breaking the coal and in moving and installing roof support, ripping roadways, and miscellaneous work.

The first step in the mechanization of long-wall operations was the introduction of the mechanical coal cutter for undercutting the face.

Next to be introduced was the conveyor installed parallel to the face, onto which coal was shoveled and which conveyed the coal to loading gates whence it was transported to the main haulage system.

Production of Coal in Great Britain[19]

Tables 8 and 9 give statistics of coal output from mechanized faces for the year 1961 (52 weeks ended December 30, 1961). The following notes indicate the basis on which statistics in the accompanying Tables are derived and the manner in

TABLE 8. SUMMARY OF MECHANIZED
(52 weeks ended December 30, 1961,

Type of machine		Number of machines in use at end of period 1961	1960
LONG-WALL METHODS			
1. Anderton Shearer Loader	– with powered supports	65	43
	– with normal supports	402	283
	Total Anderton Shearer Loader	467	326
2. A. B. Trepanner	– with powered supports	51	33
	– with normal supports	177	142
	Total A. B. Trepanner	228	175
3. **Plow Types**			
A. Plow, Rapid	– with powered supports	5	3
	– with normal supports	179	139
	Total Plow, Rapid	184	142
B. Huwood Slicer	– with powered supports	2	4
	– with normal supports	28	28
	Total Huwood Slicer	30	32
C. Scraper Box		20	23
D. M. & C. Stripper		3	3
E. Plow, Slow		2	4
	Totals 3A to 3E	239	204
4. **Cutter-chain Loading***			
(i) Conventional Conveying		92	102
(ii) A. F. Conveying		9	10
	Total 4(i) and 4(ii)	101	112
5. Meco-Moore		82	102
6. A. F. Conveyor (with prop-free front)†		53	53
7. Huwood Hydraulic Loader‡		67	51
8. Dosco Miner		11	10
9. Multi-jib Cutter Loader		33	38
10. Mawco Cutter Loader	– with powered supports	5	3
	– with normal supports	13	7
	Total Mawco Cutter Loader	18	10
11. Dranyam Cutter Loader	– with powered supports	1	1
	– with normal supports	2	2
	Total Dranyam Cutter Loader	3	3
12. Gloster Getter		2	9
13. Huwood Mechanical Loader§		5	5
14. Trepan Shearer		9	1
15. Experimental and Other Applications		23	24
16. Totals Long-wall Methods (1 to 15)**		1341	1123

COALFACE OUTPUT [19]

with comparative statistics for 1960)

Pithead coal loaded during period		Man-shifts/thousand tons pithead (MST face) (including shotfirers)		Average advance per 24 hr				Average advance per machine shift			
1961	1960	1961	1960	1961		1960		1961		1960	
tons	tons			ft	in.	ft	in.	ft	in.	ft	in.
6,141,617	3,223,601	112	118	5	3	4	9	2	6	2	6
31,897,968	26,649,282	152	146	4	6	4	4	2	8	2	8
38,039,585	29,872,883	146	143	4	8	4	4	2	7	2	8
6,195,979	2,742,380	100	106	5	7	5	2	2	8	2	8
16,760,958	12,362,763	132	130	5	1	5	0	2	9	2	9
22,956,937	15,105,143	123	125	5	3	5	0	2	9	2	9
255,201	101,253	156	166	3	6	3	5	2	3	2	0
10,365,626	6,595,050	180	181	3	8	3	7	2	10	2	11
10,620,827	6,696,303	180	181	3	8	3	7	2	10	2	10
148,548	186,100	184	178	4	4	3	5	1	10	1	10
3,063,029	2,459,239	135	138	4	1	4	0	2	3	2	4
3,211,577	2,645,339	137	141	4	1	4	0	2	3	2	3
415,326	493,914	297	281	3	3	3	4	2	9	3	0
280,018	276,618	102	110	3	10	4	9	3	10	4	9
216,976	298,943	212	218	3	7	3	2	2	2	2	7
14,744,724	10,411,117	173	175	—		—		—		—	
3,025,596	3,887,069	237	227	3	10	3	11	3	2	3	6
640,271	785,285	177	183	3	3	3	2	3	0	2	7
3,665,867	4,672,354	226	219	3	9	3	10	3	2	3	5
7,760,624	9,501,426	144	140	4	7	4	8	2	11	2	10
3,953,933	3,777,018	197	206	—		—		—		—	
2,357,052	2,418,476	197	180	4	0	3	11	3	3	3	3
1,048,915	906,284	119	99	4	7	4	7	4	0	4	1
782,399	768,047	266	251	4	0	3	10	3	5	3	3
386,290	158,833	128	117	6	1	5	5	3	1	2	10
577,665	207,573	157	158	4	11	4	9	2	8	2	9
963,955	366,406	145	140	5	4	5	0	2	10	2	9
22,228	2799	240	232	2	1	1	10	2	0	1	10
281,840	186,935	115	139	4	3	3	9	2	3	2	3
304,068	189,734	124	140	3	10	3	7	2	2	2	2
286,389	585,905	170	167	3	4	3	10	1	10	2	1
225,746	248,454	168	171	4	10	4	7	3	11	4	0
361,185	92,398	138	122	4	0	3	10	2	8	2	4
1,080,229	684,686	202	206	3	11	3	9	3	10	2	9
98,531,608	79,600,331	152	153	4	5	4	3	2	9	2	10

TABLE 8.

Type of machine	Number of machines in use at end of period	
	1961	1960
OTHER METHODS (inc. Room and Pillar)		
17. Gathering Arm Loader	102	73
18. Joy Continuous Miner	41	33
19. Cutter-chain Loading	29	17
20. Duckbill	16	13
21. Experimental and Other Applications	6	6
22. Totals Other Methods (17 to 21)	194	142
Totals Long-wall and Other Methods (16 and 22)	1535	1265

* Cutter-chain loading with or without flights includes any adaptation such as Lambton flights where a coal-cutter jib is modified for the purpose of loading.

† These figures exclude power-loaded coal. Armoured flexible conveyors (used in conjunction with power-loaders) which numbered 1006 at the end of the year 1961, conveyed an additional 80, 086, 842 tons.

which they have been evaluated. All figures are based on pithead tonnage, as distinct from saleable tonnage.

Table 8 gives a summary of mechanized coal face output by the various types of machine in use in both "long-wall" and "other methods" of working during 1961, and comparative figures are also given for 1960.

The term "mechanized output" includes the coal produced by all coal face machinery which either loads prepared coal or cuts and loads coal simultaneously. It also includes all coal obtained (including that filled by hand) where an armoured flexible conveyor is in use without a separate power-loader and where, at the completion of the shift or cycle, there are no props between the conveyor and the face (i.e. where an armoured flexible conveyor is used on a prop-free front).

Type of Machine. Machines are listed in two groups, namely for "long-wall" and "other methods" (the latter including room-and-pillar) and a grand total is given for all types of mechanized working.

Pithead Coal Loaded. This is the tonnage won by each class of machine on a pithead basis (i.e. before preparation for the market).

Man-shifts per Thousand Tons (MST Face). This is the number of man-shifts worked at the face for the production of each thousand tons of pithead output.

Average Advance per 24 hr and per Machine Shift. These figures are derived by dividing the total advance during the year by the total number of days worked and by the total number of machine shifts worked respectively.

(cont.)

Pithead coal loaded during period		Man-shifts/thousand tons pithead (MST face) (including shotfirers)		Average advance per 24 hr		Average advance per machine shift	
1961	1960	1961	1960	1961	1960	1961	1960
1,997,031	1,817,081	96	101	—	—	—	—
1,548,040	1,249,397	53	51	—	—	—	—
402,911	191,991	110	120	—	—	—	—
98,524	89,312	205	194	—	—	—	—
155,820	81,727	189	169	—	—	—	—
4,202,326	3,429,508	87	88	—	—	—	—
102,733,934	83,029,839	149	151	—	—	—	—

‡ Formally known as Huwood Ski-Hi.
§ Formally known as Huwood Loader.
** The figures given as average advance per 24 hr and per machine shift exclude armoured flexible conveyors on a prop-free front.

Armoured flexible conveyors operated on a prop-free front without the use of a separate power-loader are excluded.

Table 9 gives some additional statistics for each type of long-wall power-loader, taking the machines in use at any time during the period; comparative figures are also given for the year 1960.

Average Output per 24 hr and per Machine Shift. These figures are derived by dividing the total output by the total number of days worked and by the total number of shifts worked respectively. (Many machines work more than one shift per 24 hr.)

Average Length of Face per Project. The average length of face is derived by dividing the total length of all faces (including stable holes) which were worked during the year by the total number of projects at work during the year.

Average Area Extracted per Machine Shift and per Man-shift. These figures are calculated by multiplying the length of faces (including stable holes) by the advance of each individual project, adding the results and dividing by the total number of machine shifts and by the total man-shifts worked respectively.

Weighted Average Seam Thickness Extracted. This is derived by multiplying the tonnage by the seam thickness extracted for each project, adding the results and then dividing by the total output.

TABLE 9. ADDITIONAL DETAIL FOR

52 weeks ended

Type of machine		Average output per 24 hr per machine	Average output per machine shift	Average length of face per project
		tons	tons	yd
Anderton Shearer Loader	– with powered supports	478	230	201
	– with normal supports	414	241	182
	Total Anderton Shearer Ldr. (all supports)	423	239	185
A. B. Trepanner	– with powered supports	554	266	226
	– with normal supports	473	255	217
	Total A. B. Trepanner (all supports)	492	258	219
Plow, Rapid	– with powered supports	278	177	190
	– with normal supports	284	219	164
	Total Plow Rapid (all supports)	284	218	164
Huwood Slicer	– with powered supports	330	138	161
	– with normal supports	504	284	206
	Total Huwood Slicer (all supports)	492	271	202
Scraper Box		81	68	101
M. &. C. Stripper		377	377	186
Plow, Slow		341	207	142
Cutter-chain Loading	(i) Conventional Conveying	147	122	120
	(ii) A. F. Conveying	247	227	135
	Total Cutter-chain Loading (i) and (ii)	158	133	122
Meco-Moore		381	243	158
Huwood Hydraulic Loader*		172	140	131
Dosco Miner		456	399	146
Multi-jib Cutter Loader		93	80	117
Mawco Cutter Loader	– with powered supports	370	187	159
	– with normal supports	301	164	173
	Total Mawco Cutter Loader (all supports)	326	173	168
Dranyam Cutter Loader	– with powered supports	152	144	167
	– with normal supports	486	255	198
	Total Dranyam Cutter Loader (all supports)	419	242	190

LONGWALL POWER-LOADERS[19]

December 30, 1961				Comparative statistics for 1960						
Average area extracted		Weighted average seam thickness extracted		Average output per 24 hr per machine	Average output per machine shift	Average length of face per project	Average area extracted		Weighted average seam thickness extracted	
Per machine shift	Per man-shift						Per machine shift	Per man-shift		
ft²	ft²	ft	in.	tons	tons	yd	ft²	ft²	ft	in.
1521	59	4	0	420	216	197	1485	58	4	0
1466	40	4	6	396	245	186	1523	43	4	4
1475	42	4	5	398	242	188	1518	44	4	4
1851	70	3	11	510	263	223	1838	66	3	10
1772	53	3	11	449	248	213	1757	55	3	10
1792	56	3	11	459	251	215	1771	56	3	10
1134	41	4	2	216	126	174	889	43	4	0
1430	36	4	2	274	222	164	1453	36	4	0
1421	36	4	2	272	219	164	1438	36	4	0
759	30	5	0	345	181	189	959	30	5	2
1319	35	5	9	459	262	198	1275	35	5	8
1268	34	5	9	448	254	197	1244	35	5	8
848	42	2	4	89	80	101	900	40	2	7
2150	56	4	8	384	381	157	2264	54	4	4
854	19	6	10	259	209	164	1251	27	5	6
1113	39	3	1	157	140	128	1299	41	3	3
1327	33	4	6	267	218	158	1291	32	4	8
1135	38	3	4	169	149	131	1298	40	3	6
1376	39	4	9	388	234	167	1341	41	4	9
1264	46	3	3	164	138	129	1181	48	3	5
1779	38	6	3	484	425	145	1776	42	6	3
1070	50	2	2	93	79	123	1032	52	2	3
1486	62	3	5	386	199	181	1522	65	3	5
1341	52	3	6	317	183	164	1373	47	3	6
1395	56	3	6	343	190	169	1435	54	3	6
1006	29	4	2	97	97	170	950	43	2	9
1290	44	5	8	411	244	193	1300	38	5	5
1255	42	5	7	392	239	192	1287	38	5	5

TABLE 9.

Type of machine	Average output per 24 hr per machine (52 weeks ended)	Average output per machine shift	Average length of face per project
	tons	tons	yd
Gloster Getter	215	119	159
Huwood Mechanical Loader†	193	158	192
Trepan Shearer	386	260	190
Experimental and Other Applications	194	140	151
Great Britain – Average	353	222	171

* Formally known as Huwood Ski-Hi.

† Formally known as Huwood Loader.

Operating Cycle at a Long-wall Face

The order of operations involved in mining an undercut long-wall face is as follows:

First shift: load broken coal onto the face conveyor.

Second shift: move the conveyor over; move forward the chocks or breaker props, and withdraw back roof supports; rip roadway roofs and build packs.

Third shift: cut coal; drill and shoot the coal if undercutting does not cause it to settle and break.

The conveyor is shifted ahead each time by a distance equal to the width of the slice of coal removed from the face. A space of about 3 ft must be left between the front row of props and the face in which the coal cutter can travel.

An alternate cycle of operations could be:

First shift: load coal onto the conveyor and draw timber.

Second shift: cut coal and drill shot holes.

Third shift: shift the conveyor; rip roadway roofs, build packs; fire the shots.

Coal Cutting

About 90 per cent of the coal mined in Great Britain is machine cut and almost 50 per cent is power loaded. Mechanical cutters can be efficiently applied to most seams but in a few instances conditions may not warrant their use. Such conditions include thick friable seams where coal can be easily broken by hand and seams which are offset by numerous faults which make it difficult to traverse the face with a mechanical cutter.

(*cont.*)

December 30, 1961			Comparative statistics for 1960					
Average area extracted		Weighted average seam thickness extracted	Average output per 24 hr per machine	Average output per machine shift	Average length of face per project	Average area extracted		Weighted average seam thickness extracted
Per machine shift	Per man-shift					Per machine shift	Per man-shift	
ft^2	ft^2	ft in.	tons	tons	yd	ft^2	ft^2	ft in.
860	42	3 11	244	132	156	953	43	3 9
1712	65	2 9	169	148	211	1588	63	2 8
1656	46	4 6	343	210	203	1568	61	3 9
1226	43	3 9	203	149	151	1235	40	4 1
1467	44	4 3	327	218	170	1455	44	4 2

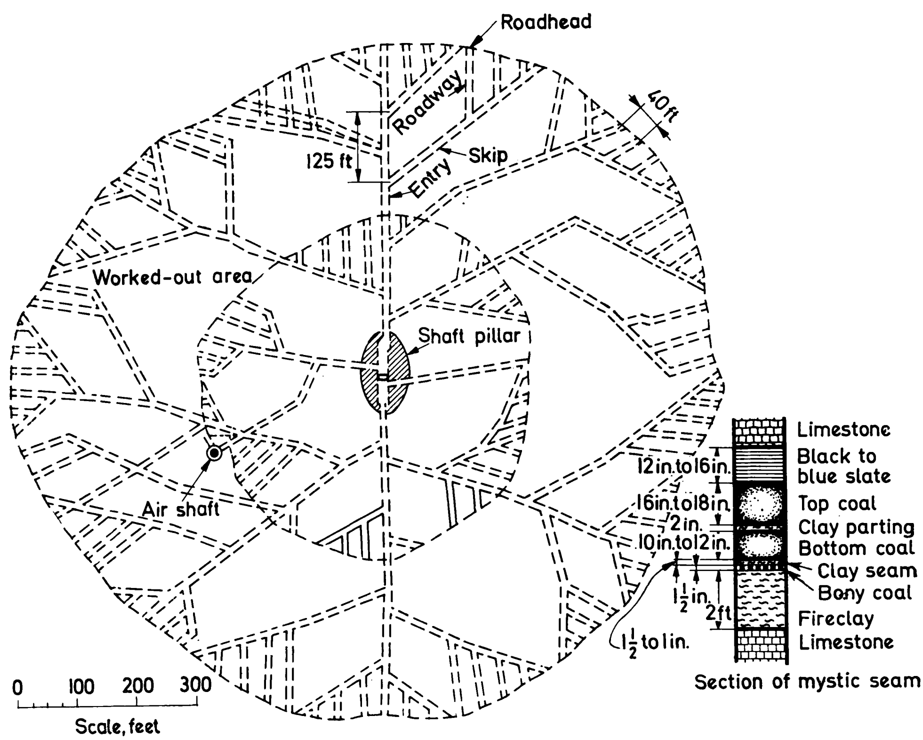

FIG. 14. Typical handworked long-wall mine in the Centerville (Iowa) District showing radial advance from the central shaft; typical of some midwestern long-wall operations in the 1930's.[18]

Most often the coal is bottom cut; however, if a seam contains a dirt band it may be desirable to make the cut in this band. If the floor is soft and tends to lift and to close the cut it may be necessary to make the cut near the top of the seam.

Machines for Long-wall Mining

Long-wall Coal Cutters

A number of manufacturers produce long-wall coal cutters in a variety of heights and powered according to the hardness of the coal to be cut.

These machines may be powered by electricity or by compressed air.

The height of the jib on a coal cutter may be varied by using different components in the turret head assembly. Some machines are fitted with hydraulic turret jibs which allow some variation of the height of cut to be made while cutting operations are in progress.

Long-wall Power Loaders

Power loaders are designed to plow or to sweep the broken coal from the mine floor onto the conveyor which runs along parallel to, and a few feet from, the coal face.

The simplest power loader is a long-wall coal cutter which has been converted for loading by attaching loading flights to the cutting chain.

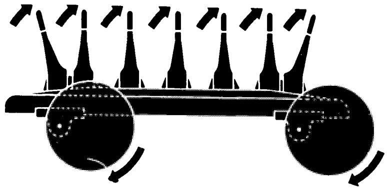

FIG. 15. Action of the Huwood Loader (for loading on long-wall faces). (Hugh Wood & Co. Ltd.)

The Huwood loader is built especially for loading coal and is equipped with two hydraulic jack anchor posts which can be tightened against floor and roof and against which the loader pushes with hydraulic cylinders to propel itself. The front portion of this loader is equipped with reciprocating arms which sweep the coal away from the face and toward the face conveyor.

In soft or friable seams undercutting may be sufficient to cause the coal to fall and to break into sizes suitable for loading while in hard seams it may be necessary to shoot the coal to break it from the seam and reduce it to sizes which can be loaded.

Mechanized Non-cyclic Mining

In pillar mining systems a machine which eliminates the drill and shoot cycles of the coal getting process is known as a continuous miner.

In British long-wall mining systems certain machines are in use which rip or cut the coal from the face and load it onto face conveyors but which cut in only one

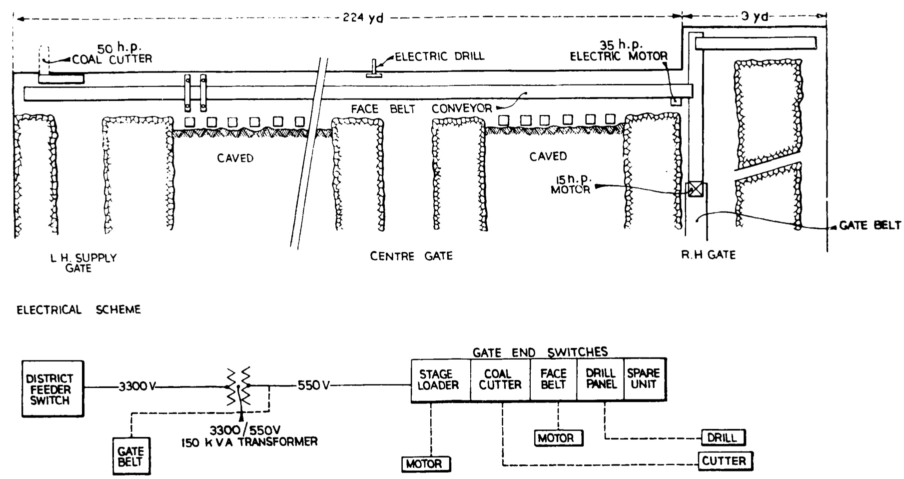

FIG. 16. Conventional long-wall face layout.[14]

direction and therefore must be flitted back idle to the starting point. In British practice such machines are considered as cyclic and only those machines which cut and/or load in both directions are classified as non-cyclic.

In this work, however, the author has classified machines for pillar mining systems as non-cyclic because they eliminate the drilling and shooting cycle of coal getting; therefore long-wall machines which eliminate the drilling and shooting cycles will also be classified herein as non-cyclic or continuous mining machines.

Machines for continuous mining on long-wall faces are usually designed to cut or rip a relatively narrow slice of coal from the face during each pass along the face. This allows a "prop-free front" to be maintained; that is the last row of props is placed about 3 or 4 ft from the face and the roof between the props and the face is supported by bars cantilevered from the last row of props.

By removing only a narrow slice of coal at each pass the roof is maintained in much better structural condition and fracturing of the roof is much less extensive that when a wide web of coal is removed from the face.

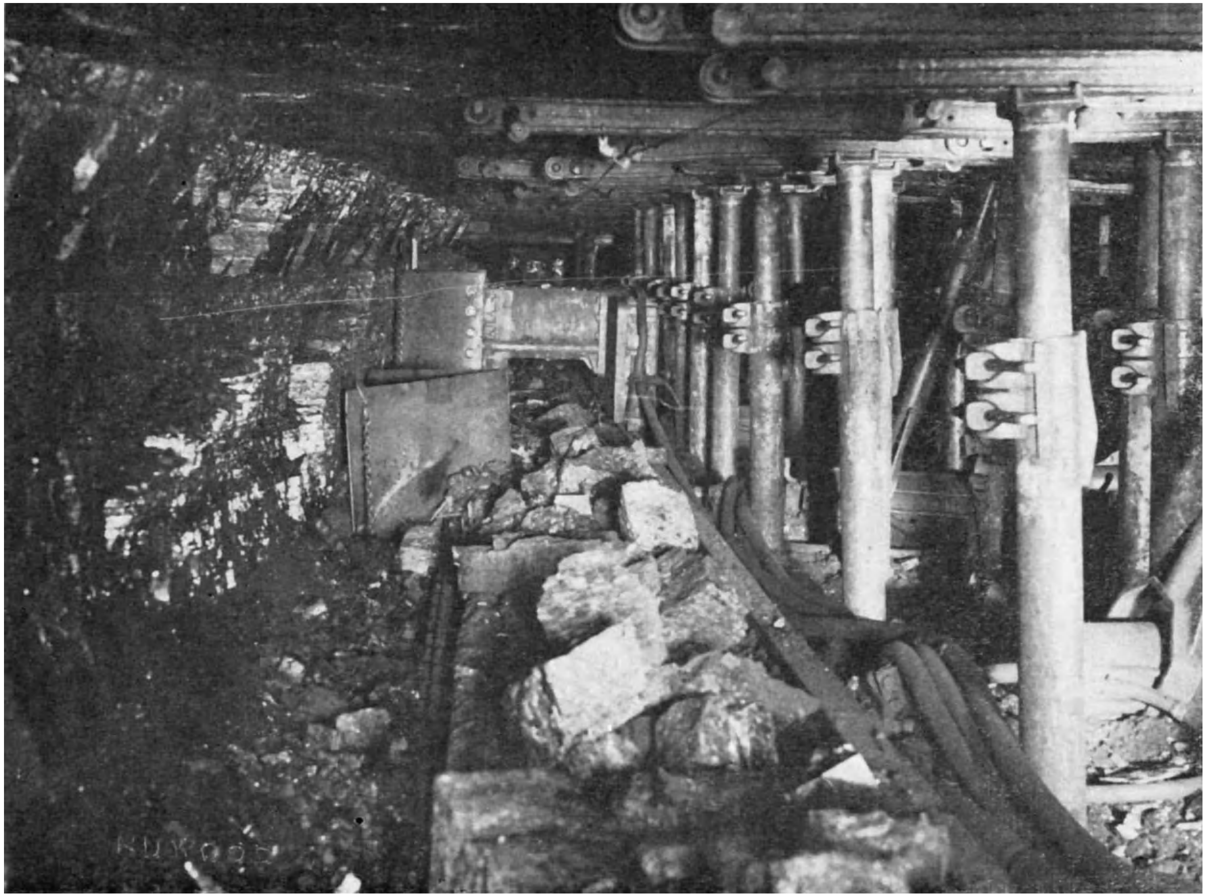

FIG. 17. Huwood slicer-loader and T.C.R. props in a Derbyshire Colliery. (Hugh Wood & Co. Ltd.)

Types of Non-cyclic Miners

Continuous long-wall miners fall into one of three classes: (1) trepanning or boring machines; (2) ripping or shearing machines; (3) coal planers or plows.

The A. B. Trepanner is a boring type machine which attacks the coal with a cylinder on the advancing edge of which are set cutting picks rotating about an axis which is parallel to the coal face. Cutting picks are fixed to the edge of this cylinder so that the action is somewhat similar to that of a boring machine. More than 22 million tons of coal was produced with these machines in Great Britain in 1961.

The Anderton Shearer-loader is also a popular machine for continuous mining on long-walls and these machines produced about 38 million tons of coal in Great Britain in 1961. The machine is equipped with a toothed drum which rotates about an axis which is perpendicular to the coal face and rips coal from the face.

The Joy long-wall Buttock Miner is a crawler-mounted machine which is equipped with twin trepanners to remove a web of coal about 5 ft wide from the face.

Coal plows or planers are simple slide-mounted devices equipped with wedge blades and designed to be pulled along the coal face and to peel from the face a layer of coal a few inches thick. The broken coal is deflected or plowed onto the armored face conveyor on which the plough is mounted.

Plows are usually classified as "high speed", "slow speed", or scraper boxes. Generally the "slow speed" plows are used in those coal beds where roof and floor partings are poor, or where the coal is hard or tough, or where conditions along the face are not uniform. These plows usually operate at speeds of about 10 ft/min to 50 ft/min.

"High speed" plows are suitable where the coal is friable and easily got. They operate in ranges up to 100 ft/min, and take off a slice of coal which is only a few inches thick during each passage along the face.

Activated Plows

Some types of coal plows are equipped with vibrating picks to aid in loosening the coal from the harder seams, and some plows are equipped with means for self propulsion so that they do not have to be pulled along the face by means of a chain.

The Samson Stripper is a self-propelled plow which wedges or peels a 2-ft slice of coal from the face by means of a wedge-shaped cutting head which is advanced into the coal step by step. The main frame of this machine anchors itself by means of hydraulic jacks which expand against the floor and roof. Horizontal hydraulic cylinders then act against these jacks to advance the wedge head into the coal.

The Huwood Slicer moves along the face on the conveyor and loosens coal from the face by means of a wedge head whose forward edge is equipped with vibrating picks.

Scraper Boxes

Scraper boxes are special types of plows which are attached to a tow chain or cable and which are dragged back and forth along the face by a winch. They scrape the coal, by stages, to a loading gate.

To obtain cutting action scraper boxes are held against the face by a guide rail which is pushed forward by pneumatic rams. Scraper boxes are most likely to be successful in thin seams and in coal which is soft or friable and is easily gotten.

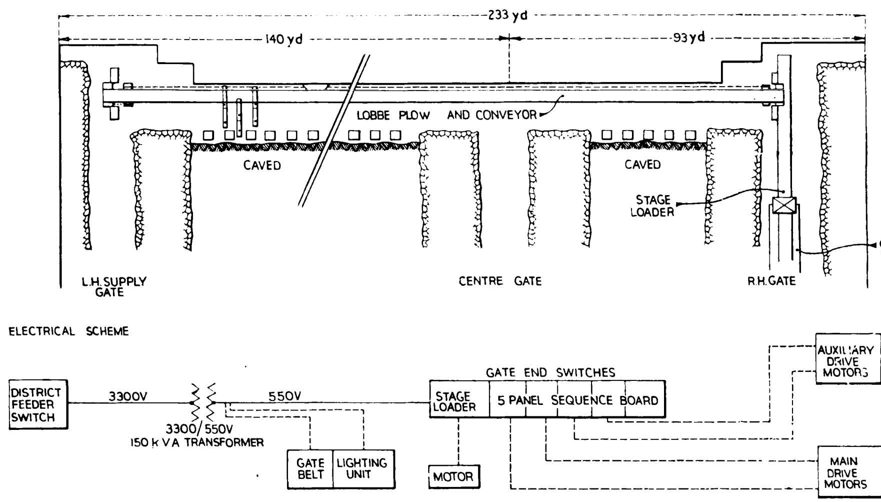

FIG. 18. A typical Lobbe plow face layout.[14]

PILLAR MINING SYSTEMS

Coal Haulage

Haulage refers to the process of moving the coal mined at the face to the mine tipple. It does not include the transportation of men and supplies that use the same equipment. For convenience the transportation system can be divided into four parts:[8] Portal haulage (shaft, slope, and drift); mainline haulage; intermediate haulage; and face haulage. These subdivisions are not necessarily made in all transportation systems as the intermediate haulage may be the mainline and the portal haulage may be a continuation of the mainline.

The portal haulage used depends on the type of mine opening. In vertical shafts, hoisting of loaded mine cars on cages to the surface is being eliminated in favor of skips that are loaded automatically from a bottom dump. Slopes sunk on dips not exceeding 17° are equipped with belts in most newly developed mines and some older mines. Where slope belts are used a supplementary track system transports men and supplies. In drift mines this haulage is a continuation of the mainline haulage whether it is done by locomotives, belts, or even shuttle cars.

Mainline haulage is the movement of mined coal from gathering stations in the mine to the portal and usually is accomplished by locomotives or by belts fed by other conveyors. Locomotives up to 50 tons operated singly or in tandem pull trips of as many as twenty-five cars ranging to 20 ton capacity. Track rails weighing up to 120 lb./yd are either bolted or welded and are laid on wood ties in a ballasted road bed. Locomotive operators are in constant touch with dispatchers and other locomotive operators by two-way radio. Block signals and automatic switch-throws are operated from the cab of the locomotive without stopping. Speeds of 15 miles, or more, per hr are maintained by the mainline trips. Belts used in main-line transportation show a trend away from the rigid frame construction to the rope type.

TABLE 10. NUMBER OF UNDERGROUND BITUMINOUS COAL AND LIGNITE MINES AND NUMBER OF HAULAGE UNITS IN USE IN THE UNITED STATES, IN SELECTED YEARS[16]*

Year	Underground mines	Locomotives				Rope-haulage units			Shuttle cars			Gathering and haulage conveyors	Animals
		Trolley	Battery	Other types	Total	Portable	Stationary	Total	Cable reel	Battery	Total		
1924	7352	12,765†	1515	443	14,723	‡	‡	649	‡	‡	‡	‡	36,352
1946	5888	14,110	1001	110	15,231	4084	1009	5093	‡	‡	‡	457	10,185
1948	7108	14,617	904	74	15,595	3886	1044	4930	‡	‡	‡	755	10,834
1949	6798	14,090	928	59	15,077	3904	1073	4977	2144	623	2767	860	10,313
1950	7559	13,822	949	62	14,833	4225	1037	5262	2872	512	3294	1013	10,033
1951	6225	13,327	900	51	14,278	3875	916	4791	3191	567	3758	1094	7478
1952	5632	12,545	812	41	13,398	3584	852	4436	3382	462	3844	1066	6555
1953	5034	11,311	678	45	12,034	2838	727	3565	3797	425	4222	1042	5354
1954	4653	10,155	762	38	10,955	1926	781	2707	4400	431	4831	1081	5409
1955	6035	9538	658	40	10,236	1327	577	1904	4375	239	4614	1002	6440
1956	6542	9445	861	102	10,408	1420	575	1995	4757	257	5014	1114	6097
1957	6512	8997	898	138	10,033	1214	616	1830	5129	257	5386	1233	5054
1958	6319	8057	920	138	9115	926	538	1464	4871	259	5130	1235	4678
1959	5815	7263	949	137	8349	900	504	1404	4795	255	5050	1416	4063
1960	5989	6922	946	173	8041	892	510	1402	4722	236	4958	1566	3503
1961	5843	6362	583	162	7107	§	§	§	4687	182	4869	1635	§

* Exclusive of lignite and Virginia semianthracite mines in 1946, 1948, and 1949.
† Includes combination trolley and battery locomotives.
‡ Data not available.
§ Canvass discontinued.

Intermediate haulage is the transportation of mined coal from the face haulage to that point where it is accessible to the mainline. It is accomplished by conveyors, belts, or locomotives and mine cars. Where gathering belts or conveyors are used, the end of this phase of haulage is at the transfer point of the coal to the mine cars or to a mainline belt. With locomotives and mine cars, it would end at the partings where the loads are assembled into trips for the mainline locomotives.

Face Haulage

Face haulage means the transportation of mined coal from the working face to an intermediate haulage. It is accomplished by shuttle cars, conveyors, locomotives, and mine cars, or by combinations of such equipment. Table 11 summarizes the face-haulage systems used at thirteen mines studied by personnel of the U.S. Bureau of Mines.[8]

Shuttle Cars

Shuttle cars were used at eight of the mines studied and received the coal directly from the continuous-mining machine or loading machine; from a surge car behind the face equipment; or from a loading machine operated behind a continuouse mining machine. All shuttle cars were the cable type, and where more than on- was required for the face equipment, separate travelways were provided.

Conveyors

Two types of conveyors were used in face haulage in the mines studied: extensible belt, and chain. The extensible belts were used in coal more than 50 in. thick, and chain conveyors were employed in thin coal. The extensible belts received the mined coal from continuous-mining machines over a bridge between the units. Equipment out by the extensible belt was not considered face haulage.

Because of the low headroom required chain conveyors are used extensively in thin coal. The face equipment can load directly on to the conveyor or, as is being done more frequently, to a bridge conveyor to the chain conveyor. Equipment out by the chain conveyor discharge was not considered face haulage.

Locomotives and Cars

Because of the trend to trackless type loading equipment, locomotive–car face haulage is on the decline. However, where mine cars of 6–10 ton capacity can be loaded at the face and where the car change can be held to a short haul, it is usable with some types of loading machines.

TABLE 11. FACE EQUIPMENT AND TRANSPORTATION TO INTERMEDIATE HAULAGE(8)

Mine No. and unit	Face mining equipment	Sequence of transfer from loading equipment to intermediate haulage	Intermediate haulage
1(a)	Auger C.M.*	Bridge conveyor — chain conveyor	24-in. belt to cars
2(a)	do.	do.	30-in. belt to storage bank
3(a)	M.L.†	do.	30-in. belt to 3-ton cars
3(b)	do.	Loading machine — chain conveyor	do.
4(a)	Ripper C.M.‡	24-in. extensible belt	30-in. belt to 5-ton cars
4(b)	do.	Surge car — shuttle car	do.
4(c)	do.	do.	do.
5(a)	Boring C.M.§	30-in. extensible belt	do.
6(a)	Ripper C.M.‡	Loading machine — shuttle car	8-ton cars
6(b)	do.	do.	30-in. belt
6(c)	M.L.†	Shuttle car	8-ton cars
6(d)	T.L.**	Locomotive and mine car	do.
7(a)	Ripper C.M.‡	24-in. extensible belt	6-ton cars
7(b)	M.L.†	Shuttle cars	do.
8(a)	Ripper C.M.‡	Loading machine — shuttle car	10-ton cars
8(b)	M.L.†	Shuttle car	do.
9(a)	T.L.**	Locomotive and mine car	do.
9(b)	do.	do.	do.
9(c)	do.	do.	do.
10(a)	M.L.†	Shuttle cars	do.
10(b)	T.L.**	Locomotive and mine car	do.
11(a)	Boring C.M.††	36-in. extensible belt	42-in. belt
11(b)	M.L.†	Shuttle cars	do.
12(a)	Ripper C.M.‡	30-in. extensible belt	30-in. belt
12(b)	do.	Shuttle cars	do.
12(c)	M.L.†	do.	do.
13(a)	do.	do.	Shuttle cars

* Shortwall auger continuous-mining machine.
† Mobile-loading units.
‡ Ripping action continuous-mining machine.
§ Boring action continuous-mining machine with a series of rotating arms.
** Track-loading units.
†† Boring action continuous-mining machine with two rotating arms.

Results of Study(8)

Four general types of face haulage were studied: chain conveyors, extensible belts, shuttle cars, and mine cars. There were variations in equipment combinations used ahead of and behind the face haulage. Table 12 summarizes the results for each unit and shows the number of tons per crew and per man-shift.

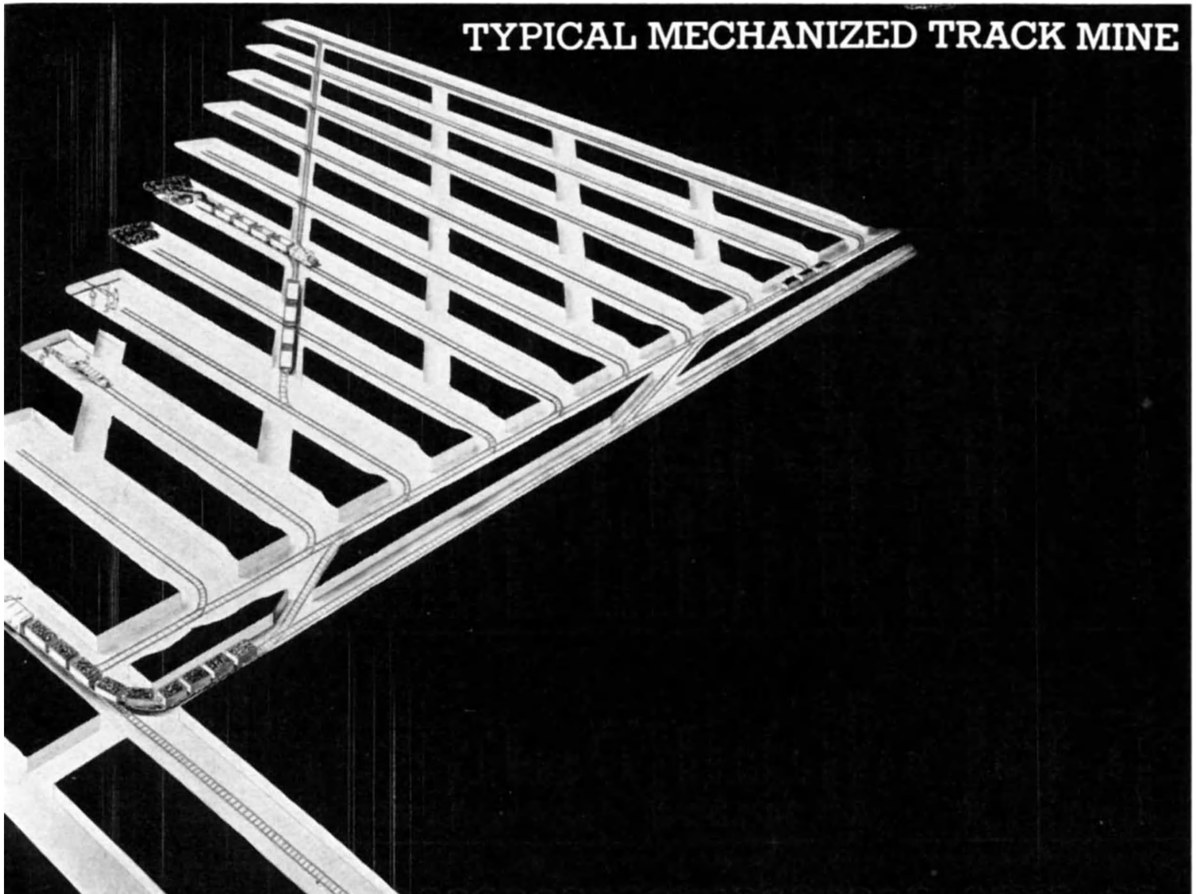

FIG. 19. Typical mechanized track mine. (Joy Manufacturing Company.)

FIG. 20. Extensible belt and continuous miner. (Joy Manufacturing Company.)

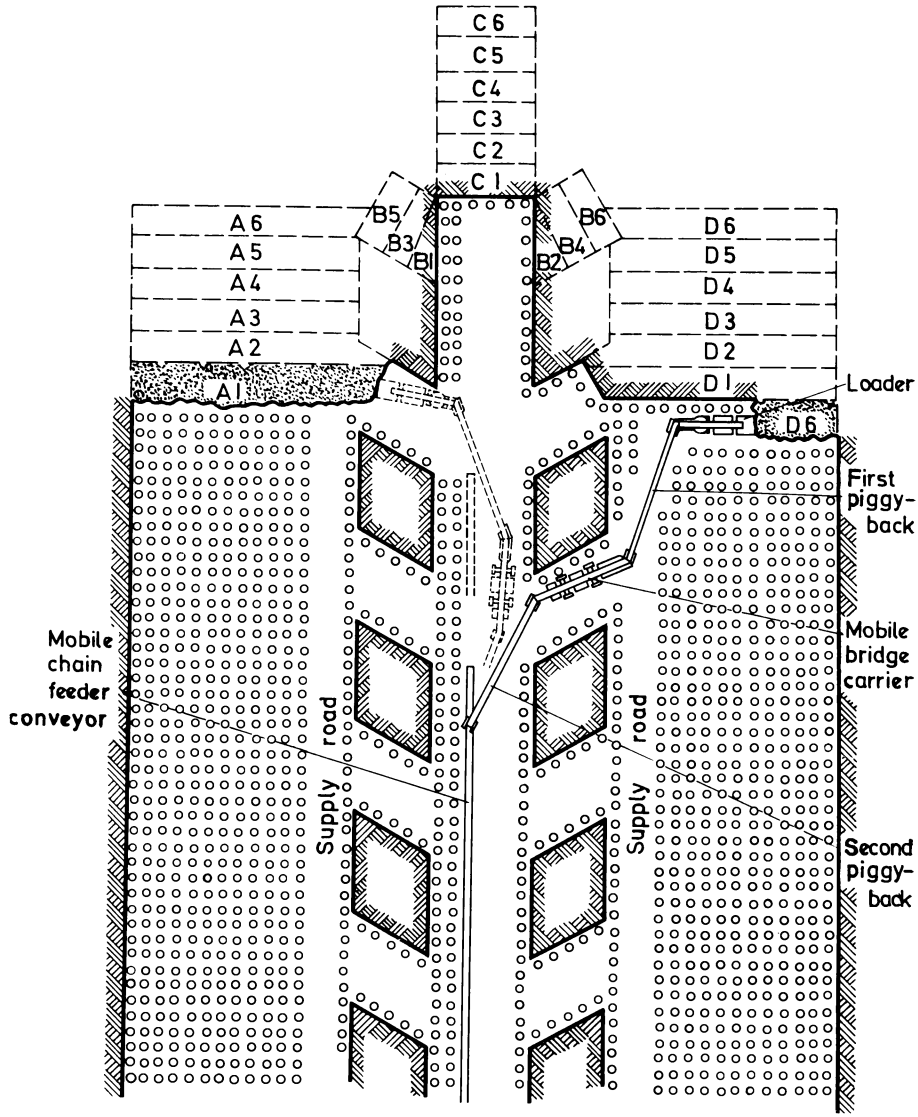

FIG. 21. Room development with a "full dimension" mining system.

The loading cycle starts in No. 3 Room *D*1, moves to No. 2 Right Breakthrough *B*2, to No. 2 Face *C*1, to No. 2 Left Breakthrough *B*1, to No. 1 Face *A*1. This continues for three cycles at which time the breakthroughs are completed. Three more cuts are taken at *D*4, *D*5, *D*6, *C*4, *C*5, *C*6, *A*4, *A*5, and *A*6, before the next extension. (The Long Company.)

In comparing the four general systems of face haulage on a basis of tons per crew man per shift, the results were:

Type of face haulage	Tons per man
Extensible belts	53
Shuttle cars	30
Mine cars	30
Chain conveyors	21

As a group, extensible belt face haulage had the best average of production per crew man-shift. This might be expected for several reasons: each was behind a continuous-mining machine; the equipment was newer than most other face haulage equipment; the delays attributable to face haulage generally were less than with other types; and the coalbeds in which they were operated were 52 in. or more in thickness.

Although the average production per crew man-shift was the same for shuttle-car haulage and mine-car haulage, shuttle-car haulage is more widely used and generally should give better results. Supervision and planning are important factors in the success or failure of these types of face haulage. The minimum thickness of coal mined where mine-car haulage was studied was 71 in., whereas with shuttle cars, six of the twelve units were operated in coal of less thickness.

All shuttle cars were cable-reel type and were used with varied equipment, both at the loading and the unloading points. Shuttle cars were loaded either by surge car, by a loading machine using the mine floor as surge capacity, or by a loading machine at the face.

Delays caused by shuttle-car haulage were longer than by extensible belt. Principal delays were caused by mechanical defects of the shuttle cars and trailing cable troubles.

Chain conveyors were most adaptable for use in coal beds less than 42 in. thick. Because of low headroom the movement of equipment and personnel is more limited than in thick coal and as a result production per man-shift is less. When used with bridge conveyors from mechanical loading face equipment, production from chain-conveyor face haulage is two to three times that for hand loading.

Principal causes of delays with chain-conveyor face haulage are: keeping the pans in alignment to prevent the flights from riding to the top of the coal in the pan line; unloading coal to get the flights back into position; and adding sections to or taking them from the pan line.

The extensible belt is the latest type of face haulage adaptable for use behind continuous mining machines in coal more than 50 in. thick. Peak loads are moved

TABLE 12. SUMMARY OF PRODUCTION PER UNIT-CREW/MAN-SHIFT[8]

Mine No. and unit	Type of face haulage	Face mining equipment	Thickness of coal mined (in.)	Minutes face haulage delays	Minutes production time	Tons per shift	
						Crew	Man
4(a)	Extensible belt	Ripper C.M.*	52	5	300	324	54
5(a)	do.	Boring C.M.†	52	14	316	300	43
7(a)	do.	Ripper C.M.*	62	13	242	200	33
11(a)	do.	Boring C.M.†	78	55	280	550	71
12(a)	do.	Ripper C.M.*	84	15	345	540	77
12(b)	do.	Ripper C.M.*	84	30	310	325	41
4(b)	Surge (car) — shuttle car	do.	52	—	294	170	24
4(c)	do.	do.	52	35	270	267	38
6(a)	Surge (floor) — shuttle car	do.	98	40	223	114	15
8(a)	do.	do.	65	30	280	255	36
7(b)	Shuttle car (to mine car)	M.L.‡	74	24	306	200	22
8(b)	do.	do.	65	26	250	255	26
10(a)	do.	do.	71	18	240	242	18
6(b)	Shuttle car (to belt)	Ripper C.M.*	51	29	191	114	17
6(c)	do.	M.L.‡	98	15	300	159	20
11(b)	Shuttle car (to mine car)	do.	78	67	350	1100	67
12(c)	Shuttle car (to belt)	do.	84	15	420	472	39
13(a)	Shuttle car (only)	do.	50	—	450	351	32
6(d)	Mine cars — locomotives	T.L.§	72	85	127	136	16
9(a)	do.	do.	74	20	330	323	32
9(b)	do.	do.	74	25	340	443	40
9(c)	do.	do.	74	10	395	608	51
10(b)	do.	do.	71	100	280	167	12
1(a)	Bridge — chain conveyor	Auger C.M.**	30	84	236	100	20
2(a)	do.	do.	42	75	300	135	30
3(a)	do.	M.L.‡	36	27	284	137	16
3(b)	Chain conveyor	do.	56	15	240	172	19

* Ripper continuous miner.
† Boring continuous miner.
‡ Mobile loader.
§ Track loader.
** Shortwall auger continuous miner.

speedily and generally without surge storage facilities which are commonly used with shuttle-car haulage.

Shuttle-car face haulage is the most widely used type; it can be adapted for use with present day face equipment, and is flexible.

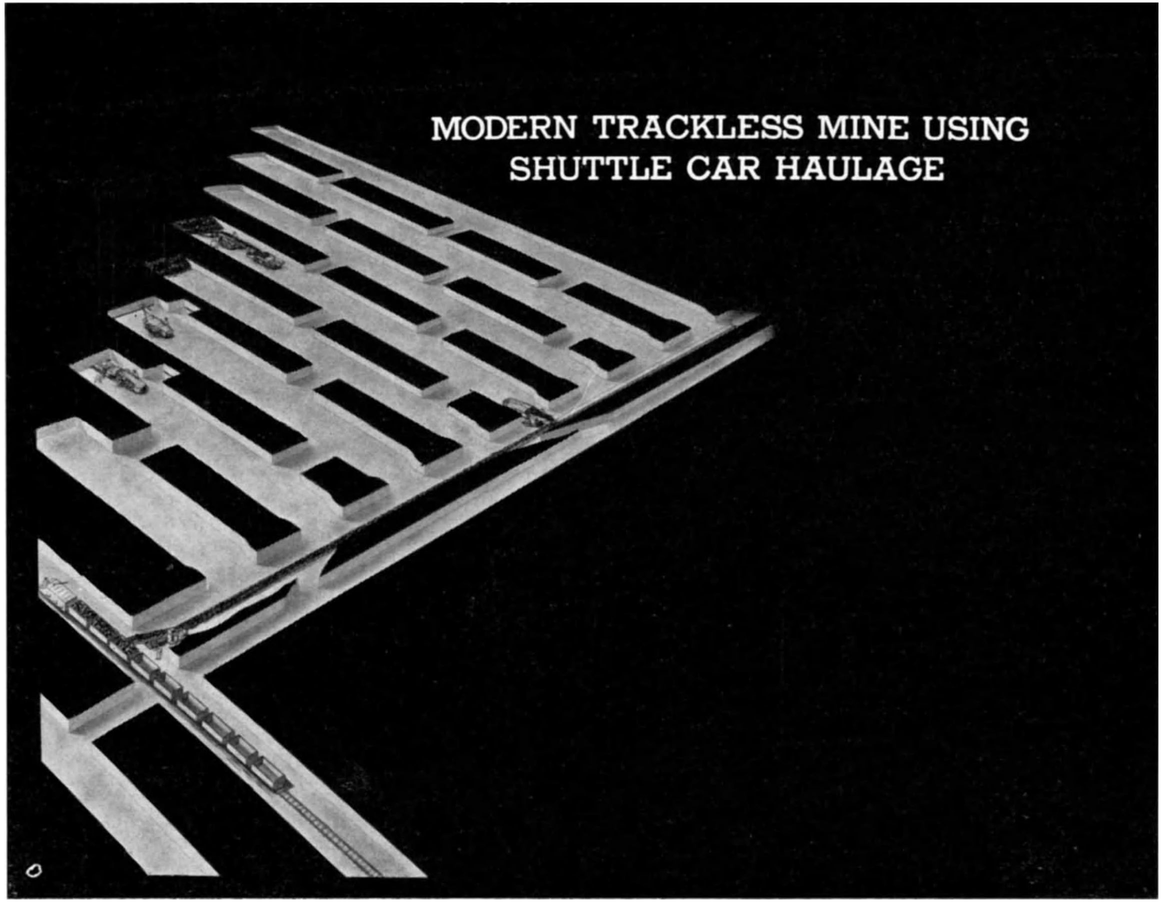

FIG. 22. Modern trackless mine using shuttle car haulage. (Joy Manufacturing Co.)

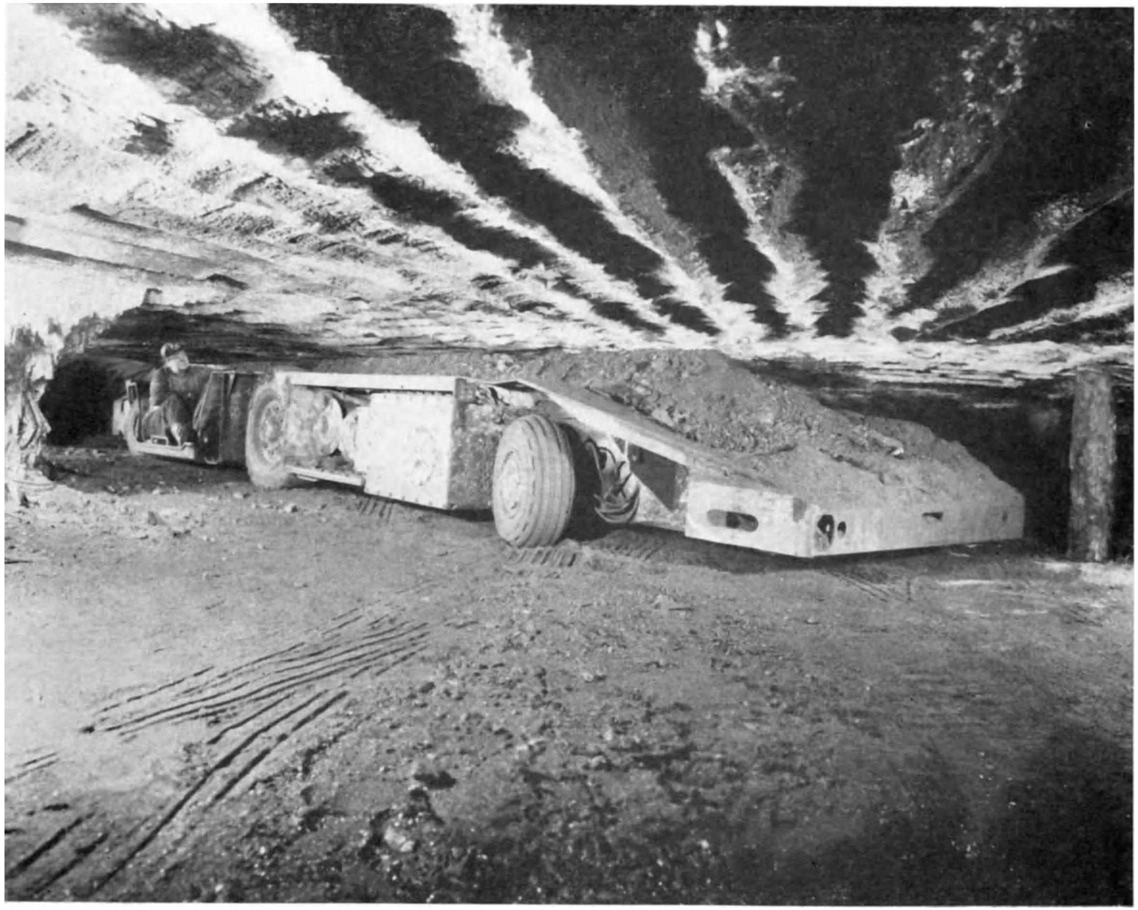

FIG. 23. Six-wheel shuttle car. (Joy Manufacturing Company.)

Although the chain conveyor has many faults and has not been improved in years, it is the best type of face haulage for mining in thin coal.

Mine car face haulage is declining in use. However, in thick coal good production rates can be maintained with this system provided that haulage is carefully planned and supervised, and that large mine cars are used.

Roof Control and Roof Action (see also Volume 1, Part B, Chapter 6 and Volume 2, Part A, Chapter 1)

The most difficult problem encountered in extracting pillars with mechanized equipment is the effective control of not only the immediate roof, but the entire overlying strata. The void created in the strata is partly filled with caved rock from the immediate overlying strata and partly by subsidence of the remaining overburden. The speed, regularity, and precision with which pillars are extracted help to obtain regular breaks and caving of overlying strata along established pillar lines, thus relieving and preventing excessive weight at the coal faces. In some mines, advantage is taken of natural cleats in roof rock to obtain frequent and systematic roof falls. Some of the more serious adverse conditions arising in pillar extraction are the results of the failure to mine all stumps, pillars, or larger blocks of coal as a section is retreated. Squeezes and other unfavorable mining conditions may arise, which result in abandonment of a pillar line and subsequent loss of large areas of coal.

The following discussion is taken from U.S. Bureau of Mines I.C. 7631.[9]

> It is generally agreed in the coal mining industry that in mining coal successfully: (1) Enough coal must be removed to permit the roof and overburden to cave and subside completely; or (2) enough solid coal must be left in place to support the overburden, at least until mining operations in a given area are abandoned. In the first method, 30–60 per cent of the coal mined is from development work, and the remaining production is from systematic pillar robbing or pillar extraction. In the second method, very little coal is recovered from pillars. The first method requires that mine section layouts be designed so that systematic pillar extraction can be planned in advance. Development and pillar-extraction plans usually are dependent upon the operator's knowledge of mining conditions in the area and upon the type of equipment to be used. Other factors that govern mine layouts are mining costs, production possibilities, recovery of coal, and safety of employees and equipment.
>
> The two systems of pillar extraction most generally used are (a) block and (b) room-and-pillar. With the block system, which is more generally used in thick coal beds, over half the production usually is from pillars. In the room-and-pillar system, which is used in both thick and thin coal beds, over half the production usually is from entries, rooms, and crosscuts.
>
> The (a) open-end and (b) pocket-and-fender methods are most generally used in pillar extraction. Where the open-end method is used, cuts or lifts are mined from one or two gob sides of a pillar and no fenders or stumps of coal are left adjacent to the gob as the lift is advanced. In the pocket-and-fender method, lifts are mined in the same way as in the open-end method, except that a fender of coal or a series of small coal stumps is left adjacent to the gob as the lift is advanced. After the lift is completed, the fender or stumps of coal are blasted, and sometimes part of this coal is recovered.
>
> In pillar mining, the roof at the pillar line is supported by timbers, fenders of coal, or stumps of coal. Timbers, where used, comprise rows of breaker props, wood crossbars, or wood cribs.

These roof-support methods often are used in combinations of two or more, depending upon roof conditions. The roof over mined-out areas must cave as pillar extraction progresses. Caving relieves weight that otherwise would be transferred to timbers and solid coal along the pillar line.

The following discussion is from U.S. Bureau of Mines I.C. 7696.[5]

As in conventional, hand-loading, and mechanical mining methods, effective roof control is most important. A method of roof support must be provided that will insure safety to workmen and equipment and allow the continuous-mining machine to operate without undue delay. Roof bolting was used to advantage in some mines; in other mines, wood and steel crossbars were set on steel jacks or wood posts; and other mine roofs required only wood or steel posts with wood cap pieces. Roof conditions generally were better in the continuous-mining sections of a mine than in the mobile-loading sections. At the cleaning plants at two of the mines studied, the reject was 8 and 13 per cent of the raw coal from the continuous-mining sections; and 30 and 20.4 per cent, respectively, from the mobile-loading sections. Both mines, widely separated, left top coal in the continuous-mining section to protect the overlying draw slate and used wood crossbars on wood posts as roof supports; whereas no top coal was left in the mobile-loading sections. Probably because of the top coal and the absence of blasting in the continuous-mining sections, fewer falls contaminated the coal, and a cleaner product to the preparation plant was assured. Roof conditions in sections where pillars were extracted with continuous-mining machines generally were better than in similar areas in the same mines where pillars were extracted by mobile-loading units. Continuous mining lends itself to the concentrated and rapid extraction of pillars (by splitting, open-end, or pocket-and-fender method). As only one working place is active at one time for each unit operating on a pillar line, better roof control can be maintained to provide safer working conditions than with conventional mining.

Weight Transfer

For successful caving of the roof following a long-wall face, or a retreating pillar line, all strong, incompressible supports must be removed from the area to be caved.

When "island" supports, such as pillars, remnants, stumps, or heavy cribs, which are too large to crush easily, are left in the goaf "the weight of the roof is thrown on the face" (to use a mining term). We may visualize the reasons for such an occurrence if we review the stress distribution in the vicinity of the face.

Figure 24(a) shows the stress distribution in the vicinity of the face for the case of full caving of the roof. Here the "abutment load" is all carried on the solid coal ahead of the face.

Figure 24(b) shows the stress conditions if a rigid "island" type of support is left in the mined area behind the face. For the conditions shown the roof is strong and the "island" support is large and relatively incompressible. The portion of roof between the "island" and the face is almost as incompressible as the seam itself. Therefore the portion of roof spanning between "island" and face becomes a part of the loaded abutment and must sustain similar abutment pressure. This overpressure may be sufficient to crush the seam at the face, buckle props and cribs, and cause the roof to fracture and the floor to heave near the face.

When weaker, or slightly more compressible, "island" types of support are left in the goaf stress conditions occur as shown in Fig. 24(c). Here the "beam" formed by the immediate roof projecting back over the goaf is prevented by the island support from breaking off cleanly. If such support yields slightly the beam deflects; again the coal at the face is squeezed between the subsiding roof beam and the floor. Props and cribs are crushed. The roof may fracture and the floor heave at the face.

A rigid island creates an abutment zone between island and face which must sustain abutment pressures proportional to its incompressibility. If the island sup-

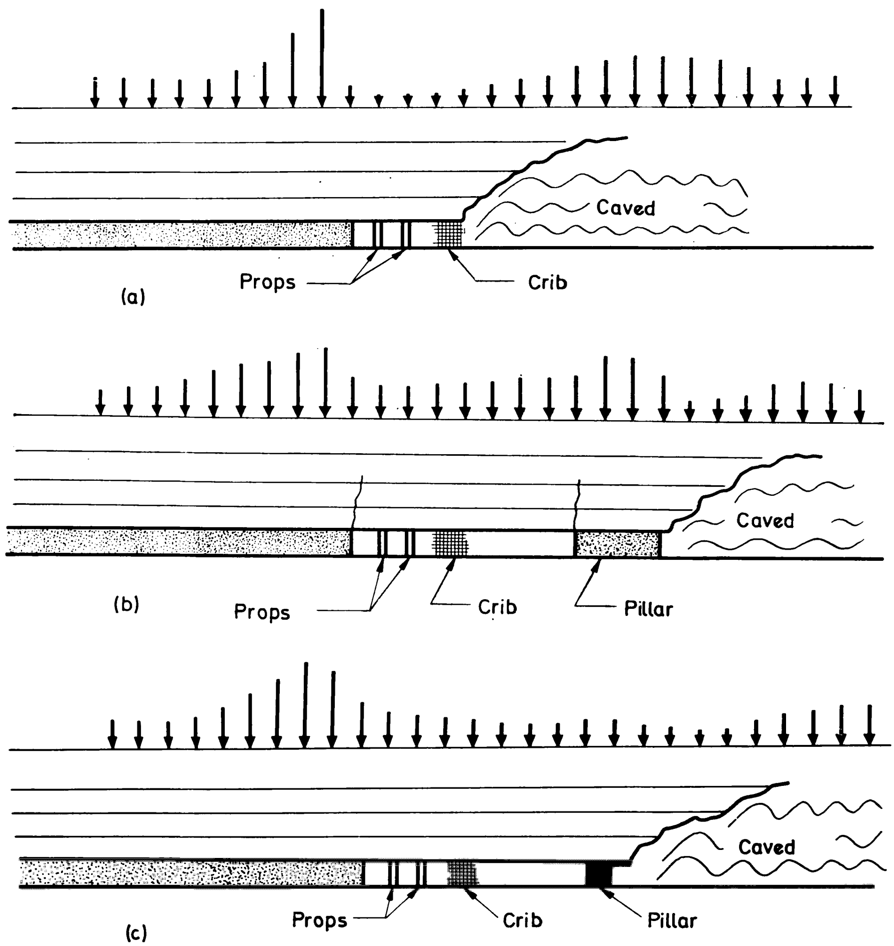

FIG. 24. (a) Vertical pressure distribution when yielding supports are used with complete caving of roof behind the face.
(b) Vertical pressure distribution after a rigid pillar has been left in the goaf.
(c) Vertical pressure distribution after a yielding pillar has been left in the goaf.

port yields sufficiently then the abutment pressure (that is, the pressure from the main roof) is less effective but the weight of the immediate roof (acting as a beam) begins to make itself felt on the face and supports. When the island support yields sufficiently the "immediate roof" beam, if it is still intact, breaks off along the crib line. Usually the damage has already been done and the immediate roof has fractured in the vicinity of the face and is resting on the props at one end and on the island at the other.

Recovery of props or cribs may be difficult or impossible, and re-establishment of the mining and support cycle is difficult.

If extraction of a seam with full caving of the roof is contemplated the seam should be mined out completely as the extraction line progresses. If a stump must be left at the outer end of each pillar as it is mined then the stump should be small enough to crush easily under the subsiding roof after extraction of the pillar is completed.

If strong stumps must be left for temporary support they should be blasted out after mining is completed on each pillar. Wooden props and cribs which are not to be recovered should be small enough to yield, or to crush, and allow the immediate roof to break off short.

Effects of Un-mined Blocks on Underlying Coal Seams

When coal seams lying on different horizons are to be worked the overlying seam should be mined by methods which do not leave large remnant pillars. The pressure of the overburden is concentrated on such pillars and is transmitted vertically to the underlying seam where zones of high stress are created underneath each of the remnants in the upper seam.

Figure 25 illustrates the stress effects of pillars which were left in the Sewickley Seam which was mined from 10 to 30 years prior to the time that the underlying Pittsburgh Seam was mined.[10]

Blocks of coal from 100 ft square to 200×400 ft and larger had been left in the Sewickley Seam. These pillars did not crush although some yield took place when they were forced into the soft bottom by the concentrated overburden load. This load was carried 85 ft down through the intervening strata to the Pittsburgh Seam with the result that headings in the Pittsburgh were subjected to excessive squeezing and roof deterioration and large amounts of timber and cribbing were required to keep entries open.

Mining conditions became normal when headings were reoriented to pass between locations of overlying pillars instead of under them.

The depth of the overburden overlying the Sewickley was most important in determining whether squeezing would take place in the Pittsburgh Seam. Where the cover varied from 50 to 200 ft little difficulty was experienced in Pittsburgh headings. Where cover exceeds 200 ft squeezes may be expected and where the cover exceeds 300 ft difficulties are almost certain to occur in headings driven under pillars left in the Sewickley.

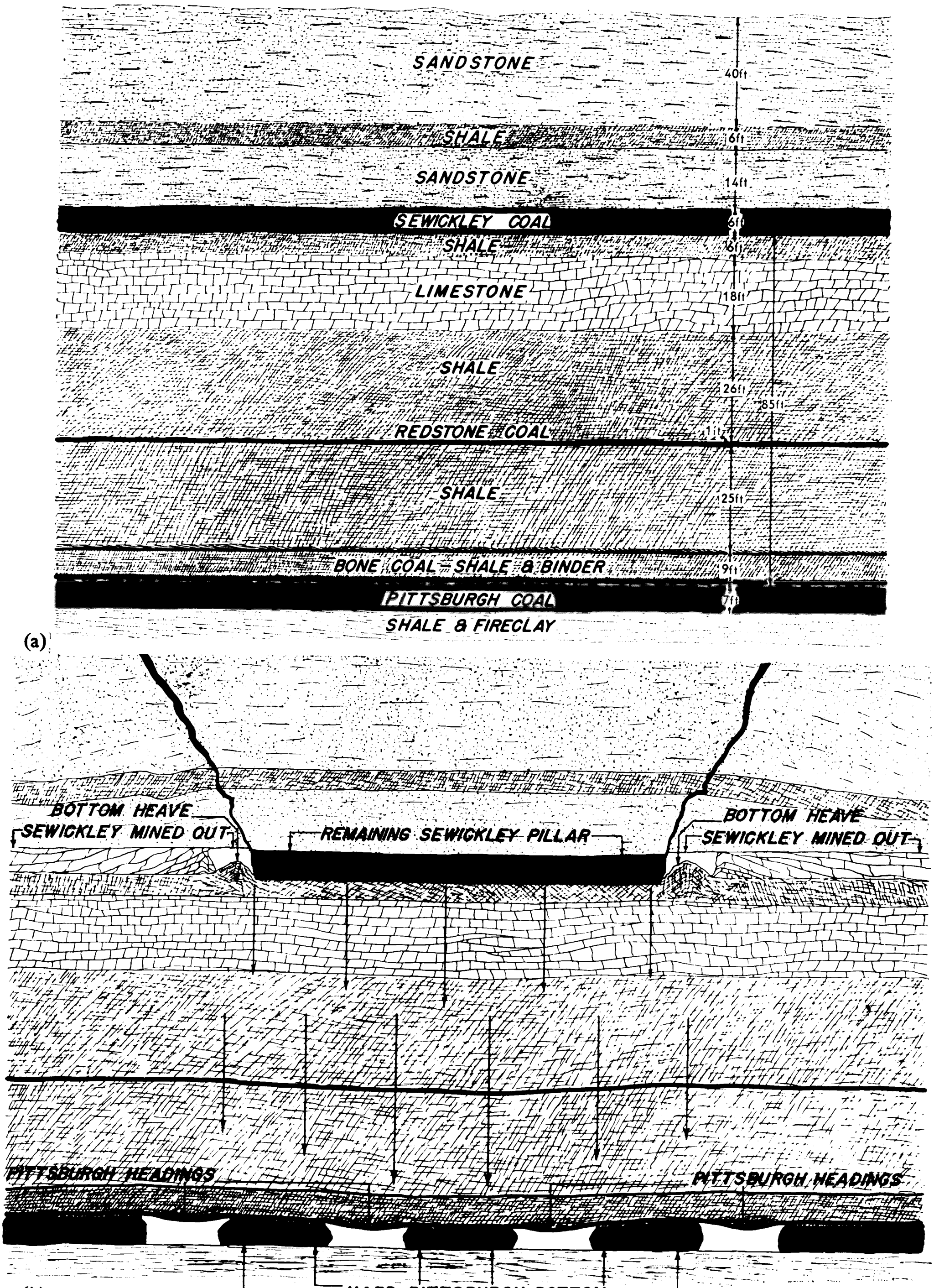

FIG. 25. (a) A typical cross-section showing strata overlying the Pittsburgh coal.
(b) Overburden load transmitted to the Pittsburgh seam through unmined Sewickley blocks.[10]

LONG-WALL MINING SYSTEM

Roof Control and Roof Action

With the system of long-wall advancing a panel of coal is extracted by advancing a face or extraction line on a broad front. Support at the working face, and immediately behind the face, is furnished by steel (occasionally wood) props. These are supplemented by timber "chocks" or "cribs" placed a few feet further to the rear. The latter furnish a line of rigid support along which the roof will break and cave. All supports are retrieved, moved ahead, and re-set as the face advances.

Behind the supported area in the vicinity of the working face the roof may be allowed to break and cave into the "goaf" (mined-out area) or "strip packing" or "filling" may be provided in the goaf area, to prevent full caving of the roof, and to allow it to settle gradually until filling is compressed (see Volume 2, Part A, Chapters 1 and 2).

Long-wall Advancing with Full Caving of the Roof (see Volume 2, Part A, Chapter 2)

This method provides no support, except at roadways. The roof is allowed to cave completely, and subsidence extends to the surface.

Figure 26(a), (b), and (c)[11] illustrates the principle of the long-wall mining system with full caving of the roof. The immediate roof fractures and falls, and the upper beds are permitted to fracture or bend until they are supported on fallen material.

The roof immediately at, and behind the long-wall face, can be considered as a cantilever beam projecting back over the goaf. A line of wooden cribs or "chocks" is ordinarily set, and spaced closely enough to produce a break line along which the immediate roof will fracture and fall. If the "cantilever beam" can be broken off short then excessive bending, and pressure, will be prevented from developing in the vicinity of the face, and excess weight will be kept off the props and chocks.

It is apparent that a thick strong roof, such as thick sandstone beds, will be difficult to break. Such a roof may exert excessive pressure on the coal and support before fracturing and break with considerable violence when it does fracture.

Long-wall Advancing with Strip Packing or Filling (see Volume 2, Part A, Chapter 2)

This method is commonly used in British and European mines. Roadways which must be kept open for access to the advancing face, such as ventilation ways, and haulageways, are supported by "rock packs" in and/or at the edges of the mined-out areas (goaf). In addition packs are built to support the general mass of strata and to control the roof near the working face.

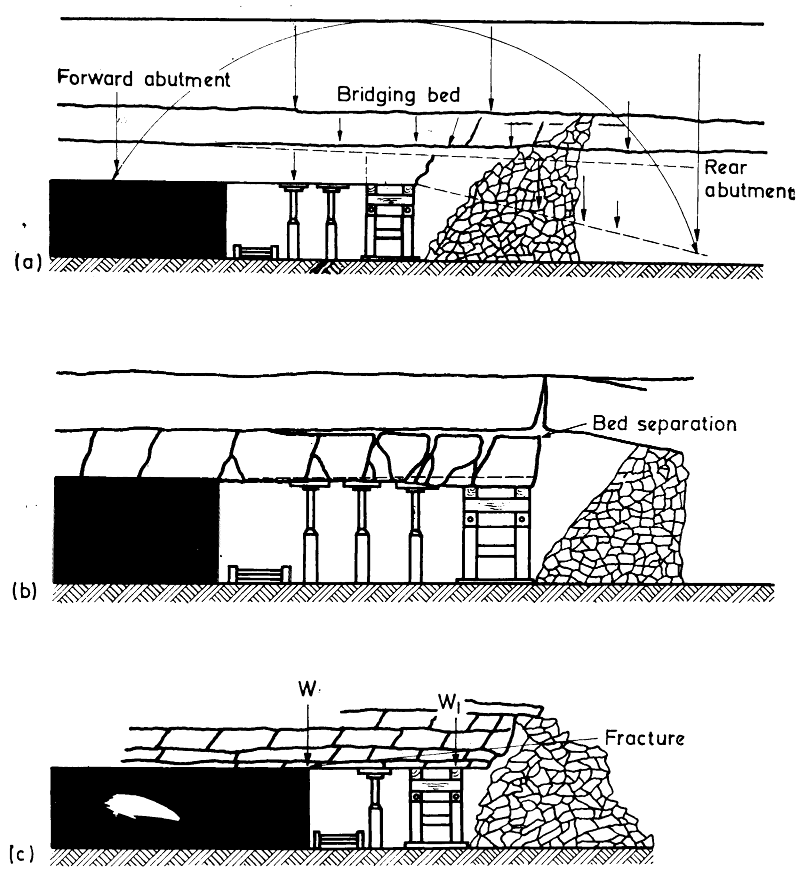

FIG. 26. (a) How higher strata bridge over a mined-out area and relieve the load on the immediate working face.
(b) How convergence affects the top at the working face.
(c) How caving is employed to fill waste behind the working face.(11)

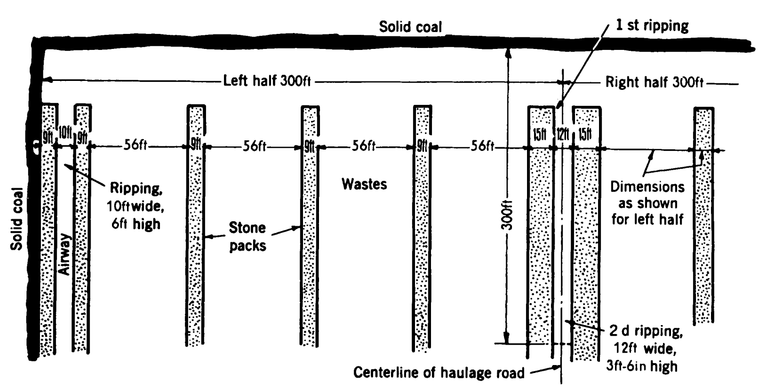

FIG. 27. Strip packing – advancing long-wall.(12)

These packs are gradually compressed by the settlement of the overlying strata and tend to reduce periodic intensive fracturing, and to minimize "bumps". Subsidence is even over the working area and the rate and amount can be regulated within limits by the quality and method of stowing of the packing.

Figure 27[12] shows a typical example of long-wall mining with strip packing.

In the example illustrated in Fig. 27 the bed thickness is 5 ft; overburden is 2275 ft. Material for the stone packs is taken from the wastes, which cave as the face advances. If the roof material does not cave readily, it must be drilled and blasted to supply this material. Support at the working face is achieved with props, either wood or steel, and chocks (wood). These face supports are reclaimed and moved forward with successive cuts. Uniform quality of packs is essential for proper control of weight, which is further accomplished, accurately, by varying the number and width of packs. The first ripping, or mining of rock sufficient for roadway height, is done at the face end of the packs; a second ripping is carried on about 300 ft behind the face, in this example, to establish final roadway height after the mined-out area has subsided.

In normal conditions, where face conveyors are used, the economic length of face has been found to be 100–150 yd for single units and twice that length for double units.

Long-wall advancing is the principal system of mining in 74 per cent of the collieries in Great Britain.

Full Caving vs. Strip Packing or Filling (see Volume 2, Part A, Chapter 2)

The "full caving" system is more economical of labor since packwalls are not used in the areas between roadways. However, packwalls for support of roof over roadways are needed with the "full caving" as well as with the strip packing or filling methods.

With the "full caving" system the span of the "pressure arch"* is greater and therefore the maximum "front abutment" pressure is greater than when strip packing or filling is used. Figure 28 shows comparisons of abutment pressures for (a) newly opened face; (b) an advancing face being caved; (c) an advancing face with strip packing behind it.

The more tightly packed and incompressible the rock packs the more will the front abutment pressure be reduced toward the values shown in Fig. 28(a).

The roof will be solidly supported in the goaf and on the coal face and length of roof span will extend from rock pack to coal face.

However, the rock packs which form the strip packing cannot be made as solid as the original coal. Therefore the effective roof span always exceeds that of the original development opening. Resultant abutment pressures also exceed the pressures which existed initially at the sides of the development opening.

* See Volume 1, Part B, Chapter 6.

The full caving system requires the least labor, but certain conditions may prevent its use. For example, a soft floor may heave at the face under the abutment pressures induced by full caving. A soft seam may crush excessively, or a roof may be excessively fractured at the working face by abutment pressures.

If the roof deflects rather than fracturing, and acts as a cantilever beam the immediate roof deflection allowed by full caving may be sufficient to buckle props at the face.

With retreating long-wall systems the access roadways pass through the unmined seam rather than through the goaf. In this case the abutment pressures on the coal seam adjacent to a roadway may cause heaving of soft roadway bottom, fracturing

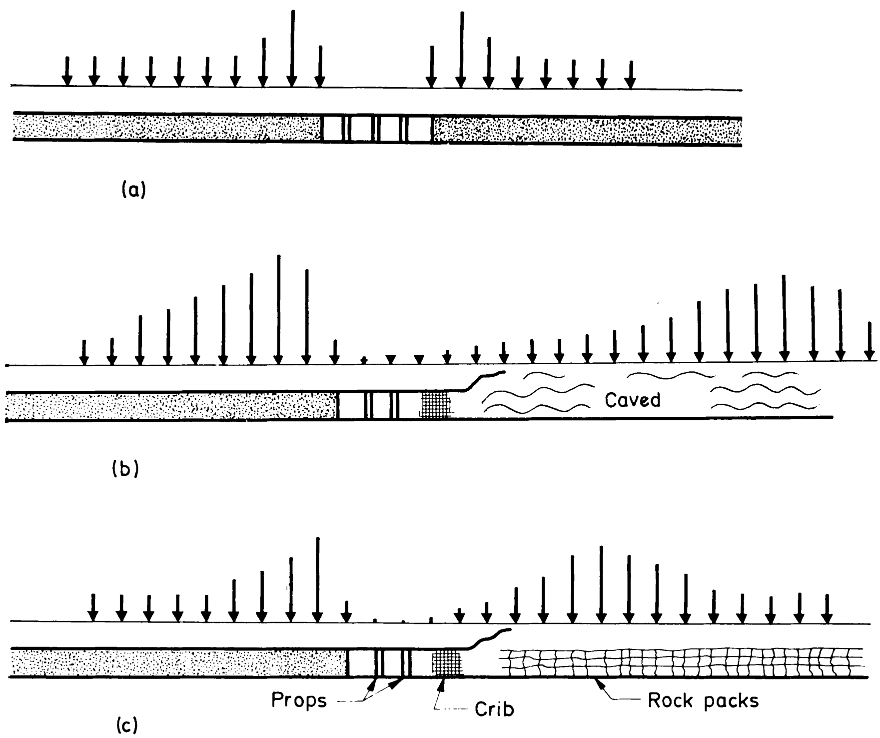

FIG. 28. (a) Abutment pressures for a newly opened face.
(b) Abutment pressures for a long-wall with full caving.
(c) Abutment pressures for a long-wall with well-made "rock packs".

of a soft roof, or fracturing and falling of coal at the sides of a roadway. Thus a long-wall retreating system may require strip packing in the goaf for the purpose of reducing the abutment pressure on the seam ahead of the face.

Any type of rock packing or filling requires much labor.

In the case of an experimental retreating long-wall system with full caving[13] it was noted that 65 per cent of the face labor was required in connection with roof support. Under less favorable conditions it appears quite likely that 75–80 per cent of face labor might be required for roof control. These figures compare with 15–25 per cent of face labor for roof control in pillar mining systems of the U.S.

It was previously noted that costs of filling alone, behind long-wall faces in the Ruhr, were approximately equal to costs of mining the coal. The filling cost did not include the cost of support at and immediately behind the face.

The foregoing factors indicate the desirability for exhausting all possibilities of employing a "full caving" system before employing a system requiring filling, or packing of the goaf.

Roadway Support (see also Volume 2, Part A, Chapters 2 and 4)

With advancing long-wall mining systems the access openings (roadways) leading to the face must pass through the mined-out area. Since all the coal has been removed from this area some other filling must be provided to prevent the roof from subsiding onto the floor and closing the opening.

Under such conditions the depth pressure or weight of overlying rock is so great that no rigid structural support is sufficiently strong to maintain the opening. Therefore "rock packs", "packwalls", are constructed alongside the roadways to prevent complete closure of roof into floor. However, any such filling is compressible and considerable subsidence takes place before the filling or packing is sufficiently compressed to prevent further subsidence.

The subsidence makes it necessary that the roof be "ripped", or the floor "dinted" occasionally to provide enough headroom for passage of men and equipment. "Brushing" or "ripping" consists of removing a portion of the roof, to provide more headroom while "dinting" is excavating in the floor for the same purpose. In effect it is the cutting of a trench in the floor, or an inverted trench in the roof.

The "packwalls" may eventually stabilize a "beam" of roof strata overlying the roadway but further protection is usually required to prevent fragments of roof, and filling from falling into the roadway. Thus some local support is needed for the roof and sides. This support must be of a type which yields with the roof or it will be destroyed.

Various types of support have been devised for such situations; some of which involve steel arches or arched beams supported on yielding cribs; some involve steel arches or beams supported on fragmented rock which is confined behind lagging and allowed to "flow" out to allow the arch to lower with the roof. The most successful of the yielding roadway supports are the "yielding steel arches" which yield under excessive ground pressure by sliding together or "telescoping" of segments. These arches possess enough strength to support the immediate roof and sides but not enough strength to support the weight of the main roof; therefore they are designed to yield under the great weight of the main roof but to hold

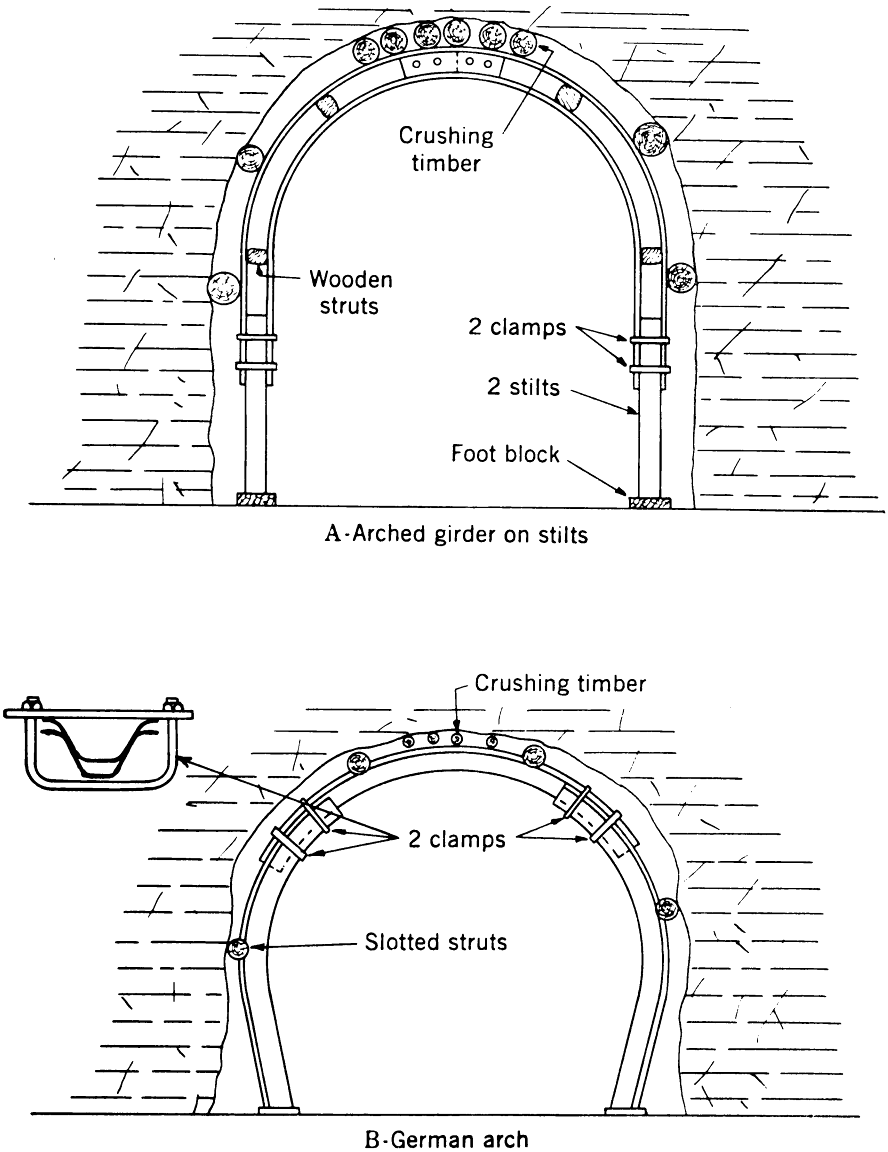

FIG. 29. Types of yielding steel roadway supports.[12]

against the lighter pressures due to loose rock of the immediate roof, and side pressures of filling.

These supports are described more fully in Volume 2, Part A, Chapter 4.

Two types of yielding roadway supports are shown in Fig. 29.

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CHAPTER 2

PILLAR MINING SYSTEMS CONVENTIONAL (CYCLIC) MINING

MINING CYCLES

The sequence of operations in cyclic mining is as follows:

Cutting: A slot, usually horizontal and from 5 to 7 ft deep, is cut across the width of the coal face. This is usually cut at floor level and provides the coal a free face to which it can break when it is shot, as well as providing a plane of separation between the coal and the floor.

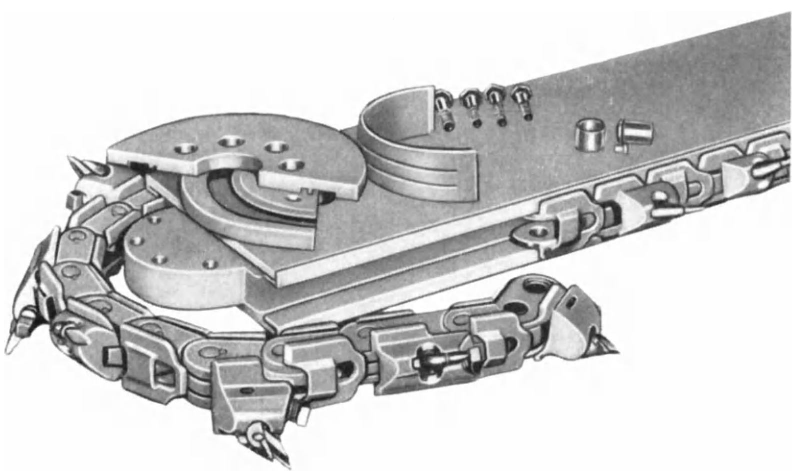


FIG. 1. Cutter bar and chain. (The Bowdil Company.)

Drilling: Rotary auger-type drills are commonly used for drilling shot holes. Drills may be hand held, or may be mounted on columns, or they may be mounted on rubber-tired drill jumbos.

Shooting: Holes may be shot with explosives, or the coal may be broken by means of compressed air or carbon dioxide cartridges.

Loading coal: Most coal is mechanically loaded by means of crawler mounted loaders which operate by thrusting a wide steel pan into the broken coal which is then swept up the pan and onto a conveyor which transports the coal to the back end of the loader and discharges it onto another conveyor, or into shuttle cars.

From the loader the coal is transported to the main transportation system which may be either a main conveyor belt or may consist of large rail mounted cars.

Roof support: Temporary support may be placed as soon as the coal is shot, and before the loader approaches the face, or support may be placed over the loader as it advances.

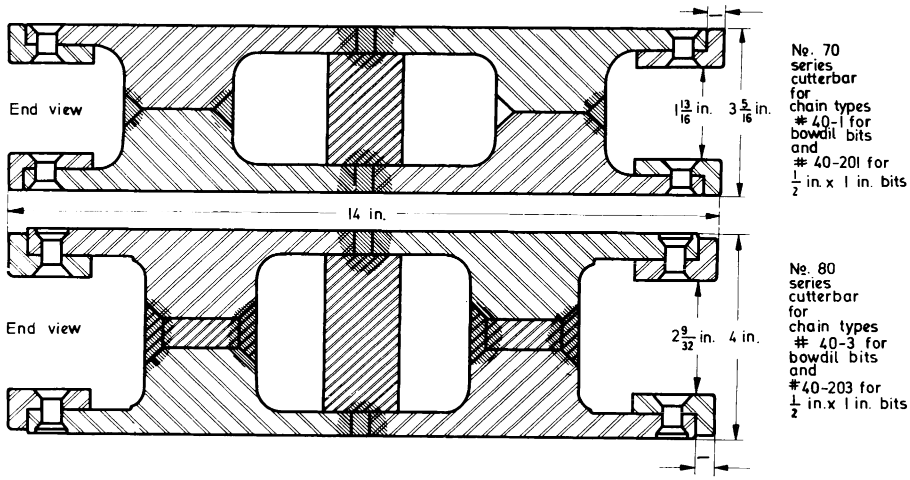


FIG. 2. Cross-sections of typical cutter bars. (The Bowdil Company.)

Support may consist of props, or of props and bars, or of hydraulic props, or of roof bolts.

Wooden roof props are still in extensive use but have been replaced to a large extent by roof bolts.

Coal Cutters

The basic cutting unit in all modern coal cutters is the "jib" around the perimeter of which a toothed chain travels somewhat in the manner of a chain saw. All coal cutters employ this same cutting principle.

Short-wall Coal Cutter:

This machine is used in cutting wide headings and production places. The jib is fixed rigidly to the body of the machine and extends through the machine body to discharge the cuttings at the back end of the machine.

The machine maneuvers by means of two ropes one of which is used to pull the machine along while the other is used to control the angle which the machine makes with the face.

Usually short-wall cutters are equipped with jibs about 7 ft long and the rate at which the machine can progress along the face while cutting is about 2 ft/min.

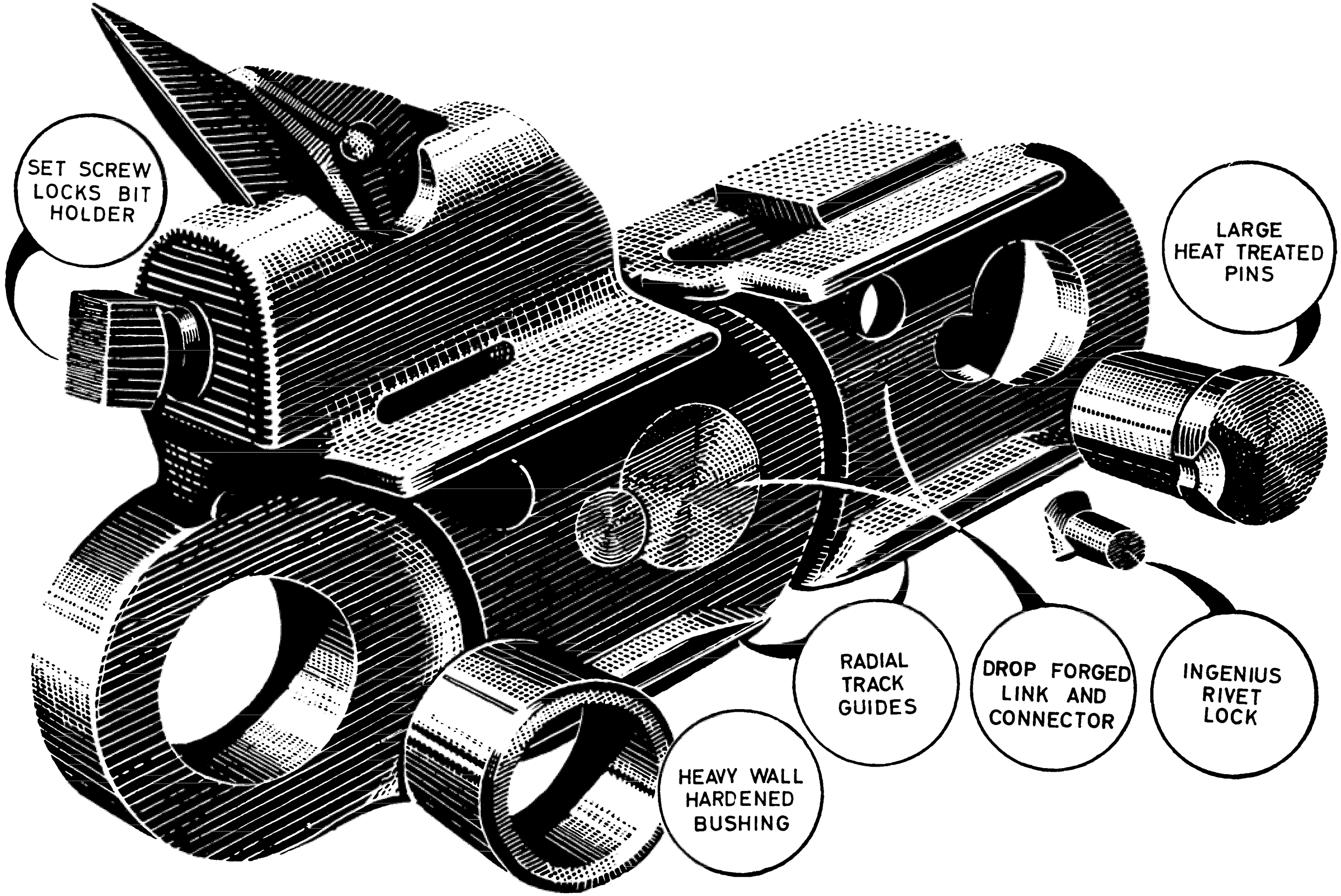
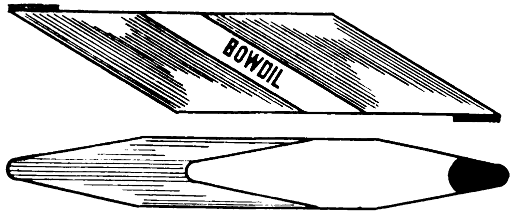
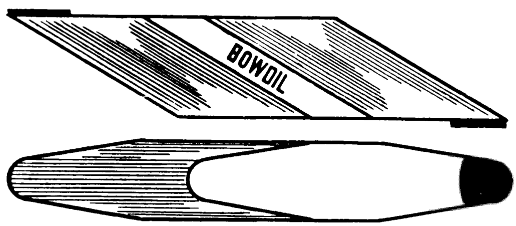


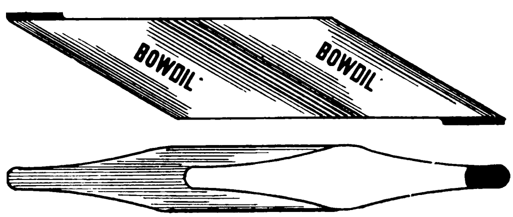
FIG. 3. Cutter chain. (The Bowdil Company.)



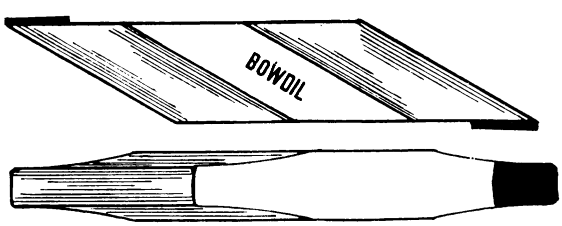
No. 1-19
Regular Diamond Bit BOROD TIPPED. Designed for medium cutting conditions, they save power, produce coarse cuttings.



No. 1-20
Heavy Diamond Bit BOROD TIPPED. Designed for severe cutting conditions, such as iron pyrites and rock.



No. 1-28
Regular Concave Bit BOROD TIPPED. Designed for maximum power savings. Concave shape automatically maintains side clearance. (Patented feature.)



No. 1-29N4
Heavy Duty Concave Bit BOROD TIPPED. Designed for very severest service.

FIG. 4. Coal cutter bits. (The Bowdil Company.)

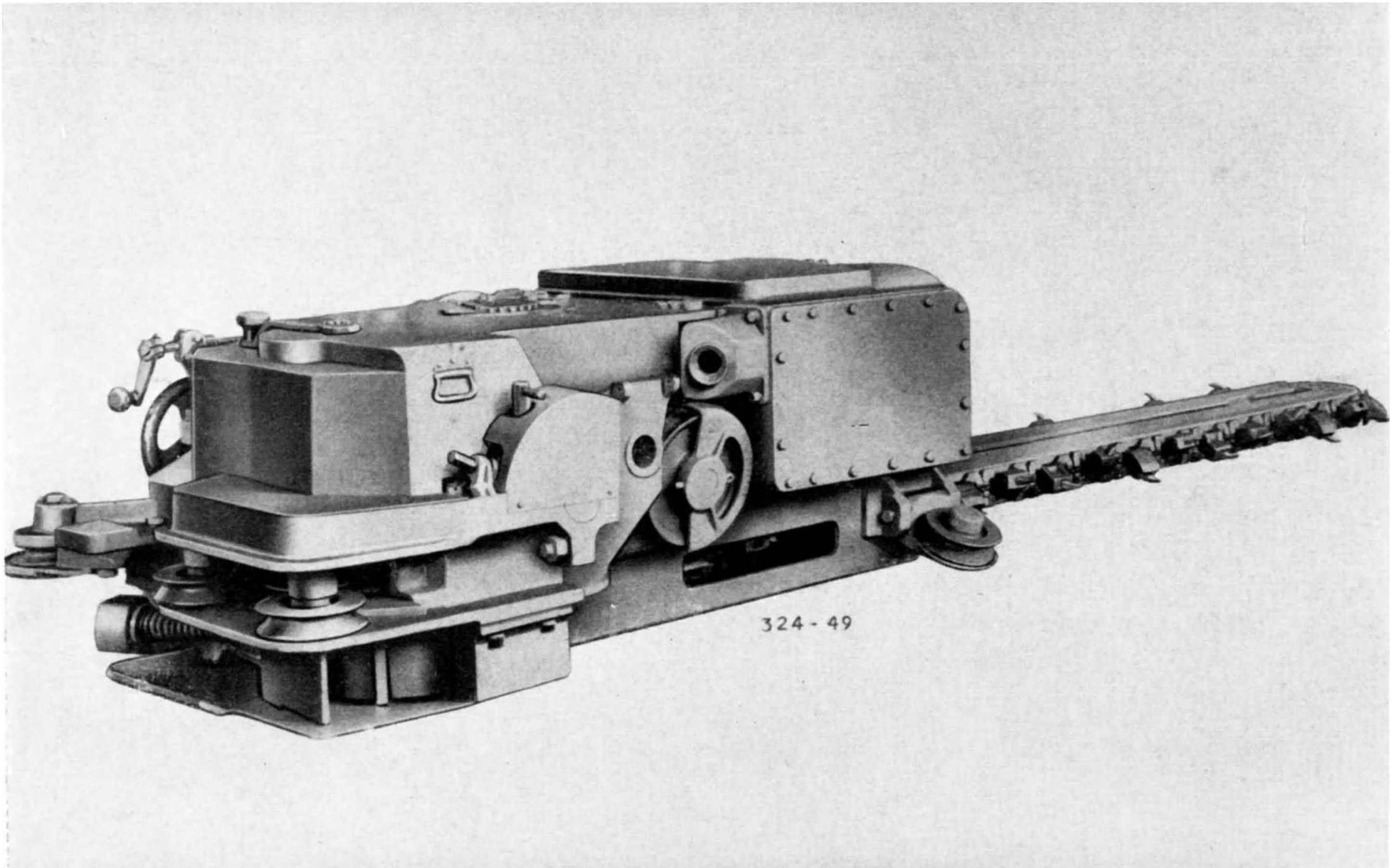


FIG. 5. Short-wall coal cutter. (Jeffrey Manufacturing Co.)

Figure 6 illustrates the method of "sumping in" and cutting across a shortwall face.

A short-wall cutter may be used to cut a number of headings during the course of a shift. Rail mounted trucks may be used (in track mines) to transport the machine from heading to heading while a crawler-mounted or rubber-tire mounted flitting truck may be used for this purpose in trackless mines.

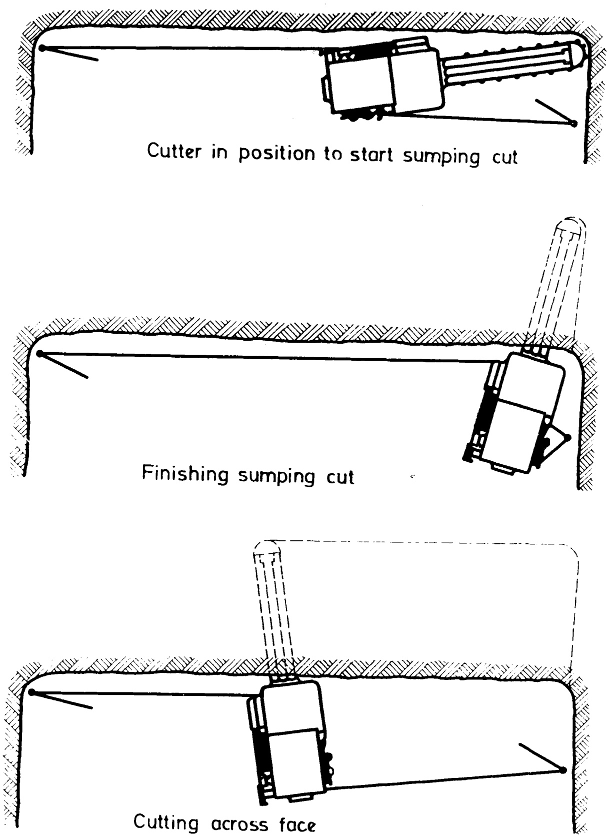


FIG. 6. Cutting the face with a short-wall cutter. (Jeffrey Manufacturing Co.)

Arc-wall Cutters

As its name implies the arc-wall cutter makes a cut by sumping in and then swinging its jib in an arc. It is most commonly used for cutting narrow places, such as those in room and pillar work.

The arc-wall cutter is designed to make a horizontal cut on one horizon. Where it is desirable to make cuts at various heights machines are available with the body mounted on four hydraulic jacks.

Arc-wall cutters may be rail mounted, mounted on crawlers, or mounted on rubber tires.

Figure 7 illustrates the reach of a modified arc-wall cutter. This type can cut a heading 30 ft wide.

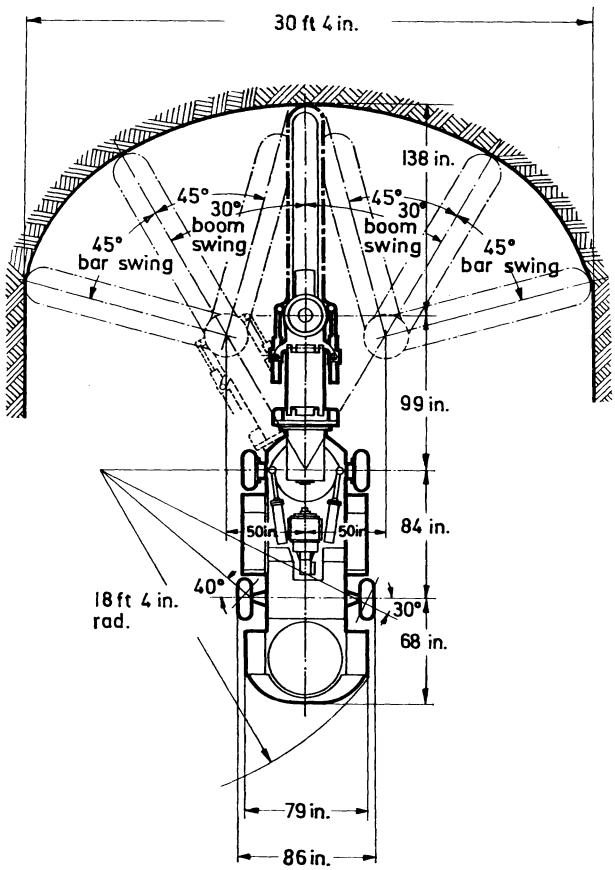


FIG. 7. Arc-wall cutter. (Joy Manufacturing Co.)

Universal Cutters

The machine can cut horizontally, vertically, or at any angle. The angle of the cut is controlled by rotating the head which carries the jib while the height of the cut is regulated in some machines by raising or lowering the body of the machine by means of integral hydraulic jacks, and in other machines is regulated by raising or lowering a head which pivots about a transverse horizontal axis.

Universal cutters may be track mounted, mounted on crawlers, or mounted on pneumatic tires.

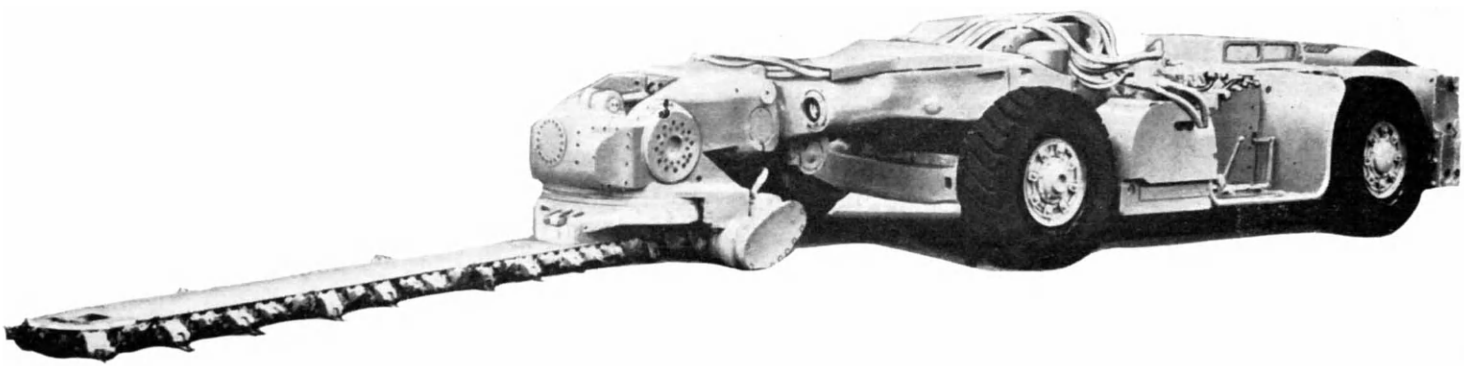


FIG. 8. Universal cutter. (Jeffrey Manufacturing Co.)

FIG. 9. Drilling the face. (Jeffrey Manufacturing Co.)

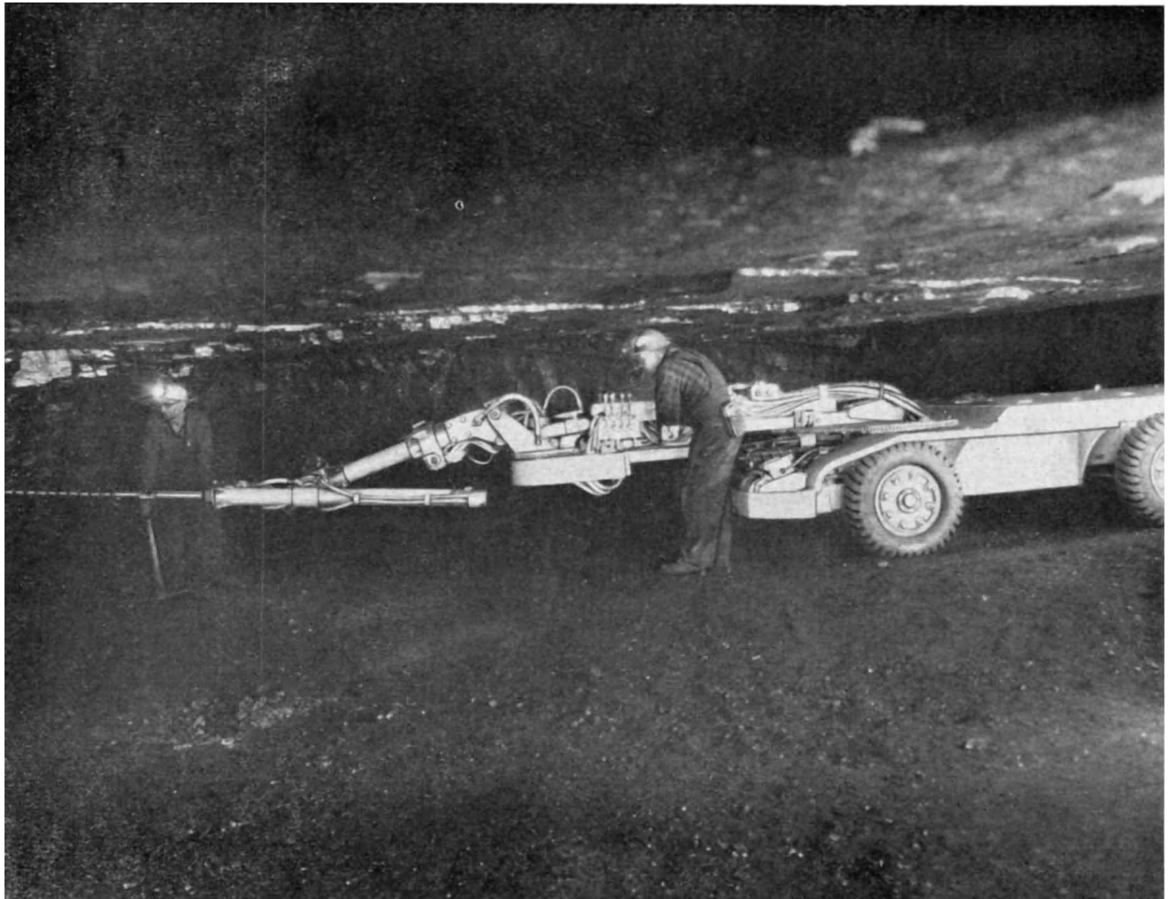
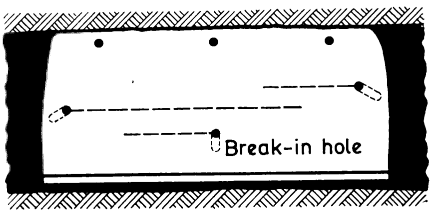


FIG. 10. Drill mounted on rubber tired jumbo. (Jeffrey Manufacturing Co.)

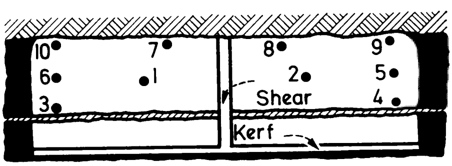
Coal Drills

In soft coal, or in thin seams, shot holes may be drilled with hand held auger-type rotary drills. Where the coal is hard or the seam is thick column-mounted drills may be used.

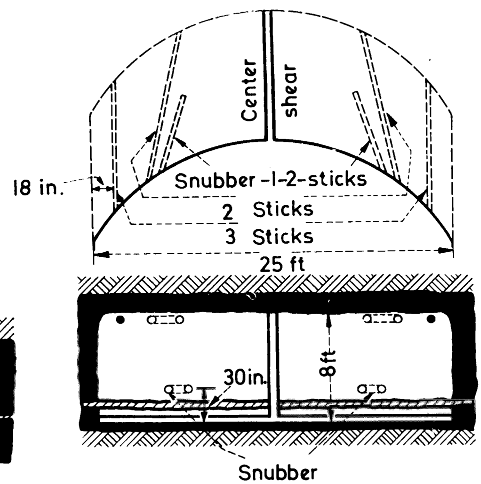
For highly mechanized operations multiple rotary drills mounted on booms which are mounted on a rubber tired chassis and are manipulated by hydraulic controls are used.



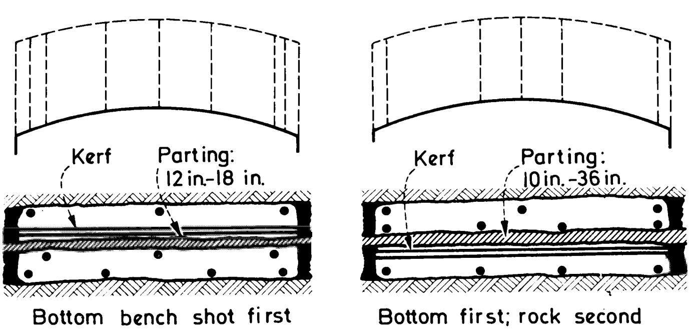
BREAK-IN-HOLE provides additional free face for relievings ucceeding holes. In this plan, rib holes are stepped up to break lower part of face in section.



EQUALIZED BURDEN is the goal in this drilling pattern in a center-sheared place, thus achieving good breakage of both coal and impurity band.



SHORT SNUBBERS break band and roll out front of fall in arc-cut center-sheared place. Shearing and snubbing reduce the number of holes required and also cut explosive requirements.



SHOOTING WITH HEAVY PARTINGS. Plan at left shows cutting above the parting, followed by shooting of the bottom bench first. Plan at right, for still-thicker parting, shows cutting underneath the rock and holes immediately above.

FIG. 11. Drilling patterns.[1]

Drilling and Shotfiring

Shot holes should be spaced in accordance with the thickness of the seam. In thin seams one row of holes placed near the top of the seam and spaced a distance apart about equal to the thickness of the seam is usually sufficient.

In seams 5 ft or more in thickness it may be necessary to use a top and a bottom row of shot holes with the holes staggered in alternate rows.

Coal Loaders

Loading machines may be classified as (1) gathering-arm loaders; (2) duckbill or shaker loaders; (3) overshot or rocker shovels; (4) slusher or scraper-loaders.

Of these only the gathering arm loaders and the duckbills have been used extensively in American coal mining practice.

7 Rocker shovels are used extensively in metal mining work but require about –8 ft of headroom and find application in very few coal mines.

Gathering-arm loaders

These are self propelled and may be mounted on rails, on crawlers, or on rubber tires. The crawler-mounted loader is the most commonly used. It is made up of three main components (a) the gathering head which is thrust into the coal and which is fitted with two rotating arms to sweep the coal onto the conveyor: (b) a chain conveyor which runs down the middle of the machine and carries the coal to the rear; and (c) the crawler chassis which carries a separate motor for each of the crawlers.

In track-mounted machines the loading head has the ability to swing 45° to either side of the center line and the tail of the machine can swing 15–20° on either side of the center line.

Gathering-arm loaders are available in a heights from about 30 in. up and in capacities ranging from less than 1 ton/min up to 12–20 tons/min.

Duckbill Loaders

Shaker conveyors can be converted to self-loaders by the addition of "duckbills". The duckbill loader consists of steel-trough sections with a shovel head at the forward end and with a gripping device at the rear end which can be clamped to the shaking conveyor so that the reciprocating motion of the shaker is imported to the duckbill.

The shovel head or "duckbill" rests on the floor and is forced forward into the loose coal and, by the nature of the shaker action coal is caused to travel up the trough and onto the shaker conveyor.

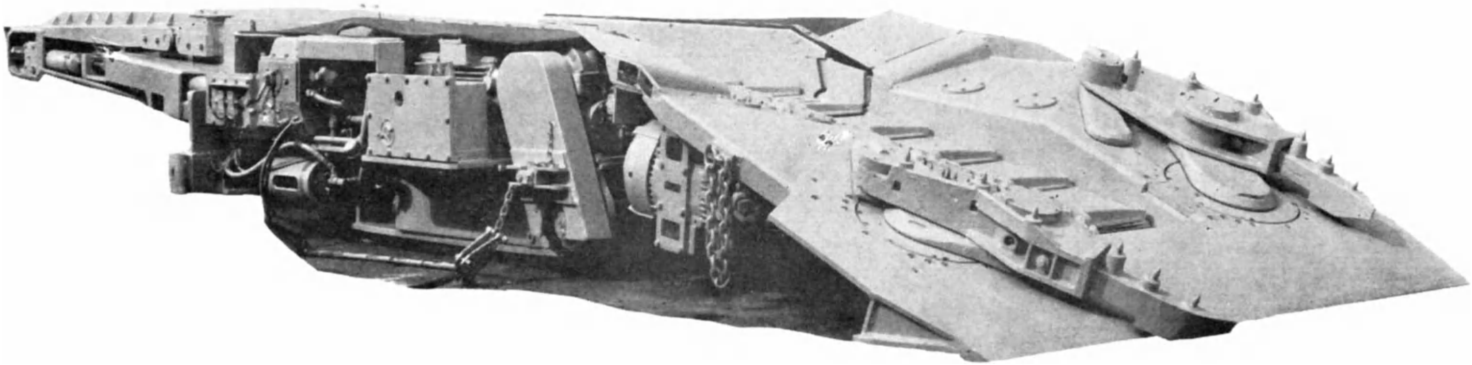


FIG. 12. Gathering-arm loader. (Jeffrey Manufacturing Co.)

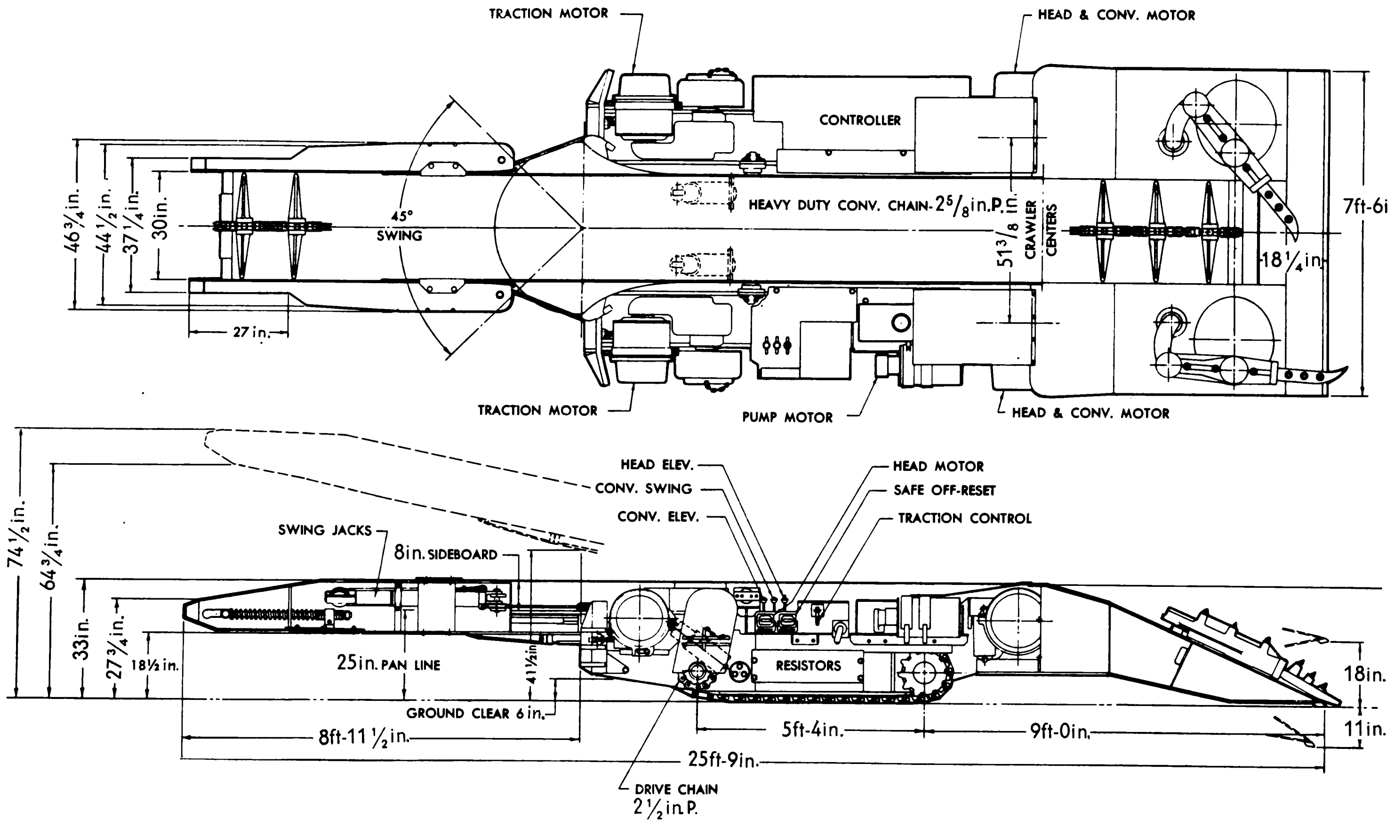


FIG. 13. Joy 14-BU loader. (Joy Manufacturing Co.)

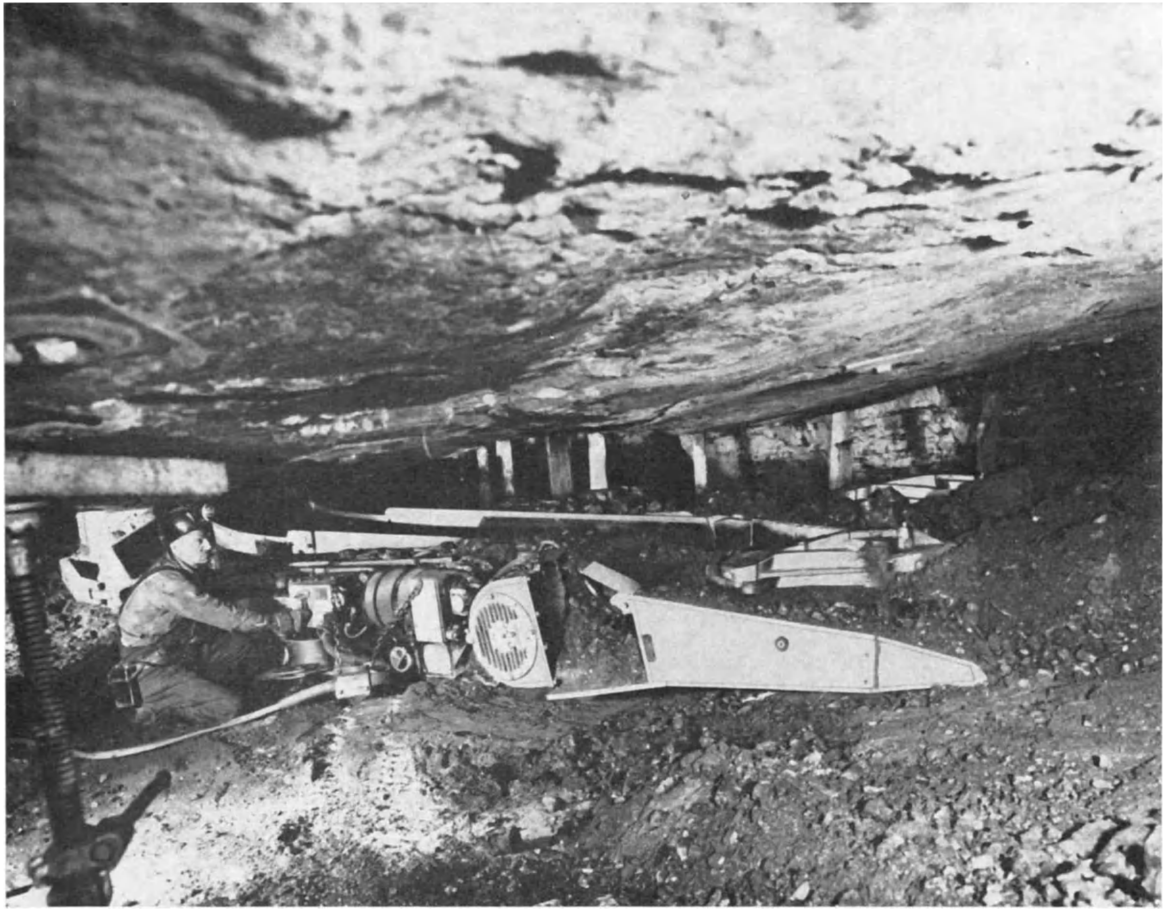


FIG. 14. Gathering-arm loader. (Joy Manufacturing Co.)

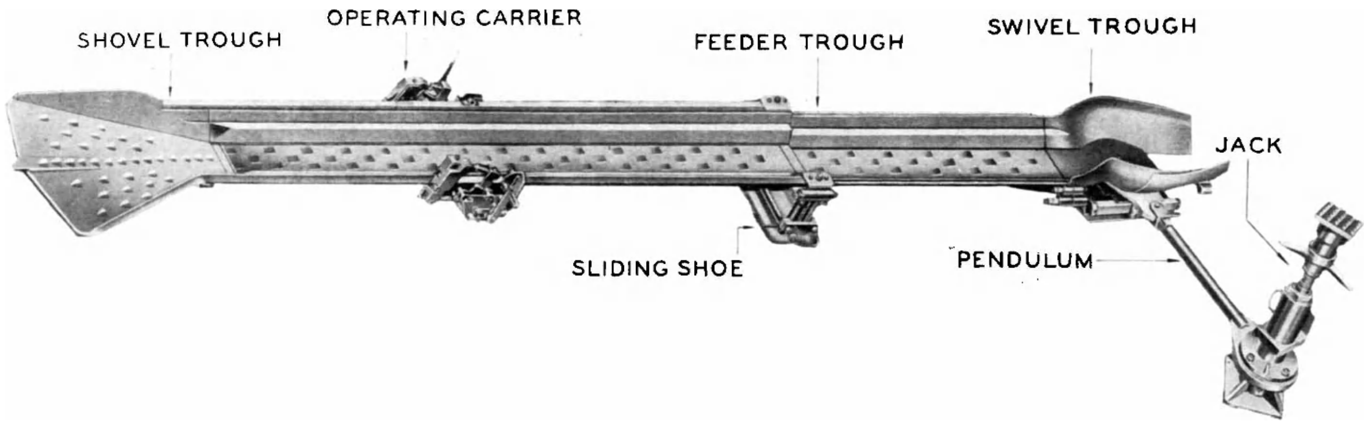


FIG. 15. Goodman duckbill. (Goodman Mfg. Co.)

The duckbill is anchored by a pendulum jack, about which it can be pivoted through an arc of 30°, or 45°, on either side of the centerline of the shaker conveyor. The duckbill requires very little headroom and is suitable for use in thin seams.

The Conveyor-loader

The Jeffrey conveyor-loader is mounted on rubber tires and is very maneuverable because the front wheels can be rotated through 90°. The machine has a gathering head which is provided with chains and flights which scrape the coal onto a chain conveyor which carries the coal back to discharge it at the rear of the machine.

The method of operation is as follows: Coal is cut and shot and the conveyor-loader is flitted to the face and maneuvered so that the gathering head is at the broken coal while the rear end is situated so that it will discharge onto the room conveyor.

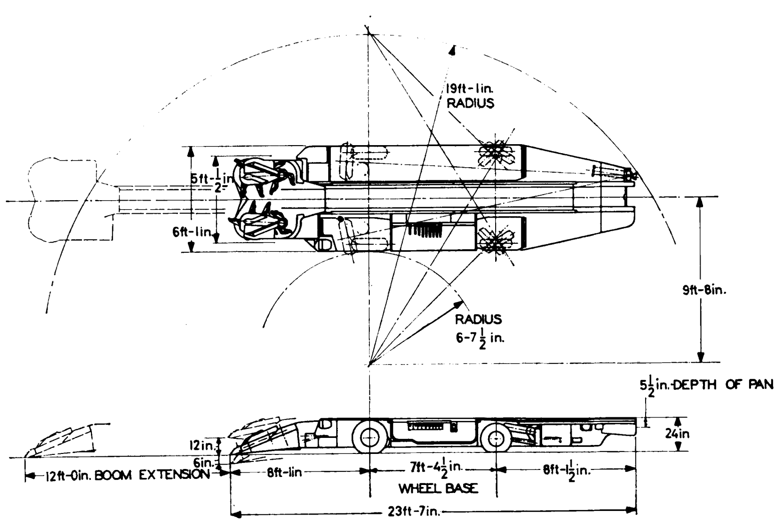


FIG. 16. Conveyor-loader. (Jeffrey Manufacturing Co.)

veyor. An hydraulic roof jack at the rear of the machine is dropped and tightened between floor and roof to provide an anchorage point about which the machine may pivot. The retractable rear wheels are then raised clear of the floor and the front wheels are rotated through 90° so that the front end of the loader can travel in an arc across the heading. The gathering head is then lowered and extended into the coal pile to start loading.

The machine has an overall height of 24 in. which makes it suitable for use in seams as low as 36 in. and has a rated capacity of $1\frac{1}{2}$ tons/min with an average loading rate of $\frac{2}{3}$ tons/min including clean-up.

METHODS AND EQUIPMENT USED IN DEVELOPMENT WORK

A number of mines were studied by personnel of the U.S. Bureau of Mines to determine what methods and equipment were used in underground development (*U.S. Bur. Mines I.C.* 7813, 1957). Following are the descriptions of the methods and equipment used in development work at two of these mines and in addition there are shown diagrams of development methods used at additional mines.

Mobile Loading onto Chain Conveyors (Mine 1)

This mine is operated in the No. 3 Elkhorn bed, which averages 35 in. thick in this area. The bed is flat lying, with some local dips. The coal is underlain by fire clay and overlain by a sandy shale that usually provides good roof. The overburden averages 315 ft thick and consists mainly of strata of shales and sandstones.

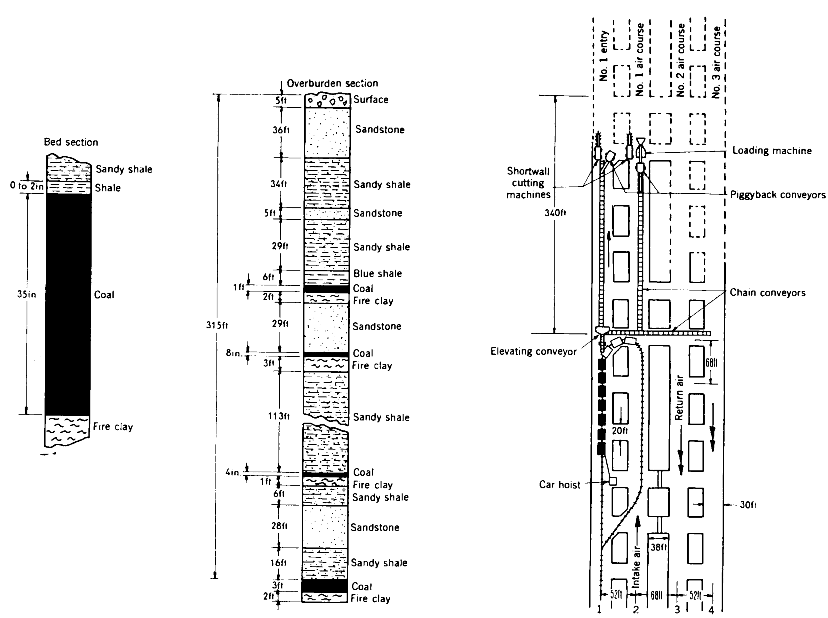


FIG. 17. Geologic section and method of development at Mine 1 — Mobile loading onto chain conveyors.

The mine was operated two shifts a day and produced an average of 1070 tons of raw coal (1049 tons of clean coal), using the following units:

Two mobile-loading piggyback to chain-conveyor units developing panel entries; nine men each.

One mobile-loading piggyback to chain-conveyor unit mining rooms; nine men.

A detailed study was made of one entry development unit only. Entries 30 ft wide were driven in sets of four on 52- and 68-ft centers, with cross cuts 20 ft wide on 68-ft centers. Entries were developed by a unit consisting of nine men, one mobile-loading machine, two piggyback conveyors, two short-wall cutting machines, two chain conveyors (in entries), one mother chain conveyor, one elevating conveyor, two hand-held electric drills, one roof-bolting machine, one supply truck, and one car hoist.

Operating Cycle

The cycle of operation was: timber, drill and cut, extend conveyor, blast and load coal. Power to operate the mining equipment was furnished at 250 V d.c.

Entries were advanced in pairs, with a 38-ft coal pillar between each pair. The first pair (entries 1 and 2) were advanced 340 ft. Equipment was then moved out and set up in entries 3 and 4. While these entries were being advanced, the bottom was brushed, the main-line track was extended in number 1 and 2 entries, and a new loading station was established 320 ft ahead of the old station. When the entries 3 and 4 were advanced 340 ft, the equipment was moved back through the last open cross cut to entries 1 and 2.

Roof Support

The roof was supported by three rows of props on each side of the chain conveyor, spaced 4 and 5 ft apart. When the roof required it, $\frac{3}{4}$-in. expansion-shell type bolts 2 ft long were installed on 5-ft centers. The roof bolts were provided with $6 \times 6 \times \frac{3}{16}$ in. steel bearing plates.

Cutting and Drilling

Five equally spaced holes 9 ft deep were drilled at a distance of 1 ft below the roof. The coal was bottom-cut and the conveyor was extended. Each hole was charged with four sticks of permissible explosives and blasted individually.

Loading and Haulage

Broken coal was loaded by a mobile-loading machine onto a piggyback conveyor which transferred it to a chain conveyor which discharged coal into $2\frac{1}{2}$-ton capacity steel, bottom-dump mine cars. Loaded cars were transported to the surface by a 15-ton trolley locomotive.

Rock dusting was done by hand on shift and by machine on week-ends, when required.

Crew Required

A unit crew consisted of the following men:

Section foreman	1
Loading-machine operator	1
Loading-machine operator's helper	1
Cutting-machine operator	1
Cutting-machine operator's helper	1
Conveyor man	1
Timberman	1
Utility or supply man	1
Boom man	1
Total	9

A unit crew produced an average of 150 tons of raw coal per shift, or 16.7 tons per man-shift. The average advance in each entry in a development group is 10 ft per shift, or a total of 20 ft for the group.

Mobile Loading into Shuttle Cars (Mine 8; Fig. 18)

This mine is operated in the Coalburg bed, which is 73 in. thick in the area. A shale parting 6-in. thick occurs 12–18 in. from the bottom of the coal bed. Only the coal above this parting (about 55 in.) was mined. The roof generally was firm sandstone, but in parts of the mine the immediate roof was shale. The overburden consisted of shales and sandstones and averaged 200 ft in thickness.

The mine was operated two shifts per day and produced an average of 1541 tons of raw coal (1376 tons of clean coal), using the following units:

One mobile-loading unit with shuttle cars, developing main entries; twelve facemen.

One conveyor mobile-loading unit with piggyback, developing panel entries and mining rooms; six facemen.

Equipment

Main entries 20 ft wide were driven in sets of six on 50-ft centers, with cross cuts 20 ft wide on 75 ft centers. These entries were developed by a unit consisting of twelve men, two mobile loading machines, two short-wall cutting machines with caterpillar trucks, three shuttle cars, two hand-held coal drills, and a 30-in. belt conveyor. Power to operate equipment is furnished at 250 V d.c.

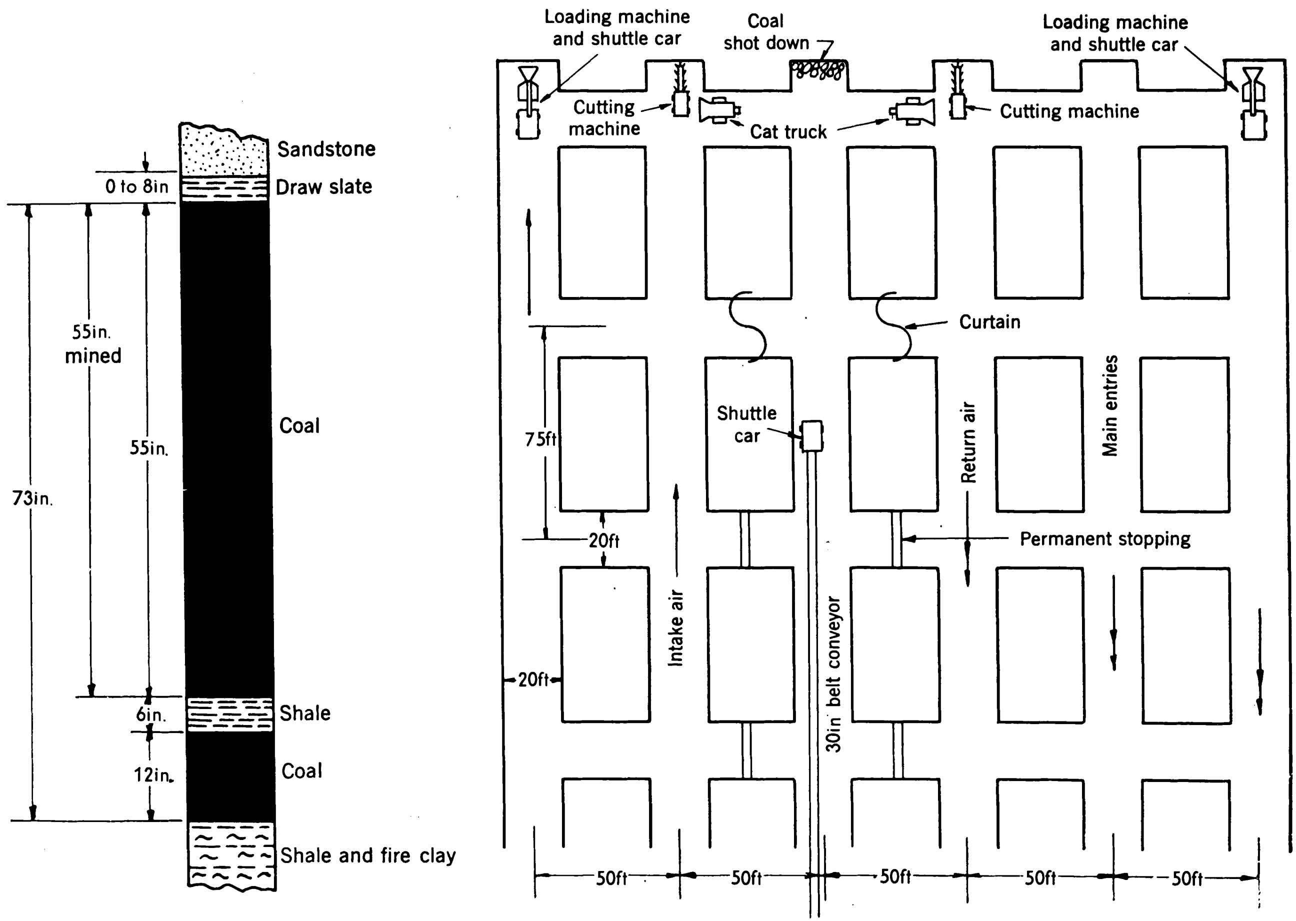


Fig. 18. Geologic section and method of development at Mine 8 — Mobile loading into shuttle cars.(2)

Operating Cycle

The cycle of operation was: Timber, drill, cut, blast, and load coal.

Drilling and Blasting

Six holes, each 7 ft long, were drilled in the coal. Three holes were drilled 1 ft below the roof, and three others were drilled 2 ft below the roof, with the lower holes being staggered between those in the upper group.

After drilling the entries were bottom cut and the drill holes were charged with four sticks of permissible explosive in each hole. Holes were then blasted individually.

Roof Support

Props were used to support the roof where it was sandstone. However where it was shale more timbering was required and 3 in. × 8 in. × 16 ft long wood crossbars were set on wood posts for roof support.

Loading and Haulage

The coal was loaded by mobile-loaders into shuttle cars and transported to the 30-in. belt conveyor in the No. 3 entry for transportation to the surface.

Rock dusting was done on shift by hand casting and offshift by machines.

Crew Required

A unit crew consisted of the following men:

Section foreman	1
Loading-machine operators	2
Loading-machine operators' helpers	2
Cutting-machine operators	2
Cutting-machine operators' helpers	2
Shuttle-car operators	3
Total	12

A unit crew produced an average of 509 tons of raw coal per shift or an average of 42.5 tons/man-shift. The average advance in each entry in a development group was 18.2 ft/shift, a total of 108.6 ft for the group.

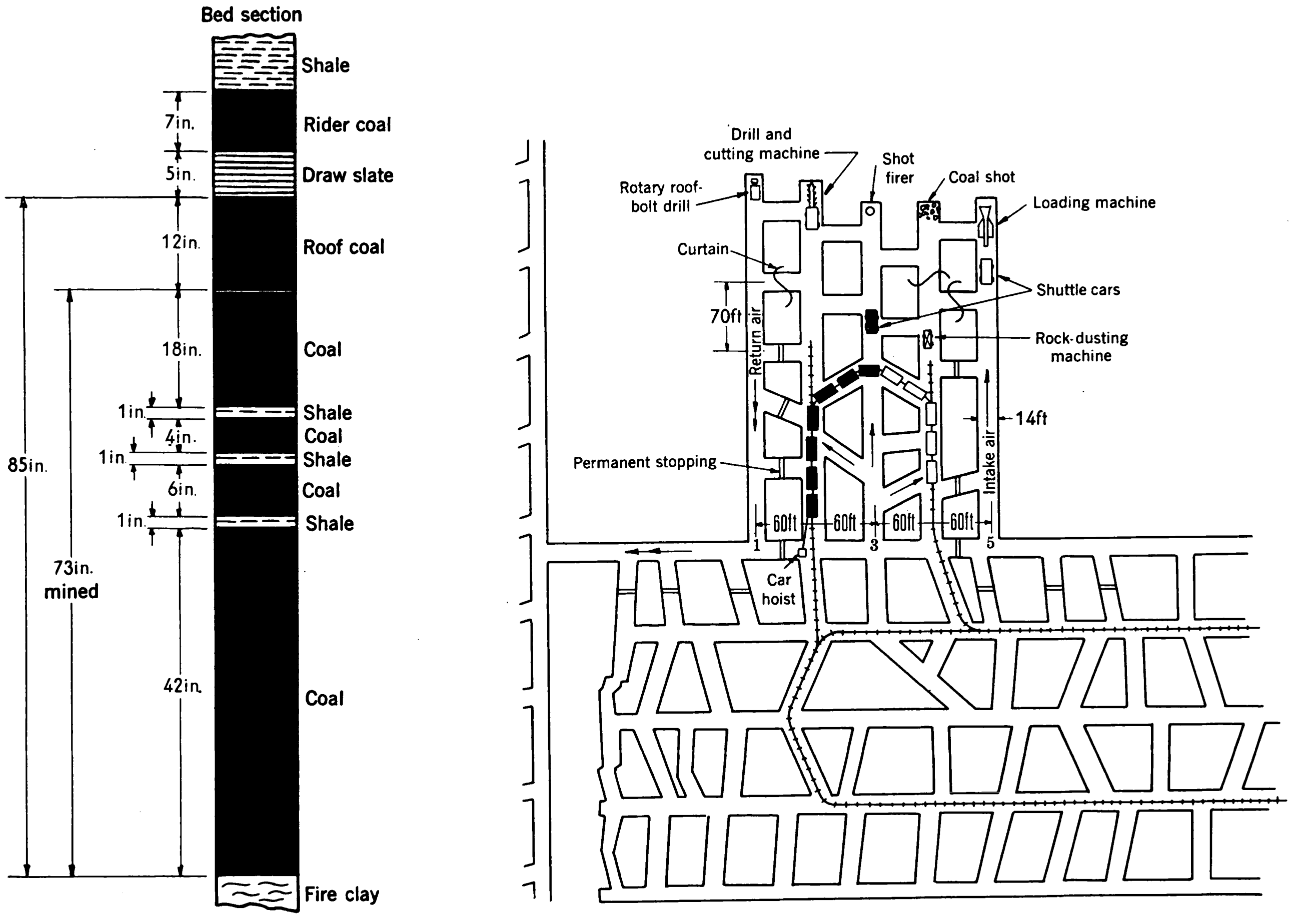


FIG. 19. Geologic section and method of development at Mine 11 – Mobile loading into shuttle cars.[2]

Mobile Loading into Shuttle Cars (Mine 11; Fig. 19)

This mine was operated in the flat-lying Pittsburgh bed, which averages 85 in. in thickness in the area. The overburden averaged 530 ft in thickness and consisted mainly of shales and sandstones. The equipment used by each development unit is shown in Fig. 19.

The cycle of operation was: Roof bolt, cut and drill, blast, and load coal.

A unit development crew, consisting of thirteen men, produced an average of 393 tons of raw coal per shift, or 26.4 tons/man-shift. The average advance in each entry in a development group is 13.2 ft/shift, a total of 65.9 ft for the group.

Track-mounted Loading into Mine Cars (Mine 9; Fig. 20)

This mine was operated in the Upper Kittanning (C′) bed, which averaged 59 in. in thickness in this area. The dip of the bed averages from 1° to 3°. The bed is overlain by an average thickness of 284 ft of overburden which consists mainly of sandstones and shales.

The equipment used by each development unit is shown in Fig. 20.

The cycle of operations was: Brush roof and bolt and timber the center entry; timber the other four entries, extend track, drill and cut, blast, rockdust, and load coal.

A unit development crew, consisting of $10\frac{1}{4}$ men, produced an average of 134 tons of raw coal per shift, or 9.7 tons/man-shift. The average advance in each entry in a development group is 6.1 ft/shift, a total of 30.4 ft for the group.

Hand Loading onto Chain Conveyors (Mine 3; Fig. 21)

This mine was operated in the Pocahontas No. 3 coal bed which averages 40 in. thick in this area. The bed is flat lying with few minor rolls or disturbances. The overburden averages 435 ft in thickness and consists of shales and sandstones.

The equipment used by a development unit is indicated in Fig. 21.

The cycle of operations was: Cut, drill, blast, load coal, extend chain conveyor, and timber. Some of these operations were carried on simultaneously.

The coal was hand-loaded into chain conveyors.

A unit development crew, consisting of $12\frac{1}{2}$ men, produced an average of 156 tons of raw coal per shift, or 12.5 tons/man-shift. The average advance in each entry in a development group is 5.5 ft/shift, a total of 33 ft for the group.

METHODS AND EQUIPMENT USED IN ROOM MINING

The following examples are taken from a study made by the U.S. Bureau of Mines (*U.S. Bureau of Mines, Information Circular* 7978).

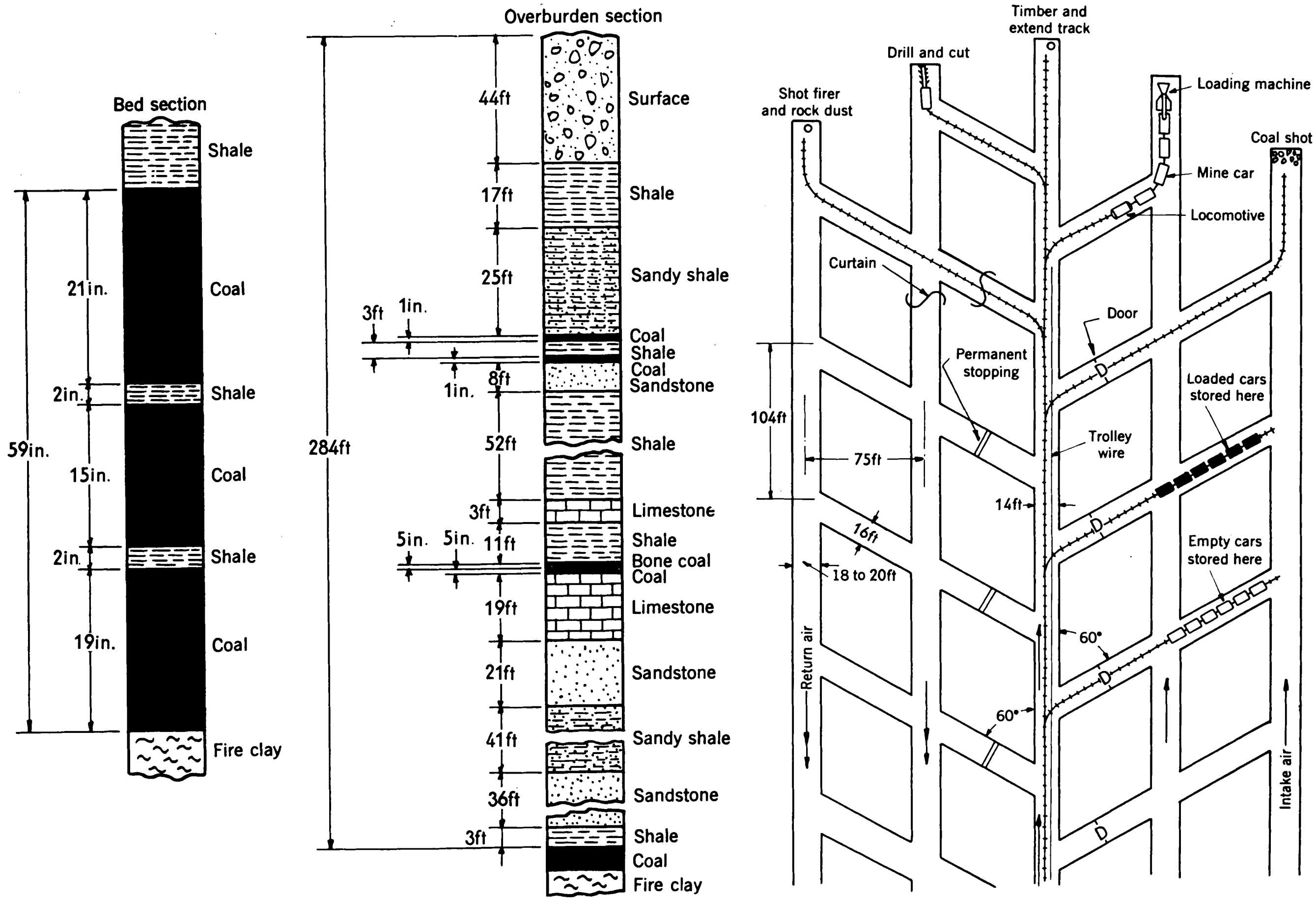


FIG. 20. Geologic section and method of development at Mine 9 – Track-mounted loading into mine cars.(2)

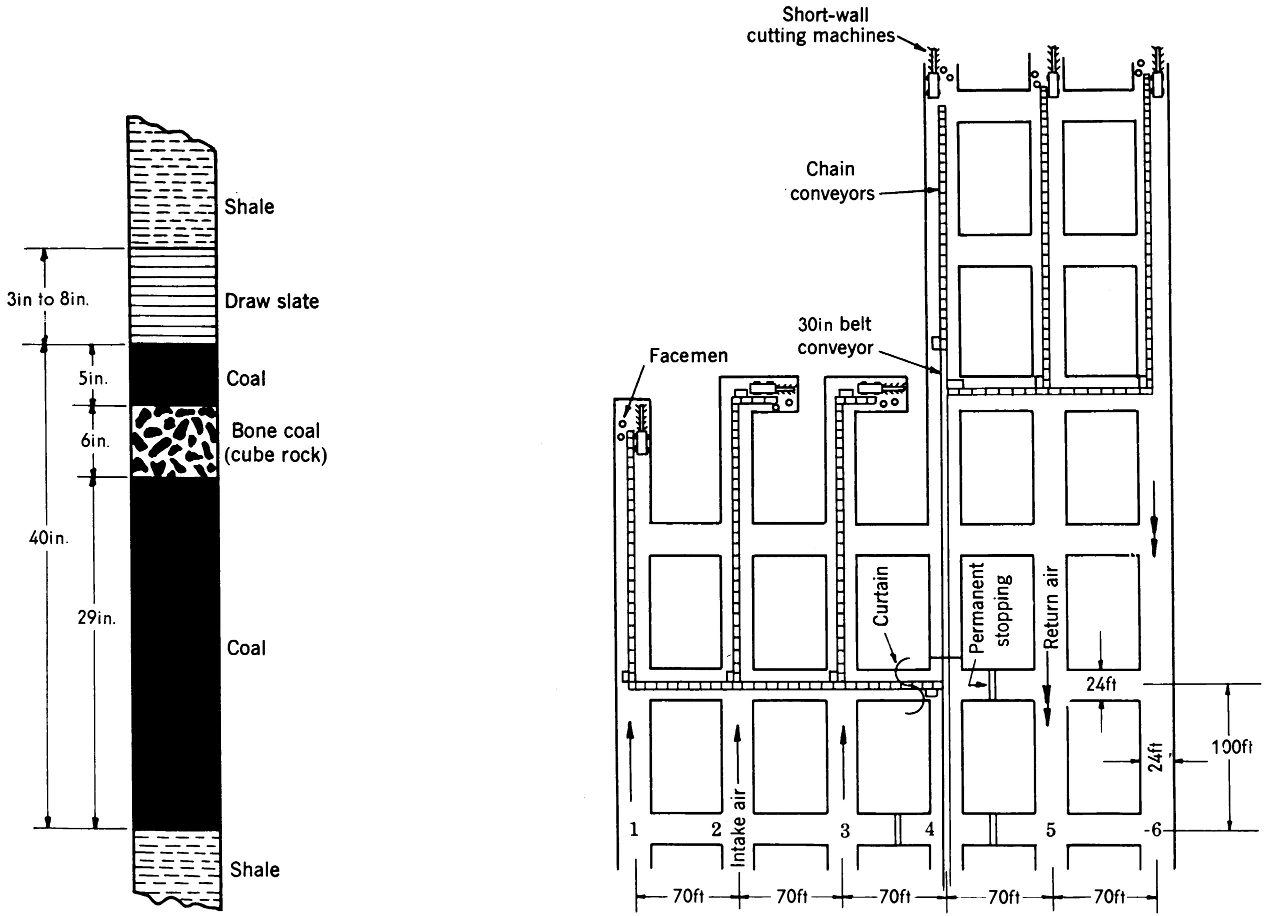


FIG. 21. Geologic section and method of development at Mine 3 – Hand loading onto chain conveyors.[2]

Mobile Loading into Shuttle-cars (Mine 7; Fig. 22)

This mine is operated in the Pittsburgh No. 8 bed which averages 62 in. thick in this area. The overburden consists principally of shales and limestones and averages 200 ft thick.

Rooms are driven 242 ft deep with cross cuts 15 ft wide on 80 ft centers, and were staggered. Rooms were mined in groups of seven alternately from one side to the other. A pillar 35 ft wide was left unmined between each group of rooms.

A room mining unit consisted of nine men, one short-wall cutting machine with truck, two post-mounted drills, three blowdown valves, one firing shell (compressed air), one loading machine, one roof-bolting machine, and two shuttle cars (one being a reserve). Power to operate the machinery was furnished at 275 V d.c. Rock dusting was done by hand on shift.

Roof Support

The draw slate was taken down, and the top coal was bolted. Roof bolts were recovered as rooms were completed.

Cutting and Drilling

Eight blast holes, each $2\frac{1}{2}$ in. in diameter and 9 ft deep, were drilled in the face of a room. Two holes were drilled 1 ft above the bottom, one on each rib; one hole in the center about 30 in. above the bottom, two holes, one on each rib, below the draw slate; and three holes approximately on the centerline of the draw slate, one near each rib, and one midway between ribs. The coal was undercut and blasted one hole at a time with a compressed air shell (8000–10,000 psi).

Loading and Haulage

The broken coal was loaded by a mobile loader into shuttle cars which transferred it to 6-ton capacity steel mine cars for haulage to the rotary dump at the slope bottom.

Crew Productivity

A unit crew, consisting of nine men, produced 200 tons of raw coal per shift which yielded 164 tons of clean coal. This was 22.2. tons of raw coal or 18.2 tons of clean coal for each man in the unit crew.

The average advance of each room or cross cut was 7.9 ft/shift or 55.3 ft for seven rooms.

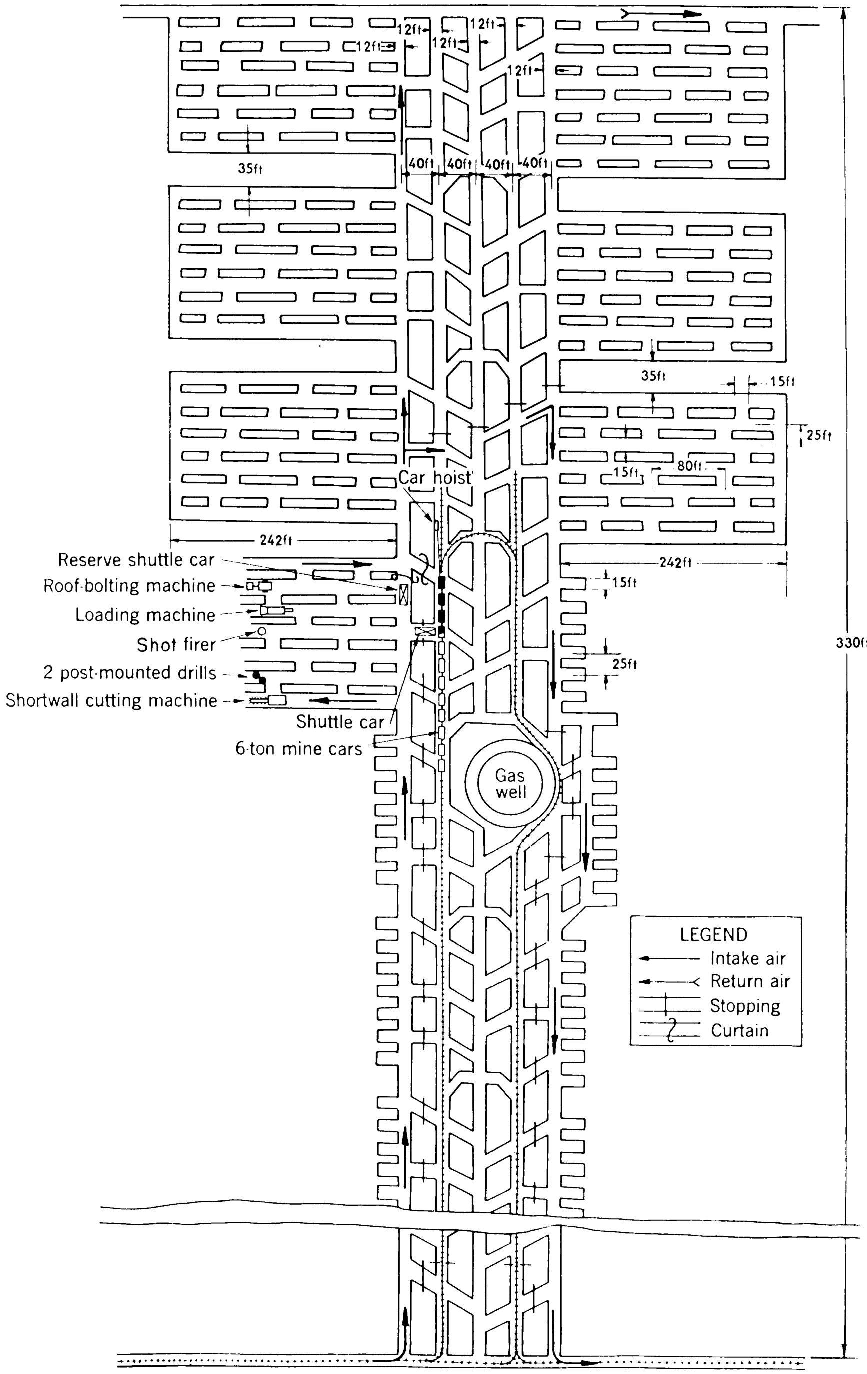


FIG. 22. Room-mining plan with a mobile-loading unit using two shuttle cars at Mine 7.[3]

Productive Time

Face haulage delays averaged 20 min due to mechanical difficulties with the shuttle car. Production delays (23 min) were caused by mechanical failure of the loading machine. Other delays included 20 min for timbering, and 51 min waiting on empties.

Travel time required 60 min, and lunch time was staggered, leaving 420 min for actual time at the face.

Total face delays were 114 min, leaving 306 min for productive time at the face per shift.

Mobile Loading onto Chain Conveyors (Mine 3; Fig. 23)

This mine is operated in the Cedar Grove bed which ranges from 36 to 56 in. thick in the area. The overburden consists of shales, sandstones, and several coalbeds and ranges up to 772 ft in thickness.

Rooms were driven in pairs 250 ft deep, and 40 ft wide on 50-ft centers with cross cuts 16 ft wide on 50-ft centers.

A room-mining unit consisted of nine men, one loading machine, two short-wall cutting machines; two hand-held electric drills; two chain conveyors, each 300 ft long; a 30-in. belt 2000 ft long, and a car hoist serving two units.

Power to operate the equipment was furnished at 250 V d.c. Rock dusting was done by hand on shift and by machine on the third shift.

Operating Cycle

The cycle of operation was: Timber, rock dust, drill, cut, pan up, blast, and load coal.

Roof Support

Rooms were timbered with posts set on 4-ft centers in four rows 8 and 10 ft from the left rib to the chain conveyor, and 8 and 10 ft from the right rib to the chain conveyor. Cross cuts were timbered with two rows of posts, each row 3 ft from the ribs and with posts on 4-ft centers. Wood crossbars 6 in. by 8 in. by 16 ft ong on 4 ft centers were set when required.

Cutting and Drilling

The face was bottom-cut and drilled with eight holes about 8 ft deep. These holes were placed about 8 in. below the roof and $2\frac{1}{2}$ ft from either rib on 5-ft centers. Each hole was charged with two to three sticks of permissible powder and detonated simultaneously.

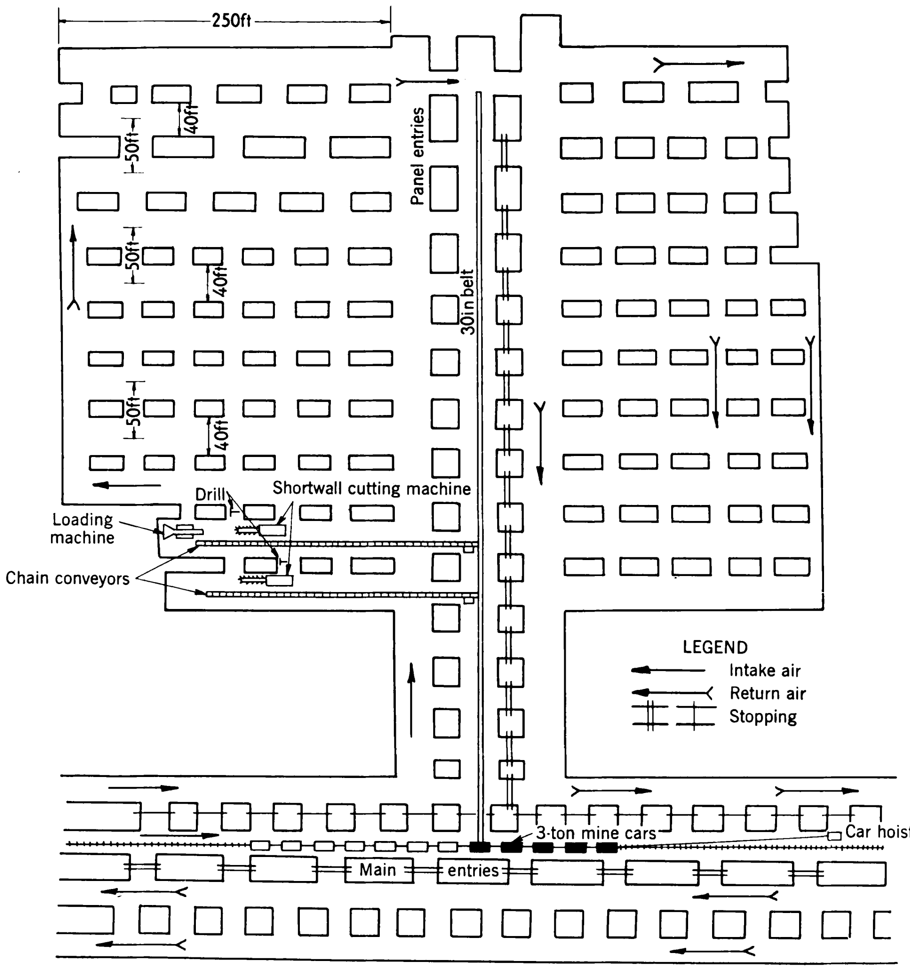


FIG. 23. Room-mining plan with mobile-loading equipment using chain conveyors at Mine 3.[3]

Loading and Haulage

Broken coal was loaded by mobile loading machine onto a chain conveyor which discharged coal to a 30-in. belt which carried it to 3-ton steel bottom-dump mine cars.

Crew Productivity

A nine man unit room crew produced an average of 172 tons of raw coal per shift which yielded 150 tons of clean coal. This is 19.1 tons of raw coal or 16.7 tons of clean coal for each man in the unit crew.

The average advance of each room or cross cut was 11.5 ft/shift or 23 ft of advance for two rooms.

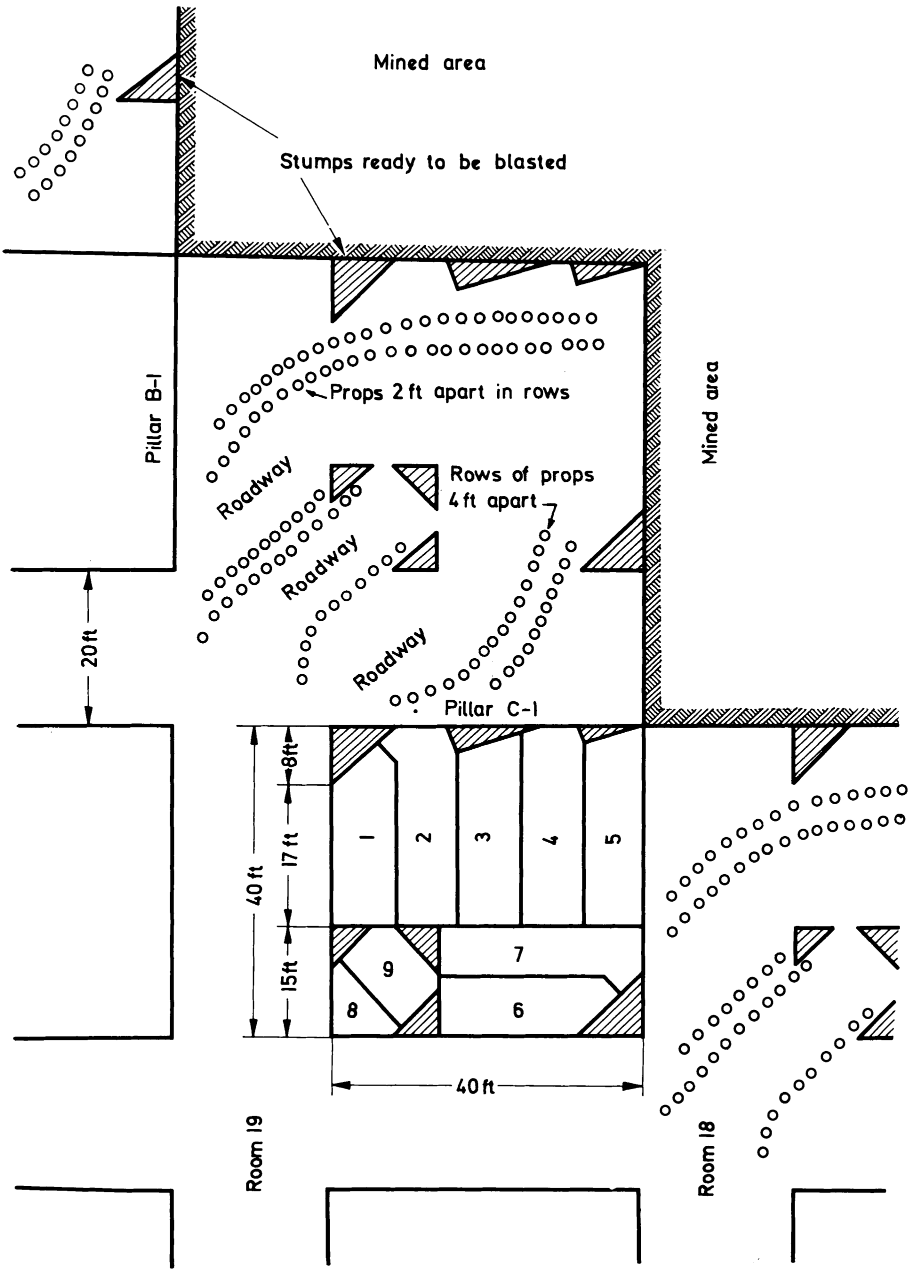


FIG. 24. The pocket-and-fender method for pillar extraction at Mine 58.[4]

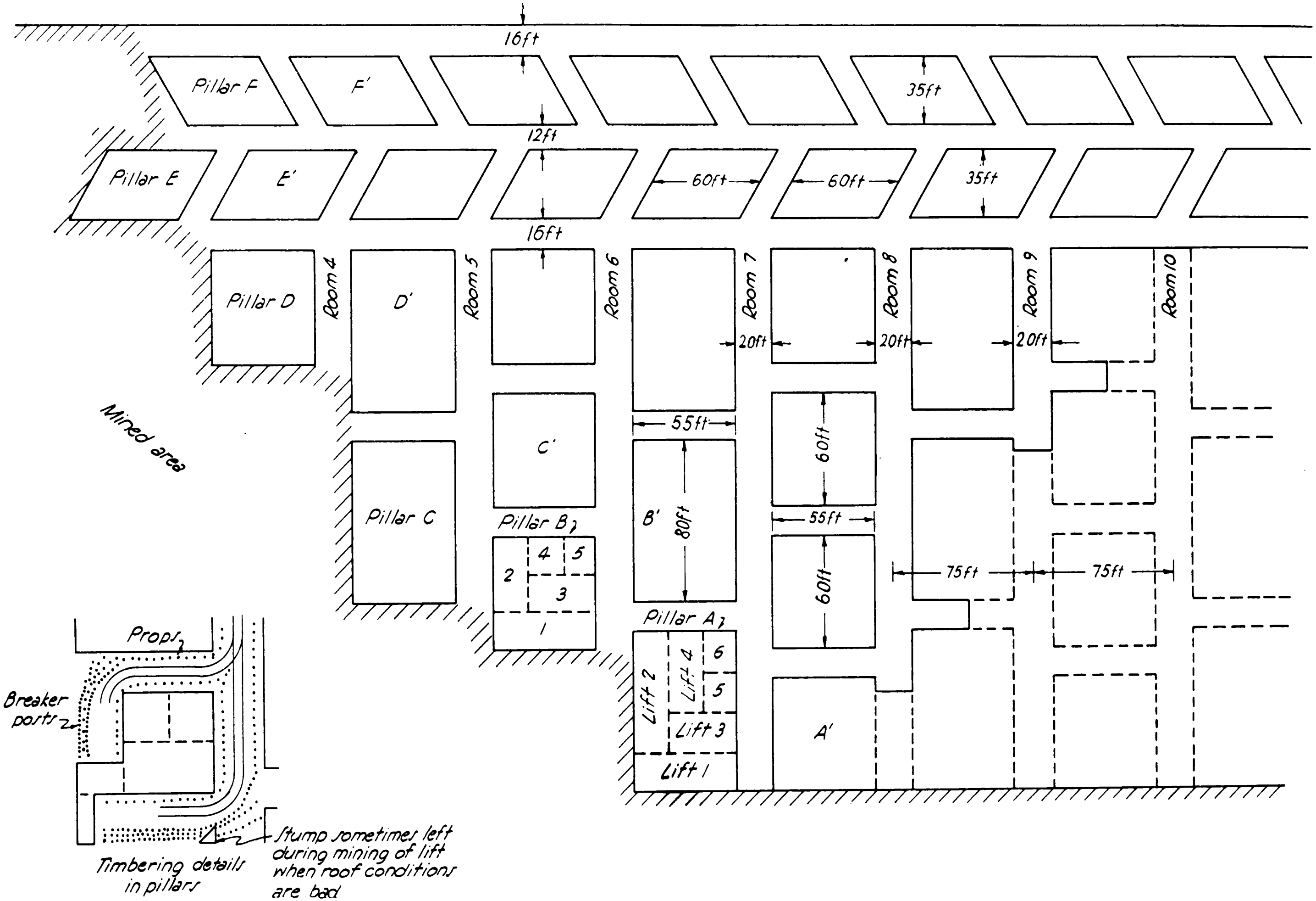


FIG. 25. The open-end method for pillar extraction at Mine 60.[4]

Productive Time

Travel time required 36 min and lunch time took 30 min leaving a maximum of 414 min available for work at the face. Face delays amounted to 174 min, leaving 240 min of actual productive time per shift.

PILLAR EXTRACTION WITH CONVENTIONAL EQUIPMENT

The two general methods commonly used in extracting pillars are: (1) The pocket-and-fender method, and (2) the open-end method.

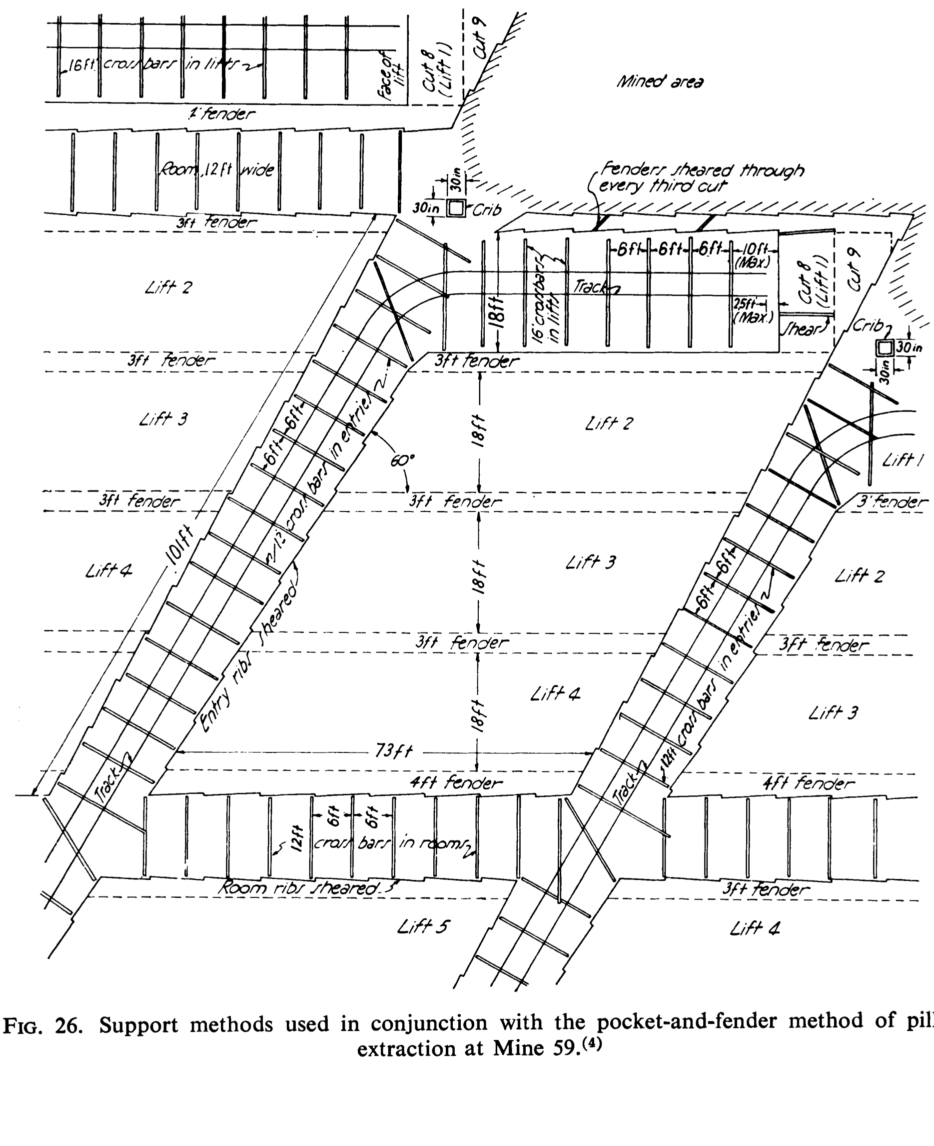


FIG. 26. Support methods used in conjunction with the pocket-and-fender method of pillar extraction at Mine 59.[4]

Pocket-and-fender Method

A series of cuts are taken into the pillar on the side away from the goaf and thin "fenders" of coal are left between the mining machine and the goaf for the support of the roof and protection of the crew from caving roof in the goaf.

This method is in widespread use but various amounts of coal are lost in these thin fenders or stumps which are left between the pockets or lifts and the gob area. This method does not provide as satisfactory a percentage of recovery as the open-end method but there are many mines where the latter system cannot be used because of adverse roof conditions.

Figure 24 shows a pocket-and-fender method used for the extraction of pillars in a mine in the upper Freeport bed which averages about 63 in. thick, and is covered with about 200 ft of overburden.[4]

The sequence of cuts is indicated by the numbers. The large triangular stumps are blasted, and the roof over the mined area crushes the small fenders, breaks the props, and caves.

Open-end Method

A series of slices are taken off the pillar on the side adjacent to the gob area. This method gives the highest percentage of coal recovery but in many cases it cannot be used because the roof is not good enough, and working conditions become hazardous.

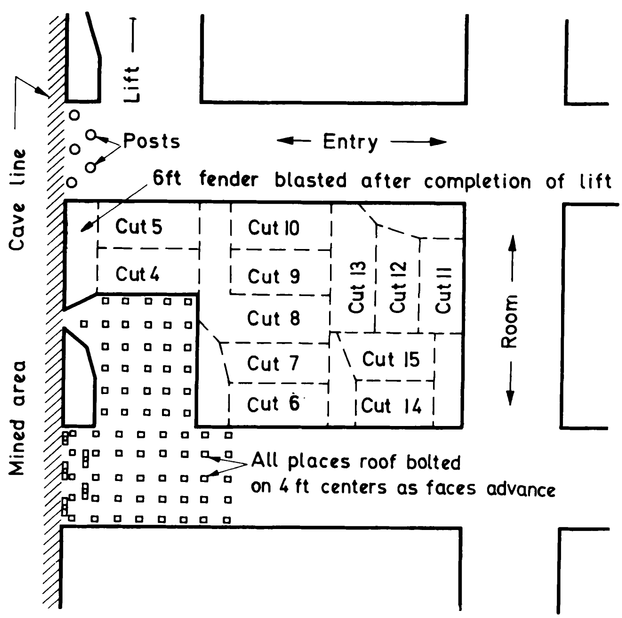


FIG. 27. Support methods used in conjunction with the pocket-and-fender method of pillar extraction at Mine 64.[4]

Figure 25 illustrates the open-end method for pillar recovery as used for extraction of pillars in a mine in the Elkhorn No. 3 bed which averages 45 in. thick and varies from 30 to 60 in. thick. Cover over this bed ranges from a few feet to 1000 ft under the mountain tops.[4]

Roof Support During Pillar Extraction

In addition to the fenders or stumps which are left to prevent premature caving of the roof and to prevent caved rock from the gob area from entering the working places additional roof support in the form of roof bolts, props, timber, cribs, or hydraulic props will probably be required.

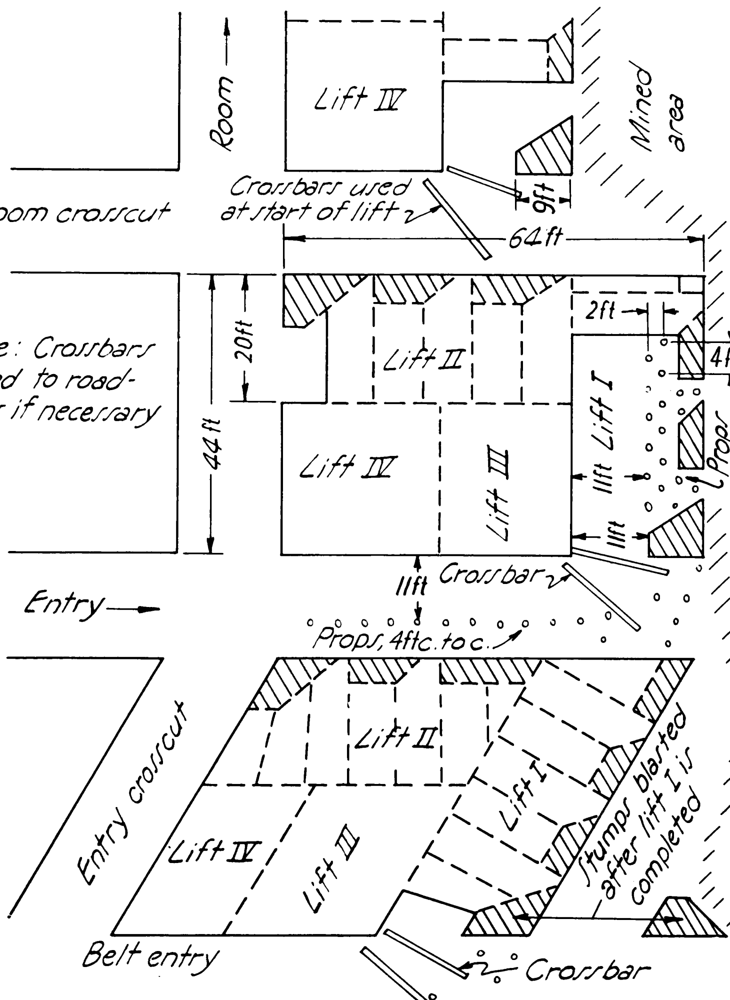


FIG. 28. Support methods used in conjunction with the pocket-and-fender method of pillar extraction at Mine 66.[4]

Figures 26–28 illustrate support methods used in conjunction with pillar extraction by the pocket-and-fender method.

Figure 29 illustrates a timbering method used in one mine where pillars are extracted by open-end methods.

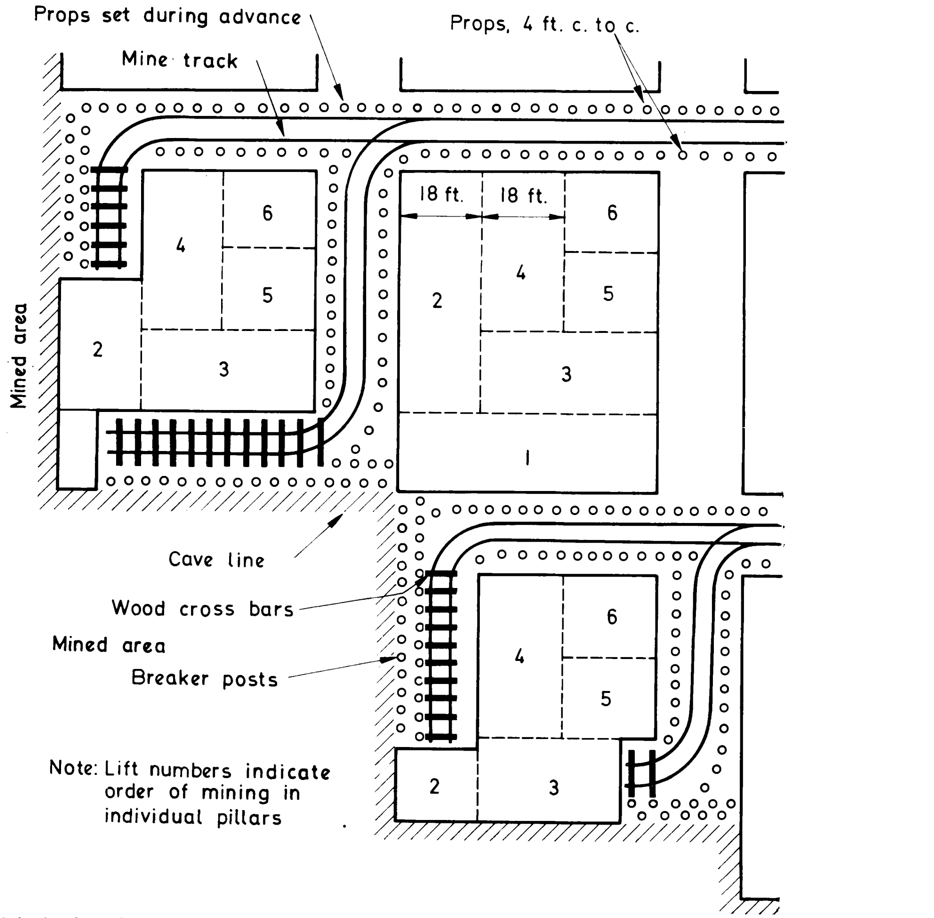


FIG. 29. Timbering details and sequence of lifts used in conjunction with extraction of pillars by the open-end method at Mine 65.[4]

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4. HALEY, W., SHIELDS, J. J., TOENGES, A. L., and TURNBULL, L. A., Mechanical mining in some bituminous coal mines. Progress Report 6. Extraction of pillars with mechanized equipment, *U.S. Bur. Mines I.C.* 7631, April, 1952.

CHAPTER 3

PILLAR MINING SYSTEMS CONTINUOUS (NON-CYCLIC) MINING

CONTINUOUS MINING MACHINES

Continuous mining machines cut or rip the coal from the seam and load it onto conveyors, or onto shuttle cars, for transportation to the main haulage system. In continuous mining the cutting, drilling, and shooting cycles required in conventional mining are thus eliminated.

Continuous mining machines may be divided into two broad classes; that is the "rippers" and the "borers".

Ripping Machines

These are characterized by a series of parallel disks or chains which are equipped with coal cutting picks. These disks or chains rotate about an axis which is parallel to the face and the picks rip the coal from the face.

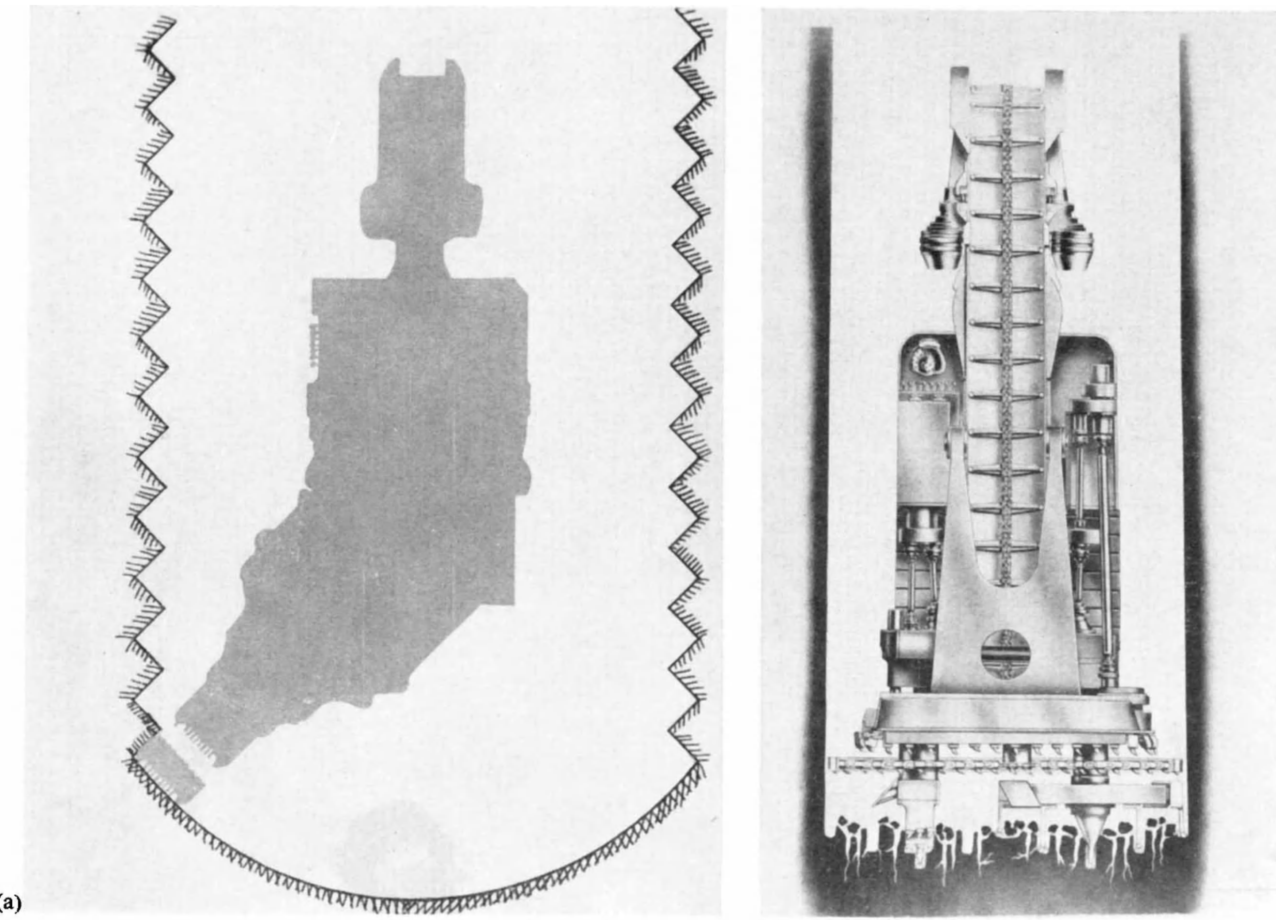
Boring Machines

These are equipped with cylindrical cutting heads with cutting picks set on the edges of the cylinders. These are advanced into the coal and cut a series of concentric annular grooves.

Most types of ripper machines are articulated, that is the head of the machine can swing to the left or right, within a limited angle, and these machines are therefore somewhat more flexible in operation than are the boring machines.

Joy Continuous Miner

There are various models of Joy continuous miners to suit different seam conditions. One of the recent models (6CM) is shown in Fig. 2. The machine consists of a cutter head or ripper head, a main chassis, and a loading unit. The ripper head is fitted with five multi-pick chains and is driven by 100 hp electric motors. A single disk containing cutter picks is fitted on either side of the ripper bar. The 6CM is fitted with a 7-ft ripper bar and can cut up to a height of 10 ft.



(a) (b)

FIG. 1. Continuous mining machines: (a) ripping machine; (b) boring machine. (Joy Mfg. Company).

The ripper jib can be swung on either side and is attached to the main chassis, which is mounted on two tracks powered by two separate motors.

Cut coal is delivered to the conveyor by the ripper head and also by two gathering arms which are fitted on either side of the conveyor. The rear conveyor can be swung 45° to either side of the machine.

FIG. 2. Ripper-type continuous miner. (Joy Mfg. Company.)

Method of Operation

The machine, with its head in the retracted position, is moved forward on its crawlers until the cutting bits are just touching the face in the center of the room. The head is then swung to its limit to the right and lowered until the bits are about to touch the floor. It is then, with the cutting chains running, advanced 24 in. into the coal and is gradually raised vertically, taking out the coal above as it rises, until the desired roof line in the coal is reached (maximum 10 ft). The head is then

FIG. 3. Joy 6CM continuous miner at the face. (Joy Mfg. Company.)

retracted, taking out any loose coal left below roof level. Thus a block of coal, 42 in. wide and 24 in. deep, is removed from the seam.

The head is again lowered, swung about 3 ft to the left and again sumped forward 24 in. in the seam, and this operation is repeated across the whole width of the room.

Lee–Norse Miner

The Lee–Norse miner rips coal from the seam by means of a cutter head which consists of four rotating disks which are equipped with standard cutter picks. The arms holding the cutter head are caused to oscillate by means of motor-driven eccentrics.

The Lee–Norse miner is mounted on crawlers and cuts a width of $8\frac{1}{2}$ ft. The cutter-head does not swing sideways therefore it cuts on a straight line until it has advanced to the end of a cut. It then moves back on its crawlers and maneuvers into position for the next cut.

Method of Operation

The machine advances on its crawlers with the cutting head operating and sumps into the coal at the top of the seam. The coal is ripped downwards and is collected by gathering arms and delivered into the single-chain conveyor and discharged at the rear of the machine.

Characteristics

The Lee–Norse miner is a relatively simple low-cost machine which has a high productive capacity and the larger models of machines have produced an average in excess of 800 tons of raw material per shift.

Where mining conditions are severe, bit costs may be as much as 10 cents per ton but costs on the order of 3–5 cents/ton are more common.

Maintenance labor runs generally from 10–15 cents/ton and parts from 10 to 20 cents/ton.

Goodman Continuous Miner

The cutting head of this machine consists of two large cutting arms rotating in opposite directions, and a trimming chain. All these are fitted with cutting picks. Each large arm is equipped with a double core barrel, two short arms, and two long hinged arms which lock in the extended positions. Wedge-shaped cutter bars at the top and bottom of the chain break off that section of the face which is not accessible either to rotating arms or the cutting chain.

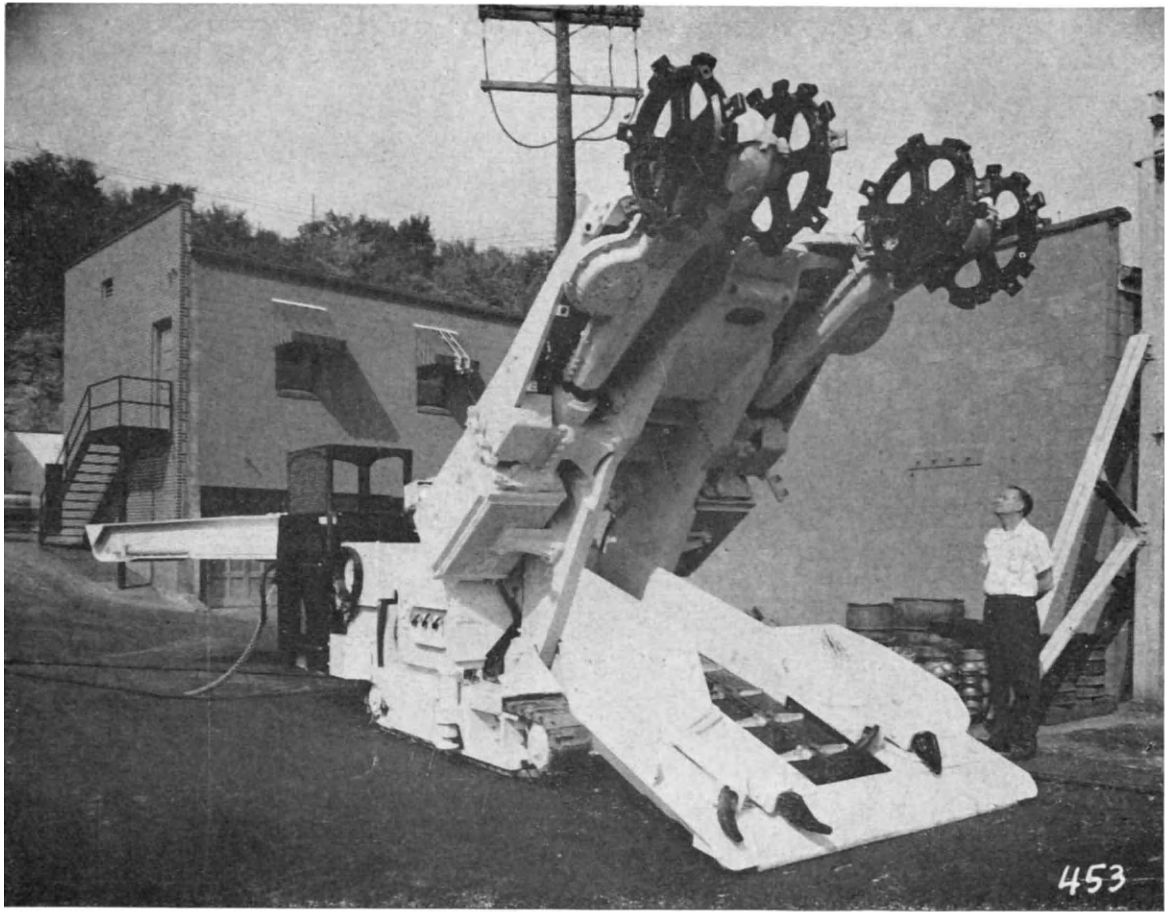


FIG. 4. Lee–Norse CM68 continuous miner. (Lee–Norse Company.)

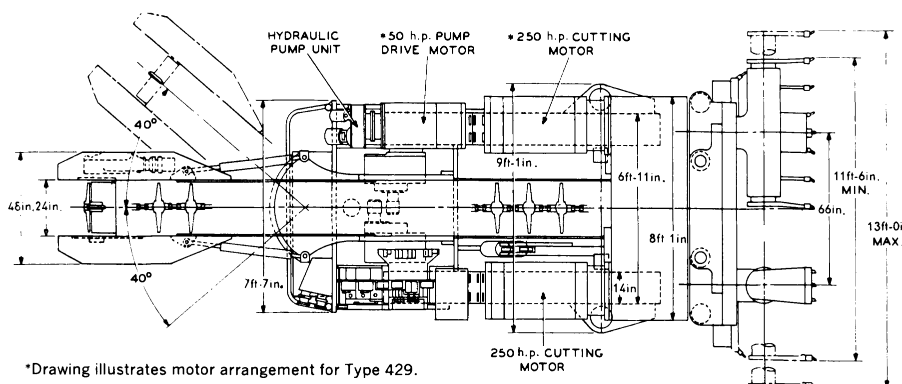
The actual section cut by the machine may be varied up to $7\frac{1}{2}$ ft high and $13\frac{1}{2}$ ft wide. The machine moves on crawler tracks. A high speed chain conveyor collects the coal in front and carries it through the machine. The discharge height can be altered as desired.

The tail conveyor can be swung 40° in either direction.

Method of Operation

The machine moves up to the face on its crawlers. The rotating units, geared together and equipped with bit-filled cutting arms, and a center core barrel, cut and break out the coal.

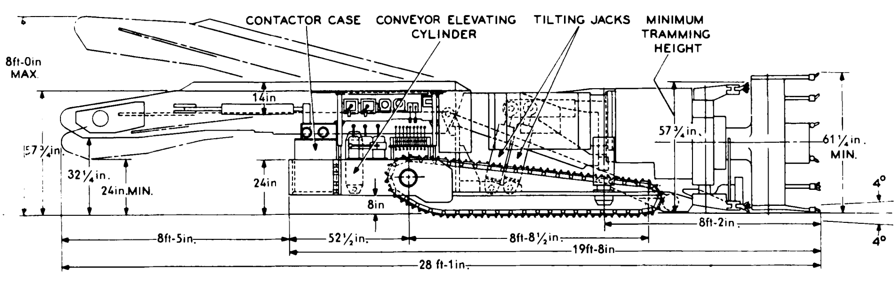
Dimensions and specifications • Types 427, 428, 429



*Drawing illustrates motor arrangement for Type 429.

Type 427 has one cutting motor only (250 h.p.) on left-hand side of machine and one hydraulic drive motor (50 h.p.) on right-hand side.

Type 428 has two motors (250 h.p. ea.), one on each side of machine. A power take-off from the left-hand motor drives the hydraulic pumps.



Rotor speed:	15 rpm.	Conveyor chain speed:	320 ft/min	
Tractor speed:		Weight:	427—91,000 lb	
cutting	0 to 6½ ft/min		428—94,500 lb	
tramming	up to 26 ft/min		429—99,000 lb	

FIG. 5. Goodman borer continuous miner. (Goodman Mfg. Company.)

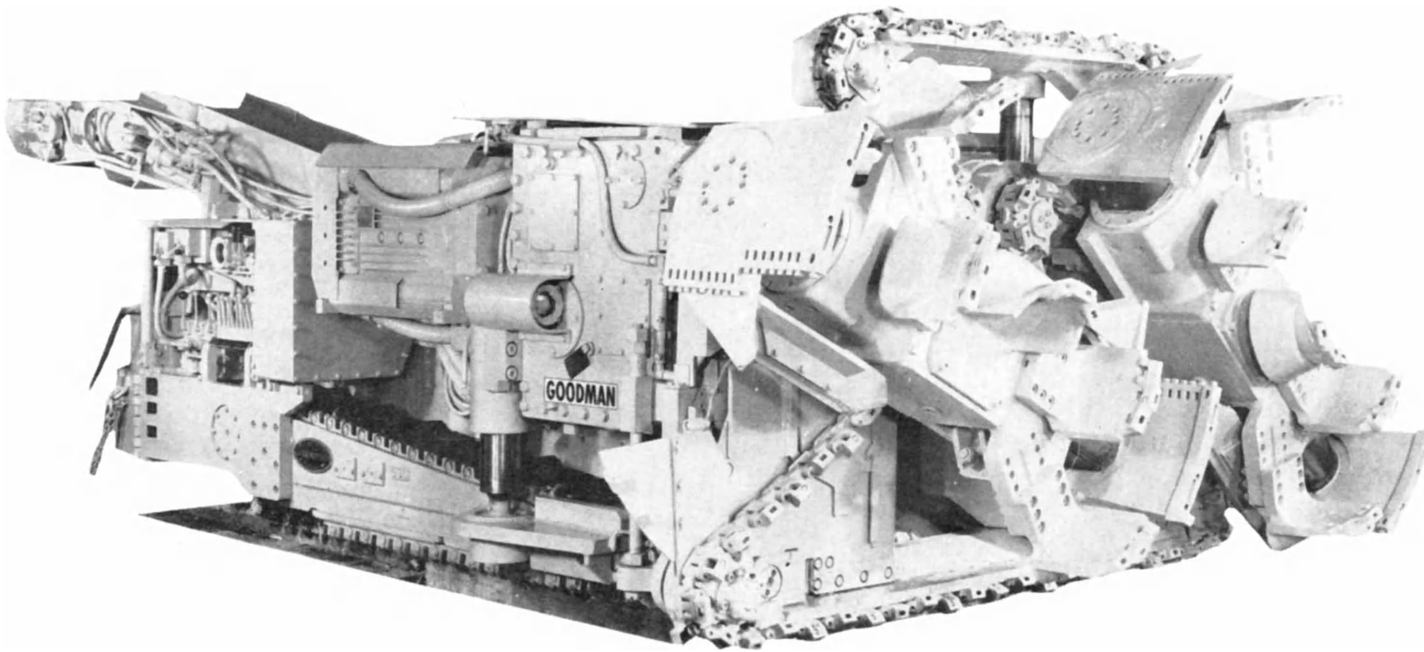


FIG. 6. Goodman Type 429 borer continuous miner. (Goodman Mfg. Company.)

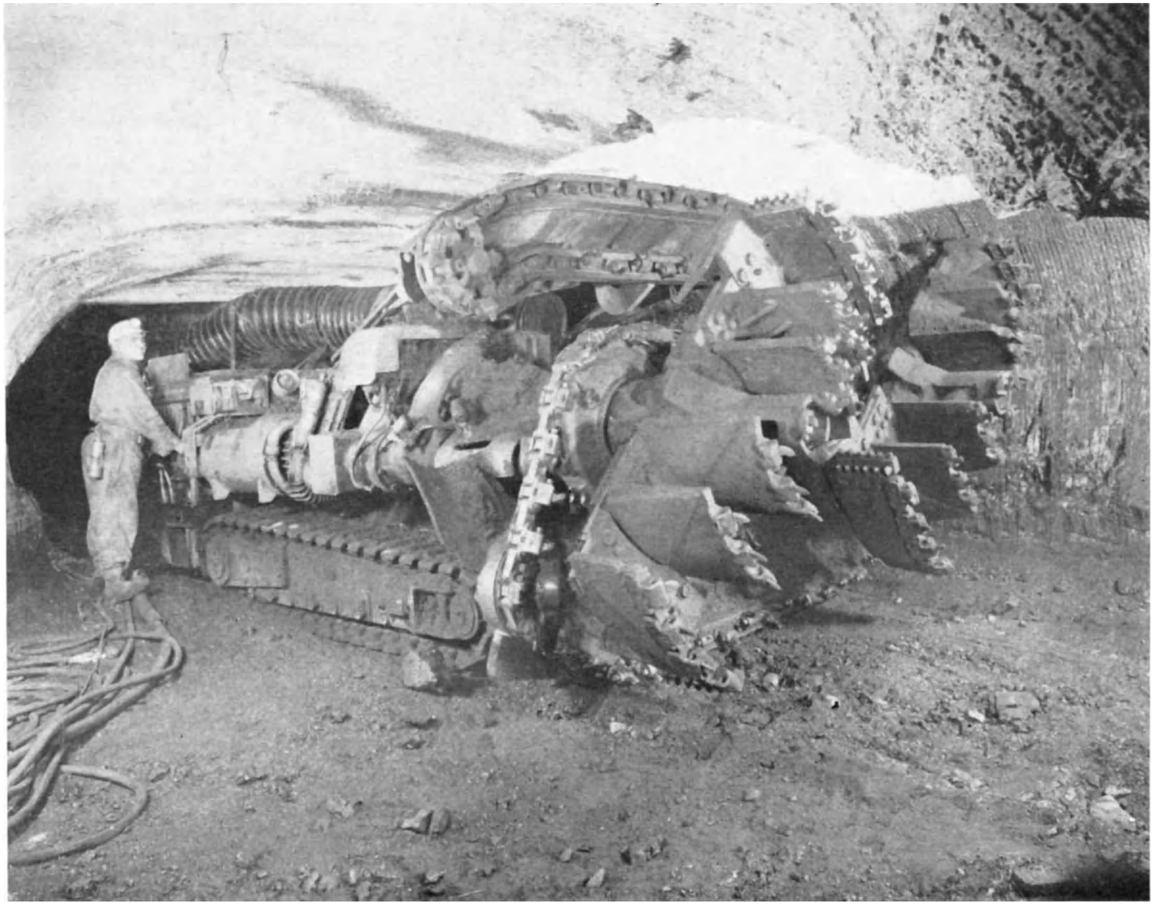


FIG. 7. Joy twin borer continuous miner. (Joy Mfg. Company.)

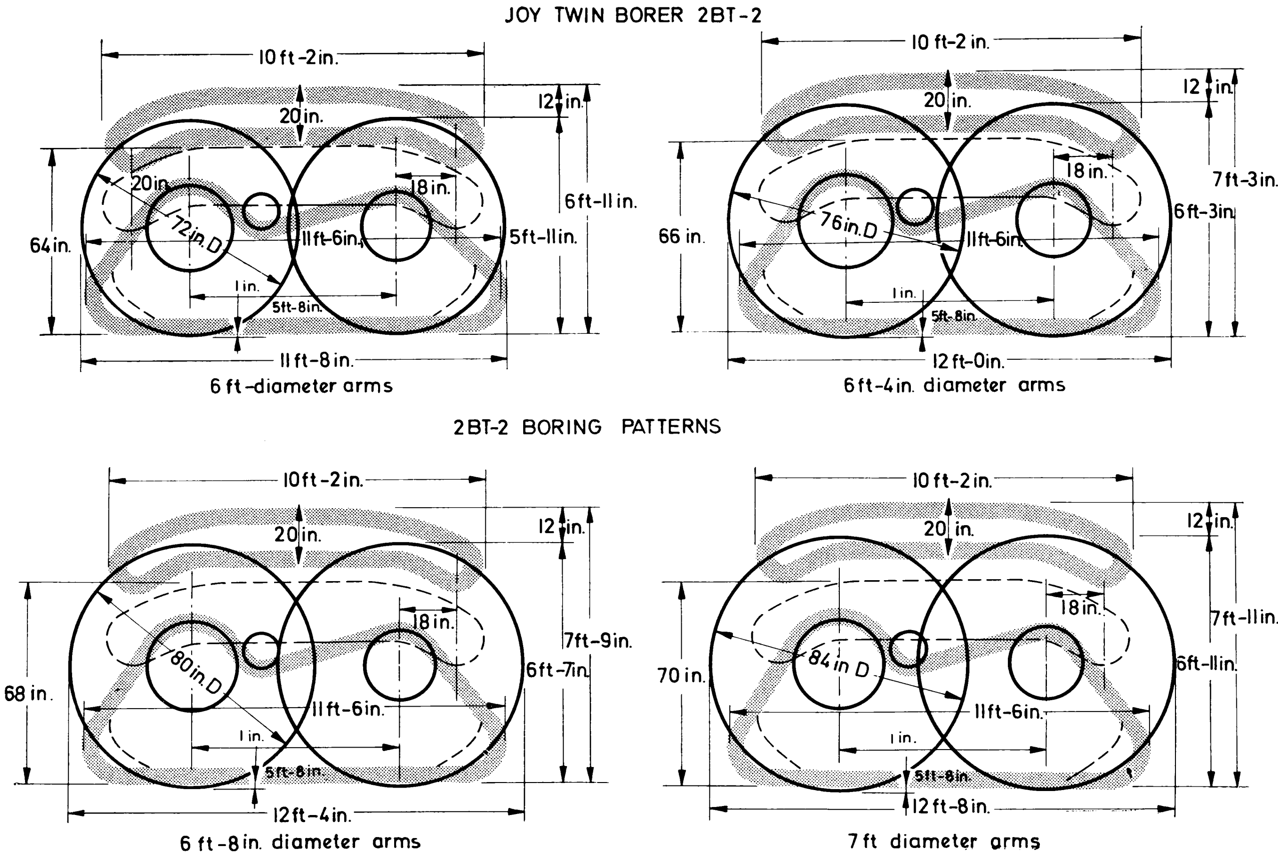


FIG. 8. Joy 2BT 2 twin borer boring patterns. (Joy Mfg. Company.)

As the machine bores ahead, the trimming chain cuts free, at the top and bottom, the center wedges of coal not reached by the rotating arms. The chain also widens the path at the bottom on each side, and can be arranged for cutting a wider than normal top to allow for cross barring.

Coal taken from the face drops to the bottom, is pushed to the center and up the throat of the conveyor, by plows on the rotating arms. The discharge conveyor can be swung 40° to either side, a big advantage when turning cross cuts or pulling pillars.

Joy Twin Borer Continuous Miner

The Joy twin borer uses the boring principle to cut a full face at a maximum rate of 8 tons a minute. The machine is crawler mounted and has two sets of boring arms and two sets of trim chains, one of which cuts an arched shape for tool control. The miner's full face cut varies from 5 ft 11 in. to 7 ft 11 in. high and 11 ft 8 in. to 12 ft 8 in wide. It trams while boring at up to 4.5 ft/min; trams from mine openings at up to 28.5 ft/min. All cutting surfaces retract hydraulically for roof and wall clearance.

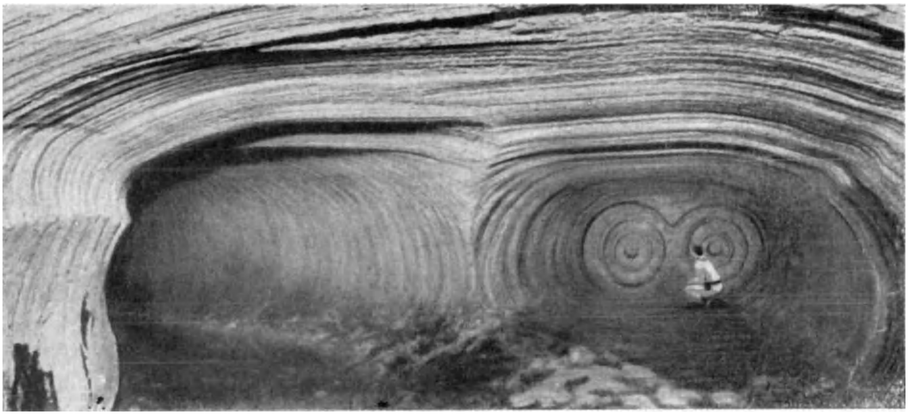


FIG. 9. Entry driven by Joy twin borer. (Joy Mfg. Company.)

Boring-arm diameters can be changed in 4-in. increments from 6 to 7 ft by using arms of different lengths, and raising or lowering the main transmission 2 in. for each change. Additional heights up to 12 in. are cut with the upper trim chain, which is quickly raised hydraulically to any desired position.

Boring arms are equipped with internally mounted hydraulic cylinders which can retract each arm 12 in. overall. The center section of each arm and the cylinders remain the same, regardless of diameter bored. The boring arm assembly is driven by an 8-in. diameter, alloy steel output shaft from the main boring arm transmission.

Jeffrey Colmol

The Jeffrey Colmol is a full face machine, and is available in different models to mine seams from 38 in. to 60 in. thick. The mining width varies from 9 ft 9 in. to 10 ft 1 in.

The Colmol is a pure rotary-head machine since it relies solely on the action of its rotors for breaking the coal. It has two heads of rotors, a top head of eight or ten single-ended rotary breaker arms and a bottom head of five of varying design.

Each breaker-arm has a central pilot tool which bores a hole ahead of the rotating arms. The core breakers on the arms break off the coal which is left protruding by the action of the bits. These arms are so arranged that they overlap to cover the entire area within the width and height of their outer reach.

The arms rotate at relatively slow speeds (less than 60 rpm) and this slow-speed operation tends to reduce degradation and to produce the larger sizes of coal.

The arms on one side of the machine rotate in one direction while the arms on the other rotate in the opposite direction. This opposed rotation prevents the machine from drifting and also sweeps the coal toward the center where it is picked up by the conveyor.

The machine has a rate of penetration of 0–36 in. per min, a tramming speed of 20 ft/min, and weighs approximately 52,000 lb.

Jeffrey 100-L Continuous Miner

The Jeffrey 100-L is equipped with twin augers which have cutting bits set around the edges of the scrolls. Cutting augers are available in 20, 24 and 28 in. diameters to give maximum mining heights of 30, 34 and 41 in. respectively. The effective cutting length is 5 ft. The machine has three principal sections: (1) the head, which utilizes the twin augers for the mining operation; (2) the central part, which contains the drive motor; and (3) the conveyor section, which delivers the mined coal onto the room conveyors.

Method of Operation

The machine is maneuvered by ropes in the manner of a short-wall cutter. In operation the machine is positioned at one side of the heading with both augers at floor level. The ribside auger is raised and the machine is sumped into the face. The ribside auger is then lowered and the other auger raised and the machine is hauled across the face by rope in the manner of a short-wall cutter.

A gathering conveyor, consisting of a single strand chain with cast-steel flights, gathers the loose coal from between the augers and delivers it to the rear of the machine where it is discharged onto a bridge conveyor.

Hydraulically operated gathering plows on both sides of the machine help to move and retain the mined coal within pick-up limits of the augers.

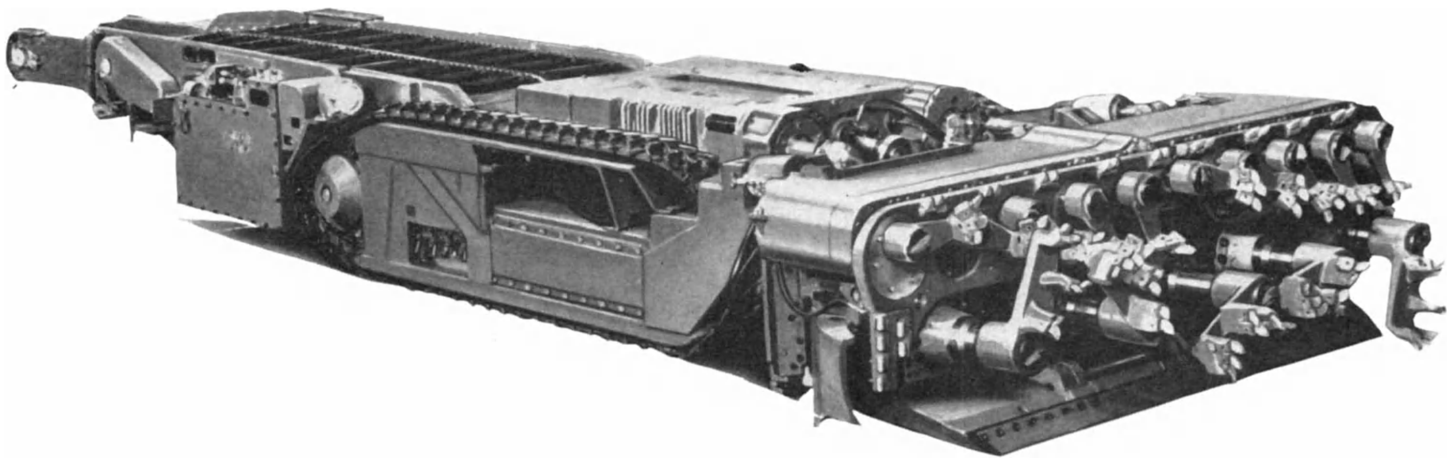


FIG. 10. Jeffrey Colmol continuous miner. (JeffreyM fg. Company.)

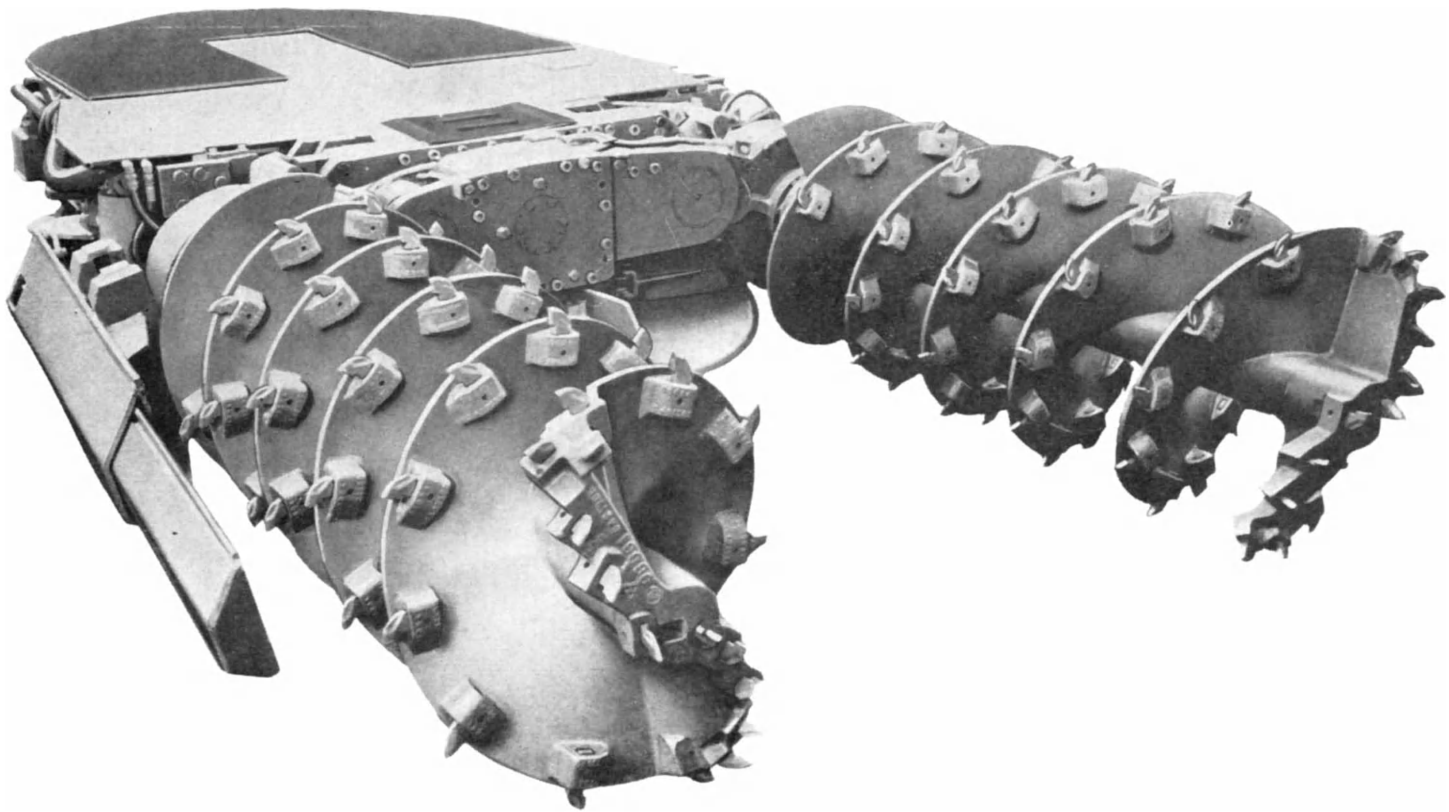


FIG. 11. Jeffrey 100-L continuous miner. (Jeffrey Mfg. Company.)

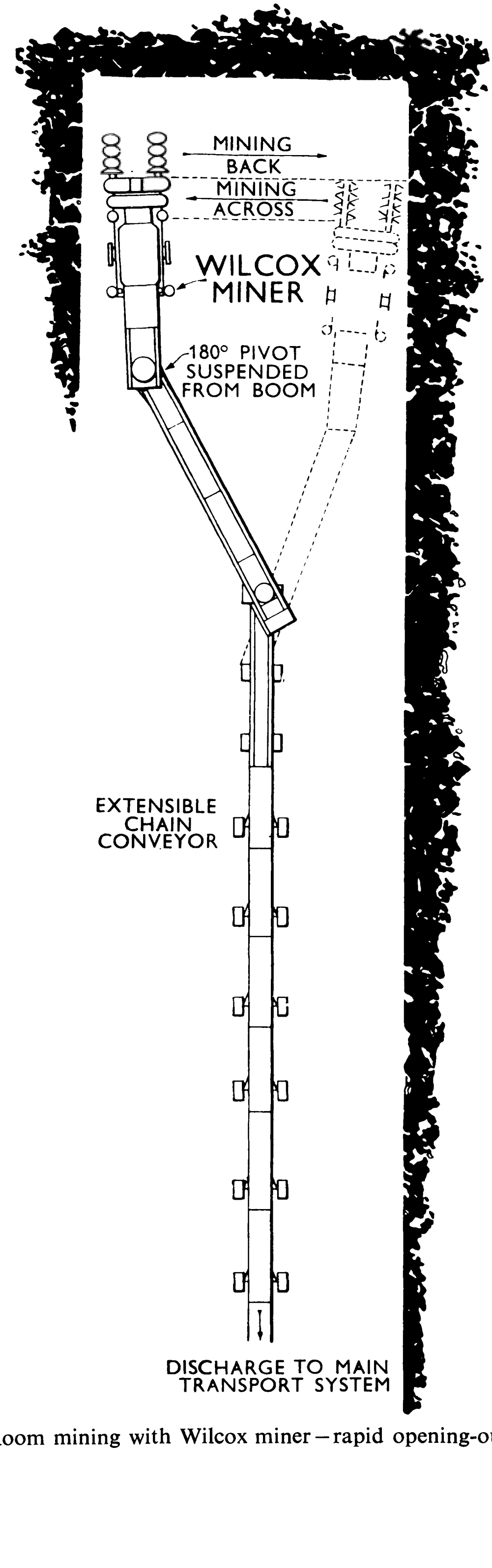


FIG. 12. Room mining with Wilcox miner – rapid opening-out system.

The machine weighs approximately 19,000 lb.

Although this machine does not have the capacity of other types of continuous miners it is simple and comparatively cheap and can mine wide faces.

The Crawley–Wilcox continuous miner is very similar in construction to the Jeffrey 100-L and the methods of operation are similar. Figure 12 illustrates a method of applying these machines.

Auger Mining

The Cardox–Hardsocg auger is equipped with a cutting head 3 ft long and 24 in. in diameter. A number of picks are fitted to the periphery of the cutting head. These cut an annular ring of coal which is broken up by a center bit and is then conveyed by archimedian screw sections to the mouth of the hole.

This machine is equipped with a 25 h.p. electric motor and can bore holes up to 80 ft deep at a rate of 2.7 ft/min.

An auger built for the Oliver Springs, Tennessee, mine of the Wind Rock and Coal Company by the Salem Tool Company bores 34-in. holes to depths of 100 ft.[1] The machine has separate power and drilling units. The power unit weighs $8\frac{1}{2}$ tons and is equipped with two 50 h.p. motors, hydraulic pumps and a 250 gal. tank of hydraulic oil. Hydraulic power is transmitted to the drilling unit through 50 ft hoses. The power unit is self-moving on two hydraulic skids.

The drilling unit weighs $9\frac{1}{2}$ tons. It is self moving on two skids and when it is positioned for drilling it is secured in place by two roof jacks at the rear of the unit. Rotational speed of the auger may be varied from 0 to 37 rpm but the best results are obtained when the head turns at the maximum speed. Auger sections are 5 ft long.

Two men are needed to operate the auger while a third drives a shuttle car between the auger surge car and the loading ramp. As the cutting head advances auger sections are pulled from the previously drilled hole and added in the new hole.

When operating in clean coal the auger takes an average of 1 hr to drill a 100-ft hole. It takes an average of 20 sec to pull back the carriage and be ready to add a new auger section. The new auger section is added in 15 sec. Each 100-ft hole yields about 23 tons of coal. Six inches of coal is left between adjacent holes. This gives a theoretical recovery of 50 per cent of the coal.

Auger Mining in Great Britain[5]

A Cardox–Hardsocg auger was installed in the High Main Seam (4 ft 3 in. thick) at Hucknall Colliery. After some modification the machine worked on a production basis of eleven shifts per week yielding an output per man-shift of 12 tons.

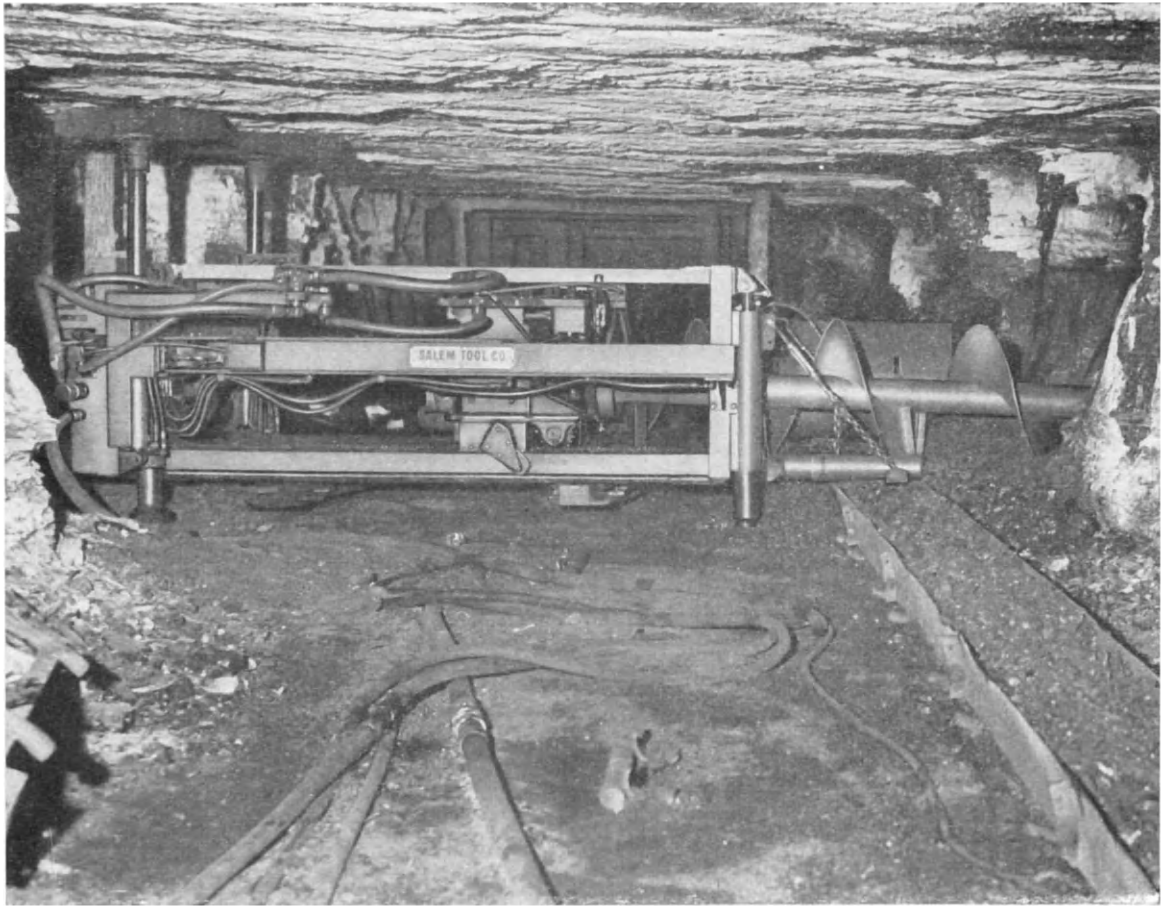


FIG. 13. Coal auger operating underground. (Salem Tool Company.)

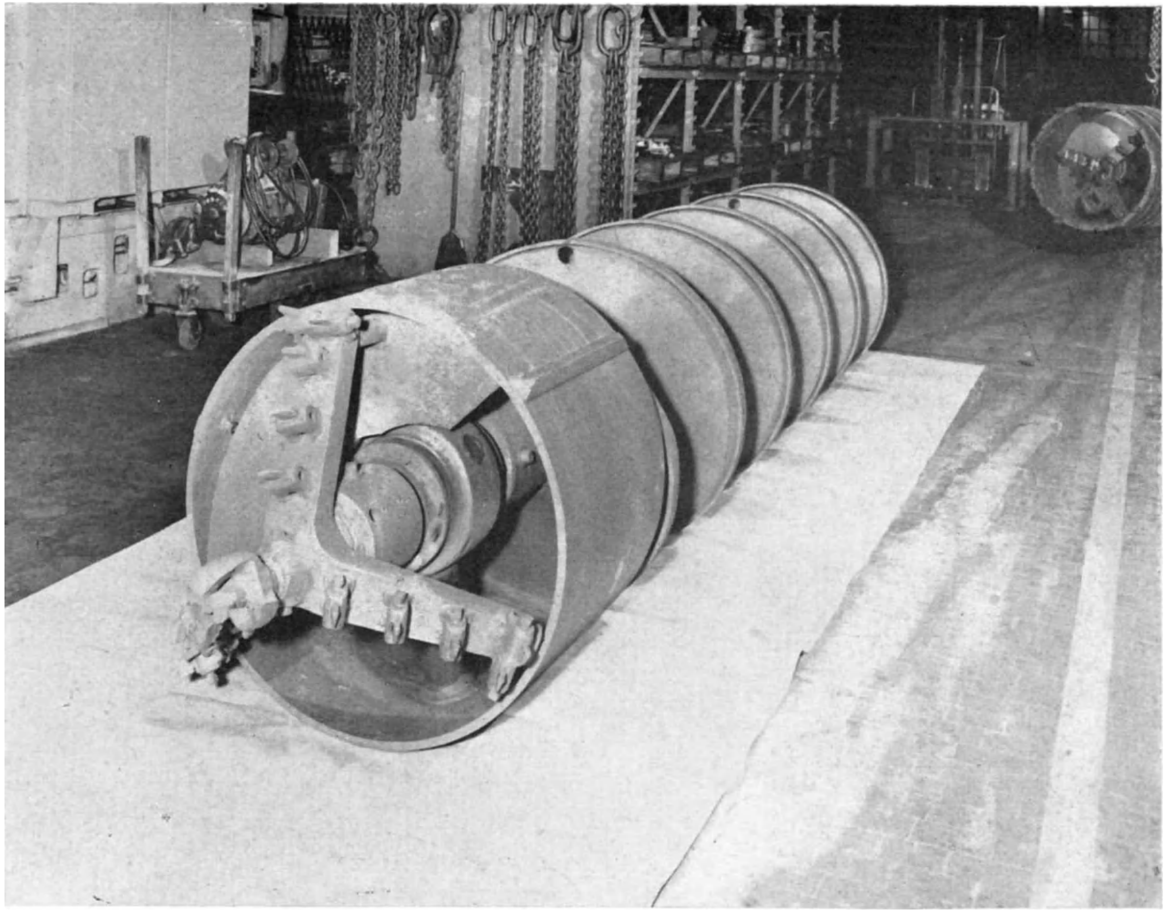


FIG. 13 (a). Non-rotating-type cutting barrel for a coal auger. (Salem Tool Company.)

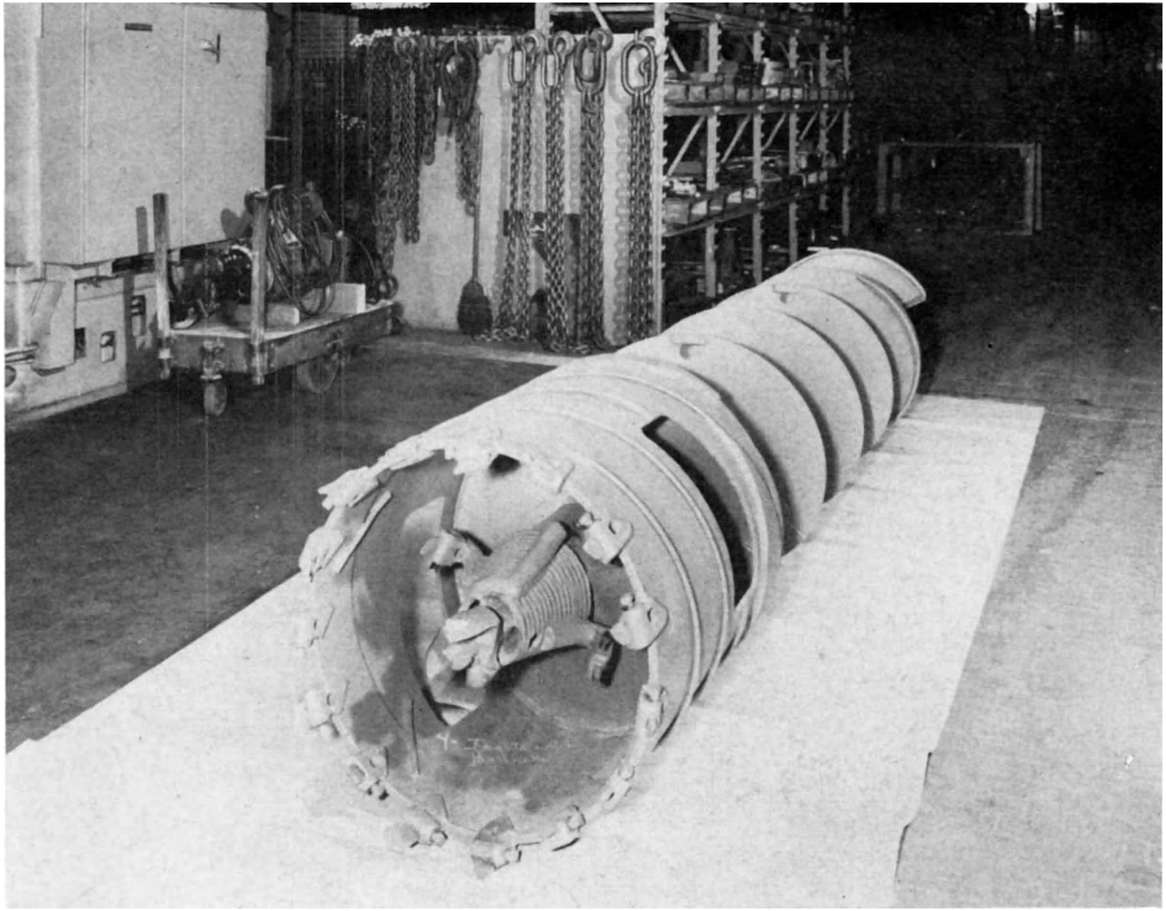


FIG. 13 (b). Rotating-type cutting barrel for a coal auger. (Salem Tool Company.)

A Joy AD2 auger at Merry Lees Colliery in the East Midlands Division was successfully operated. In this case pillars of coal were formed by driving headings 15 ft wide and 60 ft apart to enable the auger to drill from one roadway into another parallel heading. With holes on 64-in. centers, this gave a 50 per cent extraction.

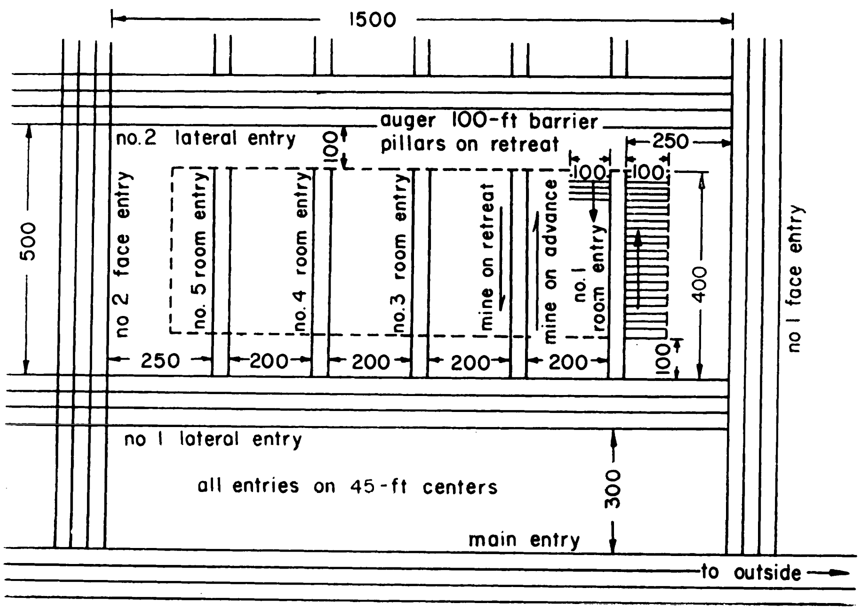


FIG. 14. Mine layout for augering.[1]

The auger cannot be used for a full production face but is useful in the following cases:

(1) Extraction of good quality coal from thin seams.

(2) The partial extraction of pillars in mines which are to be abandoned.

(3) The partial extraction of seams where subsidence cannot be tolerated.

Ventilation difficulties have retarded the use of augers in British mines.

The Collins Miner[6]

The Collins miner, which was developed in Great Britain, is essentially a remote controlled "down the hole" boring machine which is thrust into the exposed rib-side of a seam in which it bores a slot-shaped hole the thickness of the seam and about 300 ft long.

This operation is controlled entirely from the entrance of the hole since the cutting head embodying the boring unit carries with it an automatically extending belt conveyor, ventilation ducting, and cables.

The method of working is to divide the area into rectangular panels by means of roadways which are partially dinted so that the exposed coal ribs are well above roadway floor level. The Collins miner is then used to extract coal from a series of parallel holes at right angles to the roadway.

The equipment comprising the miner is largely contained in a train of nine rail-mounted bogies sited in the central roadway from which the boreholes are to be driven.

The cutting unit which enters the borehole consists essentially of a three-headed auger feeding a short conveyor passing through the center of the machine and loading onto the belt extending from the back of the cutting unit to the roadway.

The cutting unit is thrust into the coal seam by means of sectional push rods, and the main functions of the launching platform are to provide thrust to the push rods and to turn the conveyor belt through a right angle.

The control of the miner is entirely in the hands of one man who sits in the control cab and receives the information necessary for control from a series of indicating and recording instruments.

METHODS AND EQUIPMENT USED IN DEVELOPMENT WORK

A number of mines were studied by personnel of the U.S. Bureau of Mines to determine what methods and equipment were used in underground development (*U.S. Bureau Mines I.C.* 7813, 1957), and how continuous mining machines were employed (*U.S. Bureau Mines I.C.* 7696, 1954).

Following are the descriptions of methods and equipment used at some typical mines as well as diagrams showing the development patterns used.

Boring-type Mining Machine (Mine 7; Figures 15 and 16)

This mine is operated in the Sewell coal bed, which averages 44 in. thick in this area. The bed is flat lying but some rolls and faults are encountered in mining. The overburden averages 436 ft in thickness and consists mainly of shales and sandstones.

One of the continuous mining units on entry development was studied in detail.[2] Main entries 27 ft wide were driven in sets of nine on 60-ft centers, with cross cuts 18 ft wide on 75-ft centers (Fig. 15). These entries were developed by a unit consisting of nine men, one boring-type continuous mining machine, one mobile loading machine, one piggyback conveyor, two chain conveyors, and a 30-in. belt conveyor. Power to operate the equipment was supplied at 440 V a.c.

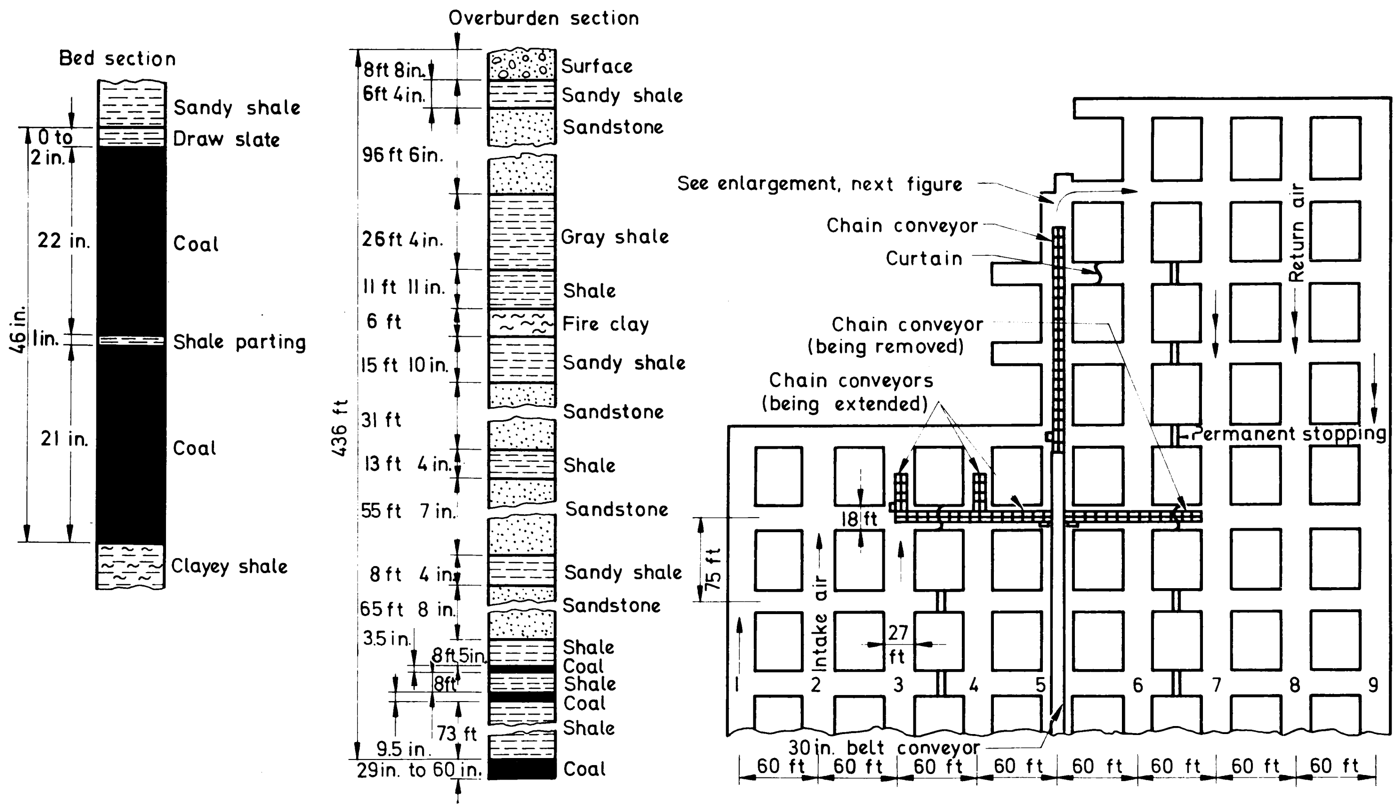


FIG. 15. Geologic section and method of development — boring-type continuous miner. Mobile loading onto chain conveyors at Mine 7.[2]

Operating Procedures

Entries were advanced with a continuous mining machine by making three successive cuts, each 9 ft wide and 42 ft long (Fig. 16). Cross cuts were advanced in a similar manner with two 9-ft cuts.

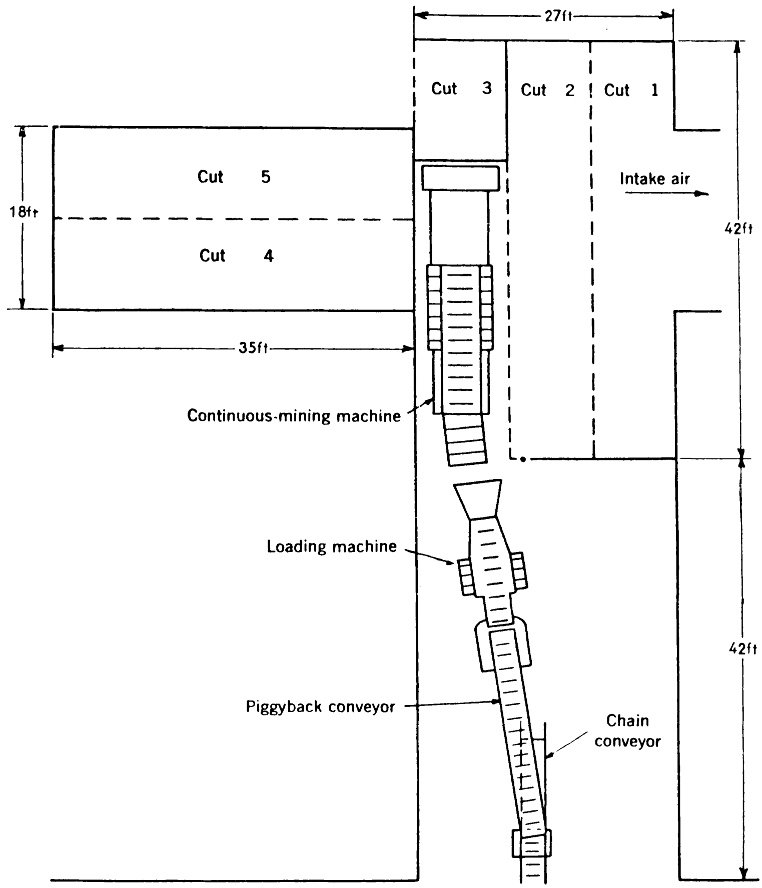


FIG. 16. Sequence of cuts at Mine 7.[2]

Loading and Haulage

Coal mined with the continuous mining machine was deposited on the floor. The coal was reloaded by the mobile-loading machine and transported by the piggy-back conveyor, two chain conveyors, the 30-in. belt conveyor, and the 36-in. main-haulage belt conveyor, to the preparation plant on the surface.

Roof Support

The roof was supported with wood props set on 5-ft centers.

Crew Required

A unit crew consisted of the following men:

Section foreman	1
Continuous mining machine operator	1
Continuous mining machine operator's helper	1
Loading machine operator	1
Loading machine operator's helper	1
Supply man	1
Mechanic	1
Total	7

A unit crew produced an average of 299 tons of raw coal per shift, or 42.8 tons per man-shift. The average advance in each entry of a development group was 6.1 ft/shift, or a total of 54.5 ft for the group.

Boring-type Mining Machine (Mine 16; Fig. 17)

This mine is operated in the Lower Kittanning Coal Bed which is from 48 to 59 in. thick in this area (42–53 in. mined). The overburden averages about 100 ft thick.

Operating Procedures

Panel entries were developed with a boring-type continuous mining machine as shown in Fig. 17. Entries were driven 18 ft wide by making alternate cuts $9\frac{1}{2}$ and $8\frac{1}{2}$ ft wide. Cross cuts were turned 45° from the center entry and driven $9\frac{1}{2}$ ft wide on 75-ft centers.

Loading and Haulage

Two shuttle cars transported the coal from the continuous mining machine to an elevating conveyor, which discharged onto a 26-in. belt. The coal was conveyed by the 26-in. belt to a 30-in. main belt, which transported the coal to the tipple.

Roof Support

All entries were roof bolted on 5-ft centers with $\frac{5}{8}$-in. bolts, 36 in. long set against $6 \times 6 \times \frac{3}{8}$ in. steel plates. The average advance of the mining machine in an 18-ft entry was 40 ft per shift.

Equipment Required

A unit crew was equipped with one boring-type continuous mining machine, two shuttle cars, one elevating conveyor, one 26-in. belt, one portable roof-bolting machine.

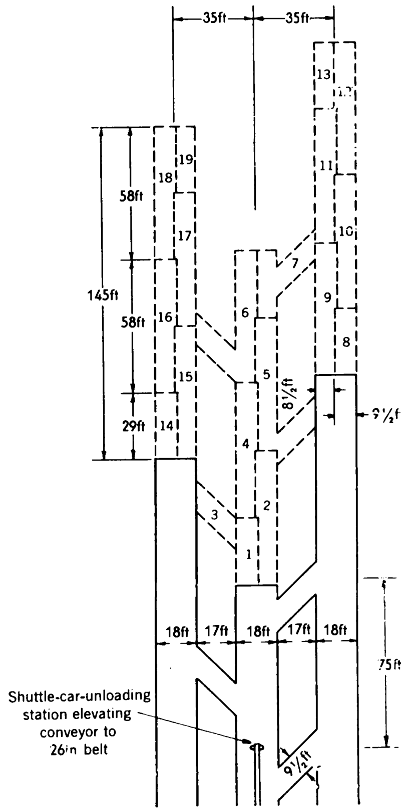


FIG. 17. Main entry development with boring-type continuous miner – shuttle car haulage to conveyor (Mine 16).[3]

Crew Required

The crew consisted of eight men. The average production of raw coal per unit crew per shift on development was 101 tons giving an average production per man on a unit crew of 12.6 tons.

Boring-type Mining Machine (Mine 18; Figures 18 and 19)

This mine is operated in the Sewickley Bed in West Virginia which has a thickness averaging 72 in. in this area. The average thickness of overburden is 320 ft. The mine uses a block mining system with pillars being extracted.

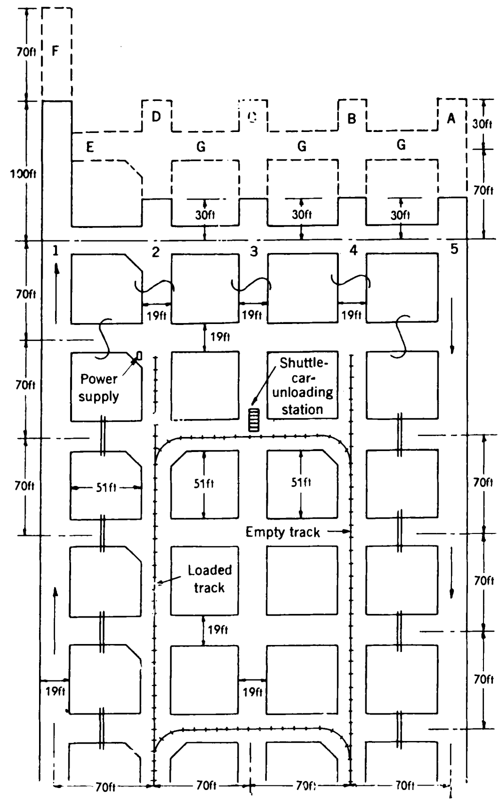


FIG. 18. Butt-entry-panel development with boring-type continuous miner—mobile loading into shuttle cars, Mine 18.(3)

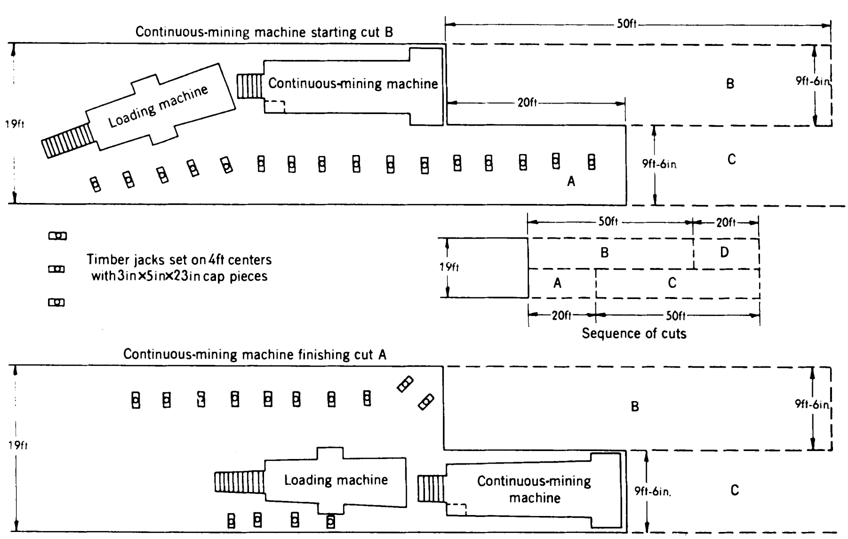


FIG. 19. Timbering plan and sequence of cuts at Mine 18.(3)

Operating Procedures

A boring-type mining machine was used for all development work. The plan for development is shown in Fig. 18. Entries and cross cuts were driven 19 ft wide by making alternate cuts $9\frac{1}{2}$ ft wide, as shown by the sequence of cuts in Fig. 19. The sequence of advance of each heading is indicated by letters in Fig. 18.

Loading and Haulage

The continuous mining machine deposited the coal on the floor which provided adequate surge capacity, and allowed the continuous mining machine to operate with a minimum of shuttle car haulage delays. The mobile-loading machine reloaded the coal into 6-ton capacity shuttle cars which transported it to the unloading station where it was discharged into 5-ton steel mine cars.

Roof Support

Steel timber jacks, set on 4-ft centers, with 3- by 5- by 23-in. wood cap pieces, were used for safety posts to protect men and equipment. These timber jacks were reset near the opposite rib for each alternate cut (see Fig. 19). Permanent timbering generally was not required for roof support.

Equipment Required

A unit crew was equipped with one continuous mining machine, one mobile-loading machine, and three cable-reel shuttle cars.

Crew Required

A continuous mining crew consisted of eight men as follows:

Section foreman	1
Continuous mining machine operator	1
Loading-machine operator	1
Shuttle-car operators	3
Timbermen	2
Total	8

The average production per unit per shift was 565 tons. This gave an average production per man-shift of 70.6 tons for each man on a development unit crew.

Full-dimension Mining

A full-dimension system is a haulage system which provides an uninterrupted flow of coal from a loader or continuous miner at the face to the main line transportation system. The equipment required for this system consists of a series of interconnected conveyors which are mobile and articulated and which will retract or extend a sufficient distance for the development of a five-entry system.

Such a system with its components is shown in Fig. 20.

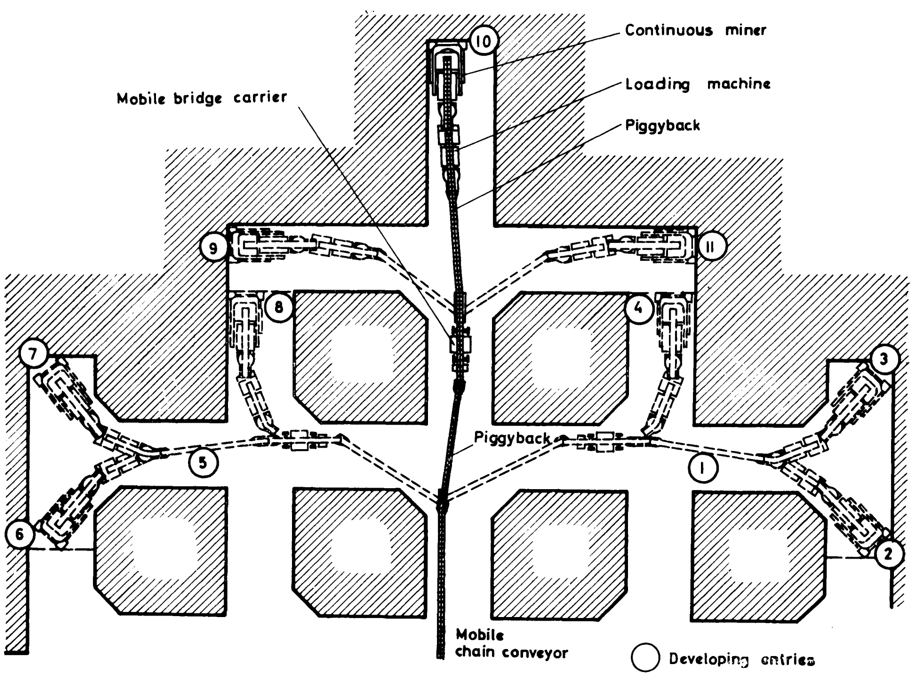


FIG. 20. Method of developing a five-entry panel.
Numbers indicate the sequence of mining (The Long Company.)

METHODS AND EQUIPMENT USED IN ROOM MINING

Following are the descriptions of room mining methods and equipment used in conjunction with continuous mining machines at some typical mines studied by personnel of the U.S. Bureau of Mines and reported in *I.C.* 7696.

Ripper-type Mining Machine (Mine 20; Fig. 22)

This mine is operated in the Pittsburgh No. 8 Bed in West Virginia which has an average thickness of 62 in. in this area (58 in. mined). The average thickness of overburden is 600 ft. Mining was by room-and-pillar system with pillars not extracted. Total recovery of coal was about 60 per cent.

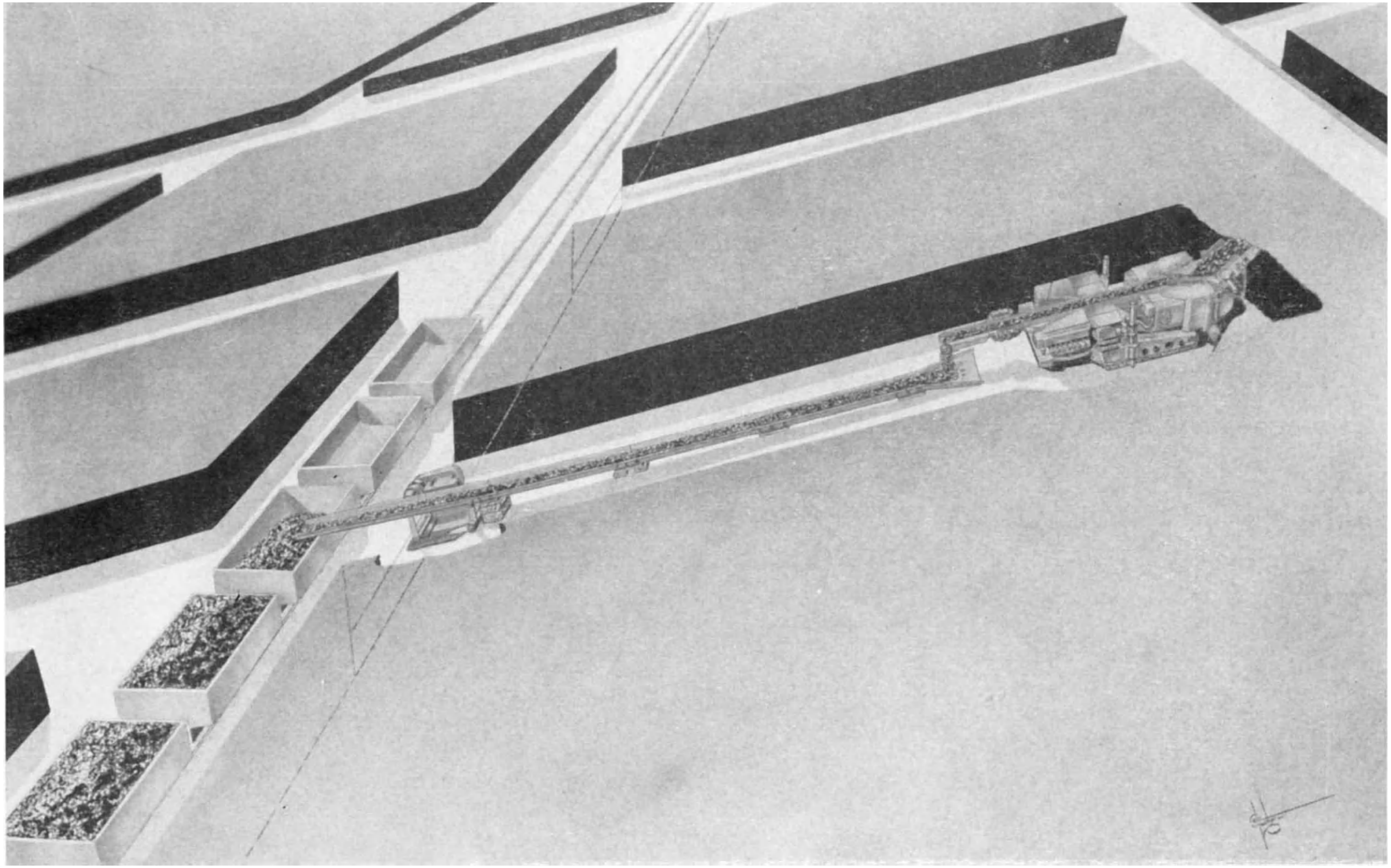


FIG. 21. Continuous mining system. (Joy Mfg. Company.)

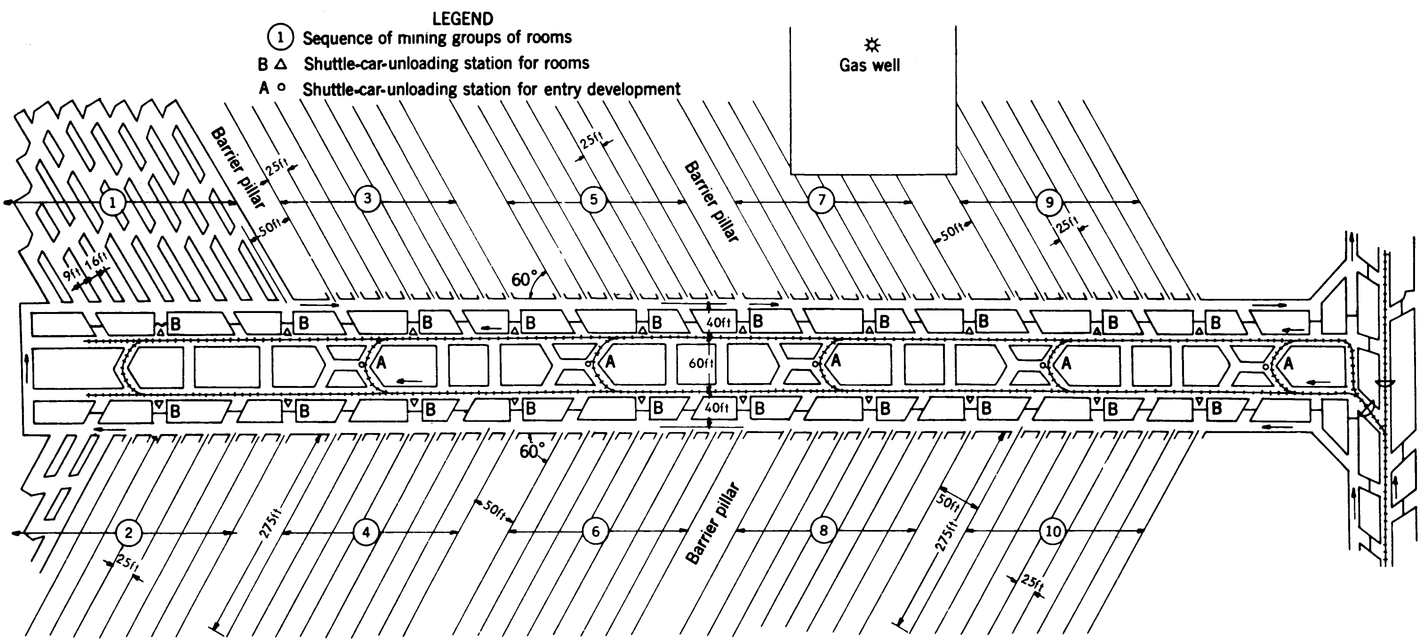


FIG. 22. Room-mining with a ripper-type continuous miner – shuttle car haulage to mine cars, Mine 20.[3]

Operating Procedure

Rooms were turned on 25-ft centers at 60° from the butt entries and driven 16 ft wide and 275 ft deep. Cross cuts between rooms were 16 ft wide and were turned on 80-ft centers at an angle of 45° (see Fig. 22).

Loading and Haulage

One of the two shuttle cars was used behind the continuous mining machine as a surge car, while the other transported coal from the surge car and discharged it into mine cars of 1.9-ton capacity at unloading stations. The unloading stations consisted of wooden ramps in cross cuts which had been top-brushed and roof bolted.

Roof Support

A 50-ft pillar of coal was left between each two adjacent groups of rooms. In mining, 4 in. of top coal were left to support the roof. Four-inch steel H-beams, 13 ft long and spaced on 3-ft centers, were set on wood posts to support the roof. These H-beams and posts were recovered when a room was finished.

Equipment Required

Each unit crew was equipped with one continuous-mining machine and two cable-reel shuttle cars.

Crew Required

A continuous-mining crew consisted of $7\frac{1}{6}$ men as follows:

Section foreman	$\frac{2}{3}$
Continuous mining machine operator	1
Shuttle-car operators	2
Timbermen	2
Electrician-mechanics	$1\frac{1}{2}$
Total	$7\frac{1}{6}$

The average daily production of raw coal per unit per shift for the continuous mining units was 141.7 tons. This gave an average production of 19.8 tons of raw coal per man-shift for each man in a unit.

Ripper-type Mining Machine (Mine 10; Fig. 23)

This mine is operated in the Lower Kittanning Bed in Pennsylvania. This bed is 48 in. thick in the locality of this mine and is overlain by about 500 ft of overburden. A room-and-pillar system was used with pillars extracted to give an overall recovery of about 90 per cent.

Operating Procedure

Rooms 16 ft wide were turned on 56-ft centers. As shown in Fig. 23 the two inby rooms were mined simultaneously by making cuts 1–6 inclusive. The pillars between these rooms were mined by extraction as shown by cuts 7–15 and Room C was mined by making cuts 16, 17 and 18.

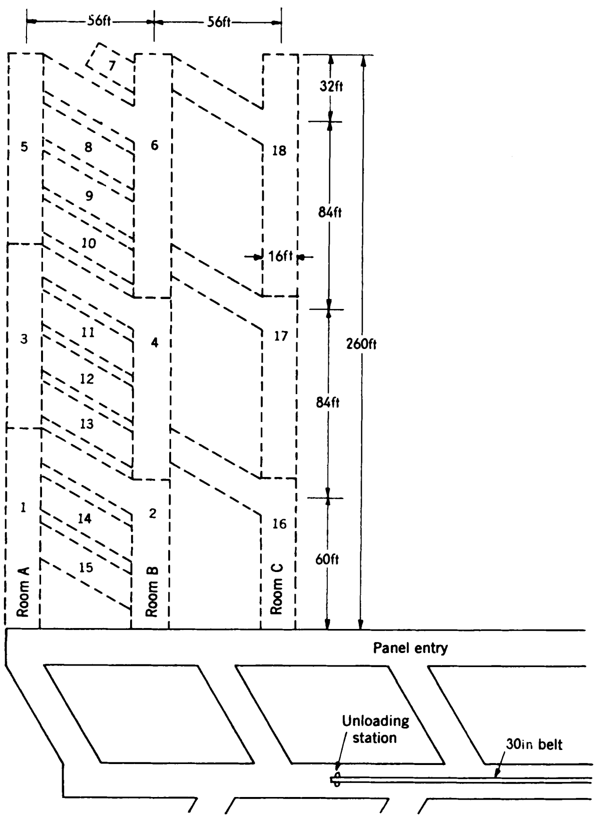


FIG. 23. Room-mining and pillar extraction with a ripper-type continuous miner – shuttle car haulage to belt, Mine 10.(3)

Loading and Haulage

One of the two shuttle cars was used behind the continuous mining machine as a surge car while the other hauled the coal to the unloading point.

Roof Support

A single row of wood props on 4-ft centers is set in rooms 4 ft from the rib, and in pillar extraction a single row of wood props is set along the rib.

Equipment Required

Each unit crew was equipped with one continuous mining machine, two cable-reel shuttle cars, and a 30-in. belt.

Crew Required

A continuous-mining crew consisted of six men as follows:

Section foreman	$\frac{1}{2}$
Continuous mining machine operator	1
Shuttle car operators	2
Timberman	1
Car trimmer	$\frac{1}{2}$
Electrician-mechanic	1
Total	6

The average daily production of raw coal per unit shift was 133.3 tons. This gave an average production of 22.2 tons of raw coal per shift for each man in a unit.

Ripper-type Mining Machine (Mine 1; Fig. 24)

This mine is operated in the American Coal Bed in Alabama. This bed is 54 in. thick and is overlain by about 216 ft of overburden. A room-and-pillar system was used with pillars not extracted. Overall coal recovery was 65 per cent.

Operating Procedure

The plan for room development is shown in Fig. 24. Rooms *A* and *B* were driven making cuts 1–5 inclusive. Succeeding rooms were driven by repeating the sequence of cuts 6–17.

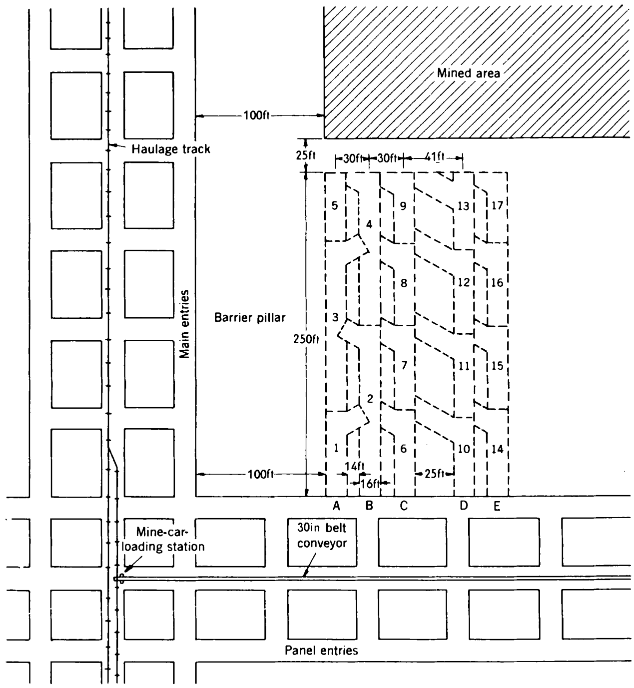


FIG. 24. Room-mining with ripper-type continuous miner—mobile loading into shuttle cars, Mine 1.[3]

Loading and Haulage

The continuous-mining machine discharged the coal to the floor from where it was loaded by a mobile-loading machine to a $3\frac{1}{2}$-ton capacity shuttle car, used as a surge car. These operations permitted the continuous-mining machine to be operated with a minimum of delay for alignment with or waiting on shuttle cars.

Roof Support

Wood crossbars, 14 ft long by 8 in. wide by 3 in. thick, set on steel screw jacks were used to support the roof. The crossbars were set on 4-ft centers as the continuous-mining machine advanced and were recovered (along with the steel screw jacks) when the room was completed.

Equipment Required

Each unit crew was equipped with a continuous mining machine, one mobile-loading machine, two shuttle cars, and one 26-in. or 30-in. panel belt.

Crew Required and Productivity

Travel time at this mine was 1 hr; no time lost for lunch (crew lunch period staggered); leaving 7 hr face time per shift. The average delay caused by mechanical failure was 39 min, 9.3 per cent of face time. Other delays, as maneuvering the continuous mining machine, timbering, power failure, bit change, greasing, and waiting for empty cars, amounted to 1 hr 24 min, 20 per cent of the face time. Actual productive time was 4 hr 57 min, 70.7 per cent of face time.

A crew of eight men in the typical continuous mining unit produced an average of 369 tons of raw coal per shift. This was an average of 46.1 tons of raw coal per man-shift per man in the production unit.

METHODS USED FOR PILLAR EXTRACTION IN CONTINUOUS (NON-CYCLIC) MINING

There are two general methods used for pillar extraction. These are (1) the open-end method; and (2) the "pocket-and-fender" or "split-and-fender" method.

(1) The open-end method involves taking lifts of coal from the side of the pillar adjacent to the mined-out area. In some cases no enders are left between the mining machine and the gob or goaf while in other cases thin pillars known as "fenders" are left to support the roof and protect the mining machine and crew from the caving roof-rock in the goaf.

(2) The "pocket-and-fender" or "split-and-fender" method involves splitting the pillar and then taking slices or lifts from the interior of the pillar.

Generally the open-end method, where it can be used, gives a higher percentage of coal recovery than the split-and-fender method because less coal is left unrecovered in the fenders or stumps. However, there are many mines where the open-end method cannot be applied because of adverse roof conditions.

Where pillars or stumps are left it is frequently necessary to blast them to secure proper caving of the roof after mining of a pillar is completed.

Auxiliary Support

Auxiliary supports used during the extraction of the last stump of a pillar or the final stump of a lift may include cribs, additional props, 20-ton hydraulic props, or 80-ton yielding-type steel props. Ropes are attached to the 20-ton hydraulic props and to the 80-ton props so that they may be tripped and pulled to a safe area when pillaring is completed.

Examples of Pillaring Methods

The figures on the following pages illustrate methods used in pillaring. Figures 26–33 inclusive are taken from *U.S. Bureau of Mines R.I.* 5631.

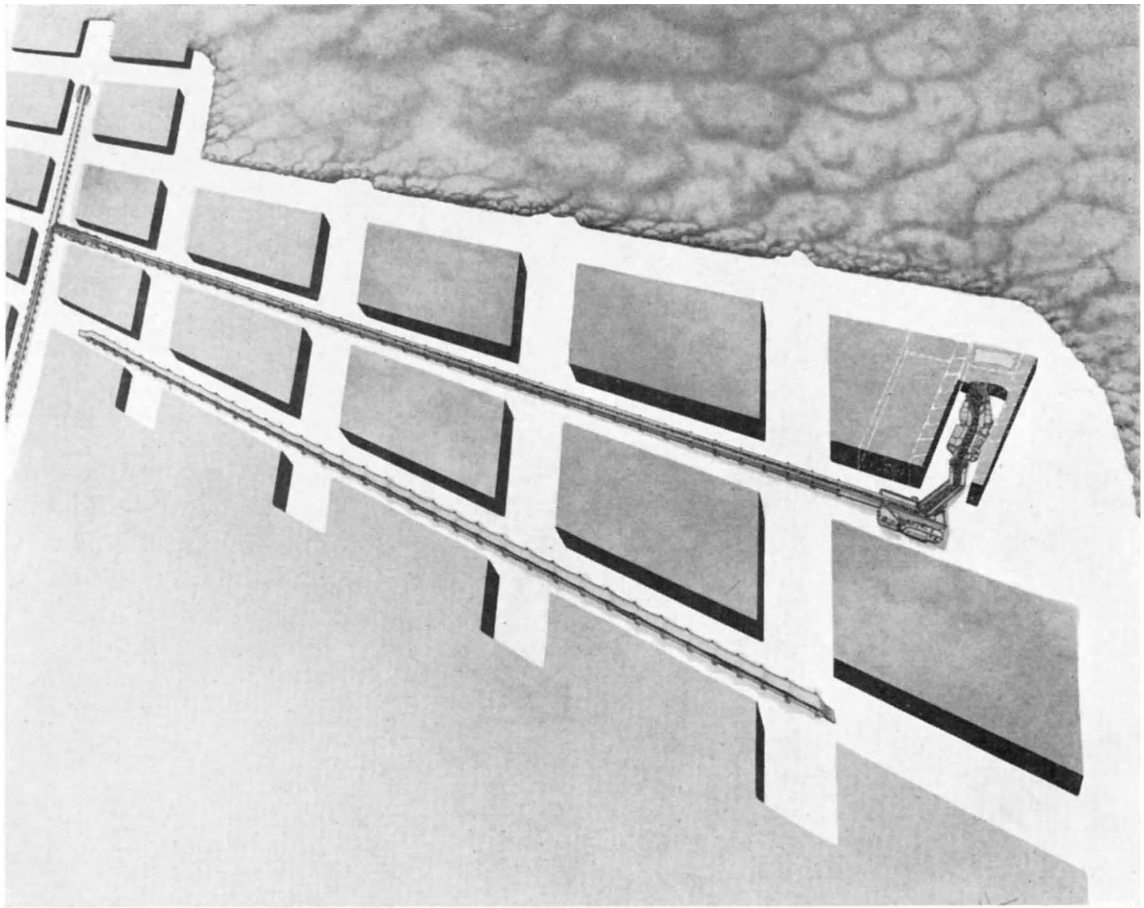


FIG. 25. Pillaring with a ripper-type continuous miner. (Joy Mfg. Company.)

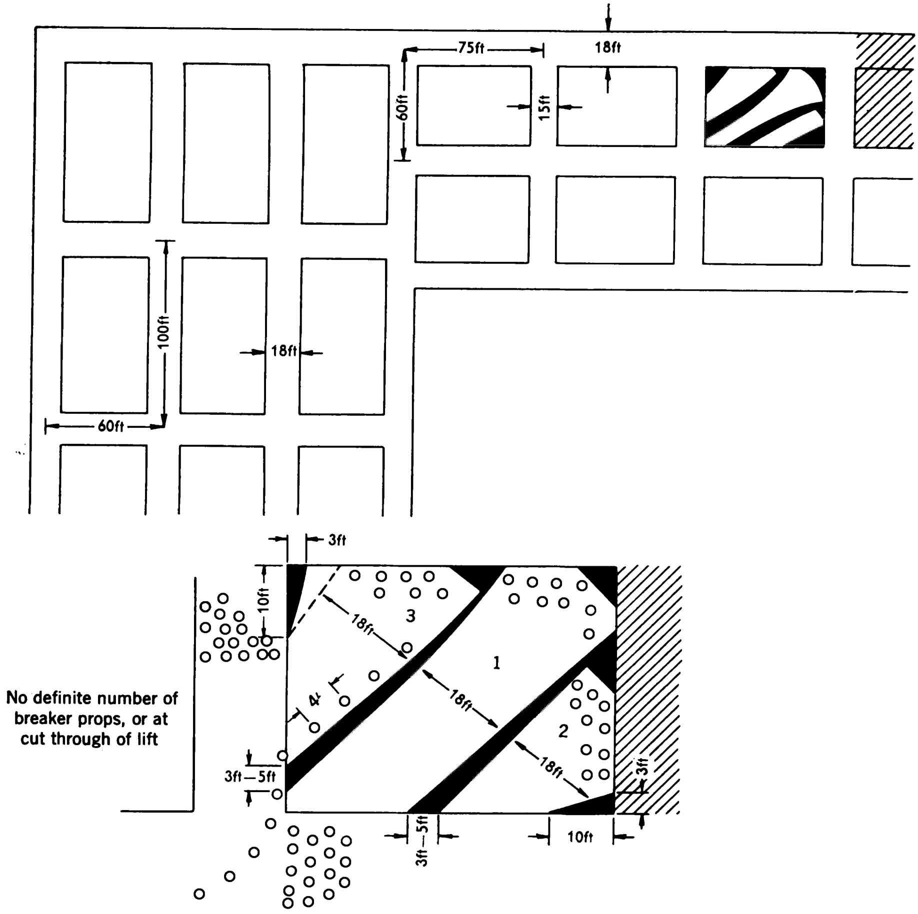


FIG. 26. Open-end pillaring with a boring-type continuous miner in Lower Kittanning coal bed using post timbering.[4]

General Requirements for Pillaring Operations

(*U.S. Bureau of Mines R.I.* 5631)

(1) After extraction of a pillar is begun speed in completing the operation is essential. If necessary it may be wise to finish a shift in another pillar, rather than leaving a small block stand to be recovered later.

(2) Complete extraction of pillars is preferable to hogging or splitting as strong remnants tend to throw weight of the roof forward onto remaining pillars and cause difficulties with roof control. It may be necessary to blast remnants to remove them as supports.

(3) An unobstructed runway is essential to allow quick removal of men and equipment if roof falls threaten.

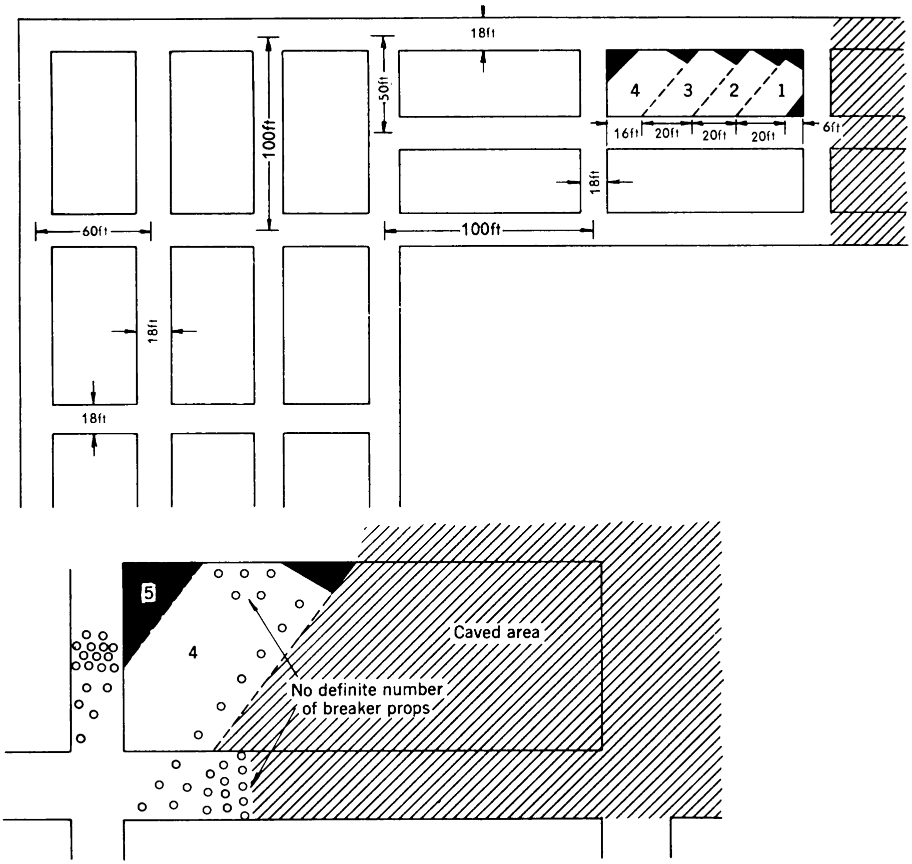


FIG. 27. Open-end pillaring with a ripper-type continuous miner in Lower Kittanning coal bed using post timbering.[4]

(4) Where a thin-fender or open-end method is employed long lifts are hazardous; it is preferable to take slices from alternate sides of a pillar rather than from the same side.

(5) Hydraulic and yielding steel props appear to be safer than breaker props and cribs since they may be tripped and recovered from a safe area outby the pillar split.

(6) In roof-bolted areas where pillars are being extracted with ripper-type continuous miners, occasional timbers set at outby intersections and in the intervening approaches to an active pillar, would warn of strata separation above the bolts, or of other roof movement not usually indicated by the bolts. Abnormal or fragile roof outby pillar places should be supported in accordance with the need.

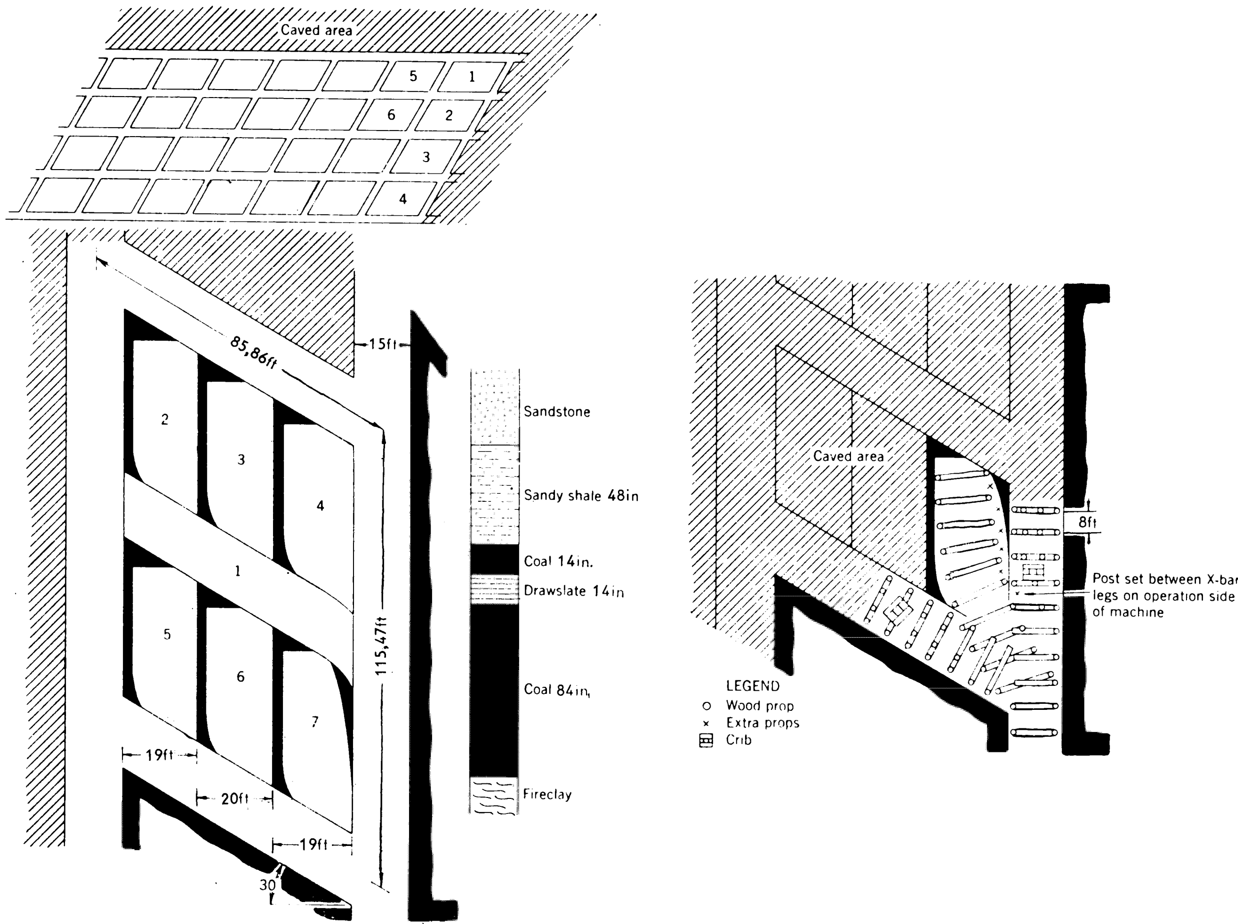


FIG. 28. Open-end pillaring with a ripper-type mining machine in Pittsburgh coal bed using crossbars on posts for roof support.(4)

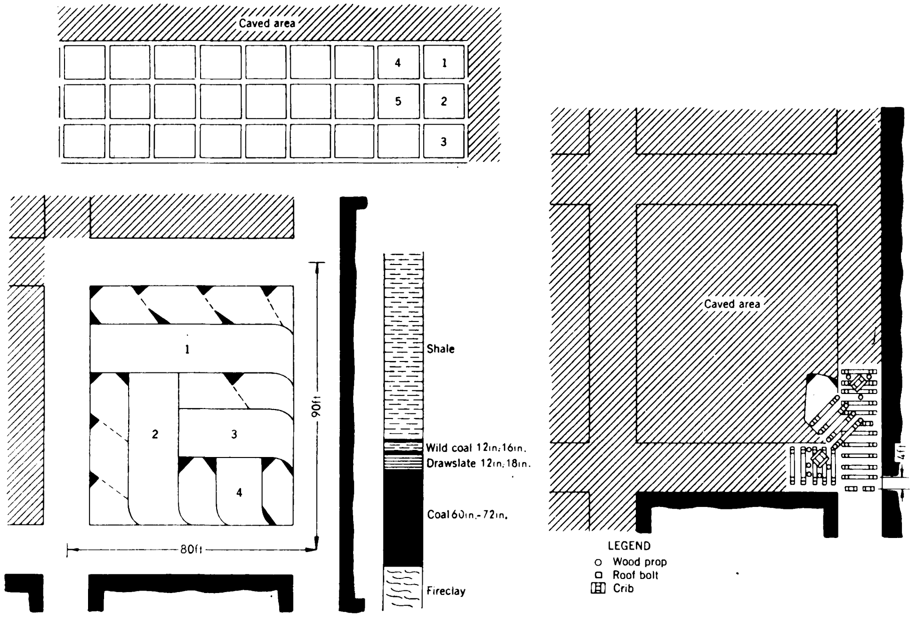


FIG. 29. Split-and-fender pillaring with ripper-type continuous miner in Pittsburgh coal bed using bolts for roof support.[4]

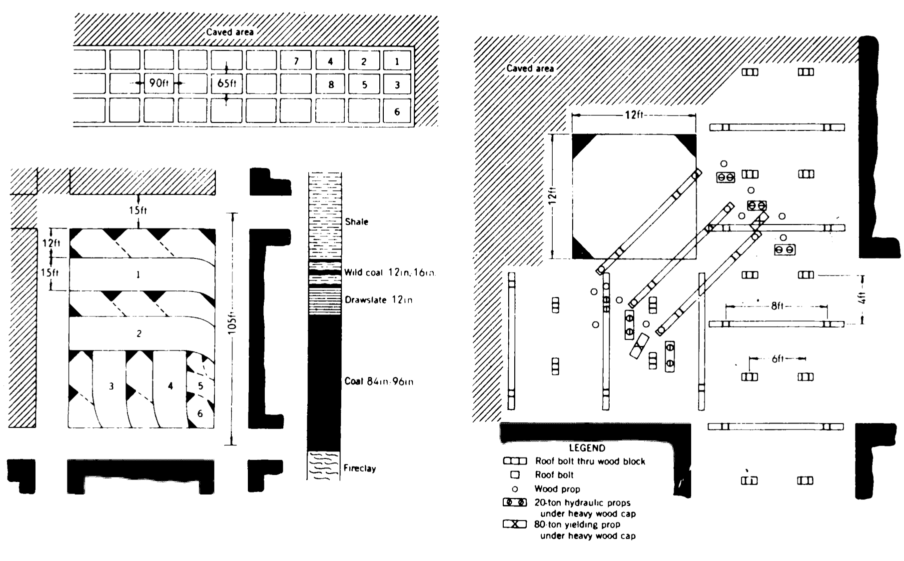


FIG. 30. Split-and-fender pillaring with ripper-type continuous miner in Pittsburgh coal bed using bolts for roof support, hydraulic props for break props, and yielding-type steel props for cribs.[4]

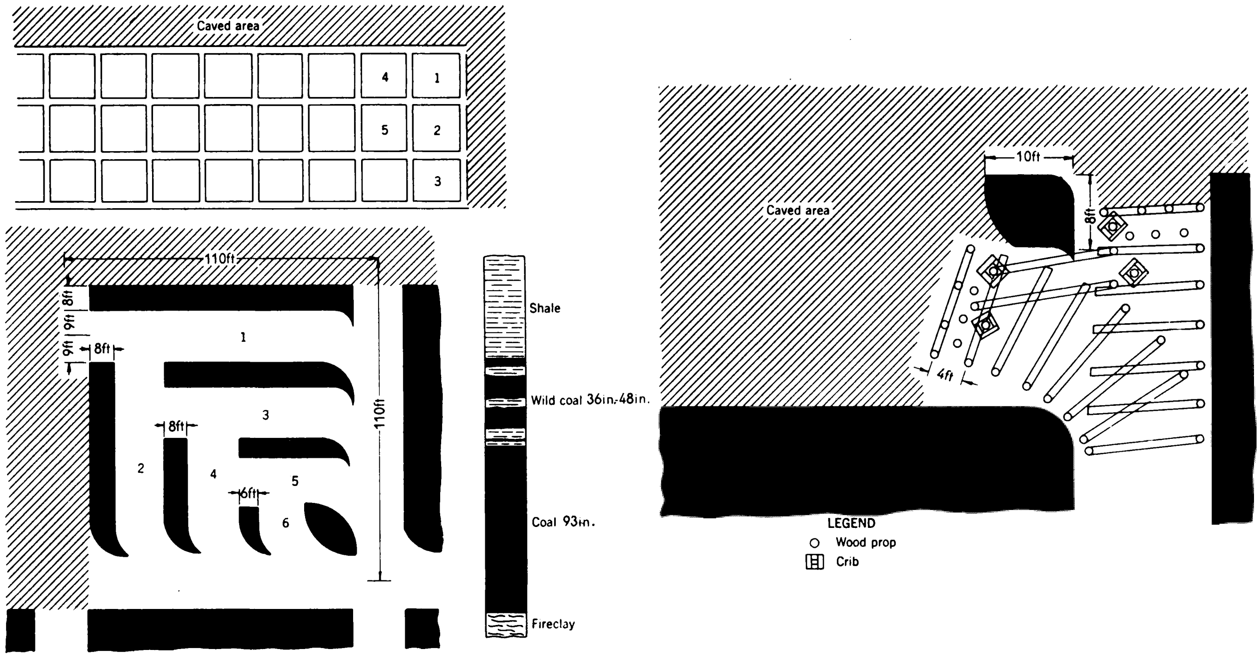


FIG. 31. Split-and-fender pillaring with ripper-type continuous miners in the Pittsburgh coal bed using crossbar and post timbering and double cribs during final stump removal.[4]

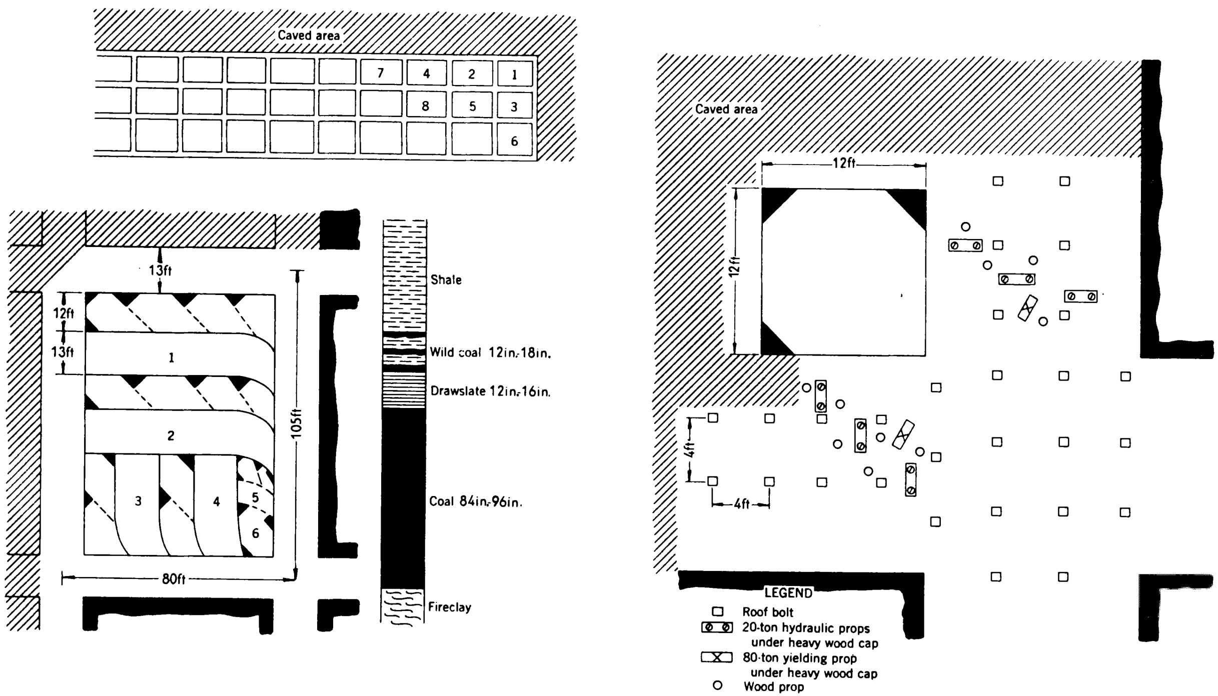


FIG. 32. Split-and-fender pillaring with boring-type continuous miner in the Pittsburgh coal bed using bolts for roof support, hydraulic props for break props, and yielding-type steel props for cribs.[4]

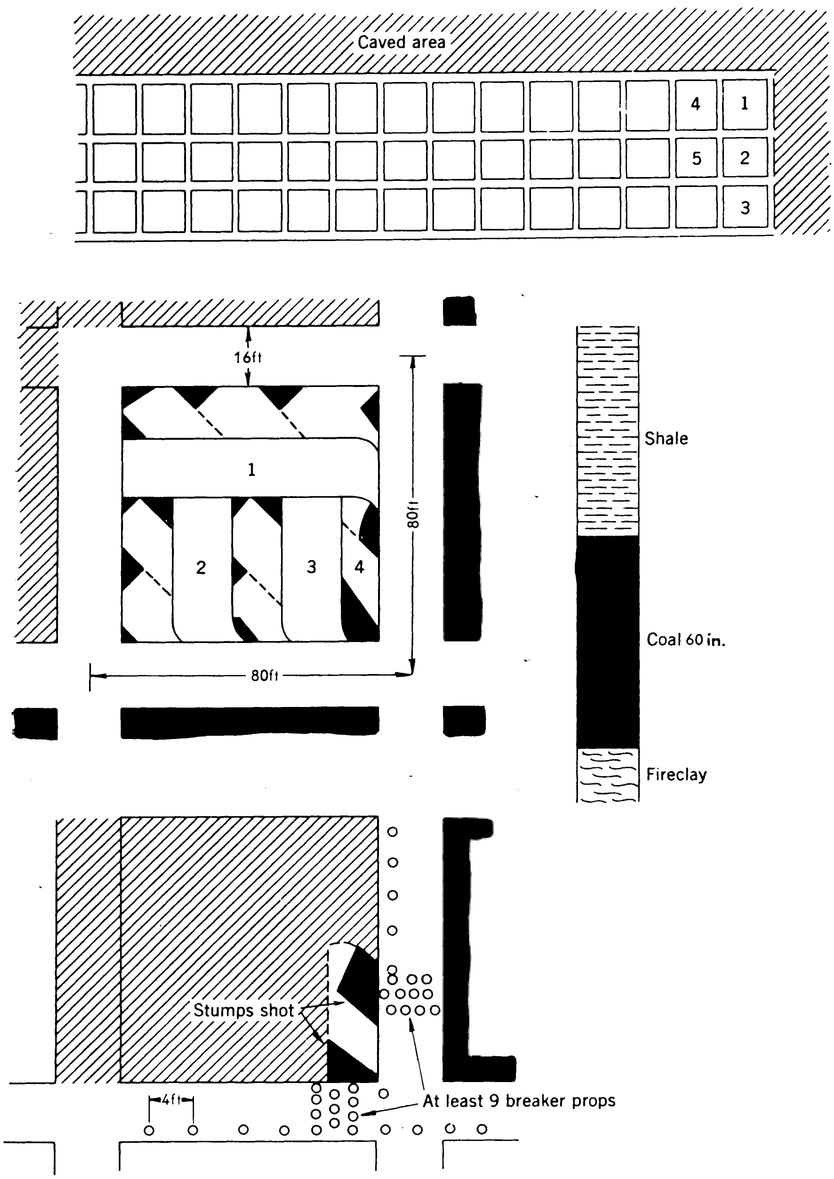


FIG. 33. Split-and-fender pillaring with ripper-type continuous miner in Sewickley coal bed using post timbering and blasting final small stumps.(4)

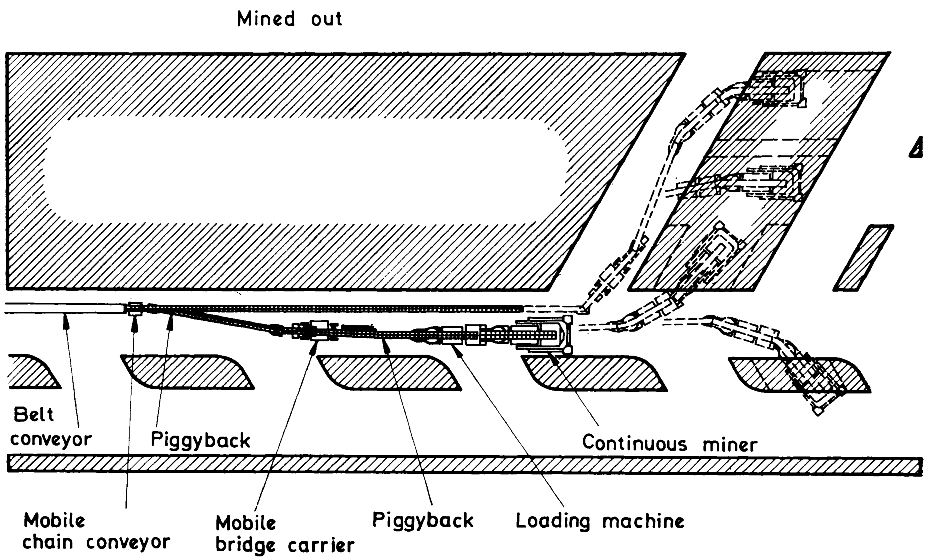


FIG. 34. Method of mining room pillars. (The Long Company.)

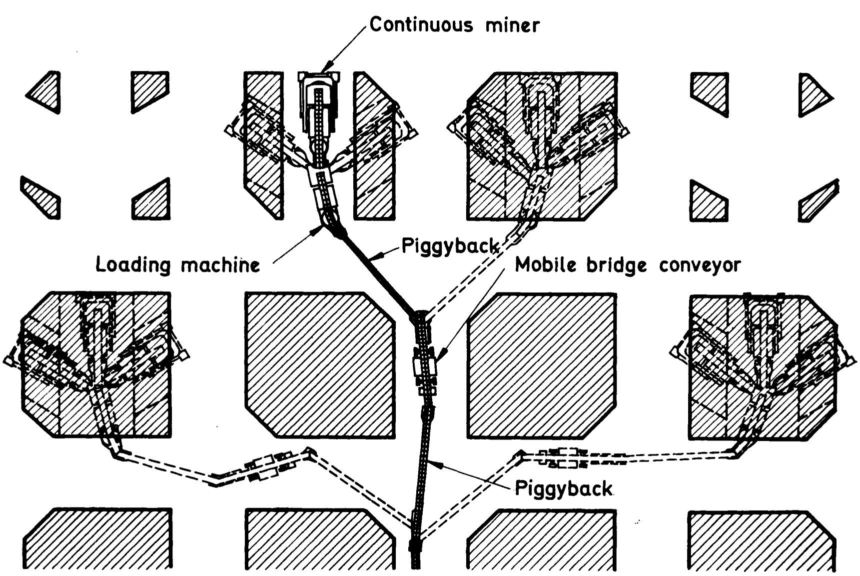


FIG. 35. Method of mining entry pillars – "full dimension" mining system. (The Long Company.)

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CHAPTER 4

LONG-WALL MINING—CYCLIC OPERATIONS

A full mining cycle at a long-wall face includes the following operations: (1) undercutting the coal; (2) drilling shot holes, loading them with explosives (or with mechanical pressure breaking devices), and shot firing to break down the coal; (3) loading (filling) the broken coal onto the face conveyor; (4) moving the conveyor over; (5) moving forward the line of chocks or props which determine the caving line of the roof, moving forward the back props, ripping the roadways, and building packs.

When coal is exceptionally soft or friable and/or when roof pressure can be brought to bear on the face coal to fracture the seam then coal may be gotten with hand-held pneumatic picks without the necessity for undercutting, drilling, or shot-firing. A large proportion of German coal production is produced in this manner and is hand-loaded onto face conveyors.

In some cases coal which is too hard for efficient getting with pneumatic picks may be gotten by undercutting the face and allowing time for the undercut block of coal to settle and fracture, after which it is filled onto the face conveyor.

MACHINES FOR CYCLIC LONG-WALL MINING

The coal cutter was the earliest form of mechanization applied to face operations in coal mining. The modern machines have evolved from early day machines which employed a horizontal rotating disk as the cutting element. The basic cutting mechanism in all modern cutters is a bar or "jib" with a toothed chain running around its perimeter. About 90 per cent of British coal output is at present machine cut.

LONG-WALL COAL CUTTERS[1]

There are a number of types of chain coal cutters in use on long-wall faces. The design varies from one to another but the basic principle is similar in all cases. The typical long-wall coal cutter consists of two parts, the cutter jib and the main body. The main body consists of three parts as follows: (1) The cutting unit, (2) the hauling unit, (3) the driving unit. These parts are located by spigot-faced joints and are held rigidly together by high-tensile steel bolts and studs.

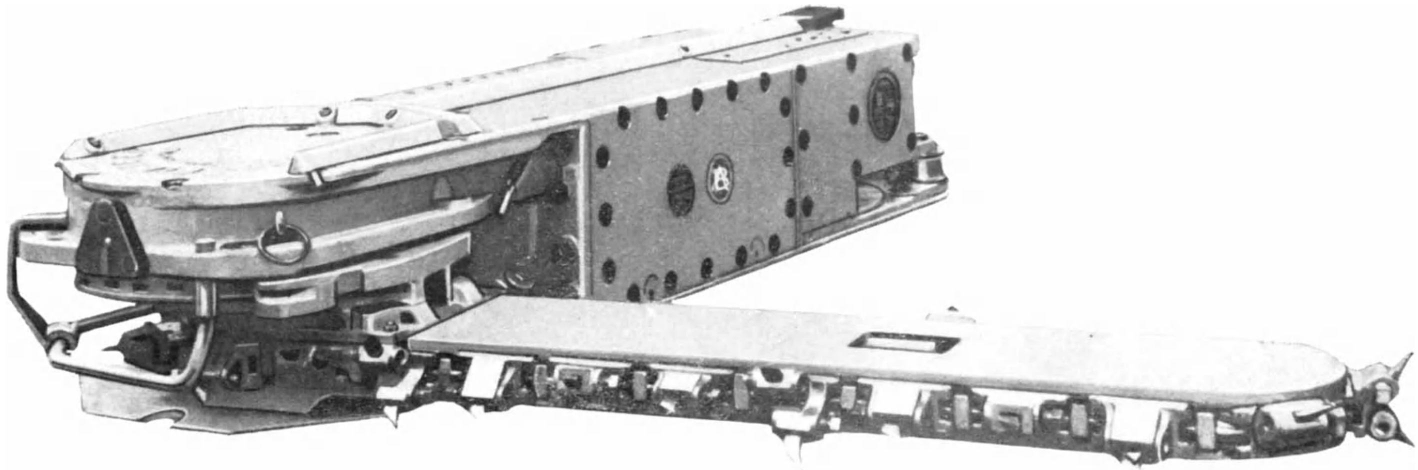

FIG. 1. Long-wall coal cutter. (Anderson, Boyes & Co., Ltd.)

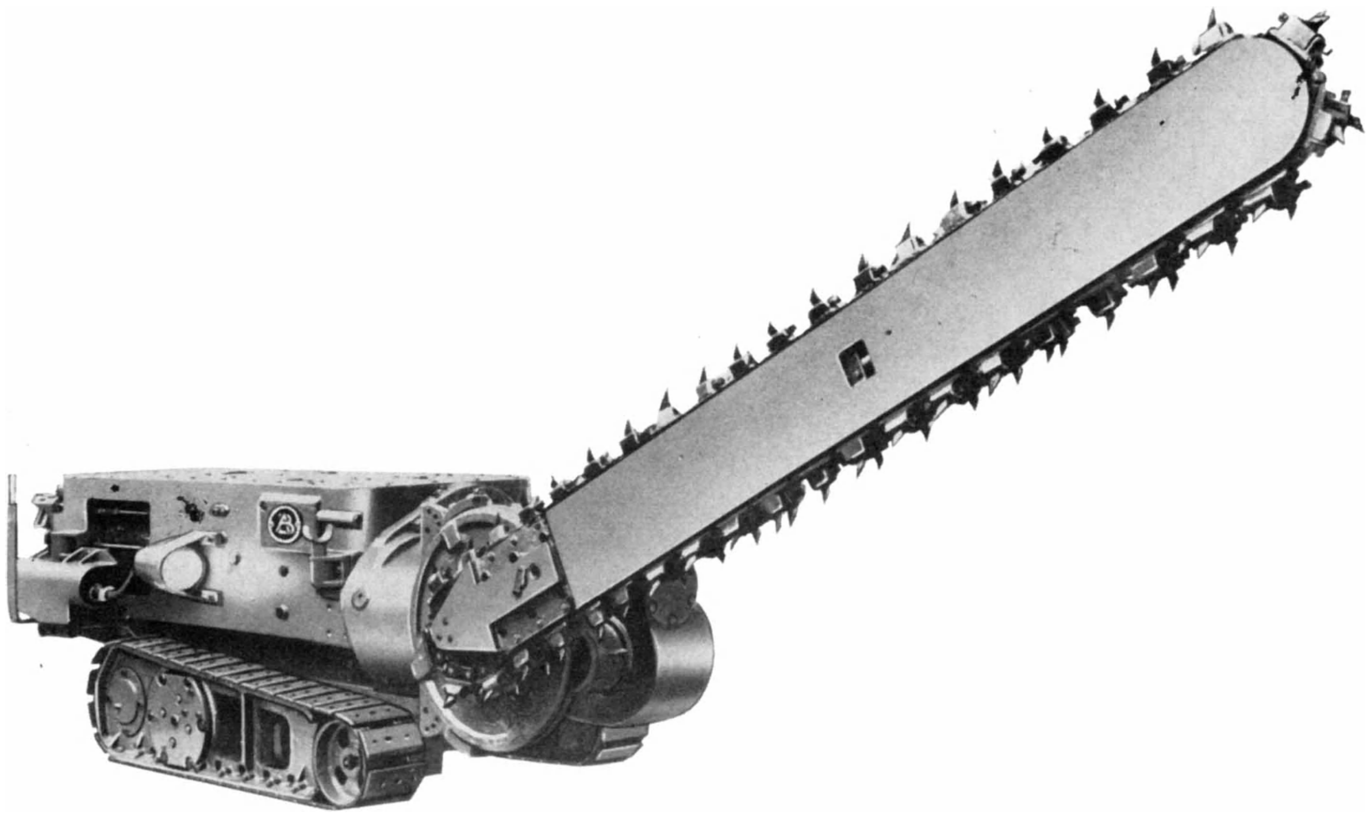

FIG. 2. Universal arc-shearer. (Anderson, Boyes & Co., Ltd.)

The cutting unit drives the cutter chain on the jib. Different makes have different specifications for cutting speeds, rate of travel, etc. The length of jib may be anything from $3\frac{1}{2}$ to 9 ft. The length of jib used depends on the nature of the coal seam, geological conditions, the amount of output which can be handled, etc.

The haulage unit consists of two rope drums, one on each side of the machine, driven by the motor through a series of gears. Each drum carries 75 ft of $\frac{5}{8}$-in. wire rope or 120 ft of $\frac{1}{2}$-in. wire rope. Cutting speed can be adjusted between 1 and 5 ft/min. Flitting speed is about 23 ft/min.

The driving unit may be either electrical (a.c. or d.c.) or compressed air.

The height of the machine may be anything from 12 to 21 in. and the width between 2 and $2\frac{1}{2}$ ft.

The speed of the cutter chain around the jib varies from 320 to 650 ft/min. The width of the kerf cut may be between 3 and 8 in. but is usually 5 in.

Height of Coal Cutters

In thin seams the height of a coal cutter is an important factor and account must be taken of the thickness of the seam, the amount of face convergence which will take place, and the types of roof support bars in use. The width of the coal cutter will affect the width of the cutting track and the ease of turning the machine.

Gummers

Cutters should be fitted with gummers which will effectively remove the cuttings from the cutter chain and will leave a clean undercut. Effective removal reduces the load on the driving motor and assists in the preparation of the coal. Gummers may be either paddle type or propeller type.

Height Adjustment

On the A. B. Fifteen coal cutter the height of the cutting jib may be adjusted to suit seam requirements. For a height of up to 13 in. the machine is raised bodily by a jack so that it rests on timber flats. For a height of 13 in. or over the gearhead is inverted.

For greater heights of cutting jibs may be mounted on hydraulic turrets. The hydraulic lifting and lowering arrangement is housed in the turret itself and is completely isolated from dirt or damage.

Types of Jibs

Single-jib machines are used successfully in many collieries but difficulties are sometimes encountered where roof is sticky and there is need for both over-cutting or cutting a dirt band simultaneously. In such cases coal cutters fitted with twin

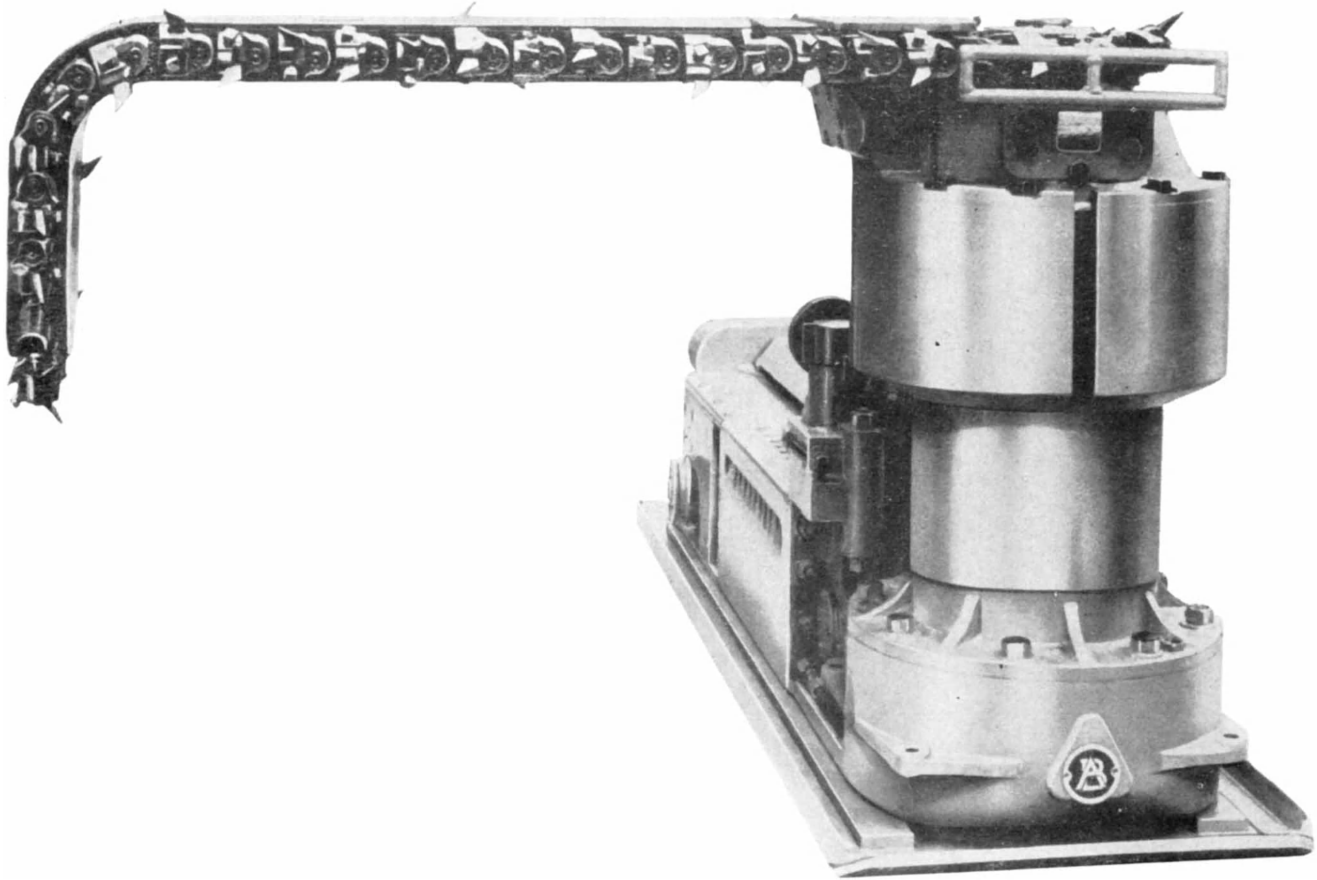

FIG. 3. Hydraulic turret overcutter with down-curved jib for roof cutting. (Anderson, Boyes & Co., Ltd.)

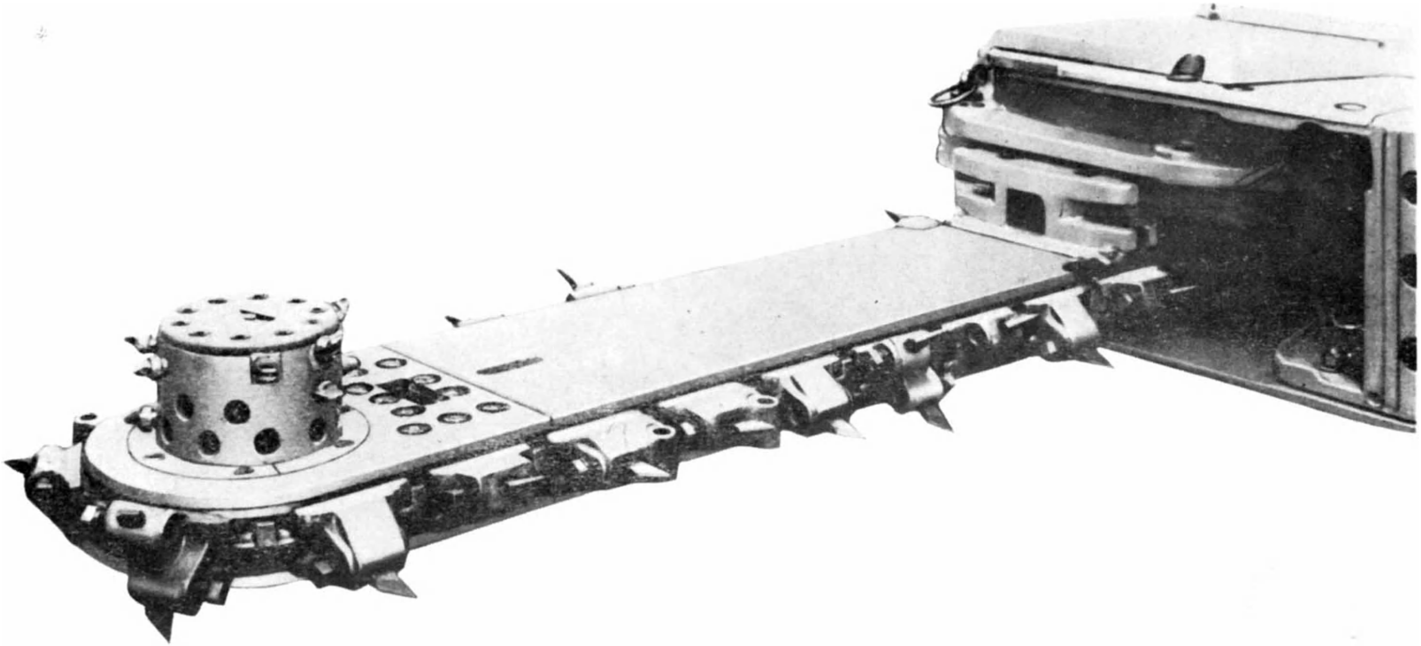

FIG. 4. Mushroom jib. (Anderson, Boyes & Co., Ltd.)

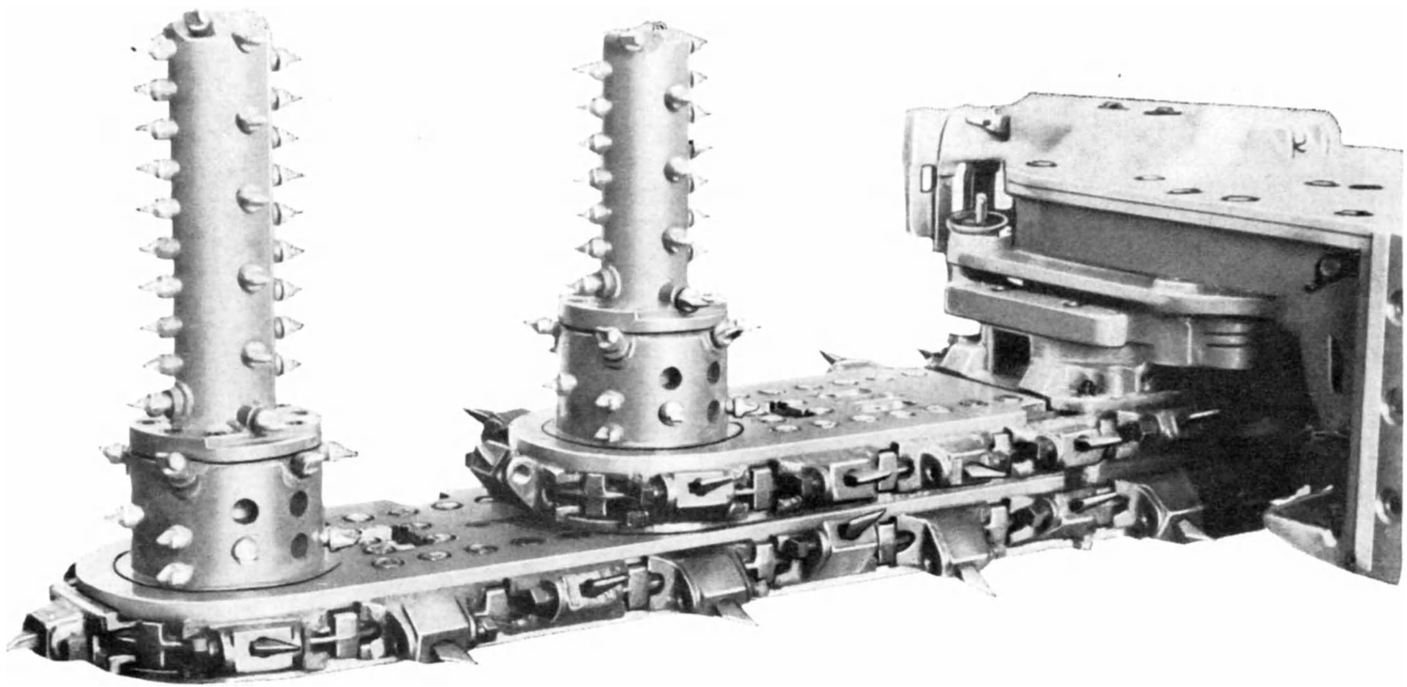

FIG. 5. Twin "XY" mushroom jib. (Anderson, Boyes & Co. Ltd.)

jibs or with stepped jibs may be used. The stepped jib is useful in allowing roof bars to be extended over the face track while cutting at roof level.

In cases where there is need for the coal cutter to be mounted on an armored conveyor, the under-cutting at floor level is facilitated by the use of bent or slightly curved jibs.

Mushroom-jib Coal Cutters

The coal-cutter jib may be so designed as to give a shear cut at the back of the web in addition to the usually horizontal cut. This shear cut may be made by using either a mushroom jib or a curved jib.

The mushroom head is incorporated in a special jib in such a manner that it can be fitted directly to any standard coal cutter. The mushroom head and the sprocket at the end of the jib are combined and thus the mushroom receives its drive direct from the standard cutting chain. The mushroom and the sprocket are mounted on a stationary vertical shaft which is supported by the bottom jib plate: this plate, together with the top jib plate, is of special construction and ensures maximum rigidity at the jib end.

The mushroom is mounted on ball bearings arranged so as to give rigidity as well as maximum protection against dust.

The A. B. mushroom jib can shear up to a maximum height of 10 in. from the floor and extension pieces are available in 2-in. steps by means of which the vertical shearing height can be extended an additional 14 in. (A mushroom jib fitted with an extension is designated as a "turret jib".)

When a shear cut is required in the middle of a web as well as at the back, a twin turret jib is useful. This jib is used in conjunction with a special deep undercut gumstower.

For satisfactory jib performance it is necessary to use a gummer to remove the cuttings to leave a clean undercut. Mushroom jibs are not recommended for overcutting with turret turned downwards because there would be no method of removing cuttings from the shear cut and they would tend to jam the machine.

Advantages of Curved Jibs and Turret Jibs

Some advantages claimed for curved jibs and turret jibs are:

(1) The shear cuts reduce the amount of shot firing required to break the coal and in some cases entirely eliminate the necessity for shotfiring.

(2) The coal sizing may be improved with a larger percentage of larger sizes being produced.

(3) Output per man-shift is increased.

(4) The shear cut gives a straight face line and makes installation of support easier.

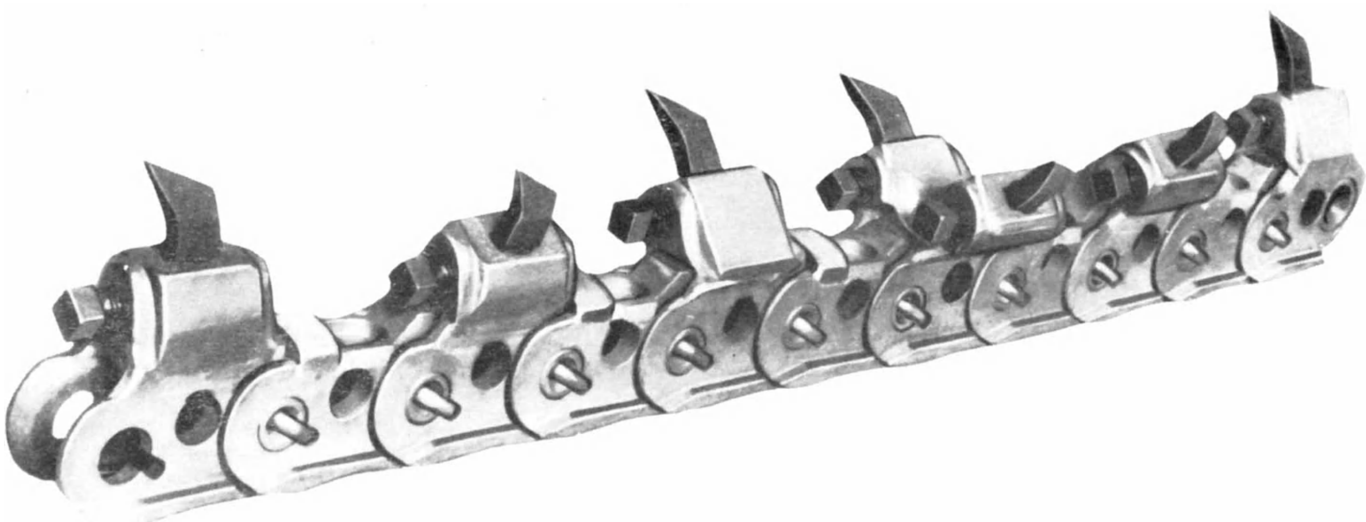

FIG. 6. A.B. Titan extra-heavy cutting chain. (Anderson, Boyes & Co. Ltd.)

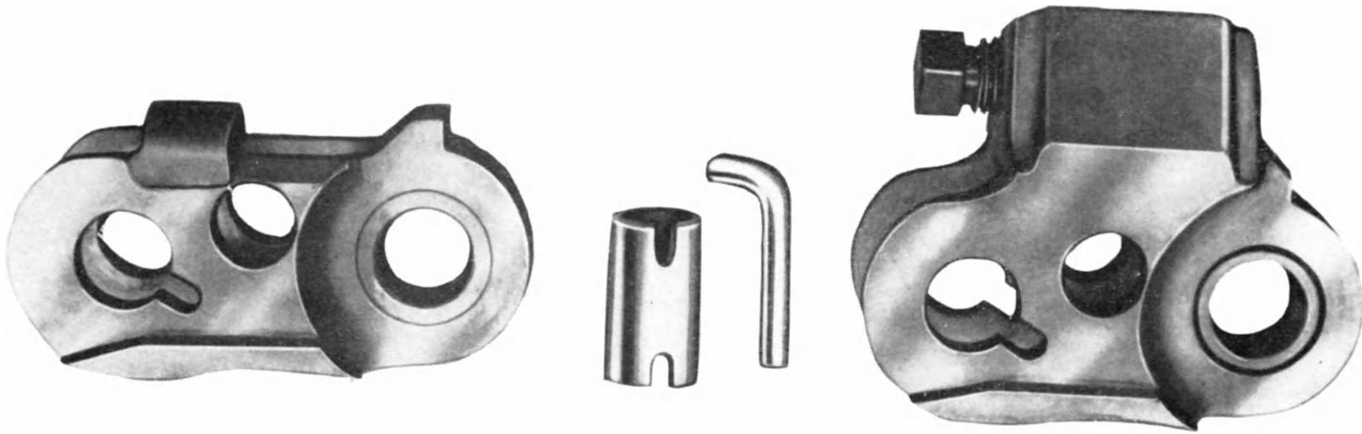

FIG. 7. Components of A.B. Titan extra-heavy cutting chain. (Anderson, Boyes & Co. Ltd.)

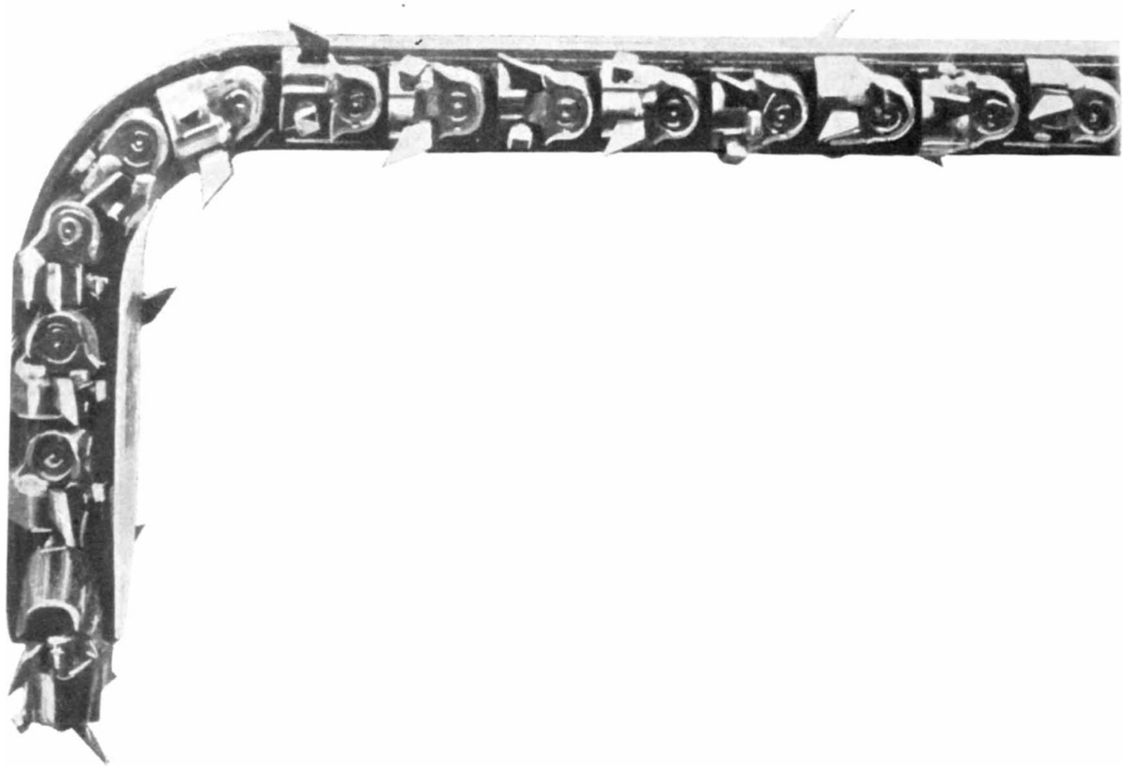

FIG. 8. A.B. curved jib with double-articulated cutting chain (Anderson, Boyes & Co. Ltd.)

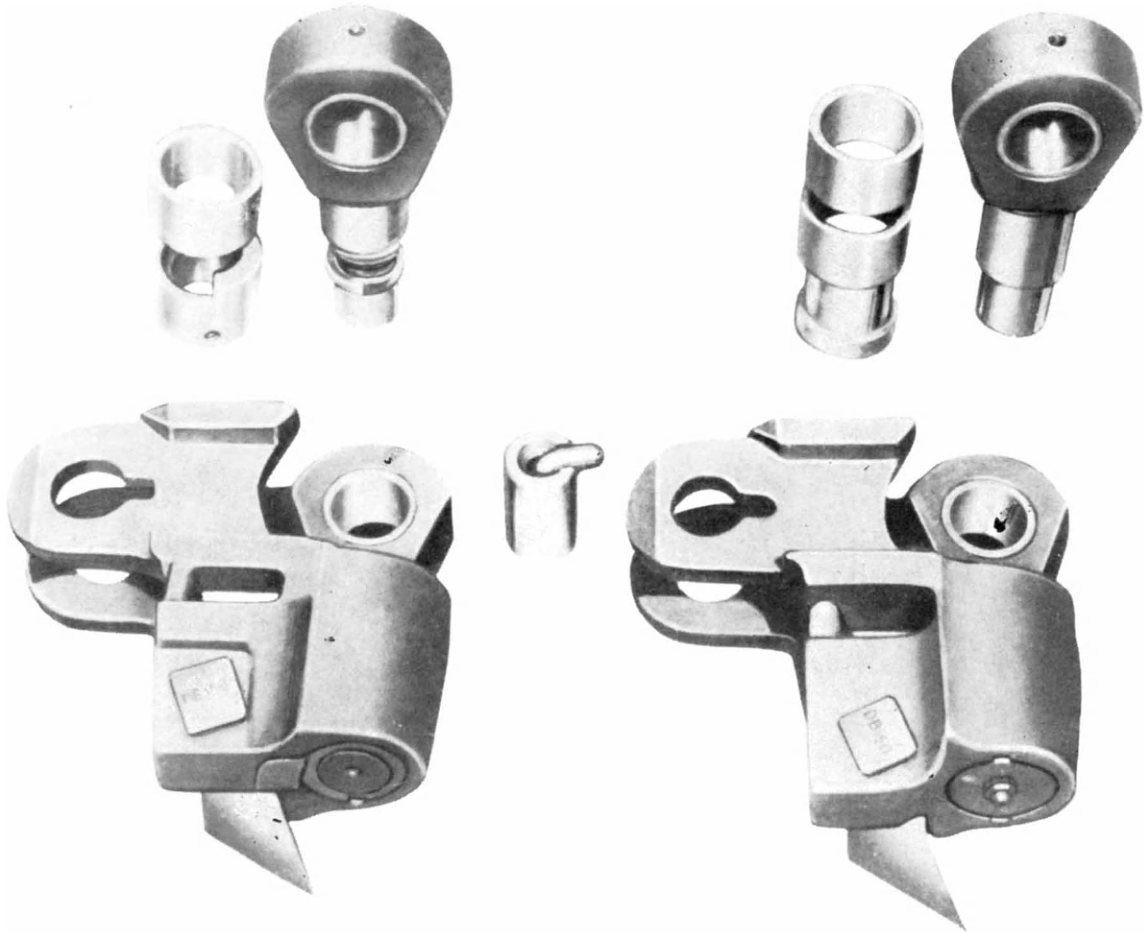

FIG. 9. Components of double-articulated cutting chain. (Anderson, Boyes & Co. Ltd.)

The following disadvantages of curved or turret jibs are noted:

(1) Great care is needed when starting and finishing a cut at the corners of the face.

(2) Power consumption is high.

(3) Chains must operate at slower speeds.

(4) A gum stower is essential.

(5) Gas may accumulate in the shear cut.

Applications for Curved Jibs

Curved jibs have been applied in a variety of ways. Some of these applications are indicated in Fig. 10.

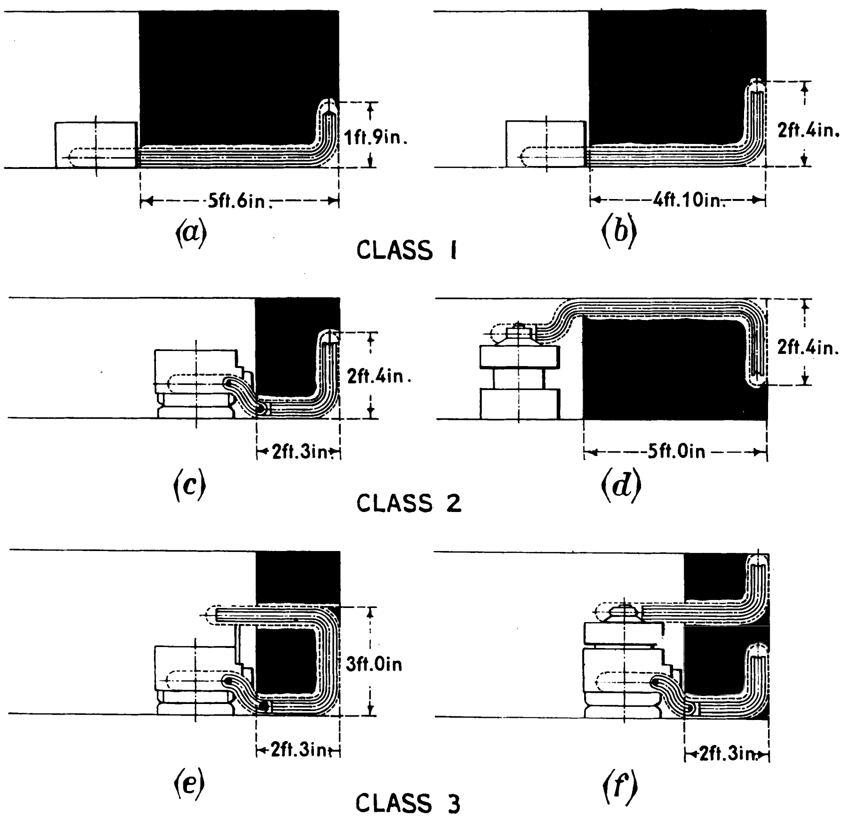

FIG. 10. Various types of curved jibs.(2)

In one instance in a 4-ft thick seam on a face 600 ft long where the coal cutter was mounted on the armoured conveyor and was equipped with a curved jib an overall face output of 7 tons/man-shift was obtained. The face was hand loaded. The cutter made a 2 ft 3 in. undercut and a 2-ft shear cut.

POWER LOADERS

Coal at the long-wall face was generally hand loaded until the period between 1940 and 1950 when power loaders for loading coal onto face conveyors began to come into general use.

Power loading machines were at first designed to load the strip of coal cut by a standard coal cutter. This strip was usually several feet wide and required wide

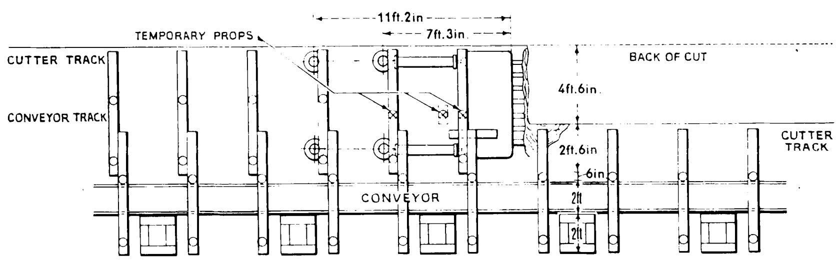

FIG. 11. Face support system with Huwood double jack-post loader.[2]

loaders. Power loaders for long-wall faces did not achieve a real break-through until the mining system was changed so that a narrow web of coal was cut along the face and narrow loaders were designed to load it. These machines allowed the space between the last row of props and the face to be narrowed so that the roof could be supported by cantilever bars and an unobstructed space (prop-free face) provided in which the face conveyor could be located and along which the power loader could pass. The conveyors used were of a semi-flexible type which could be snaked over to the face as the face advanced.

Types of Loaders

Power loaders for long-wall faces may be divided into three classes:

(1) Machines which are designed especially for loading.

(2) Coal cutters which may also be used for loading by the addition of flights to the cutter chain.

(3) Combination machines which both cut and load the coal during one pass along the face.

(1) Loading Machines

The Huwood loader is designed especially for loading. Earlier models of this machine were rope hauled along the face but more recent models are propelled by two hydraulic rams which push against hydraulic jack posts which are set between floor and roof.

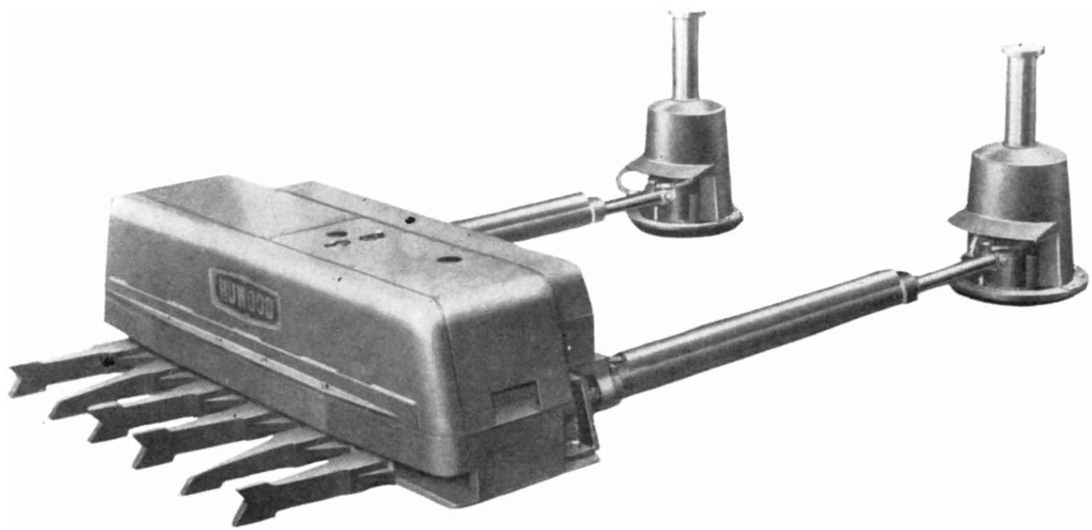
FIG. 12. Huwood loader. (Hugh Wood & Co., Ltd.)

Loading action is provided by a series of arms which project in front of the machine and which operate with a reciprocating motion to sweep the broken coal away from the face and onto the face conveyor. This conveyor may be either a chain or an armoured conveyor, or it may be a bottom belt conveyor.

The Huwood loader can load at the rate of 1 ton/min; however, the coal must be well prepared to enable the machine to function successfully.[2]

The loader can travel at a maximum speed of about 220 ft/hr but a speed of only 90 ft/hr is considered to be sufficient for the completion of loading out a face on shift. It is claimed that 360–420 ft of face can be loaded during a working shift. It has been recommended that the cutter should be kept at least 150–180 ft in front of the loader.

A seam from $2\frac{1}{4}$ to $3\frac{1}{2}$ ft thick is most suitable for this type of power loading, although a low type of loader has been made to be used in seams as low as 1 ft 11 in.

(2) Loading With Modified Coal Cutters[2]

The simplest form of power loader is created simply by reversing the picks on the cutter chain of the conventional coal cutter. A more efficient loader is created, however, by attaching loading flights to the cutter chain. The pick holders on the chain are adapted for attachment of the loading flights.

This form of loading has been successful in the thinner hard seams, such as those from 2 to 3 ft thick where conditions are not suitable for plows.

In using these loaders the face is first pre-cut and suitably prepared for loading. To convert the cutter to a loader one pick is withdrawn from one of each pair of

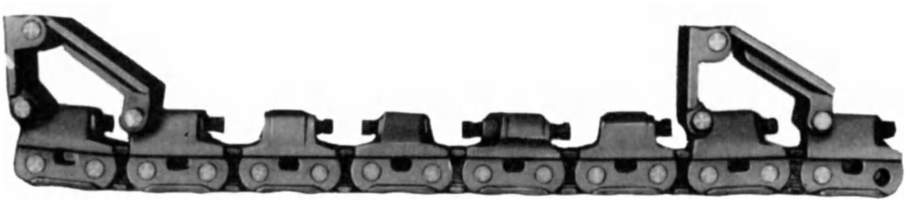

FIG. 13. Cutting chain fitted with loading flights. (Anderson, Boyes & Co. Ltd.)

the flight-carrying pick holders, and flights are attached by means of drop-in pins. Usually one flight per foot of jib length is sufficient for loading purposes. The converted coal cutter then traverses the face in the direction opposite to the cutting direction, with the jib leading and the chain running in reverse so that the broken coal is scraped onto the conveyor.

Figure 14 shows the support system for flight loading onto a bottom belt conveyor in a colliery in the Durham Division. The face was 520 ft long and the seam thickness 2 ft 10 in. Dowty props and corrugated steel bars were used. A face O.M.S. of over 6 tons was obtained with this installation using an A.B. Fifteen coal cutter having a jib fitted with a cutting turret 1 ft 3 in. high.

The whole operation was based on a rigid cyclic system, involving undercutting the coal, drilling and blasting the coal during the preparation shift, and then attaching flights to the chain ready for the next shift.

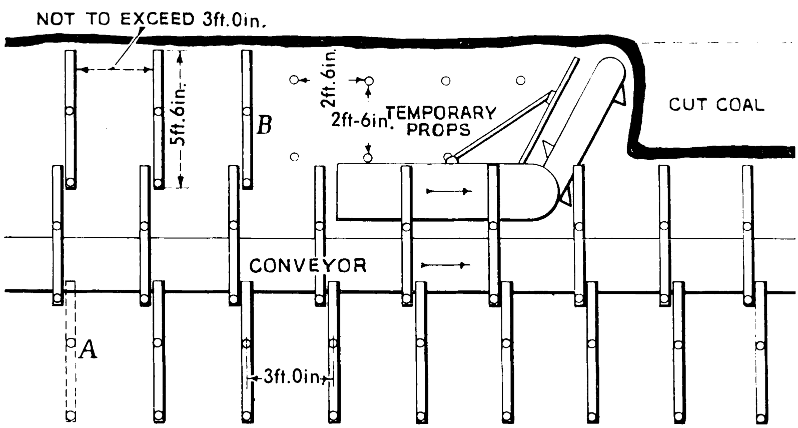

FIG. 14. Face support system for flight loading.[2]

In the second shift the coal was loaded by the flight loader and then permanent supports were set in the new track behind the machine. New stables for the cutter were also made on this shift.

The third shift consisted of moving the conveyor forward and withdrawing back supports. The roadhead was also advanced and the face packed during this shift.

Conveyor-mounted Cutters[2]

The standard chain coal cutter may be mounted on an armoured chain conveyor. Such a coal cutter was used for cutting and flight-loading in a colliery in the North-western Division (of Britain) in a 3-ft thick seam, on a face 240 ft long which was undercut to a depth of $7\frac{1}{2}$ ft. A face O.M.S. of 7.8 tons was obtained as compared with an O.M.S. of 5 tons for orthodox working.

Extent of Usage of Flight Loading[2]

The use of flight loading has been increasing continuously since 1951. Flight loading is the cheapest system of power loading and it can be used under a variety of conditions. It can also be used on faces which will have short working lives thereby avoiding the heavier capital expenditures which would be required for installation of other types of equipment.

The principal disadvantage of this system is that the flights probably cause some degradation (breakage and size decrease) of the coal.

The Scraper-box Loader[2]

The scraper-box is a relatively simple form of power loading used underground. The coal is cut and broken in the usual manner and is then scraped to the end of the face by dragging a scraper box to and fro along the face.

The box is attached to wire ropes which are actuated by a winch placed in the tail-gate road.

To hold the box against the face and in the coal a skid rail is usually set parallel to the face and is advanced by means of ratchet pushers as the face advances.

The scraper-box system eliminates the need for a face conveyor but it does have a low efficiency. This is partially offset, however, by the relative excellency of its performance in very thin seams where working conditions are very unfavorable.

A narrow design of scraper box with collapsible arms was introduced in a Scottish colliery and this enabled supports to be erected almost as close to the face as in ordinary practice. The scraper box was powered by a 100 h.p. winch with double drums. The specified rope speed was 200 ft/min and the maximum pull on the $\frac{7}{8}$ in. diameter rope was to be 15,000 lb. The equipment was installed on a double-unit face in a 19-in. thick seam, each unit being 280 ft in length. The face was cut by a 12-in. double jib machine and the coal was scraped to the center loading road by scraper boxes on each side.

SEMI-CYCLIC OPERATIONS

The preceding discussion has dealt with cutting and loading machines for cyclic operations in which cutting, breaking down the coal, and filling it onto the face conveyor were separate operations.

Non-cyclic operations will be discussed in the next chapter and involve the use of machines which rip or cut the coal from the seam and load it onto the face conveyor without the necessity of drilling and shooting to break down the coal. Generally in long-wall "continuous" or "non-cyclic" mining a relatively thin slice of coal is removed from the face at each passage of the mining machine. Since these machines are relatively narrow, are usually mounted on an armoured face conveyor, and only a narrow web of coal is removed at each pass, the roof at the face can be supported by bars cantilevered from a row of props set about 3 ft from the face. Thus coal getting operations are conducted on a "propfree face". A continuous mining (non-cyclic) system attempts to produce coal at a fairly steady rate during two, and sometimes all three shifts in contrast to cyclic operations where all coal

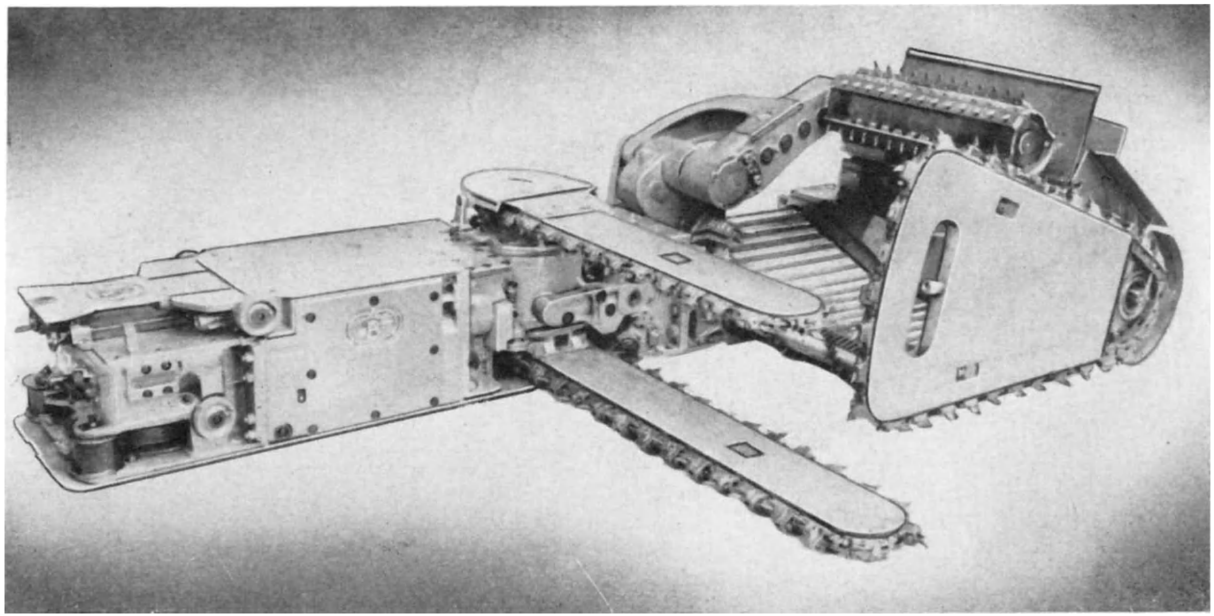

FIG. 15. Meco-Moore cutter loader. (The Mining Engineering Co., Ltd.)

is loaded out on one shift. The non-cyclic system tends to produce a relatively smooth flow of coal through the transportation system in contrast to the cyclic system which imposes peak loads on transportation during one shift.

Semi-Cyclic Systems

Between the "cyclic" systems and the "non-cyclic" systems there are some machines which operate on what might be called the "semi-cyclic" system. In these operations the coal is gotten without having to drill and shoot to break it down.

One such system is that known as "waffling". This system may be used when the coal is of such nature that machine cutting of the face produces adequate breakage and no shot firing is required. In such a situation the coal is cut by one machine and is loaded by a flight loader which follows the cutter along the face.

The distance which the loader follows the cutter is determined by the nature of the coal and whether it will settle and break immediately or whether some time must be allowed for breakage.

The Meco-Moore Cutter Loader(3)

The Meco-Moore cutter loader simultaneously cuts the coal and loads it onto a face conveyor. Initially the machine was designed to cut and load the coal alternately but it was later modified for simultaneous cutting and loading.

The machine consists of two main sections: (1) the cutting unit; (2) the loading unit.

(1) The cutting unit consists of a special long-wall coal cutter which is equipped with two horizontal jibs – one for undercutting and the other for overcutting. The chains in the two jibs travel in opposite directions. This helps to stabilize the machine and to break up the coal.

(2) The loading unit comprises the following:

(a) Left- and right-hand gummers.

(b) Left- and right-hand loader gear boxes with loader bars.

(c) Loader structure with belt.

(d) Shear jib.

Since the machine is designed for work in either direction of travel the gear boxes for the gummers, the loader bars, and the shear jib are arranged symmetrically about the center line of the loader belt. This makes it possible for the cutter unit to be attached to either end of the loader unit.

The Meco-Moore cutter loader can be used under variable conditions. It cannot be used, however, in seams less than 3 ft thick and it requires a good roof and strong floor. Where the coal does not fall easily from the roof a picked drum may be fitted to the cutter loader. The picked drum also helps to break up large lumps of coal. The machine requires stable holes 15–30 ft long at each end of the face.

An "on end" face is most suitable for this machine as an "on incline" face may give trouble in breaking the coal away when cutting against the cleat. The machine can work on slopes of 1 in 5 along the face and 1 in 15 on faces advancing to the dip.

Single unit faces may range in length from 390 to 450 ft. The length of a single face worked by a Meco-Moore cutter loader is restricted because the machine is cyclic; that is it can ordinarily cut and load on only one shift leaving the other two shifts for turning the machine, packing and drawing, ripping, etc. This machine usually takes off a 5 ft strip of coal.

Modified Operating Methods[3]

Modified methods for using the Meco-Moore machine have been devised so as to take two cuts of coal from the face each 24 hr instead of one cut. This can be accomplished by one of the following methods:

(1) The shuttle Meco-Moore system. With the shuttle system three cutter sections are used in conjunction with one loader unit. At any one time one cutter portion is incorporated with the loader unit, a second is in the approach stable being made ready for use on the next pass down the face, while the third is being used to cut the other stable before being turned and prepared for loading.

By having a cutter portion already turned and prepared for coupling to the loading section, the time taken to turn around for a return pass is greatly reduced and can be done in $1\frac{1}{2}$ hr.

With a double shift per day working face advance of 9 ft is obtained and the estimated output of 750 tons/day from a shuttle Meco face at Thoresby Colliery (E. Midlands Division) is regularly exceeded. Face output per man-shift at this face is 7.1 saleable tons.

(2) A cutter unit may be coupled to each end of the loader unit. This method eliminates the necessity for turning the machine around when it reaches each end of the face.

(3) A cutter-loader unit is used in each stable. The cutter unit is coupled to the machine when it cuts through into the stable and the cutter unit which has just cut the face is uncoupled and becomes the stable machine. This simplifies the turning operation. In this way another web can be cut as soon as the face conveyor is moved up.

Cycle of Operation for One Coaling Shift Per Day

The organization of the working cycle for one coaling shift per day may be as follows:

(1) Day shift: The face is loaded out by the cutter loader and the stables are made.
(2) Afternoon shift: The work consists of ripping, packing, and conveyor shifting. The cutter loader is turned and made ready to load the next cycle. The machine is examined and lubricated.

FIG. 16. Dosco Miner at work at the face. (Dominion Steel and Coal Corp., Ltd.)

(3) Night shift: All work is completed and made ready for the loading shift.

The normal face crew during the production shift consists of fifteen to sixteen men.

Requirements for Efficient Operation

The Meco-Moore cutter loader runs on the floor of the seam and does not use the prop-free front support system.

The efficient application of the machine depends upon the following factors:

(1) A definite system of operation and roof support should be adhered to.

(2) The face and props should be kept in a straight line. The conveyor should be laid close to the face props.

(3) The face conveyor should be adequate to deal with a large output.

In view of the rather cumbersome build of these machines and also because the Meco-Moore machine has to be turned around each time it traverses the length of the face it is not likely that there will be any great increase in the number of these machines working in the future.

The Dosco Miner

The Dosco miner is comprised of (1) the main frame and (2) a sliding section which carries the cutting head.

(1) The main frame is mounted on crawler tracks and carries the electric motors, hydraulic pumps, and the oil storage tank and all controls.

(2) The head section slides on the main frame and carries the cutting head, its elevating mechanism, and the cross conveyor. The sliding section can move backwards and forwards for a distance of 18 in. and is pivoted so that it can be swung through a vertical arc but cannot be swung from side to side.

The total weight of the machine is about 20 tons, giving a floor pressure of about 20 psi. When fully raised the cutting head is $7\frac{1}{2}$ ft above floor level.

The machine is equipped with a cutting head which is similar to the ripper heads on the continuous miners used in room-and-pillar work. The ripper head is equipped with a series of seven cutter chains which run side by side and give an effective cutting width of 4.75 ft at the front end.

Method of Operation

The crawler-mounted machine moves up against the buttock of the long-wall face and the cutting head is lowered to the floor and sumped into the coal for a distance of 18 in., then raised to rip out the coal to the desired height after which the cycle is repeated. The broken coal is discharged onto a face conveyor.

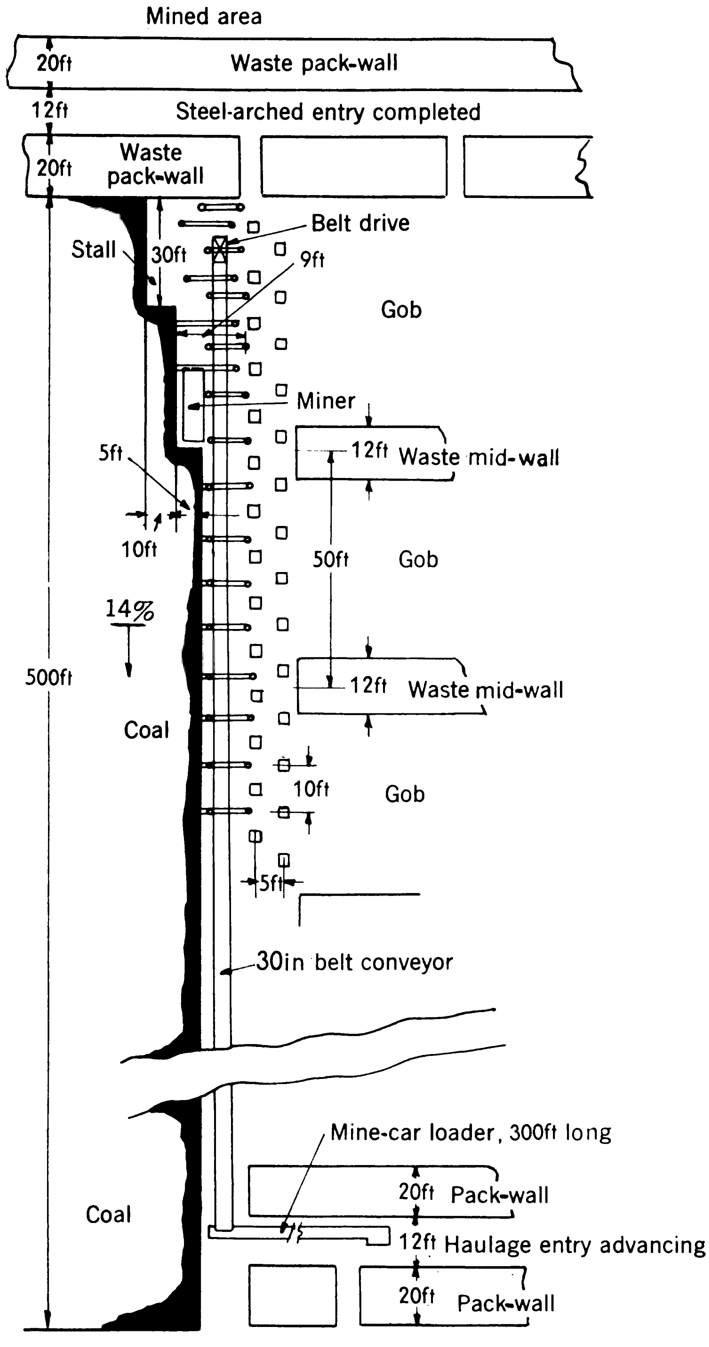

FIG. 17. Layout of a 500-ft long-wall face for mining with a Dosco miner.(4)

The machine was originally designed and built by the Dominion Steel and Coal Corporation of Canada for use in its mines in Nova Scotia. Long-wall faces were 500 ft long and the Dosco miners were operated down the dip on a 14 per cent slope.

Operating Cycle

At the beginning of the cycle of operations a machine is located in the stall at the top of the face. This stall has been prepared by three men using an undercutting machine.

The mining machine is operated down the dip and mining of the 500-ft face is scheduled for completion at the end of the first, or production shift.

On the second shift the machine is trammed up the face and placed in the prepared stall ready for the next production shift.

On the third shift the face conveyor is moved 5 ft laterally to a new position 12 in. from the face ready for the next production shift.

The face layout is shown in Fig. 17.

Roof Support

The roof between the gob and the face is supported by wood bars, posts, cribs equipped with releases, and packwalls. In the haulageways the roof is supported with packwalls and steel arches lagged with timber.

Applications of the Dosco Miner in British Practice[3]

One of the machines was delivered to the Rawden Colliery, in England, in 1953. A number of modifications were made to the machine to satisfy British Safety Regulations and in addition the cutting head was redesigned and the seven cutter chains were replaced with two wide mat chains. Bollards were mounted on the ends of the front shaft to cut additional width to obtain required clearance for the body of the machine.

Fitting of the new cutting head resulted in greater efficiency. However, the application of the machine in British mines has been limited. Although it gives a high O.M.S. it causes degradation of the coal. Because of the wide web cut its operation is cyclic in nature with production taking place on only one shift per 24 hr.

The Midget Miner[3]

The Midget miner cutter loader is designed for thin seams, and consists of a main body and the cutting head. The main body is similar to a standard coal cutter while the cutting head consists of four boring arms (resembling the cutting arms on the Jeffrey Colmol see Chapter 3, p. 121) and a peripheral cutting chain.

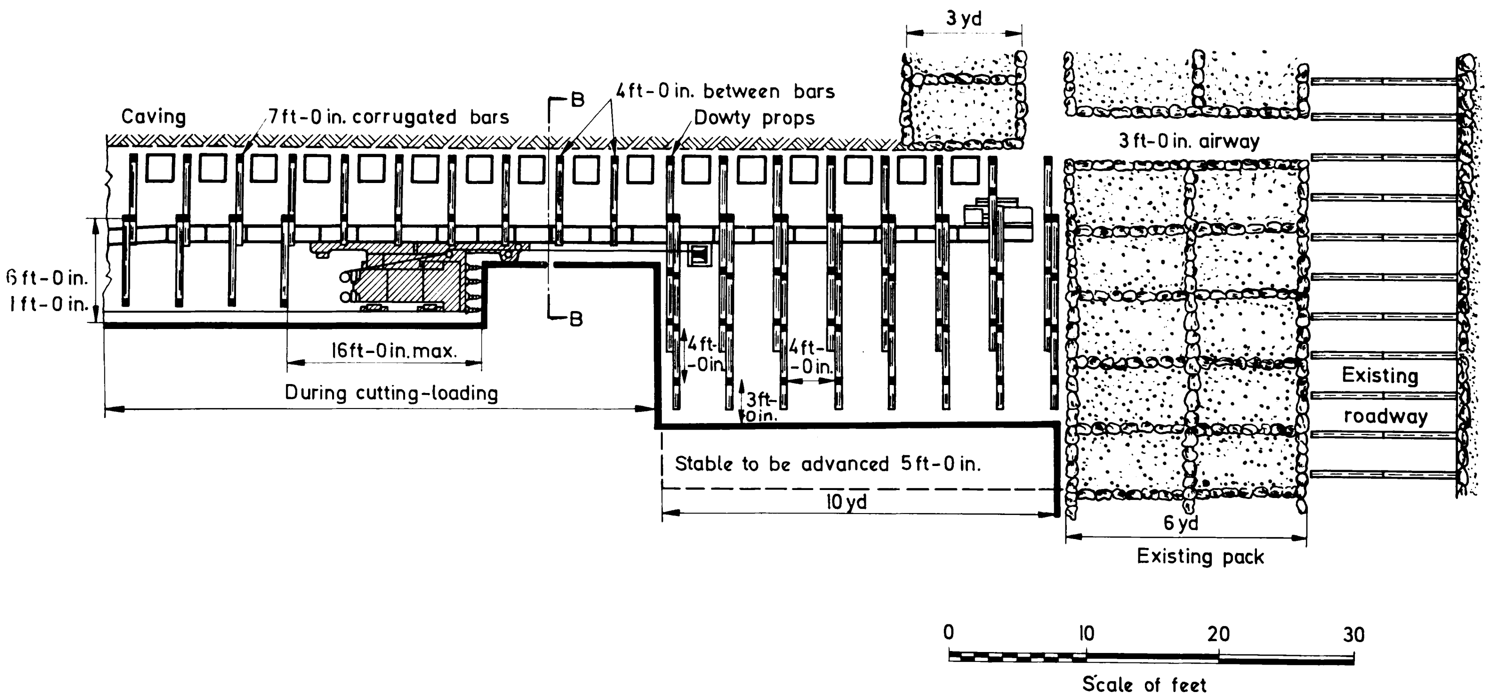

FIG. 18. Typical roof support system on a Midget miner retreating face.(3)

The boring arms are arranged so that the rotation of each arm is successively 72° out of phase with the preceding arm. The cut coal is pushed from one arm to the next until the last arm pushes the coal onto the conveyor.

Different size jibs are available to give cutting heights of 18–30 in.

The machine is mounted on skid plates fitted with four jacks, one at each corner, each of which can be independently controlled. The machine is hauled along the face by wire ropes in the same manner as a standard coal cutter.

A typical layout for a Midget miner retreating face is shown in Fig. 18.

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CHAPTER 5

LONG-WALL MINING—NON-CYCLIC OPERATIONS

MACHINES FOR NON-CYCLIC MINING

Machines for non-cyclic (continuous) operations on long-wall faces are designed to remove a relatively thin slice of coal (3 or 4 in. up to 30 in.) from the face during each passage along the face.

In contrast to this most cyclic or semi-cyclic machines cut relatively wide webs – some machines cutting webs up to 5 ft wide. Because the cut is so wide such machines ordinarily can traverse the face only during one shift in each 24 hr. The remainder of the time is used in moving the conveyor forward, moving and setting supports, ripping and packing, and turning the machine around in its stable hole. Principal disadvantage of this method of operation is that transport equipment must be large enough to handle a large volume of coal on one shift but may be only partially employed during the remainder of the 24 hr. In addition the removal of a wide web of coal tends to stress the roof to a greater degree than does the removal of narrow webs, and the difficulty of roof control is increased.

With the non-cyclic system coal may be cut and loaded on two or even three shifts each 24 hr and a steadier flow of coal is maintained which can be handled by a smaller transportation system.

Non-cyclic machines are relatively narrow and most are designed to ride on the face conveyer so that the space required between the front row of props and the face is not normally more than 3 ft. Thus a "prop-free front" can be used and the roof between the front row of props and the face is supported with bars cantilevered from the front row of props.

Non-cyclic systems are most efficient when used in conjunction with powered self-advancing roof supports which require a minimum amount of man-power for operation.

THE ANDERTON SHEARER

The Anderton shearer is the machine most widely used for continuous long-wall mining in Great Britain. In Great Britain approximately 38 million tons of coal was produced by these machines in 1961.

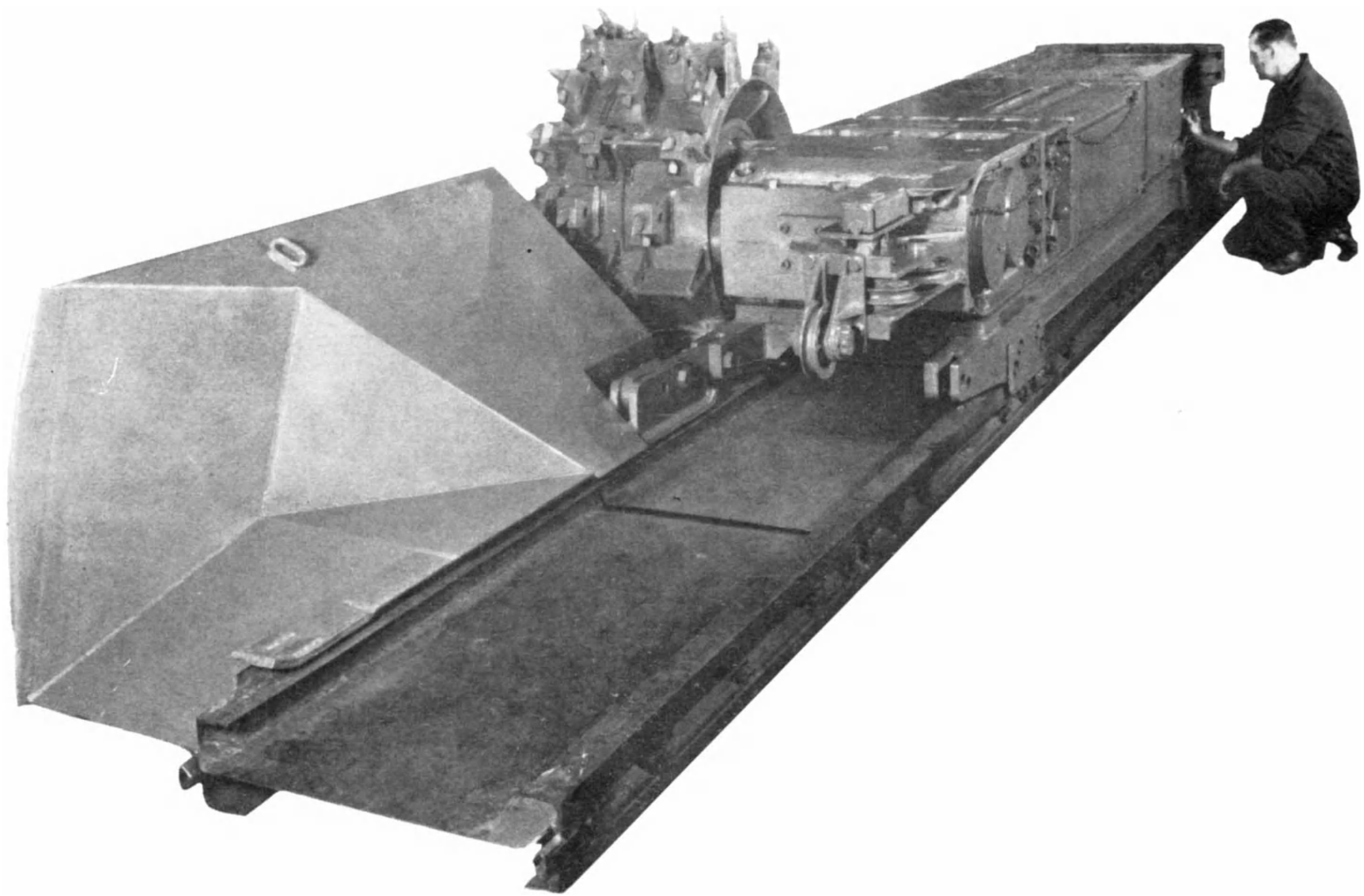


FIG. 1. B.J.D. 150 h.p. Anderton shearer on a C20 armored conveyor. (British Jeffrey-Diamond, Ltd.)

The cutting head on this machine is a drum which rotates about a horizontal axis which is approximately perpendicular to the face line. Coal picks are set around the circumference of the drum. The drum revolves so that the picks cut upward into the coal and the broken coal is thrown back over the drum onto the plow portion of the machine from whence it falls onto the conveyor. Coal which does not come down during the shearing run is picked down, or shot down and falls into the cutter track.

The machine is used in conjunction with armored conveyors and is hauled along the face by a wire rope. When the machine completes a cutting run it is pulled back down the face and the plow portion plows onto the conveyor any coal which may have been missed, or which has fallen onto the shearer track.

The depth of cut taken on each run may vary from 16 to 24 in. or more, depending upon thickness of the seam and the support problems which result from roof conditions.

Where a thick seam is to be worked, or where coal sticks to the roof, the Anderton shearer may be equipped with a jib for making a top cut, or with a curved jib which will make a top cut and also a shear cut.

Because of the nature of its ripping action the Anderton shearer produces a low proportion of large coal.

Mode of Operation

The Anderton shearer shears and loads from the buttock and during the shearing traverse of the face it travels in a direction opposite to the conveyor travel. It starts from the stable hole at the loader gate and proceeds along the face with the shearing drum revolving so that the picks cut upward into the coal and throw a large proportion of it over the drum and onto the plow which deflects it onto the conveyor.

When the machine reaches the stable at the tail gate one of two systems can be adopted:[1]

(1) The direction of the machine's travel is reversed, during which it plows the coal onto the conveyor. This is the common method of operation.

(2) In exceptional cases before starting the reverse haulage the drum may have to be taken off. This may be necessary in thin seams where roof convergence at the coal face may be enough to impede the drum progress if the drum is left in its original position. If the drum is not removed it is not possible to set bars right up to the face when the machine has passed in the cutting cycle. To overcome this difficulty segmented drums have been designed.

As the machine plows back the armoured conveyor is moved forward. In thick seams where the coal does not part readily from the roof an overcutting jib may be employed.

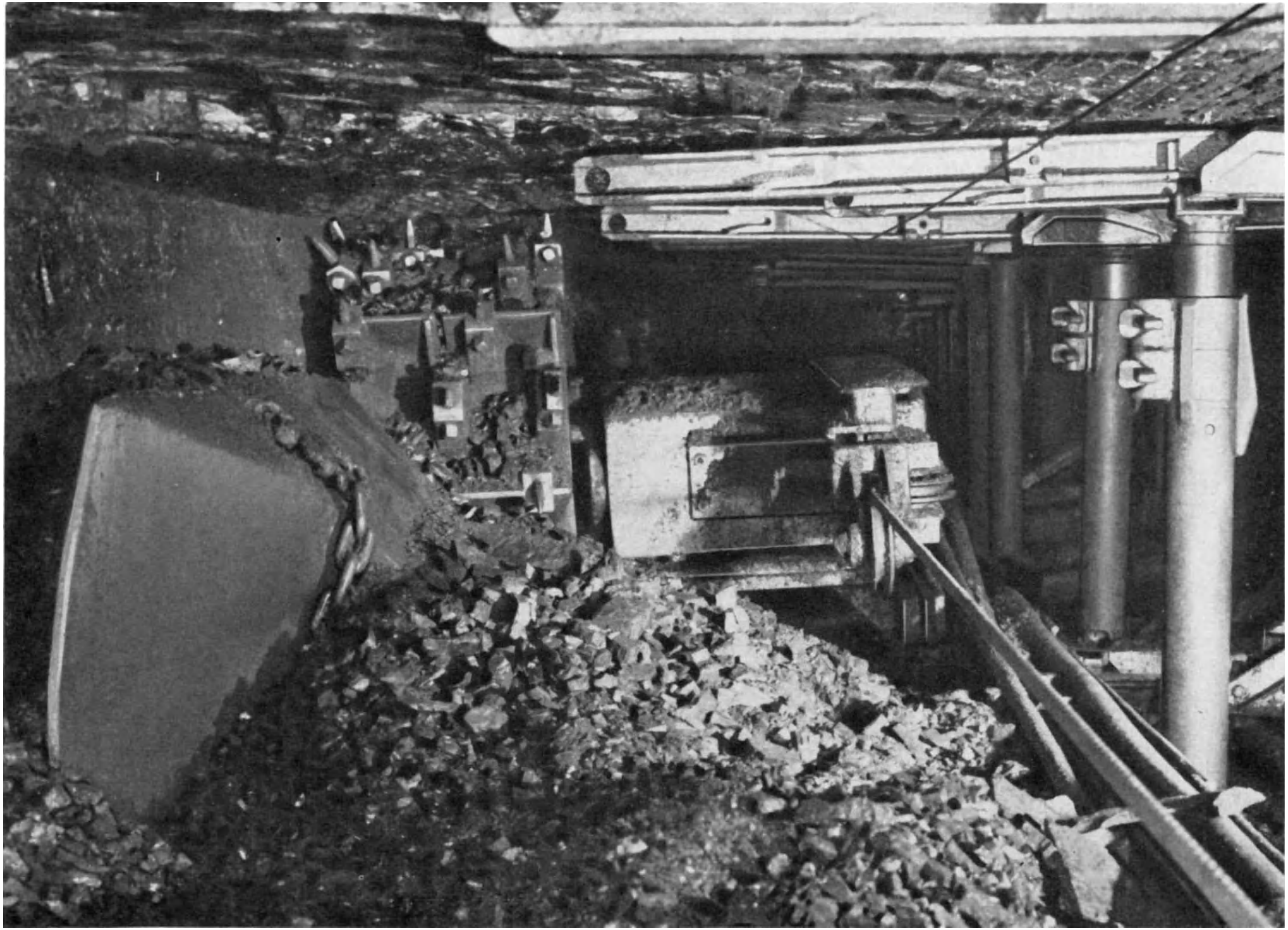


FIG. 2. B.J.D. 150 h.p. Anderton shearer with magnamatic control. (British Jeffrey-Diamond, Ltd.)

Shearing Unit

The original machine was equipped with a series of disks with the cutting picks set on their perimeters. These disks were later superseded by a drum and more recently segmented drums have been used as the rotating elements on which picks are mounted. Segments may be removed from these drums to provide clearance when the machine is flitted back along the face after a cutting run.

The largest drum attempted was 72 in. in diameter but generally if the drum diameter is more than 50 in. the seam may have to be pre-cut.[1]

Drum width is commonly 20 in. although in one case a 30 in. wide drum with a diameter of 50 in. gave good results without pre-cutting. Machines are now available up to 150 h.p.

Speed of travel during the cutting cycle may be from 5 to 30 ft/min, although the average is probably between 6 and 10 ft/min. Speed of travel during the plowing back cycle may be from 30 to 60 ft/min.

Advantages[1]

The following advantages are claimed for this machine:

(1) It is simple and versatile.

(2) It can be adapted to variable geological conditions. The nature of the roof and floor does not affect its operation; it can be used in faulted areas.

(3) It can be used for hard coal.

(4) The price is comparatively low.

The only disadvantage is that it causes degradation of the coal and a large proportion of the production is under 2 in. in size.

British Installations

A 125 h.p. B.J.D. Anderton shearer was recently installed in the Scottish Division.[4] The machine was equipped with a 50-in. diameter drum which left coal at both the top and bottom of the seam. The gradient was 1 in 12 against the cutting and the actual cutting length 500 ft.

The machine took 27–30 min on the cutting run and 18 min on the plowing run, with a complete cycle being completed in 1 hr.

Installation at Whitehill Colliery

In 1959 a 125 h.p. British Jeffrey-Diamond Anderton shearer was installed at Whitehill Colliery. The machine was equipped with 50-in. diameter disks which took a 20-in. wide cut from the face. It was also equipped with a magnamatic transmission which regulated the pull on the haulage rope in accordance with the hardness of the coal, and, in effect, regulated the rate of advance of the machine so that the load imposed on the drive motor by the cutting head was kept relatively constant.

TABLE 1. MANPOWER DEPLOYMENT FOR 21'S FACE.[2]
(Whitehill Colliery)

	Face	Elsewhere underground
Cutting Shift		
Machine operator	1	—
Cableman	1	—
Main-gate stable	5	—
Tail-gate stable	4	—
Wastemen	12	—
Tracker and grader	1	—
Shotfirers	2	—
Face conveyor operator	1	—
Steel checker	1	—
Conveyor operators	—	4
Materials supply	—	2
General worker	—	1
Overman	—	1
Deputy	—	1
Mechanical engineer	—	1
Electrical engineer	—	1
Total	28	11
First Preparatory Shift		
Main-gate rip	5	—
Main-gate pack	1	—
Tail-gate rip	3	—
Tail-gate pack	1	—
Shotfirer	1	—
Conveyor cleaning	—	2
Deputy	—	1
Mechanical engineer	—	1
Electrical engineer	—	1
Total	11	5
Second Preparatory Shift		
Face conveyor grader	1	—
Stage loader extension	1	—
Deputy	—	1
Mechanical engineer	—	$\frac{1}{2}$
Electrical engineer	—	$\frac{1}{2}$
Belt maintenance	—	2
Total	2	4

A 520 ft face could be cut in an average time of 30 min at an average speed of about 17 ft/min. Since the machine was taking a 20-in. web the rate of loading was (during actual cutting) about 260–300 tons/hr and the 30-in. wide conveyor belts were taxed to capacity to handle this amount.

All cutting and loading was done on one shift. It was found to be possible to take four cuts from the face consistently during each coaling shift. The other two shifts were devoted to preparatory work.

The manpower distribution is shown in Table 1. A face O.M.S. of 14 tons was obtained over a period of 5 months.

Huwood T.C.R. props were used in conjunction with G.H.H. bars for roof support at the face. Stables were supported by T.C.R. props in conjunction with 12 ft long 3 by 3 in. H-section girders on double slide bar heads.

A staggered-line system, without chocks, was used for support along the waste edge. The support system is shown in Fig. 3.

Maintenance Costs

An average of about twenty picks per week, or one per slice, were replaced and picks cost about $1.50 each.

It is of interest to note that this shearer cut and loaded over 250,000 tons of coal before being transferred to another colliery and in that period replacement costs were restricted to haulage ropes and a total of about 2000 cutter picks, there being no mechanical, electrical, or hydraulic faults whatsoever in the machine.

U.S. Installation[3]

An Anderton shearer was installed on a long-wall face in the Sunnyside No. 3 Mine of the Kaiser Steel Company at Sunnyside, Utah. The seam ranges from 5 to 6 ft thick with a 6–10 in. parting near the floor. Dip of the seam is approximately 7 per cent and the long-wall face was established straight up and down this dip. Cover over the seam is approximately 1100 ft.

A 125 h.p. Anderton shearer equipped with a drum 5 ft in diameter and 27 in. wide cuts the coal and loads it onto the 30-in. wide British Jeffrey-Diamond armored face conveyor. The shearer rides on the side rails of the face conveyor and pulls itself along by means of a system of sheaves acting on a $\frac{7}{8}$-in. wire rope stretched between the drive and the tail of the conveyor. The cutting speed varies from 10 to 20 ft/min.

The procedure is to cut to the tail end of the conveyor, advancing roof bars behind, remove the drum segment, and then return the shearer to the drive end at 30 ft/min. During the return pass most of the loose coal is plowed over onto the conveyor. As soon as the machine completes a run and the loose coal has been

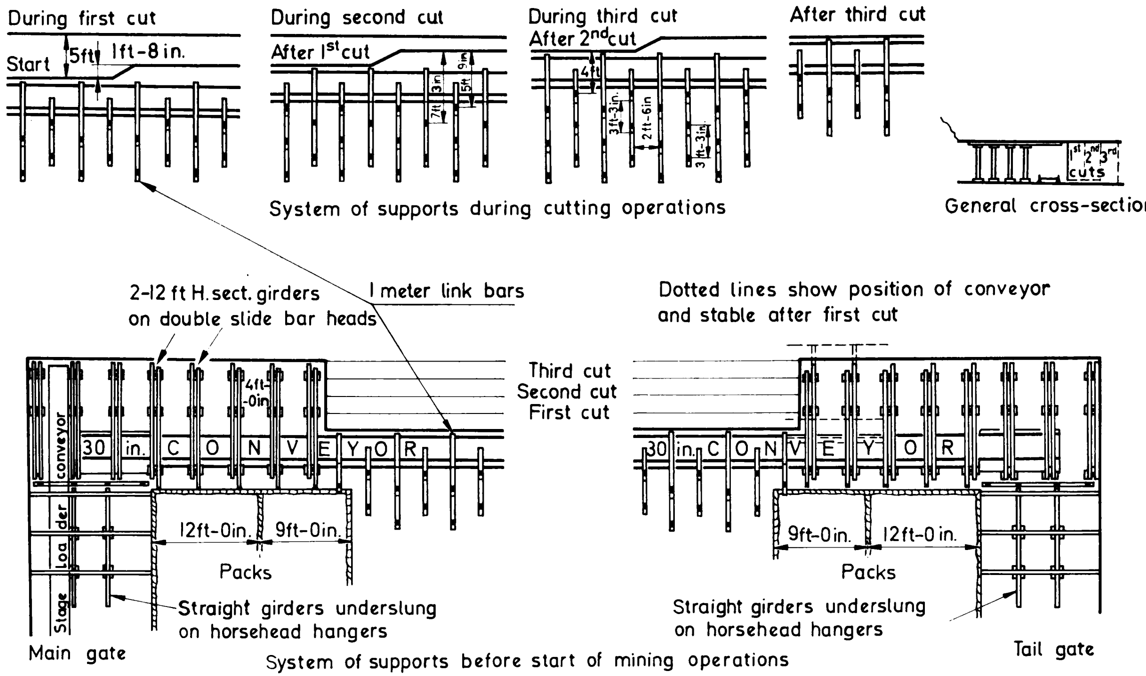


FIG. 3. Support plan with revised support for the main and tail gates.[2]

plowed onto the conveyor and the conveyor pushed up to the face the alternate roof support units are advanced and the machine is ready to begin its cutting run again.

The initial length of the long-wall face was 310 ft but it was planned to install the shearer on a 750 ft face when mining of the shorter block of coal was completed.

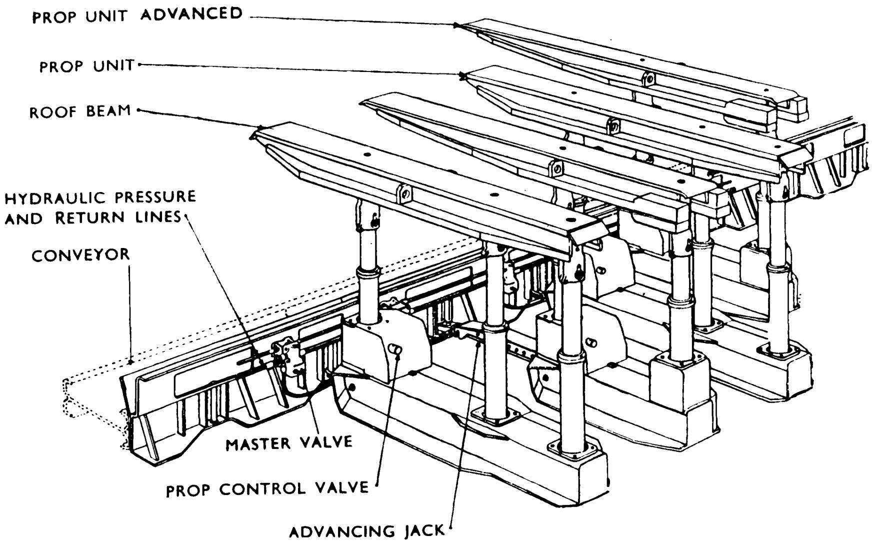


FIG. 4. Dowty self-advancing roof supports. (Dowty Mining Equipment, Ltd.)

Roof Support

Face support is by Dowty hydraulic self-advancing supports. Two and three-jack roof support units alternate on $2\frac{1}{2}$-ft centers. The base of each set accommodates hydraulic cylinders and rams for moving the support unit and also for pushing the face conveyor up to the face.

Crew Required

The face crew consists of eleven men with the following duties:

Shearer operator.
Cableman – corresponds to a helper on a conventional miner.
Cornerman – posted at the drive end of the face conveyor.
Prop men – four men handle advancement of the face conveyor and roof supports.
Mechanic.
Loader-head man – handles car loading.
Motorman – pulling trips and moving them through the loading station.
Face boss.

Productivity

The eleven-man crew consistently produced 400–500 tons of raw coal per shift during the first several months after the installation of this mining system. The best production for one shift was 700 tons. With room-and pillar-work in the same area the average production per man-shift was about 14.5 tons.

A considerable saving in manpower is accomplished by using one of the entries on either side of the long-wall block as a stable into which the shearer advances at the end of a cut. Under British mining conditions it is necessary to hand-mine stables for the advancement of the shearer and conveyor terminals at each end of the long-wall face. Several men are usually required mining and installing special supports in these stables and when they can be eliminated the O.M.S. is boosted considerably.

THE ANDERSON-BOYES LONG-WALL TREPANNER

The A. B. long-wall trepanner, which produced some 23 million tons of coal in Great Britain in 1961, is designed for narrow web non-cyclic mining in thin seams. It travels on the floor of the seam alongside the conveyor and can cut and load coal when traveling in either direction. This machine attacks the coal on the buttock.

The trepanner has a length of steel channel bolted to the bottom which bears on the face side of the conveyor and serves as a guide. The machine pulls itself along (at speeds which may vary from 0 to 6 ft/min) by means of a stationary chain which is anchored to the conveyor at both ends.

The coal cutting and breaking mechanisms are the trepanning wheels which rotate on axes parallel to the coal face. The trepanning wheel has two cutting arms which carry the cutting tools. These cutting tools cut a thin annular groove which causes a cylindrical core to form. Heavy picks are provided on the trepanning wheel to break up this core.

A bottom jib is provided to cut a level floor for the machine and shearing jibs are provided to make side cuts. The gummings (coal cuttings) from the bottom jib are loaded onto the conveyor by paddles mounted on the back of the trepanner.

Water jets are mounted on the machine and sprays are directed at the coal buttock. The machine is equipped with a roof-cutting disk which has the function of cutting roof at the desired horizon or of dressing down sticking top coal.

The depth of the web is 27 in. and normal working height is 3 ft 3 in. to 4 ft although seams up to $4\frac{1}{2}$ ft are worked where the top coal falls freely.

Method of Operation

The trepanner starts from a prepared stable and hauls itself along the face by means of a driving sprocket which winds on an anchored chain. It is guided by the face-side edge of the conveyor. The trepanner wheel takes off a 27-in. buttock of

FIG. 5. A. B. trepanner at work underground. (Anderson, Boyes & Co., Ltd.)

coal and discharges it onto the conveyor. The shear jib and the roof cutting disk cut the remaining top coal causing it to fall onto the sloping top of the machine body and slide onto the conveyor.

The floor cutting jib not only levels off the floor but also pre-cuts the face for the next run of the machine.

After the passage of the trepanner the space between the conveyor and the face is cleared of any spillage and the conveyor is snaked up to the new face again by means of power-operated rams.

New supports are set and the old supports are withdrawn from the waste edge.

When the trepanner enters the stable at the end of its run it is made ready for the return run.

Actual machine handling requires only an operator and a cable man. Almost all other personnel is engaged in setting and removing roof supports, and in preparing the stables.

Example of Installation

A trepanner was installed in Shireoaks Colliery in the Clowne Seam (4 ft thick) on a single unit face 550 ft long. The machine operated on production during two shifts out of each 24 hr, taking four strips off the face during the two shifts, each strip 27 in. wide, at an average speed of 5 ft/min.[1]

Stables at main and supply gates were 54 and 24 ft long, respectively, and were cut by standard A. B. 15-in. long-wall coal cutters.

Using a roof-cutting disk to leave a roof of inferior coal the machine cut to a total height of 3 ft. The face was pre-cut to a depth of 27 in.

TABLE 2. MANPOWER ON TREPANNER FACE, SHIREOAKS COLLIERY[1]

Personnel	Day shift	Afternoon shift	Night shift
Deputies	1	1	1
Shotfirers	2	1	
Trepanner operators	1	2	
Cable men	1	1	
Erecting supports	5	5	
Withdrawing supports	3	3	
Moving conveyor	3	3	
Main gate stable	4	4	
Supply gate stable	2	2	
Ripping main gate			3 or 4
Ripping supply gate			3 or 4
Total	22	22	7 or 9

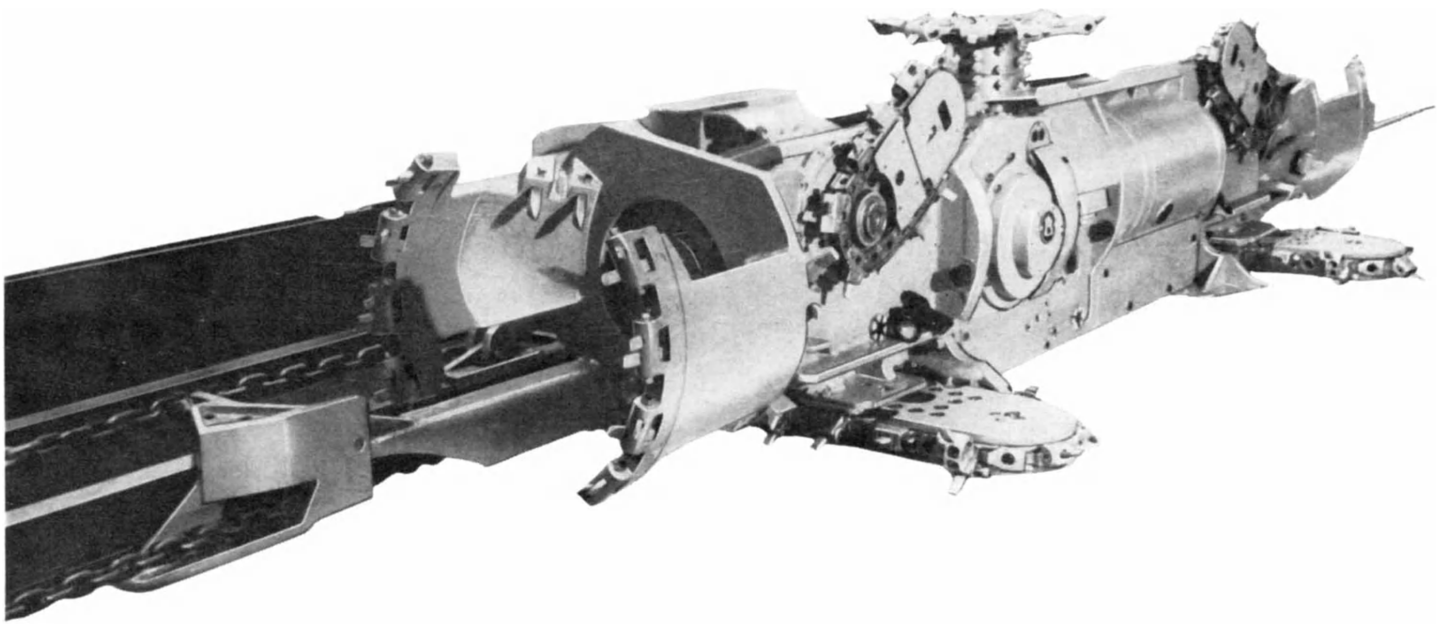


FIG. 6. A. B. trepanner. (Anderson, Boyes & Co., Ltd.)

Face Support[1]

The face was supported by 7-ft light section corrugated bars set on 4-ft centers on hydraulic props, there being three props to each bar. Another hydraulic prop was set between each prop in the rear row to act as a breaker prop, full caving being practiced between gate-side packs.

One man was responsible for setting temporary props on the face side of the conveyor immediately behind the trepanner and about 15 ft behind him four men erected the permanent supports; two setting the bar on a prop erected next to the conveyor on the waste side, a third withdrawing the temporary prop and erecting the face side leg, and the fourth man erecting the waste side leg.

Three men using hydraulic rams moved the conveyor over into its new track; the gap between the conveyor and the face being 6 in. Following some 100 to 150 ft behind the trepanner a team of three men withdrew the breaker props and the remaining two legs of the roof bar, which was set after the previous strip was cut. This team also built the gate-side packs.

Productivity

The average daily face advance was 9 ft and the average daily output was 459 tons. An average of about fifty-one man-shifts was worked at the face during each 24 hr period; thus the face O.M.S. was about 9 tons.

Table 2 is a tabulation of the men employed on each shift and the duties of each man.

Support Systems for Trepanner Faces

The support system for the A. B. long-wall trepanner should have the following basic characteristics:

(1) It should allow for a conveyor advance of about 27 in. during each cycle.

(2) It must be advanced progressively as the machine travels along the face and the rate of this advance should be at least 6 ft/min.

(3) The supports should be so placed that the trepanner operator has good access to the machine controls at all times.

Some of the systems which are being successfully used in conjunction with the trepanner include the following:

(1) Hydraulic Props and Steel Straps

In this method the steel straps may be either $5\frac{1}{2}$ or 7 ft long. Where conditions permit the face is supported on one series of 7-ft steel straps with three props to a strap. As the machine advances along the face another series of straps is erected.

As an alternative method $5\frac{1}{2}$ ft steel straps may be used. In this case the face is supported on two series of such straps, each supported by two props. As the machine advances a third series of straps is erected.

(2) Hydraulic Props and Slide Bars

In this method hydraulic props carry special bar-slide heads and three such props carry a 7-ft joist. The joist is held against the roof by the wedge action of special heads. When these wedges are released the bar is slid along towards the newly

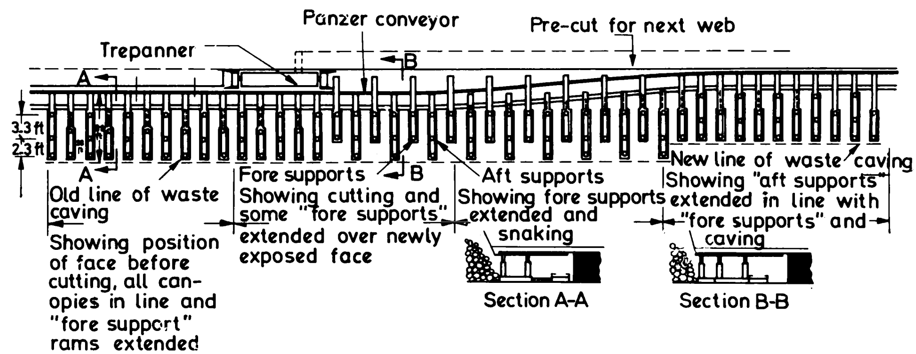


FIG. 7. Self-advancing supports on a trepanner face.[4]

exposed face and tightened against the roof by operating the wedges. This is less expensive in manpower than the application of props and steel straps described above, but requires that the roof be relatively smooth and without undulations.

(3) Link Bars and Hydraulic or Friction Props

This method is used with either hydraulic or friction props and the bar used is normally twice the depth of the web taken so that alternate settings are extended for each cut. The bar size must be carefully chosen to conform to the depth of web and under normal conditions 4 ft 4 in. bars are chosen.

MECHANIZED SUPPORT SYSTEMS

The A. B. trepanner is suitable for use with systems of mechanized support. The Dowty Roofmaster self advancing supports have been used in conjunction with a trepanner and the general layout for the operation of such a system is shown in Fig. 7.[4]

At the commencement of the cycle of operations the rear ends of the roof supports are all in line, with the roof beams set close to the face and providing cantilever support over the conveyor.

As the trepanner moves along the face both props of the two-leg hydraulic support unit immediately behind the machine are retracted and the whole unit is drawn up to the conveyor. The props are then re-pressurized and the roof bar brought up to bear on the newly exposed roof at setting loads which may be varied up to 10 tons/prop.

The conveyor is then snaked over to the face by extending a jack on the two-leg unit which pushes the conveyor over to the face.

After the conveyor is in its new position the props of the three-leg unit are retracted and the jack is closed to draw it up to the conveyor. The three props are then re-pressurized and the units are once again in line, completing a cycle.

Two men are required for the operation of the supports on a face equipped with the Dowty Roofmaster, while a third man advances the conveyor.

Productivity

Under favorable conditions on a long-wall face 600 ft long a crew of ten men would be required to operate the trepanner and the self-advancing support system. Their assignments would be as follows:

1 trepanner operator.
1 roof support man to operate the two-prop units.
1 roof-support man to operate the three-prop units.
1 roof-support man for advancing the conveyor.
1 clean-up man.
1 cable attendant.
1 auxiliary timberman.
1 conveyor operator.
1 shot-firer.
1 maintenance man.

The total investment required for this Roofmaster support system is estimated to be approximately $350,000; this does not include cost of the production machinery.

It is estimated that the foregoing equipment and crew operating on a 600 ft face in a seam 5 ft thick could produce from 750 to 1000 tons of coal per shift.[4]

THE GLOSTER GETTER

The Gloster getter consists of three parts; the motor, the cutting unit, and the haulage unit. These three parts are mounted on a common base plate.[1]

The cutting unit consists of two 2 ft 10 in. bottom horizontal jibs and a top jib which is 2 ft 6 in. long. The horizontal jibs are all in the same vertical plane. The top jib is mounted on an adjustable turret.

In addition there are two shear jibs of lengths suitable for the thickness of the seam.

The machine is powered by a 62 hp (1 hr rating) electric motor. The haulage unit consists of a vertical enclosed rope drum operating through a ratchet gear to give traveling speeds of 1–6 ft/min.

The machine is only 15 in. wide and can operate in seams as thin as 2 ft 8 in. In seams where the roof parts readily and no top jib is necessary seams as thin as 2 ft 4 in. have been worked.

The machine does not have to be turned around in the stable hole and can cut in either direction simply by reversing the direction of rotation of the motor which reverses the direction of travel of the cutting chains. The cutting section is fitted with two dog clutches which allow the shear jibs to be swiveled so that the direction of travel of the cutting chains is always downward.

The machine has been successfully used in many pits and has given saleable O.M.S. of from 4.7 to 10.3 tons; however, the bulk output is small and in certain instances it has failed because it was underpowered and could not cut at a rate fast enough to produce economical results. However, it is a versatile machine and can operate under a wide range of roof conditions and in any type of coal.

COAL PLOWS

The coal plow is designed to be pulled along the face and to cut a layer of coal from the face in a "planing" action. The broken coal is deflected or plowed onto the adjacent face conveyor.

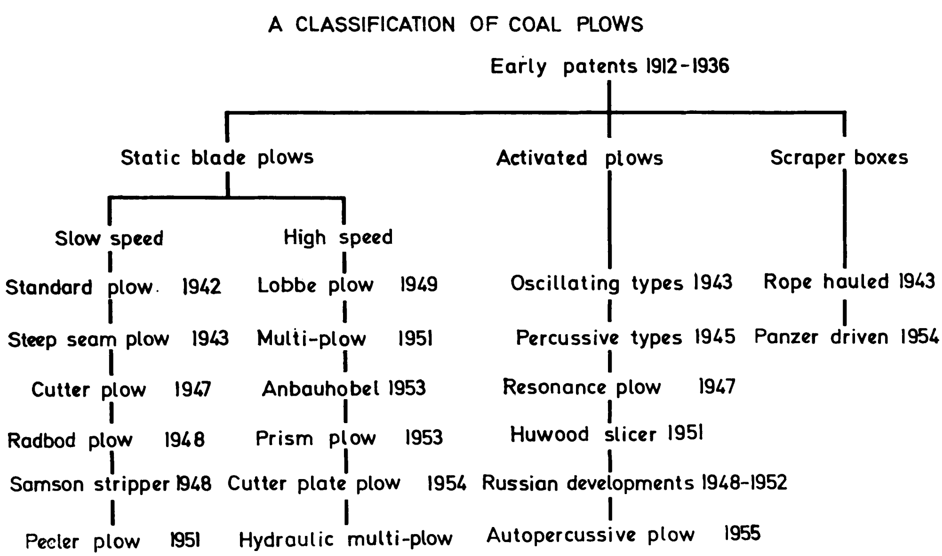


FIG. 8. Coal plough classification.[6]

Coal plows are classified as either "low speed" or "high speed". An additional classification includes coal plows which are actually scraper boxes which both cut the coal and transport it laterally along the face.

Low-speed plows may travel at speeds from 10 to 70 ft/min while high-speed plows may travel at speeds from 70 to 100 ft/min.

The coal plow travels in the space between the armored face conveyor and the coal face and is held against the face by the lateral pressure exerted on the conveyor frame by pneumatic or hydraulic cylinders which are spaced at distances from 10 to 40 ft apart depending upon the hardness of the coal seam.

Plows are pulled along the face by chains or cables. The rope pull required normally does not exceed 20 tons.

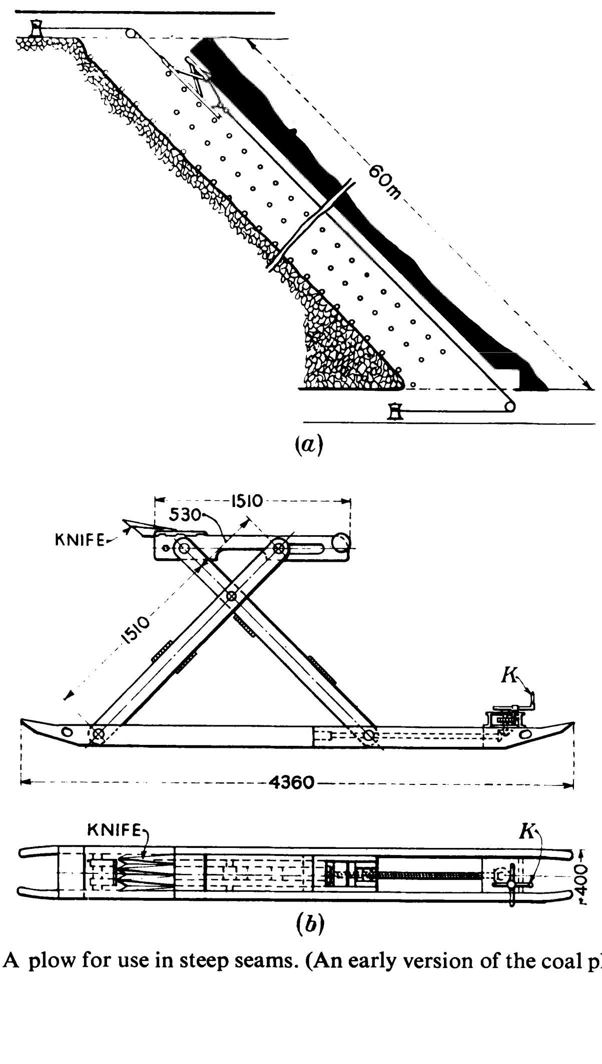


FIG. 9. A plow for use in steep seams. (An early version of the coal plow.)[6]

Figure 8 is a classification of coal plows. Figure 9 shows one of the early models developed in Germany. Because of the special conditions required for its successful application this plow is no longer in use.

The Standard Plow[5]

The standard plow was introduced into Great Britain in 1947 but was only suitable for soft coal and has been superseded by other types of plows.

The standard plow was designed to cut and load coal in both directions and the correct shape and position of the shear blade was of considerable importance. The standard plow was 20 ft long, 2 ft 4 in. wide and 2 ft high and weighed $2\frac{1}{2}$ tons.

The height of plow used depends on seam thickness and is usually one-third to one-half the seam height. The plow was pulled to and fro along the face by means of rope haulage which was capable of exerting a pull of 10–12 tons with a single rope purchase and twice that amount with a double purchase.

In the Morrison Busty Seam on a face 225 ft long the plow gave a face O.M.S. of about 12 tons and on a 420 ft long end face the O.M.S. was about 7.4 tons for the face and 3 tons for the district.

SLOW-SPEED PLOWS

Slow-speed plows include the Kohlenhobel and the Schramhobel.

The Kohlenhobel was the first plow put into service in Germany and is very simple in construction, consisting simply of a chassis on which is fixed a wedge-shaped blade designed to peel a layer of coal about 12 in. thick from the face.

The Schramhobel is similar in operation to the Kohlenhobel except that instead of a single cutting blade it is equipped with a series of blades or cutters arranged in a stepped fashion so that different blades cut to different depths. This arrangement gives a pre-cut and makes it possible to cut harder coals than with the Kohlenhobel.

Slow-speed plows equipped with stepped knives generally cut only in one direction, the cutting run being made at speeds of 15–20 ft/min and the return being made at 30–60 ft/min.

In order to maintain productivity with the slow-speed plows it is necessary to take as deep a slice as possible during the cutting run and the depth of slice may vary from 6 to 12 in., according to the hardness of the seam.

The machine is hauled along the face by a $\frac{7}{8}$-in. diameter wire rope which is wound by a 40-h.p. electrically driven winch located in the tail gate.

Slow plows have been successfully used in conjunction with pre-cutting of the face but it appears that they will be superseded by rapid plows because of the greater productive capacity of the latter machines.

RAPID PLOWS

The Loebbehobel is a rapid plow which is equipped with three or four protruding picks or bits. The plow is double-ended and cuts as it traverses in each direction along the face.

The machine normally travels at a speed of about 75 ft/min and takes a slice of coal 3–4 in. thick off the face at each pass.

It is pulled to and fro by an endless chain which is driven by the gearheads at each end of the armored conveyor. Maximum pulling force is about 20 tons.

This machine is capable of working in seams 1 ft 8 in. thick up to 7 ft thick and can produce, load, and deliver to a gate conveyor from 1.5 to 5 tons/min.[5]

The plow works well in seams which are soft or are hard but friable but will not work well in coal which is "tough" or "woody".

Example of Installation[5]

An installation of the Loebbehobel was made in the Britannia Colliery in South Wales in 1952 on a single unit face 540 ft long where the average seam thickness was 3 ft 3 in. The cleat line was at 45° to the face line. The plow was capable of operating in either direction and took between a 2-in. and a 6-in. slice of coal over the lower 15 in. of the face, the upper section falling after the plow passed.

The plow was hauled across the face by means of a $\frac{7}{8}$-in. mild steel traction chain with a breaking strength of 50 tons.

Pneumatic rams held by staker props were attached to the conveyor and pushed the conveyor and plow up to the face with a pressure of approximately 26,000 lb. Stables at each end of the face were formed by a team of men who used pneumatic picks to get the coal.

The face was supported by G.H.H. extendable roof bars and Dowty props with a prop-free front system. Waste was pneumatically stowed to minimize subsidence and to improve ventilation.

A total of fifty-eight man-shifts were worked at the face during each 24-hr period and a total of 2368 tons produced during a 5-day week. This gave a face O.M.S. of 8.2 tons.

The Loebbehobel can only be installed with its special driving head which is a part of the face conveyor. The Anbauhobel uses the same cutting head as the Loebbehobel but it uses a separate motor for driving the plow.

The Anbauhobel

The Anbauhobel is designed for installation on an existing conveyor system and consists of a plow unit complete with its own drive motors so that it is not necessary that the conveyor drive motors be used to power the plow.

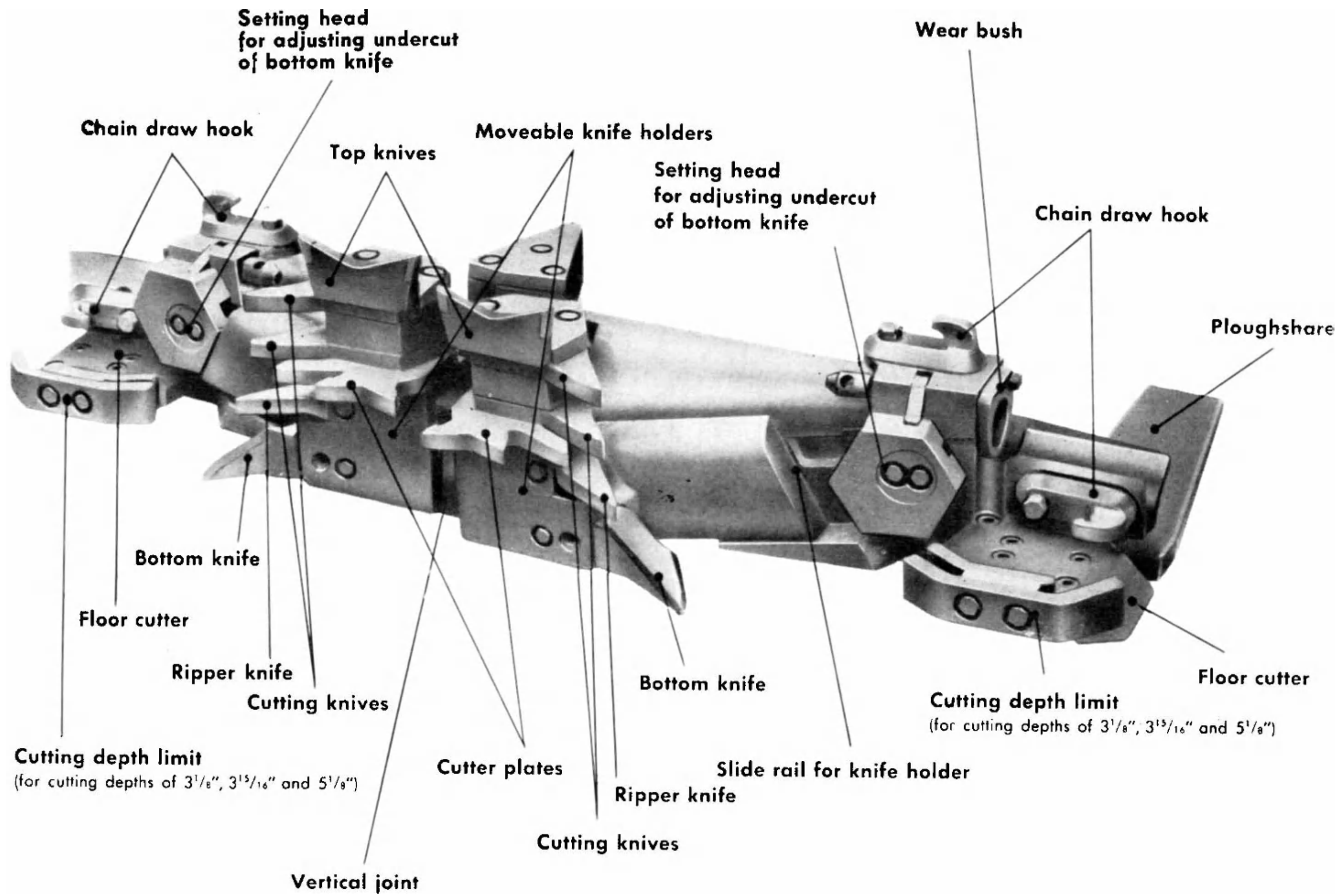


FIG. 10. Coal plow. (Maschinenfabrik und Eisengiesserei A. Beien)

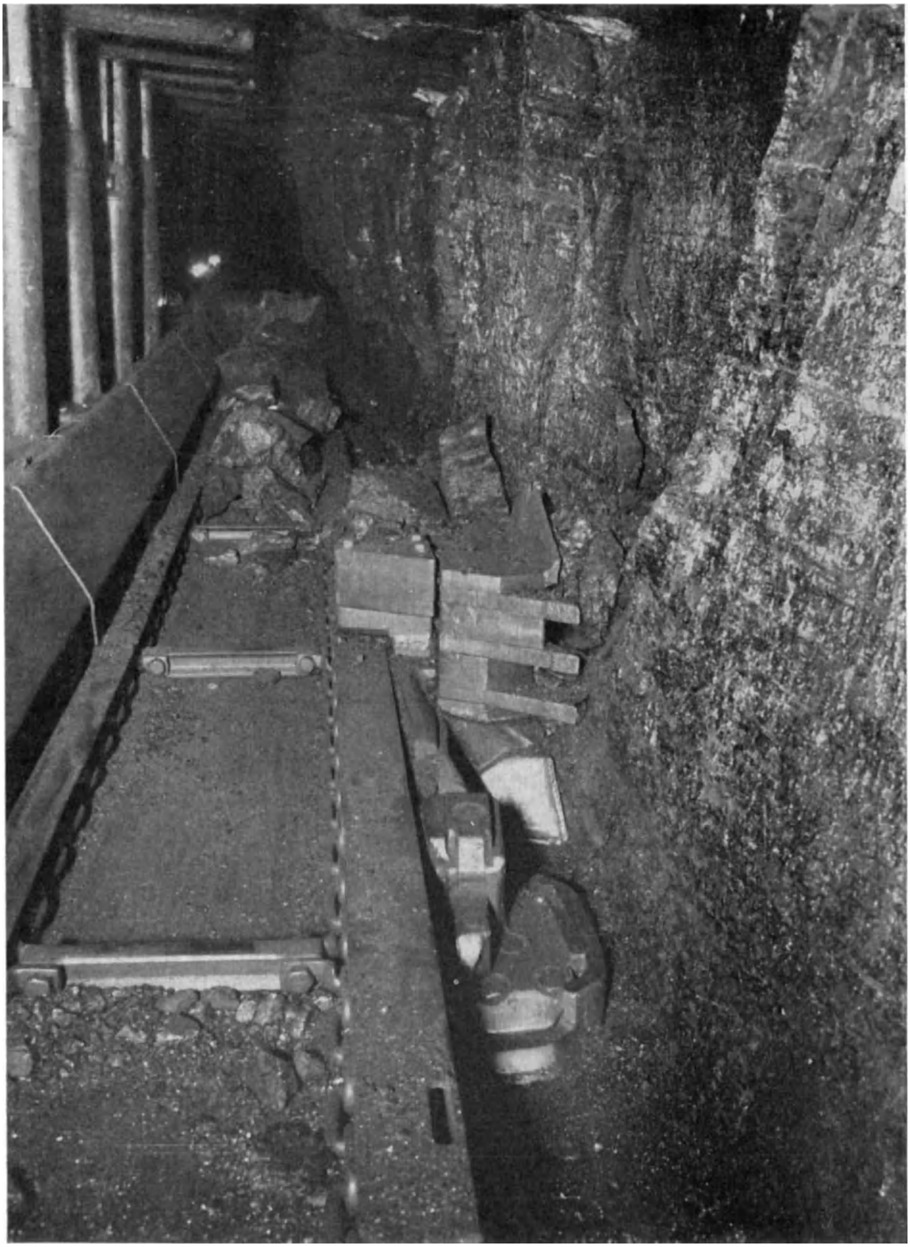


FIG. 11. A coal plow at the face. (Maschinenfabrik und Eisengiesserei A. Beien.)

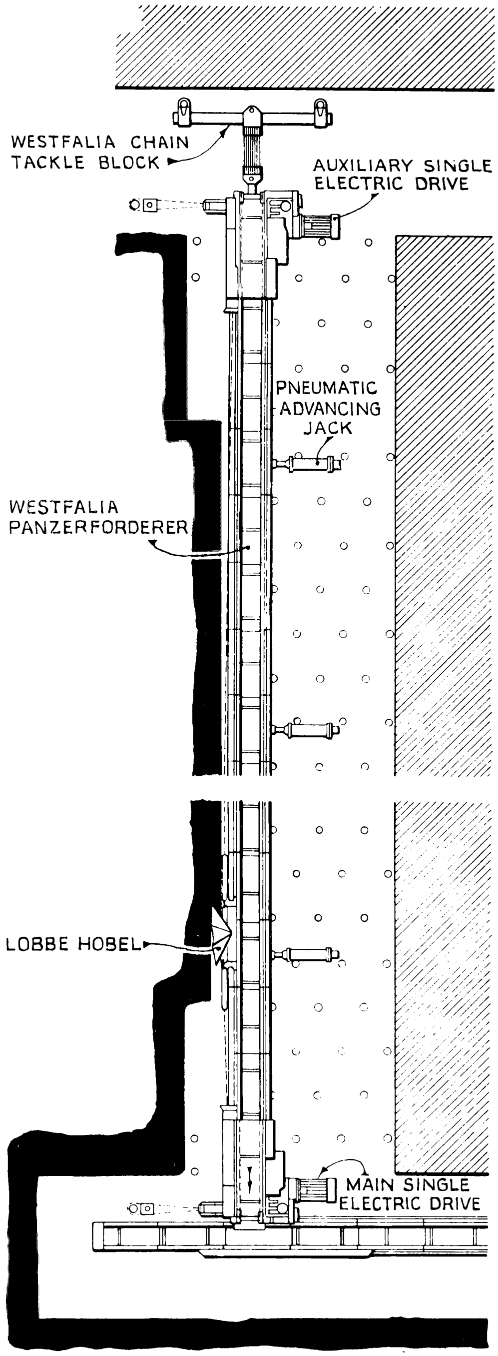


FIG. 12. Typical layout for a Loebbehobel face.[5]

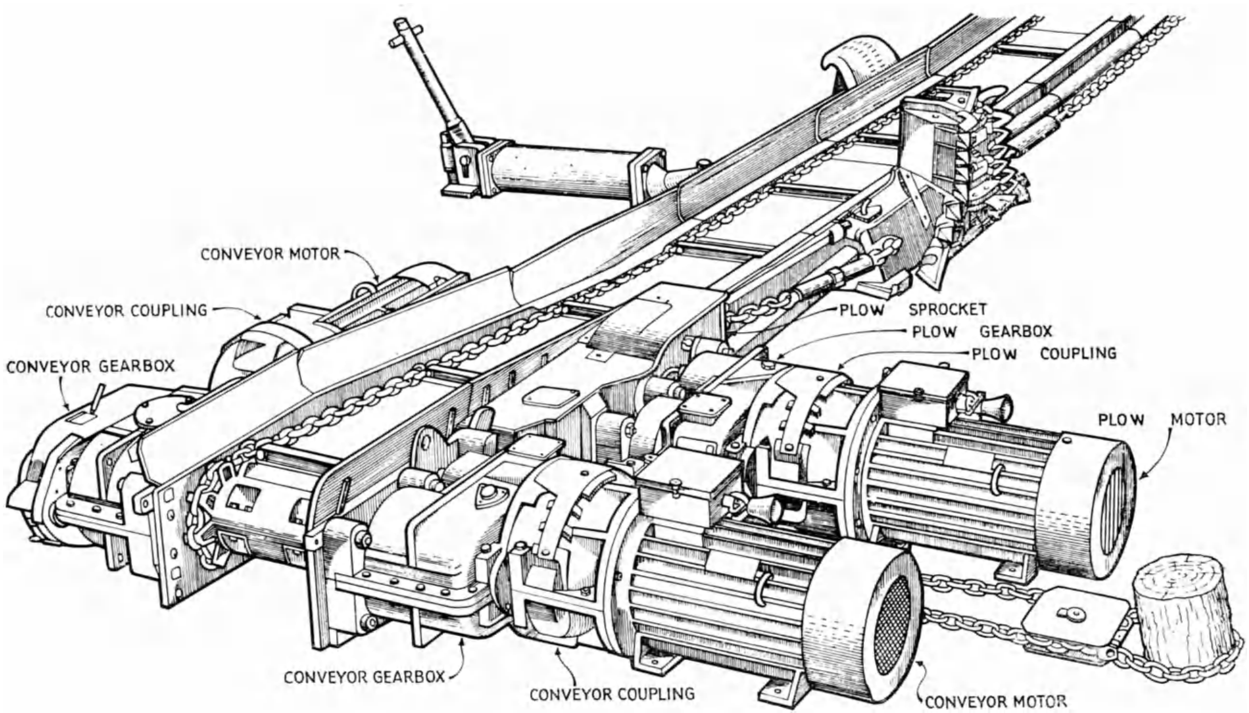


FIG. 13. Conversion plow drive unit.[6]

With the Loebbehobel the plow is pulled by one motor, that is the one toward which it is traveling. With the Anbauhobel it is possible because of the endless tow chain, to apply power from both motors to the plow at the same time.

It is possible to set up the independent plow drive at any intermediate point along the conveyor.

The Gusti Multiplow[7]

The Gusti multiplow is a type of rapid plow which was developed in Holland and was designed to work relatively thin seams under difficult conditions.

Small plow units, 3 ft in length, are spaced along the face 45–60 ft apart, and are connected by steel ropes. At each end of the conveyor the free ends of the rope are coupled to small-diameter drums driven from the conveyor motors. Plow units are pulled to and fro along guides which are integral with the conveyor.

In an experimental installation at Waterhouses Colliery Co., Durham, the multiplow face has produced a face O.M.S. of 3.8 tons in a seam 21 in. thick.

This type of plow has not found any extensive application in Great Britain, only three being in service in 1957.

ACTIVATED PLOWS

Various special plow-type machines have been designed which are equipped with pneumatic percussive picks or oscillating blades so that harder coals could be worked by plowing.

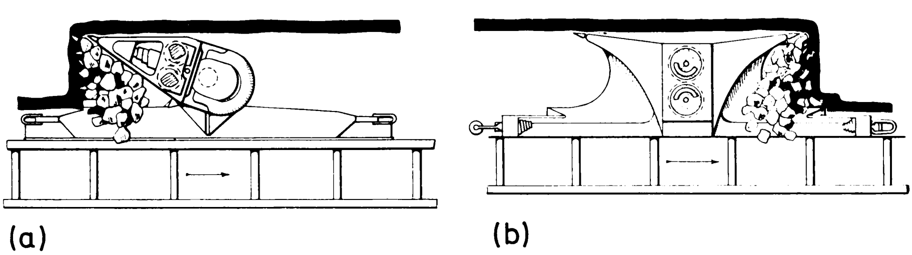


FIG. 14. Some early coal plows. (a) The single-ended Schnellhauer. (b) The Hannibal hewer.[7]

The earliest machines of this type were developed in Germany and included the Schnellhauer, the Hannibal hewer, and the Stobhauer.

The Schnellhauer depended upon two out-of-balance rotors which turned at relatively high speed and caused the main body of the plow to oscillate. These vibrations were transmitted to the cutting blade which was in contact with the coal and pressures of 100–200 tons were developed at the cutting edge.

The Hannibal hewer was a double-ended version of the Schnellhauer and gave a relatively good performance over a long period of time.

Both of these machines were eventually rejected, however, for the following reasons:[6]

(1) Studies indicated that 80 per cent of the energy was being dissipated in bearing and friction losses.

(2) The oscillations became surpressed as the cutting edges became embedded in the coal seam.

(3) Breakages of rotor bearings and bearing housings were excessive.

(4) Wear of plow and conveyor was excessive.

(5) Compressed air consumption was considered to be excessive.

Other activated plows were designed including two equipped with pneumatic picks which did not prove powerful enough to work the harder coals.

Attempts were also made by German engineers during the war years to develop self propelled plows, one of which would have been equipped with double crawler tracks one set of which would run on the floor while the other would bear against the roof. These designs were partially developed but never put into operation.

The Huwood Slicer

The Huwood slicer, developed in Great Britain, has been the most successful of the activated plows, and in 1959 some twenty-three were in use in British collieries.

This machine consists of three main units as follows:[7]

(1) The *main frame* which travels on the face side of the conveyor and carries the slicing blades. It travels on rollers.

(2) The *adjustable goaf frame* which travels on the goaf side of the conveyor.

(3) The *power unit* which joins the other two units and forms a bridge over the conveyor.

The machine is equipped with a 60 hp motor which, through a series of gears, drives ecentrics which impart a circular motion to a bearing bar which rotates at 340 rpm with a 2-in. displacement. (Reduced to $\frac{1}{2}$ in. in new models.)

To each end of the bearing bar is fixed a plate which is equipped with a series of cutting picks. These picks are given a chipping motion and as the machine advances along the face they cut a kerf at the back of the slice. The actual depth of the slice is limited to a maximum of 14 in. The broken coal is then guided by the plow-shaped wedgehead on to the armored conveyor.

Since both ends of the bearing bar hold cutter picks the machine can cut in both directions of travel along the face. Bottom coal is removed by blades fitted at the base of the wedgehead.

The machine is supported on the conveyor by four horizontal rollers and four vertical rollers are fitted to take the side thrust.

The haulage unit consists of a 25 h.p. motor mounted at the tail end of the conveyor and driving an endless chain. The return chain is carried in a special channel on the goaf side of the conveyor. Available speeds of haulage are 12.75, 17.25 and 21.25 ft/min.

The slicer can work satisfactorily in seams of $4\frac{1}{2}$ ft thickness and over although a machine is being designed which will work in seams down to 3 ft thick.

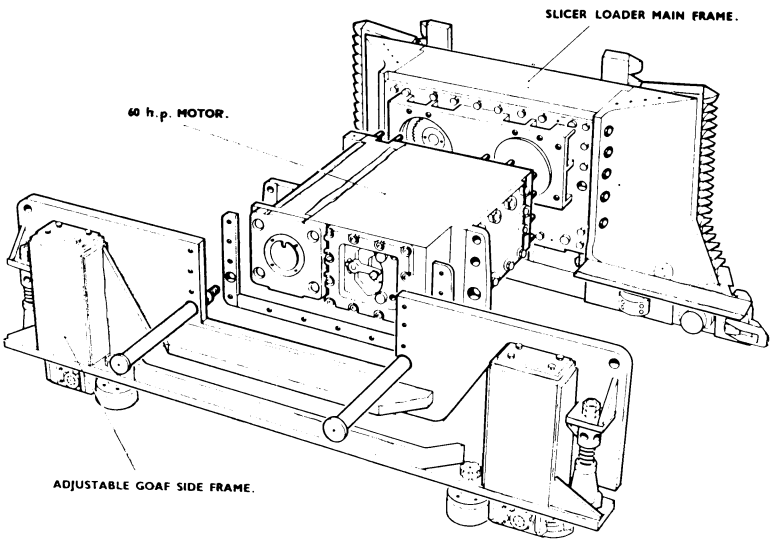


FIG. 15. Main components of the Huwood slicer-loader. (Hugh Wood & Co., Ltd.)

In hard coal pre-cutting of the face may be required.

A face length of 600 ft is considered to be satisfactory for operation of the slicer. Stable holes are required at each end of the face.

The Samson Stripper

The Samson stripper is a self propelled machine which was designed for continuous mining of long-wall faces. The machine is equipped with two bladed wedge heads which are mounted on opposite ends of a frame on which the propulsion unit is mounted.

The machine propels itself by pushing against a vertical hydraulic cylinder jack which extends to fix itself between roof and floor and form an anchor post against which the machine can push.

When the vertical jack is extended and in place, hydraulic cylinders push the cutting head forward in the coal for a distance of about 30 in., wedging off a web of coal about 24 in. thick. Maximum push available from the horizontal cylinders is about 45 tons. The machine requires very strong roof and floor so that the anchor jack can be adequately set. In 1959 there were only three of these machines at work in British collieries.

SCRAPER BOXES

The principle of the scraper box has been used for many years in metal mines for the transportation of ore along the face. The system involves the use of a number of steel boxes or scrapers which are connected in a series, and are pulled to and fro along the face so that the load deposited by each is picked up by the next box down the face and eventually is delivered to the loading gate.

By fitting the scraper boxes with knives or blades it is possible, under favourable conditions, to rip the coal from the face and transport it, step by step, to the gate. A very powerful winch is required for this duty and the cable must be capable of exerting pulls of up to 25 tons.

Scraper boxes are held against the face by a guide rail pushed by compressed air cylinders. Since no face conveyor is used the prop-free front needs to be only slightly more than 3 ft wide.

The scraper box itself consists of two side plates which are rigidly connected together at the top edge. The box is open at the bottom front and at the top. The back of the box is formed by a flap which is hinged at the top and opens towards the inside of the box only.

The length of travel usually adopted is the length of the spacing between boxes plus 15 ft. The capacity of the boxes varies from 19 to 35 ft^3.

There were ten scraper box installations operating in Britain in 1961 in seams from 12 in. to $3\frac{1}{2}$ ft thick.(7)

In 1954 the output from scraper box installations was about 150,000 tons at a face O.M.S. of a little over 4 tons.(7)

THE FLEXIBLE ARMORED CONVEYOR(6)

The face conveyor is an essential part of a coal plow installation and the failure to design a suitable conveyor was an important factor which retarded the application of coal plows.

The following are the essential requirements for a face conveyor:

(1) The structure must be such that it can form a guide for the coal plow and can withstand the heavy side pressure exerted by the plow.

(2) The joints must be flexible enough to allow the conveyor to be advanced in a snaking fashion while it is running.

(3) The conveyor must have a high capacity and be strongly constructed.

(4) The conveyor must be able to operate as a unit over a long length.

Since the war this type of conveyor has been developed into an efficient unit which satisfies all the above requirements and is universally used on plow faces.

FIG. 16. Armored conveyor with coal cutter mounted. (Maschinenfabrik und Eisengiesserei A. Beien.)

The conveyor may be driven by one, two, three, or four driving units. Each driving unit consists essentially of an electric motor, fluid coupling, and reduction gearing. If necessary compressed-air motors may be used.

Each motor has a squirrel-cage rotor, exerts high torque, is surface cooled, and normally ranges up to 55 hp. Its synchronous speed is 1500 rpm and through the reduction gearing conveyor speeds of approximately 100, 150, or 200 ft/min can usually be obtained. The traction-type fluid coupling allows the motor to start light, takes up the load gradually and equalizes the load on the driving motors.

The conveyor is actually constructed of troughed, steel place sections each 5 ft long and fabricated in one piece. This construction allows a lateral flexing of 2 ft in 25 ft and vertical flexing of 3 in. in 5 ft.

The conveyor chain is made of 18 mm in diameter 0.25 per cent carbon steel, electrically welded, heat treated, tempered, and calibrated for length. The connecting links join the lengths of chain and receive pins which can be removed while keeping the chain continuous.

Depending on site conditions and power supplied to the multi-drives, capacities of up to 300 tons/hr can be achieved and faces up to 400 yd long can be served.

The choice of devices for advancing the conveyor in modern plow installations lies between compressed-air pushers and hydraulic jacks. The selection of either type depends mainly on:

(a) Power supply. Where compressed air is available at a constant pressure of not less than 60 psi, pneumatic rams can be successfully employed, given reasonable seam conditions. Where no compressed air is available or the supply is inadequate, hydraulic rams must be used.

(b) If the coal is soft then the plow has to be held firmly against the face to obtain the maximum depth of cut and the hydraulic ram is better suited to these conditions. With hard coal a certain amount of flexibility of the conveyor is desirable in order to minimize wear of the equipment and to avoid excessive tractive pull on the plow. The pneumatic jack has definite advantages under these conditions.

In very thin seams the pneumatic ram is at a disadvantage when compared with the hydraulic type due to its size which may hamper movement of personnel in the face.

Where conditions are difficult due to a soft floor or adverse grade the hydraulic rams can develop greater thrust than the compressed air type.

(c) Generally hydraulic rams are unsuitable for high-speed coal plows due to their lack of elasticity. For slow plows and scraper boxes, however, a hydraulic ram has been designed which is provided with a cushioned movement.

Pneumatic rams may be single or double acting with thrusts varying from 0.7 to 1.4 tons at 60 psi. Compressed air for the rams is carried along the face, usually in armored hose, which is supported on the goaf side of the conveyor and advanced with it.

NEW TYPES OF CONTINUOUS MINING EQUIPMENT

The Mawco Cutter Loader

The Mawco cutter unit is designed so that it can be attached to any standard long-wall coal cutter which is equipped with a horizontal drive shaft for disk shearing.

As the cutter is drawn along the face the single cutting chain cuts a kerf completely around the perimeter of a strip of coal, freeing it from the solid at the top, bottom, and back, while at the same time a horizontal rotating breaker bar breaks the strip of coal and a plow attached behind the cutter deflects the broken coal onto the face conveyor.

The machine has shown an ability to produce a large proportion of plus 2-in. coal, and for that reason the number in use will probably increase markedly during the next few years.

The Dranyam Power Loader

The cutting unit in the Dranyam power loader is a drum, set with picks, which rotates about a vertical axis. The machine is mounted on an armored conveyor and as it is pulled along the face the rotating picks rip coal from the face and deflect it onto the face conveyor.

The principal point favoring this machine is that it makes its own stables, and therefore allows a reduction in the large proportion of the face manpower which is normally required for this operation.

The Dawson Miller Stable Hole Machine

The Dawson Miller stable hole machine is a new development and has been designed to mechanize the extraction of stable holes thereby reducing the face manpower requirements and increasing the rate of face advance.

The machine consists of a cutting unit which travels on a specially constructed rigid conveyor frame.

The cutting unit consists of a motor and a gearbox which drives a rotating cutting disk the diameter of which determines the height of the stable hole. This unit travels from end to end of the rigid conveyor frame which extends for the full length of the stable.

The machine is essentially a very narrow web Anderton shearer, which cuts a $\frac{1}{2}$-in. web each time it traverses the face. The cut coal is conveyed on the rigid frame conveyor and discharged onto an auxiliary conveyor which conveys it to the main system.

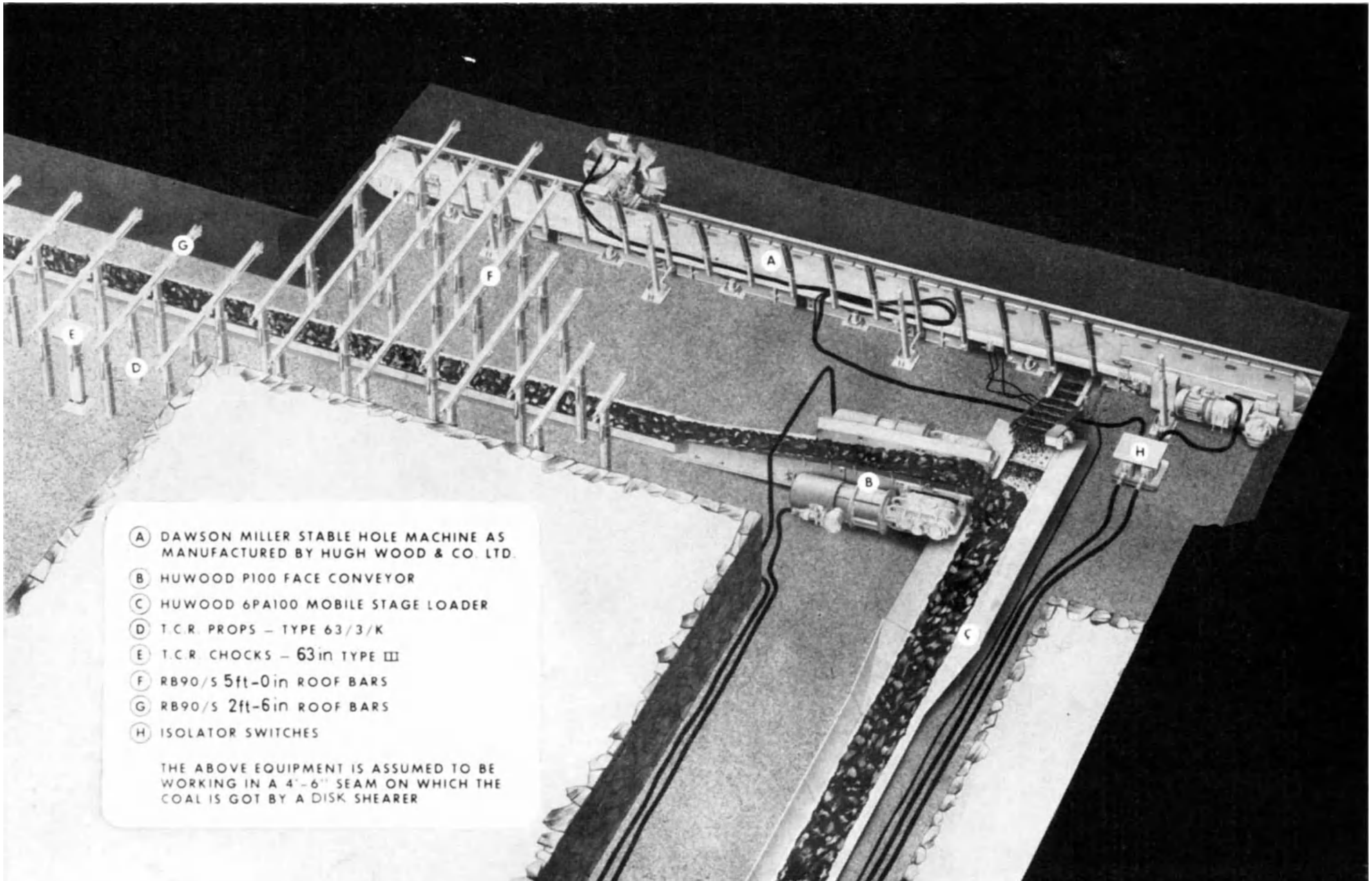


FIG. 17. Typical Dawson Miller stable hole machine installation. (Hugh Wood & Co., Ltd.)

Power Units

The conveying and traversing mechanism is driven by a $7\frac{1}{2}$ h.p. motor and gearbox arrangement. The $7\frac{1}{2}$ h.p. motor also drives a hydraulic power pack which is built into the drive head section of the conveyor and which supplies oil for push rams, for lifting jacks for horizon control, for chain tensioning rams, and to drive the auxiliary conveyor.

A four-speed gearbox fitted to the drive unit allows a choice of cutting speeds between 16 and 41 ft/min depending upon the hardness of the coal.

The cutting unit is driven by a 15 h.p. electric motor and speed-reducing gearbox.

Cutting Unit

The rotating cutting disk equipped with six arms, each holding a cutter pick, is mounted on the output shaft of the speed-reducing gearbox which is driven by a 15 h.p. motor. The whole assembly is mounted on a carriage which is traversed continuously and automatically from end to end of the rigid frame conveyor along guide rails.

Traverse Mechanism

This mechanism automatically traverses the cutting unit from end to end of the frame and automatically reverses its direction at the end of each traverse.

Advance Mechanism

The conveyor frame is automatically advanced by means of hydraulic pushing rams attached to it. The amount of advance or depth of web is controlled by toe plates at floor level, one at each end of the frame, which are maintained in contact with the stable face by the action of the pushing rams. As the cutting disk cuts by the toe plates, the resistance offered to them is removed. This allows the pushing rams to automatically advance the conveyor frame until the toe plates again contact the face, thus sumping in the cutting disk for a further web to be taken.

Conveying System

The rotating disk loads the cut material onto the deck plate of the conveyor. A 14 mm round-link chain with cantilever flights then conveys the material to a discharge port in the deck plate of the conveyor through which it passes onto the auxiliary conveyor. This conveyor is designed to allow the stable hole machine to advance $4\frac{1}{2}$ ft without requiring movement of the face conveyor or stage loader.

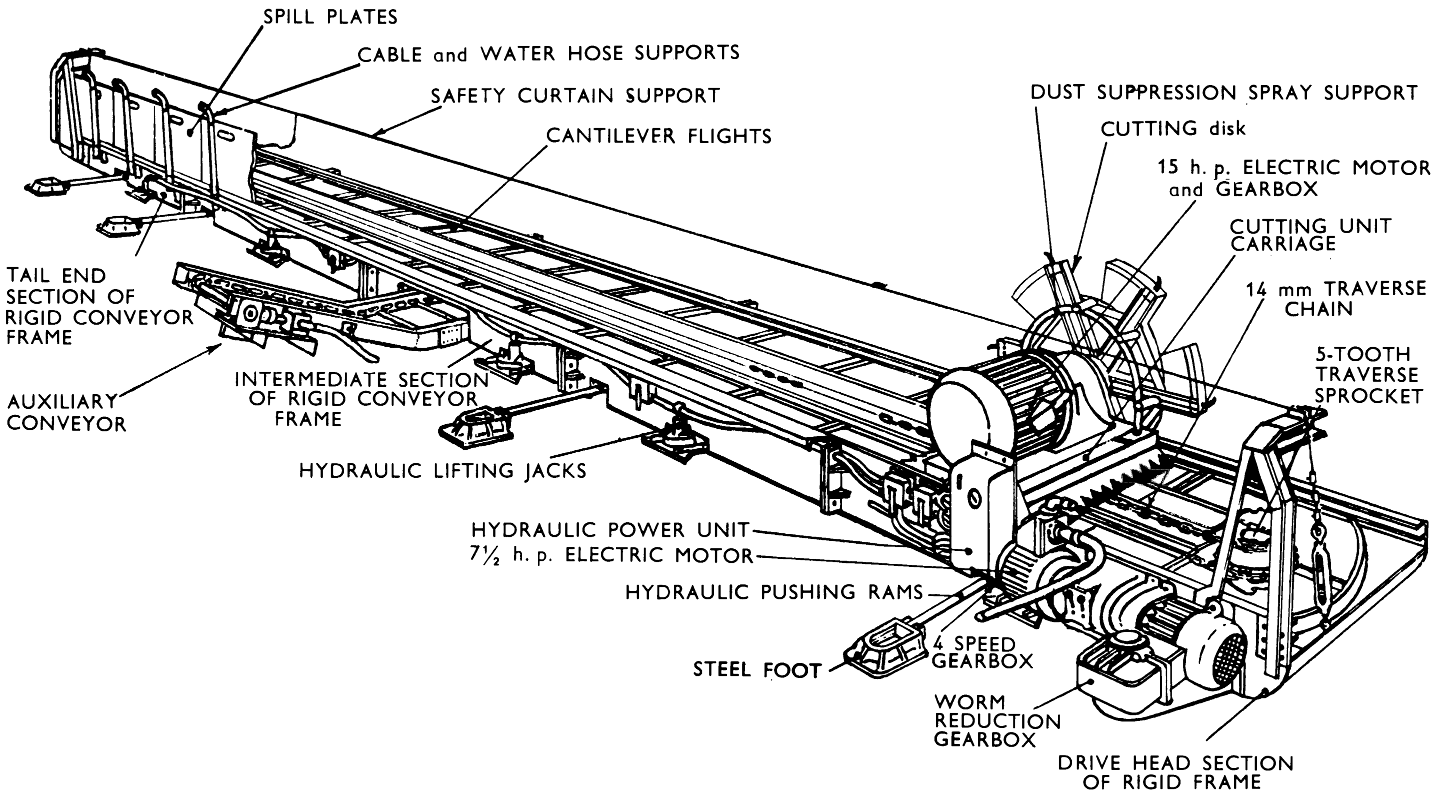


FIG. 18. Dawson Miller stable hole machine (right-hand) (National Coal Board Central Engineering Establishment.)

Joy–Sullivan Road Ripping Machine

The Joy–Sullivan road ripping machine has the following characteristics:[9]

(a) It can be mounted and operated in a standard roadway producing an excavation 13 ft wide by 10 ft high. (b) It incorporates provision for advancing the machine. (c) The cutting drums are designed for sumping as well as shear cutting. (d) Dust suppression is catered for by the use of integral water sprays; (e) Debris disposal is arranged for by scraper packing or by transference through the centre of the machine by conveyor.

The machine consists of:

(a) A skid mounted base, moved and positioned by wire ropes actuated by hydraulic cylinders. (b) A cutter head frame or carriage mounted on the base advanced or retracted hydraulically in relation to it, over a range of 2 ft 3 in. (c) A large diameter tube supported in the cutter head frame, which provides the pivot for the cutter arm and the mounting for the cutter drive. (d) A transmission drive consisting of a 60 h.p. electric motor, fluid coupling, reduction and drive shafts to the three cutter drums. (e) A cutter arm mounting the three cutter drums which is positively controlled in the vertical plane by hydraulic cylinders. (f) Three cutter drums each fitted with tungsten carbide insert picks on the periphery and face, and staggered in relation to each other to assist in the clearance of cuttings. Integral water sprays provide a jet of water at each cutter pick point. (g) A twin jack arrangement mounted at the top of the cutting head frame, giving a positive loading between the roof arch girders and the machine, to give complete stability to the machine when cutting and also to assist in the setting of the roof girders. (h) An hydraulic pump driven from the main motor and operating the hydraulic cylinders.

Operation

(1) The machine is positioned with the centre line of the drive on the centre line of the roadway, the cutter drums touching the face of the rip with the cutter head retracted on the base, and the cutter arm in the horizontal position to cut at the bottom of the rip on the operator's side.

(2) The roof jacks are raised against the roof to stabilize the machine.

(3) The machine is sumped into the face up to 12 in. by advancing the cutter head frame using the carriage jacks. An indicator is mounted on the base to show the depth of the sumping.

(4) The cutter arm is turned in the vertical plane through 180°, cutting the whole face, using the cutter arm jacks.

(5) The arm is returned to its initial position.

(6) Operations 3, 4 and 5 are repeated to take a further 12 in. cut.

(7) Chains are fastened to the front of the extension rods of the carriage jacks from anchor points under the lip, the roof jacks lowered and the machine base advanced forward.

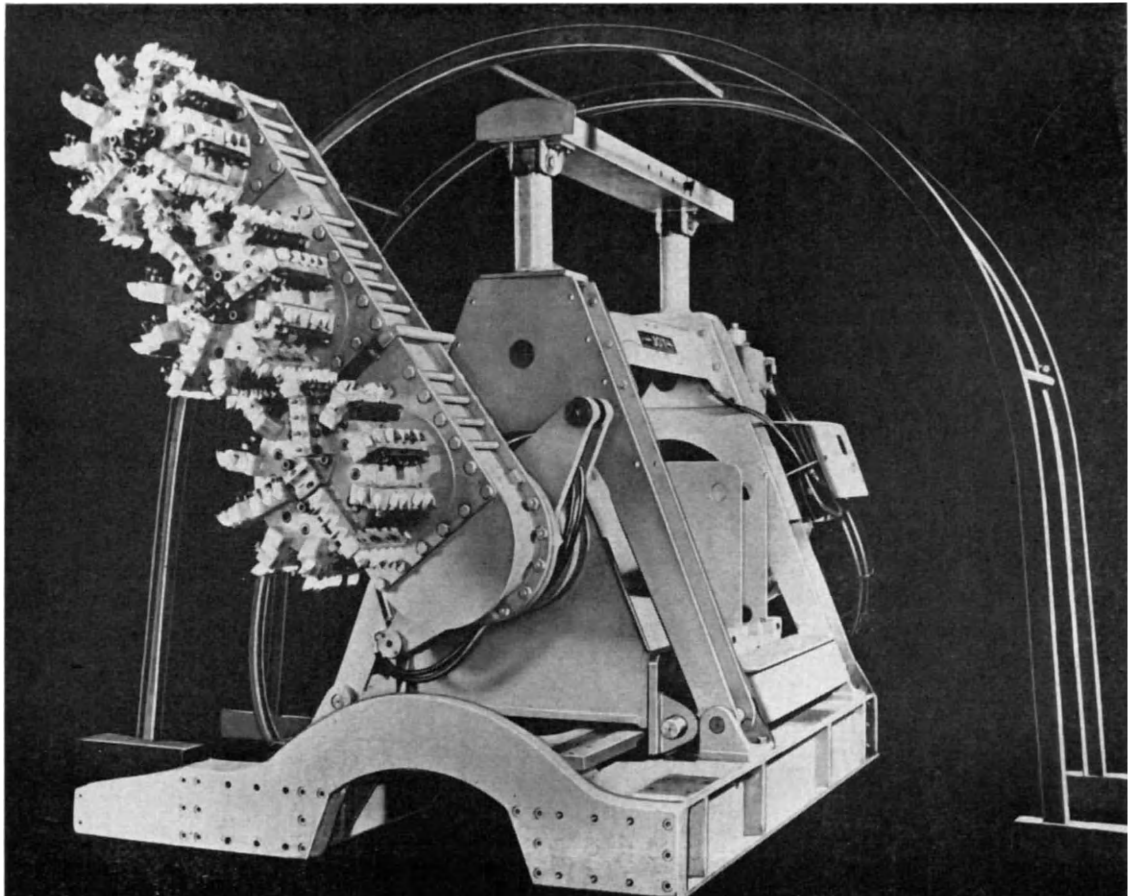


FIG. 19. Joy–Sullivan road ripping machine. (Joy–Sullivan Ltd.)

The approximate times for the above operations are:
Sumping in 12 in. — $\frac{1}{2}$ min; cutting — 2–3 min; returning cutter arm — 1 min; advancing base after fixing chains — 1 min.

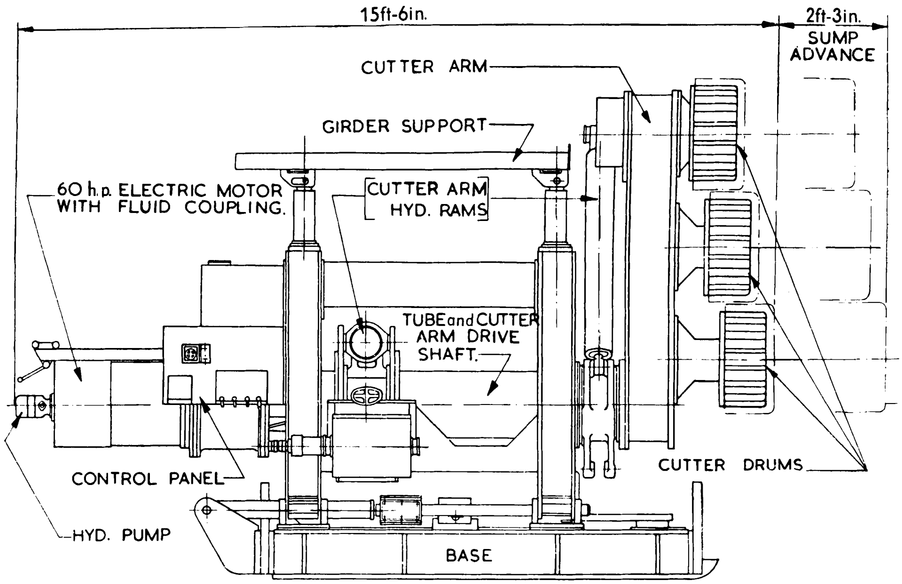


FIG. 20. Joy–Sullivan road ripping machine (side view). (Joy–Sullivan Ltd.)

Roadway Excavation Sizes

The machine is made suitable for three sizes of roadway excavation. In addition packing pieces can be used to increase the cutting height up to a further 12 in. The following list shows the various excavation sizes and the minimum height required under the ripping lip.

Model	RR.227	RR.245	RR.246
	ft in.	ft in.	ft in.
Width of excavation	13	15	17 6
Width of finished roadway	12	14	16 6
Height of excavation			
standard	10	11	12 3
with 6 in. packing	10 6	11 6	12 9
with 12 in. packing	11	12	13 3

Minimum heights of ripping lip are as follows: No packing—2 ft 6 in.; with 6 in. packing—3 ft; with 12 in. packing—3 ft 6 in.

Stone Disposal

In a 3 ft 6 in. seam a 12 in. cut gives 5 tons of stone. In a 2 ft 6 in. seam the tonnage is 6 tons. This is produced in 2–3 min plus $\frac{1}{2}$ min for sumping, so that the rate of production of stone may be at 2 and even more tons per minute. In considering methods of stone disposal it is necessary to provide a system which can operate

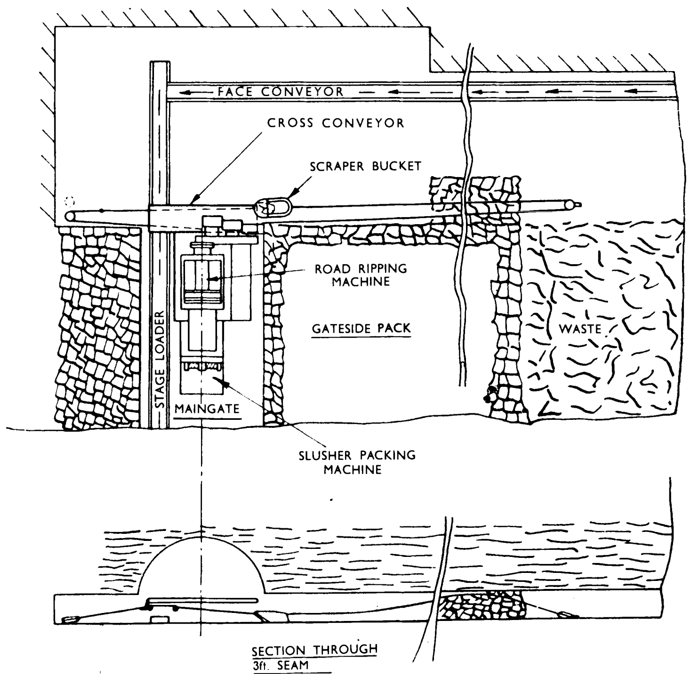


FIG. 21. Slusher packing of stone from a road ripping machine.[9]

at these rates or provide bunkerage capacity, or the cutting will have to be intermittent. It is undesirable to use the machine cutting at slower speeds because this results in increased dust production.

Four methods of stone disposal have been tried:

(a) Loading on to a center conveyor passing through the machine. This is capable of dealing with maximum rates of cutting. Chutes from the sides of the roadway to the centre conveyor eliminate all handfilling. Alternatively, a conveyor along the side of the roadway may be used. A cross conveyor can be fixed to the machine immediately below the cutting arm to give transverse loading of the stone. This cross conveyor is driven hydraulically from a power take-off from the machine.

(b) Slusher packing. This system is well known, there being about 700 units in operation. It provides a very effective means of stone disposal with the ripping machine. The rate of loading is dependent on the length of run of the packing bucket. On average the rate of packing is about 1 ton per min, but in practice the floor immediately below the cutting arm forms a bunker.

The team required is two for a 5 or 6 ft advance per shift and three for a 10 or even 15 ft advance per shift. In the first case one man is positioned on the face at the pack-hole and one man operates the two machines. About one-third of the cut is taken and then the stone cut is slusher packed. The two operations are repeated with the second and third sections of the cut. With three men operating, one man again is on the face, one man on the ripping machine, and one man on the slusher. Cutting and packing is continuous and slushing is continued whilst the cutting arm is returned to its original position and until all the stone is packed.

(c) Gob flinging. A standard long-wall coal cutter fitted with a gum-stower head was sited alongside the ripping machine. The stone was delivered to it by a cross front conveyor and then flung into the pack hole. In practice the results were not successful, mainly due to the lumps produced causing blocking, and wet dirt sticking to the gum-stower. The speed of stowing when operating was under $\frac{1}{2}$ ton/min, which necessitated a very slow cutting arm speed of 10–12 min. A 5 ft advance was obtained in $5\frac{3}{4}$ hr with four men.

(d) Pneumatic stowing. In this system the stone was again loaded by cross-conveyor onto a conveyor alongside the machine and then to a Markham crusher stower. The stowing pipe was carried through the machine to a 90° bend for stowing in the pack-hole. The results showed that a 5 ft advance may be obtained with three men per shift and a 10 ft advance with four men per shift. The main disadvantage experienced with the apparatus used was that the rate of stone disposal does not keep up with the rate of cutting. Again a cutting arm speed of 10 min was necessary giving a maximum loading rate of about $\frac{1}{2}$ ton/min. Either some form of bunkering is required or a greatly increased stowing capacity.

From the various time studies taken, rates of advance of 30 ft per shift are possible in a solid heading where the stone is loaded away from the machine by conveyor, assuming the support setting can be accommodated in 20 min/yd. On long-wall faces slusher packing can give an advance of up to 15 ft/shift.

In seams greater than 4 ft thick, slusher packing of roadside packs is not normally practiced in that the length of pack is not sufficiently long to give efficient operation. It is probable in these seams that pneumatic stowing will prove the best method of stone disposal. As explained the standard crusher-stower has a limited capacity and a larger unit of at least $1–1\frac{1}{2}$ tons/min is advisable, incorporating a crusher unit to break down any oversize obtained.

Sequence of Operations

The following is a sequence of operations for a 5-ft advance using scraper packing and is repeated for a 10-ft advance. Normally two men are required for a 5-ft advance per shift and three men for a 10-ft advance per shift.

(1) Set machine chain staker-props, return sheaves, pull forward ripping machine base, then set slide bars and props in position. (2) Cut 1 ft and scraper pack. (3) Move ripping machine base forward 1 ft. (4) Move slide bars 1 ft. (5) Cut 1 ft and scraper pack. (6) Move slide bars forward 1 ft. (7) Cut 1 ft and scraper pack. (8) Set arch girder in gate. (9) Move ripping machine base forward 2 ft. (10) Reset slide bars and props. (11) Reset left hand return sheave (if necessary). (12) Cut 1 ft and scraper pack. (13) Move slide bars forward 1 ft. (14) Cut 1 ft scraper pack and clean up. (15) Set temporary arch in gate. (16) Slide bars and props are reset to face on coal-filling shift.

The following shows a time study of a typical operation.

BREAKDOWN OPERATIONS

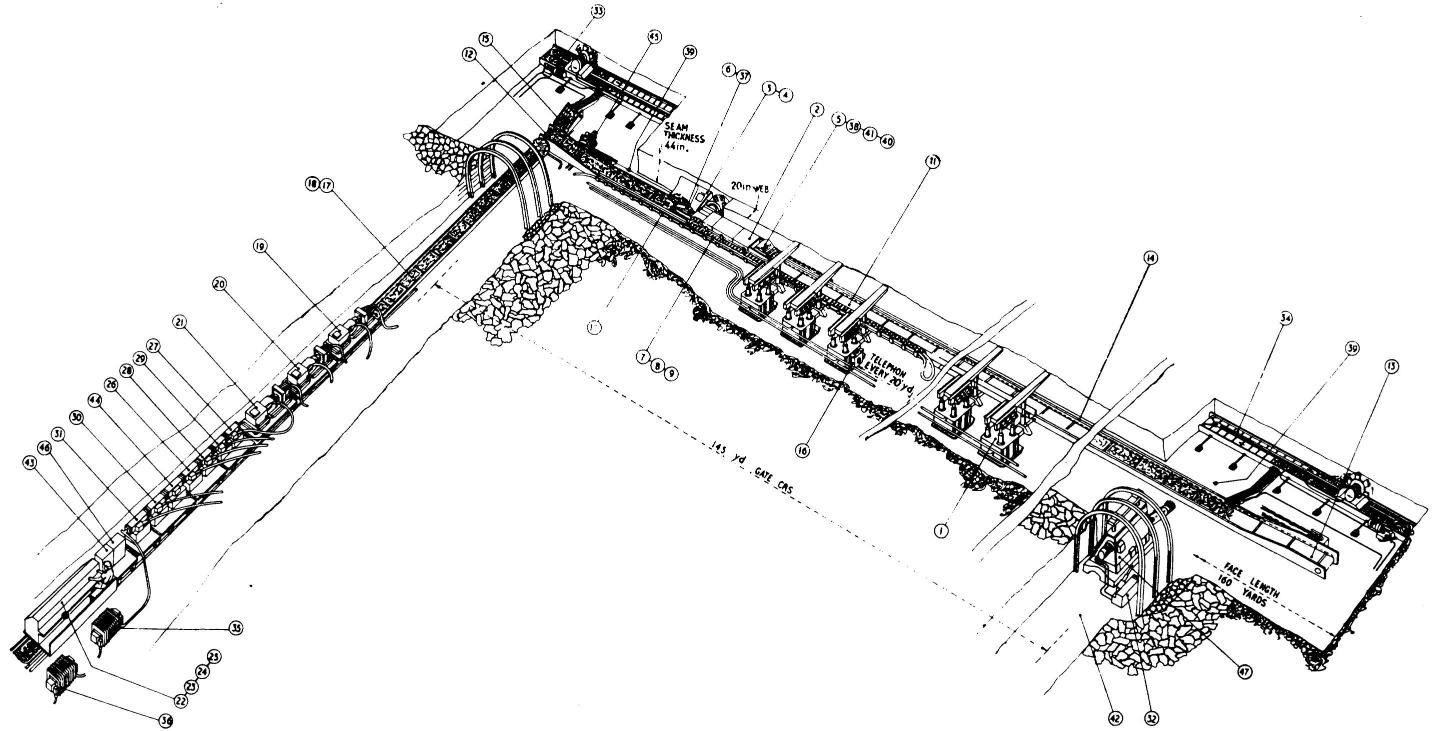
10-ft. advance – 2 packs of 5 ft. – 3 men.

			Time (min)
1	Set staker props, prepare pack hole, thread ropes for scraper packing		25
2	Prepare lip and set lip supports		20
3	Advance base	2 ft	4
4	Rip and scraper pack	1 ft	10
5	Advance base	1 ft	4
6	Move lip supports	2 ft	4
7	Rip and scraper pack	1 ft	10
8	Rip and scraper pack	1 ft	10
9	Advance base	2 ft	4
10	Move lip supports	2 ft	4
11	Rip and scraper pack	1 ft	10
12	Rip and scraper pack	1 ft	10
13	Draw rope from pack, clean-up and finish packs		10
	Total for 5-ft advance		125

Total for 10-ft advance	250 min
Setting girders – 3 at 25 min each	75 min
Approx. total time for 10-ft advance	$5\frac{1}{2}$ hr

Supports

The roof jacks with the girder support can carry the center section of a three piece arch, to support the roof between the face of the rip and the last permanent setting. When a new arch requires to be set, the centre section is then already positioned for bolting of the side legs. Alternatively, two piece arches may be used and the extension piece can carry the forward temporary support. With the profiled section of roadway cut, setting of the arches is a simple and rapid operation.



Item	Equipment	Supplier
1	Seaman Gullick powered supports with remote control	Gullick Ltd.
2	A.B. 125 hp A.S.I. (without precutting jib) 20 in web drum dia. 42	Area No. 4 E.M.
3	Plough	C.E.E.
4	Source and detector unit for vertical steering	C.E.E.
5	Hydraulic control for vertical steering	C.E.E.
6	Ranging underframe and ram for vertical steering	C.E.E.
7	Power cable	Area No. 4 E.M.
8	Water hose	Area No. 4 E.M.
9	Pilot cable	C.E.E.
10	Mk. II cable handling equipment (A.S.L. with chain haulage troughs and carrier chain)	Gullick or C.E.E.
11	Mk. III N.C.B. standard conveyor pans chains flights drive and tail end ramp pans	Area No. 4 E.M.
12	N.C.B. standard conveyor drive head frame with sprocket assy.	Area No. 4 E.M.
13	N.C.B. standard conveyor tail head frame with sprocket assy.	Area No. 4 E.M.

Item	Equipment	Supplier
14	Loading ramp pans	C.E.E.
15	Hydraulic Conveyor drive	C.E.E.
16	M.R.E. loudspeaking telephone	Gullick Ltd.
17	Stage loader	Area No. 4 E.M.
18	Gate conveyor	Area No. 4 E.M.
19	Hydraulic power pack face supports	Gullick Ltd.
20	Hydraulic power pack face supports	Gullick Ltd.
21	Hydraulic power pack A.F.C. drive	C.E.E.
22	Control console	Gullick Ltd.
23	Control box for remote control of chocks	Gullick Ltd.
24	Vertical steering control	C.E.E.
25	Remote control of hydraulic conveyor	C.E.E.
26	Gate end box (machine supply)	Area No. 4 E.M.
27	Gate end box (hydraulic power pack A.F.C. drive)	Area No. 4 E.M.
28	Gate end box (hydraulic power pack face supports)	Area No. 4 E.M.
29	Gate end box (hydraulic power pack face supports)	Area No. 4 E.M.
30	Gate end box (stage loader drive)	Area No. 4 E.M.
31	Gate end box (for stable hole equipment) loader gate	Area No. 4 E.M.

Item	Equipment	Supplier
32	Gate end box (for stable hole equipment) tail gate	Area No. 4 E.M.
33	Stable hole equipment (loader gate) Dawson Miller or Joy	Area No. 4 E.M.
34	Stable hole equipment (tail gate) Dawson Miller or Joy	Area No. 4 E.M.
35	Transformer (air cooled) 3.3 Kv to 550 V	Area No. 4 E.M.
36	Transformer 110 V	Area No. 4 E.M.
37	Obstruction detector leading on cutting stroke	C.E.E.
38	Obstruction detector leading on flitting stroke	C.E.E.
39	Face end stops	C.E.E.
40	Machine position indicator haulage end	M.R.E.
41	Methanometer (machine mounted)	M.R.E.
42	Methanometer (tail gate)	M.R.E.
43	A.C. supply unit	C.E.E.
44	Section switch	Area No. 4 E.M.
45	Independent hydraulic haulage	C.E.E.
46	D.C. supply unit	C.E.E.
47	Ripping machine	

FIG. 22. The ideal remotely operated long-wall face.(10)

The timbering system for support of the roof under the ripping lip, when stone disposal is by scraper packing, may use slide bars and cabbage heads on hydraulic props, bull-rails being added if required. When another form of stone disposal is used, straight bars or girders across the roadway supporting the lip are preferable.

REMOTELY OPERATED LONG-WALL FACES

Remotely operated long-wall faces have been developed in Great Britain and two of these faces were in operation in 1963.[(10)] All operations of the face machinery are controlled by an operator who sits at a console 60 yd away in an adjacent roadway.

The mining equipment goes through the sequence of events involved in the continuous mechanical extraction of coal, apparently by itself.

One installation, R.O.L.F. 1, is in the 3 ft 7 in. thick High Main seam at Newstead Colliery, No. 4 Area, N.C.B. East Midlands Division; and the other is in the 4 ft Piper seam at Ormonde Colliery, No. 5 Area of the same division.

The two installations have differences in manufacturing and design detail, but they are identical in concept. Each contains the elements essential to remote controlled or automatic operation which are as follows:

(a) A power loader which can steer itself. Vertical steering of the Anderson-Boyes 125 chain-hauled shearer is provided by application of a nucleonic coal sensing device. Horizontal steering and the advance of the face in a straight line depends upon periodic surveying and the moving over of the ram extensions of the powered roof supports by pre-set amounts.

(b) A cable-handling device.

(c) Means of removing fine coal from the face side of the conveyor. This would otherwise prevent the conveyor from being advanced.

(d) Remotely controlled powered roof supports. Standard units are employed in these installations.

(e) Face communications, facilities, and the ability to stop and lock out machinery in an emergency.

(f) The necessary instrumentation and monitoring system to control the sequence of operation of the face machinery.

The general layout for an ideal remotely operated long-wall face is shown in Fig. 22.

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CHAPTER 6

ECONOMICS OF COAL FACE MECHANIZATION

COAL has long been the world's basic industrial fuel and its principal source of energy. Within the past few years, however, coal has been meeting with increasing competition from natural gas and petroleum products, and the annual U.S. production of bituminous coal and lignite for the past several years has held fairly steady in the neighborhood of 400 million tons.

Table 1 shows coal production in the U.S., by methods of mining, for the years 1958 through 1961.

TABLE 1. U.S. BITUMINOUS AND LIGNITE PRODUCTION, BY METHODS OF MINING AND CLEANING, 1958–1961.

Method of mining and loading	Thousand net tons 1958	1959	1960	1961*	Percentage of total 1958	1959	1960	1961*
Underground:								
continuous mining	56,373	65,792	77,928	86,000	13.7	16.0	18.8	21.5
conventional mining:								
mechanically loaded	187,200	177,939	167,858	151,000	45.6	43.2	40.4	37.8
hand loaded	43,311	39,703	39,102	37,000	10.6	9.6	9.4	9.2
total, conventional	230,511	217,642	206,960	188,000	56.2	52.8	49.8	47.0
total, underground	286,884	283,434	284,888	274,000	69.9	68.8	68.6	68.5
Surface:								
strip mining	116,242	120,953	122,630	118,000	28.3	29.4	29.5	29.5
auger mining	7,320	7,641	7,994	8,000	1.8	1.8	1.9	2.0
total, surface	123,562	128,594	130,624	126,000	30.1	31.2	31.4	31.5
total, all mines	410,446	412,028	415,512	400,000	100.0	100.0	100.0	100.0
mechanically cleaned	259,035	269,787	273,169	266,000	63.1	65.5	65.7	66.5

* Preliminary.

On the basis of productivity in the United States the average labor cost of a ton of coal is around 45 per cent of the mine price.[2] Mines with the latest equipment and methods, and average conditions, may achieve outputs of 15–20 tons/man/day and thus may reduce labor costs to about 25–35 per cent of the average value of a ton of coal, which is about $5 per ton at the mine.

As further wage increases are granted, more mines, in order to stay in business, are forced to seek more efficient operating methods. The advent of roof bolting subsequent to 1947 was a step forward in reducing support costs. Roof bolting operations have been mechanized to the point that one man can drill a hole and install a bolt in approximately 2 min.

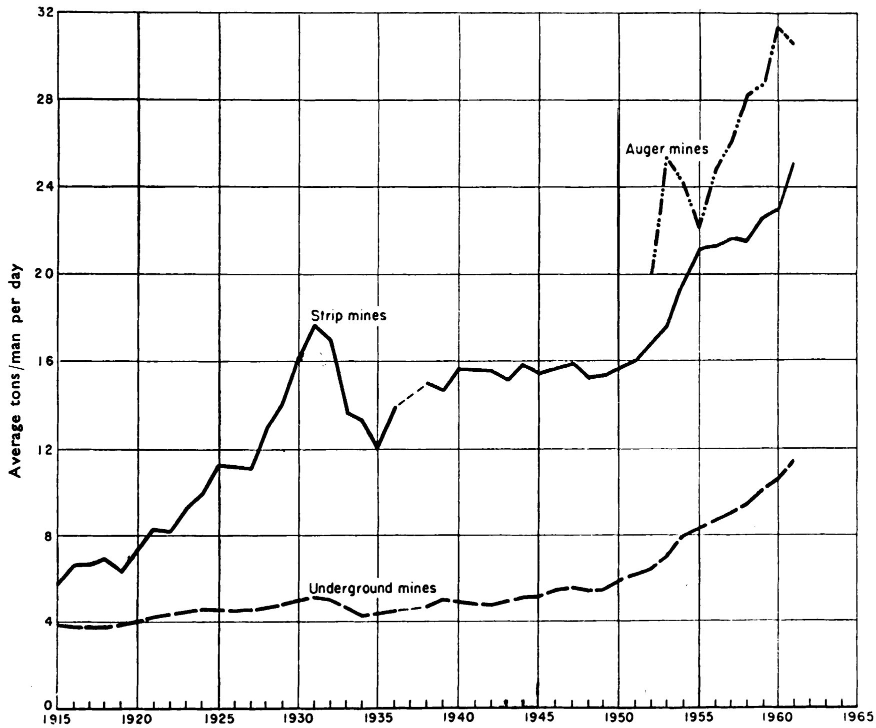

FIG. 1. Average tons per man per day at bituminous coal and lignite mines in the United States, 1915–1961, by underground, strip, and auger mines.(12)

The principal efforts at increasing efficiency and reducing costs have been centered in the development of continuous-mining machines which eliminate the cutting, drilling, and blasting cycle. In order to support these machines with their high production capacities, great improvements in transportation devices, such as shuttle cars and conveyors, have been made.

Underground productivity in terms of "tons per man-shift" has been improved year by year. In 1937, for example, the average rate of productivity was 4.50 tons per man-shift. By 1956 the overall average production of coal and lignite by all underground methods was 8.62 tons/man-shift. This figure has risen to over 11 tons per man-shift by 1961. (The tons per man-shift are calculated on the average

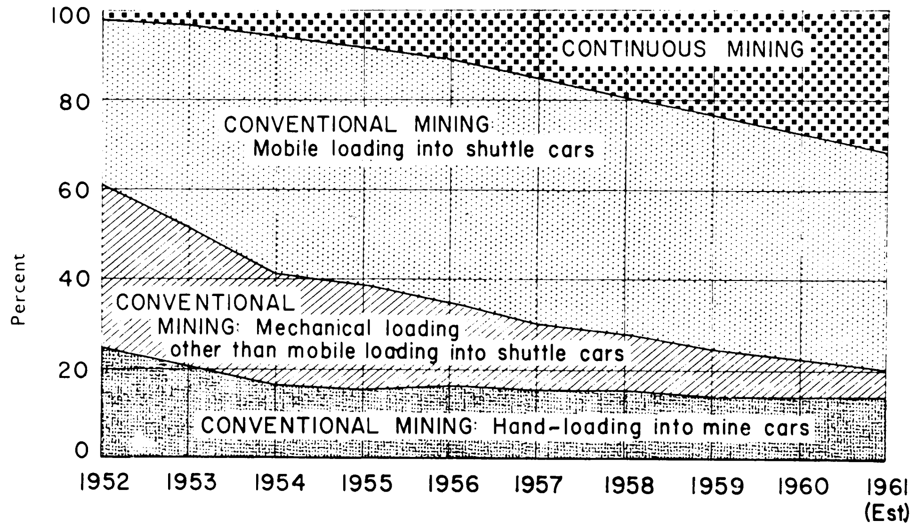

FIG. 2. Percentage of total underground production of bituminous coal and lignite mines in the United St ates by method of mining and loading, 1952–1961.[3]

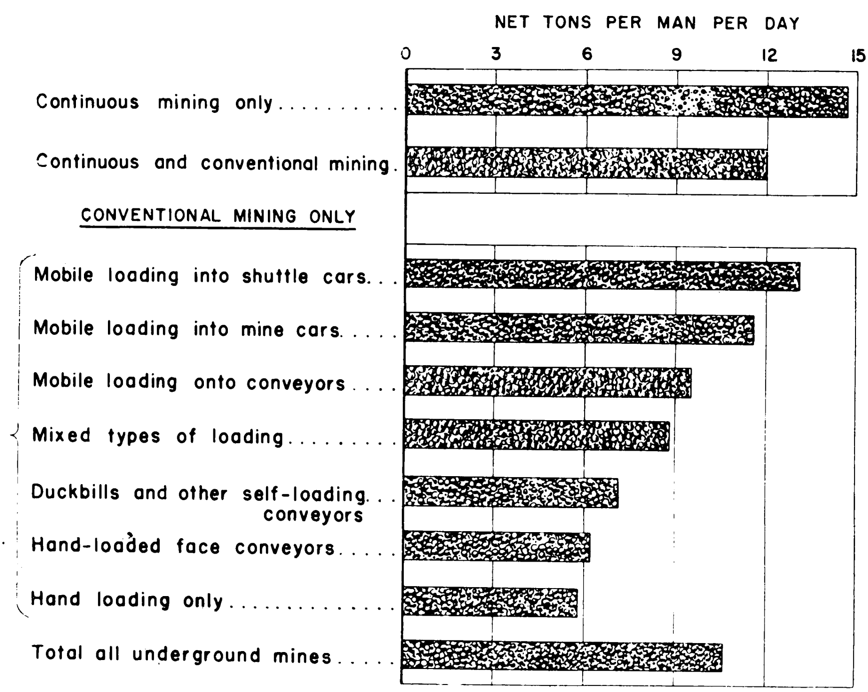

FIG. 3. Output per man-day at underground bituminous coal and lignite mines in the United States by method of mining and loading, in 1960.[3]

number of men working daily, including all men engaged in the production and preparation of coal.)

Figures 2 and 3, and Table 1 summarize coal production by methods.

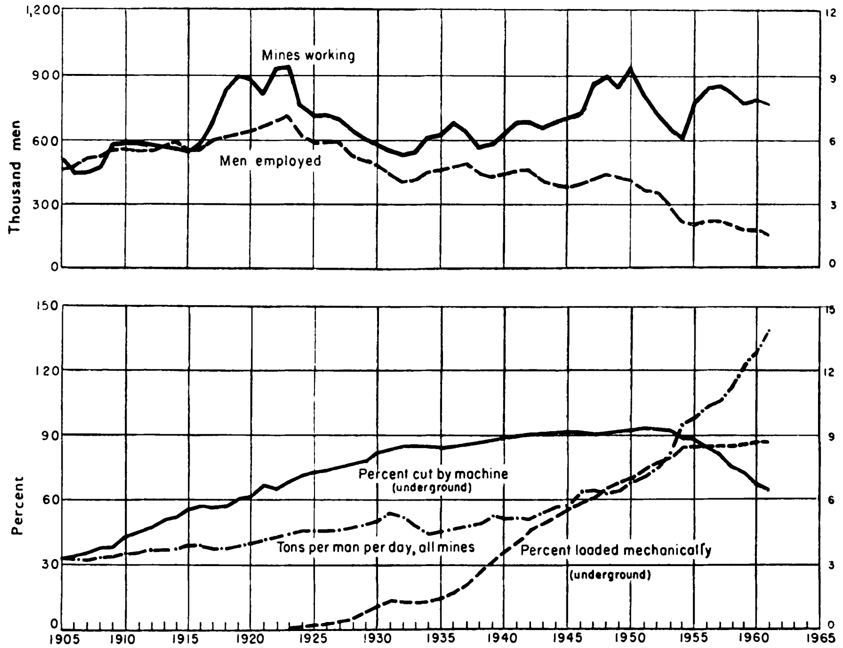

FIG. 4. Trends of employment, mechanization, and output per man at bituminous coal and lignite mines in the United States, 1905–1961.[12]

EFFECTS OF MECHANIZATION

Pillar Mining Systems

Improvements on coal cutting, loading, and transportation equipment have made it possible for a single continuous-mining unit to produce coal consistently at the rate of 400–600 tons/machine shift over a period of several months, and for conventional mechanized units to produce coal at the rate of 200–500 tons/unit shift, consistently.

This means that fewer machines are required to produce a given amount of coal, and also that a given daily tonnage may be produced from a very limited working area. Conversely, it is necessary, for maximum utilization of machine capacity, that a minimum amount of time be lost in moving equipment from one heading to another during the working shift. Therefore production from each unit must be confined to a limited area during each shift. Limiting the area, and the higher rate of extraction of coal in turn simplify the support problem.

At present 87 per cent of all underground production in the United States is loaded by mechanical loaders.[3] The present trend in mechanization is toward increased use of continuous-mining machines. In 1952, 8 million tons of bituminous coal were mined by continuous-mining machines. In 1961 it was estimated that 86 million tons were mined by this method. The trend in underground mining systems, and roof support methods, is toward those which permit most efficient utilization of these mechanical methods.

Productive Time

In twenty mines studied which employed conventional mining methods and continuous-mining machines[4] the productive time (face time less all delays) for continuous miners ranged from 2.58 to 5.67 hr (see Table 2). Major operating delays resulted from mechanical breakdowns on equipment, work stoppage to permit timbering, haulage interruptions, and interruptions due to moving and maneuvering of equipment. In five mines the lunch time loss was eliminated by staggering the lunch periods of the crew members.

TABLE 2. DISTRIBUTION OF 8-HR SHIFT[4]

Mine No.	Time (hr)					
	Travel	Lunch	Face	Mechanical delays	Other delays	Productive
1	1.00	*	7.00	0.65	1.40	4.95
2	1.00	0.50	6.50	1.00	2.50	3.00
3	1.25	0.50	6.25	0.70	1.25	4.30
4	0.67	0.50	6.83	1.50	0.50	4.83
5	0.50	*	7.50	0.28	2.52	4.70
6	1.33	0.50	6.17	1.00	0.17	5.00
7	1.67	0.50	5.83	0.83	1.80	3.20
8	0.50	0.50	7.00	0.67	1.08	5.25
9	2.25	*	5.75	0.67	2.50	2.58
10	2.00	*	6.00	1.00	0.50	4.50
11	1.25	*	6.75	0.50	1.00	5.25
12	1.00	0.50	6.50	0.83	0.17	5.50
13	1.50	0.50	6.00	0.40	1.93	3.67
14	1.25	0.50	6.25	1.00	1.50	3.75
15	1.10	0.50	6.40	0.32	1.00	5.08
16	0.80	0.50	6.70	1.10	0.50	5.10
17	0.33	0.50	7.17	1.20	0.30	5.67
18	1.00	0.50	6.50	1.50	2.00	3.00
19	1.00	0.50	6.50	1.00	1.33	4.17
20	1.50	0.50	6.00	0.67	0.33	5.00

* No time loss — crew lunch period staggered.

COMPARISON OF PERFORMANCE OF CONTINUOUS-MINING AND MOBILE-LOADING UNITS

Performance of continuous-mining and of mobile-loading units at the same mine is compared in Table 3. Mines are shown in order by thickness of the bed mined. No distinction of performance extracting pillars and mines not extracting pillars was made in this study. Productivity of raw coal per man-shift by continuous-mining units was greater than by mobile-loading units for the same thickness of coal at the same mine. These differences ranged from 2.7 to 45.3 tons of raw coal per man-shift.

TABLE 3. COMPARISON OF PERFORMANCE OF CONTINUOUS MINING AND MOBILE LOADING[4]

Mine No.	Bed thickness mined (in.)	Continuous mining Raw coal (tons)		Mobile loading Raw coal (tons)	
		Per unit per shift	Per man per shift on unit	Per unit per shift	Per man per shift on unit
9	41	179.9	27.7	90.6	9.7
16	42	101.0	13.5	*	*
7	42	104.8	17.5	123.8	14.8
11	42	121.0	22.8	89.1	11.1
8	42	139.0	23.6	183.0	15.3
10	43	133.3	22.2	100.0	9.1
5	48	150.0	23.1	*	*
16	53	105.2	19.1	*	*
17	54	250.0	38.5	200.0	16.7
1	54	369.2	46.1	190.2	19.0
12	55	180.0	27.7	155.0	15.5
6	57	199.7	30.7	130.0	17.0
20	58	141.7	19.8	141.7	11.8
18	72	565.0	70.6	304.0	25.3
19	73	209.0	29.8	306.0	21.6
13	74	188.5	26.9	*	*
15	76	210.0	32.3	277.7	17.4
4	84	278.0	61.8	385.0	39.8
2	84	313.6	48.3	*	*
14	90	206.0	29.4	154.9	14.5
3	102	293.5	48.9	373.0	25.9

* No production from mobile-loading units.

Production Curves

Production data for the two types of mining at the twenty mines studied[4] are shown graphically for comparative purposes by plotting tons of raw coal per man per shift in the unit crew against the thickness of coal mined (see Fig. 5.) Group-

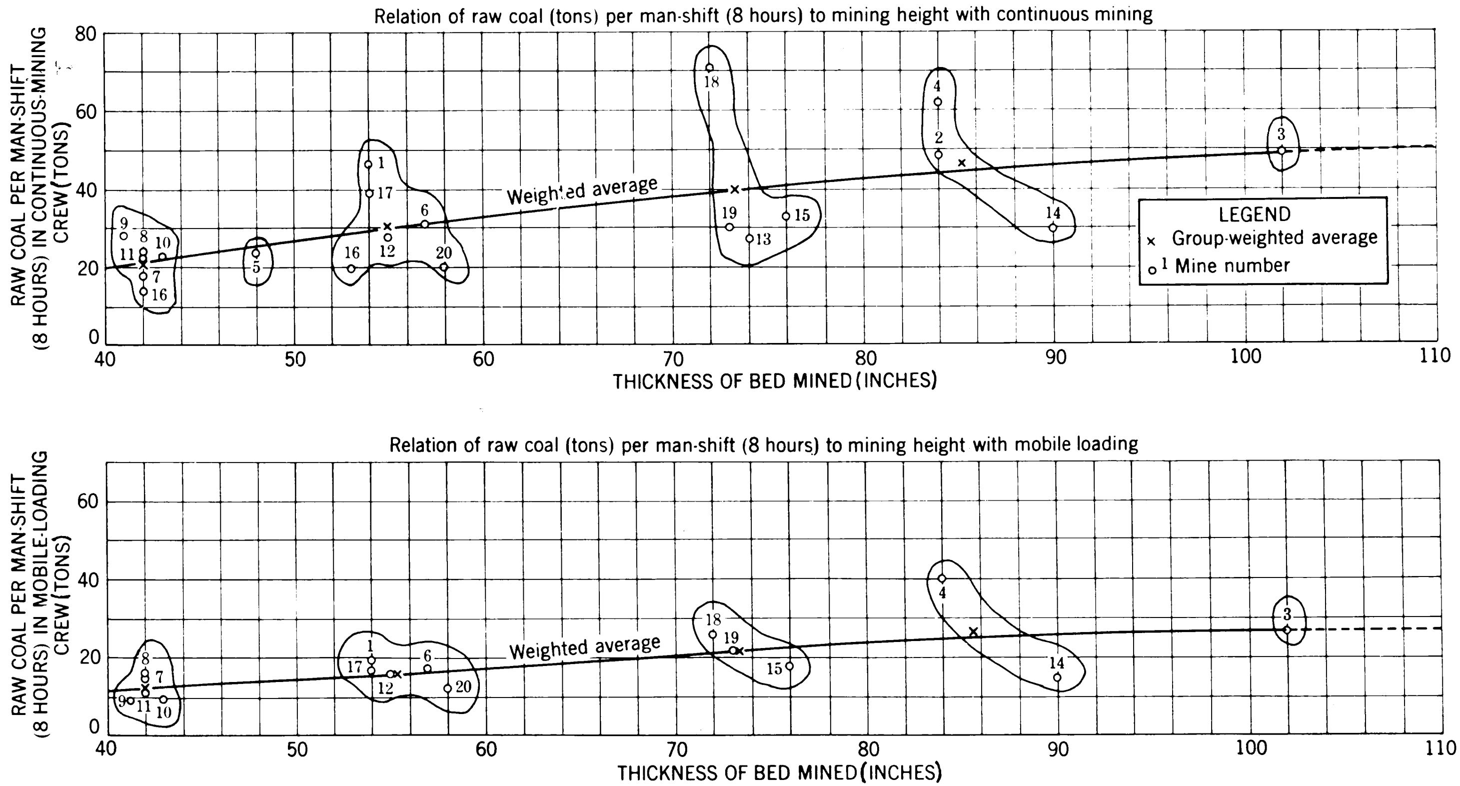

FIG. 5. Comparison of performance of continuous mining and mobile loading at twenty bituminous coal mines.[4]

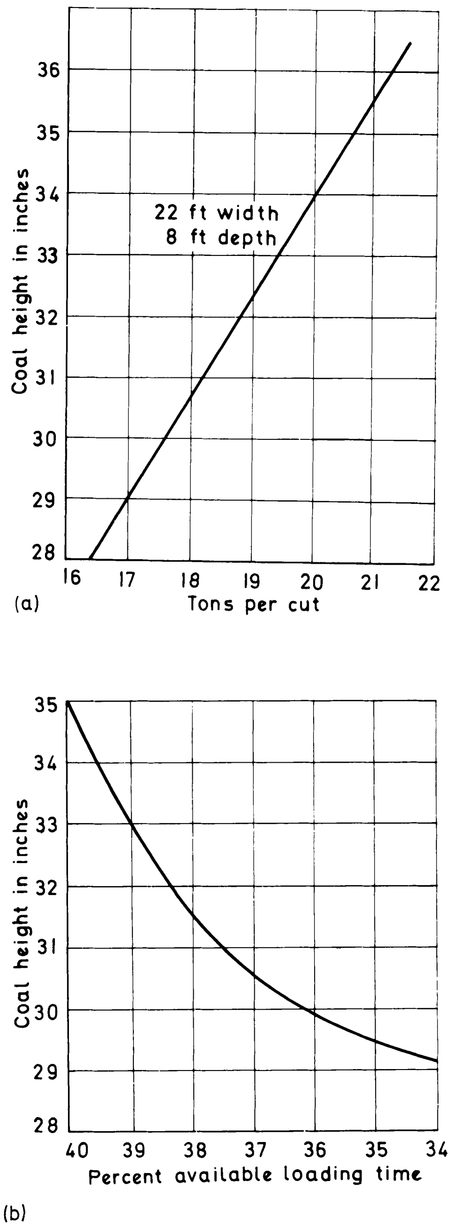

FIG. 6. (a) Assuming the same width and depth of a cut of coal, the tonnage produced from the cut is directly proportional to the height of the coal.[11]

(b) Available loading time varies with the height of the coal seam for a fixed amount and type of loading and conveying equipment.[11]

weighted average points were calculated and plotted for groups of mines having nearly the same mining thickness. The curves shown were based on these points. The curve for continuous mining tends to flatten off at 102 in.

However, this trend is not conclusive because: (1) Only one mine in this bed-thickness category was included in the study, and (2) continuous-mining machines mining greater bed thicknesses were not available for study. Probably the same gradual upward trend of the production curve would be maintained in mining bed thicknesses greater than 102 in. if continuous-mining machines were developed to mine greater bed thicknesses. In opening a new mine, or in converting an old mining system to continuous mining, it is possible from the curves shown to approximate the tons of raw coal per man-shift in unit crews that could be produced from different bed thicknesses.

For the mines studied it was apparent that continuous mining was the most productive method (per man-shift) of underground mining whether pillars are, or are not, extracted. Cleaning and sizing of the product does not seem to pose any difficult problems for most preparation plants. In fact, in two of the mines studied which were using continuous-mining and mobile-loading equipment, the reject from continuous mining was 4 and 8 per cent and from mobile loading, 10 and 20 per cent.

Limitations on Continuous Mining

In those seams, or portions of seams, which contain "sulfur balls" or other pyritic inclusions, the use of continuous-mining machines may not be economical. Because of the hardness of pyrites it is difficult to cut, even with tungsten carbide bits, and not only causes excessive wear on machines but the shift tonnage is greatly reduced.

Generally the efficiency of a continuous miner decreases as the height of the coal seam decreases. The ripper-type continuous miner uses five phases to complete a cycle; these are tram, sump, raise rippers, lower rippers, and position rippers. During a cycle coal is being produced during two of these phases only – sumping, and raising or lowering the rippers. With decreased height of seam the non-productive operation phases consume a greater percentage of the face cycle time.

The success of a continuous miner depends to a great extent on roof conditions. This machine can operate a maximum efficiency if roof conditions will permit a full breakthrough length before the machine has to move to the next place. If the roof requires support each time after the miner has advanced 10 or 15 ft then the production capacity of the machine will be greatly reduced.

Generally continuous miners can produce about 20–25 per cent more per shift when working pillars as compared with production from development work. The continuous miner can work on one pillar until it is extracted whereas conventional equipment must have at least three pillars in order to employ all section units efficiently. For this reason conditions will probably be more favorable for the use of continuous miners after a mine is fully developed and a large portion of the production comes from pillar extraction.

The continuous miner produces a large proportion of fine coal. Plant capacity for cleaning fine coal costs more than plant capacity for cleaning coarse coal and it is also a fact that the operation of the fine coal cleaning plant costs more than the coarse coal cleaning plant. This is a cost which must be taken into account when comparing overall production costs.

Generally a continuous mining section will require about six men for operation while a conventional mining section will require about nine men. The continuous mining section will usually show a higher output per face-man than the conventional but the conventional section will produce more tons per shift. Therefore it will require more continuous mining sections to produce a given tonnage. This must be taken into account when figuring overall costs.

MINING EQUIPMENT COSTS

Costs of Haulage Units

The cost per mile of main line haulage system (85 lb. rail) in a thick coal seam is about $150,000. The cost of a high speed 35-ton locomotive is about $85,000. Approximate costs of cars and rotary dumps are shown in Table 5. Modern track systems are capable of handling 400 or more tons per shift per employee.

TABLE 4. SOME HAULAGE SYSTEM DESIGN FACTORS[5]

1. Thickness of seam.
2. Total daily raw-coal production.
3. Maximum length of haulage system.
4. Available coal reserves.
5. Physical shape of property.
6. Seam contours, grades, general and local.
7. Nature of mine roof and bottom.
8. Percent of reject.
9. Size, nature and tonnage of rock to be handled.
10. Number of daily production shifts.
11. Mining plan and layout.
12. Type of mining equipment.
13. Availability of equipment.
14. Life of mine.
15. Type of power employed for face equipment.

Table 6 lists costs of various types of conveyors. In 1957 about 38 per cent of the total underground production was carried by conveyor. Belt conveyors have gained the fastest acceptance in relatively thin seams where their use eliminated the need for brushing the roof and floor. Conveyors also have an advantage over wheeled or tracked equipment where grades are irregular.

TABLE 5. APPROXIMATE 1958 MINE CAR COSTS(5)

Car description	Capacity	Type coupling	Cost
Solid body, 4-wheel, roller bearing	5 ton	Link and pin, spring bumper	$1200
Solid body, 8-wheel, roller bearing	10 ton	Automatic	$2500
Solid body, 8-wheel, roller bearing	15 ton	Automatic	$3300
Solid body, 8-wheel, roller bearing	20 ton	Automatic	$4200
Approximate cost single-car rotary dump, total			$70,000–$85,000

For drop-bottom cars, add $100–$300 to approximate costs above.

For high-strength corrosion-resistant steel bodies, cost is 5–10 per cent higher.

TABLE 6. CONVEYOR COST COMPARISONS(5)

1. Panel Belts (rope type; troughing idlers on 6-ft centers; return, 18-ft centers)
   - 36 in. × 1000 ft .... $28.61 per ft
   - 42 in. × 1000 ft .... 32.81 per ft
2. Cross Belts (conventional type, including 4-ply, 32-oz duck, $\frac{1}{8}$ in. × $\frac{1}{32}$ in. belting)
   - 36 in × 130 ft .... $56.92 per ft
   - 36 in. × 210 ft .... 44.76 per ft
3. Main Belts (all with 4-ply, 50-lb belting)
   - (a) Conventional type (troughing idlers on 5-ft centers; returns on 15-ft centers)
     - 36 in. × 2500 ft (60 hp) .... $33.59 per ft
     - 36 in. × 3000 ft (75 hp) .... 32.76 per ft
     - 36 in. × 4000 ft (100 hp) .... 31.85 per ft
     - 42 in. × 2500 ft (75 hp) .... 38.18 per ft
     - 42 in. × 3000 ft (100 hp) .... 37.56 per ft
     - 42 in. × 4000 ft (125 hp) .... 37.38 per ft
   - (b) Rope sideframe belts (troughing idlers on 6-ft centers; returns on 18-ft centers)
     - 36 in. × 2500 ft (60 hp) .... $26.60 per ft
     - 36 in. × 3000 ft (75 hp) .... 25.97 per ft
     - 36 in. × 4000 ft (100 hp) .... 25.50 per ft
     - 42 in. × 2500 ft (75 hp) .... 30.86 per ft
     - 42 in. × 3000 ft (110 hp) .... 30.50 per ft
     - 42 in. × 4000 ft (125 hp) .... 30.46 per ft

TABLE 7. COMPARATIVE FACE COSTS—CONVENTIONAL VERSUS CONTINUOUS MINING FACE EQUIPMENT FOR VARIOUS SEAMS[5]

(Figures in the following table are averages for equipment costs, crews and tonnages to be expected from a *new* mine of the seam heights shown, using the latest 250 d.c. or 440 a.c. equipment. Cost of continuous miners includes roof drills and water sprays; but no other extras, such as power take-offs or automatic lubrication, are included in any of the machine cost figures. Estimates are based on a seam height half-way between the two figures in any column, i.e. 32 in. for the 28- to 36-in. column, 40 in. for the 36- to 44-in. column, etc. If the seam is higher than this half-way mark, figure costs a little higher. If the seam is lower, reduce the costs shown slightly. As a general rule, machinery and power costs go up as the seam gets thicker, but production increases by a greater ratio.)

Type of mining	Equipment crew and tonnages	Seam heights (in.)				
		28-36	36–44	44–60	60–84	84
Conventional Mining	Face drill	$ 1000	$ 1000	$32,500	$46,000	$46,000
	Cutter	53,000	53,000	58,500	58,500	58,500
	Loader	37,000	37,000	36,000	40,000	40,000
	2 shuttle cars	55,000	52,000	55,000	75,000	90,000
	Roof drills	–	11,000	11,000	27,000	27,000
	Totals	$146,000	$154,000	$193,000	$246,500	$261,500
	Face crew	9 men and 1 boss	11 men and 1 boss	11 men and 1 boss	12 men and 1 boss	12 men and 1 boss
	Tonnage shift	300	350	450	550	700
	Tons/face man	30	29	37	42	53
Continuous Mining	Miner	–	–	$120,000	$151,000	$151,000
	Extensible belt	–	–	94,000	94,000	110,000
	Totals	–	–	$214,000	$245,000	$261,000
	Face crew	–	–	5 men and 1 boss	5 men and 1 boss	5 men and 1 boss
	Tonnage shift	–	–	450	500	600
	Tons/face man	–	–	75	83	100
	Miner	$110,000	$121,000	$120,000	$153,000	$148,500
	Loader	–	–	36,000	40,000	40,000
	2 shuttle cars	55,000	52,000	55,000	75,000	90,000
	Totals	$165,000	$173,000	$211,000	$268,000	$278,500
	Face crew	5 men and 1 boss	5 men and 1 boss	7 men and 1 boss	7 men and 1 boss	7 men and 1 boss
	Tonnage shift	250	300	450	500	600
	Tons/face man	41	50	56	62	75

In general, either mobile equipment or conveyors may be used for transportation in seams which pitch up to a maximum of about 8–10°. For seams pitching up to a maximum of about 18–20° conveyors are most satisfactory. For pitches steeper than about 20° metal slides may be used to convey the coal to entries. Coal will just slide on wet galvanized iron at a slope of about 20°.

Comparative Costs — Conventional vs. Continuous Mining Equipment

Table 7 shows comparative costs for equipping a face with conventional equipment as compared with costs for continuous equipment. The costs of the latter are higher but fewer men are required for its operation.

Mining Equipment and Production[5]

Hypothetical comparisons of equipment requirements, equipment costs, crew sizes, and production rates for continuous and conventional systems of coal mining in thick seams (5 ft and higher) and in thin seams (3 ft) are always interesting. For example, continuous mining in a production section of 5-ft coal may include two boring-type continuous miners, two seven-unit Molveyors, two sectional belt conveyors and one belt gathering conveyor, all at a cost of about $600,000. A 7-hr shift should produce 800 tons, average, with a ten-man crew.

Under conventional methods in a 5-ft seam, a production section might have two rubber-tired universal cutting machines, two high-capacity crawler-mounted loaders, four shuttle cars, one face-drilling machine, one roof-drilling machine, and one belt gathering conveyor. Total cost would be about $400,000, and average production of 700 tons per 7-hr shift could be expected with a crew of twelve men.

Continuous mining in 3-ft coal would be not unlike that for the 5-ft seam, but would differ in equipment required. At an investment of some $400,000, two thin-seam boring-type continuous-mining machines, two bridge conveyors, two mobile-head chain conveyors and one belt gathering conveyor should average 600 tons in a 7-hr shift, with ten men in the production crew.

Conventional mining in 3-ft coal probably would call for two universal cutting machines, two thin-seam conventional loading machines, four shuttle cars, two hand-held face drills, one roof drill and one belt gathering conveyor at a capital outlay of about $360,000. This equipment and a crew of eleven men should produce an average of 500 tons/shift of 7 hr.

The cost per ton mined in the continuous-mining system consistently averages less than the cost per ton in the conventional mining system—an average figure might be somewhere in the vicinity of fifty cents per ton less. One example we might cite is that a mine working a particular seam with continuous borers came up with a face cost per ton reduced to one-half that of conventional mining costs even when including the cost of labor and maintenance as well as materials such as roof support supplies, rock dust, bits, oil, and grease.

TABLE 8. COMPARISON OF THE COST OF PRODUCTION FOR MACHINE LOADED COAL FROM FIELD "A" – COMPARING THE YEAR 1949 WITH THE LAST HALF OF 1959[10]

Item	(1) 1949		(2) Last half 1959		(3) Percent increase or decrease	(4) Projected 1969		(5) Possible 1969	
	Cost per ton	% of total cost	Cost per ton	% of total cost		Cost per ton	% of total cost	Cost per ton	% of total cost
Productive labor	$1.30	29.0	$0.89	19.4	−32	$0.61	12.6	$0.21	5.3
Service labor	0.99	22.1	0.89	19.4	−10	0.79	16.3	0.66	16.7
Total direct U.M.W.A labor	$2.29	51.1	$1.78	38.8	−22	$1.40	28.9	$0.87	22.0
Mine supplies	0.66	14.8	1.03	22.6	+56	1.61	33.2	1.20	30.3
Power	0.12	2.7	0.13	2.8	+ 8	0.14	2.9	0.14	3.5
Mine supervision, clerical, and misc. Mine overhead expense	0.48	10.7	0.44	9.6	− 8	0.40	8.2	0.36	9.1
U.M.W.A. welfare and vacation pay	0.28	6.2	0.47	10.3	+68	0.47	9.7	0.47	11.9
Total mine cost	$3.83	85.5	$3.85	84.1	+0.5%	$4.02	82.9	$3.04	76.8
All taxes, insurance, compensation, dues, and assessments	0.28	6.2	0.34	7.4	+21	0.41	8.4	0.41	10.3
Administrative expenses	0.12	2.7	0.11	2.4	− 8	0.11	2.3	0.11	2.8
Depreciation	0.25	5.6	0.28	6.1	+12	0.31	6.4	0.40	10.1
Total cost of production (exclusive of depletion and tonnage royalties)	$4.48	100.0%	$4.58	100.0%	+2%	$4.85	100.0%	$3.96	100.0%

LONG-WALL MINING SYSTEMS – COSTS AND PRODUCTIVITY

The foregoing section dealt with the mechanization of pillar mining systems under the relatively favorable natural conditions found in U.S. mines. The average productivity per man-shift in European mines where long-wall mining systems are used almost exclusively is much less, being generally of the order of 1–2 tons per man-shift. Approximate average production in short tons per man-shift in

1958 in Great Britain was 1.3; in West Germany 1.3; in Belgium 0.9; in France 1.3. In Russia the production per man-shift was about 1.6 tons in 1955.

All the figures given above are based on the total number employed, both on the surface, in preparation plants, and underground, at underground mines.

Relatively unfavorable natural conditions are largely responsible for the low production figures. In many cases the seams lie under cities, canals, and highly developed areas and surface subsidence must be closely controlled. Packing or backfilling is often required.

In addition the coal seams tend to be rather irregular in dip, having been subjected to folding and faulting.

The depth pressure usually necessitates the use of extensive and carefully placed support. Thus 75–80 per cent of face labor may be devoted to roof support. This compares with the U.S. pillar systems in which only 10–20 per cent of face labor may be required for support in development headings and 15–25 per cent of face labor may be expended on roof support during pillar extraction.

METHODS FOR INCREASING PRODUCTIVITY OF LONG-WALL SYSTEMS

In the United States the relatively shallow depths of the workings and the generally favorable geological conditions have encouraged the development of machinery designed for room-and-pillar work. In Great Britain and in Europe the depths of the coal beds now being worked result in heavy ground pressures. These pressures, together with the necessity for conservation of coal by total extraction of the seams, and the necessity for protection surface improvements by limiting the amount of surface subsidence, have caused the long-wall system to be used almost exclusively.

Recent efforts to increase output per man-shift (O.M.S.) have been directed to applying mechanized equipment to the long-wall face. Recent improvements have been:

(1) The application of shearers, trepanners, plows, and similar machines which rip or cut the coal from the face and load it onto the face conveyor in a continuous operation thereby eliminating the drilling and shot-firing cycles.

(2) Improved systems of roadway supports which include the application of "double packing", the installation of yielding steel arches, and the application in a few instances of roof or floor bolting.

(3) Improved systems of face support. Yielding steel props, and hydraulic props have given greatly improved control of the roof. Systems of self-propelled hydraulic roof supports have been introduced within the past few years and where they have been used they have greatly reduced the number of men required for operation of a long-wall face.

Self-Advancing Hydraulic Props

At Sunnyside, Utah, a system of self-advancing hydraulic supports was installed in conjunction with an Anderton shearer on a 310 ft face in a seam 5–6 ft thick. With this equipment a crew of eleven men (four of whom operated the supports) consistently produced 400–500 tons of raw coal per shift.

Russian efforts at improving support systems for long-wall workings have concentrated on developing a system of mechanized self-propelled props, or a system of self-propelled shields, which can be advanced mechanically as the face advances. Shields for use in steeply dipping coal seams resemble steel canopies which support themselves on ledges of coal left in place along the hanging-wall and on the foot-wall. Under the protection of the shield a layer of coal is scraped out, the shield is then lowered and another layer of coal is scraped out, progressing thus to the level below.

Roof Support Costs on Long-wall Faces

Roof support is the principal cost item in the long-wall system. At least two, and usually three rows of props must be provided behind the advancing face. In addition, if the roof is strong, a row of breaker props, wooden cribs or chocks, may be required to force the roof to fracture along the desired line.

TABLE 9. CLASSIFICATION OF EMPLOYEES AND NUMBER OF SHIFTS WORKED[6]
(Long-wall panel mined in Pocahontas No. 4 Seam, Helen, W. Va.)

Classification	Total number of shifts worked			
	Panel 1	Panel 2	Panel 3	Total
Foremen	164	203	181	548
Operators	93	73	61	227
Headpiece cleaners	138	147	3	288
Tailpiece cleaners	93	73	60	226
Boom men	93	73	61	227
Beltmen	101	64	25	190
Loaders (clean up)	173	135	120	428
Jack setters and crib builders	1622	1402	1299	4323
Jack robbers	278	*	*	278
Rock-dust men	90	—	3	93
Rock drillers	44	2	3	49
Slate men	14	3	—	17
Timbermen	8	—	—	8
German instructor	93	—	—	93
Cylinder man	†	†	43	43
Total	3004	2175	1859	7038

* Jack setting and robbing combined.
† Cylinder man reported as jack setter.

TABLE 10. MINING SUMMARY[6]

	Panel 1	Panel 2	Panel 3	Average or total
Length of long-wall face mined (ft)	328	328	328	328
Length of block mined (ft)	1441	892	832	3165
Area of block mined (acres)	10.85	6.72	6.26	23.83
Average thickness of coal mined (in.)	34	34	34	34
Clean coal recovered from raw coal (%)	95	95	95	95
Total number of single shifts operated	93	73	61	227
Average retreat of face per shift (ft)	15.5	12.2	13.6	13.9
Number of man-shifts to mine block	3004	2175	1859	7038
Number of man-shifts to develop panel*	2537	2071	1679	6287
Total man-shifts to develop and mine block	5541	4246	3538	13,325
	Raw coal (tons)			
Production from panel	54,158	34,935	31,219	120,312
Average production per shift	582†	479	513	530
Maximum production for one shift	842	732	747	–
Average production per man-shift	18.0†	16.1	16.8	17.1
Production from development	17,667	14,588	11,402	43,657
Total production from development and mining	71,825	49,523	42,621	163,969
Average production per man-shift for developing and mining	13.0	11.7	12.0	12.3
Average production per man-shift for developing and mining by conventional method at mine	10.9	10.0	10.0	–

* Includes moving to next panel and setting up equipment.
† Includes production during training period.

Props are ordinarily set about 3–4 ft apart, measured parallel to the face. This means that an average face may require an average of one prop for every foot of face length.

In addition wooden cribs, or chocks, may be spaced at 5–10 ft apart in one or two rows behind the props. Thus every 10 ft of face may have ten props and one or two cribs. A row of props must be "leap frogged" forward each time that the face advances 3–4 ft.

Tables 9 and 10 show the labor requirements for three panels mined by retreating long-wall methods with full caving in the Pocahontas No. 4 Bed, Helen, West Virginia. In this instance almost 65 per cent of the labor time was charged to support labor; that is, to "jack-setters" and "jack-robbers" (removing and resetting steel props) and to "crib-builders".[6]

The foregoing example was for retreating long-wall with full caving. If packwalls, or filling, are used for support in the goaf area labor requirements will be greater than those for full caving. It is estimated by German mine operators in the Ruhr that the average total cost of backfilling is about equal to the costs of mining the coal.[7] In many parts of the Ruhr district backfilling is required for control of surface subsidence. Solid packing by hand permits the least subsidence (about 15 per cent of the coal bed thickness) but it is very expensive and suitable material is hard to obtain. In steep seams fill material is dumped in and allowed to flow into place by gravity. In flatter seams it may be blown in with "pneumatic packing machines" or thrown into place by "throwing machines" (see Volume 2, Part A, Chapter 2).

The backfilling of mined-out areas is general practice where the coal bed inclination is greater than 25°. Only about half of the mined areas in the flatter beds are backfilled. In many the full caving system is followed.

In advancing long-wall systems support labor additional to the face labor is required for the maintenance and construction of the roadways leading to the face. As the face advances the "rear abutment" of the "pressure arch" compresses the gob filling and tends to crush supports and to close the roadways. Thus considerable labor is required for "brushing" (ripping) the roadways, and for replacing and repairing supports.

Choice of Machine for Coal Face Mechanization

Many different types of machines are manufactured for long-wall mining and some of the factors which determine which machine will give the best service in a given seam are as follows:

(1) The hardness or toughness of the coal.
(2) Thickness of the seam.
(3) Gradient of the seam.
(4) Direction of the cleat.
(5) Depth of the seam and strength of the roof rock.
(6) Market requirements for the coal.
(7) Presence of partings at roof and floor.
(8) Presence of dirt bands in the seam.
(9) General geological conditions and regularity of the seam.
(10) Total cost of a mechanized installation.

The aim of a mechanized installation should be to get a greater quantity of coal at a reasonable cost from the same or a reduced number of men as would be employed on a conventionally worked face.

In some cases softening of the coal by water infusion, shotfiring, or pre-cutting may be required before a suitable cutter-loader will work.

Table 11[8]

Long-wall machines	Minimum seam thickness	Nature of floor	Nature of roof	Hardness of coal	Width of web	Cutting speed per min	Average performances obtained during 2nd quarter, 1959, in East Midlands Division	
							Area extracted per M/c day (yd^2)	Pithead output per M/c day (tons)
Plow	1 ft 6 in. (may be less)	Reasonably good	Not critical	Soft and Medium hard	6 in. fast plow 12 in. slow plow	72.5 ft fast plow 40 ft slow plow	277	275
Huwood slicer	4 ft (less with new model)	Not critical	Not critical	Not critical	14 in.		267	445
Meco Moore	3 ft	Strong	Good roof	Medium hard	4 ft 6 in.	30 in.	288	467
Trepanner	3 ft 3 in.	Reasonably good			27 in.	5–11 ft	402	500
Flight Loader	2 ft	Reasonably good	Not critical	Not critical (Top coal should fall)	Variable		131	124

Huwood Loader	1 ft 11 in.	Reasonably good	Medium Strong		4 ft 6 in.	1.5–3.7 ft*	–	–
Gloster Getter	2 ft 8 in.	Reasonably good	Good roof	Hard coal	30–36 in.	3–6 ft	222	261
Anderton Shearer	2 ft 6 in.	Not critical	Not critical	Not critical	20, 16 and 30 in.	10–18 ft	361	452
Dosco Miner	4 ft 6 in.	Good	Good	Medium hard	5 ft		241	514
Multi-jib cutter loader	2 ft	Not critical	Not critical	Not critical	30–48 in.		–	–

* Gives the speed of travel of the machine while loading.
The figures given in the above table are only approximate and are likely to vary.

Table 11 is a rough guide to the suitability of some of the most popular coal getting machines used in Great Britain.[3] An attempt has been made to arrange them in the order of their degradation effect.

Table 12[8] lists the maximum gradients being worked for varying conditions by the various types of power loaders. On gradients exceeding 1 in 3 power winches are needed to control the machine along its track.

TABLE 12[8]

Machine	Gradient along the face		Gradient in direction of face advance	
	With conveyor	Against conveyor	To rise	To dip
Rapid Plow	1 in 2.5	1 in 12	1 in 5.5	1 in 6
Huwood Slicer	1 in 10	1 in 7	1 in 11	1 in 14
Meco-Moore	1 in 5	1 in 6	1 in 4.3	1 in 6
A. B. Trepanner	1 in 4.7	1 in 11	1 in 8	1 in 7
Flight Loader	1 in 1.6	1 in 6	1 in 3	1 in 15
Huwood Loader (Ski-Hi)	1 in 8	1 in 6	1 in 6	1 in 30
Anderton Shearer	1 in 1.9	1 in 6	1 in 5	1 in 3.5
Dosco Miner	1 in 10	1 in 10	1 in 11	1 in 27

Once a certain type of machine has been decided on as suitable for an installation the following factors should be considered in detail:[8]

(1) Types of supports and conveyor to be used.
(2) The possible degradation of the coal and its effect on coal preparation.
(3) The requirements of the stable hole.
(4) Wages and costs.
(5) Labor relations and safety.
(6) Organization of face operations and training of crews.

The total number of coal face machines in use in 1949 was about 176. By the end of 1959 this figure had risen to 1036. The long-wall machines most commonly used in 1948 were the Meco-Moore cutter loader and the Huwood loader, but the period after 1953 saw the introduction of a number of new machines.

Table 13 shows the numbers and types of machines in use between 1947 and 1959.[3]

The main reason for the decline of certain machines in popularity appears to be the degree of degradation which they produce in coal sizes. The trend of mechanization will probably be toward the use of those machines which produce the greatest percentage of large coal. For this reason plows and trepanners will probably

TABLE 13[8]

Type of machines	Number in use at the end of the year									
	1947	1949	1951	1953	1954	1955	1956	1957	1958	1959
A. LONG-WALL MACHINES										
1. Anderton Shearer loader	—	—	—	4	25	78	192	302	294	311
2. Cutter chain loader	—	—	1	21	36	83	164	188	151	137
3. A. B. Trepanner	—	—	—	1	2	5	11	30	72	116
4. A. B. Meco-Moore	20	49	65	94	102	118	135	127	115	104
5. Rapid Plow	—	—	—	10	15	14	15	31	50	82
6. Huwood loader	N/A	27	38	24	19	14	25	18	35	62
7. Multi-jib cutter loader	—	—	—	—	2	6	19	48	33	42
8. Huwood Slicer	—	—	1	1	2	2	7	15	24	23
9. Scraper Box	—	—	—	2	2	8	9	9	15	21
10. Gloster Getter	—	—	—	3	14	19	25	18	15	11
11. Dosco Miner	—	—	—	—	1	2	7	12	11	8
12. Slow Plow	1	1	3	10	7	12	7	9	10	6
13. Samson Stripper	2	3	6	6	6	4	6	5	6	3
14. Mawco cutter loader	—	—	—	—	—	—	—	—	—	2
15. Gusto Multi plough	—	—	—	—	1	2	1	3	—	—
Total	23	80	114	175	234	417	623	815	831	928
B. ROOM-AND-PILLAR MACHINES										
1. Gathering arm loader	N/A	49	34	40	52	50	58	60	63	61
2. Joy continuous miner	—	—	—	5	3	5	11	15	23	27
3. Duckbill loader	N/A	47	34	29	31	22	20	26	22	12
4. Cutter chain loader	—	—	3	11	4	8	19	18	11	8
Total		96	71	85	90	85	108	119	119	108
Grand Total		176	185	260	324	502	731	934	950	1036

Note — The above table does not show the experimental models at the end of each year.

increase in popularity. The re-designing of the drum of the Anderton shearer may improve the sizing of the coal it produces and bring a futher increase in its popularity.

In bord-and-pillar workings there was no great change in the total number of machines working although there was a decided increase in the number of continuous mining machines in use. Only a small percentage of coal is produced from bord-and-pillar workings. In 1949, 12.4 per cent of total output came from bord-and-pillar workings while by 1959 it had dropped to 7 per cent.(8)

Coal Degradation

One of the greatest drawbacks to mechanization has been the degradation of coal size. If a proper choice of mining machine is not made it is possible that the advantages gained by increased production will be offset by the decrease in the selling price of the smaller coal which is produced.

The following precautions should be taken to reduce coal degradation on power loaded faces:(8)

(1) Use sharp cutter picks.

(2) Prevent churning of the coal by stopping the machine as soon as the conveyor stops.

(3) High chain speed and better gummers will reduce circulation of cuttings.

(4) Coal should not be broken any more than is absolutely necessary to prepare it for cutting and loading.

(5) In thin seams plows may be used to load the prepared coal rather than multi-jib cutters.

(6) Haulage speeds as high as practicable should be used. The higher the haulage speed the greater is the pick penetration and the less the degradation produced.

(7) Changes in the lacing of the cutter picks have resulted in improved sizes from the Anderton shearer.

(8) The proportion of fines depends upon the thickness of the seam. For a particular width of kerf the proportion of fines increases as the seam thickness decreases. This effect may be reduced by using thin kerf machines or by cutting into the floor.

Improvements Needed in Mechanization

There is need for improvement in certain methods and procedures on mechanized faces. Great improvement in O.M.S. figures would be brought about by successful solution of the following problems:(8)

TABLE 14. PERSONNEL REQUIREMENTS AND WAGES COST FOR CONVENTIONAL FACE[9]

Occupation	Number	Basic rate			Wage/shift			Extras		Cost per shift		
		£	s.	d.	£	s.	d.	s.	d.	£	s.	d.
DAY SHIFT:												
Colliers	31	1	6	0	3	1	6	5	7	103	19	7
Face roads	6	1	4	0	2	17	3	5	7	18	17	0
Belt tensions	3	0	17	6	1	1	0	2	5	3	10	3
Transfer point	1	0	17	6	1	1	0	2	5	1	3	5
Measurer	1	0	17	6	1	1	0	2	5	1	3	5
Supports checker	1	0	17	6	1	1	0	2	5	1	3	5
Belt attendant	1	1	7	9	1	13	4	5	7	1	18	11
Supplies	4	1	7	9	1	13	4	5	7	7	15	7
Deputy	1	16	5	0	2	9	0	5	7	2	14	7
AFTERNOON SHIFT:												
Conveyor turners	4	0	18	0	2	4	4	5	7	9	19	8
Pipe fitters	3	0	14	0	1	15	9	5	7	6	4	0
Withdrawers	6	0	17	0	2	2	2	5	7	14	6	6
Chocks	6	0	18	0	2	4	4	5	7	14	19	6
Face roads	3	1	4	0	2	17	3	5	7	9	8	6
Face roads assistants	6	1	10	9	1	16	11	5	7	12	15	0
Deputy	1	16	5	0	2	9	0	5	7	2	14	7
NIGHT SHIFT:												
Coal cutter operator	1	0	18	0	2	4	4	5	7	2	9	11
Coal cutter assistant	1	1	7	9	1	13	4	5	7	1	18	11
Coal cutter trackman	2	0	18	0	2	4	4	5	7	4	19	10
Water infusion	2	0	19	0	2	6	6	5	7	5	4	2
Belt attendant	1	1	7	9	1	13	4	5	7	1	18	11
Supplies	2	1	7	9	1	13	4	5	7	3	17	10
Face roads	3	1	4	0	2	17	3	5	7	9	8	6
Face roads assistants	6	1	10	9	1	16	11	5	7	12	15	0
Deputy	1	16	5	0	2	9	0	5	7	2	14	7

Total face personnel = 97
Output per day = 400 tons
Face O.M.S. = 4.1 tons
Total wages cost per day = £258 1*s*. 7*d*.
Wages cost per ton = 12*s*. 10.3*d*.

(1) Elimination of stable holes

A disproportionate number of men is required for the excavation and support of stable holes. The Dawson-Miller stable hole machine which mechanizes the excavation of stable holes is a partial answer to this problem (see Chapter 5).

Another possible solution is a power loader which excavates its own stable hole such as the Dranyam machine which has been undergoing tests during recent years.

(2) Mechanization of rippings

A new ripping machine has been designed with a small diameter picked drum mounted upon a radial arm contrived to arc over to cut the roadway roof in an

TABLE 15. PERSONNEL REQUIREMENTS, O.M.S. VALUES AND WAGES COST FOR LOBBE PLOW FACE[9]

Occupation	Number	Rate/shift			Wage/shift			Extras		Total cost		
		£	*s.*	*d.*	£	*s.*	*d.*	*s.*	*d.*	£	*s.*	*d.*
DAY SHIFT:												
Captain	1	1	5	0	3	1	6	5	7	3	7	1
Plow operators	2	1	1	0	2	10	9	5	7	5	12	8
Supports and pushers	14	1	1	0	2	10	9	5	7	39	8	8
Preparing stables	4	1	6	0	3	1	6	5	7	13	8	4
Preparing stables	3	1	7	9	1	13	4	5	7	5	16	9
Supports checker	1	0	17	6	1	1	0	2	5	1	3	5
Supplies	2	1	7	9	1	13	4	5	7	3	17	10
Electrician	1	1	12	11	1	19	2	5	7	2	4	9
Face roads	3	1	4	0	2	17	3	5	7	9	8	6
Face roads	3	1	10	9	1	16	11	5	7	6	7	6
Deputy	1	16	5	0	2	9	0	5	7	2	14	7
AFTERNOON SHIFT:												
Chocks	6	1	1	0	2	10	9	5	7	16	18	0
Face roads	3	1	4	0	2	17	3	5	7	9	8	6
Face roads	6	1	10	9	1	16	11	5	7	12	15	0
Deputy	1	16	5	0	2	9	0	5	7	2	14	7
NIGHT SHIFT:												
Water infusion	2	1	1	0	2	10	9	5	7	5	12	8
Fitter	1	1	12	11	1	19	6	5	7	2	5	1
Face roads	3	1	4	0	2	17	3	5	7	9	8	6
Face roads	6	1	10	9	1	16	11	5	7	12	15	0
Deputy	1	16	5	0	2	9	0	5	7	2	14	7

Total face personnel	=	64
Output per day	=	400 tons
Face O.M.S	=	6.2 tons
Total wages cost per day	=	£175 2*s.*
Wages cost per ton	=	8*s.* 10.8*d.*

arched cross-section. A second picked drum rotates at the center of the arc. This machine is still in the experimental stage and methods have to be devised to dispose of the ripping dirt.

TABLE 16. COMPARISON OF COST—CONVENTIONAL FACE VS. PLOW FACE[9]

Capital cost of machinery and equipment for conventional face	
Electrical equipment	
Switchgear	£
District feeder switch	296
150 kVA transwitch, 3.3 kV/550 V.	675
Gate end panel for gate conveyor	220
Gate end panel for stage loader	220
Gate end panel for face belt	220
Gate end panel for coal cutter	220
Gate end panel spare	220
Gate end lighting unit	230
Total for switchgear	2301
Cables	
0.1 in.^{2} 3-core P.I.L.C.D.W.A. 3.3 kV grade, 300 yards	450
Connecting cable transwitch to gate end panel (type 20, 0.1 in.^{2} T.R.S.), 75 yards	120
Cable. Gate end panel to stage loader and face belt (type 22, 0.0225 in.^{2} T.R.S.), 30 yards	51
Coal cutter trailing cable (type 4, 0.0225 in.^{2} T.R.S.), 200 yards	500
Cable joint boxes (type E.S.B.5), 4	64
Total for cable	1185
Motors	
35 h.p. motor for gate belt	320
20 h.p. motor for stage loader	250
35 h.p. motor for face belt	320
Total for motors	890
Mechanical equipment	
Coal cutter	1900
Face belt conveyor	2500
Rigid steel props (900)	675
W—section steel roofing bars (475)	475
Mechanical chocks (150)	2250
Compressed air pipes, 230 yd.	92
Water pipes, 230 yd.	50
Compressed air picks (10)	326
Electric drill	52
Boring machines with air leg mounting	360
Total capital cost (electrical and mechanical)	£12,554

Capital cost of plough face and equipment

Electrical switchgear	£
District feeder switch	296
150 kVA transwitch, 3.3 kV/550 V	675
Gate end switch for gate belt, Gate end switch for stage loader — 2 at 220	440
M.U. 81 sequence board (5 panel) for conveyor	1077
Lighting unit	230
Total for switchgear	2718
Cables	
0.1 in.^{2} 3-core P.I.L.C.D.W.A. 3.3 kV grade, 300 yards	450
Connecting cable to gate end panel (type 20, 0.1 in.^{2} T.R.S.), 75 yards	120
Cable for conveyor motors (type 22, 0.0225 in^2. T.R.S.), 500 yards	850
Cable joint boxes, 4	64
Total for cable	1484
Motors	
Plow motors (45 h.p.)	1480
35 h.p. motor for gate belt	320
20 h.p. motor for stage loader	250
Total for motors	2050
Mechanical equipment	
Approximate cost of plow complete with conveyor and pushers, telephones, compressed air hose, etc.	27,000
Dowty props	10,080
Hinged bars	3234
Bar shoes	640
Chocks	2250
Pneumatic picks	163
Electric drill	52
Boring machines with air leg equipment	360
Total capital cost of equipment	£50,031

Extra capital cost of equipment for plow face

Electrical	
Cost of switchgear for plow face	2718
Cost of cables for plow face	1484
Cost of motors for plow face	2050
Cost of electrical equipment for plow face	6252
Cost of switchgear for conventional face	2301
Cost of cables for conventional face	1185
Cost of motors for conventional face	890
Cost of electrical equipment for conventional face	4376

Extra cost of electrical equipment for plow face = £6,252 − £4,376 = £1,876

Mechanical

	£
Cost of Lobbe plow unit complete	27,000
Cost of coal cutter, face conveyor	4400
Extra cost of power loading equipment	22,600
Cost of roof supports, etc., for plow unit	16,204
Cost of roof supports, pipes, etc., for conventional face	3542
Extra cost for plow face	12,662

Total extra capital cost for plow unit =
£1876+£22,600+£12,662 = £37,138

Of this some £22,600 is for equipment classed as capital plant and machinery such as conveyors, cutters, and power loaders. The remainder is classed as revenue expenditure, e.g. roof supports, L.T. switchgear and cables.

Power costs

Assuming a charge of 1*d.* per Board of Trade Unit:

	Per ton
Power cost for conventional face =	0.84*d.*
Power cost for plow face =	1.67*d.*
Extra cost for plow unit	0.83*d.*

Depreciation, interest, and maintenance charges

These charges are made on the excess capital incurred by the plow installation and are based on annual costs.

Depreciation

For power loading machines at $18\frac{3}{4}\%$ p.a. on £22,600	£4120 p.a.
For other machinery at $12\frac{1}{2}\%$ p.a. on £15,000	£1880 p.a.

Interest

For capital expenditure, i.e., 5% p.a. on £22,600	£1130 p.a.
Annual costs for depreciation interest	£7130
Annual output	100,000 tons
Cost per ton for interest and depreciation	1*s.* 5*d.*
Cost per ton for maintenance spares	6*d.*

Summary and comparison

Saving in wages cost/ton for plow face = 12*s.* 10.3*d.*−8*s.* 10.8*d.* = 3*s.* 11.5*d.*

Extra cost per ton due to more expensive machinery

	s.	*d.*
Interest and depreciation	1	5
Maintenance spares		6
Power costs		0.83
Total	1	11.83

This represents a saving of 3*s.* 11.5*d.*−1*s.* 11.83*d.* = 1*s.* 11.67*d.* per ton of output.

(3) Goaf control

In many cases when hand packing is used in the waste area it is extremely difficult to keep pace with the advance of the power loader. Where it is possible caving is the best method to follow; but where the goaf must be packed this should be done by power stowing.

Economics of Face Productivity[9]

The economics of face productivity is influenced by the following factors:

(1) Personnel requirements and O.M.S. values.
(2) Wages cost.
(3) Capital cost of machinery and equipment.
(4) Interest and depreciation charges.
(5) Maintenance charges.
(6) Power costs.
(7) Installation and withdrawal costs for equipment.
(8) Any alteration to the value of the coal produced due to degradation.

Table 14 lists the personnel requirements and wage costs for a typical hand-loaded face about 700 ft long. Table 15 lists the personnel requirements and wage costs for the same face equipped with a Lobbe plow. Mechanization of this face reduced wage costs by about one-third. Table 16 summarizes the effects of wage cost savings and capital cost increases.

Effect of Face Length[9]

Table 17 illustrates the possible effects of varying lengths of plow faces and refers to a seam 3 ft thick from which 8-in. slices are planed. This table indicates that the length of face has little bearing on the actual output of the plow itself. However, other factors which depend upon the face length have an important influence on the O.M.S. If a given output is obtained from a short face that face must advance a greater distance than a long face and the extra advance involves more work in preparing stables, driving face roads, packing, and erecting supports.

Thus for best results a plow face should be as long as is possible consistent with face gradient, geological conditions, ventilation considerations, and supervision convenience.

TABLE 17[9]

Face length (yd)	Plow run	Maximum cuts per shift	Plow output (tons/shift)	Stall output (tons/shift)	Percentage output by plow	Cubic yards ripping/shift
215	185	6.3	259	42	86.5	15
265	235	5.0	261	33	89.0	12
315	285	4.2	264	28	90.0	10

Productivity of Plow Faces in Germany

Coal plows are widely used in Germany where the soft and friable nature of the coal and the inclination of the coal seams makes plows the favored machines for face mechanization.

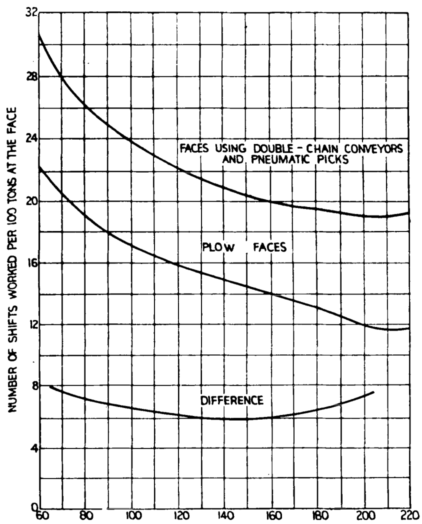

FIG. 7. Effect of seam thickness on productivity. (

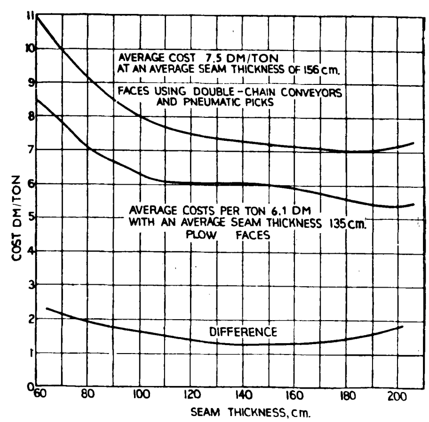

FIG. 8. Effect of seam thickness on productivity costs.[9]

Figure 7 shows the results of a study made to determine the relative productivity of plows as against the productivity of faces where coal is won by hand-held pneumatic picks and hand loaded onto face conveyors. The efficiency of the operation is expressed as the number of shifts worked at the face per 100 tons produced. It will be noted that the efficiency increases with increasing seam thickness and that the O.M.S. on plow faces was consistently greater than on hand worked faces.

The highest productivity for hand worked faces occurs at a seam thickness of $6\frac{1}{2}$ ft while a comparable productivity is obtained from plow faces operating in seams only 2 ft thick. Figure 8 summarizes the same information on a cost basis. Table 18 lists the O.M.S. for three different types of plow installations which are in use in Germany.

TABLE 18. PRODUCTIVITY OF PLOW INSTALLATIONS IN GERMANY[9]

Machine	Number of installations	Average daily output (tons)	Average face O.M.S. (tons)
Lobbe plow	52	451	6.37
Anbauhobel	21	458	6.45
Step knife	2	—	4.9

Coal Plows in The United States[6]

At an experimental installation in the Pocahontas No. 4 Coal Bed (35 in. thick) three panels were mined on retreating faces (each 330 ft long) with a rapid plow and the average O.M.S. for the three panels was 17.1 tons. The average rate of face advance over the whole extraction period was 14 ft per shift.

In this installation yielding steel props were used for face support and timber cribs were used for support at the caving line of the roof.

No stable holes were required as the triple entry previously driven at each end of the long-wall face was used as space for the conveyor and plow drive heads. Chain pillars which supported the triple entries were not recovered.

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CHAPTER 1

PILLAR-SUPPORTED (OPEN) STOPES

STOPING SYSTEMS

In general, vein type metalliferous deposits lack the regularity of form and the continuity of the bedded, non-metallic mineral deposits. Therefore, advance planning of metal mining systems is often limited by the difficulty of obtaining adequate advance information as to characteristics of such a deposit. This applies particularly to steeply dipping vein deposits. Usually only the upper portion of such a deposit can be adequately explored by drilling prior to the commencement of mining. In many cases mining is begun with only enough ore in sight to pay current expenses, and additional ore is blocked out as stoping operations proceed to greater depths.

Some metalliferous deposits are entirely situated within a few hundred feet of the surface. The zinc–lead deposits of the Tri-State District, and of the Illinois–Wisconsin fields, uranium deposits of the Colorado Plateau and of the Echo Lake District of Canada, and bedded iron ores, in Alabama, and of France, are typical of shallow deposits. Such deposits can be rather thoroughly explored, in advance of mining, by core drilling, or by churn drilling, from the surface. The porphyry copper deposits of the western United States usually have a large lateral extent as compared with their vertical extent and can be thoroughly sampled by drilling prior to commencement of mining.

For discussion purposes herein stoping systems will be divided into the following classifications based upon the support methods used:

(1) Pillar-supported stopes (open stopes).
(2) Timber and/or fill supported stopes.
(3) Slicing or caving systems.

PILLAR-SUPPORTED STOPES (OPEN STOPES)

Open stoping is usually restricted to deposits with strong wall and/or roof rocks In the Tri-State District the flat-lying zinc–lead deposits in limestone are mined by open stoping methods, with pillars left at intervals to support the roof. These deposits are located at depths of a few hundred feet and pillars from 20 to 60 ft in diameter are spaced 40–100 ft apart (center to center). The character of roof and floor, character of ore, and depth of deposit are some of the factors which determine how much ore must be left as pillars to support the roof. It has been estimated that in the Tri-State District about 15 per cent of the ore was left as pillars.[1]

Figure 5 shows methods typical of those employed in large ore bodies whose roofs and walls are strong enough to permit open stoping. Such operations have become highly mechanized, and heavy excavating and haulage machinery is employed. High rates of production and very low costs are obtained with these methods.

FIG. 1. Slusher scraping ore to an ore pass. (Ingersoll-Rand Co.)

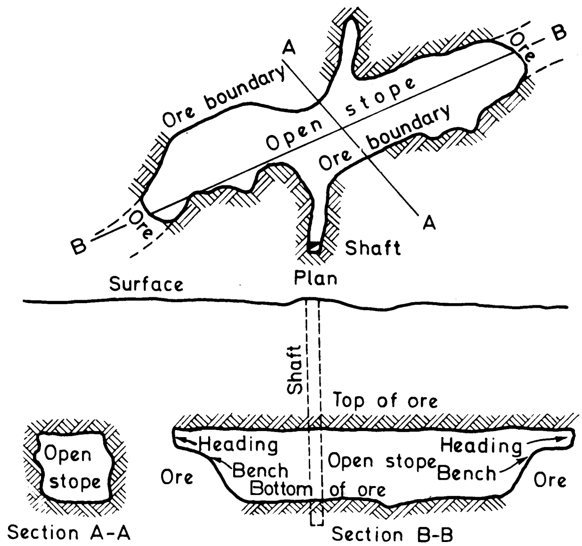


FIG. 2. Open stoping without pillars in a small ore body.(8)

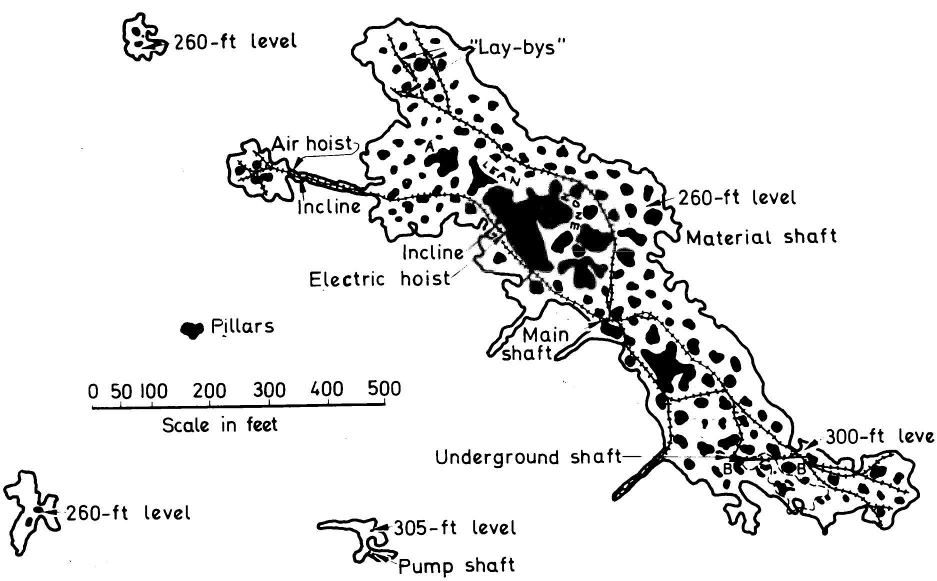


FIG. 3. Open stoping with casual pillars.(8)

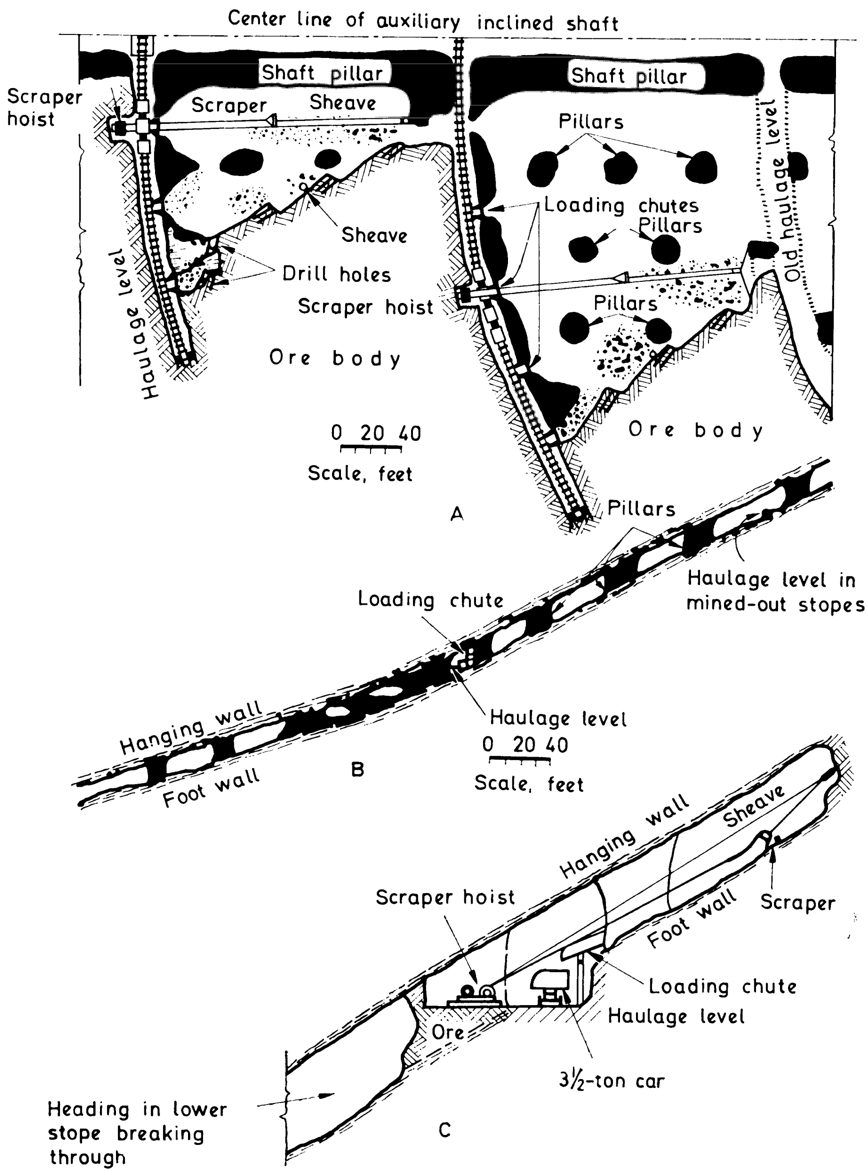


FIG. 4. Scraper stopes, Mineville, N.Y., (A) Plan of a stope. (B) Cross-section through stope. (C) Enlarged cross-section showing scraping to chute.[8]

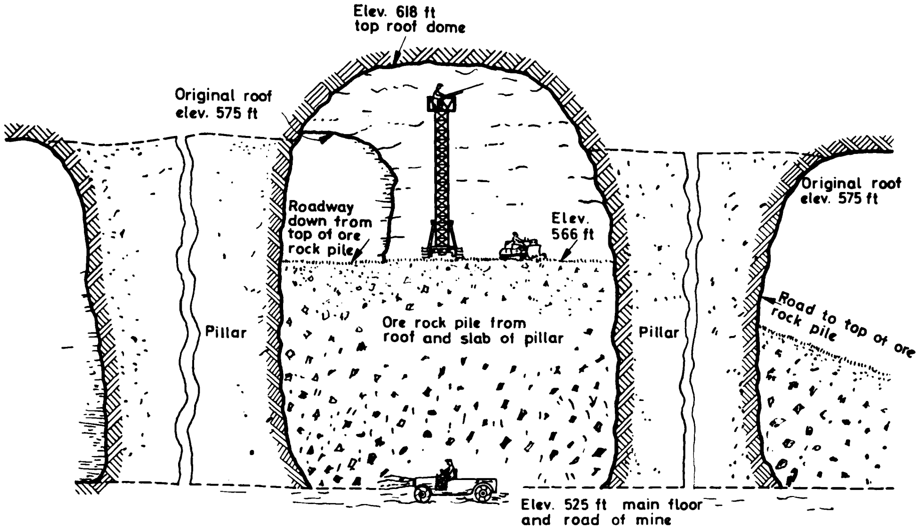


FIG. 5. A mining view in the Westside Mine.(2)

TABLE 1. SUMMARY OF COSTS(2)

Name of mine: Westside. Period covered: Year 1954

Tons of ore hoisted during period: 241,616 Mining method: Open stopes with pillar support

(1) Underground costs per ton of ore hoisted:

	(1) Labor*	(2) Supervision†	(3) Compressed-air drills and steel	(4) Power	(5) Explosives	(6) Other supplies	(7) Total
Development	—	—	—	—	—	—	—
Mining	$0.543		$0.078	$0.006	$0.134	$0.073	$0.834
Transportation (underground)	0.123					0.075	0.198
General underground expense	0.061			0.006		0.066	0.133
Surface expense (directly applicable to underground operation)	0.070			0.006		0.028	0.104
Total	$0.797		0.078	0.018	0.134	0.242	1.269

* Includes 7.5 per cent of direct labor costs for social security, compensation insurance, and unemployment benefits.

† Included under labor.

MINING METHODS AND COSTS

Westside Mine — Cherokee, Kansas

Figure 5 illustrates the mining methods used at the Westside Mine of the Eagle-Picher Co., at Cherokee, Kansas. The mining costs at this mine (in 1954) are summarized in terms of dollars per ton in Table 1 and costs are summarized in terms of units of labor, power, and supplies in Table 2.

TABLE 2. SUMMARY OF COSTS IN UNITS OF LABOR, POWER, AND SUPPLIES(2)

Name of mine: Westside. Period covered: Year 1954.
Tons of ore mined and hoisted: 241,616 Mining method: Open stopes with pillar support

	Mining
A. *Labor (man-hours per ton)*	
Breaking (drilling, blasting, trimming)	0.124
Loading	0.062
Haulage and hoisting	0.082
Supervision	0.021
General	0.075
Total labor	0.364
Average tons per man-shift (underground)	24.91
Average tons per man-shift (surface)	193.6
Average tons per man-shift (total)	21.97
Labor, percent of total cost	62.8
B. *Power and supplies*	
Explosives (40 % strength) (lb. per ton)	0.4435
Total power (kWh. per ton)	3.76
(1) Air compression — 1.25	
(2) Hoisting — 1.26	
(3) Pumping — 0.94	
(4) Ventilation, Lighting — 0.31	
Other supplies in percent of total supplies and power	51.3
Supplies and power (percent of total cost)	37.2

Zinc Mine — Jo Davies County, Ill. (Gray Mine)

Figure 6 shows the methods used in room-and-pillar open stoping in a lead–zinc orebody which was about 150 ft beneath the surface at its shallowest end and about 500 ft beneath the ground surface at its deep end. This mine was located in Jo Davies County, Illinois.

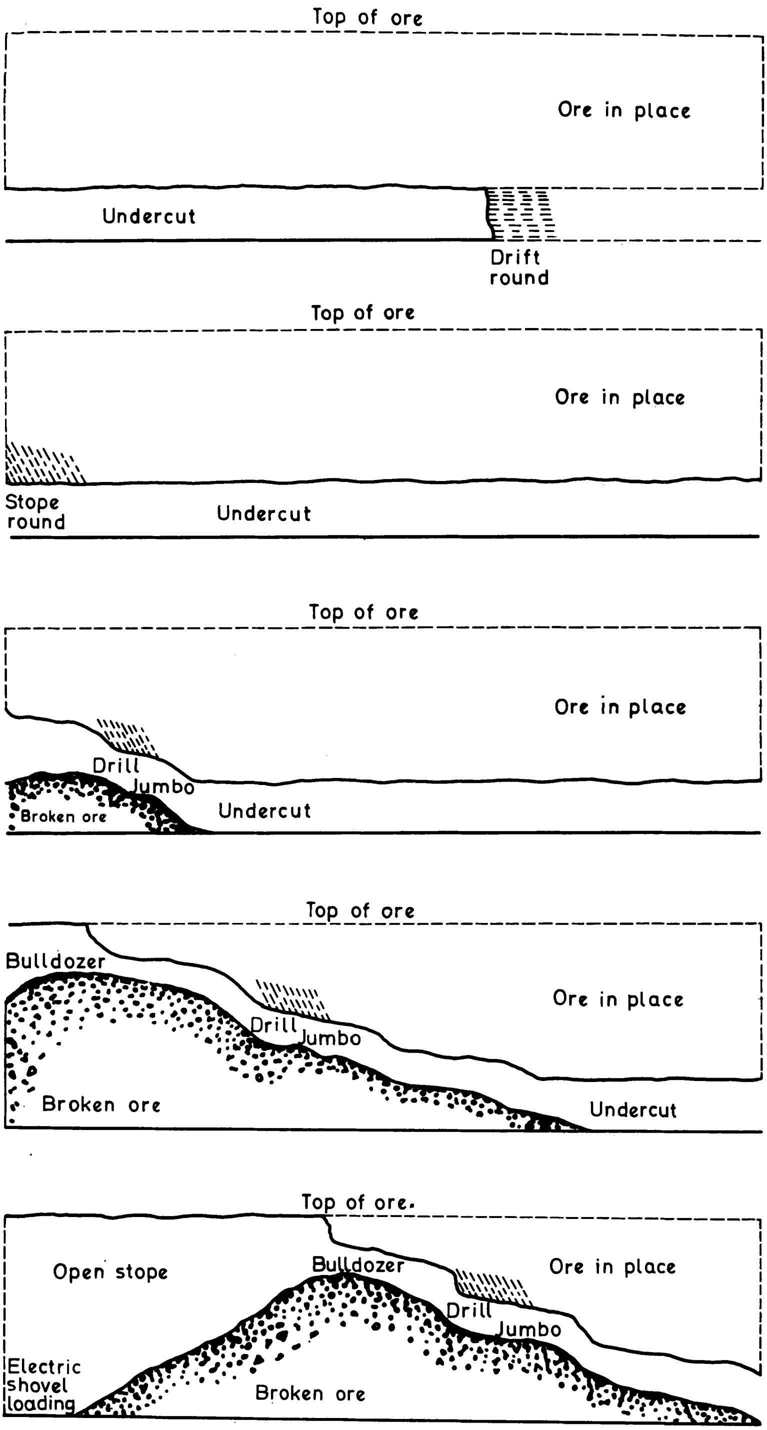


FIG. 6. Sequence of steps in room-and-pillar open stoping (Gray Mine).[3]

Table 3 shows the mine organization; Table 4 lists the equipment used in mining operations; Tables 5 and 6 list the mining costs and productivity respectively.

TABLE 3. MINING CREW (GRAY MINE)[3]

Mine superintendent	1
Foremen	2
Jumbo operators (drillers)	8
Diamond drillers	2
Shovel operators	4
Truck drivers	4
Bulldozer operator	1
Powdermen	2
Compressor-man	1
Maintenance-men (shop)	3
Total	28

TABLE 4. SUMMARY—EQUIPMENT, POWER, AND SUPPLIES (GRAY MINE)[3]

1	24,000 cfm, low-pressure type fan.
1	800 ft^3 compressor
1	360 ft^3 compressor (portable).
1	6-in. centrifugal pump.
1	8-in. American turbine (deep-well type).
4	two-jib drill jumbos, crawler-mounted.
1	HD-5 Allis-Chalmers front-end loader.
1	104 Eimco Rocker Shovel.
1	105 Eimco Rocker Shovel.
2	$\frac{1}{2}$-yard Koehring electric shovels.
1	$\frac{1}{2}$-yard Bay City electric shovel.
5	6-yard Koehring Dumptors.
2	LeTourneau-Westinghouse Tournapull Rear-Dumps.

Summary of Costs[3]

The tabulation in Table 5 covers operating, mining and milling costs at the Gray mine of Tri-State Zinc, Inc., for the fiscal year ended June 30, 1955. Costs do not cover management, mine-office, and miscellaneous general expenses, such as depreciation, depletion, taxes, and insurance (other than compensation and social security).

Costs are based on the tonnage of dried ore mined and milled. Two mining methods were employed — breast stoping and room-and-pillar open stoping, but costs are not broken down for the two methods. Current exploration and development costs are included.

TABLE 5. SUMMARY OF COSTS, IN DOLLARS (GRAY MINE)[3]

Gray Mine Operation

Tons mined: 305,181 (short tons, on dried basis) — Period covered: Fiscal year ended June 30, 1955

Mining		
Breaking		
drilling		
labor	0.1397	
compressor	0.0434	
drill steel	0.0188	
repairs and supplies	0.1045	
miscellaneous	0.0009	
Total drilling	0.3073	
blasting		
labor	0.0295	
explosives	0.1674	
repairs and supplies	–	
miscellaneous	–	
Total blasting	0.1969	
Total breaking		0.5042
loading, haulage and general	0.5184	
Pumping		
labor	0.0007	
power (electric)	0.0568	
repairs and supplies	0.0060	
miscellaneous	–	
Total pumping		0.0635
Milling		
labor	0.3072	
power (electric)	0.1675	
crushing	0.0169	
ball mill	0.0291	
reagents	0.0930	
repairs and supplies	0.1109	
assaying	0.0036	
water supply	0.0317	
pond	0.0105	
miscellaneous	0.0144	
Total milling		0.7848
Total mining, pumping and milling		1.8709

TABLE 6. PERFORMANCE DATA — OUTPUT PER DRILL-SHIFT AND PER MAN-SHIFT (GRAY MINE)

Total tons mined	305,181	
Total drill shifts	2304	
average tons per drill shift		132.46
Total number mine man-shifts	8431	
average tons per man-shift		36.21

TABLE 7. SUMMARY OF COSTS (LA SAL URANIUM MINE)[4]

Name of mine: La Sal.
Tons hoisted: 33,098 stope ore
1796 development ore
2941 development waste
37,835 total

Mining method: Panel retreating.
Period covered: October 1956 through February 1957

Summary of direct costs in units of labor, power, and supplies:

	Development*	Stoping†	Total‡
Labor:			
production, man-hour per ton:			
drilling, blasting, barring	0.34	0.24	0.25
scraping	0.15	0.17	0.17
loader-machine mucking	0.05		0.01
hand mucking	0.03	0.01	0.01
roof bolting	0.03	0.03	0.03
general	0.13	0.05	0.06
Total	0.73	0.50	0.53
production (tons per man-shift)	11.0	16.0	15.2
other, underground (tons per man-shift):			
general, bull gang§			120.5
haulage			73.5
hoisting and caging			73.6
supervision**			71.0
all labor, underground (tons per man-shift)			8.7
surface labor, properly chargeable to underground operation (tons per man-shift)			62.1
all labor and supervision, surface and underground (tons per man-shift)			7.6

* Development unit costs were calculated on a basis of 1796 tons of development ore and 2941 tons of development waste. Waste tons were included to show true costs, as the waste varies considerably each month.

† Stoping unit costs were calculated on 33,098 tons of stope ore.

‡ Total unit costs are prorated averages for total ore and waste produced from development and stoping.

§ Miscellaneous underground work including long-hole drilling.

** Includes full time of foreman and night shifter and part time of engineer, sampler, asst. superintendent, and superintendent chargeable to La Sal operation.

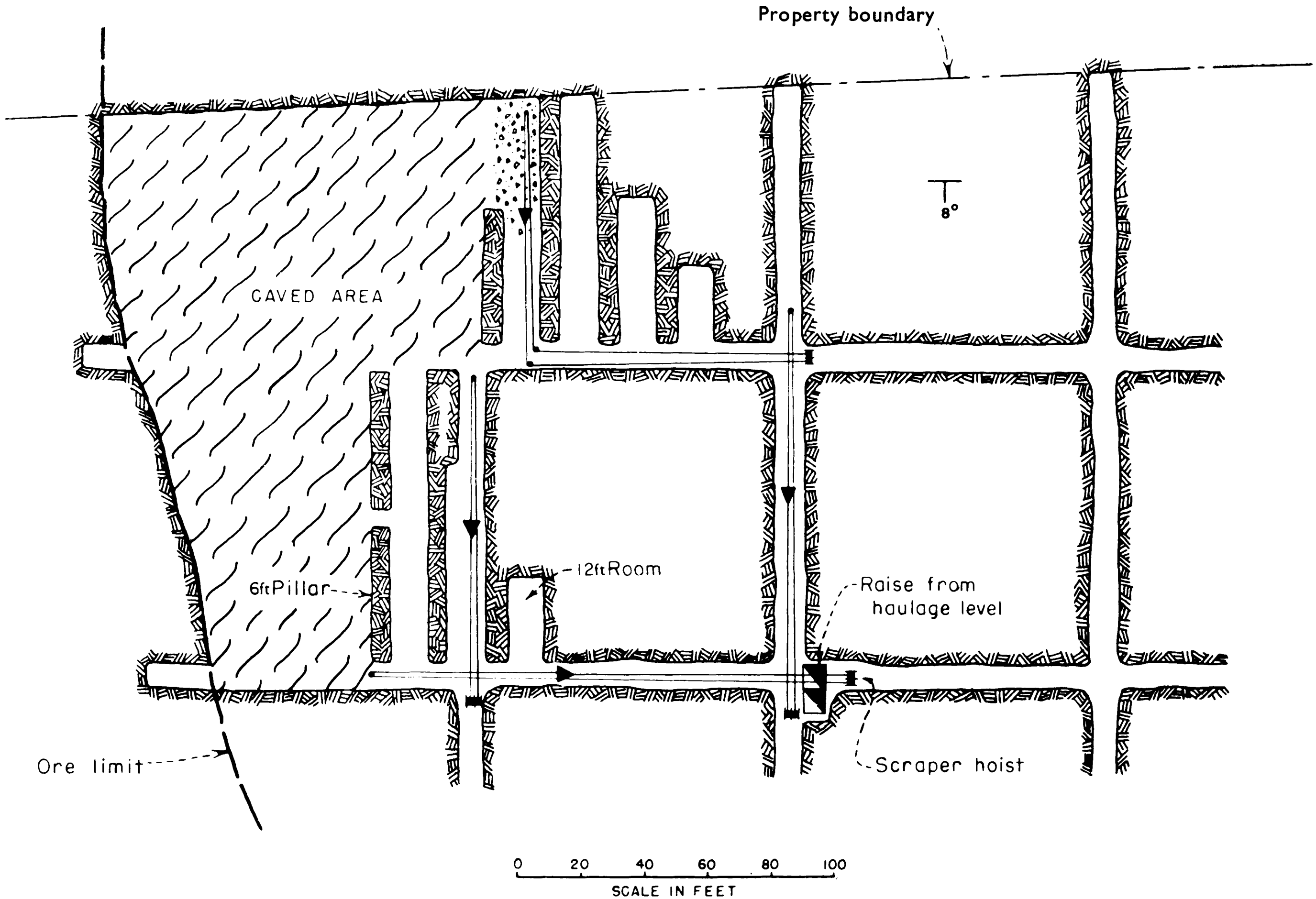
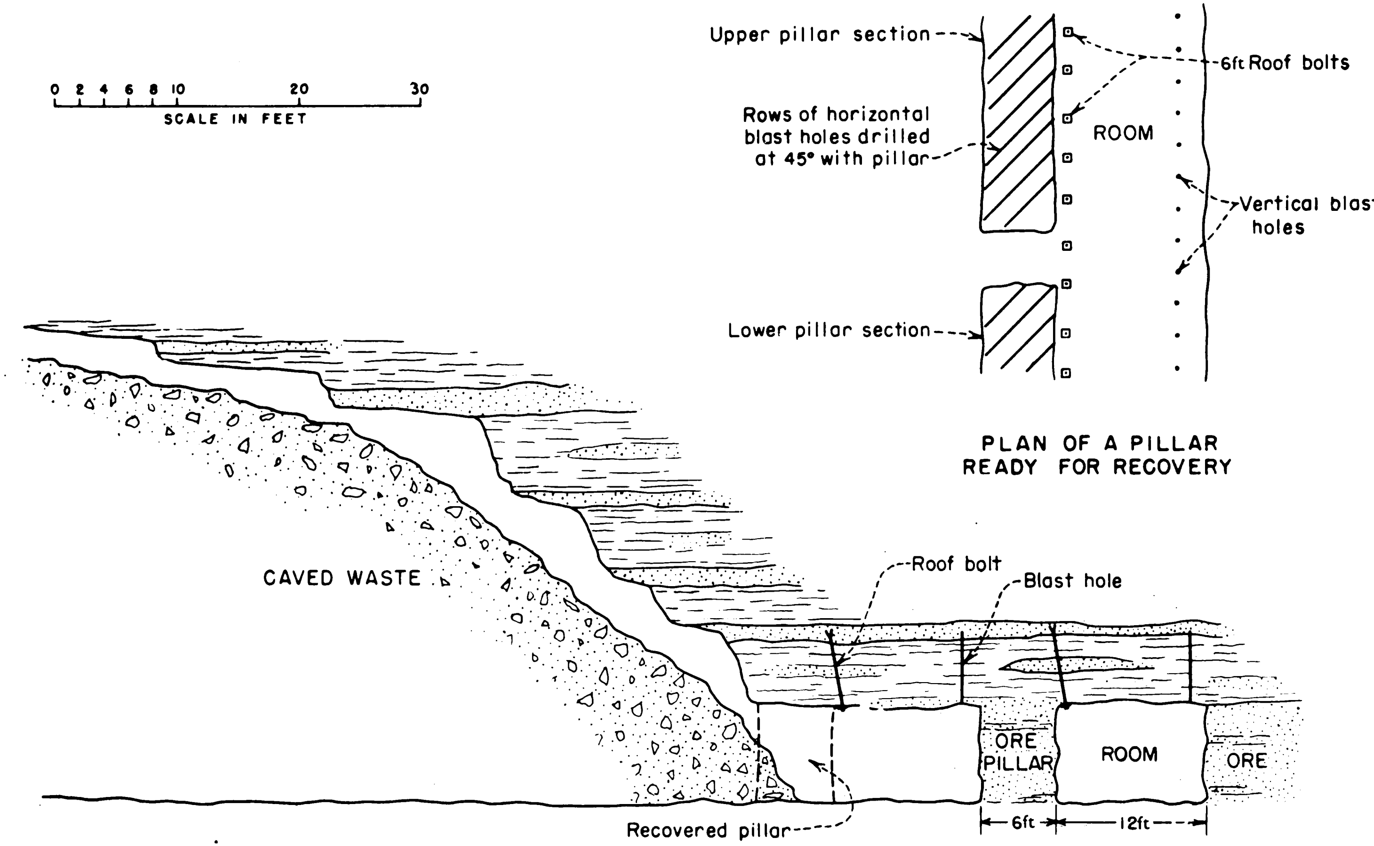


FIG. 7. Idealized sketch of mining method at La Sal Uranium Mine.(4)



After a pillar has been recovered the vertical holes along the next pillar are blasted to induce caving.

FIG. 8. Vertical section through stope blocks at La Sal Uranium Mine.[4]

La Sal Uranium Mine

Figures 7 and 8 illustrate the methods used in mining a flatlying deposit at the La Sal Uranium Mine, San Juan County, Utah. This deposit was situated about 500 ft beneath the surface. The mining method resembled a room-and-pillar system

TABLE 8. MINING COSTS AT THE CALYX No. 3 MINE[5]

Mine: Calyx No. 3
Ore hoisted: 26,803.22 tons
(1) *Direct cost per ton of ore hoisted:*

Mining method: Horizontal open stoping with casual pillars
Period: September 1955 to February 1957

	Labor	Super-vision	Com-pressed air*	Bits and steel	Explo-sives	Fuel and oil†	Water	Re-pairs and expend. equip.	Other sup-plies total	Total
Development‡	§$0.44	$0.09	$0.07	$0.03	$0.09		$0.01	$0.08	$0.03	$0.84
Mining:										
drilling and blasting	0.67	0.11	0.13	0.15	0.63		0.05	0.09		1.83
scraping and loading	1.25	0.22	0.19					0.14	0.07	1.87
tramming	0.72	0.11				$0.03		0.22	0.03	1.11
Hoisting	0.41	0.07				0.08		0.05	0.03	0.64
Equipment maintenance	0.46	0.08				0.02		0.20		0.76
Ventilation						0.05				0.05
Total	$ 3.95	$0.68	$0.39	$0.18	$0.72	$0.18	$0.06	$0.78	$0.16	$7.10

(2) *Indirect cost per ton of ore hoisted:*

Payroll taxes and workman's comp	$0.50
Utah production tax	0.11
Welfare and vacation pay	0.01
Surface structures	0.03
Engineering and surveying	0.04
Truck and traveling expense	0.18
Legal, accounting, and manager's expense	0.41
Office expense	0.06
Assaying and umpire costs	0.04
Total	$1.38

* Consists principally of fuel-oil costs.
† Fuel oil and lubricants used in equipment other than for compressor.
‡ Development costs are not unit costs per ton of development ore but are costs of development work charged against total tons mined.
§ Development labor includes drilling, blasting, scraping and tramming, and underground exploratory drilling.

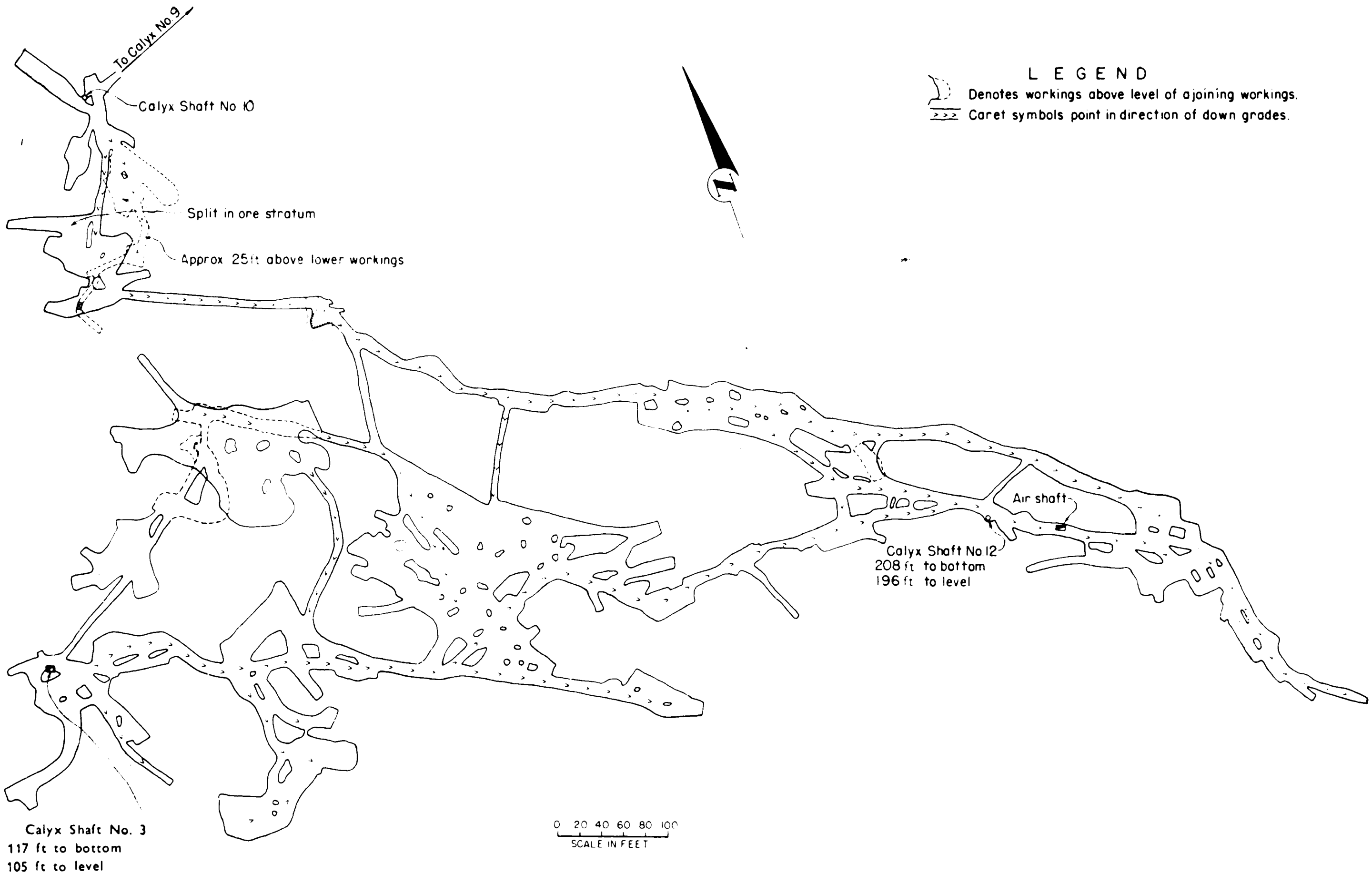


FIG. 9. Plan map of underground workings, Calyx No. 3 Mine, Emery County, Utah.[5]

with full extraction of pillars and caving of the roof on retreat. No timber was required.

Table 7 gives productivity figures for the La Sal Mine.

Calyx No. 3 Mine — Emery County, Utah

Figure 9 shows the system of open stoping with random pillar support which is used for mining irregular deposits of uranium ore in the Temple Mountain District

TABLE 9. GENERAL COST SUMMARY (HOLDEN MINE)[6]*

	1956	1955	1954	1953	1952	1951	1950	1949	1948
Dry tons milled	309,435	398,738	444,694	433,717	545,776	550,530	657,634	627,316	232,158
Mining expense:									
direct	53.3	51.5	54.8	51.5	46.4	46.9	44.7	44.8	44.5
development	2.6	3.1	3.9	6.8	5.8	6.7	7.0	7.7	9.2
Core drilling	0.5	0.2	0.6	0.9	1.5	1.4	2.1	1.7	1.5
Ore transportation (underground)	1.4	1.3	1.4	1.5	0.8	0.6	1.0	1.7	1.4
Milling expense	18.4	20.0	18.2	18.2	24.3	24.5	26.9	25.3	24.6
Surface transportation (to railhead)	5.0	5.5	4.5	4.9	5.2	4.9	4.8	4.9	4.2
Administration	4.5	4.5	4.1	4.0	4.2	4.0	3.7	3.8	4.1
Property tax†	1.2	1.1	1.0	1.0	1.0	0.9	1.0	0.9	0.9
Other	13.1	12.8	11.5	11.2	10.8	10.1	8.8	9.2	9.6
Total	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Comparative expense‡	156.2	126.5	123.3	126.9	105.4	103.7	95.7	93.9	100.0

* Percentage of total cost.
† Excluding income tax and depreciation.
‡ Base year 1948.

of Utah. These deposits are mined through shafts sunk by 36-in. diameter calyx drills.

Table 8 summarizes the mining costs at the Calyx No. 3 Mine.

Steeply Dipping Ore Deposits

Steeply dipping deposits are also mined by open stoping methods, with support by intermittent pillars. Usually such methods are feasible only down to depths of about 1000–1500 ft.

Open stopes in steeply dipping veins are sometimes classified according to whether the ore is attacked from above or from below. If stoping proceeds downward from a level, with the miners standing on unbroken ore to work, then the method may be referred to as "underhand" stoping.

TABLE 10. MINING EXPENSE SUMMARY (HOLDEN MINE)[6]*

	1956	1955	1954	1953	1952	1951	1950	1949	1948
Tons broken	253,531	425,294	349,822	381,515	479,894	501,404	393,991	452,617	416,327
Tons drawn from stopes	283,812	374,553	407,268	398,041	520,666	521,515	614,399	594,289	587,559
Tons drawn from development	23,735	25,734	38,564	38,629	24,316	29,691	39,054	35,601	22,165
Total tons drawn	307,547	400,287	445,832	436,670	544,982	551,206	653,453	629,890	609,724
Ore breaking, direct expense:									
labor	8.59	7.81	9.70	9.19	4.75	6.02	6.52	7.22	6.37
drilling and blasting	13.93	12.86	12.89	11.39	7.49	7.19	7.08	10.92	14.70
explosives	4.82	4.57	4.24	3.71	2.36	2.77	2.96	3.67	3.08
timber	1.24	0.06	0.06	0.16	0.06	0.08	0.51	0.44	0.33
other	3.29	0.23	0.23	0.30	0.17	0.11	0.13	0.04	—
Total	31.87	25.53	27.13	24.75	14.82	16.17	17.20	22.28	24.48
Ore drawing:									
labor	9.45	11.13	10.23	11.00	9.63	10.20	9.78	9.64	8.70
explosives	4.78	5.76	6.89	6.88	6.63	9.69	10.97	10.26	9.08
other	2.72	2.47	1.73	2.46	1.95	3.20	2.14	2.48	2.29
Total	16.95	19.36	18.85	20.34	18.21	23.09	23.89	22.38	20.07
Total ore breaking	31.87	25.53	27.12	24.75	14.83	16.17	17.20	22.29	24.48
Total ore drawing	16.95	19.36	18.85	20.34	18.21	23.09	23.89	22.38	20.07
Tramming	11.38	11.38	11.55	11.52	7.78	8.55	8.47	8.54	7.66
Hoisting	7.50	7.36	7.01	6.77	7.02	7.15	6.78	6.50	7.24
Mine general expense	26.71	30.16	25.39	21.93	19.13	21.60	21.35	21.14	21.02
Development	5.59	6.21	10.08	14.69	13.83	17.93	22.31	19.15	19.53
Other	—	—	—	—	19.20	5.51	—	—	—
Total mining expense	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Comparative expense†	168.32	129.43	139.51	160.32	130.56	106,58	98.29	94.13	100.00

* Percentage of total expense.
† Compared with base year 1948.

If stoping proceeds upward from a lower level then the method may be designated as "overhand" stoping. The back may be carried upward either as a series of steps, or the back may be carried up in a series of inclined slices, or in a series of horizontal slices.

TABLE 11. LABOR DISTRIBUTION AND PRODUCTION, JANUARY 1 TO DECEMBER 31, 1955 (HOLDEN MINE)[6]

		Man-shifts per day			Man-hours per day			Man-hour per ton		
		Production	Development	All	Production	Development	All	Production	Development	All
Breaking		8.9	14.1	23.0	71.2	112.8	184.0	0.0540	0.0856	0.1396
Timbering		8.8	2.3	11.1	70.4	18.4	88.8	0.0534	0.0140	0.0674
Mucking		14.4	11.5	25.9	115.2	92.0	207.2	0.0874	0.0698	0.1572
Hauling and hoisting		17.6	6.3	23.9	140.8	50.4	191.2	0.1068	0.0382	0.1451
Supervision		3.3	3.4	6.7	26.4	27.2	53.6	0.0200	0.0206	0.0406
Crushing		1.8	—	1.8	14.4	—	14.4	0.0109	—	0.0109
Diamond drilling		—	2.2	2.2	—	17.6	17.6	—	0.0134	0.0134
Maintenance		—	—	13.2	—	—	105.6	—	—	0.0801
Total underground labor		54.8	39.8	107.8	438.4	318.4	862.4	0.3326	0.2416	0.6543
Chargeable surface labor		—	—	43.0	—	—	344.0	—	—	0.2609
Total labor				150.8			1206.4			0.9153
Tons broken	425.294									
Tons drawn	400.287									
Work days	303.7									
Avg. daily production	1318.03									
Avg. daily underground shifts	107.8									
Avg. chargeable surface shifts	43.0									
Tons per man-shift:										
underground		12.23								
surface		30.65								
overall production		8.74								

SUB-LEVEL STOPING

In very wide veins, dipping at steep angles in strong rock, the method of "sub-level" stoping may be employed. This involves the driving of sub-level drifts, and cross-cuts within the orebody. Miners working within these sub-level openings drill a pattern of holes which allows successive vertical slices of ore to be blasted off, the ore falling into the large open stope created by previous blasts.

TABLE 12. JOB CLASSIFICATIONS AND WAGE SCALES (HOLDEN MINE)[6]

Job classifications			Wage scales	
Mine	Shops	Surface	Effective Feb. 1. 1957*	Effective July 1, 1957
Shaft miner (senior)	Mechanic (senior), electrician (senior)	—	$19.87	$20.55
—	—	Power-shovel operator	19.41	20.07
Shaft miner, timber framer, diamond driller, hoist-man	Welder, mechanic, blacksmith, elec-trician, machinist	Heavy-equipment ope-rator, mechanic, carpenter, painter, plumber, first cook	18.95	19.59
Longhole loader	—	—	18.49	19.11
Miner (raise, drift, stope), longhole driller, timber-man	Bit sharpener	Heavy-truck driver	18.03	18.63
Mucking-machine operator (production), motorman (main level), slusher oper-ator (production), trackman	Welder, mechanic, electrician, black-smith (second class), machinist	Mechanic, carpenter (second class)	17.57	18.15
Cage or skip tender, pipe-man, mucking machine operator (development)	—	Baker	17.11	17.67
Slusher operator (develop-ment), fuseman, motor-man (sub-level), brakeman, powder nipper	Lampman	Truckdriver, hook-tender, oiler	16.65	17.19
Helpers, mucker, nipper	Helpers	Second cook	16.19	16.71
	—	Watchman	15.73	16.23
—	—	Surface labor, dish-washer, waiter, janitor	15.27	15.75

* Retroactive to July 1, 1956.

Figures 10, 11 and 12 illustrate "sub-level stoping" as practiced at the Holden Mine, Chelan County, Washington. A copper–zinc orebody about 2500 ft long, extending to a depth of about 2500 ft, and with mineralized widths up to 80 ft, was mined between 1938 and 1957. About 10 million tons of ore were produced.

The ore zone was enclosed in quartz amphibole schist – a very strong rock – which favored the use of open stoping methods.

Originally ore was broken into the stopes by "coyote hole" or "powder blast" mining. This method involved the driving of undersized drifts in the ore zone

TABLE 13. DISTRIBUTION OF ELECTRIC POWER (HOLDEN MINE)[6]*

	1956	1955	1954	1953	1952	1951	1950	1949	1948
Dry tons milled	309,435	398,738	444,694	433,717	545,776	550,530	657,634	627,316	608,431
Mine:									
compressors	10.38	9.14	8.83	9.14	7.33	7.79	7.05	6.56	7.57
hoists	2.34	2.07	2.07	2.13	1.92	2.10	1.98	1.99	1.51
slushers	0.35	0.29	0.24	0.21	0.23	0.72	1.23	0.91	0.50
pumps	2.23	2.04	2.59	2.69	2.51	1.94	1.79	0.95	0.74
ventilation	4.65	3.23	2.33	2.53	2.12	2.36	2.17	1.80	1.99
crusher	0.50	1.30	1.06	1.92	1.54	0.70	0.38	0.23	0.17
tramming	1.02	0.83	0.71	0.86	0.72	0.67	0.53	0.64	0.67
stope fill	–	–	0.32	0.70	1.29	0.59	1.17	0.74	–
lighting and other	0.61	0.47	0.43	0.41	0.23	0.18	0.13	0.18	0.14
Total	22.08	19.37	18.58	20.59	17.89	17.05	16.43	14.00	13.29
Mill:									
coarse crushing	2.08	2.55	2.74	3.03	2.85	2.73	2.69	2.64	3.06
fine grinding	12.57	12.04	11.79	11.49	12.97	12.50	12.19	12.53	13.36
flotation	6.21	5.39	5.21	5.84	5.57	5.03	4.11	4.46	5.13
other	1.92	1.77	1.78	2.48	1.68	1.71	2.97	4.38	3.84
Total	22.78	21.75	21.52	22.84	23.07	21.97	22.68	24.01	25.39
General, surface	8.77	6 60	5.99	5.82	4.83	4.30	3.55	3.32	3.47
Total	53.63	47.72	46.09	49.25	45.79	43.32	42.66	41.33	42.15

* kWh per dry ton milled.

which were then filled with explosives and blasted. Because of the shattering effect of these large blasts, as well as other factors, the method was abandoned in favor of longhole blasting. This is the method shown in Figs. 11 and 12.

Tables 9–18 give a comprehensive breakdown of mining expense, labor productivity, wage scales, and power and powder consumption, etc., at this mine.

TABLE 14. POWDER CONSUMPTION (HOLDEN MINE)[6]

	To date	1956	1955	1954	1953	1952	1951	1950	1949	1948
Ore broken:										
by longholes	4,846,562	193,367	394,211	303,037	328,562	452,959	462,715	341,215	409,947	380,169
from other sources	3,700,089	60,164	31,083	46,775	52,953	26,935	58,800	52,776	42,670	36,153
Total	8,546,651	253,531	425,294	349,822	381,515	479,894	521,515	393,991	452,617	416,322
Ore drawn:										
from stopes	—	283,812	374,553	407,268	398,041	520,666	521,515	614,399	594,289	587,559
from development	—	23,735	25,734	38,564	38,629	24,316	26,691	39,054	35,601	22,165
Total	10,188,383	307,547	400,287	445,832	436,670	544,982	551,206	653,453	629,890	609,724
Ore broken (tons)	8,546,651	253,531	425,294	349,822	381,515	479,894	521,515	393,991	452,617	416,322
Ore drawn (tons)	*10,188,383	307,547	400,287	445,832	436,670	544,982	551,206	653,453	629,890	609,724
Apparent dilution (%)	19.5	21.3	−5.9	27.4	14.5	13.6	5.7	65.9	39.17	46.5
Mine development (ft)	—	1262	1315	2298	3827	2795	3814	4835	5611	6077
Stope development (ft)	—	5709	5600	8023	7316	3844	4816	7006	8071	4607
Exploration (ft)	—	300	3846	1086	1017	1144	2344	2308	—	—
Total (ft)	—	7271	10,761	11,407	12,160	7783	10,974	14,149	13,682	10,684
Primary breaking powder factor†	0.23	0.45	0.33	0.32	0.28	0.18	0.14	0.18	0.18	0.26
Secondary breaking powder factor‡	—	0.85	0.84	1.03	1.29	1.13	1.30	§	§	§
Total powder for production	—	1.16	1.19	1.26	1.51	1.29	1.42	—	—	—
Powder per foot of development	—	29.3	22.3	25.63	23.9	23.8	27.4	§	§	§
Total powder consumption per ton production	—	1.76	1.71	1.81	2.04	1.57	1.89	2.12	1.88	§

* Plus 24,985 tons reserve.
† Pound per ton broken.
‡ Pounds per ton drawn.
§ Not available.

APPLICABILITY OF OPEN STOPING METHODS

The open stoping methods described above are obviously most efficient in those ore deposits which have strong walls and/or roof, since the amount of ore which must be left in pillars varies with the strength of ore, as well as strength of walls, and/or roof.

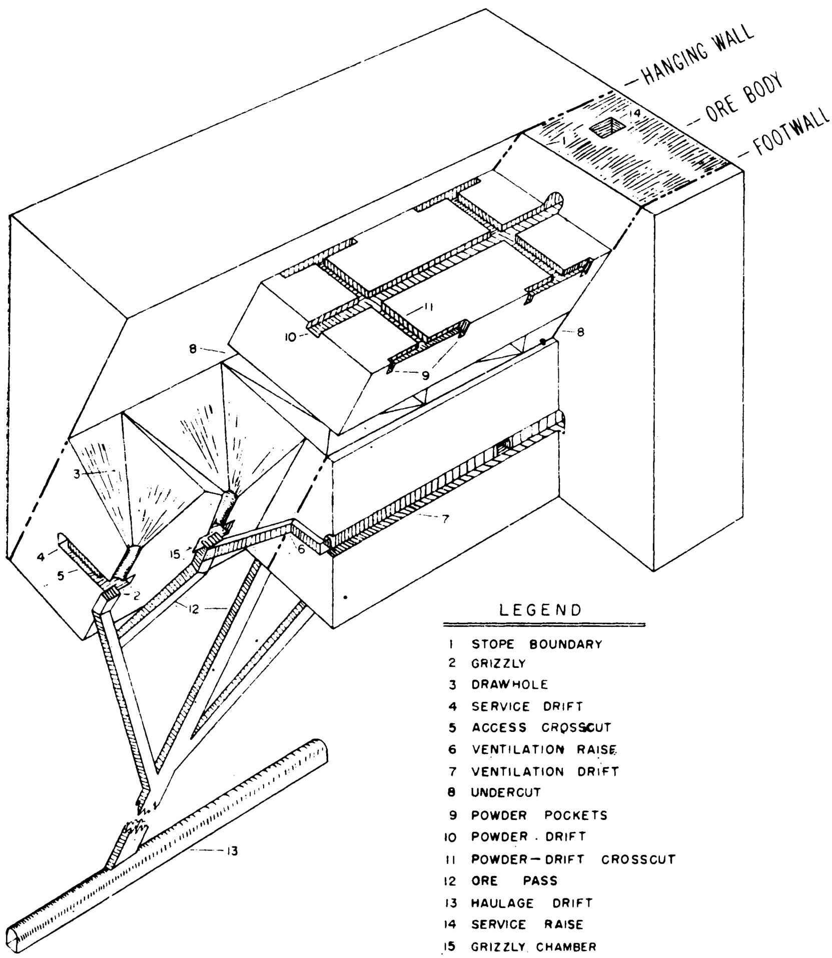


FIG. 10. Idealized powder-blast stope development.[6]

Success of open stoping methods depends also on the depth of mining and pre-existing stress conditions in the rock. At great depth even strong roof and/or walls will fracture, and quickly close any sizable unfilled opening.

In flat-lying deposits stopes may be left unfilled and no harm is done if eventually they cave to the surface provided that the surface is not built upon. If stopes in

TABLE 15. BLASTHOLE

	To date	1956	1955
Drilled (ft)	1,362,315	83,588	130,831
Blasted (ft)	1,292,798	83,745	133,807
Broken (tons)	4,846,562	193,367	394,211
Blasted (tons/ft)	3.75	2.31	2.95
Powder (lb)	1,099,871	87,600	131,550
Powder (lb/ton)	0.23	0.45	0.33

TABLE 16. APPORTIONMENT OF LONGHOLE DRILLING COSTS (HOLDEN MINE)

	Since inception	1956	1955
Ore breaking (ft)	664,169	83,588	130,831
Development (ft)	60,587	4148	6412
Total drilled (ft)	726,493	87,736	137,243
Bits used	3835	337	452
Feet per bit	189.4	260.3	303.6
Direct expense:*			
labor	58.65	60.91	64.20
new bits	14.29	11.27	11.78
grinding	2.44	2.72	2.77
couplings	3.20	2.18	3.23
new drills	3.76	–	2.54
new rods	5.64	6.92	4.16
rod maintenance	1.69	0.55	0.93
drill repair	5.08	5.45	5.77
other labor and supplies	5.26	10.00	4.62
Total	100.00	100.00	100.00
	72.98†	76.39†	72.90†
Air	14.13	13.89	14.14
Air and water lines	12.89	9.72	12.96
Total	100.00	100.00	100.00
Total drilling expense‡	§	17.82	19.10

* Percentage of direct drilling expense.
† Percentage of total drilling expense.
‡ Percentage of total mining expense.
§ Not available.

PERFORMANCE (HOLDEN MINE)

1954	1953	1952	1951	1950	1949	1948
106,296	75,205	65,721	49,657	48,112	84,638	71,889
85,243	70,415	76,091	52,289	48,963	68,290	82,777
303,037	328,562	452,959	462,715	341,215	409,947	380,169
3.53	4.67	5.95	8.85	6.97	6.00	4.59
95,090	87,750	83,276	64,100	62,375	75,575	98,465
0.32	0.28	0.18	0.14	0.18	0.18	0.26

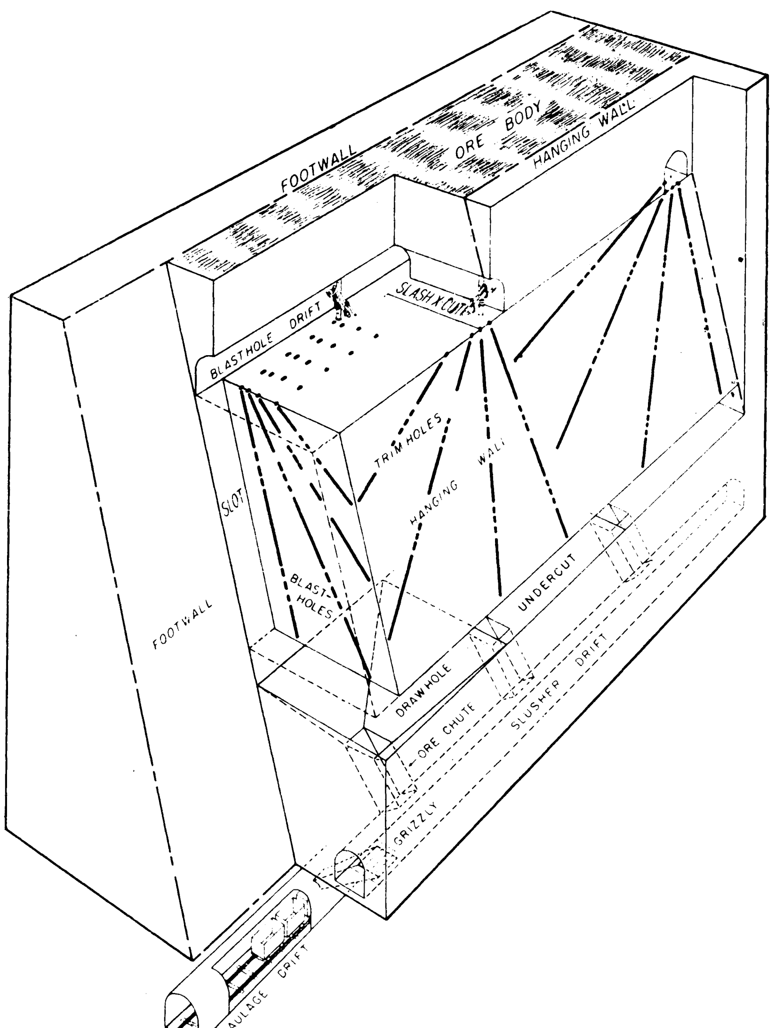


FIG. 11. Idealized longhole stope.[6]

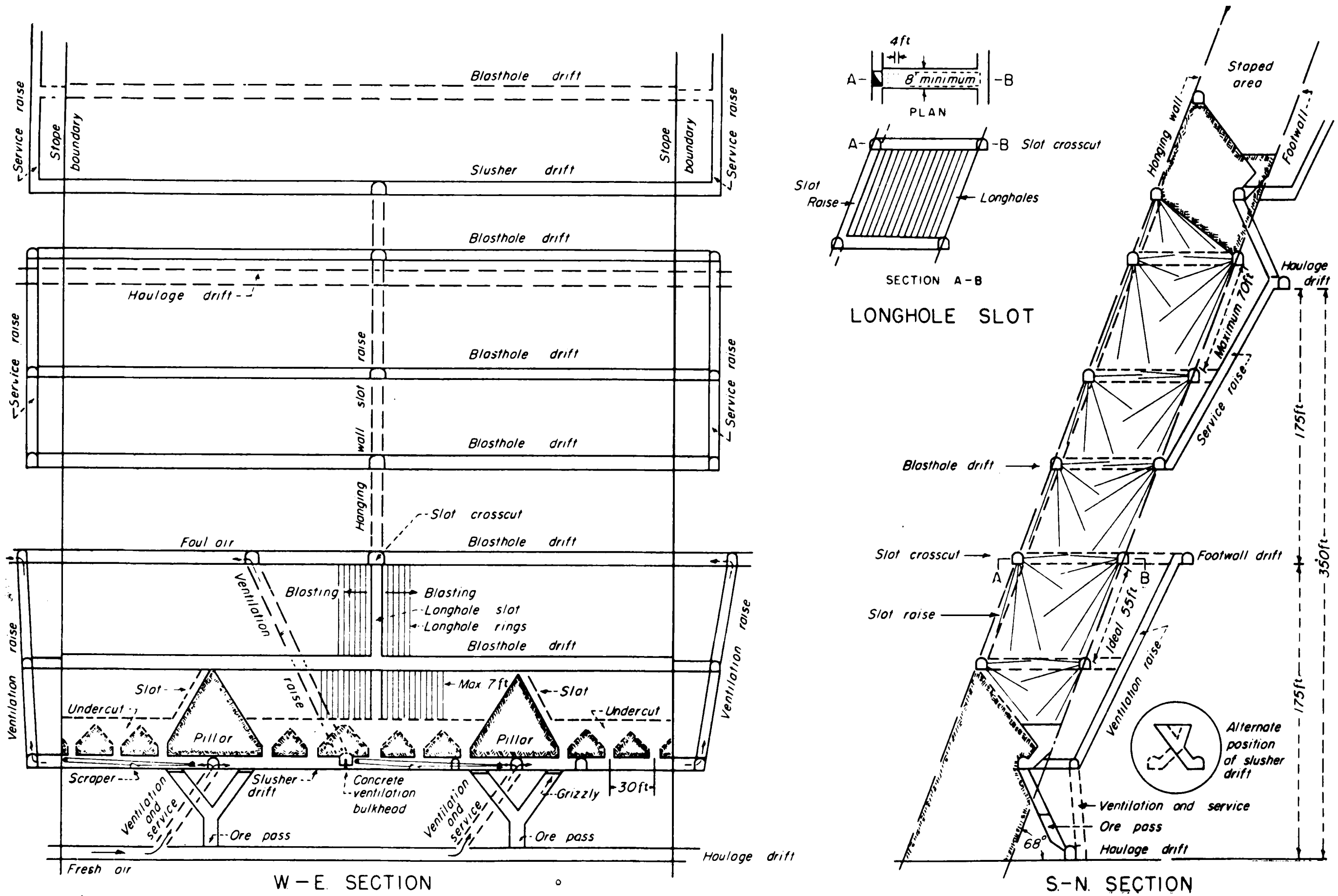


FIG. 12. Generalized stope development, Holden Mine, Chelan County, Wash.[6]

TABLE 17. APPORTIONMENT OF COSTS* AND COMPARISON OF ORE-DRAWING METHODS (HOLDEN MINE)[6]

	Slushing to ore passes			Slushing to cars			Mucking into cars		
	1956	1955	1954	1956	1955	1954	1956	1955	1954
Ore drawn (tons)	118,103	325,573	201,570	41,349	—	33,701	33,915	68,638	67,766
Labor	32.26	37.43	37.24	42.82	—	56.35	53.71	61.40	56.81
Supplies	2.45	1.23	1.40	0.14	—	—	—	—	—
Explosives	28.44	31.81	41.60	25.18	—	31.55	29.71	22.40	21.48
Power	0.46	0.35	0.35	—	—	0.40	—	—	—
Maintenance:									
mechanical	16.67	16.17	16.26	5.97	—	11.70	—	—	—
general	19.72	13.01	3.15	25.89	—	—	—	—	—
Other	—	—	—	—	—	—	16.58	16.20	21.71
Total	100.00	100.00	100.00	100.00	—	100.00	100.00	100.00	100.00
Cost of ore drawing†	16.18	18.30	17.06	17.40	—	15.04	22.99	23.83	25.42
Tons per hour per unit	34.6	35.8	‡	22.4	—	‡	15.5	14.0	12.2

* Percentage of ore-drawing expense.
† Percentage of mining expense.
‡ Not available.

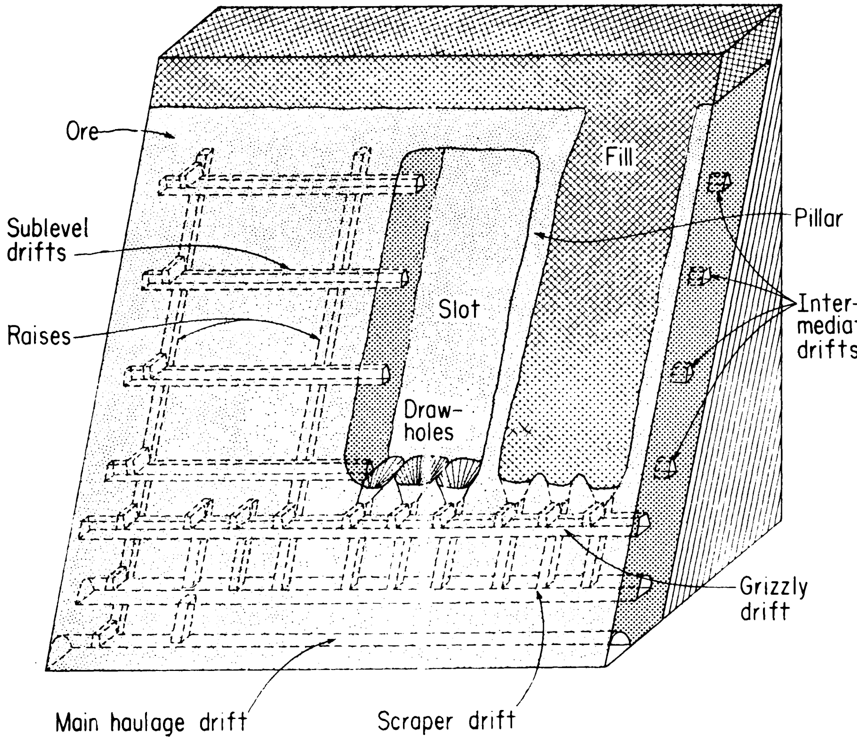


FIG. 13. Sub-level stoping method used in mining the "brown iron ore" of the Salzgitter District near Braunschweig, Germany.

steeply dipping deposits are not filled, caving eventually occurs as mining proceeds to greater depths. The resulting disturbances to hanging-wall, and sometimes to foot-wall also, may affect access openings. If the orebody is known to extend to great depths, it may be desirable to mine the upper portions by fill methods in order to avoid later disturbances due to caving of open stopes.

TABLE 18. APPORTIONMENT OF TRANSPORTATION COSTS, 1500 LEVEL (HOLDEN MINE)[6]

	1956	1955	1954
Ore hauled (tons)	309,657	398,556	444,808
Cars	39,312	51,943	58,736
Tons per car (dry)	7.88	7.81	7.57
Direct operating costs:*			
labor	37.17	44.21	45.92
supplies	–	–	0.14
power:			
air	4.09	5.00	5.12
electric	2.32	2.50	2.49
Total direct operating costs	43.58	51.71	53.67
Track and rolling stock maintenance	56.42	48.29	46.33
Total, 1500 level transportation costs	100.00	100.00	100.00
Cost of 1500 level transportation†	19.65	21.47	18.69
Cost of 1500 level transportation‡	2.24	2.44	2.16
All tramming costs as percentage of mine expense	11.38	11.38	11.55

* Percentage of transportation costs, 1500 level.
† Percentage of total tramming cost.
‡ Percentage of total mine expense.

Since open stoping methods are most efficiently applied to flat-lying deposits at depths of not more than a few hundred feet, or to steeply dipping deposits at depths of less than about 1500 ft, in strong rock, they tend to revert to filled stoping methods when shallow ore is worked out and mining extends to greater depths.

Open stoping methods provide a low cost mining method by sacrificing part of the ore in pillars for support. The method requires strong walls and/or roof, and strong ore, for pillars.

The open stoping methods are most efficient in tabular deposits with regular, well-defined walls, but are sometimes used in large massive deposits with irregular walls.

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CHAPTER 2

TIMBER AND FILL SUPPORTED STOPES

WHEN the walls and/or roof of an orebody are weak or "slabby" then some means of support must be provided to prevent waste rock from falling into the stope, or from squeezing in and closing up the stope.

In the past, many near-surface mining operations have used timber alone for support. In relatively narrow veins support may be provided by timber stulls, caps, or by cap-and-post timbering. The simplest type of timber support is the stull, wedged between hanging-wall and foot-wall, and installed either at random, or in a pattern. Figure 1 illustrates stull stoping in a narrow vein and this system is satisfactory for narrow, near-surface ore deposits where the walls are strong and do not slab or squeeze excessively.

TIMBERED STOPES

For stope openings whose width exceeds the length of a single long timber cap some system of square-set timbering has usually been employed.

Orebodies many tens of feet wide, hundreds of feet high, and hundreds of feet long have been mined out by square setting, using only the timber for support. Such methods have ordinarily been confined to orebodies located in ground which stood well and which did not extend to more than a few hundred feet of depth. Where timber only was used for support it was necessary that the orebody be mined out and the workings abandoned before the timber decayed and the whole timbering system collapsed.

Most of the shallow ore deposits have been exhausted and as mining operations progress to greater depths it becomes necessary to provide support additional to that provided by timber alone. Timber support is normally supplemented by waste rock, or sand fill, placed inside the timber sets, and carried up to within a few sets of the stope back.

For the past 25 years the trend has been toward the use of less timber and toward the use of cut-and-fill stoping methods with filling kept close to the back and with walls supported by rock bolts until filling can be placed.

During the past 10–15 years this trend away from timber support and toward straight cut-and-fill methods has been accelerated by the following factors:

(a) Development of rock bolting which allows support of extensive areas of wall and roof, either permanently, or until filling can be placed.

(b) Increasing cost of timber as timber supplies become more remote from mining areas.

(c) Increasing labor costs; a large part of timbering cost is the cost of transporting, cutting, and placing the timber.

(d) The use of larger and more efficient scrapers (slushers) for distributing waste filling in the stopes.

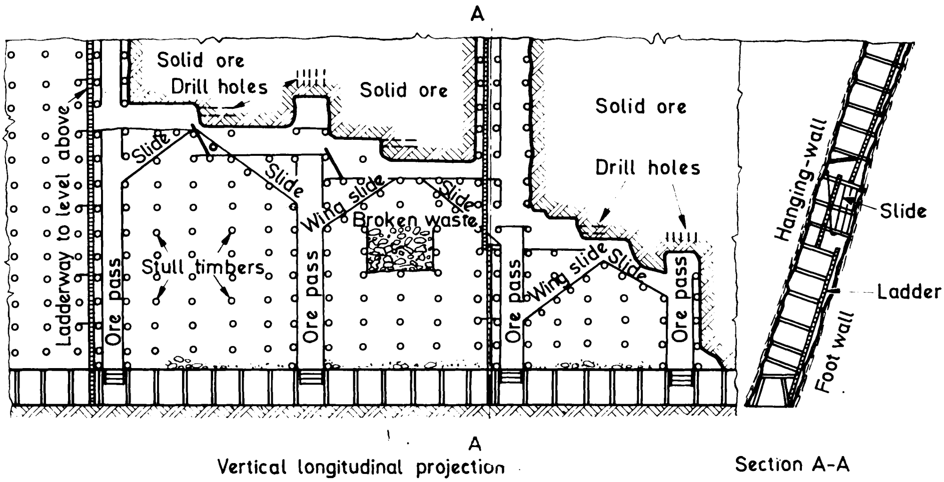


FIG. 1. Example of stull stoping in a narrow vein.[8]

(e) Renewed interest in hydraulic sand filling methods. When mill tailings are available they make the cheapest and most easily placed form of stope filling. Such filling furnishes tight support to the walls, fills voids, forms a hard smooth working floor, and is the least compressible of all types of filling.

A combination of rock bolting and hydraulic sand filling for stope support has resulted in very low mining costs in some recent instances. Rock bolts may be installed to furnish support to walls until sand filling can be run into the stope.

Attack Methods

As with open stoping methods, supported stopes are sometimes classified in accordance with the direction of attack on the ore. Nearly all systems requiring square set timbering, or cap-and-post timbering, are mined by "overhand" methods. An "underhand" stoping system known as the "Mitchell Slicing" system has on occasion been employed in wide orebodies with weak walls. This consists of cutting a wide orebody into pillars by means of narrow transverse timbered stopes, after which each pillar is mined out by working from the top down (underhand). Other combinations of square set timbering with underhand stoping have been used for special conditions, but such instances are rare; and nearly all supported stopes are mined by overhand methods.

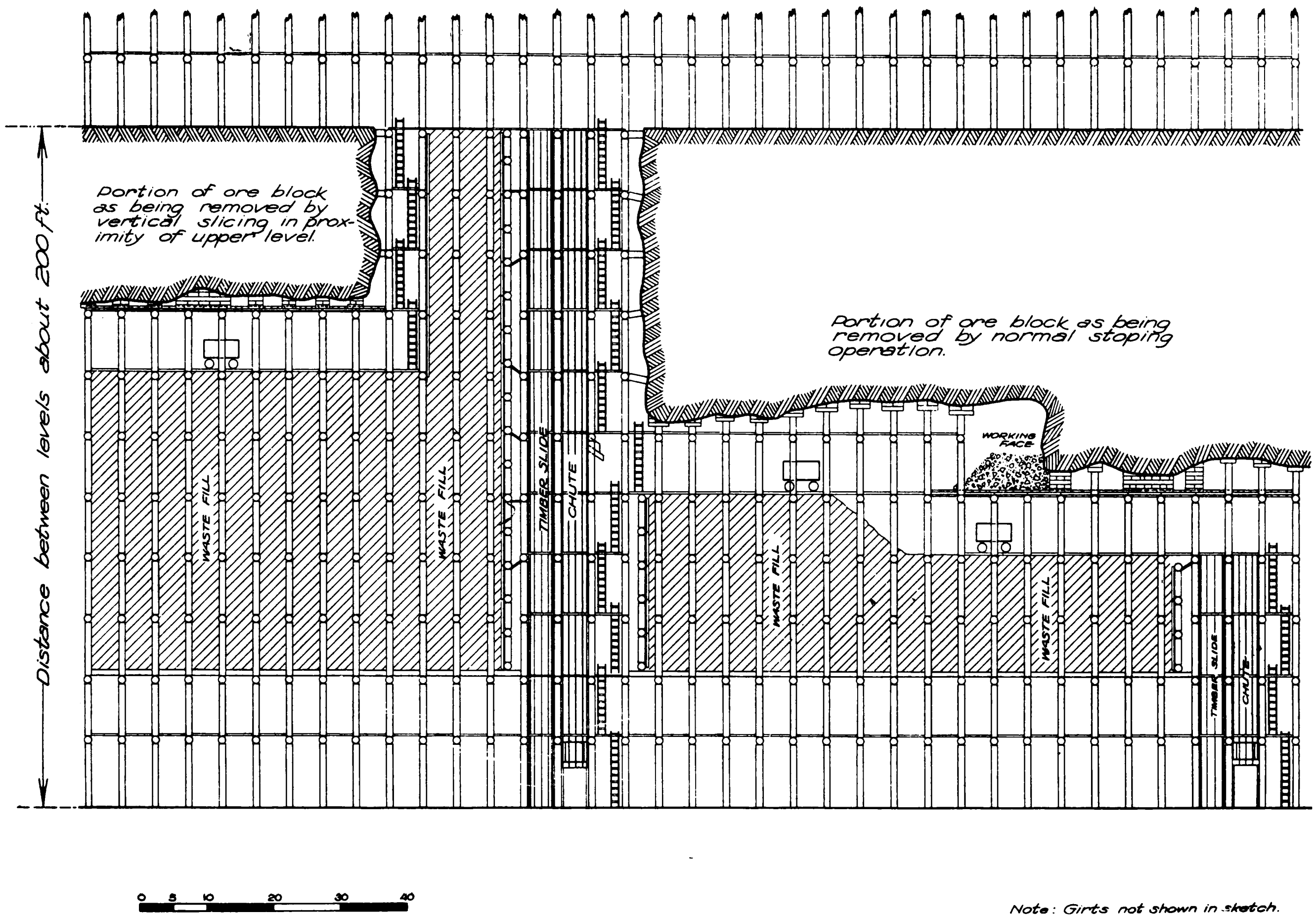


FIG. 2. Generalized sketch showing method of stope mining as formerly practiced at the Hecla Mine.(1)

TIMBER AND FILL SUPPORTED STOPES

Figure 2 shows typical cap-and-post timbering with waste filling. Figure 3 shows a typical square set-and-fill stoping system. Figure 4 shows the evolution of stoping methods during the productive life of the Morning Mine at Mullan, Idaho. The Morning vein (lead–silver–zinc) was located in a shear zone and ground was highly fractured. Vein widths varied from 3 to 40 ft, with an average of about 13 ft. Close timbering was required and heavy side pressures developed on timber. Between 1897 and 1953 the vein was stoped to a vertical depth of 6400 ft below the outcrop. Cap-and-post timbering was used throughout. The method of stoping and chute spacing originally used are shown as Method No. 1. Timber chutes were difficult to maintain and the system was modified to that of Method No. 2. With the adoption of slushers the timbering system of Method No. 3 was adopted. As soon as a cut was completed the tramming, or slushing, floor was filled with waste before another cut was started.

Mining Methods and Costs — Morning Mine

Tables 1, 2 and 3 show comparative stoping costs, supply costs, and cost distribution for Period 1 (1939) and Period 2 (August 1, 1950 to July 31, 1951).

TABLE 1. STOPING COSTS (MORNING MINE)[3]*

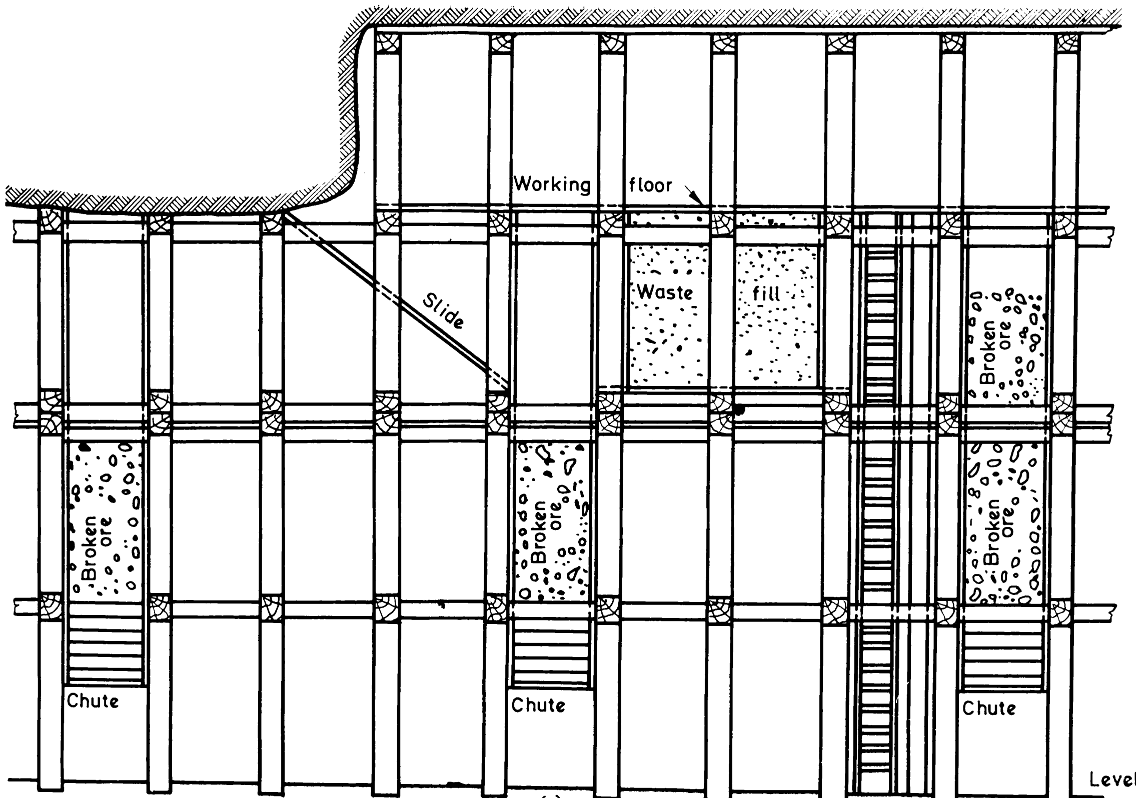
Period 1: 249,400 tons mined by company
Period 2: 93,637 tons mined by company

Cost per ton labor	Timber	Explo-sives	Other supplies	Power	Total
Period 1. $1.44	$0.30	$0.09	$0.14	$0.19	$2.16
Period 2. 6.12	0.82	0.17	0.79	0.57	8.48

* Mining conditions were more severe in 1951; this is partly responsible for increased stoping costs.

TABLE 2. COST OF SUPPLIES (MORNING MINE)[3]

Item	Unit	Period 1	Period 2
Power	50-lb. box	$ 6.25	$ 9.68
Lagging	M	19.00	65.00
Stull	16-in. dia./ft	0.1872	0.5635
Rail	2240 lb.	55.45	176.62
2-in. pipe	100 ft	16.09	39.04
$1\frac{1}{8}$-in. drill steel	100 lb.	12.59	20.48
Grinding balls	100 lb.	3.73	5.75


Working floor
Slide
Waste fill
Broken ore
Broken ore
Broken ore
Broken ore
Chute
Chute
Chute
Level

(a)

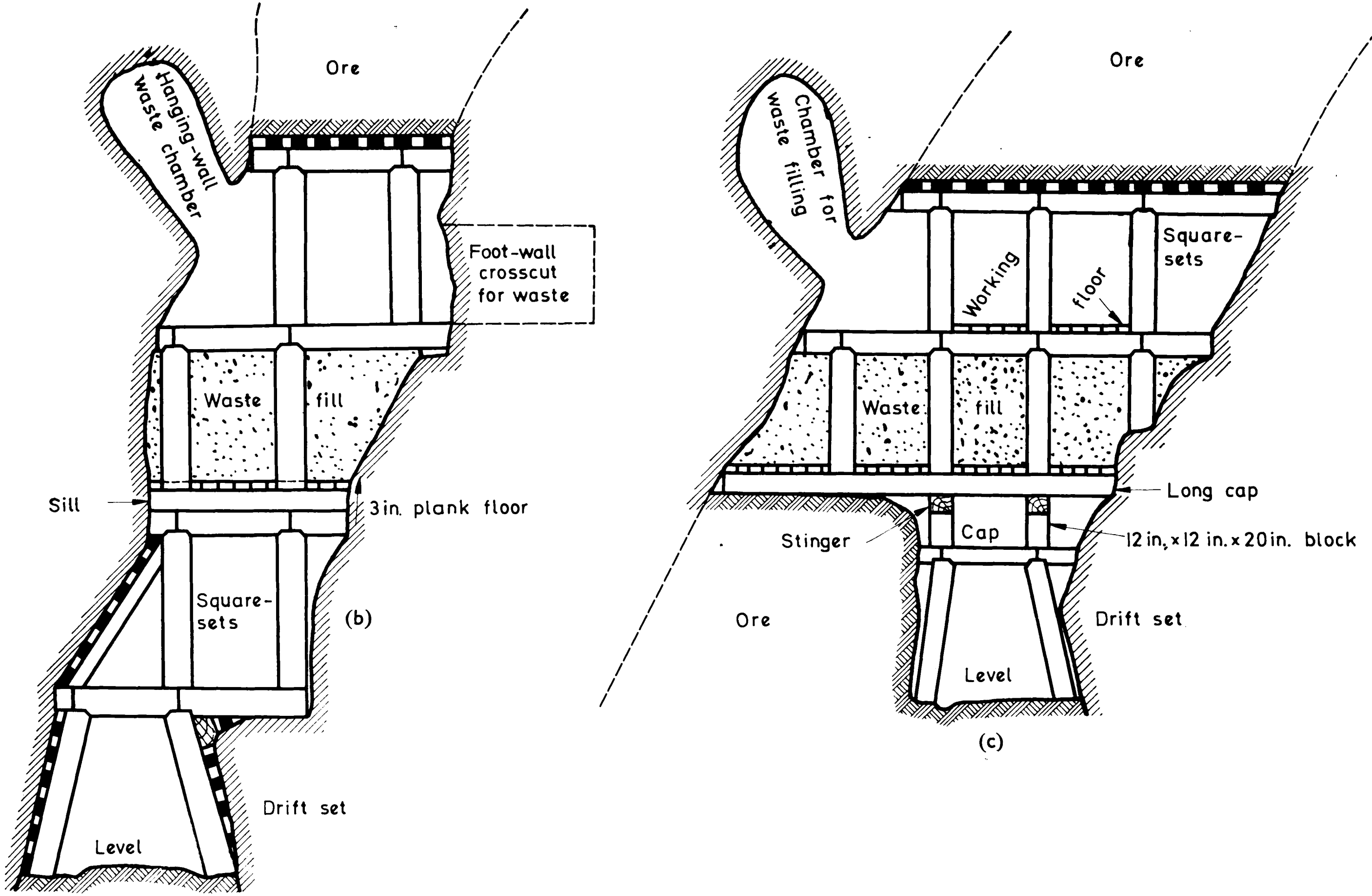


FIG. 3. Square-set stoping, Argonaut Mine, California. (a) Vertical longitudinal projection. (b) Vertical cross-section showing square sets directly on drift sets. (c) Vertical cross-section showing square sets on long caps and stringers.[8]

TABLE 3. TOTAL COST DISTRIBUTION (MORNING MINE)(3)

Mining	Period 1	Period 2
	(1939)	(1950-1951)
Ore mined (tons):		
company	249,400	93,637
leasers	113,722	3221
Total	363,122	96,858
Based on ore mined by company only:		
Superintendency plus vacations except develop. and shaft maint.	$42,118.70	$53,464.43
cost per ton	0.17	0.57
Stoping	538,281.92	793,798.27
cost per ton	2.16	8.48
Tramming on levels	60,203.03	88,312.86
cost per ton	0.24	0.94
Hoisting	101,969.84	151,427.41
cost per ton	0.41	1.62
Haulage on main tunnel	24,790.47	55,152.60
cost per ton	0.10	0.59
Drainage	15,059.26	49,677.90
cost per ton	0.06	0.53
Shaft and station maintenance	144,679.91	228,766.20
cost per ton	0.58	2.44
Ventilation	23,559.61	31,343.88
cost per ton	0.29	0.33
Lighting	8,734.43	7,940.03
cost per ton	0.03	0.08
General underground maintenance	118,859.58	95,261.64
cost per ton	0.48	1.02
Miscellaneous	49,823.98	93,434.21
cost per ton	0.20	1.00
Development	80,577.14	79,638.43
cost per ton	0.32	0.85
Mine dry and headhouse	11,755.55	24,159.36
cost per ton	0.05	0.26
Paid to leasers	436,436.94	41,768.50
Total mining cost	$1,656,850.36	$1,794,145.72
cost per ton	4.563	18.523
Milling		
Tons milled from Morning Mine	363,122	96,858
Tons milled from other mines	–	75,020
Total tonnage milled	363,122	171,878

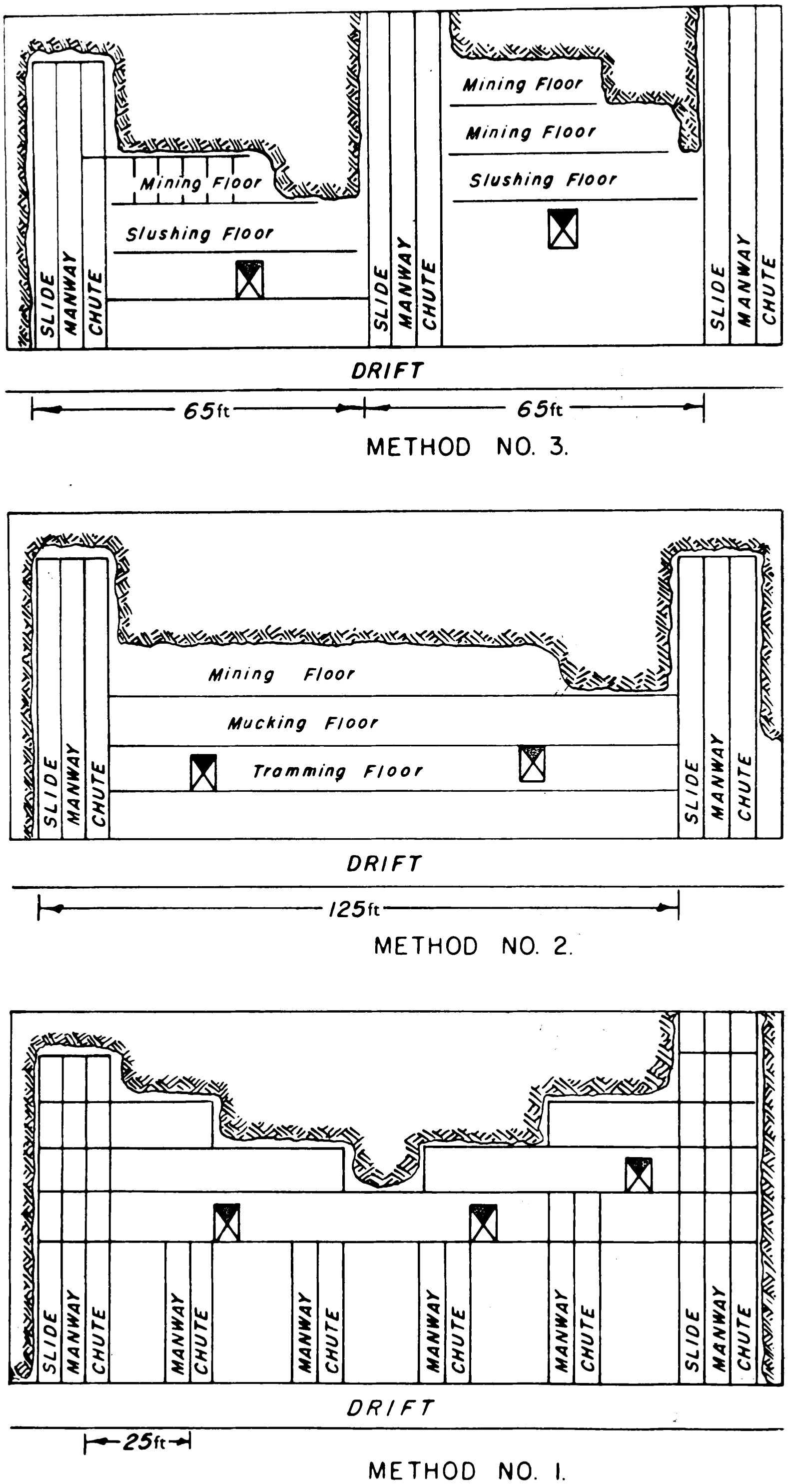


FIG. 4. Evolution of stoping methods at the Morning Mine, Mullan, Idaho.[3]

CUT-AND-FILL STOPES

Cut-and-fill Stopes Using Sand Fill (Hydraulic Fill)

Figure 5 illustrates conventional cut-and-waste fill stoping method used at the Dayrock Mine which was supplanted by the cut-and-tailings fill method, shown in Fig. 6. The lead–silver vein averages about 3 ft wide in stopes, with a maximum

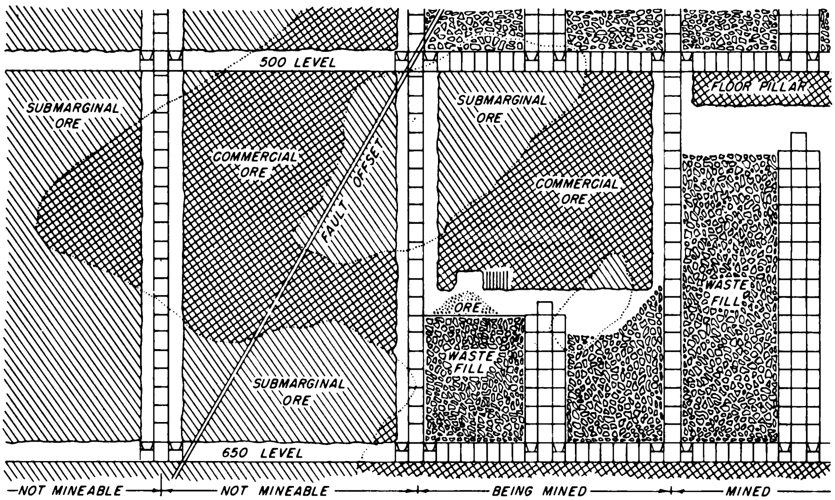


FIG. 5. Conventional cut-and-waste fill stoping as formerly practiced at the Dayrock Mine.[4, 5]

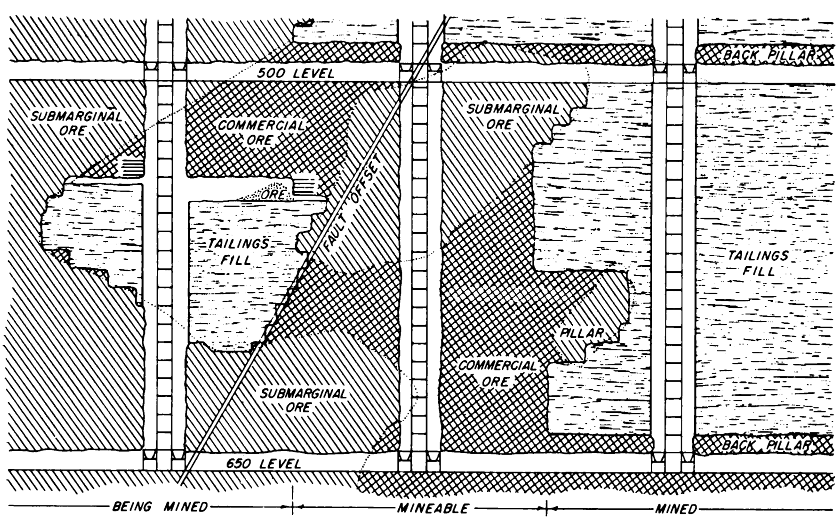


FIG. 6. Dayrock cut-and-tailings fill stoping, showing flexibility of this system.[4, 5]

width of 8 ft. Mineralization along the vein is irregular. The wall rocks, of fractured quartzite, are weak and tend to slab, requiring close filling to prevent dilution by wall rock.

Figure 7 illustrates the method of cut-and-sand fill stoping employed in a wide orebody at the Homestake Mine.

Filling Costs

Table 4 compares costs of sand filling vs. waste filling at the Dayrock Mine.

Mechanized Mining on Sand fill[7]

Figure 8 shows the methods used in mining a nearly vertical copper ore body 25–40 ft wide. Mining is by cut-and-hydraulic fill methods with a transloader (a self loading ore transport machine) being used to transport the ore from the face to ore passes.

TABLE 4. COST OF FILLING AT DAYROCK MINE[4]

	Cost of filling per ton of ore extracted	Cost of filling per ton of fill emplaced
Sand fill stopes	$0.67	$1.23
Waste fill stopes	1.21	1.81
Quarried waste fill stopes	6.01	9.02
	Cost of stope chutes and fill lacing	
Sand fill stopes	$8.80 per vertical foot	
Waste fill stopes	12.57 per vertical foot	
	Installation cost of sand fill plant	
Surface plant	$3485	
Underground trunk pipe line	3345	

(Salvage materials used extensively in the surface plant.)

The transloader operates in the stope on top of a de-slimed sand fill and despite its weight of 25,000 lb. plus the weight of 5.5 yd^3 of ore, the floor fill has proved sufficiently firm to permit excellent mucking and tramming rates to be achieved. The machine is capable in mucking in short bursts at a rate of 150 tons/hr while tramming 300 ft to the ore pass.

Production in the transloader stope is 23.9 tons/man-shift compared with 13.8 tons/man-shift in comparable cut-and-fill stopes in the same mine and the savings in production costs are calculated to be $1.10 per ton.

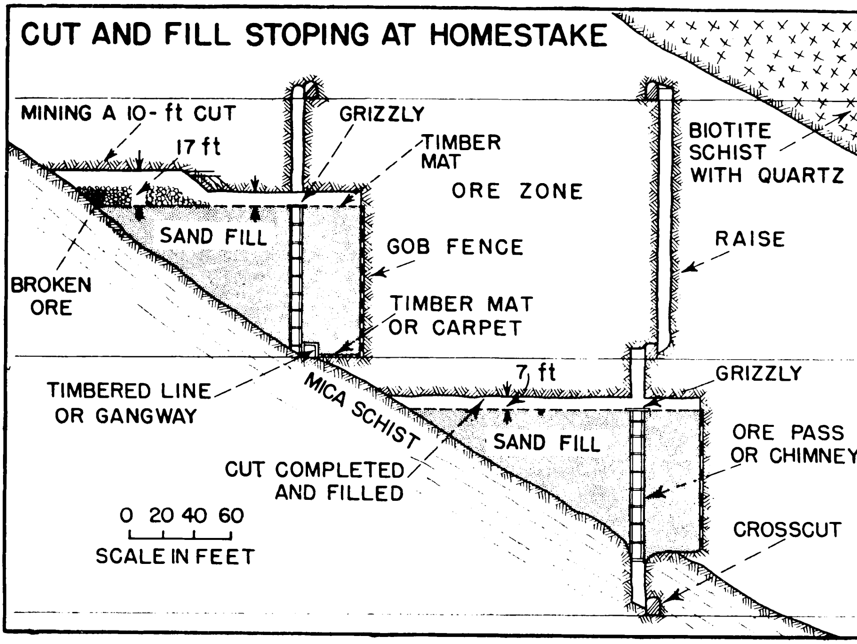


FIG. 7. Homestake Mine stope development starts by driving cross cuts 200 ft apart then raising 150 ft to the level above. Drifts, cross cuts, and gangways are 7×7 ft; raises are 5×7 ft.[6]

Cut-and-fill Stopes Using Waste Fill

Filled stopes are worked by overhand methods, the ore usually being removed from the back in a series of horizontal, or inclined, slices; or the ore may be removed in a series of steps which form an inclined line.

The stoping process consists of removing a slice of ore from 5 to 7 ft thick from the stope back, and then putting an equal depth of filling into the stope for support of the walls, and as a base on which the miners can work to drill and blast the next slice.

A "rill" stope is a stope with an inclined back; another name for such a stope is "inclined cut-and-fill". The inclined back has the advantage that ore may be removed from the stope, and waste run into the stope by gravity, and the amount of scraping required is reduced to a minimum. The disadvantages of the inclined stope are that: (1) working conditions for the miners on the sloping waste fill are difficult, in that they must drill and move equipment on an inclined surface, and (2) a greater vertical expanse of wall is exposed. This encourages slabbing off

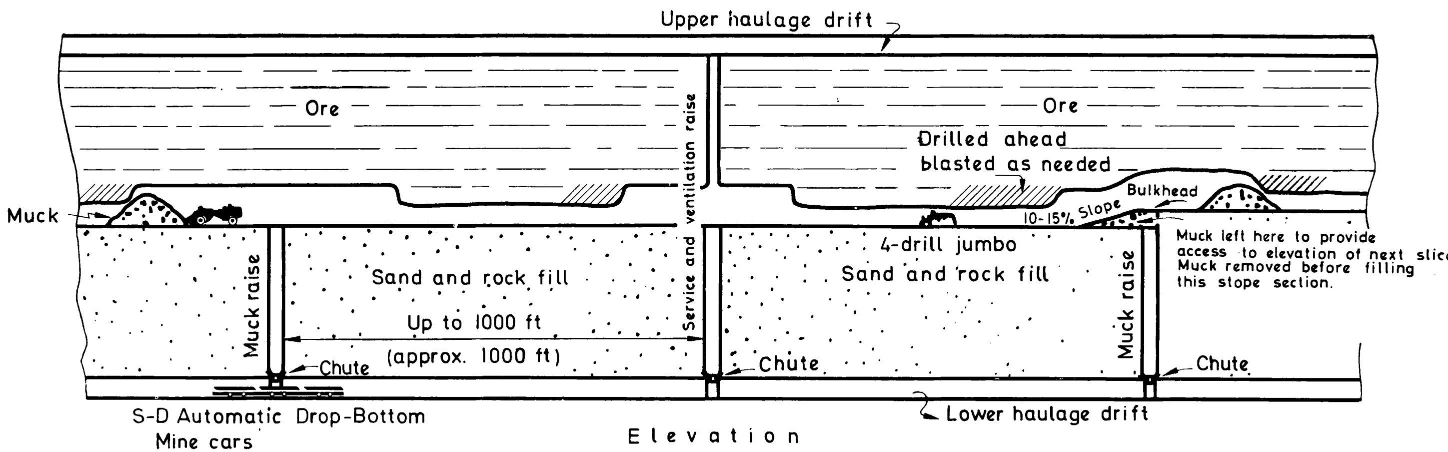


FIG. 8. Ore haulage equipment operates on the surface of the sand and rock stope filling at this Canadian copper mine.[7]

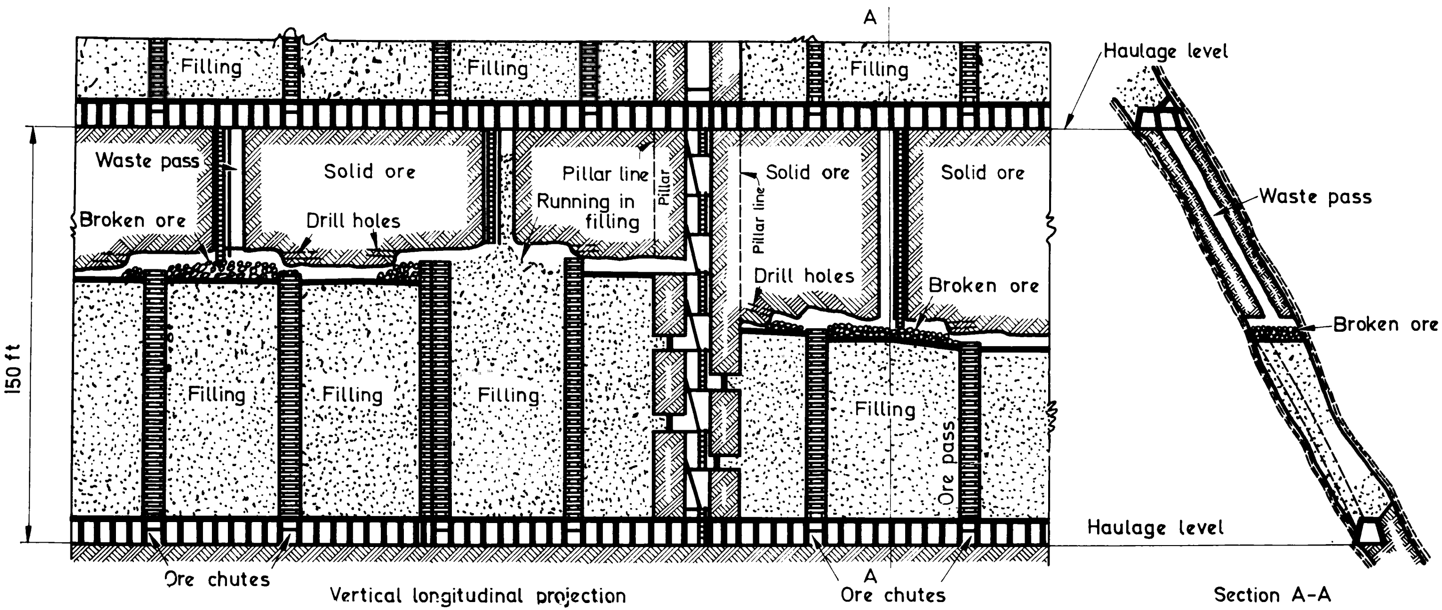


FIG. 9. Example of horizontal cut-and-fill stoping.[8]

the walls, and slabs which fall tend to roll down the inclined surface onto men working below.

The "flat-back" cut-and-fill stope has the advantages of: (1) working conditions are better, since men and machines work on a level surface; (2) less vertical expanse of wall is exposed so slabbing is reduced to a minimum. However, ore must be

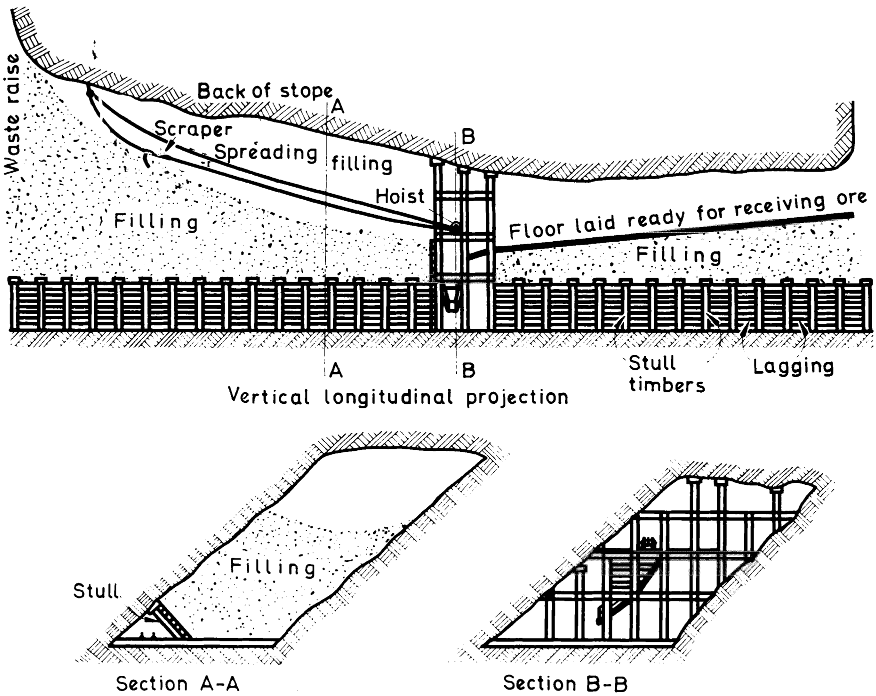


FIG. 10. Cut-and-fill stoping on stull timbers, Park Utah Mine, Utah.[8]

"slushed out" of a flat-back stope, and waste filling distributed in the stope, with power scrapers.

With large modern scrapers costs for flat-back stoping should not exceed those for inclined stoping.

In systems employing hydraulic sand fill, "flat-back" stoping methods are used because the surface of a sand fill is essentially horizontal after it solidifies.

Examples of cut-and-fill Stoping Operations Using Waste Fill

Figures 9–13 illustrate variations of cut-and-fill stoping methods (using waste fill) which have been used successfully in the past.

It has been noted previously that methods which require a great deal of timber cannot compete economically with methods which do not require as much material or labor to produce a ton of ore. Thus the trend is away from the use of waste fill and toward the use of a hydraulically placed fill consisting of sand or mill tailings. Waste fill may be economical when the waste is a product of the normal development or exploration program and does not have to be broken especially for filling material.

Sand fill and waste fill procedures and costs are treated in more detail in the chapter titled "Backfilling of Stopes", Volume 2, Part A, Chapter 9.

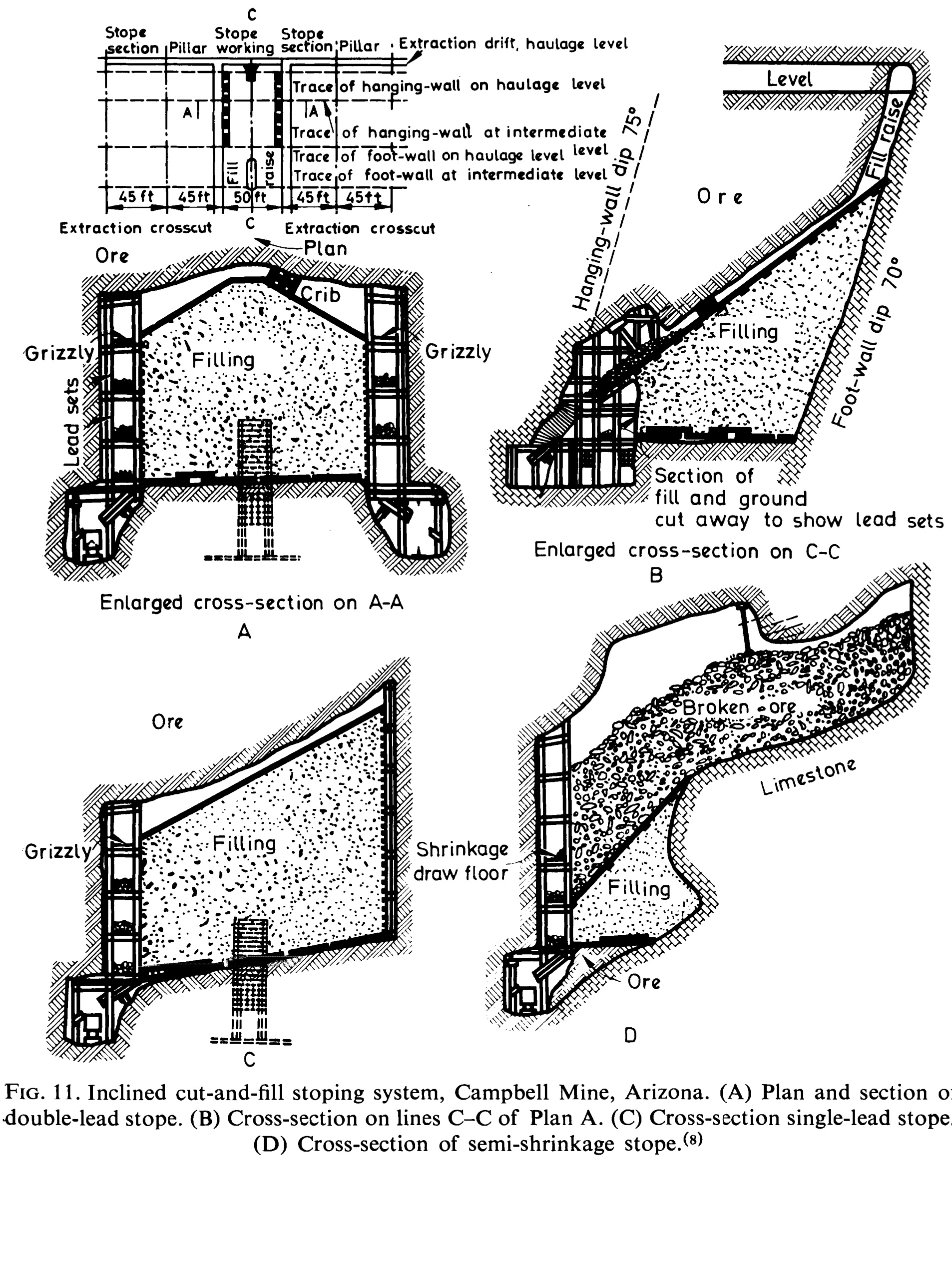


FIG. 11. Inclined cut-and-fill stoping system, Campbell Mine, Arizona. (A) Plan and section of double-lead stope. (B) Cross-section on lines C–C of Plan A. (C) Cross-section single-lead stope. (D) Cross-section of semi-shrinkage stope.[8]

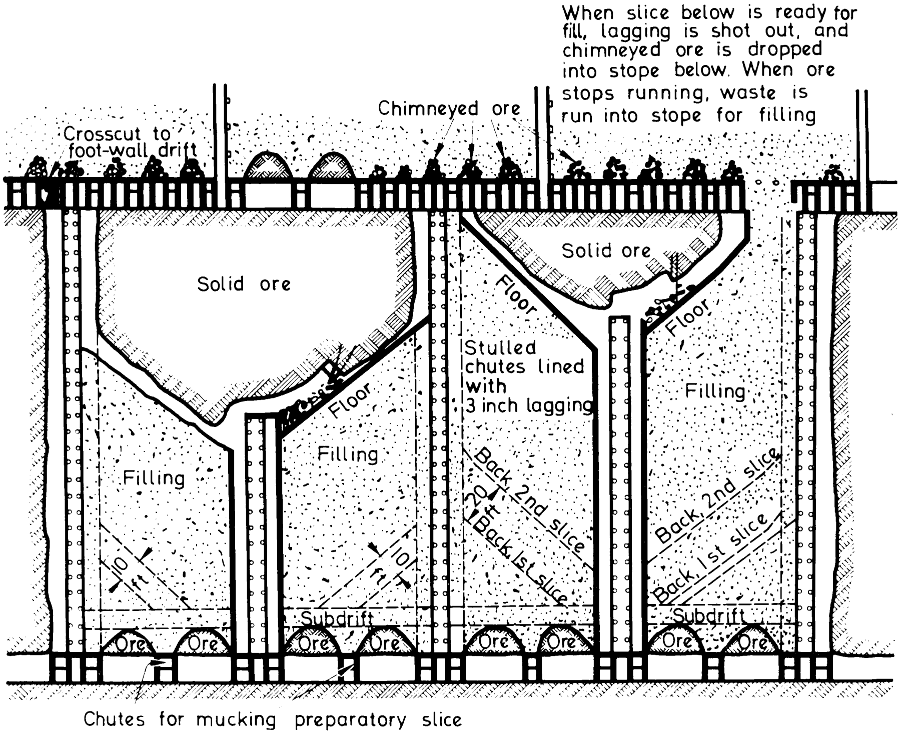


FIG. 12. Details of inclined cut-and-fill stoping, Eighty-Five Mine, New Mexico.[8]

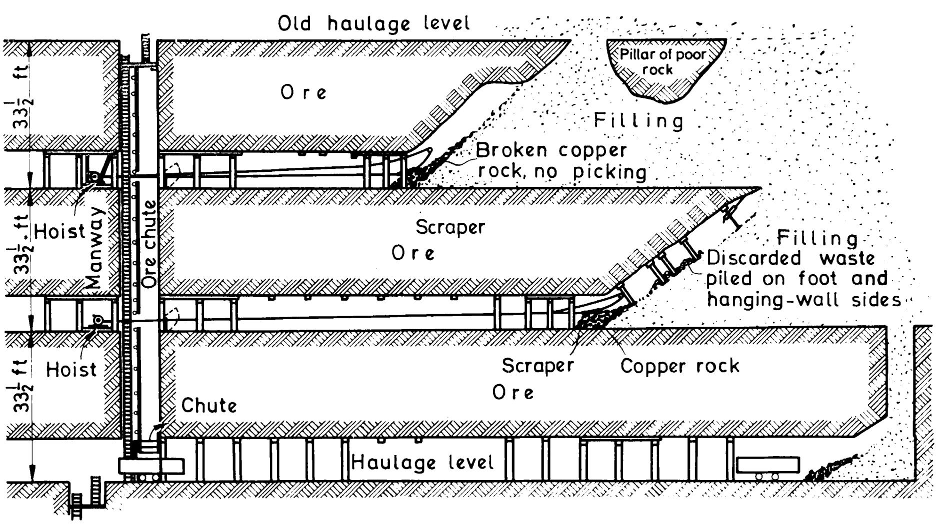


FIG. 13. Sublevel, inclined cut-and-fill system, Champion Mine, Michigan.[8]

STOPE-FILLING MATERIALS

The most common filling material is waste rock from development and exploration work in the mines. This is trammed and dumped into the raise leading to the stope to be filled. Waste for stope filling is frequently obtained also by driving cross cuts into stope walls and slushing waste into a stope, or by driving inclined raises into the stope walls and allowing the waste rock to run into the stope.

Waste obtained from the surface is also frequently used. This may be natural fragmented material, such as sand, gravel, or talus, or it may be quarried rock, broken by blasting. Such material is usually dropped through from the surface through special "waste raises" to levels above the stopes and then transported in level cars and dumped into the stope raises.

"Caving for waste" is sometimes practicable; with this method two parallel timbered drifts are driven in country rock a safe distance from the mine workings and inclined stopes started from the drifts. In fractured rock, which will cave when thus undermined, broken rock may be drawn off continuously without further blasting.

When a mine is nearly worked out, so that the upper workings may be allowed to cave, filling material may be drawn from old stopes on upper levels and dropped to lower levels to be re-used in lower stopes.

Mill tailings are one of the best sources of stopes filling. When part of the "slimes" are removed, and tailings are thickened to about 70 per cent solids, they may be carried long distances by pipeline to be discharged into stopes. In many cases mills are located too far from mining operations to allow use of this method. Consideration should be given to advantages of tailings for stope filling when selecting the location for a mill.

In some cases the additional water introduced with hydraulic tailings fill may produce deleterious results, such as softening of walls, or drift floors, etc., so that such filling is not practicable.

Sand filling equipment, procedures, and costs are treated in detail in Volume 2, Part A, Chapter 9.

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CHAPTER 3

SHRINKAGE STOPES

SHRINKAGE stopes are in a class between open stopes and filled stopes. In shrinkage stoping the ore is broken by overhand methods and accumulates in the stope until the stope is completed. As the ore increases in volume when it is broken, about one-third to one-half the broken ore must be drawn off as the stope progresses in order to leave room between the top of the broken ore and the stope back in which men can work.

The stope walls are supported by the broken ore until ore breaking is completed. Ore may then be drawn off completely, leaving the stope standing empty, in which case it is an open stope. If eventual caving of the stope is to be prevented it will be filled by dumping in waste from above, in which case it becomes a filled stope.

The shrinkage method is simple and requires little timber, and on this account it is suitable for use in steeply dipping deposits which have strong, regular walls.

ORES SUITABLE FOR SHRINKAGE STOPING

Characteristics of the ore for successful shrinkage stoping should be such that when broken it does not pack as do some ores which contain clay or much fine material. The ore must not "freeze" because of oxidation of pyrite, and also it should be of a type which will not oxidize badly while it is kept in storage in the stope as most ore-dressing plants are designed to recover sulfides, rather than oxides.

An objection frequently cited to the shrinkage stoping method is that considerable money must be invested in breaking of ore before any considerable amount of ore can be recovered. Thus the cost of broken ore in the stope represents invested capital on which interest must be charged.

As with open stoping methods the feasibility of shrinkage stoping decreases with increasing depths, since lateral earth stresses generally tend to cause walls to squeeze in, and to compress broken ore so that it cannot be drawn from the stope.

EXAMPLES – SHRINKAGE STOPING METHODS AND COSTS

Stormy Day Tungsten Mine

Figure 1 shows a typical shrinkage stope at a small tungsten mine in Pershing County, Nevada.

The ore occurs as tactile bodes mineralized with scheelite along the contact between marbelized limestone and granite. Most of the orebodies are from 4 to 10 ft thick.

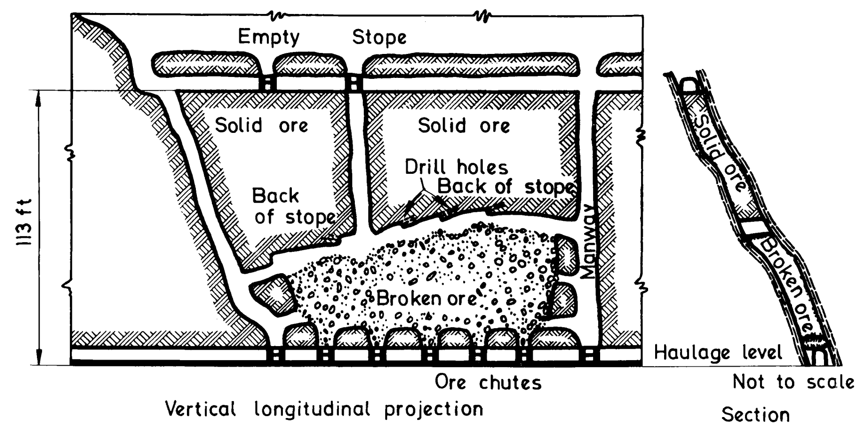


FIG. 1. Shrinkage stoping, Stormy Day Tungsten Mine, Pershing County, Nevada.[1]

Virtually all of the loading from drifts, cross cuts, and draw points was done by a Model 12B Eimco loader. The ore was hauled approximately 1000 ft by an Eimco air trammer on a track having a grade of $1\text{–}1\frac{1}{2}$ per cent in favor of the loaded cars. The track was laid with 16-lb. rails on wooden ties.

Surface Plant

The surface plant included a combination dormitory and cookhouse equipped with necessary facilities for housing employees at the property. The office building was a 21-ft trailer. The shop, well equipped with tools and machines needed to conduct mining operations at a small property, was in a 20- by 40-ft frame building where repairs and blacksmith work were done.

Summary of Equipment

Mining equipment at the Stormy Day mine comprised the following items:

Number	Equipment
1	100-ton steel ore bin.
1	Compressor, 315-cfm, Ingersoll-Rand.
1	Compressor, 315-cfm, Atlas-Copco.
1	Compressor, 210-cfm, Schramm.
1	Trammer, compressed-air, Eimco.
1	Loader, Model 12B, Eimco.
2	Mine cars, 22-ft^3, rocker-type.
2	Pusher legs.
1	Slusher, double-drum, Ingersoll-Rand.
1	Tugger, single-drum, Ingersoll-Rand.
2	Stope hammers, Ingersoll-Rand.
1	Stoper, Copco B-2.
3	Drills, jack-leg-type.
1	Generating plant, gasoline, small.
2	Steel-sharpening jibs.
	Air receivers.
	Water tanks.
	Bit grinder, portable.

Ventilation and Drainage

The mine had good natural ventilation and no water-drainage problem.

Labor Classification and Wages

An average of seven men, working one shift, was employed during development. All drilling and blasting were done during the first half of the shift.

Bunkhouse, boardinghouse, and family accommodations were provided, and employees were furnished accommodations and board in addition to their monthly wage. Working conditions were good, and labor turnover was relatively low. Job classifications and monthly wages were as follows:

Classification	Monthly wage	Classification	Monthly wage
Lead miner	$475	Trammer (mucker)	400
Miner (stope)	416	Surfaceman	325
Miner (development)	450	Cook	235
Trammer (development)	350		

Summary of Costs

Table 1 shows the summary of costs during the period from September 1955 to October 1956.

The charge for trucking ore to the United States Vanadium mill at Bishop, Calif., was $ 0.035 per ton-mile; to the Getchell mill at Redhouse, Nev., $ 0.0325 per ton-mile; and to the Toulon mill $3.50 per ton.

Mine: Stormy Day — Mining method: Shrinkage-stope.
Ore mined: 15,000 tons — Period: September 1955 to October 1956.

Size of excavation (ft): Drifting, cross cutting, raising – 5 by 7.

TABLE 1. PROPORTIONATE COSTS OF LABOR, POWER, AND SUPPLIES (%)[1]

	Development	Mining	Total
Labor:			
drilling and blasting	18.83	22.48	21.42
mucking	7.32	5.58	6.09
haulage	7.32	6.87	7.00
supervision	7.32	9.87	9.13
general	6.70	6.86	6.81
Total, all labor	47.49	51.66	50.45
Power and supplies:			
rental of mining equipment	22.42	18.39	19.56
explosives, fuse, caps	6.54	8.51	7.93
timber	1.50	1.53	1.52
air compression	3.55	3.52	3.53
drill steel	3.92	3.37	3.53
miscellaneous repairs and spares	1.50	1.53	1.52
miscellaneous	5.60	7.66	7.07
track and ties	5.61	–	1.63
water and air lines	1.87	–	0.54
vehicle travel	–	2.68	1.90
gas and power	–	1.15	0.82
Total, labor, power, and supplies	100.00	100.00	100.00

TABLE 2. LABOR, MAN-HOURS PER TON, AND TONS MINED PER MAN-SHIFT(1)

	Development	Mining	Total
Labor (man-hours per ton):			
underground:			
blasting and drilling	0.571	0.400	0.432
mucking	0.286	0.133	0.162
haulage	0.286	0.133	0.163
supervision	0.133	0.102	0.108
surface: general	0.327	0.190	0.216
Total man-hours per ton	1.603	0.958	1.080
underground:			
number of employees (excluding supervision)	2	5	7
Average tons mined per man-shift	7	12	10.57

Highland Surprise Mine

Figures 2 and 3 show shrinkage stoping and cut-and-fill stoping as practiced at the Highland Surprise Mine (zinc–lead), Shoshone County, Idaho.

Productivity and cost figures for the Highland Surprise Mine were as follows:

Shifts Worked in the Last Quarter of 1948

	Oct.	Nov.	Dec.	Total	Average month
Shift bosses	90	55	52	107	65.67
Ore breaking and transportation	700	646	618	1964	654.67
Development and exploration	475	476	421	1372	457.33
Other underground	219	420	505	1144	381.33
Mechanical and surface	503	503	439	1445	481.67
Total	1987	2100	2035	6122	2040.67
Production in tons broken per man-shift:					
ore breaking	6.11	6.69	6.72		6.51
development	5.61	6.65	7.33		6.53
all mine employees	3.49	3.56	3.56		3.54

Direct Mining and Milling Costs per ton in Last Quarter of 1948

	Labor	Explo-sives	Comp. air bits, steel	Timber	Misc.	Total
Mining:						
stoping, combination of shrinkage, cut-and-fill	$1.43	$0.29	$0.27	$0.38	$0.03	$2.40
raising in ore	1.90	0.32	0.41	0.58	0.07	3.28
drifting in ore	2.41	0.68	0.41	0.02	0.20	3.72

	Per ton
cost of underground transportation .	$0.20
hoisting	0.20
surface transportation	0.09

Summary — mining costs:			
stoping	$1.44		
raising in ore	0.66		
drifting in ore	0.80		
transportation and hoisting	0.49		
total stoping and development		$3.39	
exploration		3.30	
surface		1.05	
Total mining cost per ton			$6.74
Milling:			
labor	$0.48		
supplies	0.75		
power	0.18		
Total milling cost			$1.41
General expense			0.75
Total direct mining and milling cost per ton			$8.90

Little Pittsburgh Mine

Figure 4 shows a modified shrinkage stoping method which was tried at the Little Pittsburgh Mine, Shoshone County, Idaho. This modification was tried experimentally because standard methods of shrinkage stoping resulted in excessive dilution from slabbing off of the walls as stopes were pulled. The smaller stopes of the modified method reduced dilution and reduced amount of ore in storage. Savings were not sufficient to compensate for the additional "dead work" and for such dilution as still occurred, and the mining system was converted to the modified square set system shown in Fig. 5.

Miscellaneous Mines

Figures 6, 7 and 8 show variations of shrinkage stoping methods which have been used at various mines.

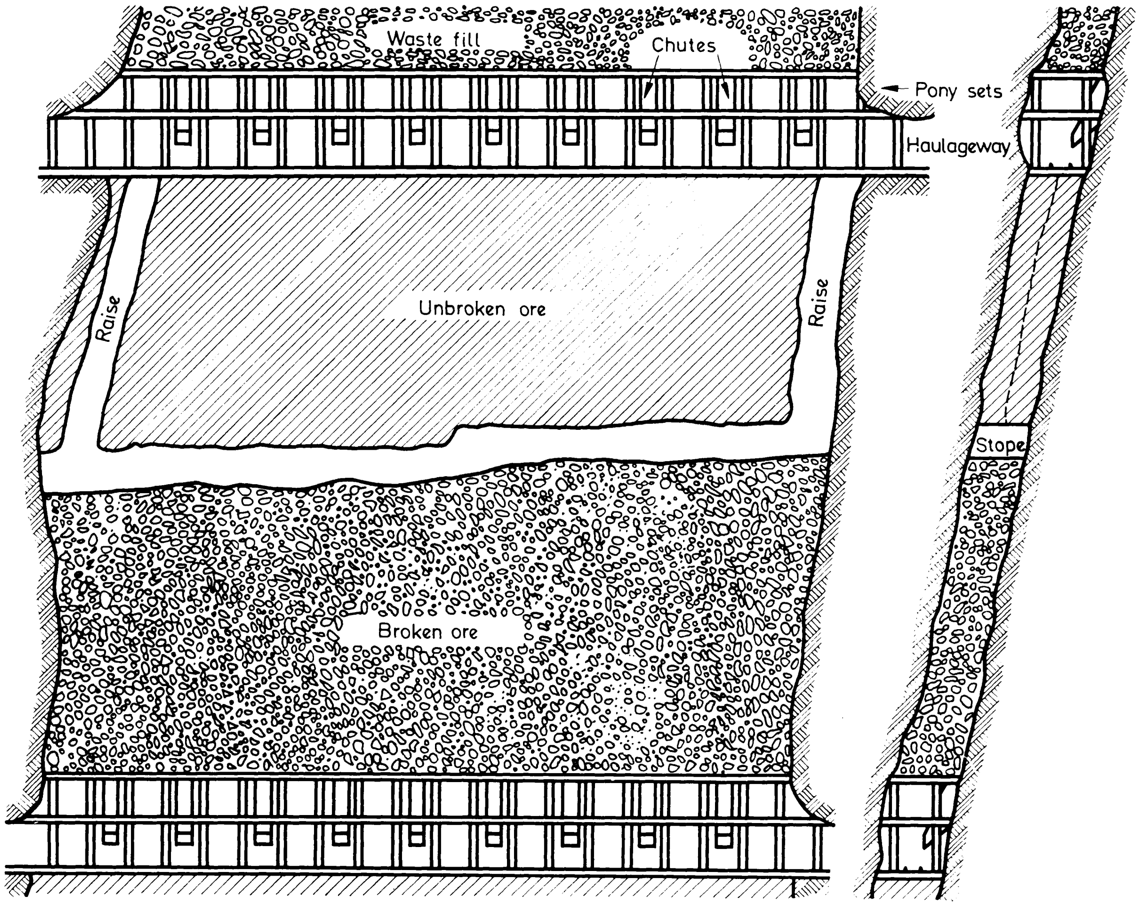


FIG. 2. Shrinkage stoping, Highland Surprise Mine, Pine Creek, Idaho.[2]

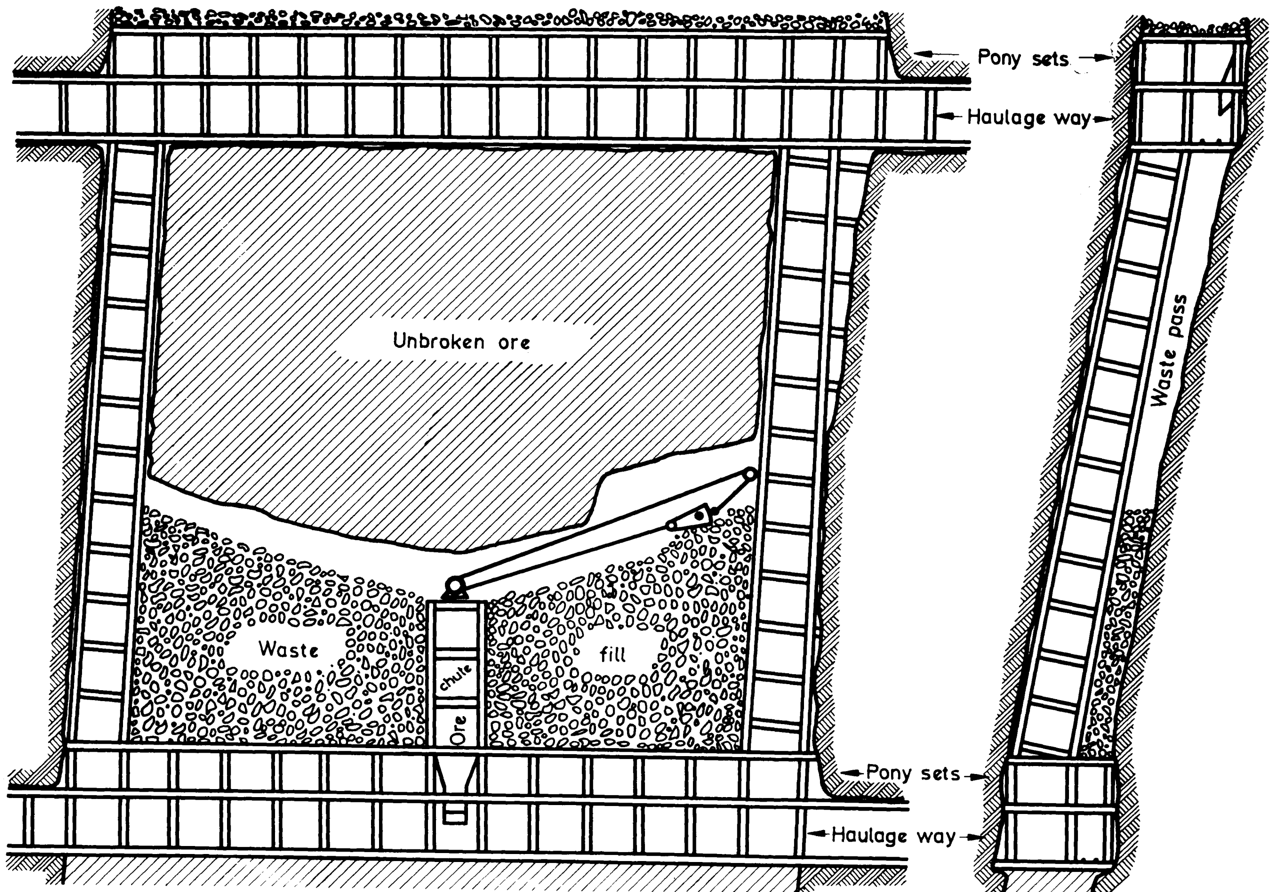


FIG. 3. Cut-and-fill stope, Highland Surprise Mine.[2]

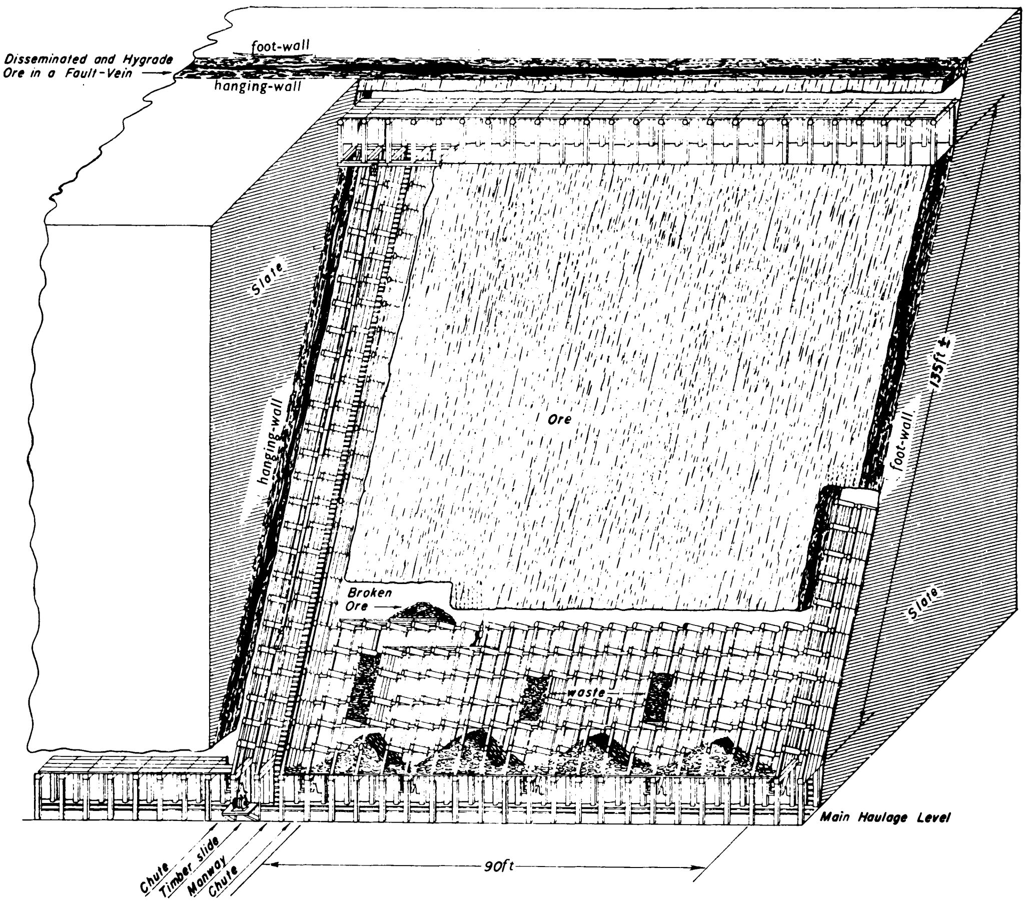
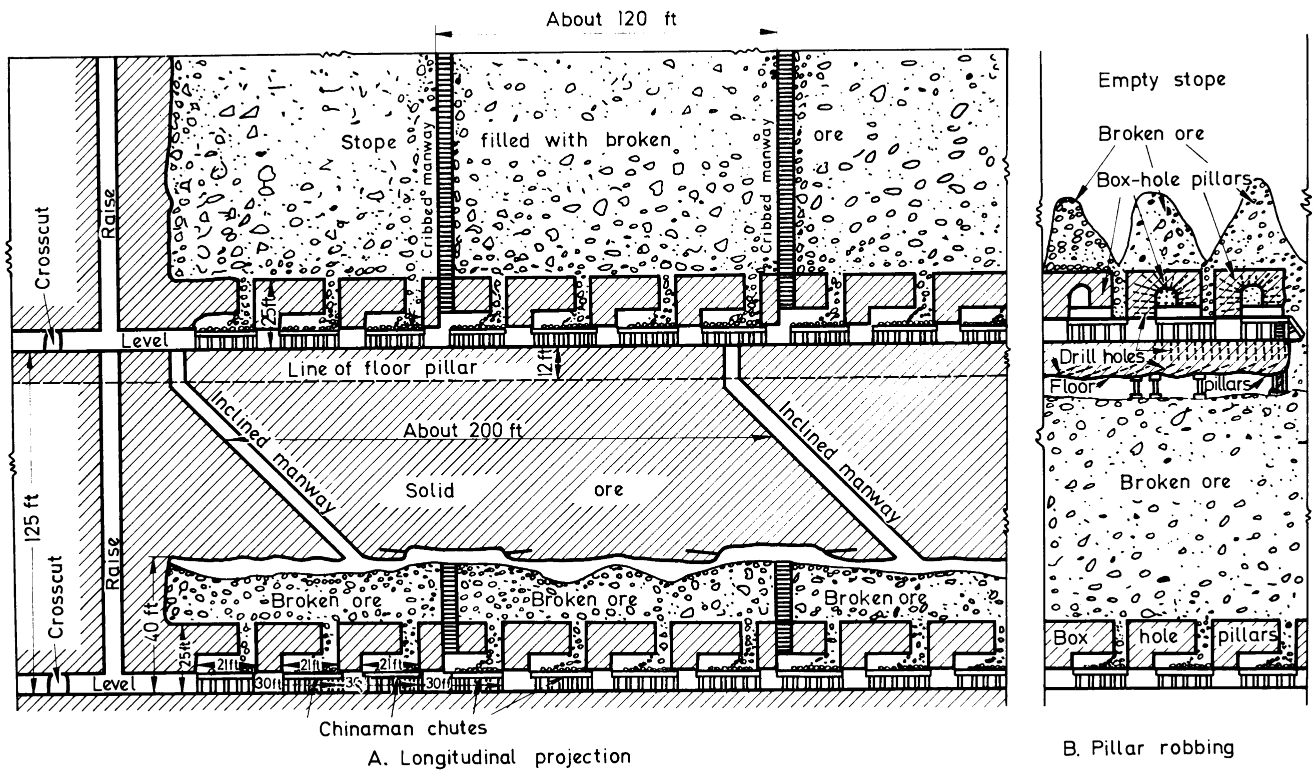


FIG. 5. Typical stoping method used at the Little Pittsburgh Mine.[3]



A. Longitudinal projection

B. Pillar robbing

FIG. 6. Shrinkage stoping over boxhole pillars, using chinaman chutes, Teck-Hughes Mine, Kirkland Lake, Ontario.(4)

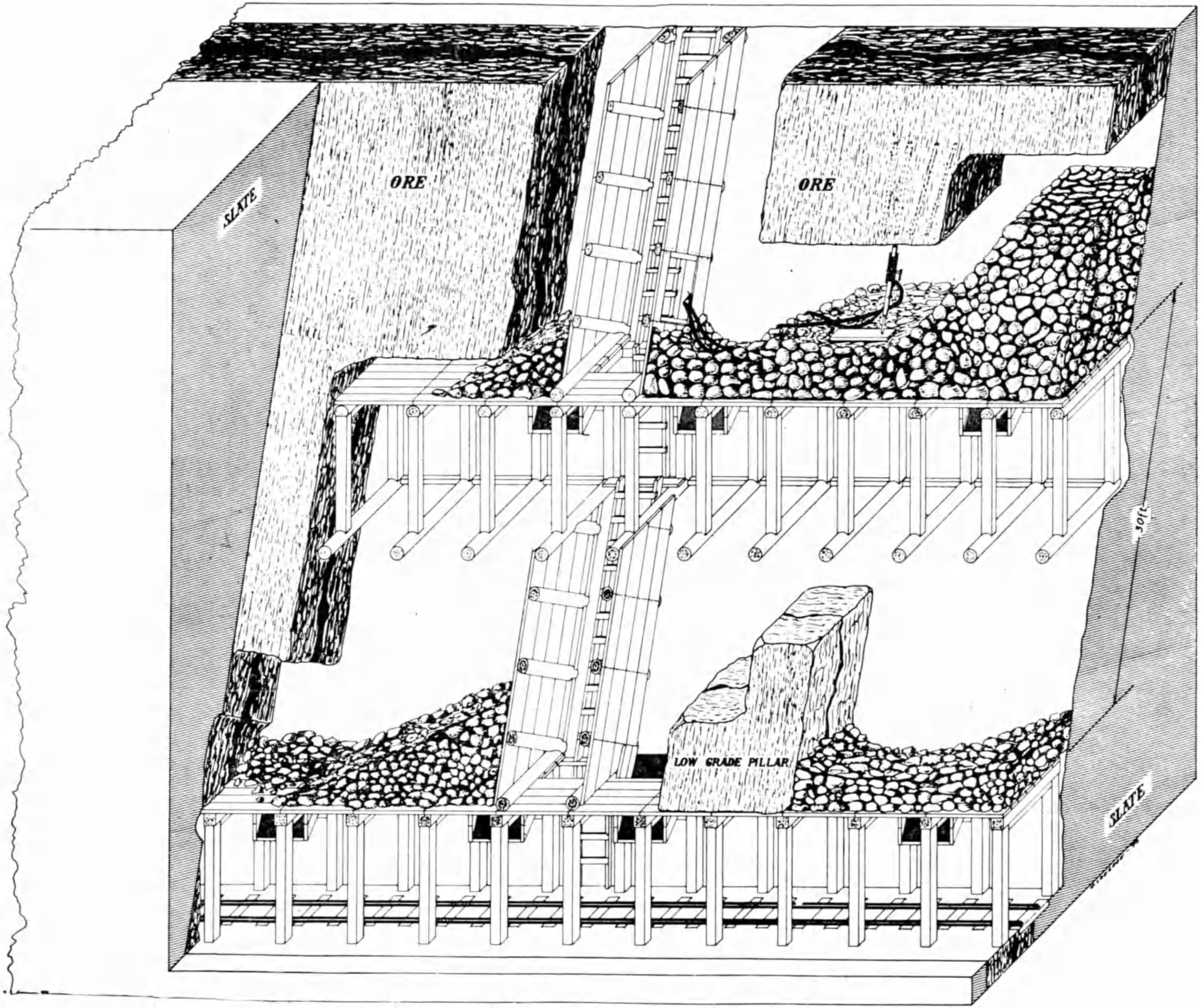


FIG. 4. Experimental shrinkage stope, Little Pittsburgh Mine, Pine Creek, Idaho.(3)

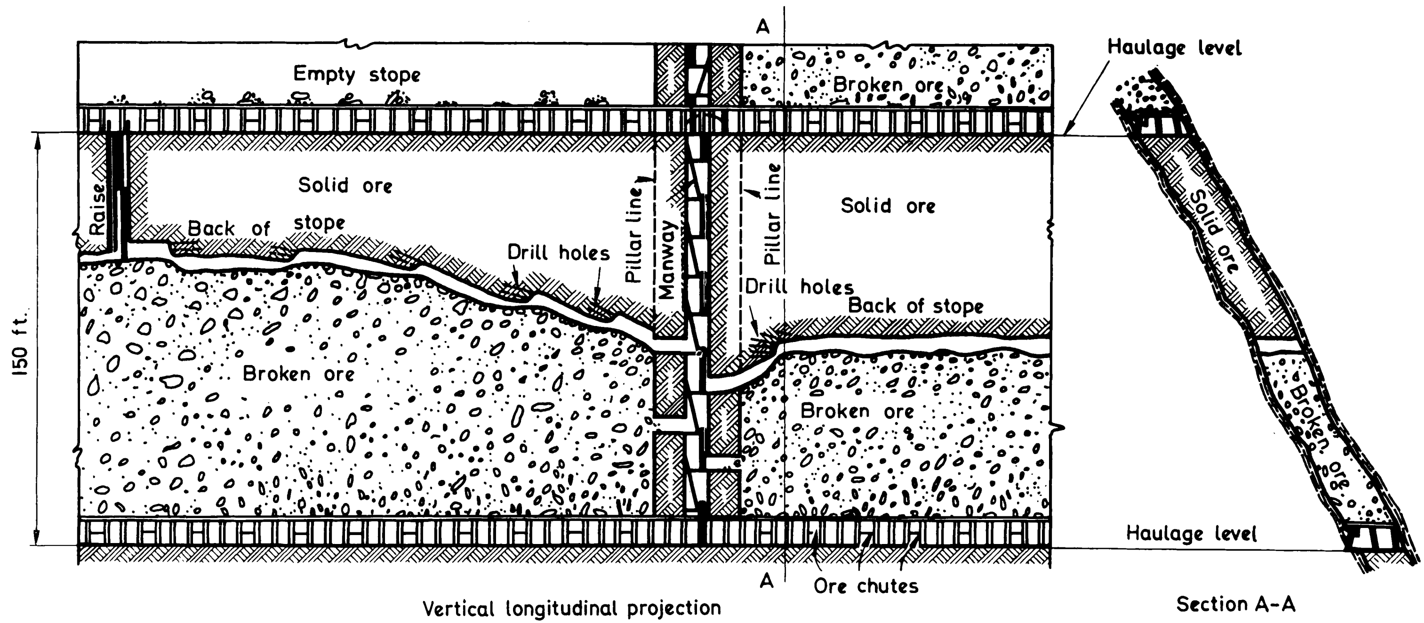


FIG. 7. Example of shrinkage stoping; stoping on drift sets.[4]

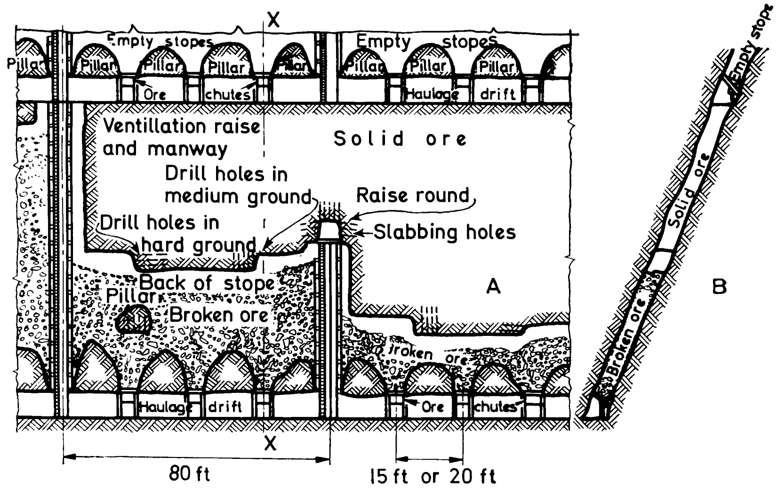


FIG. 8. Shrinkage stoping on drift pillars, Nevada–Massachusetts Mine, Nevada. (A) Vertical longitudinal projection. (B) Vertical cross-section X–X.[4]

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CHAPTER 4

SLICING AND CAVING SYSTEMS

THESE methods include systems in which ore is extracted from beneath the waste capping which rests on it. The capping is allowed to subside as the ore is extracted.

TOP SLICING

Top slicing consists of removing ore by driving a series of timbered drifts and cross cuts in the top of the orebody, just under the waste capping. These openings are driven side-by-side. As each is completed the supporting timber is blasted and overlying material is allowed to subside onto what was formerly the floor of the opening. Another cross cut, or drift, is then driven alongside the opening previously caved and the process repeated, working downward in successive slices.

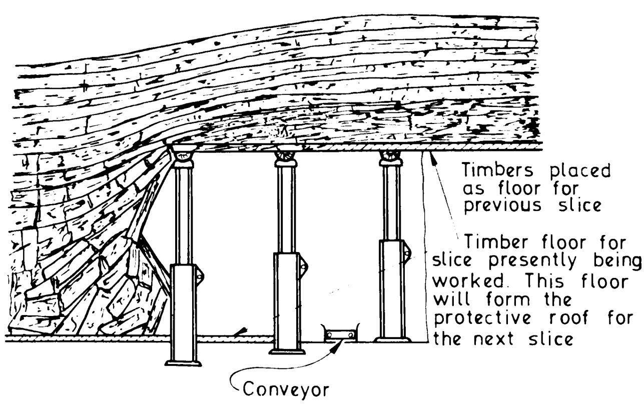


FIG. 1. Top slicing in a thick seam using yielding props for support.[1]

A thick mat of timber builds up over the unbroken ore. This helps to prevent overburden from sifting through into the workings.

Mining costs for top slicing are relatively high because of the large quantities of timber required. The restricted size of working faces at which ore is broken, and the distances which ore must be transported to raises, are other factors which contribute to high costs.

The method is best adapted to wide veins, or to thick beds of weak ore where a high percentage of extraction is desired and waste dilution is to be kept to a minimum.

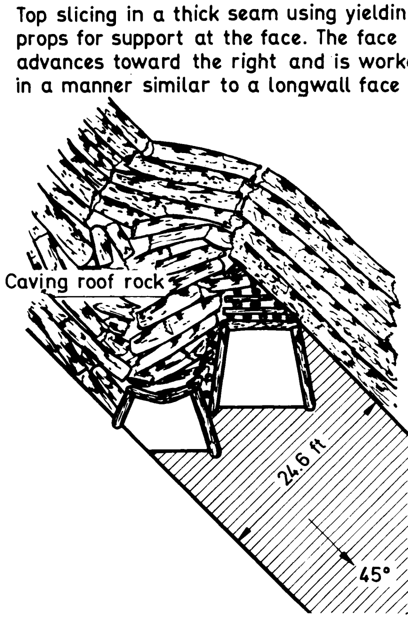


FIG. 2. Top slicing in a thick inclined coal seam.(1)

Since the surface subsides as the ore is mined this method cannot be used if the surface must be maintained. The overburden, or cap rock, must be of such nature as to break and subside evenly, as arching over, or "hang-ups", followed by sudden collapse might seriously damage the workings below.

Figures 1 and 2 illustrate variations of "top slicing" methods used in thick coal seams in France.

SUB-LEVEL CAVING

Sub-level caving is an extension of top-slicing. Drifts or cross cuts are driven in a manner similar to top slicing. After each drift is completed to the edge of the orebody, the ore over and at the sides of the drift is allowed to cave on retreat from the drift.

The height of the slices taken in sub-level caving is usually 15–25 ft, as compared with 10–12 ft in top-slicing.

Sub-level caving may be applied to types of orebodies similar to those suitable for top-slicing except that sub-level caving requires ground which will break readily, but coarsely, to form a capping which will arch to support itself readily over small openings.

Mining costs are less with sub-level caving than with top-slicing, since less timber, powder, labor, etc., are required per ton mined, but the ore is subjected to some waste dilution during caving, and some ore is lost in the gob.

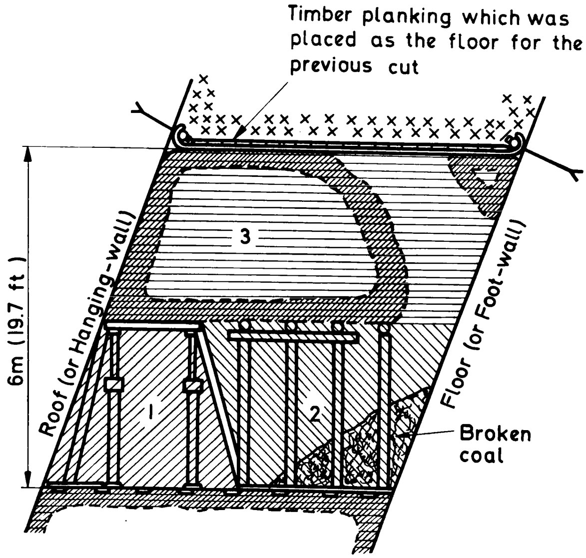


FIG. 3. Mining coal from a steep seam by sub-level caving.[1]

Figure 3 illustrates a variation of sub-level caving as used in mining a steeply dipping coal seam at Firminy, France. The order of extraction is indicated by the numbers 1, 2 and 3.

Figure 4 shows a method of recovering top coal from a thick seam by caving it behind the face. Figures 5 and 6 show the sub-level caving system as used in the iron mines at Kiruna, Sweden.

BLOCK CAVING

Block caving involves the undercutting of large blocks of ore. The settlement, following undercutting, causes crushing and fracturing of the ore. As broken ore is drawn off the "block" settles continuously and subsidence eventually extends to the surface. Ore is drawn off at the bottom of the block until cap rock, or overburden, appears at the drawpoints, indicating that ore in the block has been exhausted.

The orebody may be divided into blocks, by narrow shrinkage stopes, or a series of drifts may be driven, one above the other, to form lateral boundaries.

Blocks are commonly about 300 ft high; the King asbestos mine in Quebec has successfully caved blocks 400 ft high. Blocks 600 ft high are being successfully caved at the San Manuel Mine (copper), Pinal County, Arizona.

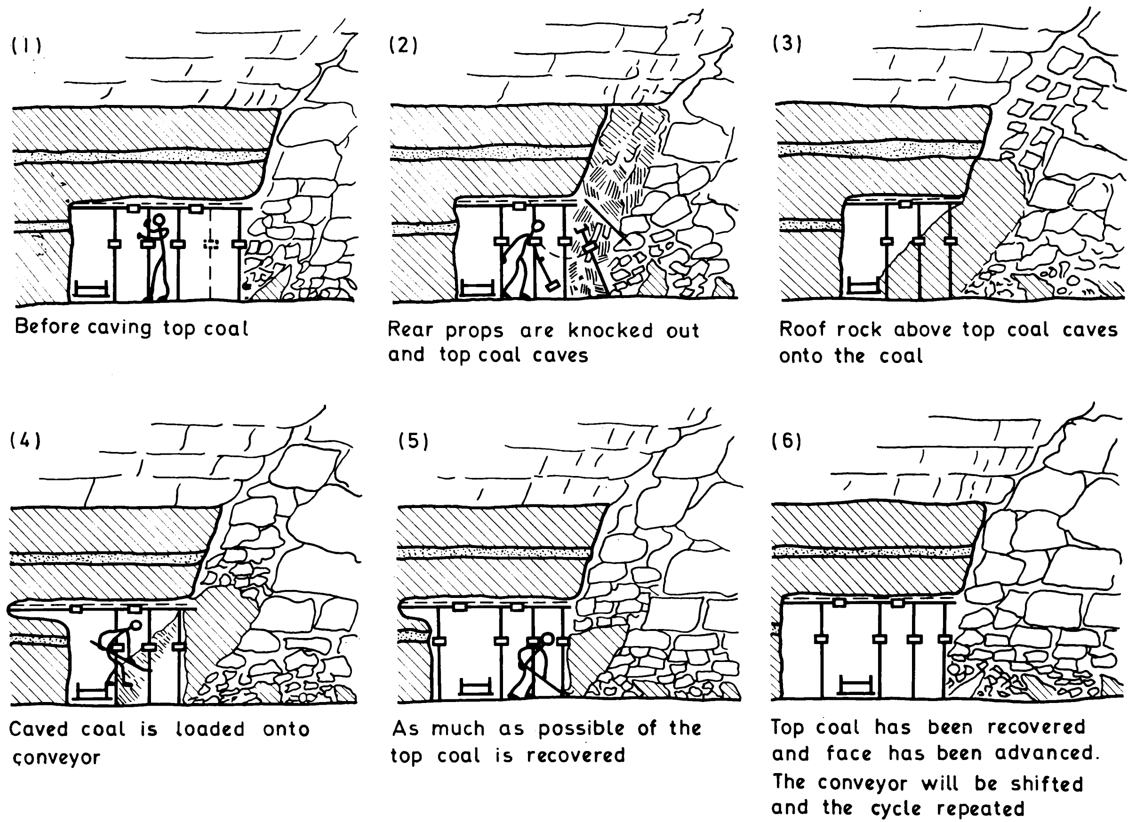


FIG. 4. Mining in a thick seam by caving top coal behind the face.[1]

In horizontal area a block may extend the full width of the orebody but the tendency has been to reduce the size of blocks. Magma copper reduced blocks to 150 ft square.

A block should be large enough so that the ore caves freely when undercut, but small enough that excessive weight is not thrown on supports of extraction openings below.

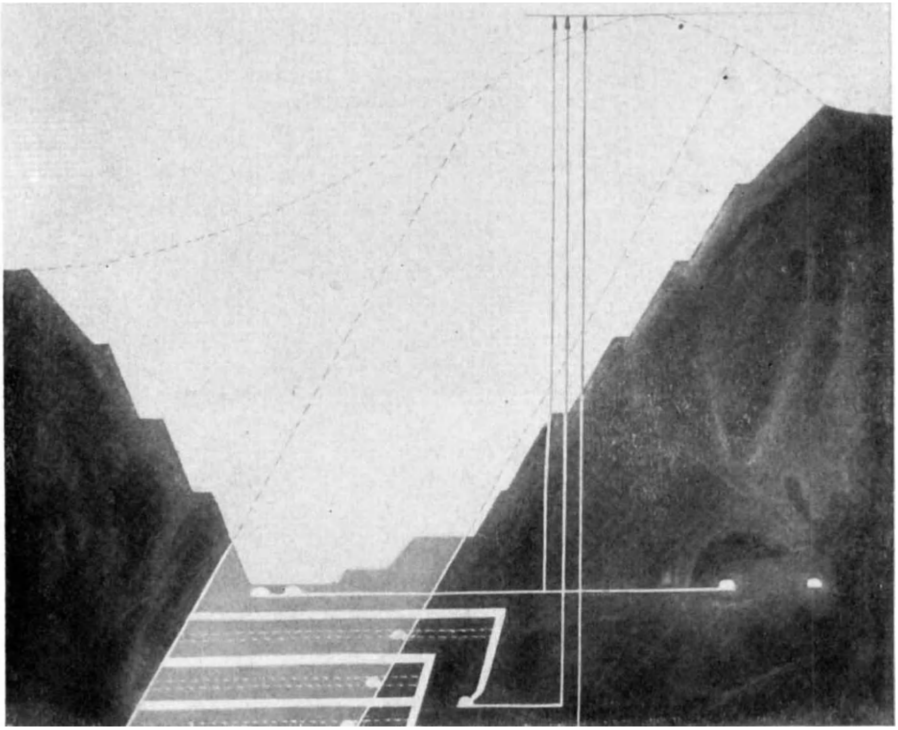
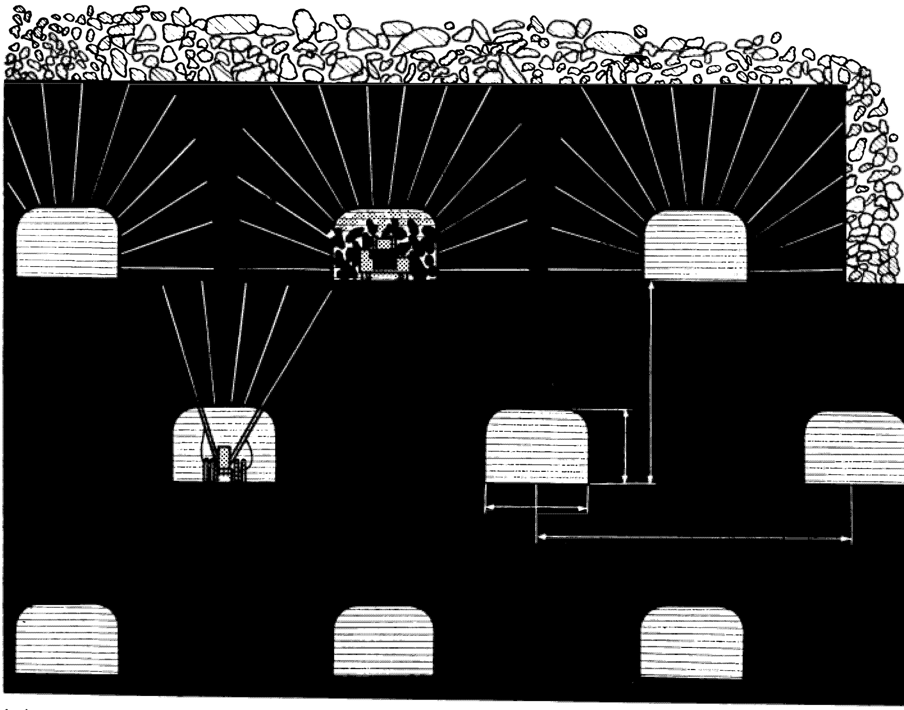


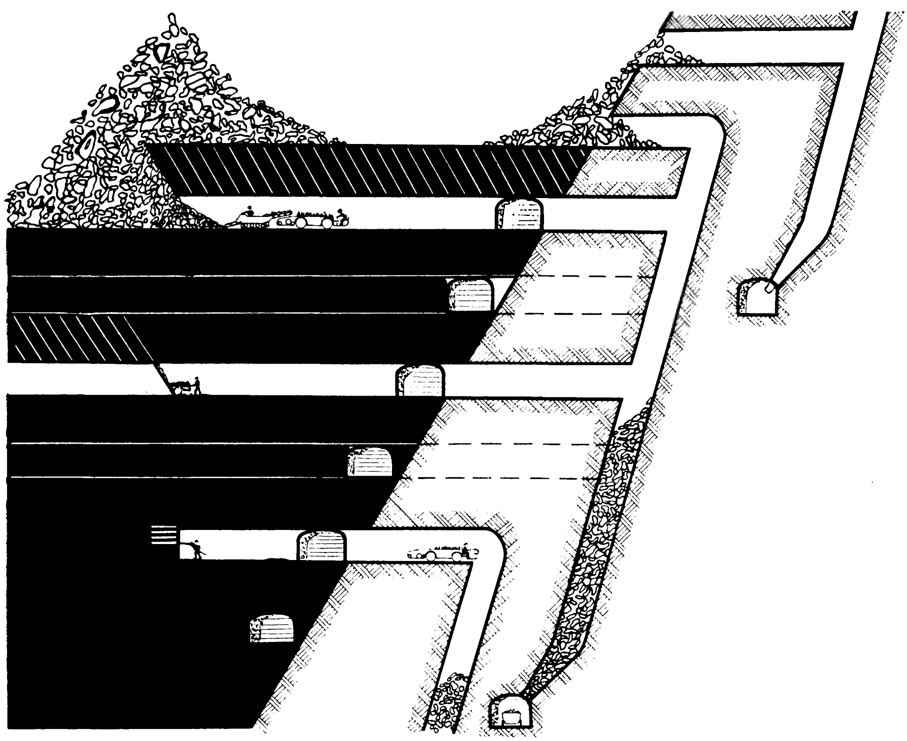
FIG. 5. Underground mining plan at Kiruna, Sweden. Cross-section of pit and location of underground development for sub-level caving.[2]

Orebodies suitable for caving are thick beds or massive deposits of homogeneous ore overlain by ground which will cave readily. Character of the ore must be such that it can be supported while blocks are developed and undercut, but will break up when caved.

Block caving has been used principally in the Lake Superior iron districts, in the "porphyry" copper ores of the Western United States, and in the asbestos mines of Quebec. Efficiency of modern heavy excavation machinery tends to tip the economic balance in favor of stripping and mining by open pit methods of many deposits which formerly might have been mined by block caving methods.



(a)



(b)

FIG. 6. Sub-level caving in iron ore at Kiruna, Sweden. (a) Sub-level caving with 30-ft high slice showing slice drilling pattern from sub-levels. (b) Transverse sub-levels with mining retreat from hanging-wall.[2]

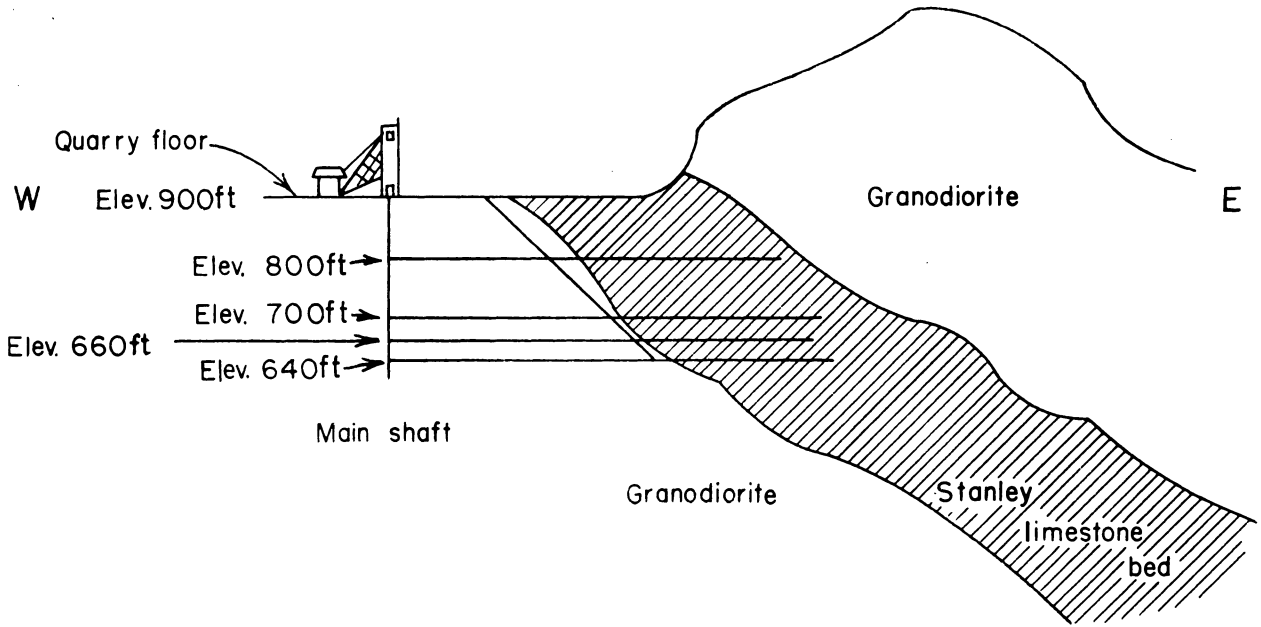


FIG. 7. East–west section showing placement of principal mine workings for block caving in relation to lower Stanley limestone bed at Crestmore Mine of the Riverside Cement Co., Riverside, Calif.(3)

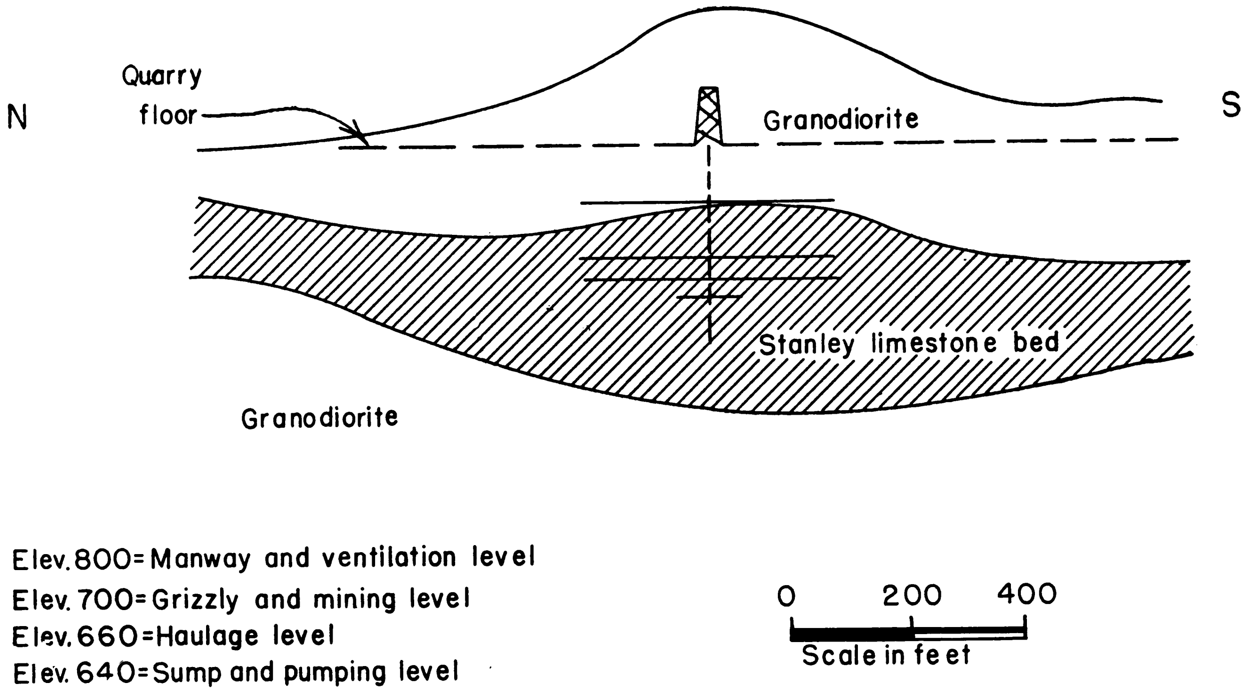


FIG. 8. North–south section, showing placement of principal mine workings for block caving in relation to the lower Stanley limestone bed, Crestmore Mine.[3]

BLOCK CAVING IN LIMESTONE

Block caving operations are usually confined to massive deposits of structurally weak ores. An exception to this rule was the block caving operation at the Crestmore Mine of the Riverside Portland Cement Co., at Riverside, California. This is the only known example of block caving having been used to mine a limestone deposit, and in fact the only known example in which the method has been successfully applied in structurally strong rock. Strength of the mined rock is indicated by the physical property data of Table 1. Between 1930 and 1954, 7,882,000 tons of limestone and granitic materials were extracted from the Crestmore Mine by block caving.

The caving procedures and experience resulting from their application to caving limestone are of particular interest because they increase the range of ore strengths to which block caving may be applied.

Figures 7 and 8 show the general outlines of the limestone and the placement of mine workings. Figures 9–11 show plans and cross-section through some of the mined blocks.

Characteristics of the Limestone Deposit

Although the Crestmore limestone contained a network of watercourses and accompanying areas deteriorated by solution, the rock did not contain, to any significant degree, the fracturing and jointing normally characteristic of block-caving ground. The cover overlying the limestone was thin, ranging from 0 in some areas to about 375 ft overlying block 4C. This maximum depth of cover affected only a small percentage of the total area caved. Hence, the weight on the blocks and correspondingly the stress in the limestone was slight, a factor that makes caving more difficult.

TABLE 1. PHYSICAL PROPERTY DATA[3]

Specimen No.	Specific gravity*	Compressive strength (psi)	Modulus of rupture (psi)	Young's modulus† (psi × 10^6)	Modulus of rigidity (psi × 10^6)	Young's modulus‡ (psi × 10^6)	Scleroscope hardness, scleroscope units
1	2.68	7700	2170	9.10	4.02	9.02	42
2	2.72	7900	–	6.58	3.15	6.8	39
3	2.71	8500	2260	11.1	4.17	11.0	45
4	2.68	11,000	2300	9.82	4.11	5.82	44
5	2.70	6700	–	2.41	2.29	5.82	39
6	2.72	24,000	2360	11.4	4.37	11.3	56

* Apparent.
† Sonic.
‡ Static.

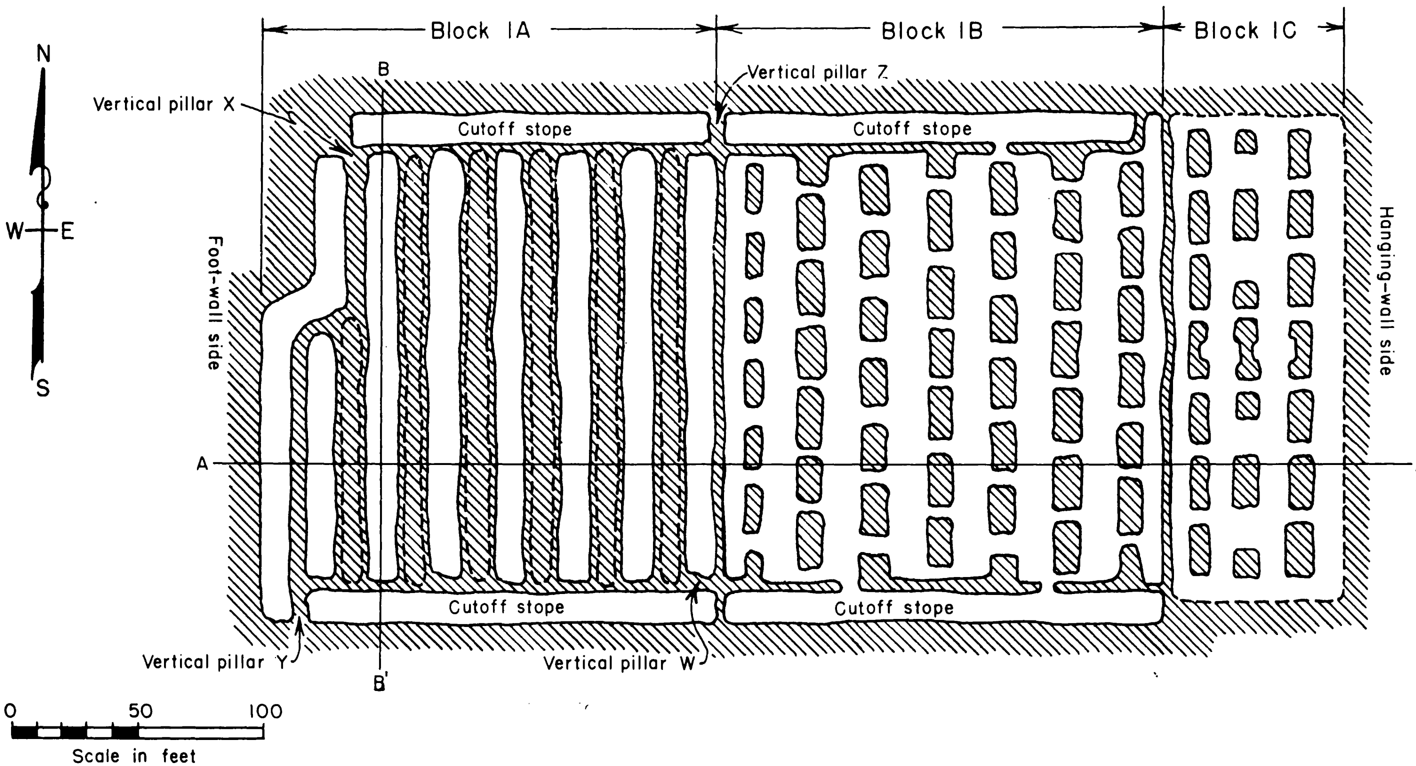


FIG. 9. Plan of undercut level of Blocks 1A, 1B, and 1C, Crestmore Mine.(3)

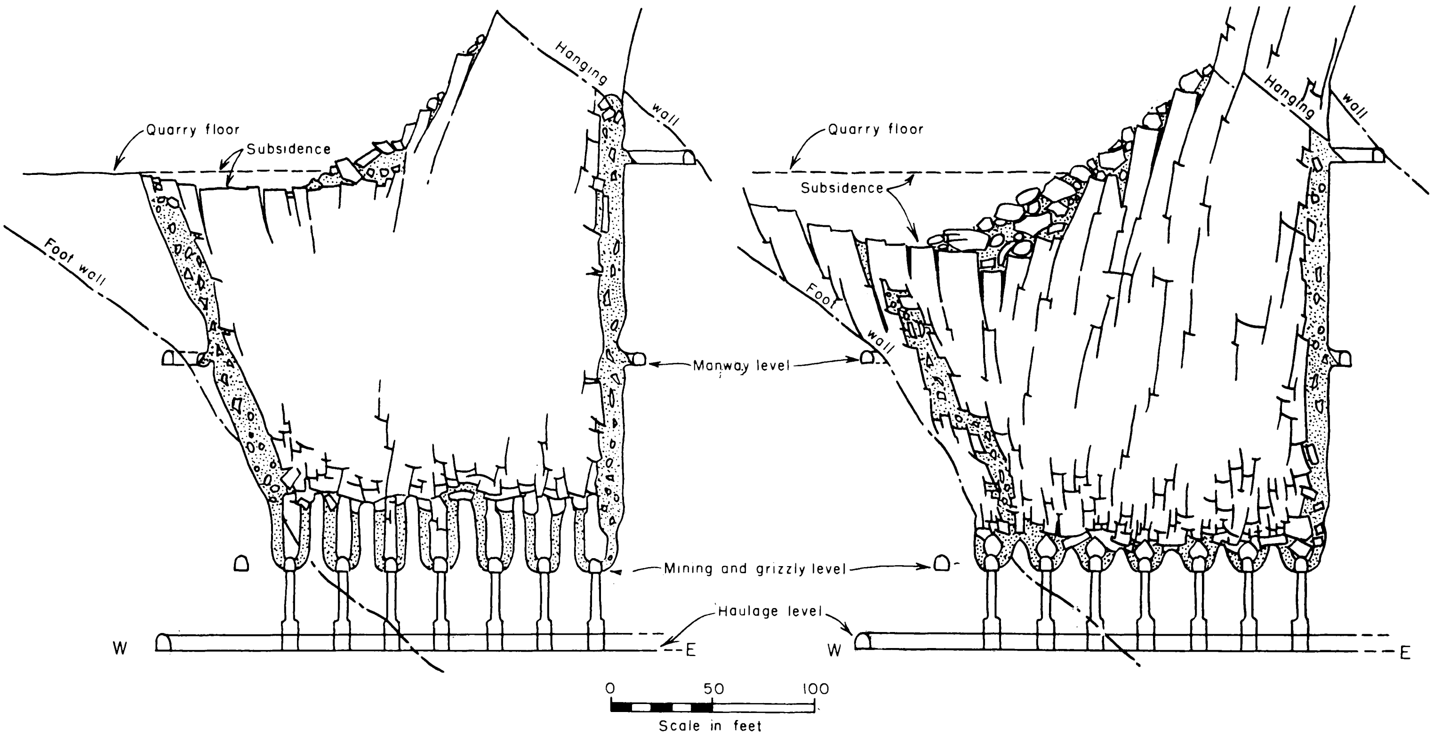


FIG. 10. Section A–A′ through Block 1A after vertical pillars *Z* and *W* had been broken by blasting, Crestmore Mine.(3)

FIG. 11. Section A–A′ through Block 1A, showing the initiation of caving action after remnants of the undercut pillars had been removed by blasting, Crestmore Mine.(3)

Caving Methods

To induce and sustain caving in the ground it was necessary to effect measures that would concentrate the stress generated by the weight of a thin cover on localized areas near the lower surface of the blocks. These measures were either unique to this operation or were carried out to a degree greater than was required in normal caving. They were as follows:

Undercutting

(1) The blocks were completely cut off on all sides by either vertical shrinkage (cutoff) stopes or by caved areas. The cutoff stopes extended from the mining and grizzly levels upward to the quarry floor or to the hanging-wall contact.

(2) The area that undercut each block to start the caving was much greater than that undercut in normal block caving. At the Crestmore mine an undercut area about 80×200 ft (16,000 ft^2) was generally required to initiate the cave, and as much as 35,000 ft^2 was undercut in one instance without producing a cave. In some porphyritic copper ores, caving can be induced by undercutting an area as small as 625 ft^2.

Draw Control

(3) To sustain caving, a radically different system of draw control was developed. In normal block caving the most widely accepted practice is to pull equal tonnage from each drawpoint according to a predetermined daily schedule. By following this procedure, funneling is minimized. At Crestmore no draw schedule was used, and pulls were confined to those drawpoints underlying the areas of the block highly enough stressed to fracture and break the rock. The area under high stress was usually indicated by the presence of a relatively copious amount of powdered rock in the drawpoint chutes. Pulls were made and continued only from drawpoints where the powdered rock was evident until a substantial void, possibly 75 ft in diameter, had been created. This arch over the void thus transmitted and concentrated the load to other localized areas in the block, and the pulling accordingly was shifted to the drawpoints, where the presence of powdered rock in the chutes indicated a high stress area. Once the draw was started from an area, it had to be continued at a rate sufficient to maintain a void. If the blocks were allowed to settle on the broken rock, it would reconsolidate, and a considerable effort would be required to re-initiate the cave. Sometimes reconsolidation was extensive enough so that the block had to be almost completely reundercut to restart the caving action.

Unusual Features

The mining method developed at the Crestmore mine was characterized by a number of factors, some of which are seldom experienced or applied in mining porphyry copper and other similar deposits by block-caving methods. Some of these factors that contributed to the success of the operation at Crestmore might be advantageously applied to other caving operations where the characteristics of the ore and country rock materials are similar to those at Crestmore. These factors are:

(1) The rock arched instead of funneling when a sustained pull was confined to a particular set of contiguous drawpoints.

(2) Even though most of the daily production could usually be drawn from a single producing block, the amount pulled per day ranged from 700 to about 3000 tons during the life of the mine. Under average caving conditions in other formations, as the porphyry coppers, this tonnage is a limited production. Factors limiting daily production in the Crestmore mine were:

(a) After an arched void developed, some time was required to allow the stresses in the block to readjust and for them to transfer and be concentrated to some area in the rock near the periphery of the void. The material in this peripheral area was overstressed and crushed, and some powdered rock was produced. This powdered rock filtered downward through the broken zone toward the drawpoint chutes, where it accumulated. A considerable time might elapse before enough powdered rock accumulated in one or several drawpoints to become noticeable and hence serve as an indicator that the rock overlying these drawpoints was broken enough to be drawn. The readjustment period and the time spent in seeking the drawable chutes, as evidenced by the accumulated rock powder, was non-productive.

(b) The draw must be confined to a restricted area of the block that can be pulled only from a few, usually contiguous, drawpoints.

(c) The location of the drawable area shifted erratically. No set routing of mine cars to specific transfer raises was possible, and a continual, time-consuming rerouting of car trains was unavoidable.

(3) About 30 per cent of the limestone drawn was oversize. These oversize pieces were broken, either in the drawpoint chutes or on the grizzly level, by blockholing, mud capping, and bombs.

(4) Difficulty was experienced in caving the last 25 per cent of the block. This was especially true where the granodiorite cover was thin or nonexistent. Evidently the weight of the last 25 per cent of the block and cover material was not great enough to cause good fragmentation.

(5) The major portion of the granodiorite cover crushed to minus-$\frac{1}{4}$-in. material that dribbled down through the coarsely broken limestone blocks. As a result, the limestone was substantially diluted by the granodiorite. Dilution is a serious disadvantage in most mines, but at the Crestmore mine it was advantageous because

20 per cent of a material containing alumina and silica was needed, plus the Stanley limestone, to make a product from which a portland cement could be produced. The granodiorite, which was drawn with the limestone as a diluent, served as the source of the needed alumina–silica material.

(6) The strength of the limestone was such that little support was needed in either the haulage or grizzly levels. As the first blocks were developed, the grizzlies and drawpoint chutes were reinforced with concrete. After a period of operation this concreting was found to be unnecessary and was discontinued. The fact that little support and virtually no maintenance were required in the underground workings contributed substantially to the economic efficiency and safety of the Crestmore operation.

Advantages

Block caving in hard rock as practiced at the Crestmore mine had several advantages over conventional caving as practiced in the porphyry copper ores:

(1) The block development could be made well in advance of putting the block into production, thereby allowing flexible mine production.

(2) The method was satisfactory for a relatively low daily production. The draw from one block could be maintained at a rate as low as 700 tons/day.

(3) Timber and other support in haulage, grizzly, and undercut levels and in shrinkage stopes were generally unnecessary.

(4) Good efficiency was achieved, as indicated by an 8-year average production of 32.25 tons/man-shift.

(5) Stable ground conditions contributed to achievement of an exceptional safety record.

Disadvantages

(1) A large proportion of the draw was oversize. About 30 per cent of the total draw had to be broken in the chutes or on the grizzlies by secondary blasting.

(2) Considerable difficulty was experienced in recovering the final 25 per cent of each block because the internal stress in the remnant block resulting from the overlying rock and cover was too low to cause the block to fragment.

(3) Considerable dilution from the cover must be expected, especially during the final phase of the draw from the block.

Block caving was originally chosen (in the 1930's) over open stoping for mining this limestone deposit because it was the most economical mining method at that time. The improved, large scale underground mining machinery introduced since the war, together with the introduction of low cost fertilizer-grade ammonium nitrate for blasting have now tipped the economic balance in favor of open stoping with heavy equipment in mineral deposits such as this and the limestone is now mined by open stoping methods.

BLOCK CAVING IN COPPER ORES

Miami Copper Co., Ariz.

Figures 12 and 13 illustrate block caving methods formerly used at Miami Copper Co., Miami, Arizona, and the revised method adopted to allow mining of ore

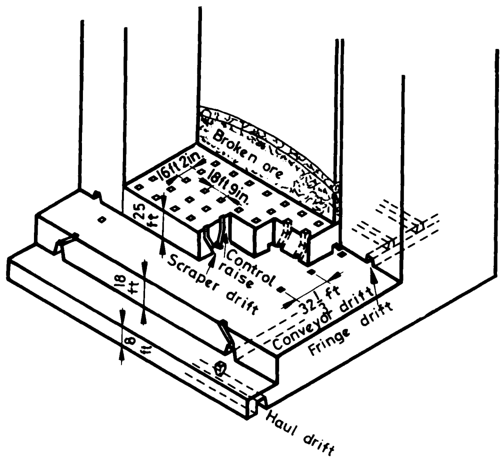


FIG. 12. Scraper–conveyor system of block caving developed to extract ore which bottomed close to the haulage level, Miami Mine.[4]

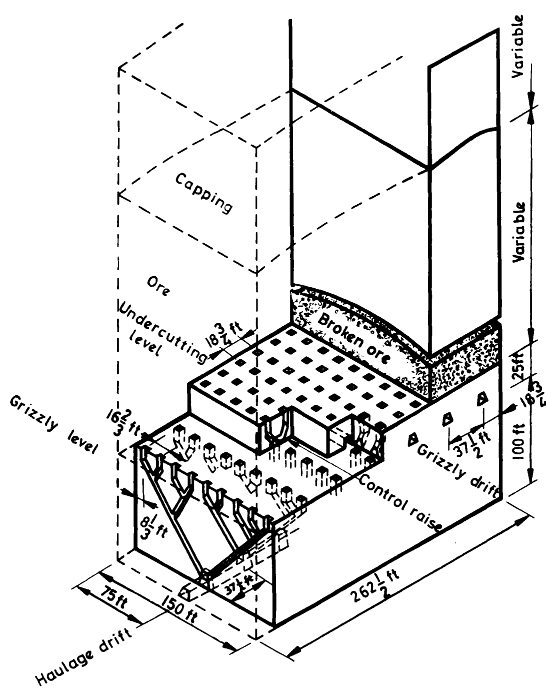


FIG. 13. Full-gravity transfer system of block caving at Miami Mine, which was replaced by the scraper–conveyor system. The thick pillar required by this method could not be used where ore bottomed close to the haulage level.[4]

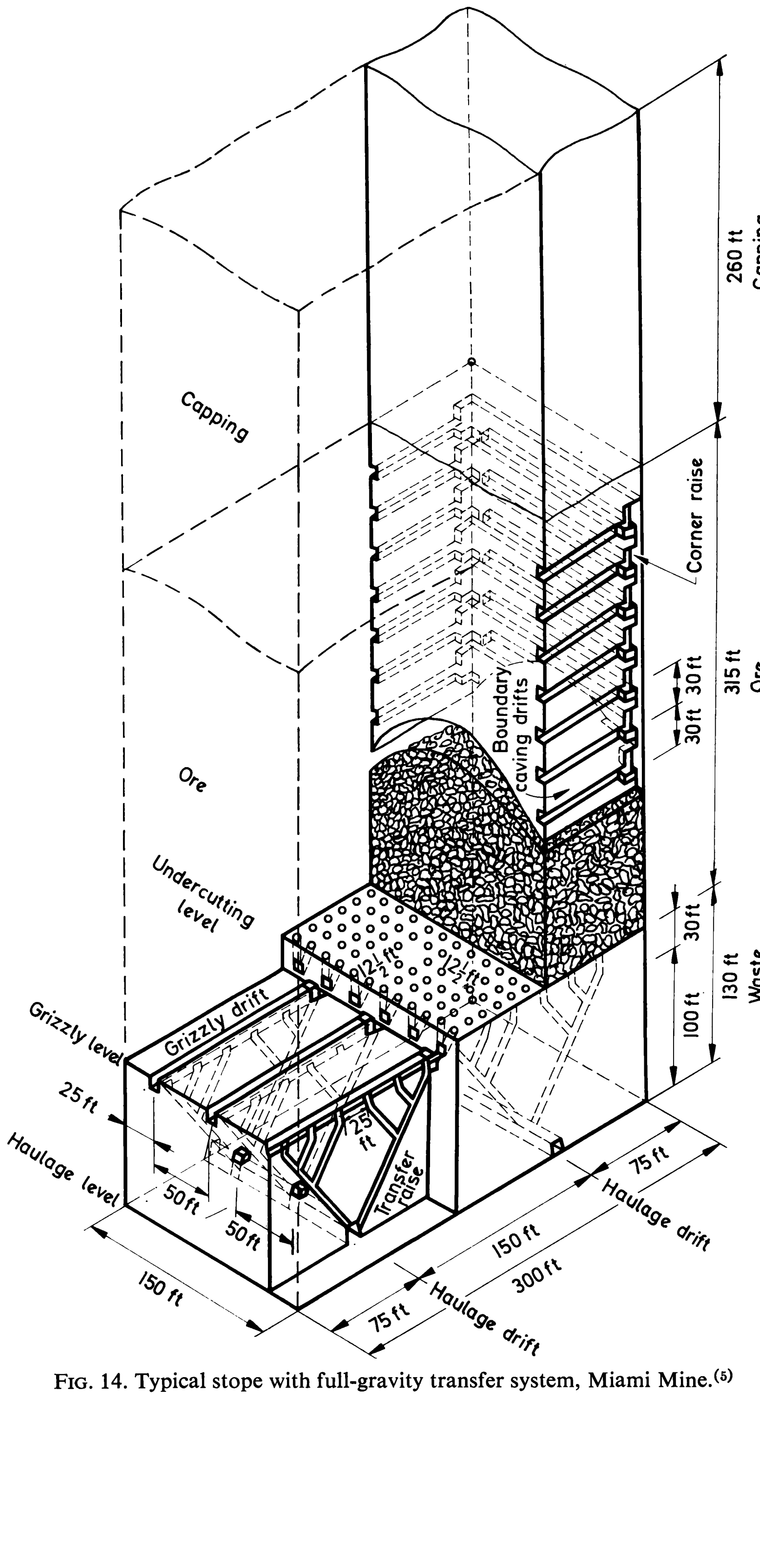


FIG. 14. Typical stope with full-gravity transfer system, Miami Mine.[(5)]

which bottomed a short distance above the main haulage level. In the 40 years from 1910 to 1951 nearly 125 million tons of copper ore were mined from the massive deposit.

Figure 14 is an isometric drawing of a typical stope at the Miami Copper Co. The Miami orebody averaged from 500 to 600 ft in thickness and was overlain by from 250 to 500 ft of barren capping. The orebody, viewed in plan, was a triangle, 3700 ft at the base and 2500 ft in altitude. Average copper content was about 0.8 per cent.

For most of its productive life the caving blocks were extracted through a pillar, 125 ft thick, which was left between the main haulage level and the undercut level (bottoms of the caving blocks) above. When it developed that a large part of the remaining ore reserves bottomed too close to the main haulage level to allow use of the thick pillar, pillar thickness was reduced to 50 ft and scrapers in scraper drifts scraped ore to conveyor drifts in which ore was conveyed to main haulage drifts.

TABLE 2. STOPE EFFICIENCY (1944), BAGDAD MINE[6]

Tons hauled in 2-week periods	Muck crew		Repair crew		Total	
	Shifts	Tons per man-shift	Shifts	Tons per man-shift	Shifts	Tons per man-shift
29,968	972	30.83	72	416.2	1044	28.7
28,272	887	31.87	53	533.4	940	30.4
24,175	779	31.03	44	549.4	823	29.4
25,795	855	30.17	63	409.4	918	28.1
31,216	1009.75	34.87	86	363.0	1095.75	28.5
32,054	984	32.58	71	451.5	1055	30.4
28,539	769	37.11	62	460.3	831	34.3
29,805	879	33.91	99	301.0	978	30.5
35,108	959.50	36.59	72	487.6	1031.50	34.0
29,798	874	34.09	67	444.7	941	31.7
32,408	870	37.25	76	426.4	946	34.3
25,358	738	34.36	72	352.2	810	31.3
21,980	643.50	34.16	58	379.0	701.50	31.3
23,105	650	35.54	69	334.9	719	32.2
27,164	841.50	32.28	66	411.6	907.50	29.9
25,463	754.50	33.75	57	446.7	811.50	31.4
30,928	873.75	35.40	53	583.5	926.75	33.3
28,570	753	37.94	50	565.4	803	35.6
29,910	875	34.18	43	695.6	918	32.6
29,601	805	36.77	75	394.7	880	33.6
34,204	878	38.96	115	297.4	993	34.4
29,703	749	39.65	78	380.8	827	35.9
29,346	723	40.58	81	362.3	804	36.5
29,201	730.50	39.97	35	834.3	765.50	38.1
691,671	19,853.0	34.8	1617	427.7	21,470	32.2

The thinner pillar resulted in increased weight on slusher, conveyor, and haulage drifts, with consequent crushing of supports. This problem was overcome by installing circular steel sets in slusher drifts, and I-beam steel sets for support of haulage and conveyor drifts.

Bagdad Mine, Arizona

Tables 2–6 summarize the productivity and cost records for the block caving operation of the Bagdad Mine of the Bagdad Copper Corp., Yavapai County, Arizona.

TABLE 3. COMPARISON OF STOPE AND GLORY-HOLE EFFICIENCIES (METHODS USED DURING 1945 AND 1946), BAGDAD MINE[6]

Stope			Glory hole		
Tons in 2-week periods	Shifts	Tons/shift	Tons in 2-week periods	Shifts	Tons/shift
2714	296	9.17	35,800	379	94.46
1609	254	6.33	33,417	361	92.57
3596	325	11.06	36,787	418	88.01
2368	259	9.14	35,507	410	86.60
1830	216	8.47	33,310	343	97.11
2629	270	9.74	35,249	370	95.27
6670	326	20.46	35,875	384	93.42
4318	308	14.02	34,662	350	99.03
5561	191	29.12	29,900	316	94.62
9504	441	21.55	31,799	355	89.50
11,013	645	17.07	34,500	376	91.76
11,649	615	18.94	26,356	275	95.84
7777	458	16.98	25,427	283	89.85
10,414	545	19.11	23,945	310	77.24
13,939	616	22.63	14,469	193	74.97
12,251	547	22.40	13,579	190	71.47
107,842	6312	17.09	480,582	5313	90.45

Block caving operations were first started in 1937 and operated at a relatively moderate production rate until high copper prices during the war stimulated expansion. Production was expanded but the rock was hard and the orebody was thin and ore to supplement underground production was obtained by "glory hole" operations on the surface.

In 1948 underground mining was discontinued and the operation was converted to an open pit mine.

TABLE 4. STOPE RECORD, BAGDAD MINE[6]

Stope No.	Estimated			Drawn			Extraction (%)		
	Tons	Copper (%)	Copper (lb.)	Tons	Copper (%)	Copper (lb.)	Tons	Grade	Cop-per
1	102,300	1.495	3,058,770	104,808	1.396	2,926,239	102.4	93.4	95.7
2	98,800	1.697	3,353,272	103,571	1.525	3,262,486	104.8	89.9	97.3
3	102,470	1.890	3,874,000	118,533	1.370	3,237,672	115.6	72.5	83.5
4	99,800	1.530	3,084,442	66,249	0.860	1,142,100	66.4	56.2	37.0
5	95,950	1.150	2,229,850	37,390	0.910	679,700	38.5	79.1	30.5
6	110,000	1.975	4,345,000	113,807	1.380	3,142,364	103.5	69.9	72.4
7	98,800	1.851	3,657,576	111,981	1.490	3,342,170	113.3	80.5	91.4
8	97,200	1.602	3,114,290	104,465	1.230	2,574,172	107.5	76.8	82.7
9	96,100	1.370	2,634,000	36,546	0.920	676,100	38.0	67.2	25.7
10	161,800	1.200	3,883,200	134,359	0.740	2,000,700	83.0	61.7	51.5
11	138,760	1.330	3,699,342	123,500	1.100	2,722,650	89.0	82.7	73.7
12	132,720	1.805	4,777,920	166,150	1.290	4,301,287	125.1	71.5	90.0
14	94,650	1.270	2,404,110	77,639	0.770	1,195,961	82.0	60.6	49.7
15	93,470	1.360	2,542,380	77.174	0.930	1,432,427	82.6	68.4	56.3
16	100,050	1.569	3,139,570	65,880	0.890	1,169,420	65.8	56.7	37.6
Total	1,662,760	1.534	49,797,722	1,442,052	1.172	33,805,448	88.9	76.4	67.9

TABLE 5. PRODUCTION BY YEARS, BAGDAD MINE[6]

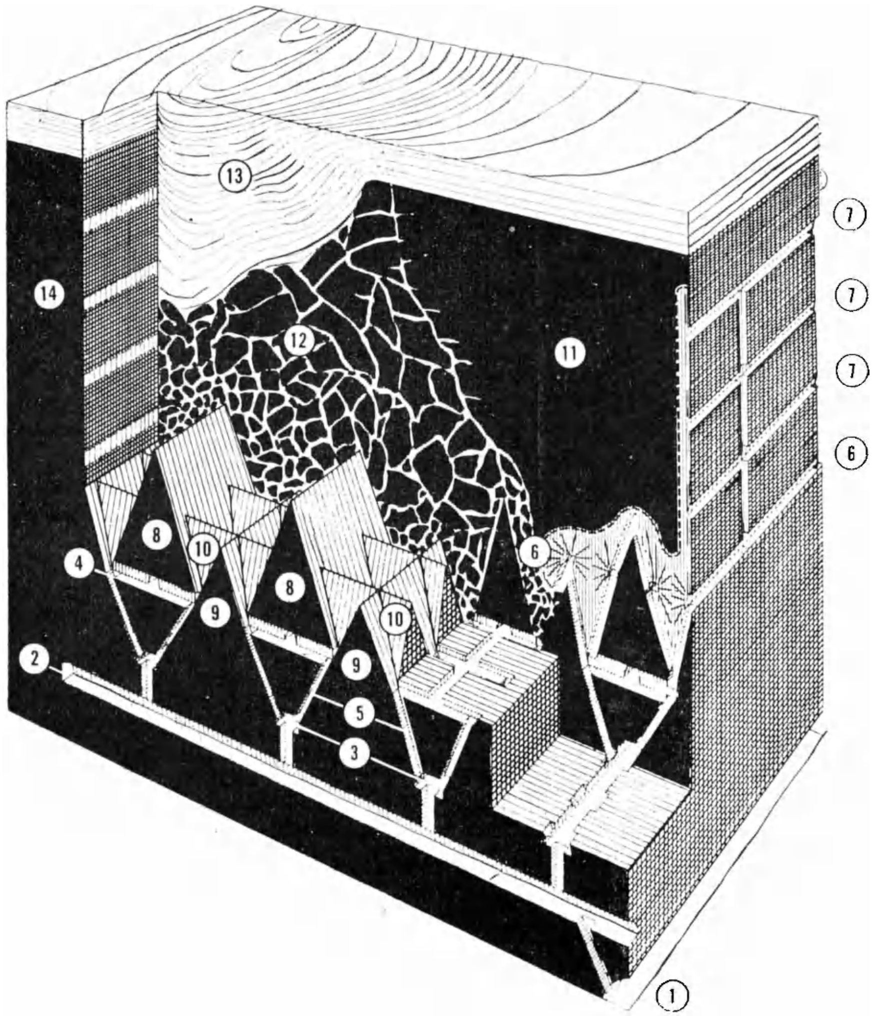
Year	Mill capacity (tons per day)	Tons milled	Copper produced (lb.)
1937	250	75,512	1,537,396
1938*	250	29,200	681,000†
1939*	275	14,196	297,000†
1940	275	73,600	1,540,000
1941	275	88,200	2,298,000
1942	275	67,900	1,090,000
1943	2000‡	377,271	7,370,934
1944	2000	682,484	9,818,181
1945	3000	618,711	8,229,049
1946	3000	862,535	12,226,848
1947§	3000	957,302	13,569,945
Total	–	3,846,911	58,658,353

* Operations terminated May 1, 1938, and resumed Nov. 1, 1939.

† Estimated.

‡ New mill opened March 1943.

§ Partly by open-pit method.



LEGEND

1. Main haulage drift
2. Scraper drift
3. Shaker drift
4. Grizzly drift
5. Ore passes
6. Undercuts
7. Undercuts
8. Main roofs above grizzly and shaker drifts
9. Intermediate roofs parallel to main roofs
10. Cross roofs at right angles to 8 and 9
11. Standing block
12. Broken block
13. Caving mass
14. Leg pillar, or adjoining block

FIG. 15. Block caving method used in mining the "brown iron ores" of the Salzgitter District near Braunschweig, Germany.(7)

Although the new concentrating plant was completed and placed in production in March 1943, the shortage of skilled miners and the lag in stope development prevented daily mine production from reaching the 2500-ton rate until production from caving was augmented by some ore from glory-hole operation. Production by years during the life of the underground operation is shown in Table 5.

TABLE 6. OPERATION COSTS, 1943 (TYPICAL MONTH, 38,245 TONS MILLED), BAGDAD MINE[6]

Mining	Labor	Supplies	Power	Miscel- laneous	Total	Cost per ton
Stope preparation	–	–	–	$9,070.89	$9,070.89	$0.237
Drawing and tramming	$12,576.10	$2,805.60	$84.37	30.00	15,496.07	0.405
Maintenance	1241.13	60.00	–	–	1,301.80	0.034
Hoisting	1740.98	207.90	162.50	–	2,061.38	0.054
Pumping	6.82	64.55	30.80	–	102.17	0.003
Ventilation	13.64	52.86	42.64	–	159.14	0.004
Proportion of general	–	–	–	8187.76	8,187.76	0.214
Total	15,578.67	3,191.58	320.31	17,288.65	36,379.21	0.951

Milling	Labor	Supplies	Parts	Power	Miscel- laneous	Total	Cost per ton
Crushing	$1134.51	$493.48	$489.91	$211.60	–	$2329.50	$0.061
Ball mills	774.26	5311.05	661.44	1468.05	–	8214.80	0.215
Flotation, copper	720.03	1512.00	618.16	432.72	–	3282.91	0.086
Flotation, moly	1095.56	558.52	464.02	55.28	–	2173.38	0.057
Filtering	917.73	22.17	145.72	78.63	–	1144.25	0.029
Tailings disposal	1380.68	19.19	236.34	–	–	2136.21	0.056
Water	921.44	304.86	24.56	500.39	–	1751.25	0.045
Proportion of general	–	–	–	–	$4371.19	4371.19	0.114
Total	6944.21	8201.27	3190.15	2746.67	4371.19	25,403.49	0.662
Cost per ton	$0.118	$0.214	$0.081	$0.072	$0.114	$0.662	

BLOCK CAVING IN A BORATE MINE

At the Jennifer Mine, in Kern County, Calif., an attempt was made (1952–1954) on an experimental basis to mine sodium borate ores by block caving.[8]

A block 272 ft long and ranging from 113 to 149 ft wide was laid out and development work started late in 1952. Because of the low-strength shale footwall most of the sub-block development openings (extraction haulageway drifts and draw raises) were located in the lowermost 22 ft of the ore body.

Depth of cover over the caving block was 300 ft. The thickness and types of strata overlying the orebody are indicated in Fig. 18.

Development of the cave block was started in 1952. All the extraction, undercut, cutoff, and observation drifting was done with continuous miners. All draw or finger, ventilation, and observation raises were developed by conventional drilling and blasting.

No difficulty was experienced in either the drifting or raising operations, and virtually no roof support problems were encountered. As a precautionary measure, the 12-ft wide by 8-ft high extraction drifts, directly under the area to be caved were timbered on 4 ft centers and lagged. These supports proved to be purely

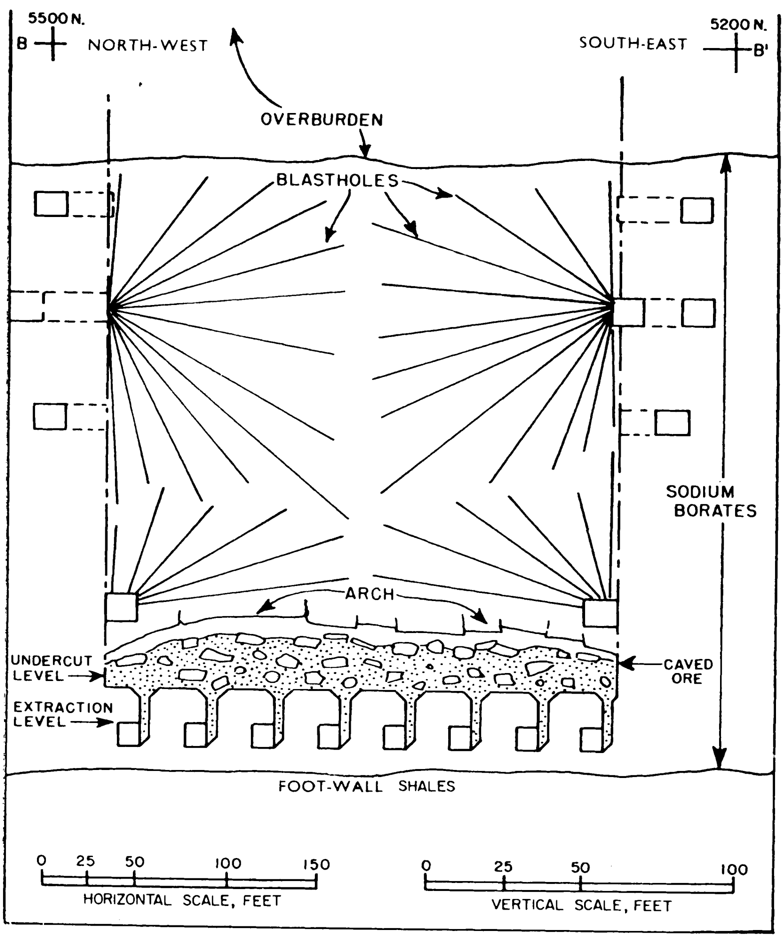


FIG. 16. Idealized cross-section showing arch which developed after the support pillars had been removed leaving an undercut area 272 × 130 ft (35,000 ft^2).

Also shown are some of the blastholes which were drilled, loaded with dynamite, and blasted to break up the borate ore and re-initiate caving.[8]

precautionary as no failures in the roof pillars over the extraction drifts were noted during the draw, or as late as 1959 when it was easy to salvage the timber.

The first of the undercut pillars was blasted out on April 6, 1953 and by April 15 approximately 65 per cent of the total area of the block had been undercut, and draw from the block was started immediately on a small scale.

Remaining undercut pillars were removed during the latter part of July, 1953, and at this time an area approximately 272 ft $\times$ 130 ft had been undercut and a systematic draw from the block initiated. However, the fragmentation was poor and the draw contained large slabs which had to be broken by blasting in the chutes.

Late in August, after some 15,000 tons of material had been drawn, it was found, through inspections conducted in the observation stub drifts, that the block was not caving. An arch had formed and the void between the top of the broken material and the arched roof was being formed at approximately 40 ft above the formed floor of the undercut level, as shown in Fig. 16.

Because a sustained cave could not be produced, it was decided to break the block by longhole blasting. About 150 holes, $1\frac{7}{8}$ in. in diameter and 20–70 ft long, were fanned out in all directions from the ends of the stub observation drifts and blasted on October 24, 1953.

The blast resulted in the immediate collapse of the entire block which was accompanied by a weak, underground air blast. Subsidence extending to the surface as a downward surface displacement of about 6 in. occurred almost simultaneously with the blast.

Fragmentation

The fragmentation produced by both self caving and induced caving contained an excess of large slabby blocks. This was due partly to the absence of joints or fractures in the ore, and partly to the plastic qualities of the clay–borate rock. These large blocks (estimated to represent about 30 per cent of the draw) had to be broken by blasting them into fragments small enough to pass through the 5-ft diameter draw raises and the 3-ft wide chute openings.

The caving characteristics of an ore body depend on the stresses imposed on the block and on the macroscopic strength of the ore, including the fracturing and jointing. Laboratory tests on a few samples gave compressive strengths ranging from 1100 to 6400 psi, with the weaker ores having the higher borate content.

The ore failed to cave more freely because this block was only under cover of 300 ft, and hence under low average stress, and because the macroscopic strength of the ore was greater than that of most orebodies because of the absence of preexisting fractures, and joints.

Surface Subsidence

Throughout the draw period the area which initially subsided at the time of the blast continued to sink at a uniform rate until the subsidence pit was bounded by steep-faced (practically vertical) walls ranging from 30 to 50 ft in height. The

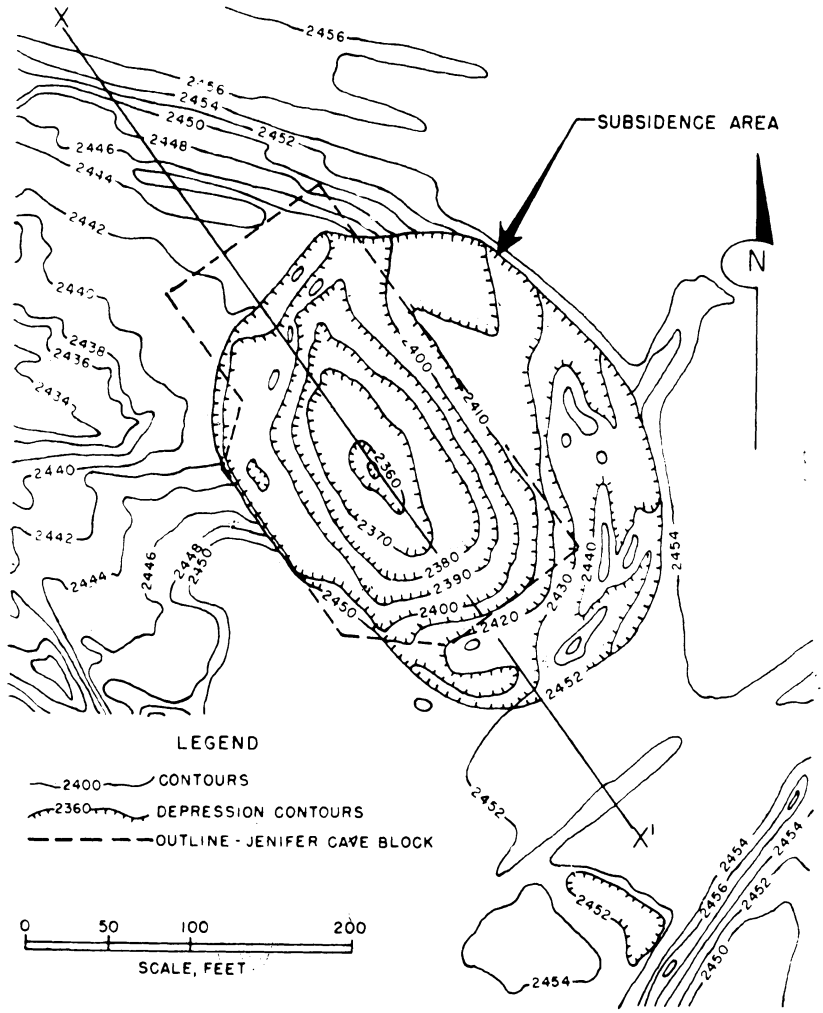


FIG. 17. Contour map showing the areal extent and configuration of the subsidence pit at the surface over the Jennifer Mine caving block.[8]

appearance and configuration of the subsidence pit changed very little between October 1955 and the last observation made in October, 1961.

One of the unique features of this subsidence pit was the fact that its periphery was marked by a single, continuous, steep walled face. Normally a subsided area, and especially its periphery, consists of a series of steplike terraces occurring at more or less progressively higher elevations as the horizontal distance outward from the center of the pit increases.

Figure 17 shows the contours of the subsidence pit while Fig. 18 is a cross-section through the caved block.

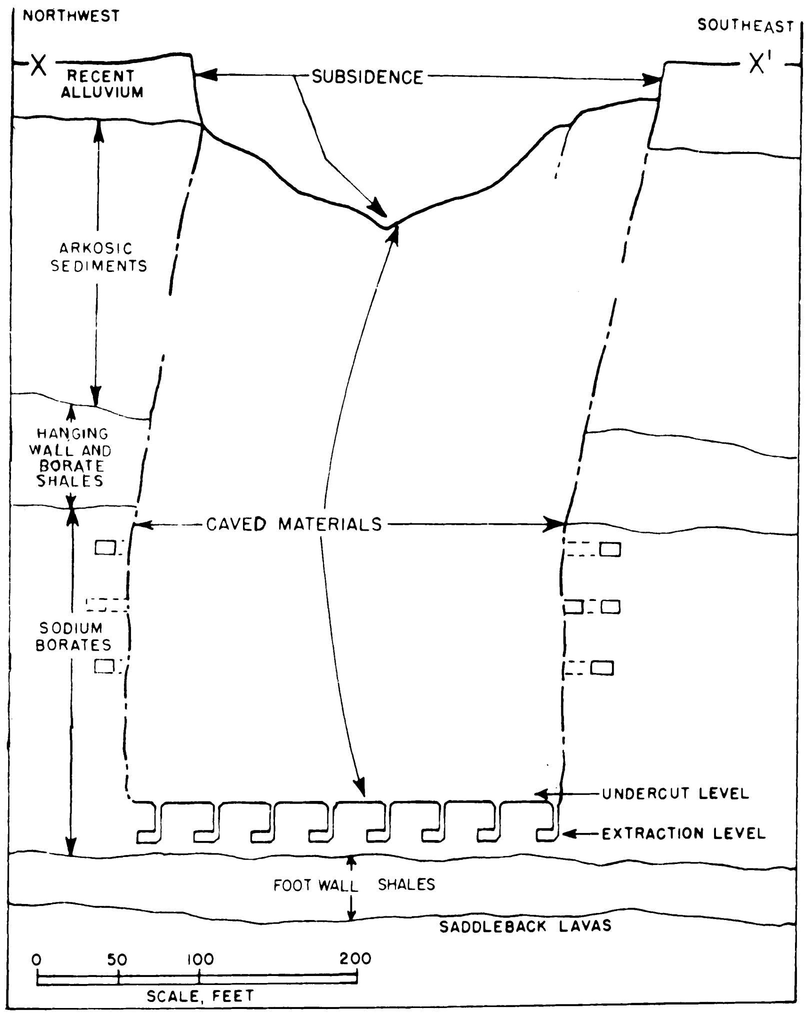


FIG. 18. Idealized cross-section, Line X–X′, Fig. 17, through the Jennifer caved and subsided area.[8]

Extraction

Total extraction from the block, including ore from development on the undercut level, was about 216,000 tons. This was about 70 per cent of the 308,000 tons calculated as available for draw in the block above the floor of the undercut level.

Hence some 30 per cent of the estimated extractable borax was not recovered. This loss resulted from the draw being discontinued either because dilution became apparent and/or because the approximately 25 per cent B_2O_3 grade ore then being produced by caving operations was less desirable as a mill feed than the 30 per cent ore that could be obtained from the fully developed and producing Jennifer Mine.

OBSERVATIONS OF GROUND MOVEMENT AND SURFACE SUBSIDENCE AT BLOCK CAVING OPERATIONS

The Angle of Draw

The term "angle of draw" is, by its nature, ambiguous and various writers have applied the term to different angles. Some writers use the term to describe the angle of inclination of a line drawn to connect the edge of an underground working with the outermost limit of surface subsidence, while others draw the line to the outermost limit of surface movement. Most writers measure the angle of draw from the horizontal but some measure it from the vertical.

The author prefers to take the angle of draw as being the angle of inclination (measured from the vertical) of a line drawn to connect the edge of the mine working with the outermost limit of surface subsidence.

The Angle of Break and the Angle of Subsidence

Johnson and Soulé[(10)] employ two very appropriate terms to describe angles between limiting points of ground movement in block caving operations.

The angle of break is the angle (measured from the horizontal) of inclination of a line drawn to connect the edge of the nearest underground working with the outer limit of surface fracturing. Usually the outermost surface fractures will be tension fractures which are the result of lateral surface movement rather than subsidence movement.

The angle of subsidence is the angle (measured from the horizontal) of inclination of a line drawn to connect the nearest underground working with the outer limit of the subsidence movement.

Thus the angle of draw as the term is used herein would be the complement of the angle of subsidence.

Ground Movement Observed Underground

The following observations of underground movement in connection with block caving operations are taken from a paper by Fletcher,[(11)] and pertain to observations at the Miami Mine of the Miami Copper Co., Miami, Arizona. This mine has been in operation since 1910, and between 1910 and 1925 24 million tons of high grade ore was mined by top-slicing, sub-level caving, etc. Between 1926 and 1954, 102 million tons of low grade ore was mined by block caving with an additional 10 million tons of mixed ore mined by the same method.

Between 1955 and 1960 an additional 23 million tons of low grade ore was mined by block caving.

Effect of Boundary Cutoff Drifts

The early stopes all had boundary cutoff drifts which served as observation points and in no case did the stopes cave outside a vertical line. In only one case was there cracking outside the stope boundary. A drift driven through a completely drawn stope, 80 ft above the undercut, showed that there had been no caving outside the vertical boundaries of the stope.

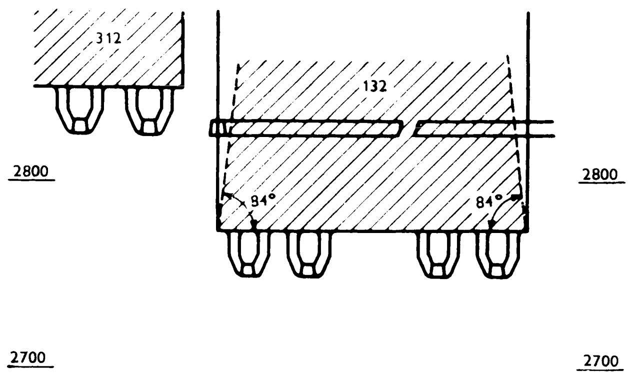


FIG. 19. Cross-section showing a drift driven through a caved block. The boundary planes of the caved zone were found to be inclined inward.(11)

All of the later stopes on the 1000 level were mined without boundary cutoff. In every case where it was possible to observe these stopes they caved inside the vertical stope boundaries. In No. 132 stope a drift was driven through the caved material and the boundary plane of the caved material was found to be inclined at an angle of 83° inside the stope boundary. This is illustrated in Fig. 19.

Caving Next to Broken Ground

Stopes next to broken ground tend to draw from outside the limits of the undercut block. Pillar stopes (ore left as pillars while blocks were caved between them) were structurally weaker than the original stopes. Where pillar stopes were mined in weak ground the ground was heavier than in the original stopes and a great deal of repair was required. This resulted in a poor draw.

Stresses in the Ground Around a Caving Block

The rock surrounding a caving block is under stress. The distance from the caving block that this stress is noticeable depends upon the strength of the rock.

A drift was driven through caved material in one block, and that portion located in the firm rock adjacent to the stope showed no timber failure 20 ft back from the caving ground.[11]

On the other hand the timber in an elevator penthouse in weak ground 100 ft from a block and 75 ft above the undercut level failed when the block was mined.

After a stope has been mined, the waste-fill consolidates and the stress in the adjacent rock is relieved. Workings can be driven alongside and into this material.

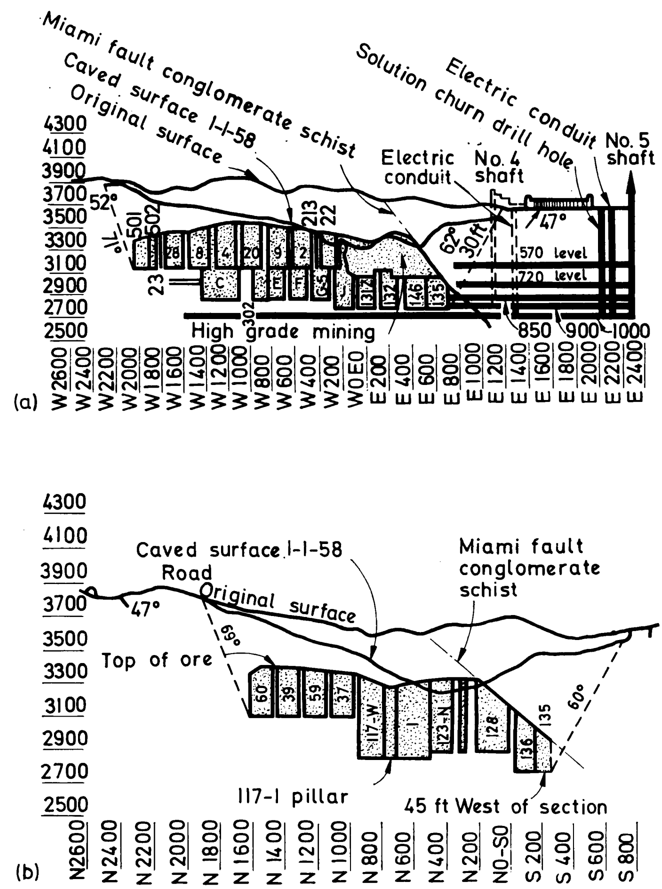


FIG. 20. (a) Cross-section through a caved area at the Miami Mine showing surface subsidence as of January 1, 1958 (N 400 Section). (b) Cross-section through caved area at the Miami Mine showing surface subsidence as of January 1, 1958. (E 800 Section).[11]

Ground Movement and Subsidence of the Surface

Observed Angles of Subsidence and Angles of Break

Figure 20 shows typical examples of the angles of subsidence and angles of break observed at the Miami Mine.[11] The angle of break, that is the angle from the undercut level to the outermost limits of surface cracking, averages about 45° in both schist and conglomerate.

The angle of subsidence generally ranged from 60 to 70°.

Figure 21 shows cross-sections through the ore body at the San Manuel Mine. The angle of break and the angle of subsidence are represented by dotted, and by dashed lines, respectively. It will be noted that the angle of break is generally much smaller than the angle of subsidence and generally is about 50–70°. The angle of subsidence generally lies in the range from 70 to 90°.

Figure 22 shows the subsidence of the ground surface at the Athens Mine, Negaunee, Mich. This was an iron ore deposit which was mined by top slicing methods rather than by block caving. However, the net effect of the ore removal is probably similar to that which would be produced by block caving. It will be noted that neither the angle of break, nor the angles of subsidence deviate very far from 90°.

Figures 17 and 18 show a plan and cross-section through a caved block at the Jennifer borax mine. In this case the angles of subsidence on two sides of the block were equal to 90° or more while the angle of subsidence on the remaining two sides ranged from about 70 to 90°.

Time Element in Subsidence

Observations at the Miami Mine,[11] indicated that the rate of progress of caving action from the undercut level upward to the surface depended to a great extent on the strength of the rock, although it is also influenced by the size of the caving block and the rate of draw; both of these latter factors are, however, largely determined by the strength of the rock.

In general when blocks of weak to medium rock were undercut 600–700 ft beneath the surface, and the normal pull was about 9 in./day, the cave would reach the surface in from 100 to 150 days.

Increase in Volume of Subsided Capping

At San Manuel[10] the ratio of the volume of ore mined to the volume of subsidence for a given period was about 1.44 to 1. It is expected that this ratio will decrease somewhat when mining in an area is stopped and some subsidence continues, largely because of the consolidation of the broken rock and the filling of existing voids.

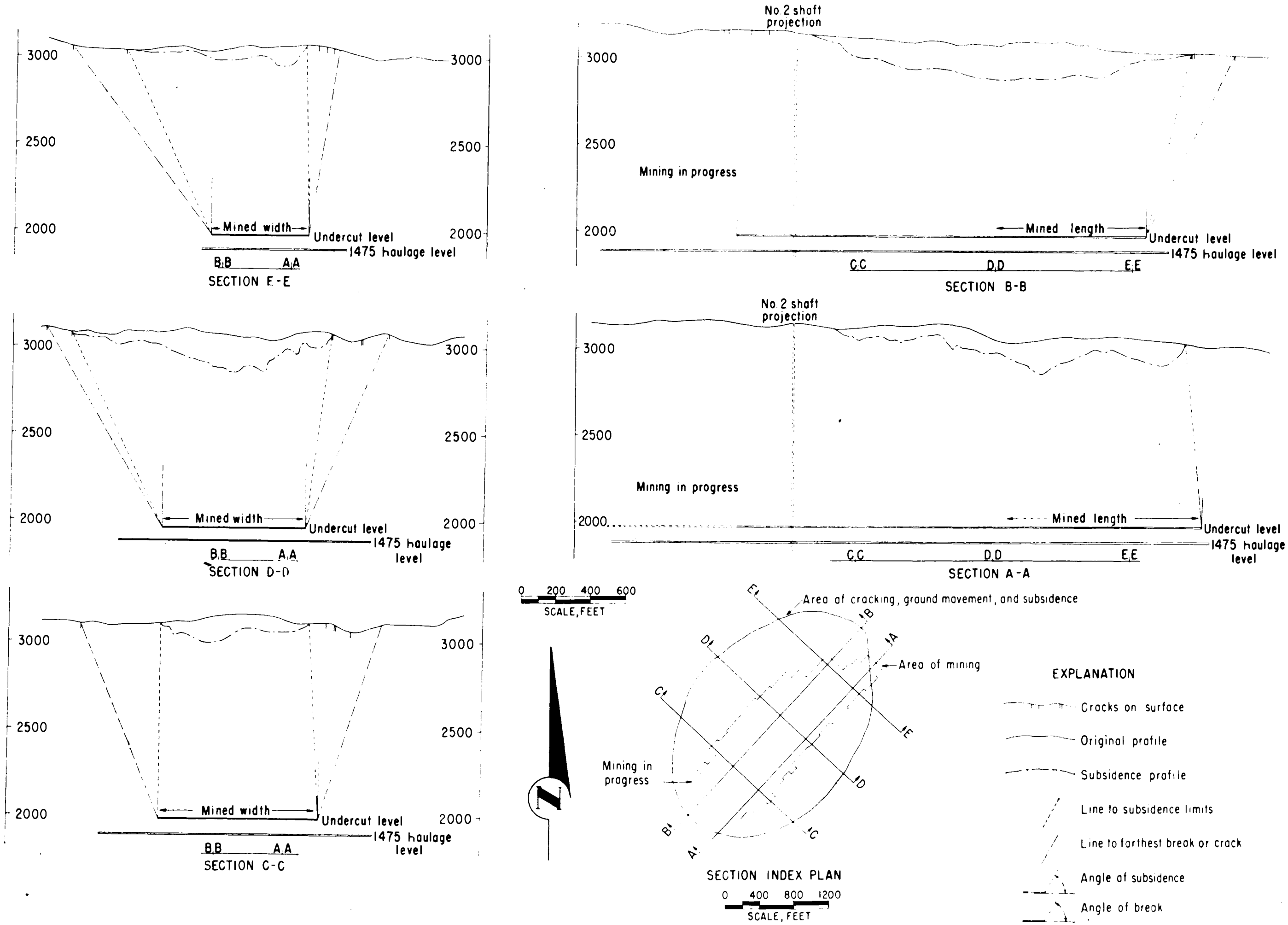


FIG. 21. Angles of subsidence and break along various cross-sections through the South Ore Body, San Manuel Mine.[10]

At Miami the rock in place averages about 12.5 ft^3/ton while the broken rock averages about 20 ft^3/ton. This would indicate a volume increase of about 60 per cent from the solid state to the loose state. However, caved material tends to re-

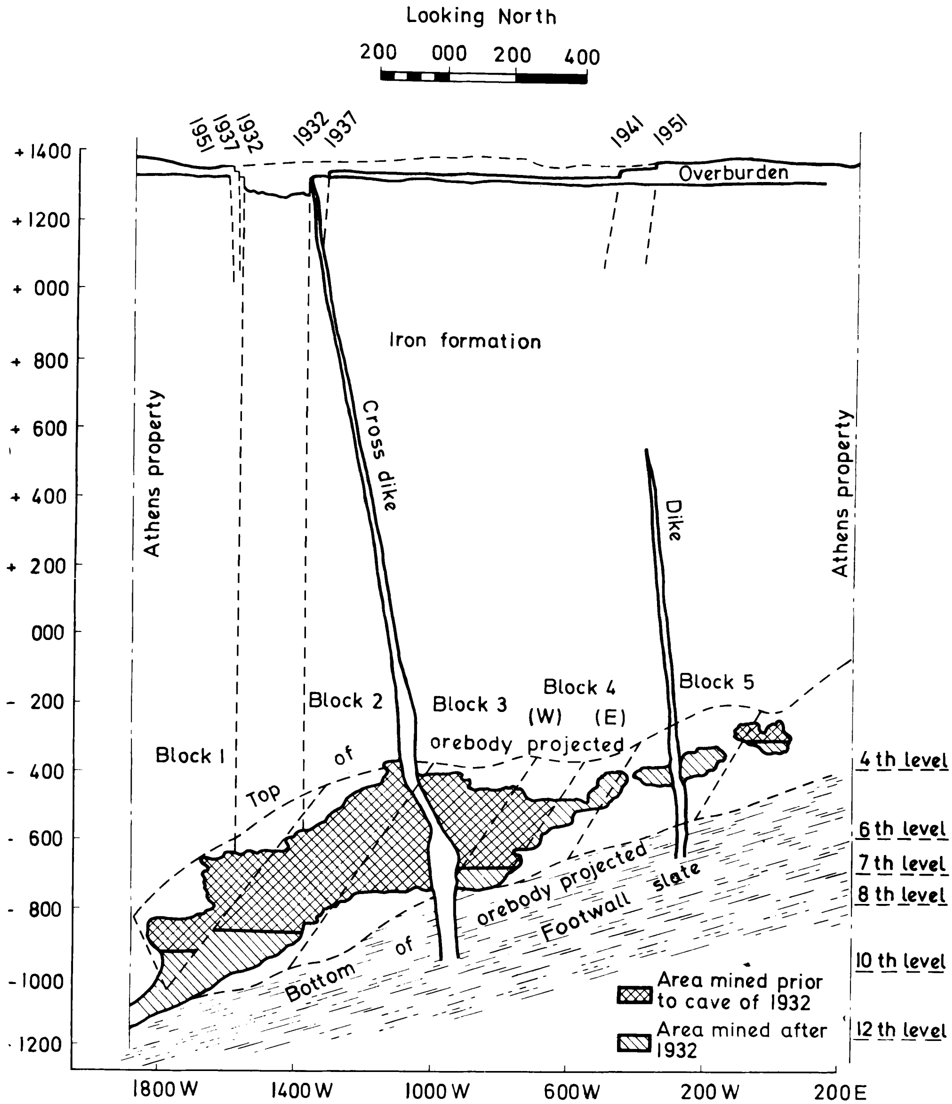


FIG. 22. East–west longitudinal section through the Athens Mine showing mined areas and surface subsidence.(12)

consolidate under the pressure of the rock column. Thus material at the bottom of the column will approach its original density while that at the top will be the least dense and the average density will lie somewhere between the two extremes. It has also been observed that the amount of swell varies with the type of rock.

At the Inspiration Consolidated Copper Co. it has been found, in open pit mining adjacent to block-caved stopes, that this caved and re-consolidated material averages about 16 ft^3/ton as compared with 12.5 ft^3/ton for rock in place. This is equivalent to a volume increase of about 28 per cent.

Surface Features of Subsidence

Surface features of subsidence include a central depression, usually centered near the center of the mined-out area, which is bounded by steep scarps along

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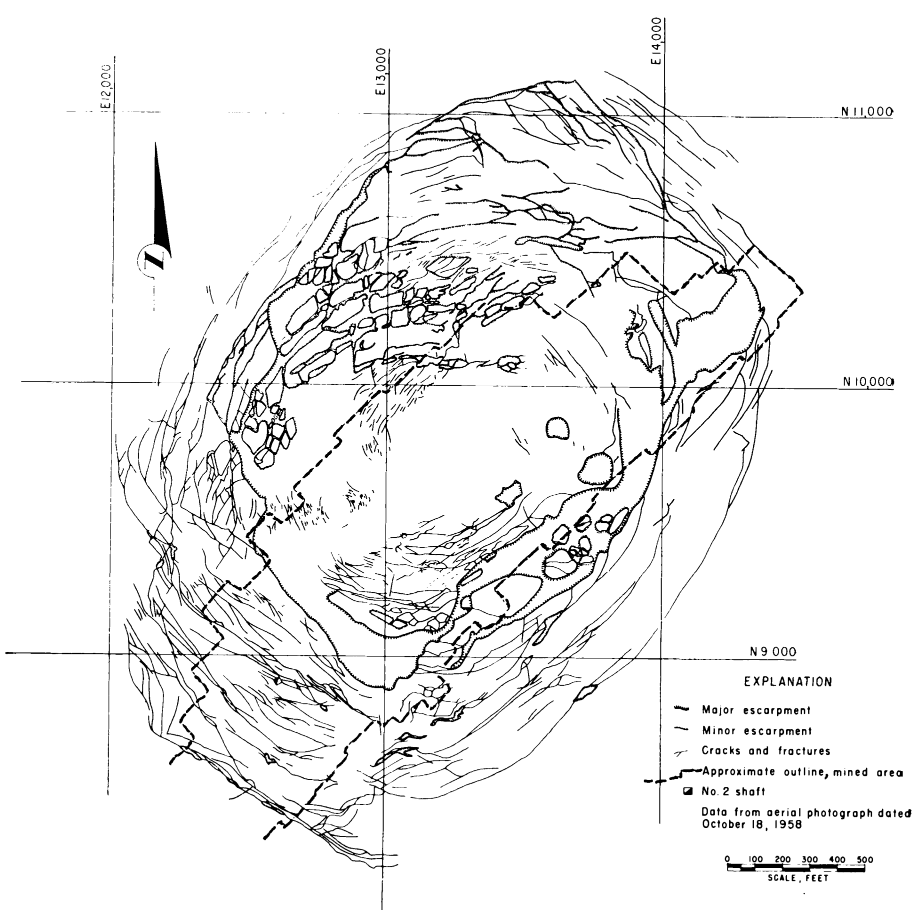

FIG. 23. Surface subsidence and fracture pattern over the South Ore Body, San Manuel Mine.[10]

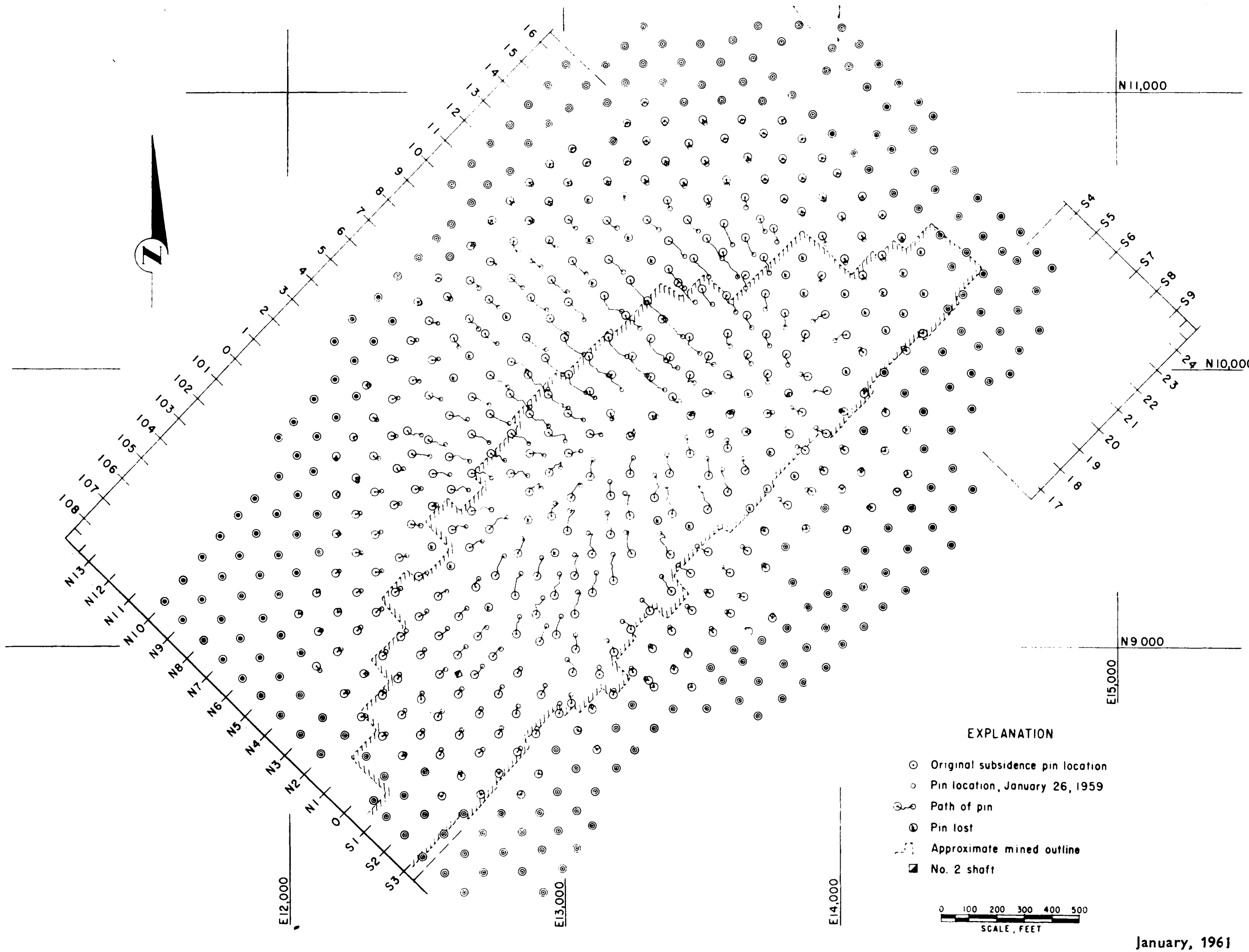

FIG. 24. Locations of triangulation pins for determining surface movements and pin movements to January 26, 1959.[10]

which shearing movements have taken place as the central core moved downward. Surrounding this central area are minor scarps, decreasing in height as they become more distant from the central area and finally grading into tension cracks which have been formed, not by vertical movement but by lateral movement as the ground surface has tilted toward the central depression.

Figure 23 is a plan view showing the subsidence and fracture pattern over the South Ore Body at the San Manuel Mine. Figure 24 shows the horizontal components of ground movement up until January 26, 1959. A grid of triangulation pins

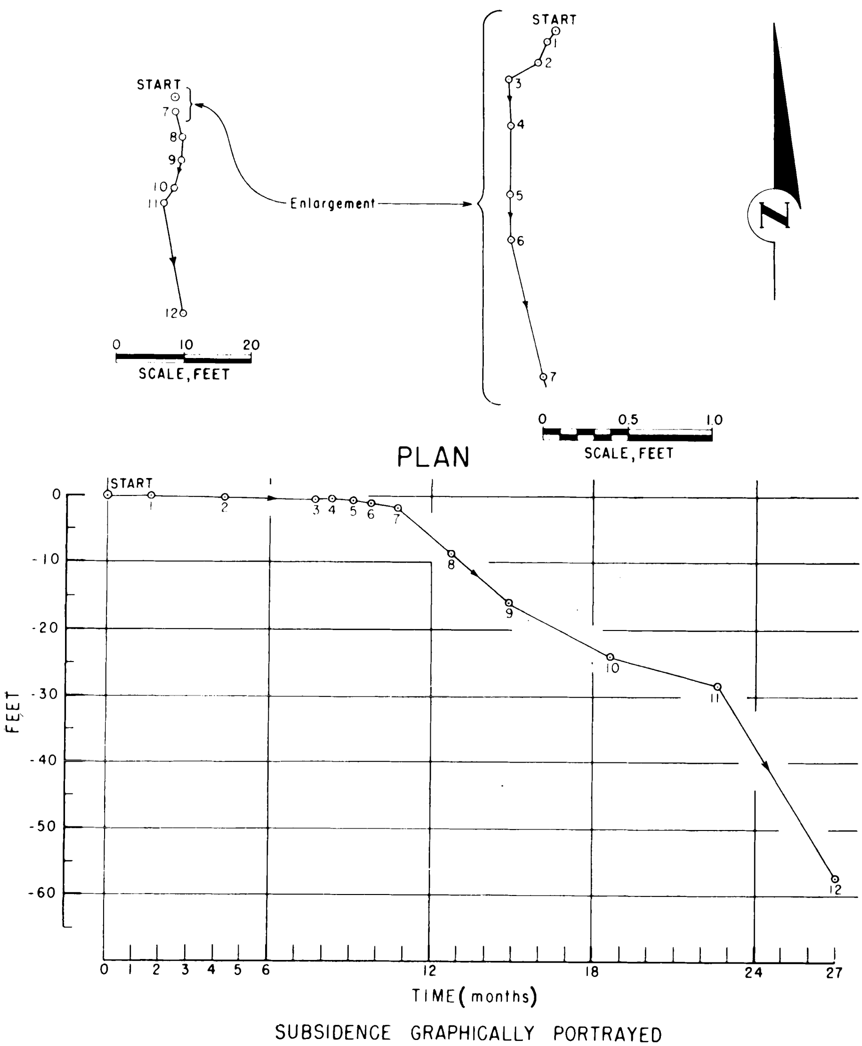

FIG. 25. Subsidence-pin movement, Pin No. 18.[10]

was established on the surface and the position of each pin was determined by triangulation measurements at intervals after the start of mining.

Figure 25 is a record of the lateral and the vertical movement of one triangulation pin during a 27-month period.

The Mechanics of Subsidence

Fletcher[11] has made the following observations regarding the mechanics of block caving subsidence at the Miami Mine.

The great number of tension cracks in the subsidence area and the absence of any observed shear planes (except along the Pinto fault) suggest that the tensile and compressive strength are the governing factors in subsidence and block caving at the Miami mine. Immediately after undercutting a block the roof begins to act as a beam and fails in tension on the underside and forms an arch (or dome) which continues to fail as the span of the arch is too great for the strength of the rock.

When the arch reaches the waste capping the crown of the arch breaks into the capping before the haunches. The waste thus broken in the crown of the arch mixes with the ore in the haunches of the arch and is a major source of ore dilution.

"Piping" is the term used to describe small break-throughs to surface. Piping is caused by small arches occurring in the broken material and working their way through this material up to the surface. Piping may occur after the entire stope has broken through to the surface. The best preventative is a uniform draw although piping can also be caused by too-wide spacing of the draw points.

As a caving arch approaches the surface a small sag area forms and this causes tension cracks to open up at the outside of the stope limits. When this point is reached the collapse of the surface is very rapid.

Stress Conditions

There is considerable evidence that the ground surrounding a caving block is under stress, and from observations of timber failures it is evident that there is a small movement of solid ground toward the caving block. This small movement of the surrounding ground causes the halo of tension cracks to form at the surface. These tension cracks form blocks which fail by tipping toward the caving block. In some cases they may tilt away from the caving block, or individual pie-shaped wedges may drop into the tension cracks.

Figure 26 shows the stress conditions which are believed to exist in the ground around a caving block.

The rock failure observed outside the caving block is typical of failures in cohesive materials which are adjacent to steep slopes. At times during caving the entire side of a block, from the undercut to the surface, is subjected to stresses. The sub-

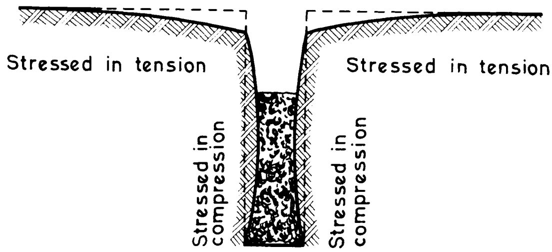

FIG. 26. Stress conditions in the ground surrounding a caving block.[11]

sided capping and the broken ore do support the walls; however, the fact that the angular measurements from the undercut to the most distant escarpments and the most distant tension fractures are fairly consistent indicates that these surface features are a function of the depth of mining rather than the depth of the unsupported sides of the caving block.

Summary[11]

(1) Observed data indicates that a block will usually cave vertically, or inside the undercut limits, unless there is some strong structural feature which tends to divert the movement of the rock. In the case of the Miami orebody ore tended to pull from outside the vertical block limits where a major fault, dipping at about 45°, cut through the block.

It was also noted, however, that when blocks bounded by broken ground were caved this surrounding material tended to be drawn with the caving ore. Thus in weak or fractured ground it appears that there is a tendency for the surrounding fractured rock to migrate into the caving block.

(2) Ground movement does occur outside the boundaries of the caving blocks in all cases and the magnitude of this movement depends upon the strength of the rock.

(3) The tension cracks surrounding a caved block do not represent planes of movement extending from the ground surface down to the undercut level. The ground enclosed by the halo of tension cracks is not necessarily a zone of highly fractured ground. This is illustrated by the following example cited by Fletcher[11] at the Miami Mine.

In 1956 an attempt was made to leach material outside of the caving limits by introducing leaching solution into the surface cracks along escarpments some distance north of four caving blocks. The solution worked its way under the surface end, entered the caved blocks high up on the column and leached copper from the northern portions of these blocks. These results demonstrated that: (a) the ground between the peripheral cracks and the lower portions of the caved blocks had not fractured sufficiently to allow solutions to penetrate it; (b) the peripheral cracks penetrated only to relatively shallow depths and had no direct connections with the undercut level or with the lower portions of the caving blocks.

Thus it must be remembered when lines are drawn to represent the angle of subsidence, the angle of break, or the angle of draw, that these lines *do not* represent planes of movement or planes bounding continuous zones of fractured rock.

Support For Development Openings Under Caving Blocks

At the Crestmore Mine no support was required for the grizzly drifts or for the haulage levels. In softer rocks a variety of types of support is used, depending upon the pressures encountered and the physical nature of the rock.

Timber was for many years the only support material used in grizzly and haulage levels. In most operations continuous repair and replacement of timber sets is required to keep these development openings passable.

Steel: In recent years steel sets have come into use for maintenance of development openings. Among the first types used were the cap-and-post timbering with steel H-beams for both posts and caps. Installed in haulage levels, they deformed under the weight of the ground but lasted much longer than equivalent timber sets and could remain in service much longer, even when badly deformed. Under conditions of moderate pressure they proved more economical than timber because of decreased maintenance costs.

Circular steel sets have proven economical at Miami Copper Co. They are particularly effective where the rock is finely broken and where pressure tends to be of the squeezing type, that is, relatively uniform on all sides. In this mine the ground pressure is too great to be successfully resisted permanently by the circular steel sets, but by proper spacing of the sets the lagging can be so proportioned that it will fail before the steel sets. Thus, the maintenance problem is reduced to that of replacing broken lagging rather than replacing entire sets.

Yielding steel sets have also proved successful in keeping open the development headings under caving blocks. Under excessive ground pressure these sets yield enough to allow a ring of compressed ground to form around the circumference. This ring then carries a large part of the ground load. As with the circular steel sets, the yielding steel sets are most suitable where the ground is finely broken and the pressure is uniformly distributed around the circumference.

Concrete support is used for slusher or grizzly drifts under conditions where a caving block contains relatively large rock fragments and rock blocks which create concentrated "point" loads on drift support. Such concentrated loads quickly destroy both timber and steel sets by causing them to buckle at the points where the loads bear on the supports. At the Jeffrey Mine slusher drifts are lined with as much as 3 ft of concrete, designed to support a static head of 500 ft of rock. These drifts are damaged by concentrated loads created at the abutments of natural arches which form when the block of coarsely broken asbestos ore subsides as ore is pulled from the slusher drifts.

Support Methods at San Manuel

At the San Manuel Mine in Pinal County, Arizona, blocks 600 ft high are being mined. On the second level the 600 ft high column of ore with a waste column about 670 ft high on top of it creates extreme ground pressures on the panel drifts.[9] The $8\frac{1}{2}$-ft inside diameter circular panel drifts on the grizzly level are lined with a minimum of 18 in. of concrete, and a regular sequence of repairs has been established as ground pressure develops.

The first repair step, when concrete cracks and breaks up, is to drill holes through the concrete and into the rock and install rock bolts with the objective of holding the coarse chunks of concrete in place.

If the weight increases, the second step is to install yielding steel rings inside the concrete lining. These rings are prestressed against the broken concrete with a 50-ton hydraulic jack to insure uniformity in ring loading. Ring spacing, determined by pressure, normally is about 24 in. in heavy ground. Cement grout is sometimes injected into the ground behind the concrete.

If the pressure continues to increase after yielding steel sets have been installed then the drift is gradually squeezed closed. The third step is to reopen the drift, install heavy yielding steel sets which are used as forms, and again place concrete lining.

Usually the stress pattern changes and after a period of waiting mining can be resumed without difficulty.

The ore-bearing quartz monzonite and quartz–monzonite porphyry rock in the areas of heavy ground pressure becomes almost plastic and literally disintegrates and flows as a compact cohesive mass when drawing is stopped in a block due to repair.

Unbroken boulders in the mass, sometimes with voids around portions of their surfaces, turn the ore into a breccia. These areas must be reopened by drilling, blasting and timbering, before the unconsolidated broken caved rock above will again flow through the finger raises. Re-drilling has been necessary to as high as 50 ft above the grizzly level in some cases.[9]

APPENDIX

THE ROCK MECHANICS OF BLOCK CAVING OPERATIONS[17]

The following treatise on "The Rock Mechanics of Block Caving Operations", was presented at the International Symposium on Mining Research which was held at the Missouri School of Mines and Metallurgy, University of Missouri, in February, 1961.

This paper has previously been published in Volume 2, *Mining Research*, which was edited by George B. Clark and published by Pergamon Press, 1962.

This section deals with the stresses which are involved in the fracturing of ore, and subsidence of blocks, in caving operations and with the pressures which develop on workings underlying a caving block.

Block caving involves the undercutting of large blocks of ore. Stresses induced in the ore after the undercutting cause it to crush and fracture as the broken ore is drawn off at the bottom. As the ore is drawn off the block settles and subsidence eventually extends to the surface.

The sizes of individual blocks which are caved vary with the physical characteristics of the ore and the dimensions of the orebody. Blocks in various mines have varied from about 37×75 ft in lateral dimensions up to about 250×250 ft in lateral dimensions. Heights of blocks are generally between 100 and 300 ft although blocks up to 500 ft high have been successfully caved in the asbestos mines of Quebec.

The pressure exerted by the subsiding ore on the supports of openings beneath caving blocks varies with the lateral dimensions as well as with the height of a block. A block should be small enought to avoid excessive crushing of supporting timber. On the other hand, a block must be large enough so that stresses created in the ore will be sufficient to fracture the ore.

Some porphyry copper ores will cave when as little as 700 ft^2 of a block is undercut. On the other hand, in a strong rock, such as limestone, undercuts about 80×200 ft have been required to initiate caving. Long and Obert cite an instance in which 35,000 ft^2 of a block was undercut without producing a cave.

Mechanics of Ore Fracture

Self-supporting Rock Structures

A block of ore which has been undercut has certain self-supporting tendencies. To obtain successful caving it is necessary to create stresses which are sufficiently large to overcome this self-supporting tendency of the ore block.

A strong, intact rock may support itself over an opening in the manner of a beam. In fractured rock, or rock which has little tensile strength, a beam cannot exist. Under such conditions the rock derives its strength (that is its tendency to support itself) from "arching effect". The simplest form of arch is the "linear arch" which is formed when a beam is cut in the center as shown in Fig. 27. This same beam may be cut transversely in a number of places and as long as the direction of the cuts does not approach the direction of the maximum shear stress the "linear arch" will not be appreciably weakened.

(Note: The term "arching effect" as used herein does not signify the formation of a vaulted or "peaked" roof or back. Rather, the term refers to the stress distribution within the rock. "Arching effect" signifies that the rock tends to support itself because the vertical forces due to the weight of the rock are resolved into diagonal thrust forces. These forces are carried by the rock at the sides of the undercut opening.)

A man-made masonry arch – the voussoir arch – has been used in civil engineering structures since early Roman times. Such a structure consists of a series of shaped rock, or concrete, blocks as shown in Fig. 28. Such a structure tends to form naturally in the rock when an opening is excavated in fractured rock.

In badly fractured rock the roof tends to stabilize itself in the form of a vaulted (arch shaped) roof. Rock tends to fall from the roof until all tensile stresses are

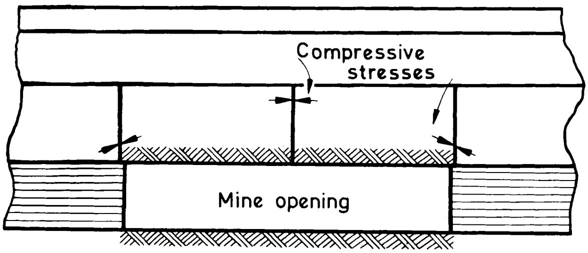

FIG. 27. A simple "linear arch" formed by fracture of the roof beam.

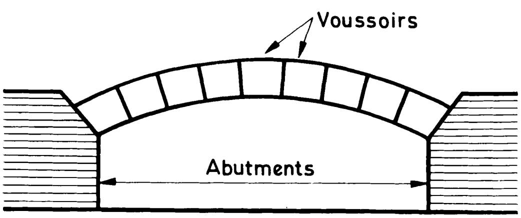

FIG. 28. A typical Voussoir arch.

eliminated and only compressive stresses exist along the arch line. Thus the rock tends to cave to form a natural voussoir arch which will be self supporting.

In caving operations it is necessary to create stresses which are sufficient to overcome the strength of such naturally formed arch structures.

Caving in Weak Rock

Since the back of an undercut is essentially a flat surface it resembles the bottom of a beam and initial failure of the undercut block occurs at this surface in a manner similar to failure of a beam. Stresses are the greatest at the center of the back and most ore fragments fall from this location. The vaulted opening so formed works its way up until it reaches the natural arch line of the ore. Along this line stresses are compressive and parallel to the arched rock surface. This caving process is illustrated in Fig. 29. The back has arched up until stresses parallel to the back are compressive. Under such conditions shear failure of the rock under compressive stress becomes important in causing breakdown of the structure. Shear stresses are maximum at the ends, or abutments, of the arched back.

It is not necessary that any visible opening exist between the pile of broken ore and the ore which is supporting itself in place. In weak rock, or in a granular material, there may be no visible plane of discontinuity between the de-coupled ore and

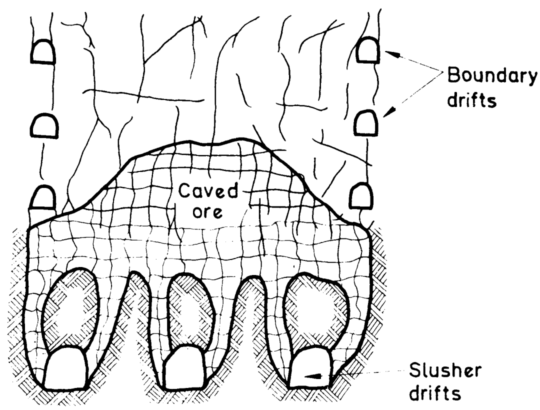

FIG. 29. Block caving action in weak ground.

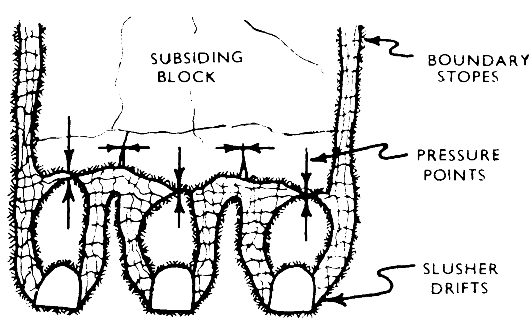

FIG. 30. Fracturing produced by block subsidence in strong rock.

that which still appears to be in place. A partial removal of the supporting pressure of de-coupled ore may be brought about by subsidence of the de-coupled ore. Even a slight subsidence may be sufficient to cause the disaggregation of more of the overlying ore while all ore fragments are still in contact with each other.

Caving in Strong Rock

When the ore to be caved is strong in tension then the cave line does not form a vaulted arch but the ore subsides instead as an integral block and fracturing takes place under compressive stress in the lower portions. The mode of fracture is illustrated in Fig. 30. As the block settles after undercutting is completed it contacts the pillars at random points and the concentration of compressive stresses at these points cause the rock to fracture by shearing and crushing. Other points of high compressive stress occur where individual large ore blocks contact each other in forming crude linear arches. Each of these contact points is eroded away by progressive shear fracturing until other random support points are established. These in turn are reduced by shear induced by excessive compression with the process repeating itself as long as the resultant broken material is taken from the appropriate drawpoints. If the broken material is not drawn off as it forms it may be so compacted by the weight of the overlying block that it will be impossible to draw.

Example — Caving in Strong Rock vs. Caving in Weak Rock

Block caving operations are usually confined to massive deposits of structurally weak ores. An exception to this was the block caving operation in limestone at the Crestmore Mine of the Riverside Portland Cement Co., at Riverside, Calif.[13] A limestone bed with a thickness of about 300 ft was mined. The rock had a compressive strength of about 7000–11000 psi. In order to secure subsidence of a block the following measures were used.

(1) The blocks were completely cut off on all sides by either vertical shrinkage stopes or caved areas. In block caving in weak rocks only portions of block boundaries are cut by boundary drifts.

(2) The area of the undercut required to start a block caving was much greater than that in weaker ores. An undercut area of about 16,000 ft^2 was generally required to start caving and as much as 35,000 ft^2 were undercut in one instance without causing a cave. In contrast to this in some porphyry copper ores caving can be induced by undercutting an area as small as 625 ft^2.

(3) In normal block caving the most widely accepted practice is to pull equal tonnage from each drawpoint according to a pre-determined daily schedule. At Crestmore rock could only be pulled from those drawpoints which were situated under those points which were highly stressed enough to fracture the rock. Such points were indicated by the relatively copious amount of powdered rock in the drawpoint chutes. Pulls were made and continued only from drawpoints where the powdered rock was evident until a substantial void, perhaps 75 ft in diameter, had been created. The weight of the block was thus transmitted to other localized areas around the perimeter of the void and pulling was shifted to drawpoints where the presence of powdered rock in the chutes indicated a high stress area. This arching to form a void was in contrast with the usual action in porphyry copper ores where sustained draw from one set of drawpoints usually results in funneling, or piping, up to the surface. Distribution of maximum pressure points could not be predicted in advance and such points could only be located by searching to find the locations where rock powder had accumulated in the chutes.

Stresses in Broken Rock at Drawpoints

As the broken ore converges on a drawhole transverse stresses are brought into play. These are due to "arching effect" as the fragments crowd together in entering the drawhole. As shown in Fig. 31 a few large pieces can form an arch across the entrance to obstruct the drawhole. Similar stresses exist in the fragmented ore even

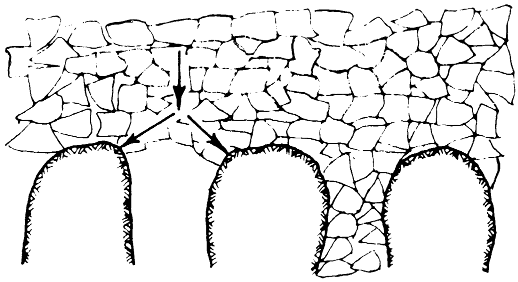

FIG. 31. Drawpoint stoppage by arching.

when it is flowing freely through the raise. However, the raise will not clog because a stable arch cannot form as long as the shear stresses in the broken ore exceed the shear strength of the mass.

Whether or not broken ore will form an arch to obstruct a drawhole depends upon the relative size of the fragments, the coefficient of friction between fragments, and the vertical pressure to which the fragmented ore is subjected.

Pressures on Support Due to "Arching Effect"

Severe local pressures on drift support may develop where the ore breaks into relatively large fragments, or where it contains a percentage of large strong inclusions. Pulling at one drawpoint causes arches to form which have abutments on support for other surrounding drawpoints. Even heavy concrete lining for slusher drifts may be broken by the intense point pressures developed by lateral thrust due to such arching. Figure 32 illustrates this type of pressure development.

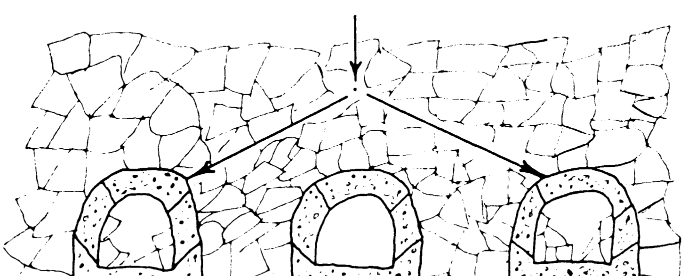

FIG. 32. Destructive localized pressures due to arching effect in coarsely broken rock.

Boundaries of Caving Blocks

Each block is ordinarily separated from surrounding ground by planes of weakness created by boundary shrinkage stopes or by driving a series of drifts, one above the other, along the selected boundary. These boundaries ordinarily suffice to provide a barrier along which broken ore subsides and which served to prevent most exterior rock from mixing with the ore in the block.

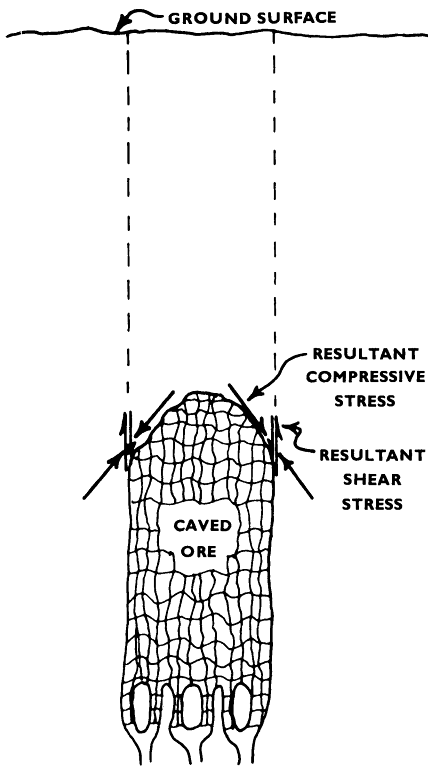

FIG. 33. Arching action in a narrow block in fractured rock. The boundaries are steeply inclined, or vertical.

Even when caving of a block progresses upward into the overlying capping the limiting planes of subsidence usually remain essentially vertical if the cross-sectional area of the block is relatively small as compared with its depth beneath the ground surface.

Figure 33 shows the theoretical stress conditions at the boundaries of a caving block. In this case the lateral dimensions of the block are small as compared to its depth. The narrower the width of a block relative to its depth the more does the subsiding ore tend to arch across. This "arching effect" results in diagonal thrust stresses against the adjacent rock. The resultant principal compressive stress is inclined rather than vertical, and the resultant fracture occurs along shear planes inclined at $45° - \phi/2$ to this principal compressive stress. The resultant planes of movement will be steeply inclined, or vertical, as shown in Fig. 33. (ϕ = the angle of internal friction of the ore.)

Examples of "Chimney Caving"

Figure 34 is taken from a curve plotted by Marr[14] which shows the "angle of draw" observed over mined-out areas in British coal mines. (Note: Marr measured the angle of draw from a vertical line drawn through the edge of the mined-out area rather than from a horizontal plane.) Marr observed that the angle of draw decreased as the ratio of the width of the mined-out area to its depth increased.

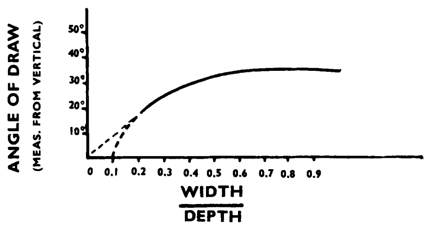

FIG. 34. Angle of draw vs. the width/depth ratio.[14]

That is, the boundaries of the subsiding block approach the vertical as the width/depth ratio of the mined-out area decreases. Such subsidence progressing upward between nearly vertical planes has been termed "chimney caving" or "piping".

An example of chimney caving has been cited by Rice.[15] In this instance an opening 14 ft wide and 28 ft long was made in Hanbury slates in order to obtain filling material by caving. The caving proceeded, filling material being drawn off, until after about a year's time the caving had worked through to the surface in the form of a vertical chimney similar in cross-section to the opening 900 ft below. The dip of the slates was about 60° from the horizontal but the caving cut diagonally across the bedding.

Still another example of chimney caving is shown in Fig. 35. In this case the subsidence of the block (along nearly vertical planes) of jasper capping nearly 1900 ft thick resulted from the extraction of a block of ore by top slicing.

Arching Effect in Horizontal Planes

When a block of ground is of limited length, as well as of limited width, the effect of "arching effect" or "ring action" in a horizontal plane must be considered. This tends to prevent movement of ground toward an excavation and is probably also a factor in causing subsidence to follow nearly vertical planes. The ground surrounding the subsiding block may be considered as being a horizontal ring in compression.

Landslide-type Subsidence

Once vertical subsidence has broken through to the surface vertical banks may form temporarily. When these become high enough a "landslide-type" of slope slump occurs. Non-cohesive material cannot form vertical banks but if the material

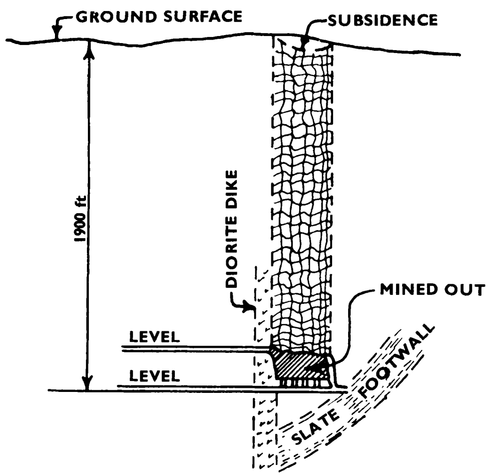

FIG. 35. An example of "chimney caving" at the Athens Mine, Negaunee, Mich.[15]

possesses some cohesion then high vertical banks may form before the weight of the material causes a landslide to develop. The progression of subsidence to the surface and the slumping toward the surface depression are shown in Fig. 36.

Pressures at the Bottom of a Caving Block

As shown in Fig. 37, the subsidence of an isolated block of fractured ore may be likened to the drawing off of a granular material from a bin. The boundaries of the block represent the sides of the bin and the drawholes would correspond to bin hoppers. It is of interest to calculate average pressures at the bottom of such a block by means of Janssen's formula. This was derived for the purpose of calculating pressures of granular materials on the bottoms of bins. This formula has been modified by Terzaghi for application in the field of soil mechanics.

The example which follows Fig. 37 assumes that the material being caved is a fragmented material and that an equal amount of ore is drawn from each point so that uniform subsidence takes place. In the example the average unit vertical pressure at the bottom of the ore is computed to be equivalent to the weight of a depth of 137 ft of rock, or to about 68 per cent of the actual rock depth. The remainder of the weight of the rock block is transferred by "arching effect" to the rock at the boundaries of the caving block.

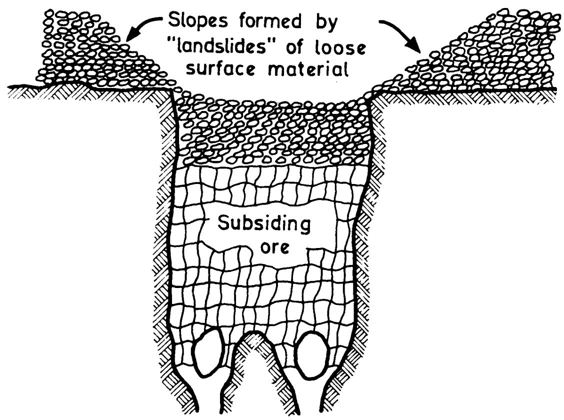

FIG. 36. Subsidence of surface over a caving block.

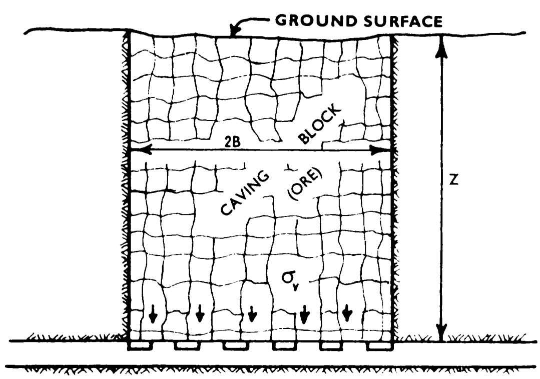

FIG. 37. Pressure at the bottom of a caving block.

Example: Calculation of theoretical average pressure, σ_v, at the bottom of a caving block.

$$\sigma_v = \frac{\gamma R}{i \tan \theta}\left[1 - \exp\left(-i \tan \theta \frac{Z}{R}\right)\right] \qquad \text{(Janssen's formula)}$$

where:

σ_v = Average unit pressure at bottom of caving block.

γ = Density of ore in pcf.

$$i = \frac{1 - \sin \phi}{1 + \sin \phi}$$

ϕ = Angle of internal friction of the fractured ore.

Z = Depth from surface to undercut level.

R = "Hydraulic radius" of the caving block

$$\text{that is: } R = \frac{\text{(horizontal cross-sectional area of block)}}{\text{(perimeter of block)}}$$

$\tan\theta$ = Coefficient of friction between the caving block and adjoining walls.

Assuming a caving block 150 ft square and 200 ft high and taking $\gamma = 170$ pcf, $\phi = 45°$; and $\tan\theta = 0.90$ then:

$$R = \frac{150 \times 150}{4 \times 150} = 37.5$$

$$\sigma_v = \frac{(170)(37.5)}{(0.17)(0.90)} \left[1 - \exp\left(-(0.17)(0.90)\frac{200}{37.5}\right)\right]$$

$= (41{,}600)\ [1 - e^{-0.82}]$
$= (41{,}600)\ [1 - 0.44]$
$= 23{,}300$ psf which is the average vertical pressure for the assumed conditions.

This unit pressure is equivalent to a column of rock of $23{,}300 \div 170 = 137$ ft depth and the pressure is equal to $137 \div 200 = 68$ per cent of the original depth pressure.

A great deal of uncertainty exists as to the actual values which should be assigned to the quantity ($i \tan\theta$) in Janssen's formula. Lucas and Verner obtained experimental values for ($i \tan\theta$) of about 0.18. They used rounded flintshot silica sand and a steel bin for their experiments.[16]

Terzaghi *(Theoretical Soil Mechanics*, John Wiley, New York, 1956, pp. 73, 196) substitutes for the theoretical value i an empirical constant K. According to Terzaghi, experimental investigations in sand have shown that this coefficient K should be assigned a value of about 1 when Janssen's formula is used for computing pressures on tunnel supports.

If we substitute K for i in Janssen's formula we have:

$$\sigma_v = \frac{\gamma R}{K \tan\theta}\left[1 - \exp\left(-K \tan\theta \frac{Z}{R}\right)\right]$$

If we retain all other conditions of the previous problem but assume that $K = 1$ then we have $K \tan\theta = (1)(0.9) = 0.9$

and
$$\sigma_v = \frac{(170)(37.5)}{(1)(0.9)}\left[1 - \exp\left(-(1)(0.9)\frac{200}{37.5}\right)\right]$$

$= 7100\ [1 - e^{-4.8}]$

$= 7100\ [1 - 0.008]$

$= 7000$ psf for the assumed conditions.

Pressure Variations

The foregoing computations do not give any information as to the actual variation of pressure across the bottom of the block. Rather the computed result is an average pressure across the entire area of the block bottom.

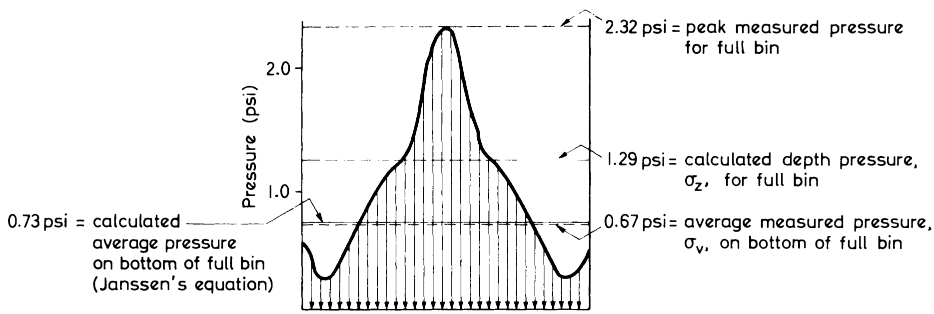

FIG. 38. Vertical pressure distribution on the bottom of a full non-flowing bin.[16]

Figure 38 shows the pressure distribution across the bottom of an experimental model circular bin 12 in. in diameter and 24 in. deep filled with dry rounded flintshot silica sand.[16] The average measured vertical pressure on the bottom of the full bin was about 0.67 psi or equivalent to 52 per cent of the total weight of the sand. The remaining 48 per cent of the weight of the sand was transferred to the walls of the bin by friction and "arching effect". Maximum vertical pressure occurred at the center of the bin. A similar pattern of pressure distribution might be expected at the bottom of a caving block of similar dimensions. Values of σ_v as measured by Lucas and Verner[16] for various depths of sand corresponded closely to values computed by means of Janssen's formula.

Some important conclusions which may be drawn from these experiments are:

1. The maximum vertical pressure occurred at a center "pressure abutment", with lesser pressure peaks at the walls.

2. A semblance of a center area of overpressure (pressure abutment) appeared when the depth of the sand reached a Z/R (depth/hydraulic radius) ratio of about 0.6 and pressure abutments became well defined for Z/R ratios of 3.0 or more. This Z/R ratio would correspond to a depth of rock of about (3) (37.5) $=$ 113 ft in the caving block of Fig. 37.

3. For various depths of sand the measured values of the peak pressure in the center abutment ranged from about 1.8 σ_z to about 2.5 σ_z where $\sigma_z = W/A$ (weight of sand divided by area of bin bottom).

Peak pressures also ranged from about 2.5 σ_v to 3.5 σ_v where σ_v represents the average measured pressure on the bin bottom for a certain depth of sand.

4. Measured peak abutment pressures at the walls were about 1.5 σ_v for Z/R ratios less than 5.0. These decreased to values less than 1.0 σ_v for Z/R ratios greater than 7.

The foregoing measurements are significant in indicating that localized unit pressure at the center of a caving block may be as much as twice the depth pressure, σ_z (depth pressure is the unit pressure obtained by dividing the total weight of a rock "block" by its bottom area).

Pressure abutments also occur in underground mining other than block caving operations. These pressure abutments are discussed in Volume I, Chapter 6, page 257.

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CHAPTER 5

MINING BEDDED METALLIFEROUS ORES

BEDDED deposits of the metalliferous ores rarely have the uniformity in thickness or the regularity of roof and floor (hanging-wall and foot-wall) which is found in coal beds. In addition the dips of such metalliferous deposits generally vary from point to point and flat-lying bedded deposits are the exception.

As with coal beds such deposits may be mined by room-and-pillar methods, or by long-wall systems although some of the shallow deposits are mined by open stoping methods with support by random pillars.

Because the minerals which compose ores are normally much harder than coal most ores must be broken by drilling and blasting and there is no opportunity to use continuous-mining machines such as those which are employed in coal mining.

Coal-mining types of loaders, such as gathering-arm loaders, and shuttle cars are used for ore transportation in flat lying veins. Because ores are much heavier and more abrasive than coal, such machines must be "beefed up" for metal mining service.

OPEN STOPING

This may be considered as a modification of the room-and-pillar system with pillars being left in place permanently. It is used most often in small, irregular deposits containing many lean or waste areas. The workings follow ore and the barren areas are left as pillars for roof support. If there are not enough waste areas available for adequate roof support then pillars may be left temporarily while the workings are advanced to the limit of the ore and the ore pillars may be extracted on retreat with the roof being allowed to cave behind the retreating workings.

Example of Open Stoping in Bedded Uranium Deposits

Open stoping with occasional timber support, or with roof bolts, is used extensively in the small and irregular bedded uranium deposits of the Colorado Plateau.

Figure 2 shows the outline of the stoped areas in the Continental No. 1 Mine. The mineable ore averaged about 7 ft thick and occurred at the base of the Moss Back sandstone.

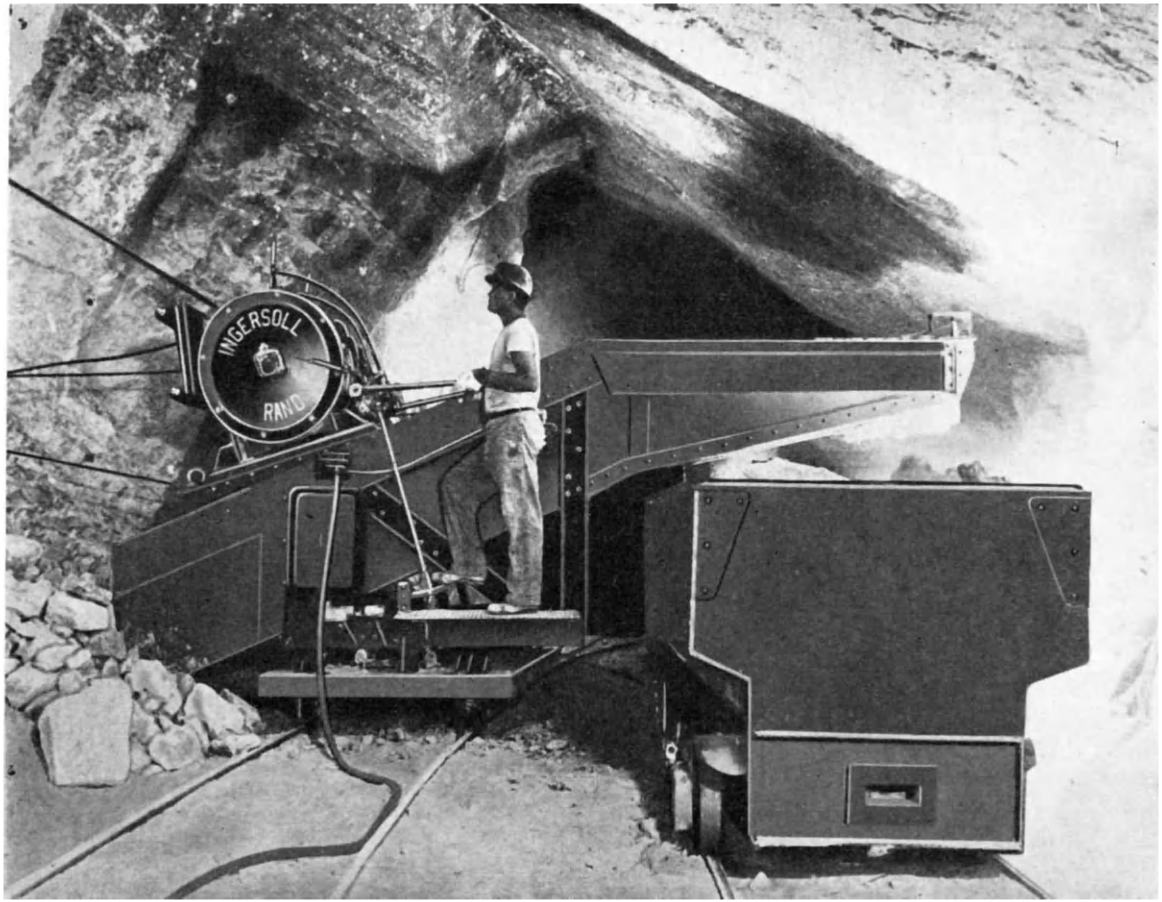


FIG. 1. Scraping ore from a flat stope. (Ingersoll-Rand Co.)

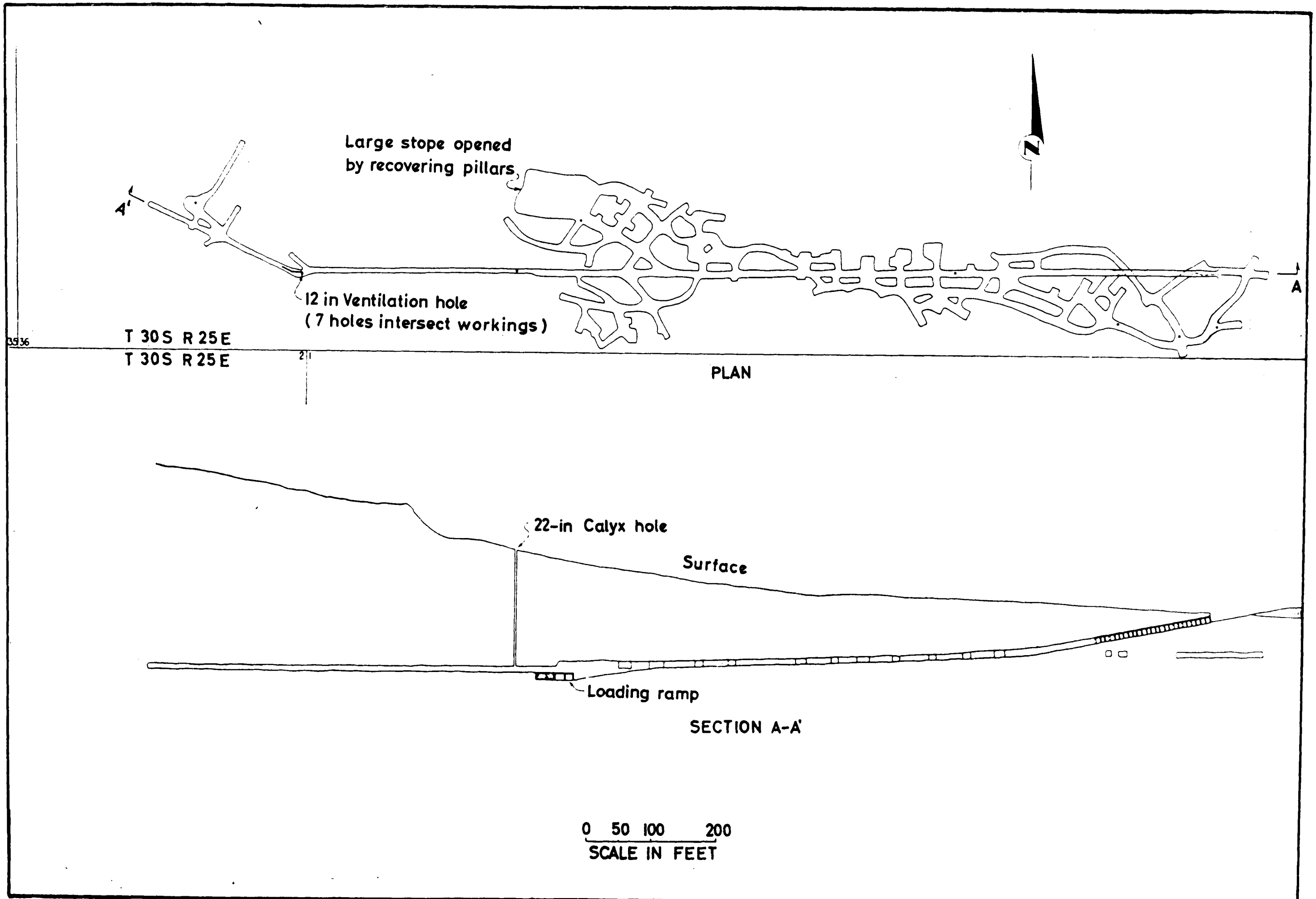


FIG. 2. Plan map and setcion of underground workings, Continental No. 1 Uranium Mine.[1]

The workings followed the irregular outline of the payable ore and the ore was mined by horizontal open stoping with random pillar support.

Drilling was done by means of light drills, such as air-leg stopers and some of the ore was slushed to loading points to be picked up by an overshot loader while the remainder was loaded by means of a slushing ramp onto cars or buggies.

Transportation in such mines is commonly done by means of modified "concrete buggies", shuttle cars, or trucks (where headroom is sufficient).

Example of Open Stoping in a Bedded Zinc Deposit

At Mascot, Tenn., zinc ore averaging 2.9 per cent zinc (1929) was mined by a mill-hole version of the heading and bench system. The method is illustrated in Fig. 3.

The ore body was wide and long, dipped at about 20°, and ranged from a few feet thick up to 150 ft thick. The ore occurred in a dolomitic limestone and the mineralized ground would not stand well without support whereas the unmineralized rock composing the roof and walls of the stopes would often stand indefinitely over spans of 100 ft or more.

The mine was developed by a vertical shaft 612 ft deep and stopes were mined both above and below the shaft level.

Pillars were roughly circular in cross-section and proportioned to the character of the ground and the height of the ore. Ordinarily they were 25–30 ft in diameter and the stopes between pillars were 40–60 ft wide.

The method as illustrated in Fig. 3[(16)] is particularly adapted to a large rather thick deposit in which the ore may not stand well but which has a very firm and strong roof.

ROOM-AND-PILLAR MINING

Room-and-pillar mining methods are used in mining the bedded iron ores of the Birmingham District in Alabama, and in mining the bedded iron ores of the Lorraine District in France.

Room-and-Pillar Methods in French Iron Mines

Figure 4 ((a), (b), and (c)) illustrates the methods used in mining flat lying bedded iron ores in the Lorraine District of France.

Three-entry main headings are driven and pairs of secondary headings are turned off from these. From the secondary headings single tertiary headings are turned off. The effect is to divide the bed into blocks about 300 ft square. A series of parallel rooms is driven through each block and pillars are recovered on retreat, allowing the roof to cave behind the retreating pillar line.

On the average rooms are 260–330 ft long and 16 ft wide while the long pillars are 40–65 ft wide.

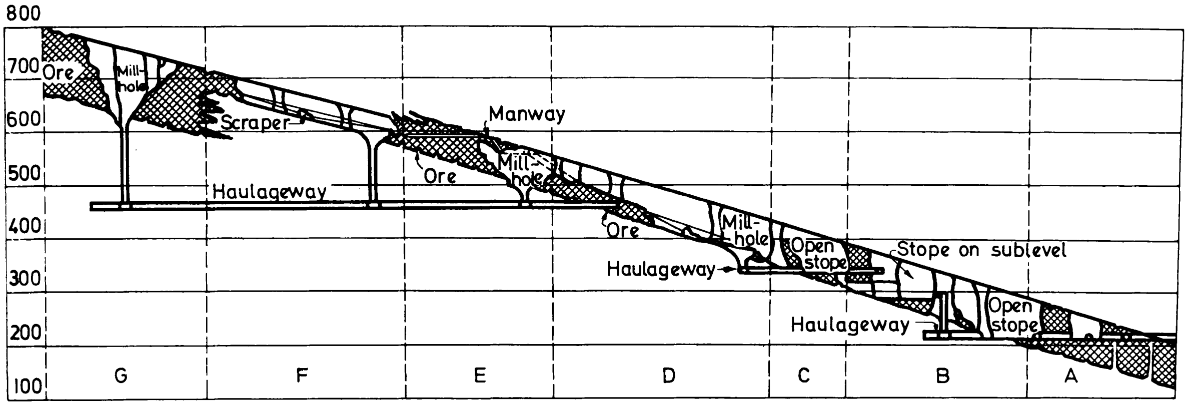
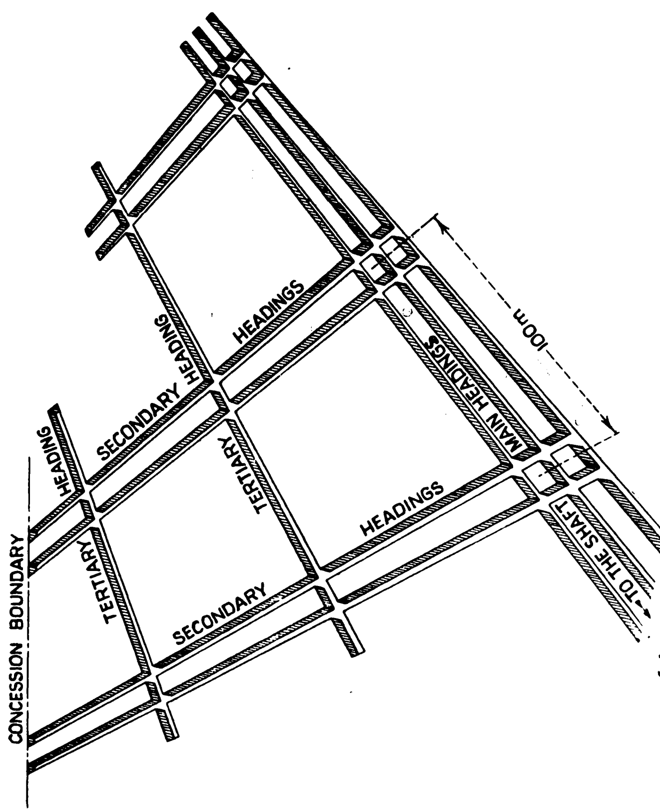


FIG. 3. Steps in development of mining processes, bedded zinc deposit, Mascot, Tenn.(16)


CONCESSION BOUNDARY
HEADING
TERTIARY
SECONDARY
SECONDARY HEADING
TERTIARY
SECONDARY
HEADINGS
HEADINGS
MAIN HEADINGS
100m
TO THE SHAFT

(a)

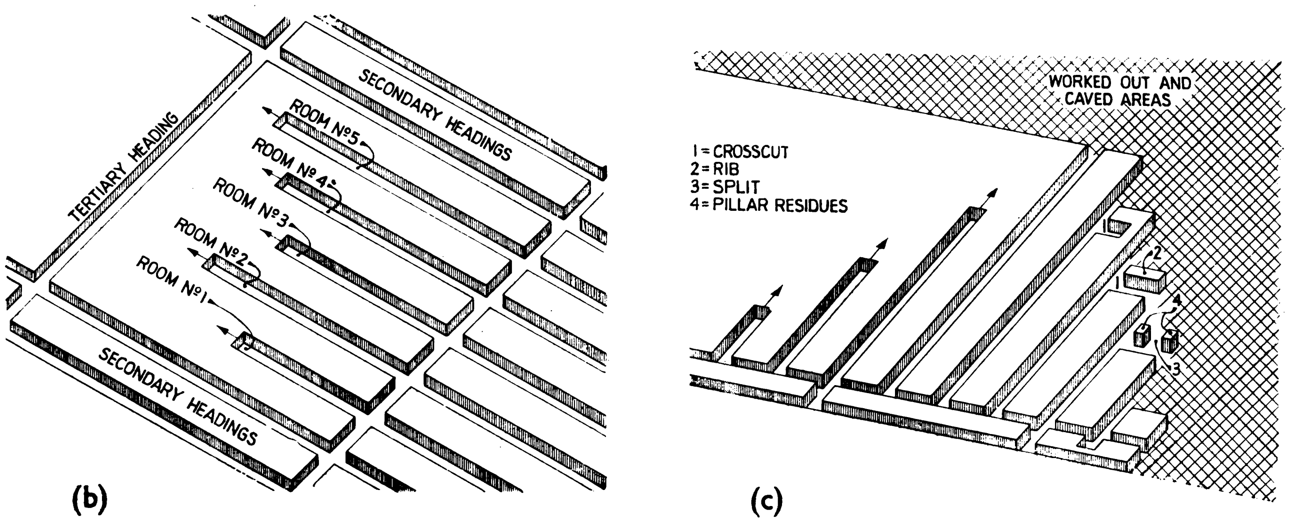


FIG. 4. Three stages in working by the room-and-pillar method used in the Lorraine iron ore mines.[2, 3]

(a) Blocking out the deposit.
(b) Driving the rooms.
(c) Pillar extraction.

This system provides for an average recovery of 85 per cent in seams more than 16 ft thick, 90 per cent in seams 11–16 ft thick, and 95 per cent or more in thin seams.

The number of slices worked simultaneously during the extraction of a pillar varies with individual conditions and the length of the pillar-extraction line is, on occasion, as much as 1300 ft.

The roof and floor rocks are principally shales and sandstones which are somewhat calcareous. These rocks are fairly strong and stand well when bolted. More than 17,000 roof bolts are used monthly in these mines. Roof conditions vary greatly from place to place but on the average one roof bolt is used for every 11 ft^2 of exposed roof.

The iron ore has a hardness about the same as that of limestone and can be drilled with rotary drills.

A large proportion of the loading and transportation equipment is of coal mining types with gathering arm loaders and shuttle cars predominating.

TABLE 1. CHARACTERISTICS OF A TYPICAL LORRAINE IRON ORE MINE[2, 3]

	Average	Maximum	Minimum
Overburden (m)	150	230	0
Dip (deg.)	3	5	0
Thickness of worked seam (m)	4	9	1.80
Mechanical strength of the seam rock (kg/cm^2)	200	600	60
Mechanical strength of the strata in the immediate roof (kg/cm^2)	180	850	50

Room-and-Pillar Methods in Alabama Iron Ore Mines

The bedded iron ores of the Birmingham District are mined under about 1600 ft of cover and the ore generally ranges from 6 to 14 ft in thickness. Entries and rooms are generally driven in widths from 20 to 32 ft and the roof is bolted.

The slabbing method of pillar recovery is usually followed, with single 6–7 ft slabs taken from each side of the pillars left standing.

Figures 5 and 6 show methods of roof bolting used in the iron ore mines.

LONG-WALL MINING METHODS FOR BEDDED METALLIFEROUS DEPOSITS

Long-wall methods may be applied in mining bedded ore deposits when the deposit has sufficient regularity of roof and floor to make it feasible. Such methods have been applied in mining uranium ores in the Colorado Plateau; in mining copper ores from the conglomerate beds of Michigan; and in mining gold from the deep "reefs" of the Witwatersrand District of South Africa.

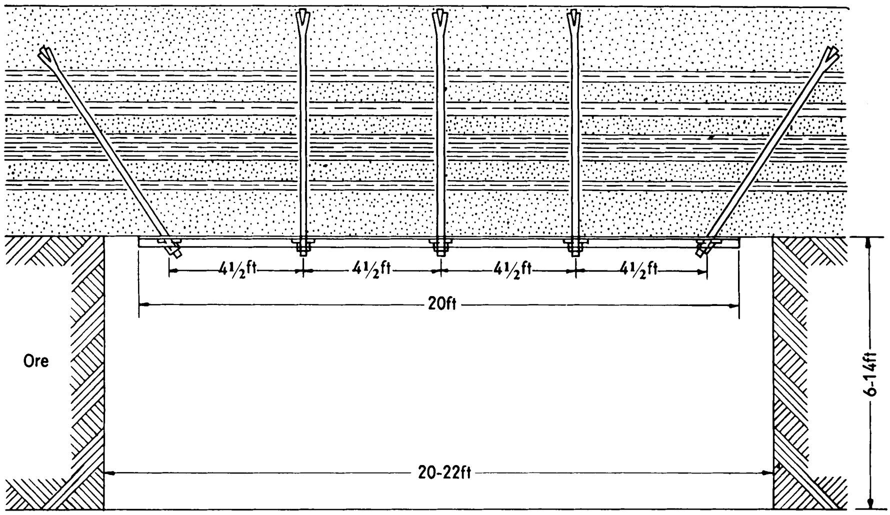


FIG. 5. A combination of vertical and angle bolting in the iron ore mines, Alabama.[4]

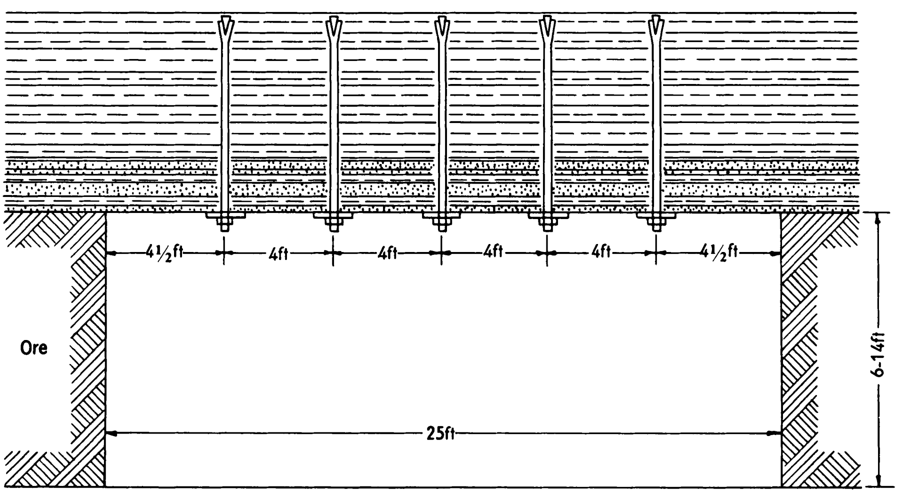


FIG. 6. Vertical bolting in the iron ore mines, Alabama.[4]

Long-wall Mining of Uranium Ores

Long-wall methods were used in extracting the Radon orebody which had an average thickness of $4\frac{1}{2}$–5 ft and was situated at a depth of about 550–750 ft beneath the surface and had a dip of about 7°. This orebody was 2150 ft long and from 400 to 700 ft wide.[7]

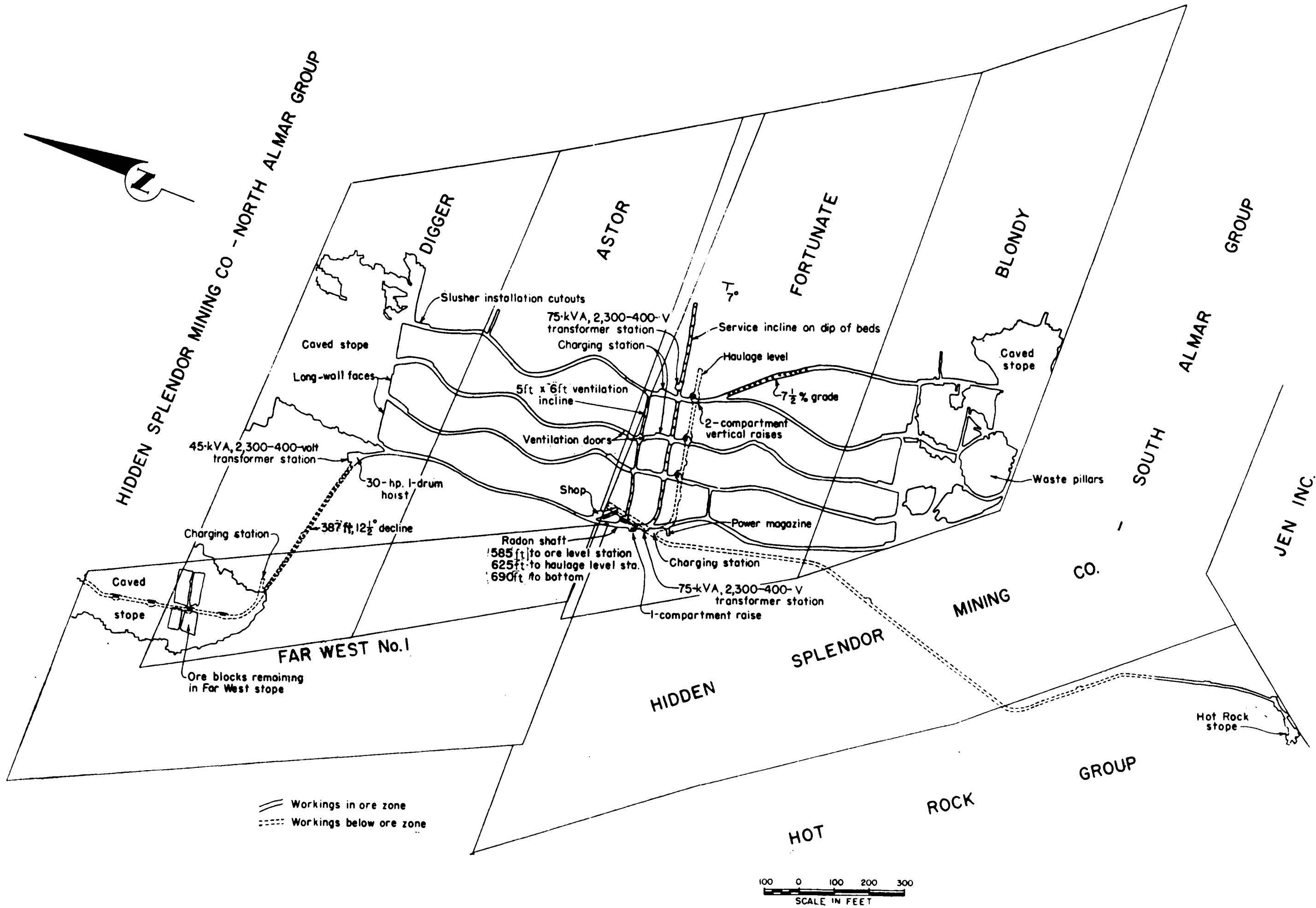


FIG. 7. Plan of workings, Radon Uranium Mine.(7)

The room-and-pillar system was first considered for mining this orebody but was rejected because of the weak roof rock and the uncertainty of full pillar recovery. A primary consideration in selecting the long-wall system was the relatively high per-ton value (about $65) of the ore which made it desirable to extract all ore, leaving none in pillars.

Drifts were driven to the ends of the orebody and ore was extracted by retreating long-walls.

The roof was supported by two rows of yielding steel props. Props were set $3\frac{1}{2}$ ft apart while the rows were 4 ft apart. This gave an average of one prop for each 14 ft^2 of roof. In addition removable timber cribs were built between the two rows of props to furnish additional support at the line where it was desired that the roof should break.

The mining cycle was:

(1) Drill.
(2) Lace.
(3) Load and blast.
(4) Slush, load, and tram.
(5) Move lacing.
(6) Clean up.
(7) Pull and re-set props and cribs.

Drilling

Short, telescopic air feed legs were used because the cave line was so close to the face line. The long-wall face was advanced 4 ft with each round.

Lacing

Douglas Fir 4×8 in. timber was fastened to the props with steel straps so that it formed a continuous wall which prevented blasted ore from being thrown into the goaf.

Loading and Blasting

About forty holes along the face line were loaded and blasted at a time.

Slushing, Loading and Tramming

Ore was scraped along the face to a loading point by a light-weight slusher–scraper traveling about 300 ft/min. Ore was loaded into 40 ft^3 cars by overshot mucking machines.

Moving Lacing

After all ore was slushed from the face the lacing was cut loose with an axe and dropped into piles. If there was not room along the long-wall the lacing was dragged to the nearest drift with the slusher.

Clean up

The stope was cleaned by hand shoveling before props were moved.

Pulling and Setting Props and Cribs

Several props were set in place close to the face, 4 ft ahead of the front row, before props in the back row were pulled. Two men worked at pulling and setting props.

Caving usually followed the removal of each prop and was generally complete by the time that all props in the line had been moved.

Long-wall Mining of Bedded Copper Ores

Figure 8 is an isometric sketch showing the configuration of the lode in the Conglomerate Mine of the Calumet and Hecla Consolidated Copper Co., in Michigan.

Copper occurred in the conglomerate bed which varied from 10 to 20 ft in thickness and which had a dip of from 35 to 40°. The hanging-wall was a hard, strong, trap rock and the foot-wall was an amygdaloid which was somewhat softer.

Mining was by a system of stull supported stopes down to a vertical depth of about 3500 ft. Below this depth the pressure of the overburden began to cause rock bursts and foot-wall heave and the mining system was changed to a retreating long-wall method using stulls for support and allowing the hanging-wall to cave behind the retreating pillar line. This system was used down to the bottom levels at a vertical depth of about 4900 ft.

Figure 9 illustrates the placement of the support timbers in the stopes and Fig. 10 shows the general configuration of the retreat line of several stopes.

The lower 40 ft of each stope was timbered with pairs of stulls each $1\frac{1}{2}$–2 ft in diameter, placed on 10-ft centers. Above this height single stulls were placed on about $10\frac{1}{2}$-ft centers.

Experience showed that during the time required to complete a stope (approximately 100 days) the maximum thrust of the hanging-wall manifested itself at a point about one-third the total height of the completed stope, or just below the fourth double set above the level.

It is a matter of considerable interest that open stopes of the dimensions shown could be successfully used at depths of from 3500 to 4900 ft below the surface.

An attempt to excavate openings of these dimensions in coal-measure strata would quickly result in caving of the roof and loss of the opening. This is a good illustration of the effect which hard, strong roof beds can have on the permissible size of mine openings at depth.

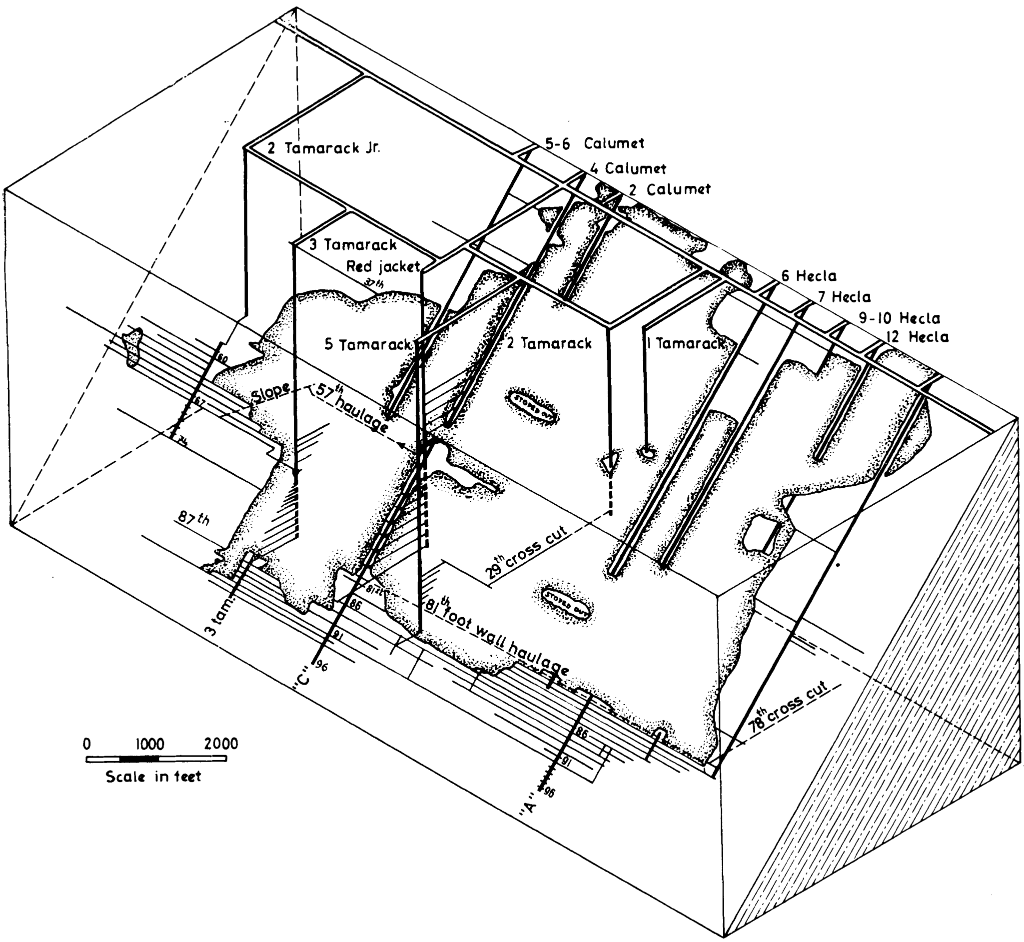


FIG. 8. Isometric sketch of Conglomerate Lode, Michigan.(8)

At these operating depths pillar spalling, foot-wall heave, and rock bursts were troublesome, but after the adoption of the retreating long-wall system rock bursts were mainly confined to the shaft-pillar recovery operations.

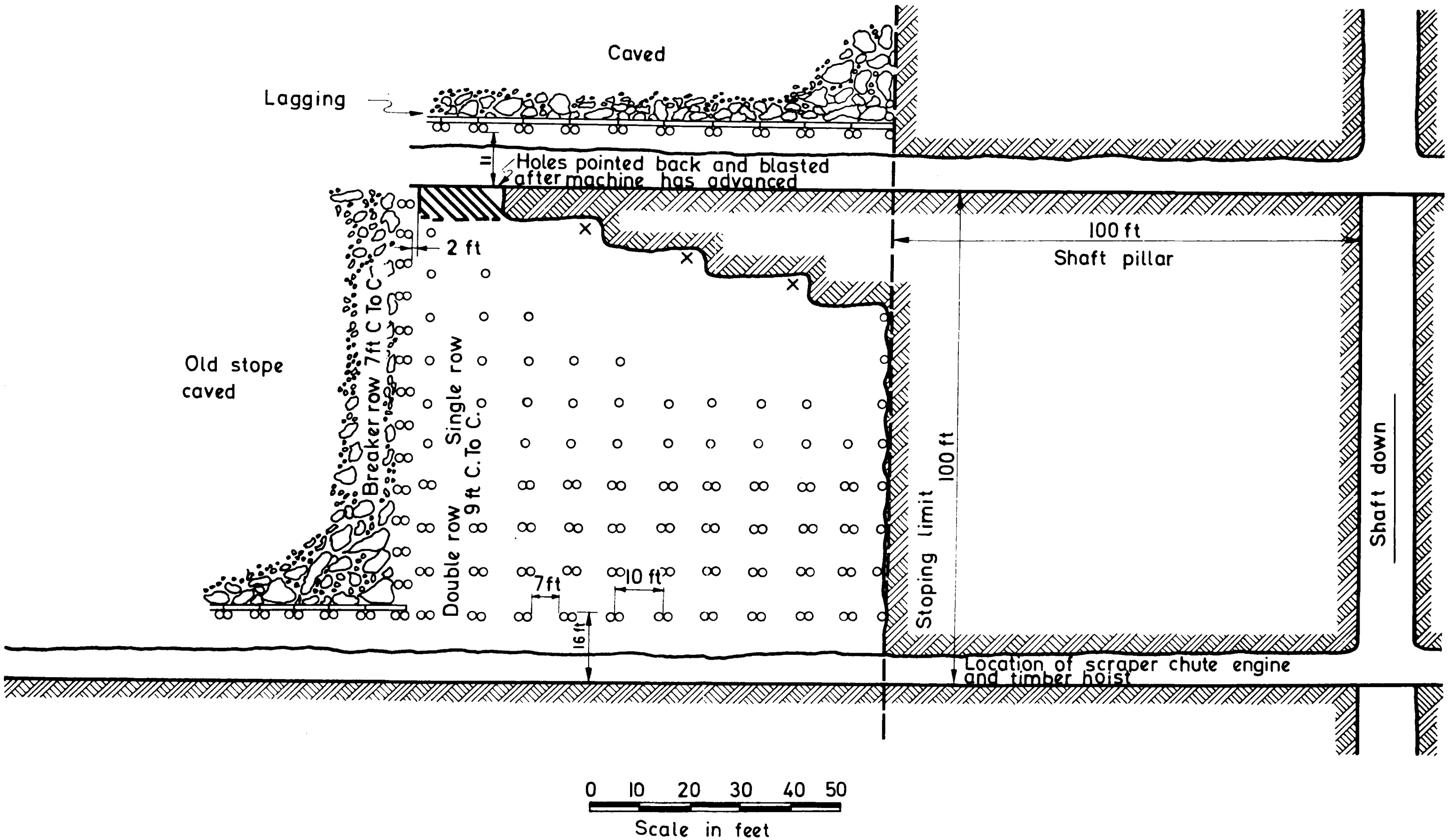


FIG. 9. Open stope, Conglomerate Lode, Michigan.(8)

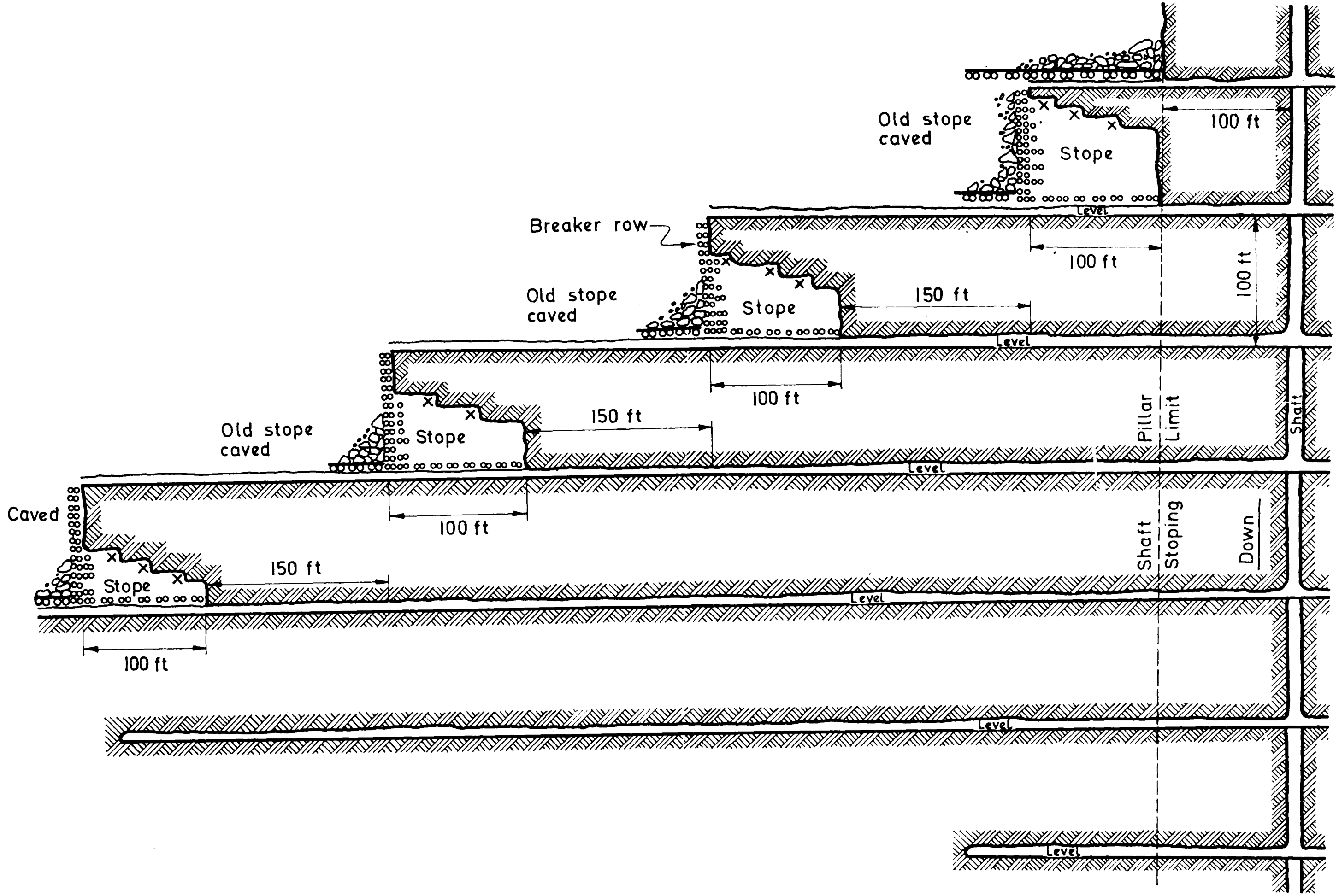


FIG. 10. Retreating stoping system, Conglomerate Lode, Michigan.[8]

Long-wall Mining in The Witwatersrand Gold Mines

The orebodies of the Witwatersrand consist of conglomerate beds of quartz and pebbles cemented together with predominantly siliceous material. The pay streak varies in thickness from a pencil-line contact to 10 ft, or more. The greater part of the production is taken from a "channel" less than 4 ft thick. The "reefs" dip at angles which vary from 0 to 40°.

Both the hanging-walls and the foot-walls of the reefs generally consist of hard brittle quartzites. Figure 11 is a typical cross-section showing some of the reefs which are mined.

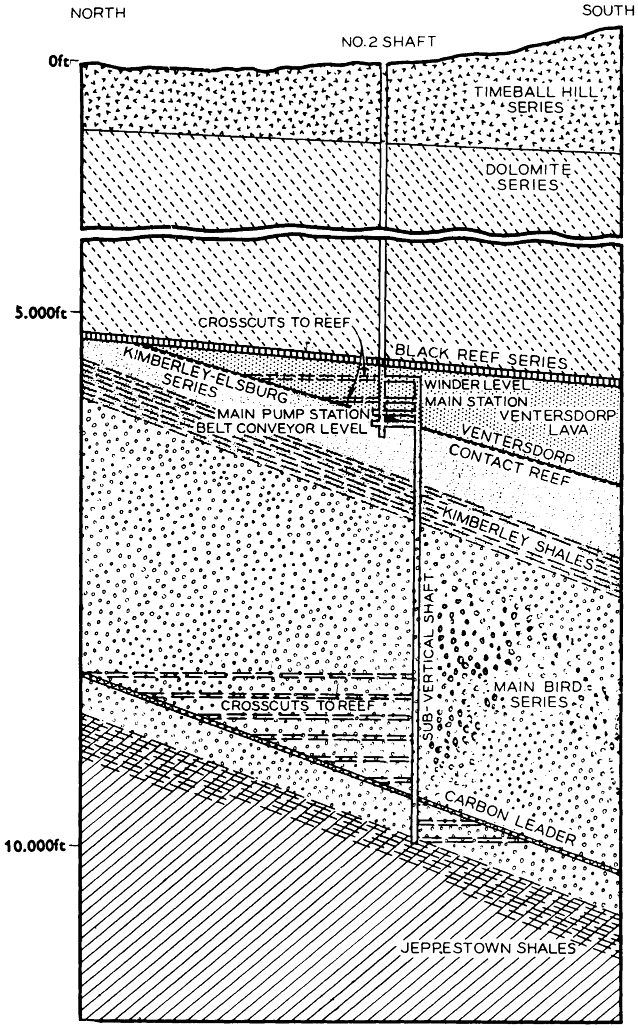


FIG. 11. Section showing the geological structure in the vicinity of No. 2 Shaft system at Western Deep Levels, looking east. (Witwatersrand).[11]

Mining has extended to a maximum depth of 11,500 ft and many mines are operating at depths between 5000 and 9000 ft. Long-wall systems are used because of the extreme depth pressures on the reefs.

FIG. 12. East Rand Proprietary Mines, Ltd., plan showing localities of rock bursts ① and ② in stopes 49/50 E6E of the Angelo and 50W3W of the Hercules Section.[12]

Figure 12 shows a somewhat irregular arrangement of long-wall faces. At greater depths it is desirable to maintain a more regular arrangement of faces in order to avoid creating remnants which would be subject to bursting.

Figure 13 shows the more regular progression of long-walls which is desirable to avoid creation of projecting points which would be subject to bursting.

Long-wall operations have traditionally used systems of mat packs, or tight timber cribs for support. These are about 2 or 3 ft square and are placed on 10 or 12-ft centers. At some distance behind the face the hanging-wall subsides onto these supports crushing and compressing them until there is almost complete closure between hanging-wall and foot-wall. During this closure process the hanging- and foot-walls have in many cases remained almost intact although commonly cut by numerous steeply-dipping induced cleavage planes.

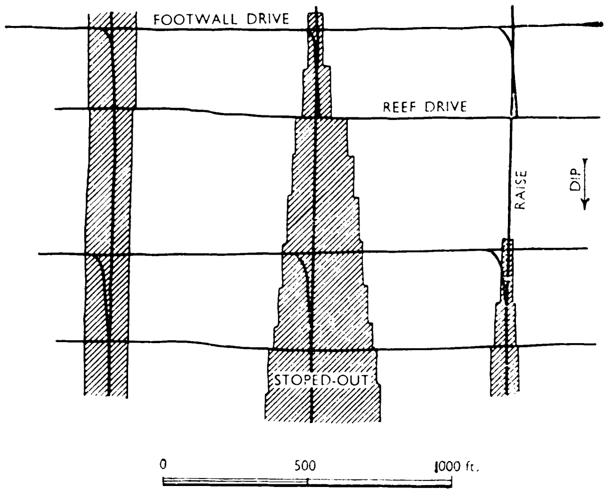


FIG. 13. Stoping plan for a mine at intermediate depth, Witwatersrand.[13]

Hydraulic Props

Experimentation has begun in some of these mines with a view to full caving of the roof as is done in a large proportion of European coal mines.

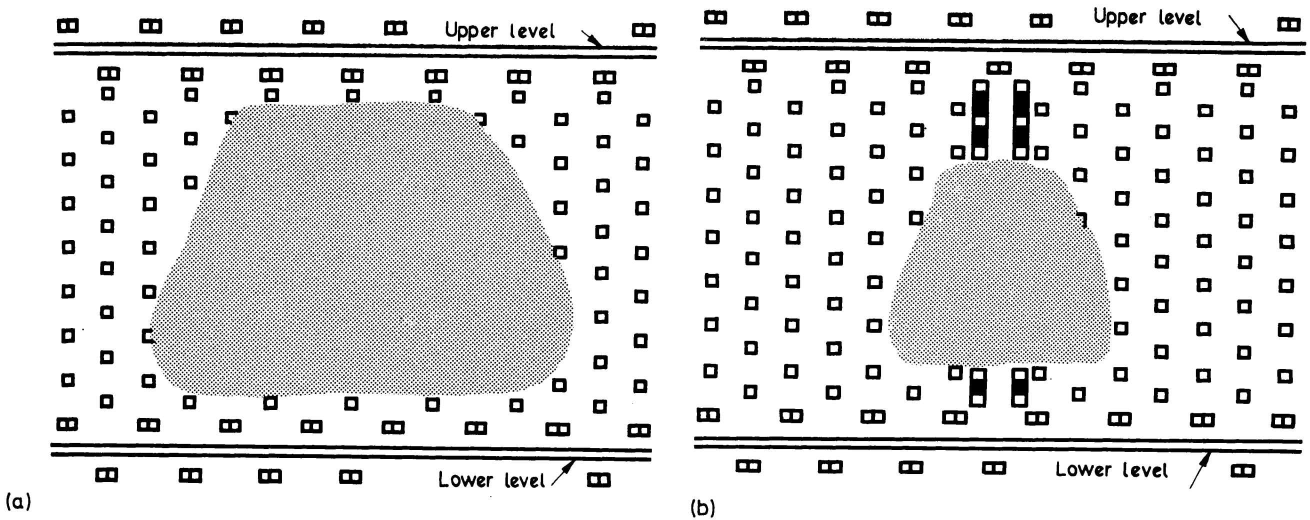
Four rows of hydraulic props placed parallel to the face with the props spaced at 3-ft centers on both strike and dip has been sufficient to produce caving of the roof at the Stilfontein Mine.[15]

At Free State Geduld four rows of hydraulic props have been used and spaced 3-ft apart on dip (normal to the face) and 4 ft apart on strike. It has been found that doubling the back row of props improves the caving.

Mat packs are still used for supporting the strike "gullies" (slusherways).

Timber Cribs

Figure 14 (a, b, and c) shows methods used to remove remnants which are under high pressure and are susceptible to bursting. Timber cribs are distributed uniformly, in a checkerboard pattern to provide for uniform convergence of hanging-wall and foot-wall.



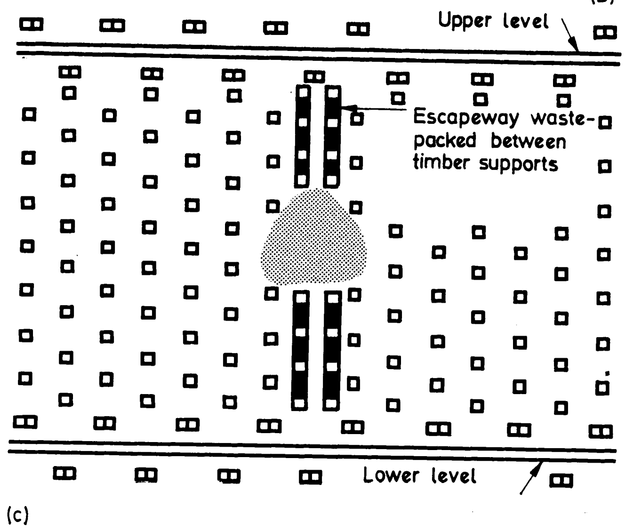


FIG. 14. Stages in remnant removal, Witwatersrand district.(14)

(a) First stage in mining remnant area. Block is being mined faster at the apex and the timber lines are being changed to run more into the face instead of parallel.

(b) Second stage in mining remnant shows how escapeways are being established above and below the block, as mining continues to shrink its size.

(c) In the third and final stage, two different escapeways have to be kept open. They are usually waste-packed between timber supports as shown.

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CHAPTER 6

COSTS AND OTHER FACTORS AFFECTING CHOICE OF MINING SYSTEMS

FACTORS AFFECTING PLANNING AND OPERATION

In the following tabulation are listed most of the factors which must be taken into consideration in determining whether or not a mineral deposit can be profitably mined and/or in selecting a mining system.[1]

(1) External Factors

These control access, location and type of surface construction required. They may be determined by observation and by mapping.

(a) Transportation facilities.

(b) Topography as it may affect tunnel or shaft locations, open pit operations, drainage, and location of plant and storage for supplies, product and waste.

(c) Ownership of surface rights and their value.

(d) Climate, and appropriate type of construction, seasons when transportation and construction are easiest.

(2) General Factors

These may exert distinct control on many phases of design. They must be sought out.

(a) Market for principal and byproduct minerals.

(b) Availability and delivered cost of power, timber, water, waste fill, other supplies, and equipment.

(c) Economic status of the area, type of labor available, wage scales, and living costs.

(d) Housing, school, and recreational facilities.

(e) Financial condition of operating company.

(f) Political factors and governmental restrictions.

(g) Taxes and manner of assessment.

(h) Insurance and compensation laws.

(3) Dominant Features or Geometry of the Mineral Deposit and Enclosing Rocks

These control major outlines of underground layout. Models will illustrate them excellently. They may be determined from core drilling.

(a) Size, shape, dip, rake of deposit.
(b) Type and depth of cover.
(c) Depth of water level.
(d) Presence of multiple deposits.
(e) Unit value (grade) of deposit or deposits, and distribution of value, including low-grade sections that may be maintained intact for later exploitation.
(f) Sharpness of cutoff between valuable and waste rock.
(g) Regularity of deposit, presence of pinches or swells, offshoots of valuable mineral, offsets by faulting.
(h) Structural environment; for instance, massive beds adjacent to shales, competent intrusives in weak rocks, folding.

(4) Physical and Chemical Character of Deposit and Surrounding Rocks

These factors control or influence competence, subsidence, ease of drilling, breaking characteristics, means appropriate to handling, need for ventilation and pumping. They may be determined or approximated by core drilling.

(a) Rock types.
(b) Type and extent of alteration.
(c) Weaknesses, such as bedding, schistosity, consistent mineral cleavage, faulting, jointing, cavities, and the spacing, regularity or pattern they show.
(d) Weaknesses along walls of the deposit.
(e) Tendency of valuable mineral to produce rich fines or mud.
(f) Tendency to pack or be sticky.
(g) Tendency to oxidize and heat or adversely affect treatment or use of product.
(h) Tendency to "air slack".
(i) Presence of "swelling ground".
(j) Abrasiveness.
(k) Silica or silicate content.
(l) Temperature and gradient.
(m) Water occurrence, porosity and permeability of deposit and surrounding rocks, corrosiveness and turbidity of water, its occurrence in pores, in joints, in occasional fissures, channels or pockets, and the interconnection of water-bearing openings.
(n) Ease of drilling.

(5) Treatment of Mine Product

This determines net recoverable value and one component of the cost of the final product. It, therefore, limits the means which may be employed in development and production. It may restrict the type of mine product or require several products. Treatment may be determined by laboratory study of core or assay rejects.

(a) Processes to be used, operating and plant costs, net recoverable value.

(b) Possibilities of separating waste or valuable mineral early.

(c) Desirability of uniformity in value or in physical properties to increase recovery in treatment.

(d) Undesirability of contaminants, as wood chips in crude asbestos.

(e) Undesirability of fines, as in lime for burning.

(f) Necessity of handling more than one type of mine propuct.

(g) Possibilities of cheap recovery from low-grade mine product, as by heap leaching or leaching in place.

Useful Lives of Mine Openings

In general the sub-surface openings and sub-surface structures required for a mining operation may be divided into two classes: (1) permanent facilities and (2) temporary facilities.

The first class includes all those openings and structures which provide access to the mineral deposit during the duration of extraction operations. In some instances, as for example some shafts for European coal mines, the facilities were expected to have useful lives of 50 years or more.

Generally, the average length of time for which a main shaft, for example, is in service is probably no more than 20–30 years. During this time the portion of the mineral deposit tributary to the shaft will be mined out.

Permanent facilities include:

(a) shafts—for hoisting and ventilation;

(b) adits, entries, slopes, or tunnels—for haulage, drainage, or ventilation.

(c) shaft appurtenances—hoist rooms, shaft stations, shaft pockets, pump rooms.

Temporary openings are those created during actual extraction of the mineral. In bedded deposits such openings include rooms and appurtenant entries or cross cuts as well as long-wall faces and appurtenant roadways.

The life of rooms and connecting openings may be only a few days during which pillars are mined out and the roof allowed to cave. Openings giving access to a panel of rooms may be kept open for a few months while main entries may be in service for several years.

The actual required life of the opening at a long-wall face is not more than a few days since the average long-wall face advances from 3 to 6 ft/day and the roof caves 15–20 ft back of the face. Roadways leading to a long-wall face must ordinarily be kept passable for a period up to several years, depending upon the size of the panel being mined.

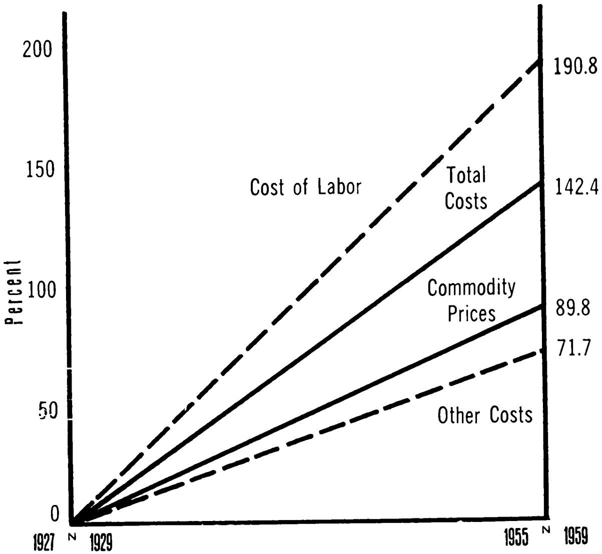

FIG. 1. Percentage increase in costs of square set mining, from 1927–1929 period to 1955–1959 period compared to the general price level.[2]

Average tons per man-shift, 3.9		
	1927–29	1955–59
Cost of labor per ton	$ 2.496 (59 % of total)	$ 7.258 (71 % of total)
Other costs per ton	1.711	2.938
Total cost per ton	4.207	10.196
Wholesale price index (1947-1949 = 100)	66.4 (1928)	126.0 (1958)

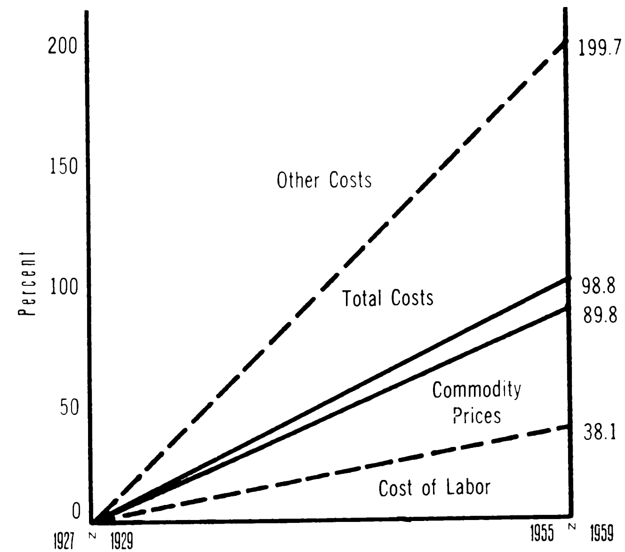

FIG. 2. Percentage increase in costs of room-and-pillar mining from 1927–1929 period to 1955–1959 period compared to the general price level.

Average tons per man-shift, 23.3		
	1927–29	1955–59
Cost of labor per ton	$ 0.648 (63 % of total)	$ 0.895 (44 % of total)
Other costs per ton	0.381	1.142
Total cost per ton	1.029	2.046
Wholesale price index (1947-1949 = 100)	66.4 (1928)	126.0 (1958)

In metalliferous deposits the life of a stope may range from a few months up to a few years depending upon the width of the deposit and method of stoping. Access raises in the stope must be kept open until adjacent ore is stoped out.

Drifts giving access to several stopes must remain passable for a period of several years, during the time required to work out a series of stopes.

Cross cuts which give access to all stopes on a level may require a life of 10 years or more depending upon size of orebody and rate of stoping.

The ideal support system aims at providing just enough support to stabilize the ground for the time during which the opening is required for extraction of mineral. Thus main access openings are provided with permanent types of support while extraction openings are provided with temporary support.

GENERAL MINING COSTS

On the following pages are tabulated general costs for different systems of mining. The rising cost trend which has been in progress for the past 30 years is shown graphically.

TABLE 1. EXPLOSIVE CONSUMPTION FOR VARIOUS MINING METHODS[(2)]

Mining method	Ground condition	Lb. explosives per ton broken: Pillar mining	Stoping: Range	Stoping: Average
Square setting	Weak	0.20	0.30–1.08	0.5
Cut-and-fill	Medium	0.30	0.50–1.23	0.7
Sub-level stoping	Strong	0.26	0.33–0.59	0.4
Room-and-pillar	Strong		0.67–1.14	0.8
Block caving	Weak		0.08–0.19	0.14
Block caving	Medium-strong		0.23–0.47	0.35
Open pit			0.10–0.53	0.28

TABLE 2. TIMBER CONSUMPTION FOR VARIOUS MINING METHODS[(2)]

Mining method	Board ft per ton mined: Range	Average
Square setting	12.0–19.7	15.1
Mitchell slice	7.5–10.5	9.3
Cut-and-fill	0.8– 7.0	4.5
Shrinkage	0.4– 3.9	1.9
Open stopes (small)	0– 1.7	0.7
Sub-level stoping	1.0– 2.0	1.5
Block caving	0– 2.0	1.2

TABLE 3. DRILL BITS AND RODS AVERAGE COST PER FOOT OF HOLE FOR VARIOUS DRILLING RATES(2)

Type drill—bore & rate of penetration	Bit size (in.)	Bit life (ft)	Bit cost (each) ($)	Cost per ft hole ($/ft)	Steel size	Steel life (ft)	Steel cost ($)	Cost per ft hole ($/ft)
Jackleg $2\frac{5}{8}$ in.								
Over 30 in./min	$1\frac{3}{8}$	1500	12.85*	0.009	$\frac{7}{8}$ in. $\times 4\frac{1}{4}$ in. sh	3000	17.00†	0.009
20–30 in./min	$1\frac{3}{8}$	700	12.85	0.018	alloy steel	1500	9.00	0.017
15–20 in./min	$1\frac{3}{8}$	500	12.85	0.026	8 ft section	800	26.00	0.033
Drifters $3\frac{1}{2}$ in.								
Over 30 in./min	$1\frac{5}{8}$	1500	15.15*	0.010	$1\frac{1}{4}$ in. lug	3000	31.10†	0.014
20–30 in./min	$1\frac{5}{8}$	700	15.15	0.022	carbon steel	1500	9.50	0.027
15–20 in./min	$1\frac{5}{8}$	500	15.15	0.030	12 ft section	800	40.60	0.051
Drifters L.H. 4 in.								
Over 30 in./min	2	1500	18.05*	0.012	1 in. hex. exten.	2000	17.50	0.009
20–30 in./min	2	700	18.05	0.026	carburized steel	1500	17.50	0.012
15–20 in./min	2	500	18.05	0.036	4 ft section	1000	17.50	0.018
Crawl-IR $4\frac{1}{2}$ in.								
Over 30 in./min	3	3000	47.20*	0.016	$1\frac{1}{4}$ in. hex. exten.	4500	45.00	0.010
20–30 in./min	3	1500	47.20	0.031	carburized steel	2500	45.00	0.018
15–20 in./min	3	500	47.20	0.094	10 ft section	1500	45.00	0.030
Drillmaster (DHD) $3\frac{1}{4}$ in. §								
Over 30 ft/hr	6	10,000	425.00*	0.043	4 in. OD alloy	47,000	470.00	0.010‡
20–30 ft/hr	6	2500	425.00	0.170	steel tubing	47,000	470.00	0.010
15–20 ft/hr	6	500	425.00	0.850		47,000	470.00	0.010

* Bit cost includes reconditioning cost on the following basis: grinding wheel cost $5.00 each—Labor at $2.00 per hour.

$1\frac{3}{8}$ in. Bits—100 regrinds per wheel

$1\frac{5}{8}$ in. Bits—100 regrinds per wheel

2 in. Bits 50 regrinds per wheel

3 in. Bits—25 regrinds per wheel

6 in. Bits—3 regrinds per wheel

† Reconditioning cost at $1.50 per thread or per shank for over-all of six reconditionings per rod.

‡ Low steel cost because down-the-hole drill transfers no energy through steel.

§ "Down-the-hole" type drill.

TABLE 4. DIRECT MINE OPERATING COSTS FOR VARIOUS MINING METHODS, PERIOD 1955–59[2]

Mining method	Tons mined per month*	Direct mining costs/ton†			Labor percent of total cost‡
		High	Low	Average‡	
Square setting	439,330	$18.72	$6.22	$10.20	71.2%
Cut-and-fill	585,300	14.73	3.07	6.69	56.7
Shrinkage	305,820	8.12	1.75	3.92	N.A.
Room-and-pillar (trackless type)	733,229	2.41	1.16	2.05	43.7
Sub-level stoping	1,547,410	4.71	1.06	2.37	56.9
Sub-level caving	118,150	N.A.	N.A.	4.97	63.3
Block caving	1,803,150	2.25	1.15	1.41**	54.2
Open Pit§	5,198,060	1.15	0.21	0.32	36.4

* Total aggregate tonnage of ore mined each month except for open pits (see footnote §).

† Includes exploration and development, stoping, haulage, hoisting, pumping, and general underground and surface costs.

‡ Weighted average on the basis of tons produced from each mine.

§ Cost is per ton of "material" and is based on total tons of ore and waste handled.

** This average may be on the high side due to lack of information covering a number of efficient operations.

N.A.—not available.

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CHAPTER 1

STRIP MINING (OPEN-CAST MINING) OF COAL

In 1962 30 per cent of the 416 million tons of bituminous coal mined in the United States was produced by "strip mines".

Since 1911 when the first steam shovels were introduced to strip mining there has been a steady increase in the proportion of coal produced by surface mining methods. In 1914, 0.3 per cent of the total bituminous output was produced by strip mining and in 1937, 7 per cent was produced by this method.

The tons per man-shift produced by strip mining methods has always been consistently higher than the tons per man-shift produced by underground methods. In 1960 the average bituminous production per man-shift was 22.9 tons in strip mining as compared with a production of 10.6 tons/man-shift by "deep mining" methods. ("Deep mining" as used in the coal mining industry refers to underground mining methods exclusive of that mined by underground augering from "highwall" faces). During the same year the average production by augering from highwall faces was 31.4 tons/man-shift.

The coal beds which were under the shallowest depths of cover and most easily accessible were the first to be mined and the average depths of overburden which must be removed to expose coal seams has increased year by year. At present some operators are stripping up to 100 ft of overburden and are making plans to strip as much as 120 ft. To cope with the increased amount of rock and dirt which must be handled to expose each ton of coal, stripping machinery of ever increasing size is introduced year by year. A shovel with a dipper capacity of 115 yd^3 designed to move 36 million cubic yards of material yearly was put into service in 1962, and a dragline equipped with a bucket of 85 yd^3 capacity was put into service in 1963 and larger machines are in the design stages.

PLANNING A STRIPPING OPERATION

Assuming that a block of land has been acquired which contains sufficient coal reserves to warrant the investment in stripping equipment and cleaning plant then the next step is to accumulate information on the surface topography, and the depth and thickness and characteristics of the coal seam as well as the nature and thickness of the overburden overlying the coal seam.

Aerial topo maps make good base maps on which to plot coal outcrops, and on which to lay out existing property lines, and to locate proposed roads, plant, spoil areas, etc. Locations for exploratory bore holes may also be spotted on these maps.

The nature and thickness of the various strata which must be stripped from the coal seam may be determined by means of diamond drill cores and the physical characteristics and composition of the coal seam may be determined by examination and analysis of diamond drill cores.

Plant Location

The factors which influence the location of the plant will be the same as those which affect the locations of a plant for a deep mining operation and include the influence of topography, access to rail or water transportation, sufficient area available for plant construction, availability of a site for refuse disposal, and a favorable location which will keep the haulage distance from the coal face to the plant to the practical minimum.

STRIPPING EQUIPMENT

Ratio[1]

The universal factor most used to determine the economics of strip mining is called the "ratio". This refers to the cubic yards of overburden which must be dug to uncover 1 ton of coal. "Ratios" are as high as 20 : 1 at some stripping operations. At one Illinois mine as much as 80 ft of overburden is stripped to recover 28 in. of coal.

Many other factors must also be considered, such as the proportion and hardness of the rock in the overburden, the thickness and quality of the coal seam, costs of labor and materials, sale price of the coal, but basically the "ratio" is of the most importance.

Size and Type[2]

Topography, coal reserves, expected selling price of the coal, type of overburden, spoil area, and tonnage of coal desired per shift are some of the major factors influencing the selection of equipment.

Once the capacity of a mine has been decided and the major stripping machine chosen, other units such as drills, power shovels for loading coal, and trucks should be selected to build a balanced production cycle.

The equipment must be in balance ratio-wise with the maximum output of the mine and at the same time capable of handling the maximum depth of overburden. In other words, maximum yardage determines size and maximum depth determines the range of the machine.

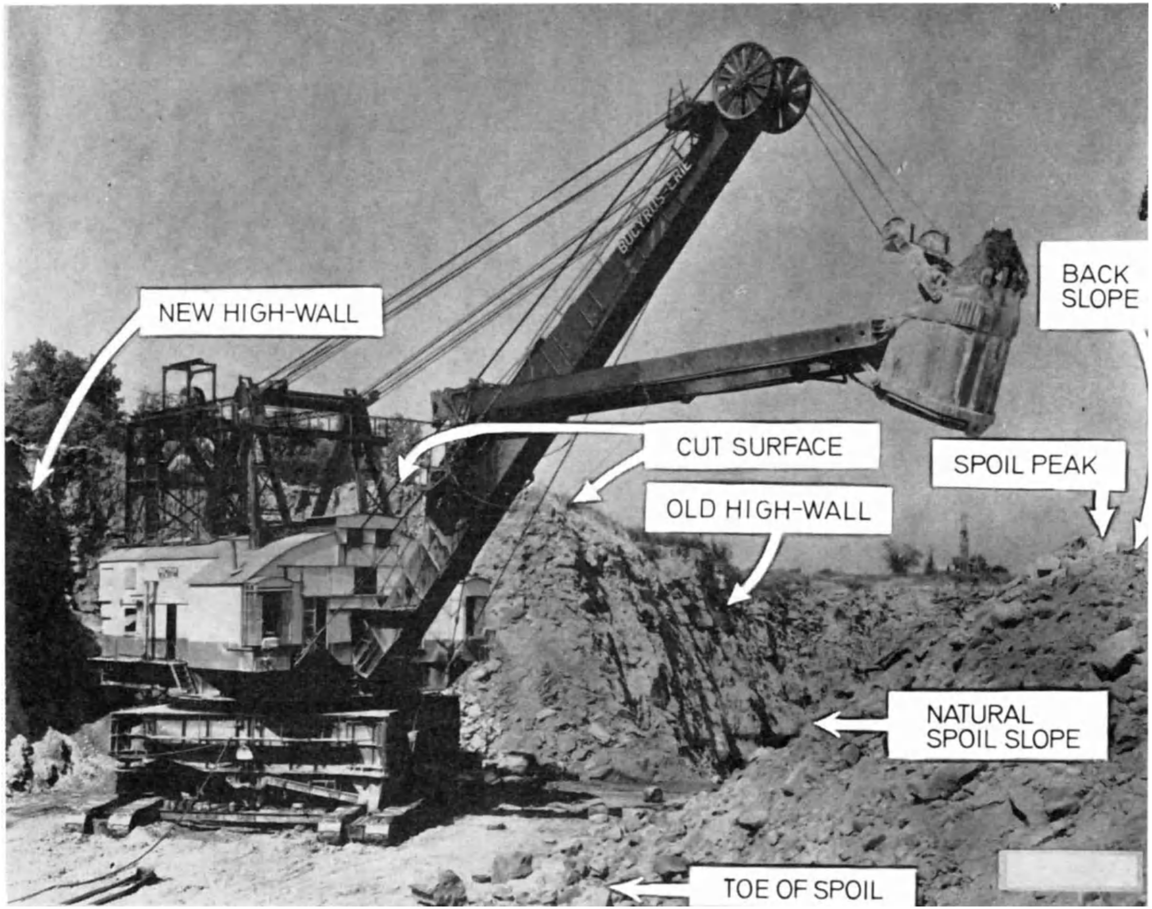


FIG. 1. A typical strip mining operation. (Bucyrus Erie Company.)

Coal is a seasonal product and it is necessary to choose equipment which meets maximum production periods; this equipment is usually larger than would be required to maintain a steady average production rate.

Shovels[2]

Shovels are available in a wide range of designs and capacities to meet most stripping conditions.

For example a 3-yd^3 shovel with a 28-ft boom and a 20-ft dipper handle can cut to a 32-ft height. A 45-yd shovel with a 120-ft boom and a 79-ft dipper handle can cut to a height of 107 ft. The 60-yd shovel which went into service in 1956 has a 150-ft boom and can pile spoil 97 ft high. The 70-yd shovel which began stripping in 1957 has a 140-ft boom and a maximum dumping height of 96 ft.

The giant 115-yd^3 shovel which went into service in 1962 has a reach of more than 460 ft and is powered by fifty motors ranging from $\frac{1}{4}$ to 3000 h.p. It will remove 3 million yd^3 of overburden per month while working in banks more than 100 ft high.

At an Ohio operation a 65-yd shovel operating around the clock exposes 2 million tons of coal per year. This shovel is equipped with a 135-ft boom and a 91-ft handle and can stack spoil 103 ft high. Its dumping reach from the outside of its crawlers is 121 ft. Overburden thickness ranges from 39 to 120 ft.

At another operation a beefed up old 33-yd shovel digs 50 ft of shale and some sandstone without the aid of explosives, averaging 10,050 bank yards per shift or 900,000 yd^3/month. The company uses a three man crew on this unit, two operators and a ground man on each of the three shifts. The two operators alternate running and oiling and thus operator fatigue does not become a limiting factor in production.

Relative Costs[2]

Big shovels in the 33- to 70-yd range usually work to a maximum of 70–80 ft of cover. Removal of overburden between 0 and 50 ft thick by a 45-yd shovel costs about 45 per cent of what it would cost to do the same job with an 8-yd shovel; in overburden between 9–90 ft, about 77 per cent of the cost with an 8-yd shovel, and in overburden between 50 and 80 ft, about 25 per cent more than an 8-yd shovel working in a bank 50 ft high.

Draglines[2]

Draglines are available in a wide range of sizes to meet varying conditions.

A $2\frac{3}{4}$-yd dragline with a 110-ft boom can dig to a depth of 58 ft and pile spoil to a height of 49 ft above the bottom of the bench on which it is working. A 35-yd unit

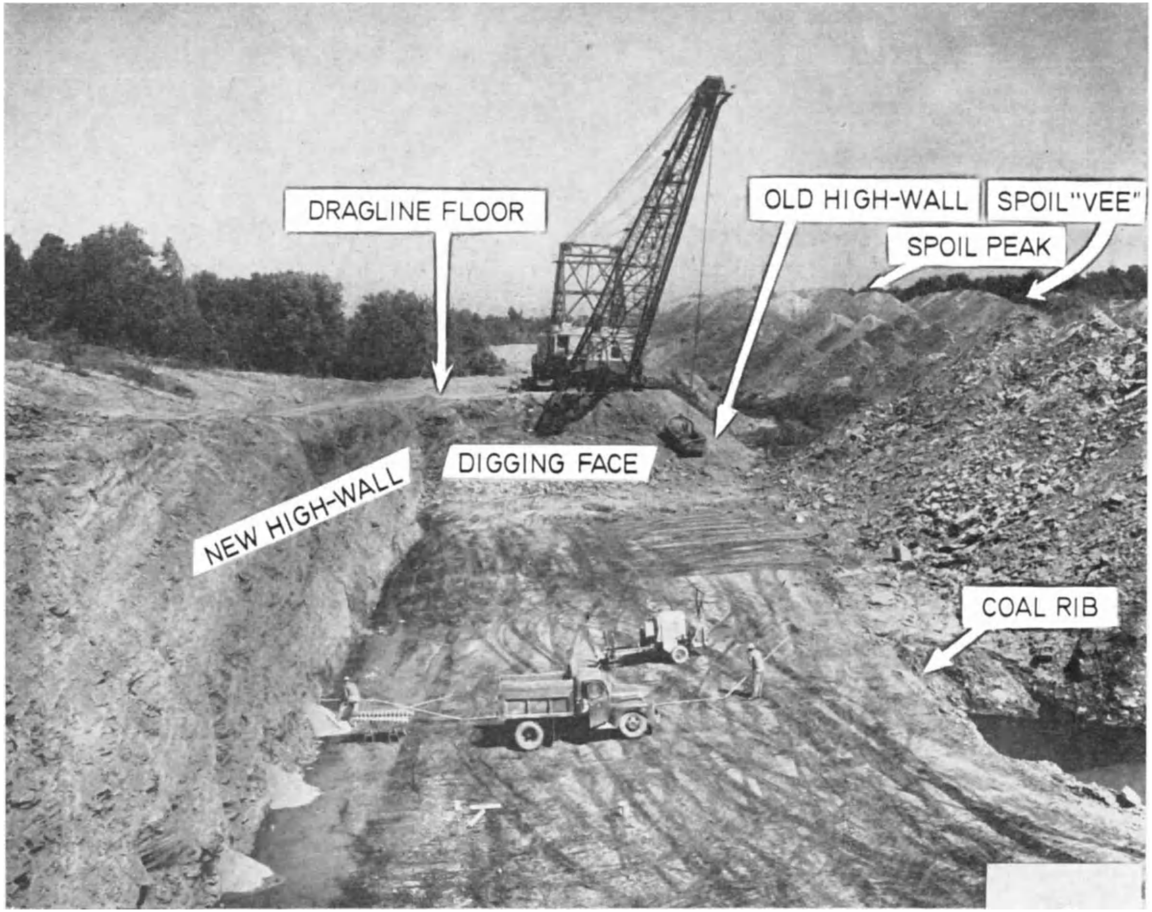


FIG. 2. Stripping operations using a dragline. (Bucyrus Erie Company.)

with a 220-ft boom can dig to a depth of 94 ft and pile spoil 98 ft above the tub (on which the dragline sits).

An 85-yd^3 machine was completed in 1963. It had a 275-ft boom, a 248-ft dumping radius and a 143-ft dumping height. Its working weight was about 5500 tons.

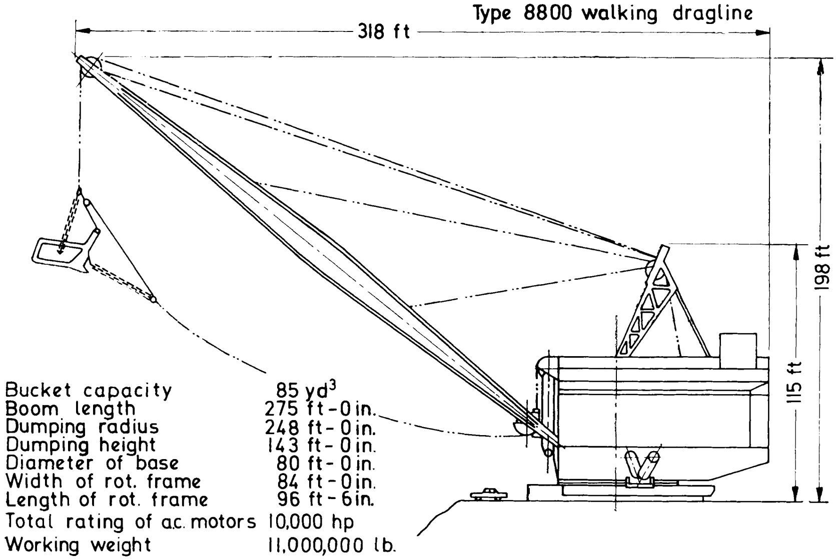


FIG. 3. The largest dragline – bucket capacity 85 yd^3.[4]

Typical Operation

A typical operation using bulldozers and draglines for stripping[3] removes overburden consisting of from 50 to 60 ft of medium hard shale and 20–30 ft of sandstone which is drilled and shot so that about 25 ft of it is cast to the spoil area by the blast. Two heavy bulldozers then team up to remove about another 25 ft of overburden to the spoil area. A diesel dragline with a 165-ft boom and a $14\frac{1}{2}$-yd bucket follows the bulldozers and throws material as far away from the highwall as possible. This dragline casts an average of about 800 yd^3/hr and the two bulldozers push this material to the spoil area.

Flat coal seams and steep slopes cause overburden thickness to increase rapidly as successive cuts advance into the hillside. To meet these difficult conditions the large walking dragline is most useful because of its long dumping range. The disadvantage in using the large dragline is that it must have a suitable base on which to sit and this is sometimes difficult to provide in rocky overburden. This factor must be considered in choosing between a dragline and a shovel.

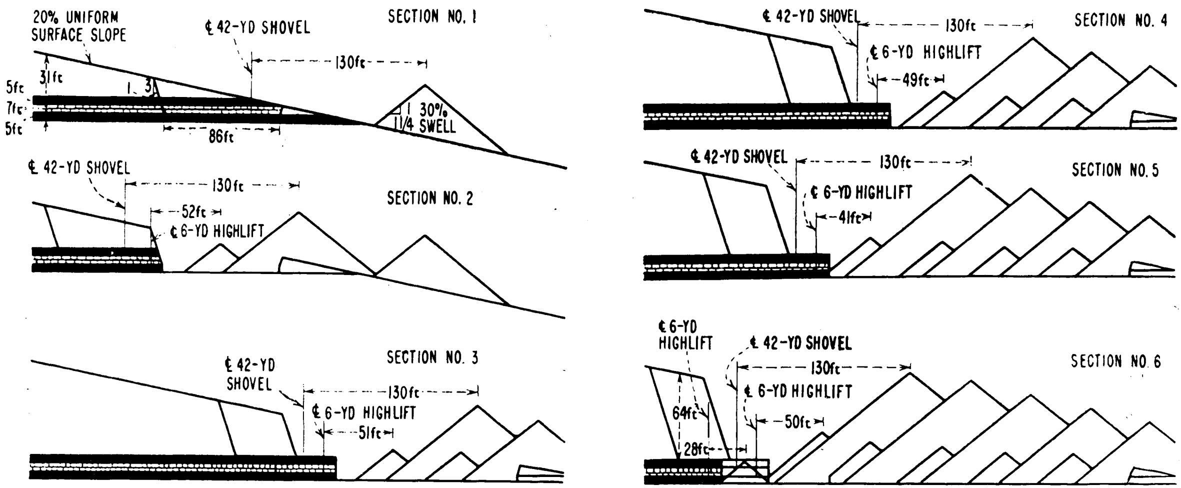


FIG. 4. Typical cross-sections of a two-seam stripping operation show how a 42-yd shovel with a 130-ft dumping radius takes the main portion of the overburden on the top seam and a 6-yd highlift machine skins off the thin layer of rock covering the lower coal.[5]

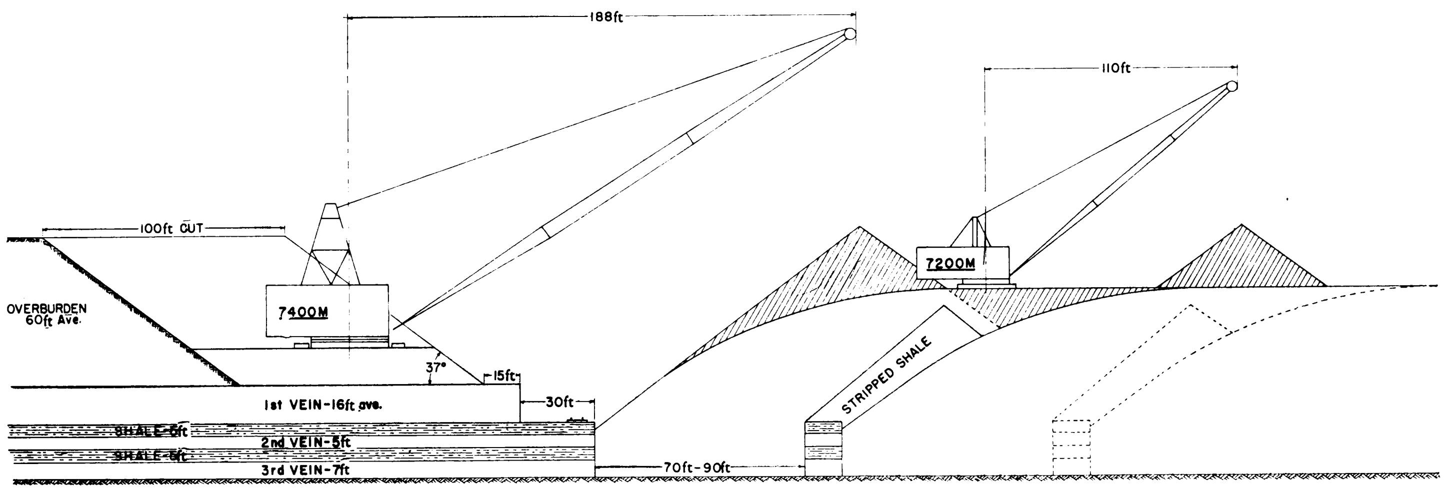


FIG. 5. Typical cross-section showing the stripping of gypsum veins, Alabaster, Michigan.[6]

STRIPPING METHODS

The three basic stripping systems are the following:

(1) A single stripping shovel traveling on the exposed coal seam digs and removes the overburden ahead of it and piles it in the cut from which coal has previously been removed.

(2) A single dragline traveling on a bench above the coal strips overburden to widen the bench for its travel way for the next cut and also removes the high-wall bench over which it has just traveled to expose the coal seam.

(3) A shovel and a dragline are used in tandem with both traveling on exposed coal. The shovel works ahead of the dragline removing the lower bench to expose the coal seam and piling the spoil in the cut from which coal has previously been removed. The dragline removes the upper portion of the overburden to form another bench and casts the spoil over and behind the spoil piled by the shovel.

Numerous combinations of shovels, draglines, bulldozers, scrapers, and wheel excavators are possible.

Back-cast Stripping

At its Alabaster, Michigan, quarry, the U.S. Gypsum Co. mines three gypsum veins by stripping methods using two draglines, one of which strips and casts the overburden while the other re-casts a portion of the overburden.(6)

Figure 5 is a cross-section through the stripping operation showing the 7400M dragline which is equipped with an 11-yd bucket on a 200-ft boom and the 7200M dragline which is equipped with a 5-yd bucket on a 135-ft boom.

The gypsum veins are loaded out onto rail cars by power shovels.

The stripping ratio, including both primary overburden and shale dividers, averages 1.85 yd^3/ton of gypsum rock. To produce the average annual production of a million tons of gypsum requires stripping about 1,850,000 yd^3 of waste.

The percentage breakdown of costs is as follows:

	% of total
Stripping	32
Drill and blast	15
Rock loading	17
Haulage	21
Supv. and engrg.	7
Miscellaneous	8
Total	100%

Shovel-dragline Combination

Figure 6[7] shows the stripping sequence used in stripping and mining the 36-in. No. 6 bed at Blue Crystal Mines, Inc., in Coshocton County, Ohio.

Figure 6 (a) shows the method used in making the initial cut near the outcrop. The diesel dragline is equipped with a 90-ft boom and a 4-yd bucket. Using a 30° boom angle this unit has a maximum digging reach of 114 ft but is given a restricted reach of 80 ft for vertical dropping of the bucket. It can dump at any distance up to 90 ft and at a maximum height of 38 ft.

As shown in Fig. 6(a) the initial cut is made at such a location along the outcrop as to provide for natural drainage from the pit, if possible. The dragline is positioned so as to develop a 100-ft wide bench of coal, not including the coal near the outcrop which would be soft and of poor quality.

The first cut C_1 is made by downcutting along the high-wall 80 ft from the machine and dumping the spoil S_1, 90 ft to the rear. Assuming a surface grade of 15 per cent which is the average for the area, a 28-ft high-wall with a 1 in 3 slope is developed and a 35-ft wide coal bench is exposed.

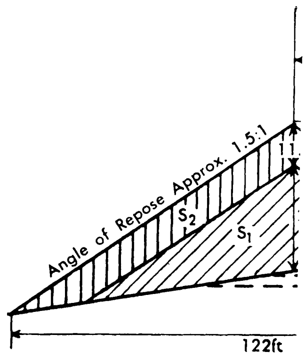
The second cut, C_2, can be made from the initial position if that is desirable, opening a coal bench 65 ft wide. S_2 spoil is dumped at the same point, thereby developing a bank 39 ft high and 122 ft wide. These values are based on the loose soil, clay, and soft shale of the overburden increasing (swelling) 15 per cent in volume, and on an assumed slope of $1\frac{1}{2} : 1$.

The high sulphur material which lie immediately above the coal bed are the last materials excavated and are buried by the S_2 spoil. The toxic materials from C_2 are dumped along the side of the spoil bank so as to permit later coverage.

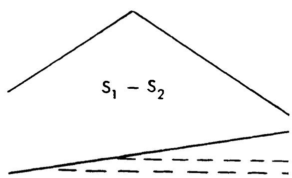
A diesel shovel follows up the initial dragline work. This shovel is equipped with a 48-ft boom, a 34-ft dipper handle or "stick", and a $3\frac{1}{2}$-yd dipper. With a 45° boom angle the shovel has a maximum digging radius of 62 ft at a maximum height of 49 ft, a floor level digging radius of 33 ft, and a maximum dumping radius of 52 ft at a maximum height of 38 ft.

As shown in Fig. 6(b) the shovel is positioned on the pit bottom 7 ft from the toe of the high-wall left by the dragline. The shovel removes C_3 and dumps the spoil to form S_3, thereby uncovering approximately 24 ft of coal bench. This material is somewhat tougher than that handled by the dragline, but swell is still 15 per cent. The slope of the piled material is steeper than that of S_2, being $1\frac{1}{4} : 1$, so S_3 becomes a spoil bank 30 ft high and 74 ft wide.

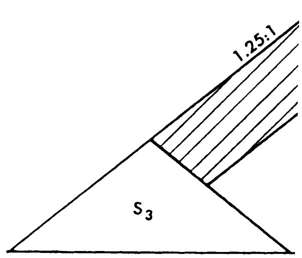
After removing C_3 for as great a distance along the high-wall as is desirable the stripping shovel returns to the initial operating point for removal of C_4. The material along the toe of the high-wall is cast into the bottom following removal of the coal under C_3, thereby providing for coverage of the acidic materials. From position 3 the shovel removes C_4 material which is still largely unconsolidated material which does not require blasting, and dumps it to form pile S_4, uncovering the re-



(a)



(b)



(c)

(a) Op
(b) Un
(c) Stri

maining 41 ft of the 65-ft wide pit. This spoil bank is 37 ft high (based on 15 per cent swell and $1\frac{1}{4}:1$ slope) and together with S_3 gives a total bank width of 122 ft.

Removal of overburden beyond the average 35-ft high-wall left by the stripping shovel is accomplished by an electric dragline equipped with a 140-ft boom and a 10-yd³ bucket. With a 30° boom angle this unit has a maximum reach of 152 ft but a restricted reach of 131 ft for vertical dropping of the bucket. It can dump at any distance up to 131 ft and at a maximum height of 65 ft.

As shown in Fig. 6(c) a 100-ft bench of coal is uncovered by the dragline in several steps. The overburden is tough and requires blasting for fragmentation.

From position 4 the dragline digs a "keyway", shown as C_5, with an average width of 25 ft and from 25 to 35 ft deep, adjacent to the higher ground so as to develop a practically vertical high-wall. This keyway provides two faces for attack on the remainder of the cut face. Spoil from C_5 is dumped into the pit adjacent to the old high-wall to provide a future operations footing for the dragline.

Due to the nature of the material, the swell figure is 20 per cent while the spoil bank slope is $1\frac{1}{4}:1$. The space occupied by the C_5 material is indicated as S_5, and includes the cross-hatched area adjacent to the old high-wall.

The dragline then moves to any position above C_6 convenient for removal of that material. Figure 6(c) indicates a position 5 which could be used to remove all of C_6, the spoil being dumped to form bank S_6. However, the dragline could move toward the spoil band and work from a position on S_5 if necessary, casting material beyond S_6. Removal of C_6 will involve recasting of the cross-hatched portion of S_5 adjacent to the old high-wall in order to remove coal adjacent to S_5. The spoil banks will stand with practically vertical faces for short periods of time, but the coal must be removed quickly to avoid trouble from slides.

Figure 6(c) indicates a 52-ft high-wall on completion of the C_5–C_6 pit. A maximum of 60–65 ft of overburden can be removed economically under today's market conditions. If the grade conditions of Fig. 6(c) remained constant, a 75-ft wide pit could be developed to the right with a 65-ft high-wall. Frequently the grade diminishes as the crest of a knob is approached, and another 100-ft pit might be developed without exceeding 65 ft of overburden.

Draglines vs. Shovels

Some advantages of draglines over shovels for stripping applications are the following:

(1) The dragline moves overland with less difficulty than the shovel and is therefore more useful where the coal reserves lie in various scattered small bodies.

(2) The dragline is more effective in performing other work required in connection with the stripping operations such as building roads and ditches.

(3) The dragline can clean a rough coal seam and leave it ready for loading.

The principal advantage of the shovel is that it can dig overburden which is tight or poorly broken and may make unnecessary the shooting of a portion of the overburden, or reduce the amount of shooting which is required.

Comparative Stripping Costs — Large Shovels vs. Small Shovels

Stewart and McDowell[8] have used the actual annual output of a 70-yd shovel equipped with a 140-ft boom as compared with the annual output of a 13-yd shovel equipped with a 95-ft boom to compute typical comparative costs of moving overburden.

During the year of 1956 the 13-yd shovel moved an average of 360, 000 yd^3/month and 600 yd^3/digging hour. The average overburden depth was 33 ft.

During 1957–1958 the 70-yd shovel moved an average of 1,582,000 yd^3/month or 3250 yd^3/hr. Average overburden depth was 61 ft.

The average output of the 70-yd machine was 4.4 times the output of the smaller shovel. Based on the same three-man operating crew and the same pay scale the direct operating labor cost for the large machine was only about 24 per cent of that for the small machine.

Cost of Moving a Cubic Yard of Overburden

The estimated operating costs have been prepared on the basis of the following (the 13-yd shovel is designated as Type 5320 and the 70-yd shovel is designated as Type 5760):[8]

(1) Labor—assume the same for both shovels as shown on the following schedule.

(2) Electric power—Cost equals \$0.0125 per kWh.

(3) Maintenance—for the Type 5320 over a 12- to 15-year period, a cost figure of \$0.03 per yd^3 has been used, which is a reasonable average. There are no long time averages available as yet for the Type 5760. Because of such things as modern design, welded construction and better steels, it is reasonable to assume an average maintenance cost of \$0.025 per yd^3 for the Type 5760.

(4) Amortization—It is assumed that the Type 5320 has been completely written off at this time and there would not be amortization charges applicable to this unit. The *guessed* present day installed cost of the Type 5760 is \$3,300,000, and amortizing this over 15 years of 11 months of 720 operating hours each, the amortization charges on this unit will be \$27.78 per operating hour.

The following tabulation shows the estimated operating cost for the two shovel units covered by this analysis study:

			Type 5320	Type 5760
Operator	at	$4.5000	$4.50	$4.50
Oiler	at	4.0000	4.00	4.00
Pitman	at	3.7500	3.75	3.75
200 kWh	at	0.0125	2.50	–
1000 kWh	at	0.0125	–	12.50
Maintenance	at	0.0300	15.00	–
Maintenance	at	0.0250	–	56.50
Amortization			–	27.78
Total per hour			$29.75	$109.03
yd^3 per hour			500	2250
per yd^3			$0.0595	$0.0485

The summary indicates that in spite of the high cost of amortizing the $3,300,000 estimated cost of the large shovel the overall total cost for moving a cubic yard of overburden is less than for the older and smaller shovel which had previously been completely amortized.

Of more importance than the lower cost, however, is the fact that the calculated maximum overburden depth which can be dug and spoiled by the Type 5320 is about 48 ft while under the same conditions the Type 5760 can dig and spoil overburden 66 ft deep. Thus the larger shovel makes available reserves which could not be handled by the smaller machine without the use of auxialiary equipment which would in turn raise the handling cost per cubic yard.

PERFORMANCE OF LARGE STRIPPING EQUIPMENT

The following tables summarize the results of a survey made by the American Mining Congress Committee on Strip Mining on shovels and draglines in operation in 1959.[9]

TABLE 1. CAPACITY PER HOUR PER YARD OF BUCKET OR DIPPER CAPACITY[9]

Shovels		Draglines	
Dipper capacity (yd^3)	Capacity /hr/yd	Bucket capacity (yd^3)	Capacity /hr/yd
10–15	38	10–15	39
16–20	42	16–20	–
21–25	39	21–25	40
26–30	42	26–30	44
31–35	43	35	33
36–40	37		
41–45	42		
55	42		
65	46		
70	44		
Average	42	Average	39

TABLE 2. BOOM LENGTH[9]

Shovels		Draglines	
Dipper capacity (yd^3)	Length (ft)	Bucket capacity (yd^3)	Length (ft)
10–15	90	10–15	172
16–20	96	16–20	–
21–25	102	21–25	189
26–30	108	25–30	196
31–35	108	35	220
36–40	119		
41–45	117		
55	145		
60	158		
65	140		
Average	113	Average	184

TABLE 3. PERCENT RUNNING TIME[9]

Shovels		Draglines	
Dipper capacity (yd^3)	%	Bucket capacity (yd^3)	%
10–15	75	10–15	81
16–20	76	16–20	–
21–25	78	21–25	82
26–30	80	26–30	83
31–35	77	35	85
36–40	77		
41–45	78		
55	78		
65	77		
70	–		

COMPUTER METHOD FOR ESTIMATING PROPER MACHINERY MASS FOR STRIPPING OVERBURDEN

The following treatment, dealing with a computer method for determining the proper sizes of equipment to select for the most economical stripping operations, is taken from a thesis by Henry Rumfelt which was submitted to Texas A. & M. College as a requirement for a professional degree. The paper was subsequently published as Preprint No. 60-F-300, and was presented at the St. Louis Section AIME

TABLE 4. OVERBURDEN DEPTH[9]

Shovels			Draglines		
Dipper capacity (yd^3)	Avg. (ft)	Max. (ft)	Bucket capacity (yd^3)	Avg. (ft)	Max. (ft)
10–15	32	46	10–15	47	67
16–20	36	48	16–20	—	—
21–25	47	72	21–25	58	73
26–30	44	55	26–30	59	71
31–35	50	65	35	85	130
36–40	44	61			
41–45	47	82			
55	41	60			
65	55	80			
70	55	90			

TABLE 5. AVERAGE MACHINE[9]

	Shovel	Dragline
Bucket or dipper capacity	35 yd^3	19 yd^3
Boom length	113 ft	184 ft
Overburden handled/hr	1471 yd	765 yd
Running time	77%	81%
Avg. depth of overburden	44 ft	54 ft
Max. depth of overburden	65 ft	72 ft

and Coal Division of SME, September, 1960 and was also published in *Mining Engineering*, May, 1961.[10]

"Overcasting" is used in certain mining activities including strip mining and it is used in some heavy construction work such as channel excavation. The stripping shovel and stripping dragline are the most common machines. In addition an overcast wheel device, known as the Kolbe type wheel excavator, is very useful in certain types of overburden that are particularly suited for it. The obsolescent tower machines have been advantageously used in the past and some are still successfully employed. Overcasting may be performed in a simple operation consisting of digging out the material, lifting it from one position, moving it over, and dumping it in the spoil position where it remains, for practical purposes, indefinitely. The mechanics of the operation, regardless of the machines used, may be called "simple overcasting".

At times more than one machine is required to effect the cut. Then the procedure is no longer considered simple overcasting. Rehandling may be involved and

TABLE 6. SURVEY OF DRAGLINE

Machine No.	Bucket capacity (yd^3)	Boom length (ft)	Overburden handled (yd^3)			Running time (%)	Overburden depth	
			Per hr	Per month	Per year		Ave.	Max.
1	12	160	550	350,000	3,750,000	85	50	65
2	12	165	371	155,344	1,864,124	80	73	100
3	12	175	529	258,542	3,102,500	76	48	54
4	12	175	444	222,524	2,670,289	75	47	60
5	12	175	304	128,000	1,540,000	75	51	79
6	12.5	160	475	300,568	3,606,816	94	55	80
7	12.5	165	619	336.008	4,032,100	94	32	90
8	12.5	170	486	259,008	2,978,601	83	44	65
9	13	165	599	388,453	4,467,219	89	41	70
10	13	175	256	146,212	1,681,445	79	—	—
11	13	180	431	204,684	2,456,210	82	32	44
12	13	—	350	—	1,753,302	64	(A)	(A)
13	13	—	801	444,610(B)	—	86	73	80
14	14	160	650	420,000	4,500,000	85	40	50
15	14	215	683	400,000	3,592,225	83	24	30
16	20	195	1039	442,050	5,304,600	80	30	45
17	21	195	478	271,427	3,257,123	85	27	—
18	21	200	851	379,000	4,544,000	61*	42	56
19	23	200	946	432,000	5,199,800	62*	35	55
20	(D)	185	1286	579,773	6,667,394	81	60	75
21	25	180	748	420,000	4,289,740	80	69	80
22	25	180	865	505,000	5,579,410	81	86	100
23	25	180	835	499,351	5,992,207	89	87	97
24	27	180	1161	516,897	2,842,931(E)	83	35	42
25	30	200	763	440,000	4,836,950	80	86	100
26	30	200	770	338,061	4,056,731	86†	71	110
27	30	200	1052	612,204	7,346,455	86	77	84
28	30	200	1112	650,000	—	91	40	60
29	35	220	1167	566.402	6,796,826	82	85	130

(A) Handling a 12 to 15-ft parting.
(B) Operated 1 month in 1959.
(C) In tandem with bulldozers.
(D) Machine had a 23-yd bucket for 8 months and a 27-yd bucket for four months in 1959.

this additional work may be accomplished by another machine, or it may be performed by the prime excavator. Another procedure which employs the dragline involves a temporary fill extending from the bench of the cut over to the spoil and requires a certain amount of rehandle by the prime excavator. It is called the

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Overburden type			Quality of shooting			Hours machine down due to market conditions or holidays	Machine working		Machine No.
Rock	Shale	Other	Good	Fair	No shoot-ing		Alone	In tandem	
x	x		x			–	x		1
x	x		x			–	x		2
	x	x			x	985	x		3
x	x	x			x	738	x		4
x	x	x	x			–	x		5
x			x			362	x		6
x			x			362	x		7
x	x		x			543	x		8
x	x		x			8	x		9
x	x		x			125		x	10
x	x		x			–	x		11
x				x		485	x		12
x	x		x			24		(C)	13
x	x		x			–	x		14
	x	x			x	1680	x		15
x	x	x	x			2160	x		16
	x		x			1008		x	17
x	x	x		x		1968	x		18
x	x	x	x			1416	x		19
x	x			x		2360	x		20
	x	x			x	1208	x		21
	x	x			x	416	x		22
	x				x	–	x		23
x	x		x			–	x		24
	x	x			x	368	x		25
x	x		x			2664		x	26
	x				x	–	x		27
	x		x			–	x		28
x			x			362	x		29

(E) Operated 5½ months in 1959.

* Running time includes normal operating delays such as oiling and mechanical, electrical and maintenance repairs.

† Running time includes deadheading.

"extended bench" or "filled pit" procedure. "Tandem" operations involve one machine taking a part of the cut and another, following the first, taking the remainder. Yet another common type operation involves two machines with partial rehandle by one. It involves the prime excavator, a shovel for example, doing all

TABLE 7. SURVEY OF SHOVEL

Machine No.	Dipper capacity (yd^3)	Boom length (ft)	Overburden handled (yd^3)			Running time (%)	Overburden depth	
			Per hr	Per month	Per year		Ave.	Max.
1	10	85	400	75,000	–	75	45	65
2	13	85	409	200,000	1,770,553	70	22	25
3	15	87	434	240,000	2,639,536	83	29	35
4	15	90	762	311,966	3,743,591	68	33	55
5	15	94	632	370,000	2,444,566	78	20	35
6	15	–	426	206,798	–	67	22	28
7	15	96	610	273,043	3,276,514	85	50	80
8	16	95	723	312,970	3,755,640	79	21	39
9	17	95	655	276,000	3,308,850	58*	32	47
10	17	95	707	237,653	–	75	43	46
11	18	105	866	406,723	4,880,679	78	39	75
12	20	89	855	459,562	5,514,746	80	50	–
13	20	96	745	420,000	4,636,778	84	29	35
14	22	95	601	249,755	2,747,301	80†	52	70
15	22	110	978	495,765	5,949,185	74	52	70
16	24	100	1102	566,019	6,792,233	79	37	75
17	26	95	996	523,311	6,279,740	76	46	60
18	26	115	1058	625,000	6,879,408	81	35	50
19	28	123	1451	890,000	9,801,887	80	49	55
20(B)	29	–	1405	709,145	–	81	54	60
21(B)	29	–	1394	886,189	–	86	62	70
22	30	102	1174	700,000	8,020,466	83	24	30
23	30	105	1362	675,000	8,127,500	68*	42	62
24	30	–	1031	540,712	6,488,541	84	36	55
25	(C)	133	1275	547,855	6,300,330	72	33	40
26	33	105	1210	634,473	7,296,441	75	49	59
27	33	108	1336	555,210	6,662,500	77	29	41
28	33	113	1504	805,412	9,664,946	79	77	85
29(B)	33	–	1825	1,164,094	–	86	33	50
30	35	105	1405	594,040	7,128,985	69	34	60
31	35	108	1311	699,935	8,399,221	80	76	80
32	35	108	1640	717,551	8,610,607	65	53	80
33	36	100	997	570,000	6,289,871	79	41	70
34	36	113	1459	506,589	6,079,068	77	33	40
35	40	113	1530	609,054	7,308,644	71	49	60
36	40	120	1321	568,626	6,823,517	71	57	75
37	40	120	1408	–	8,576,795	74	45	76
38	40	120	1329	686,838	8,242,058	75	67	72
39	40	120	1940	1,200,000	13,186,639	82	40	60
40	40	127	1339	610,000	5,052,242	76	24	35
41	40	135	1749	1,000,000	10,938,417	86	37	65

PERFORMANCE FOR 1959(9)

Overburden type			Quality of shooting			Hours machine down due to market conditions or holidays	Machine working		Machine No.
Rock	Shale	Other	Good	Fair	No shooting		Alone	In tandem	
x	x	x	x			–	x		1
x			x			1784	x		2
x	x		x			1062	x		3
x	x	x	x			1488	x		4
x	x		x			3171	x		5
x	x		x			None	x		6
x	x		x			–	x		7
x	x	x	x			1944	(A)		8
x	x	x		x		1920	x		9
x	x		x			–	x		10
x			x			361	x		11
x				x		1008		x	12
x	x		x			1024	x		13
x	x		x			3061		x	14
x	x	x	x			512	x		15
x	x	x	x			–	x		16
x	x	x	x			504	x		17
x	x		x			360	x		18
x	x		x			25	x		19
x	x		x			120	x		20
x	x		x			8	x		21
x	x		x			309	x		22
x	x	x	x			888	x		23
x	x	x		x		1960	x	x	24
x				x		1856	x		25
	x	x			x	696	x		26
x	x	x	x			2120	x		27
x	x		x			–		x	28
x	x	x	x			–	x		29
x	x	x	x			1408	x		30
x	x	x	x			794		x	31
x	x	x			x	714		x	32
x	x		x			461		x	33
x	x		x			–	x		34
x	x		x			–		x	35
x	x	x	x			1456	x		36
x	x	x	x			342	(D)		37
x	x	x	x			458		x	38
x	x		x			91	x		39
x	x		x			3138	x		40
	x		x			1133	x		41

TABLE 7 (*cont.*). SURVEY OF SHOVEL

Machine No.	Dipper capacity (yd^3)	Boom length (ft)	Overburden handled (yd^3)			Running time (%)	Overburden depth	
			Per hr	Per month	Per year		Ave.	Max.
42	42	135	2205	730,664	8,767,972	73	37	43
43	45	105	1479	750,000	6,433,459	83	31	60
44	45	113	1662	865,390	10,384,682	79§	46	82
45	45	120	1795	817,067	9,804,801	77	39	110
46	45	120	1765	798,985	9,587,814	76	56	105
47	45	105	2263	1,310,859	15,730,310	87	44	90
48	45	113	1540	722,587	6,503,285	81§	76	103
49	45	120	1916	786,548	9,438,570	75	34	63
50	45	120	1928	1,050,809	12,609,709	81	44	90
51	45	120	1966	869,116	10,429,396	69	57	90
52	45	120	1972	1,103,441	13,247,295	84	59	90
53(E)	45	–	1820	850,000	–	68	40	60
54	55	145	2321	1,300,000	13,550,169	78	41	60
55	65	–	2768	1,200,000	12,182,347	71	41	60
56	65	150	2958	1,592,473	19,109,678	80	80	120
57(F)	65	165	3165	1,900,000	–	81	44	60
58	70	140	2879	1,600,000	14,662,648	–	60	100
59	70	140	3135	1,600,000	20,918,728	78	66	104
60(G)	70	140	3200	1,534,755	13,380,279	58	38	65

(A) Machine worked alone 92% of the time in 1959; in tandem 8% of the time.
(B) Operated only 1 month in 1959.
(C) Operated 7 months with a 32-yd dipper and 5 months with a 26-yd dipper in 1959.
(D) Shovel operated alone most of the time, however, in deep material a dragline worked on the spoil.
(E) Operated 3 months in 1959.
(F) Operated 2½ months in 1959.

the digging and another machine, usually a walking dragline, rehandling that part of the spoil the shovel cannot put away. It is generally called "shovel-pullback" operations.

In the past 12–15 years there has been a rapid increase in size and models of stripping machinery. During this time draglines have almost doubled in size and shovels have almost quadrupled. An English manufacturer has entered the field and is offering a complete line of walking draglines. Also, it is reported that the Russians have already produced large shovels and draglines. Furthermore, during this period the overcast wheel excavator moved into the category of an acceptable stripping machine from that of an experimental model.

The trend in the coal industry is to increase the proportion of the overall pro-

PERFORMANCE FOR 1959[9]

Overburden type			Quality of shooting			Hours machine down due to market conditions or holidays	Machine working		Machine No.
Rock	Shale	Other	Good	Fair	No shooting		Alone	In tandem	
x	x		x			–	x		42
x	x			x		2990	x		43
x	x		x			760	x		44
x			x			361.5	x		45
x			x			361.5	x		46
x	x	x		x		–	x		47
x	x		x			1272‡		x	48
x	x		x			–	x		49
x	x	x		x		–	x		50
x	x	x	x			–	x		51
x	x	x	x			–	x		52
x	x			x		40	x		53
x	x			x		888	x		54
x	x			x		59	x		55
x	x	x	x			–	x		56
	x		x	x		–	x		57
x	x			x		297	x		58
x	x			x		29	x		59
x	x	x			x	1020	x		60

(G) Low operating time, which affected yardage, due to crews being used to replace redesigned parts. Operator reports data does not present true picture of machine capacity.

* Running time includes normal operating delays such as oiling and mechanical, electrical and maintenance repairs.

† Running time includes deadheading.

‡ Shovel also parked 3 months during 1959.

§ Running time includes regular oiling and deadheading.

duction that is obtained by strip mining. For example the Bureau of Mines statistics show that in 1958 the percent of total production mined by stripping increased to 30 per cent from a 22 per cent figure in 1951. The tendency to favor strip and open pit mining exists in other mining activities but their trends may not be as clearly outlined as the coal trend.

Yet another significant development is that the depth of overburden in coal properties considered suitable for stripping is increasing. Near the early part of the period a depth of 50–60 ft of overburden was considered to be approaching the top limit while today a depth of 80–100 ft is frequently taken into consideration for simple overcasting operations.

With these changes there have been continuing studies searching for procedures to economically treat with the deeper stripping problems. In nearly every instance, a sufficiently large machine that would accomplish the task by simple overcasting was indicated where the reserves were sufficiently large. The only known practicable way to evaluate a proposed stripping venture to high order accuracy is to construct studies including cost estimates, step by step. Nevertheless, a quick method for making appraisals for preliminary evaluations and orientation purposes has been in demand. Further, it is not unreasonable to expect that the demand for such evaluation methods will increase.

The foregoing circumstances seem not only to tend to enhance the demand for quick approximations but also to have conveniently produced appropriate experience data suggesting possible approaches to the problem.

Analysis Methods

The objective is to find a method of providing a preliminary simple overcasting analysis for a stripping prospect. The approach to the problem employs indicated trends in the relationship of the weight of the machine to its ability to do stripping work. The ability to do work is established through "MUF" numbers. Also, the approach employs situations where the geometry of each cut and spoil section is assumed to take certain defined relationships for varying overburden depths. Slopes are considered to be stable which means such practical factors as the mechanics of soils are neglected for the sake of convenience.

Assumptions

Pits that follow a straight line when projected on a horizontal plane are referred to as "straightaway". It is assumed that all sections pertain to straightaway cuts, which in turn means that areas can be treated relatively as volumes. Volumes of overburden are expressed in virgin cubic yards. When material is displaced from its virgin state to a spoil section it usually occupies a larger volume than *in situ*. The difference is called "swell" and it is expressed in percentages of the original volume. For example, if the original cut volume is denoted V yd^3 and the spoil volume is 1.2 V yd^3, then the "swell" is a positive 20 per cent.

Usefulness Factor Concept (Maximum Usefulness Factor or "MUF")

A value concept of a machine is arbitrarily established, consisting of the product of the nominal dipper size of the shovel times a functional dumping reach. A shovel's capability to negotiate cuts in deep overburden is limited by its ability to dispose of the spoil in most instances. Thus the dumping reach, along with its respective dumping height, is significant.

The angle of repose of the spoil material must be taken into consideration. It will vary in actual practice among different mines, jobs, and materials. However, a slope frequently found in practice and used in planning of 1.25/1 is used.

Shovels

The relationship of the shovel's geometry in a strip pit is shown diagrammatically in Fig. 7. To gain maximum advantage in constructing spoil piles the shovel is placed so that its tracks which are adjacent the spoil are as close as possible

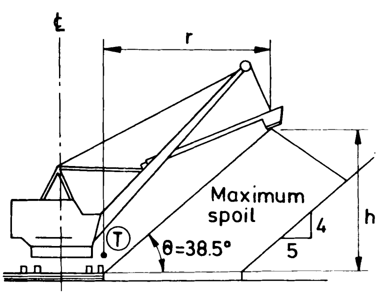


FIG. 7. Shovel reach diagram.[10]

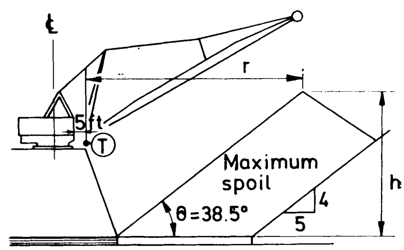


FIG. 8. Dragline reach diagram.[10]

TABLE 8. SUMMARY TABULATION SHOVEL DATA[10]

Shovel designation	Working weight (lb.)	MUFs (yd^3. ft)
a	1,785,000	2156
b	2,030,000	1740
c	3,070,000	3938
d	3,345,000	4350
e	4,950,000	6734
f	5,790,000	7630
g	6,000,000	8264
h	6,558,000	8580
i	7,700,000	9900
j	7,875,000	11,720
k	8,585,000	–
l	11,600,000	15,419
m	13,900,000	20,102

to the deposit rib. A vertical plane passed through the line of the rib of the deposit would contain the point designated as circle *T*. The dumping reach *r* is thereby established for each shovel analyzed. The MUFs for any one shovel is defined as the product of the nominal size of the dipper in cubic yards and the dumping reach in feet, shown as r. In other words the MUFs is equal to the load moment about point circle T in terms of cubic yards times feet. Letter designations are assigned the machines studied, and the results of the computations together with machine gross weights are tabulated in Table 8.

MUF vs. Gross Shovel Weight

The graph shown on the rectangular coordinate line chart, Fig. 9, depicts the trend where the ordinate represents the MUFs and the abscissa represents the gross machine weight (shovel). The result is significant in that the curve appears to follow

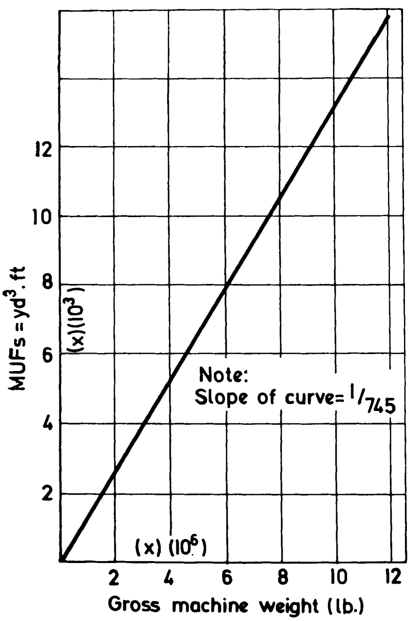


FIG. 9. Relationships of gross machine weight and "MUF" numbers.[10] Graph for shovels.

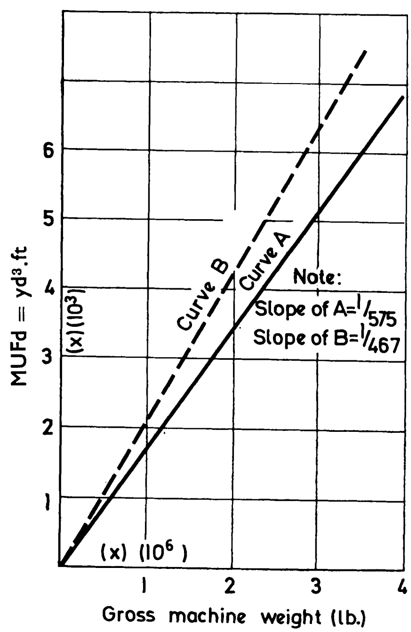


FIG. 10. Relationships of gross machine weight and "MUF" numbers. Graph for draglines.

generally a straight line regardless of manufacture or size range of machines. According to the curve, each MUFs unit requires 745 lb. gross weight in the shovel. In determining the curve, the data were first plotted on a large scale work graph. It was then observed that the points representing shovels actually working in the field were more faithful to the trend line (i.e. they had less "scatter") than shovels existing only on paper. Consequently, all points, both those representing existing and "paper" machines, are not weighed equally. The points for existing machines are given more "weight" than those for "paper" machines. For this reason the curve was not determined by strict use of the method of least squares as is the practice in certain statistical plotting.

All recent published specifications on U.S. manufactured stripping shovels and draglines available to the writer[10] have been analyzed in the manner described in the process of making this study.

TABLE 9. SUMMARY TABULATION DRAGLINE DATA[10]

Dragline designation	Working weight (lb.)	MUFd (yd^3. ft)
a	375,000	587
b	450,000	760
c	550,000	835
d	640,000	1240
e	695,000	1080
f	840,000	1370
g	1,274,000	2060
h	1,299,000	1970
i	1,460,000	3680
j	1,467,000	2560
k	1,600,000	2610
l	1,750,000	4330
m	1,900,000	3100
n	1,965,000	3440
o	2,460,000	4410
p	2,650,000	4130
q	2,930,000	5120
r	3,050,000	6240
s	3,175,000	6320
t	3,200,000	7110
u	3,335,000	6500
v	3,730,000	7640

Draglines

As with the shovel, the dragline's ability to negotiate cuts in deep overburden is limited by its ability to dispose of the spoil in most circumstances. The dragline normally works from the surface of the cut or a bench floor slightly below the surface. The machine's dumping height in this situation is important but it is not an influencing factor. For the type cut section and operation visualized, the dumping reach *r*, Fig. 8, is the controlling factor.

Walking draglines are mounted on circular bases called "tubs". Different models within any one manufacturer's line and variations among the manufacturers result in different ground to base bearing pressures. In order to have a standardized basis for comparison, the tub diameters are varied from actual to a hypothetical whereby bearing pressures are uniformly maintained at 10 psi.

With the hypothetical diameter and allowing 5 ft safety distance between the top edge of the old high wall and the near point of the tub a moment center (circle T) is established. Measuring from circle T to the dumping point gives a moment arm r for each dragline considered.

A figure of 4750 lb./nominal dragline bucket size is taken to represent the unit weight of bucket plus load contents. The specified suspended load of the machine is divided by 4750 to give the nominal capacity in cubic yards. This resulting bucket capacity times the arm r (distance in feet) gives the maximum usefulness factor for each stripping dragline. Thus, the MUFd, maximum usefulness factor for a dragline, is defined as the product of the nominal bucket size in cubic yards and the dumping reach r. It is equal to the load moment about circle T in terms of cubic yards times feet.

MUF vs. Gross Dragline Weight

Letter designations are also assigned the draglines studied. Tabulations of MUFd figures for the analyzed draglines are shown in Table 8. Only electric powered draglines are considered.

The graphs shown on the rectangular coordinate line chart, Fig 10, depict the trends where the ordinate represents the MUFd figures and the abscissa represents the Gross Machine Weight (dragline). There are two curves shown, each of which follows generally a straight line.

The solid line, curve A, represents the trend resulting from the specifications data of the dominant manufacturers' models which are in operation. As with the operating shovels' data, the data of existing dragline models when plotted are quite faithful to the line. The dash line, curve B, represents a more advantageous trend which could possibly become the significant one for newer and especially larger draglines rather than curve A. Curve B is not as well defined as curve A and is established more or less arbitrarily by taking into account recent specifications of newly announced larger and some uprated machines.

The slope of curve A figures 1/575 and the slope of curve B figures 1/467 which means that 575 lb. of machine weight are required for each MUFd in the former case and 467 lb. would be required similarly in the latter.

Pit Section — MUF Relationships

The foregoing demonstrates unexpectedly simple but at the same time logical relationships between the usefulness numbers (MUF) and the gross weights of the machines analyzed. To accomplish the stated objective it is necessary to next demonstrate a relationship between the pit section geometry and the required MUF numbers for varying depths of overburden. The demonstrations take into account both shovel and dragline type operations. The numbers would be related, in turn, to projected machine gross weights which, in effect, establishes relationships between stripping machinery mass and overburden volumes (or depths). Therefore, those mentioned in this section are the required numbers for the hypothetical sections being studied, whereas the ones determined in the preceding section are taken from current or recent machines for the purposes of defining trends.

Assumptions

In order to determine a relationship between the MUF numbers of the required machines for different depths of overburden hypothetical situations with a number of assumptions are established. From Fig. 11 "Shovel section", a generalized formula to

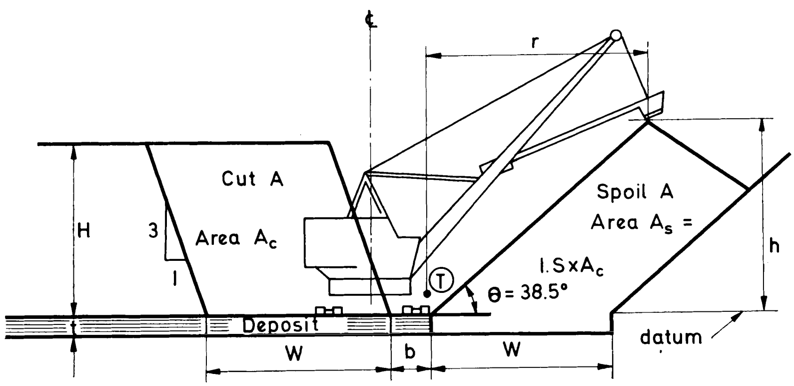


FIG. 11. Shovel cross-section.[10]

give dimension r in terms of H, t, and W is derived. First, assume the deposit is bituminous coal, of thickness t feet and the cut width of W feet. The spoil angle of repose is 38.5° (1.25 in 1 slope). The berm width (b) does not enter into the computation because it is assumed to be sufficiently small that the digging effectiveness of the shovel from its indicated position is not affected. A similar viewpoint is taken in assuming the high wall slope of 1 in 3. The derivation is shown on p. 426.

If

r = reach (as shown in Fig. 11) in feet,
S = swell in percentage,
H = depth of overburden in feet,
W = cut width in feet,
t = deposit thickness in feet,

the resulting formula is as follows:

$$r = [1.25]\times[(1+S/100)\,(H)-t+W/5] \tag{A}$$

In order to deal with varying dipper sizes "Estimated Shovel Output Table" is constructed. To facilitate computations, selected factors are assumed constants. The factors selected are those which can usually be closely approximated for a given prospect once the type machine is selected and a general operating approach is determined. For this example, it is assumed that regardless of shovel size the average cycle time will be maintained at 56 sec, the dipper factor will always be

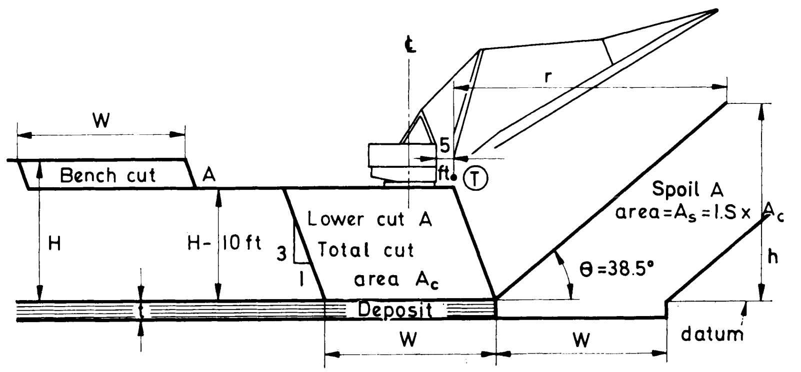


FIG. 12. Dragline cross-section.(10)

80 per cent and the monthly operating factor will be kept at 85 per cent. The construction in the table shows that in line with these assumptions the expected monthly output will be 31,200 D yd³, were D is the nominal dipper size in cubic yards.

ESTIMATED SHOVEL OUTPUT TABLE

Dipper size (yd³)	D
Dipper factor	80%
Dipper load, (yd³), bank measure	$0.8D$
Cycle time (sec)	56
Passes per hour	64
Theoretical hourly output (yd³) bank measure	$51D$
Scheduled monthly hours of operation	720
Theoretical monthly output (yd³), bank measure	$36{,}800D$
Monthly expected operating factor	85%
Expected actual output per month (yd³) bank measure	$31{,}200D$

Where Qc = the required net tons of cleaned coal required per month,
H = the overburden depth in feet, and
L = the yield in net tons of cleaned coal per acre,
then $D = (Qc)(1613)(H)/(L)(31{,}200)$ (B)

The required MUFs for the shovel at any overburden depth, H, is given by the product of equations (A) and (B). The following equation results, the derivation of which is shown on p. 427.

$$\text{MUFs} = r\cdot D$$
$$= \{[1.25]\times[(1+S/100)(H)-t+W/5]\}\times\{(Qc)(1613)(H)/(L)(31{,}200)\} \qquad \text{(C)}$$

From Fig. 12 "Dragline Cross-section" a generalized formula to give dimension r in terms of H, t, and W is derived on p. 427. First, it is assumed the high-wall slope will remain at a fixed figure of 1 in 3 and that 10 ft of surface will be removed to provide an operating bench. Then, the following are taken as variables: the deposit is bituminous coal of thickness t ft, and the cut width is of W ft. Angle θ of 38.5° represents the angle of spoil repose.

If

r = reach (as shown in Fig. 12) in feet,
S = swell in percentage,
H = depth of overburden in feet,
W = cut width in feet,
t = deposit thickness in feet,

the resulting formula is as follows:

$$r = [0.33H - 3.3] + [1.25] \times [(1 + S/100)(H) - t + W/5] \tag{D}$$

In order to deal with varying bucket sizes, "Estimated Dragline Output Table" is constructed in which a number of factors are also made constant. For example, it is assumed that regardless of size of dragline, the cycle time will be maintained at 58 sec, the bucket factor will always be 80 per cent, and the monthly operating factor will be kept at 85 per cent. In line with the method of computing and the assumptions, the expected monthly output will be 30,400 B yd³ where B is the nominal bucket size in cubic yards.

ESTIMATED DRAGLINE OUTPUT TABLE

Bucket size (yd³)	B
Bucket factor	80%
Bucket load (yd³), bank measure	$0.8B$
Cycle time (sec)	58
Passes per hour	62
Theoretical hourly output (yd³) bank measure	$49.5B$
Scheduled monthly hours of work	720
Theoretical monthly output (yd³) bank measure	$35{,}800B$
Monthly expected operating factor	85%
Expected actual monthly output (yd³) bank measure	$30{,}400B$

Where Qc = the net tons of cleaned coal per month,
H = the overburden depth in feet, and
L = the yield per acre in net tons of cleaned coal,
then $B = (Qc)(1613)(H)/(L)(30{,}400)$ (E)

For the derivation of equation (E) see p. 428.

The required MUFd for the dragline at any overburden depth, H, is given by the product of equations (D) and (E). The following equation results which is derived on p. 428.

$$\begin{aligned} \text{MUFd} &= r \cdot B \\ &= \{[0.33H - 3.33] + [1.25] \times [(1 + S/100)(H) - t + W/5]\} \\ &\quad \times \{(Qc)(1613)(H)/(L)(30{,}400)\} \end{aligned} \tag{F}$$

Illustrative Example

The employing of the developed information may be demonstrated in part by an example. The example selected is well within the range of the prepared data.

Take an imaginary coal strip mining prospect compatible with the output tables and the cut sections already constructed. Assume information is known about the property which permits establishing as constants certain of the variables in the equations which have been derived. Further assume the natural dispositions of the coal and overburden are such as to appear at the outset to provide approximately equal advantage to shovel and dragline. However, it is tentatively decided to first treat with the shovel type operations.

The 5 ft (t) coal seam is expected to yield 7500 (L) net tons of clean coal per acre. The spoil angle of repose remains 38.5° from horizontal. If mined with a dragline the cut widths (W) would be 80 ft and if with a shovel they would be 60 ft wide. The average production rate being considered is 83,500 (Qc) net tons of cleaned coal per month. The overburden depths (H) vary from 60 ft to 120 ft.

Since there are several other involved aspects of the problem, it is desired to know prior to more detailed study the approximate specifications of the shovels that would be required for different overburden depths. For comparison purposes it is desired to know the relative MUF requirements of the shovels and of the draglines for the same overburden depths. Furthermore, the swell is expected to be 20 per cent but it is desired to know what effect would result should the swell turn out to be something greater than the 20 per cent – say 25, 30, or even 35 per cent.

Appropriate conversions from variables to constants were made in the formulae and the digital computer was employed through the following programming:

Program A. Equation (A) which gives the reach requirement (r) for the shovel section was solved for different overburden depths (H) from 60 to 120 ft in increments of 2 ft in four separate groups:

First group, holds the swell (S) to 20 per cent.
Second group, holds the swell (S) to 25 per cent.
Third group, holds the swell (S) to 30 per cent.
Fourth group, holds the swell (S) to 35 per cent.

Program B. Equation (B) which gives the dipper size requirement (D) for the shovel section is solved for different overburden depths (H) from 60 to 120 ft in increments of 2 ft.

Program C. Equation (C) which gives the MUFs numbers [a product of equation (A) and equation (B)] for the shovel section is solved for different overburden depths (H) from 60 to 120 ft in increments of 2 ft in four separate groups:

First group, holds the swell (S) to 20 per cent.
Second group, holds the swell (S) to 25 per cent.
Third group, holds the swell (S) to 30 per cent.
Fourth group, holds the swell (S) to 35 per cent.

Program D. Equation (D) which gives the reach requirement, (r), for the dragline section for different overburden depths (H) from 60 to 120 ft in increments of 2 ft in four groups.

First group, holds the swell (S) to 20 per cent.
Second group, holds the swell (S) to 25 per cent.
Third group, holds the swell (S) to 30 per cent.
Fourth group, holds the swell (S) to 35 per cent.

Program E. Equation (E) which gives the bucket requirement (B), for the dragline section is solved for different overburden depths from 60 to 120 ft in increments of 2 ft.

Program F. Equation (F) which gives the MUFd numbers [a product of equations (D) and (E)] for the dragline section is solved for different overburden depths (H) from 60 to 120 ft in increments of 2 ft in four separate groups:

First group, holds the swell (S) to 20 per cent.
Second group, holds the swell (S) to 25 per cent.
Third group, holds the swell (S) to 30 per cent.
Fourth group, holds the swell (S) to 35 per cent.

Summary

The results of the computer work are not given because of their bulk. However, they are discussed and some are shown graphically.

Effect of Swell

The results indicate the percentage variation in required operating reach figures for different swell percentages are substantially uniform at all overburden depths for each machine. For example, at all intervals between 60 ft and 120 ft overburden the 25 per cent swell group shows 4 per cent more reach requirement for the shovel, the 30 per cent swell group shows 7.5 per cent more reach required, and the 35 per cent group shows 11 per cent more reach required than the 20 per cent group. The dragline results show 3.5 per cent more reach required for the 25 per cent group, 6.5 per cent more for the 30 per cent group, and 9.5 per cent more for the 35 per cent group than for the 20 per cent group.

Shovels vs. Draglines

Another significant finding is the uniform relationship between the MUFs curve and the MUFd curve shown in Fig. 13. The dragline usefulness numbers (MUFd) are approximately 1.25 times the shovel usefulness numbers (MUFs) at correspond-

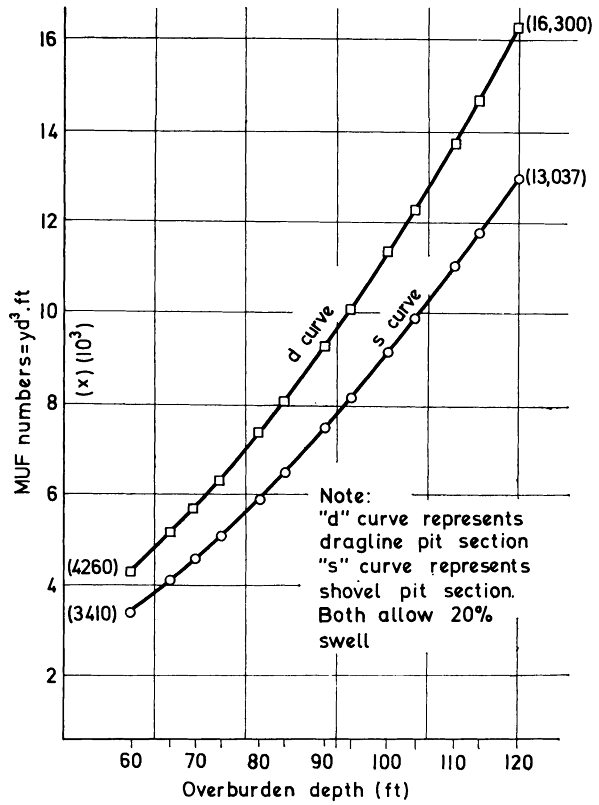


FIG. 13. Graph showing relative "MUF" requirements for shovel type and dragline type simple overcast excavation.[10]

ing overburden depths when each series is analyzed on the basis of 20 per cent swell. On the basis of weight comparisons alone, the dragline would have a slight advantage over the shovel. However, at this point in the analysis the dragline is eliminated from further consideration due to insufficient advantage indicated over the shovel and to other factors beyond the scope of the study.

Shovel Requirements as Affected by Overburden Depth

A set of curves designated H, $r \cdot D$ curves, Fig. 14, are next considered. They consist of three curves, each related to the overburden depth H, which is represented on the abscissa. The three ordinates represent separately the figures obtained from digital results for the indicated reach r, the indicated dipper size D, and the indicated gross machine weight. The latter are obtained by applying the 745 multiplier (see Fig. 9) to the MUFd digital results. At a glance the three requirements for the shovel can be found for any one overburden depth. For example at H equals 90 ft a 52 yd^3 dipper size is indicated. Also indicated are the operating reach r of 144 ft and the gross machine weight of 5,540,000 lb. At a unit price per pound for the

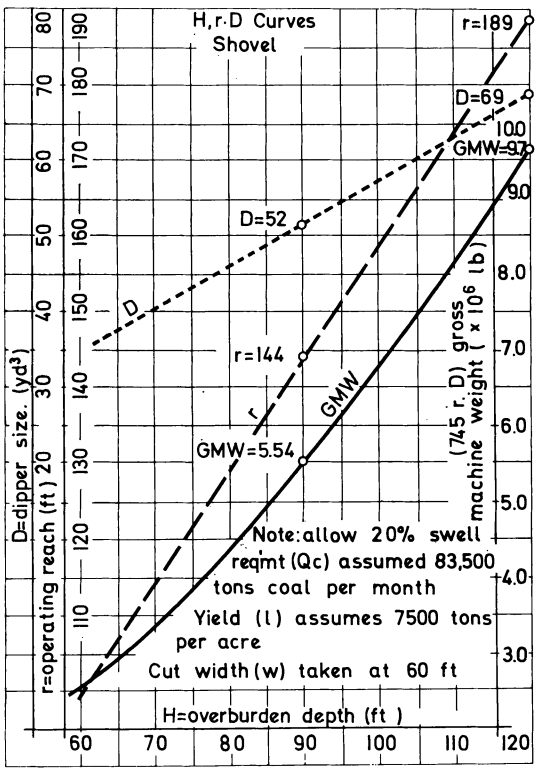


FIG. 14. Curves for shovel requirements as affected by overburden depth.[10]

machine of, say, 65 cents the price of the shovel would be approximately \$3,600,000. At 120 ft overburden depth the indicated dipper size D is 69 yd^3, the operating reach r is 189 ft and the gross machine weight is 9,700,000 lb. The approximate price at 65 per cent pound would be \$6,300,000. From these data, together with other pertinent aspects, an optimum overburden depth would be tentatively determined and the conventional type detailed stripping analysis including cost estimates undertaken.

Discussion and Conclusions

The true value of the demonstrated approach to preliminary stripping analysis cannot be determined at this time. Only experience and trial with real problems can give the answer. It must be emphasized the aim here is to demonstrate an approach for attacking such problems. The examples and the assumptions employed are to be considered only as the media through which the approach and procedure are demonstrated. They are deliberately selected slightly on the unrealistic side so as to focus attention on the approach and procedure.

It should be emphasized that this study does not take into account the many variables which are encountered in real stripping problems. The formulae derivations and the illustrative example require hypothetical situations. Actually the entire study is based upon situations and trends which are not necessarily fixed. While the trends seem quite clear there is no proof that they are absolute and they could conceivably change. Therefore, the study should be accepted with the understanding of the existence of this possible limitation.

Derivation of Formulae

Equation (A) – Derivation

Refer to Fig. 11.

Let t = thickness of deposit (ft)

W = width of cut (ft)

h = height of spoil above datum (ft)

r = reach as shown schematically in Fig. 11 measured from circle T (ft)

H = cut height (ft)

As = spoil area (ft^2)

Ac = cut area (ft^2)

S = swell (%)

$$As = (t)(W) + (h - W/25)(W) + (W/25)(W/2)$$
$$= (t)(W) + hW - W^2/5 \tag{1}$$
$$Ac = (W)(H) \tag{2}$$
$$As = (1 + S/100)(Ac) \tag{3}$$
$$= (1 + S/100)(W)(H)$$

Combining (1), (2), and (3)

$$(1 + S/100)(W)(H) = (t)(W) + (h)(W) - (W^2/5)$$
$$(1 + S/100)(H) = (t) + (h) - (W/5)$$
$$(h) = (1 + S/100)(H) - (t) + (W/5)$$

but $r = 1.25\,h$. Therefore,

$$r = [1.25] \times [(1 + S/100)(H) - (t) + (W/5)] \tag{A}$$

Equation (B) — Derivation

Refer to Estimated Shovel Output Table, p. 420.

Let Qc = the required net tons of cleaned objective (coal) required per month

L = the yield in net tons of cleaned objective (coal) per acre

H = cut height in feet

$4840 = \text{yd}^2/\text{acre}$

$(H/3)(4840) = 1613\,H = \text{yd}^3/\text{acre}$

$$(Qc/L)(1613H) = \text{yd}^3/\text{month required stripping}$$

$$\text{but } 31{,}200D = \text{yd}^3/\text{month stripping}$$

$$31{,}200D = (Qc/L)(1613)(H)$$

$$D = (Qc)(1613)(H)/(L)(31{,}200) \tag{B}$$

Equation (C) — Derivation

$$\text{MUFs} = r\cdot D \text{ by definition}$$

$$\text{MUFs} = \{[1.25]\times[(1+S/100)(H)-(t)+(W/5)]\} \times\{(Qc)(1613)(H)/(L)(31{,}200)\} \tag{C}$$

Equation (D) — Derivation

Refer to Fig. 12

Let t = thickness of deposit (coal) (ft)

W = width of cut (ft)

h = height of spoil above datum (ft)

r = reach as shown schematically in Fig. 12 (measured from circle T) (ft)

H = cut height (ft)

As = spoil area (ft^2)

Ac = cut area (ft^2)

S = swell, percentage

$$As = (W)(t)+(W)(h-W/2.5)+(W)(W/2.5)/2$$

$$= W+Wh-W^2/5 \tag{4}$$

$$Ac = (W)(H) \tag{5}$$

$$As = (1+S/100)(Ac) \tag{6}$$

Combining (4), (5), and (6)

$$(1+S/100)(WH) = Wt+Wh-W^2/5$$

$$(1+S/100)(H) = t+h-W/5$$

$$h = (1+S/100)(H)-t+W/5$$

but

$$r = (H-10)/3+(1.25)(h)$$

$$= \{0.33H-3.3\} + \{[1.25]\times[(1+S/100)(H)-t+W/5]\}$$

$$= [0.33H-3.3] + \{[(1.25)(1+S/100)(H)-1.25t+1.25\,W/5] \tag{D}$$

Equation (E) — Derivation

Refer to Estimated Dragline Output Table, p. 421.

Let Qc = the required net tons of cleaned objective (coal) required per month,

L = the yield in net tons of cleaned objective (coal) per acre

H = cut height (ft)

$4840 = \text{yd}^2/\text{acre}$

$(H/3)\ (4840) = 1613H = \text{yd}^3/\text{acre}$

$$(Qc/L)(1613H) = \text{yd}^3/\text{month required stripping}$$

but $30{,}400B = \text{yd}^3/\text{month stripping}$

$$30{,}400B = (Qc/L)\,(1613)\,(H)$$

$$B = (Qc)\,(1613)\,(H)/(L)\,(30{,}400) \qquad \text{(E)}$$

Equation (F) — Derivation

$$\text{MUFd} = \text{r}\cdot B \text{ by definition}$$

$$\text{MUFd} = \{[0.33H-3.3]+[(1.25)\,(1+S/100)\,(H)-1.25t+1.25\,W/5]\} \times\{(Qc)\,(1613)\,(H)/(L)\,(30{,}400)\} \qquad \text{(F)}$$

WHEEL EXCAVATORS

There are two types of wheel excavators of interest to the strip miner.[11] One is the large type developed for the brown coal industry in Germany; and the other is the type developed mostly in the coal strip mines of Fulton County, Ill., usually referred to as the Kolbe wheel excavator.

Both types of machines are electrically powered and are equipped with a digging wheel and an internal belt conveyor. The domestic machines are used only in overburden excavation while the German machines may be used either in overburden removal or in coal reclamation.

German Machines

The German-type wheel excavator is usually crawler mounted with the mountings arranged to give three-point suspension. These machines are designed so that there are large crawler areas in contact with the ground so that ground bearing pressures are usually kept to between 16 and 19 psi.

This enables the machines to operate on bench surfaces in the overburden where the supporting soil strength is low without the necessity of laying mats. It is reported that travel gear may account for as much as 30 per cent of the total weight of one of these machines.

To give a general idea of the spread in sizes of machines available it may be stated that one of the smaller machines has a service weight of about 55 tons and a theoretical output of about 260 yd^3/hr, and the digging boom has a range from 20 ft above to 2.5 ft below track level; one of the larger machines has a working weight of 5500 tons, a theoretical output of 13,000 yd^3/hr, and a digging range from 165 ft above to 65 ft below track level.[11]

Operation

Generally the overburden to be stripped from the brown coals of Germany is a uniform, easily-dug material such as sand or other friable material which can be dug easily and which will flow easily in the conveyor system.

The German strip mine usually involves removing thick overburden from a thick seam, or seams, of brown coal and the wheel excavator may be required to dig the overburden, or the coal.

In one method of operation the wheel excavator progresses along the bank removing material from ground surface to base in a wide strip. Crowding action may be provided by a boom equipped with an in and out thrust, or it may be accomplished through the travelling action.

The spoil area may be a nearby worked-out pit or it may be on the opposite side of the same pit being worked if conditions allow it.

Transportation of the spoil may be by highly organized rail haulage, or by a system of conveyor belts, or by a combination of the two systems. Conveyors are generally favored for the shorter hauls and rail haulage for the longer hauls.

Example of Installation

Figures 15 and 16 show a German wheel excavator installed in a lignite mine at Arjuzanx, France.[12] This installation will supply about 1 million tons of lignite a year to a 120,000 kW steam power station which was constructed to utilize the lignite in this deposit.

Lignite beds are from 5 ft to 20 ft thick and are situated under 50–100 ft of cover. It is estimated that 7,200,000 m^3 of overburden will have to be moved for every 1 million tons of coal mined.

The digging unit illustrated can deal with a maximum overburden thickness of 100 ft and can dig about 12 ft below the operating surface on which it sits. Maximum theoretical capacity is 2530 m^3/hr. The digging unit weighs 1400 tons and its power requirements are 1237 kW.

The spoil disposal machine illustrated also weighs 1400 tons and requires 1226 kW. It is equipped with a boom 110 m in length. The total distance from the centerline of the digging wheel to the end of the spoil disposal boom is about 750 ft.

After the lignite is exposed it is excavated and loaded onto conveyor belts by two bucket chain excavators, each of which has a capacity of 400 tons/hr.

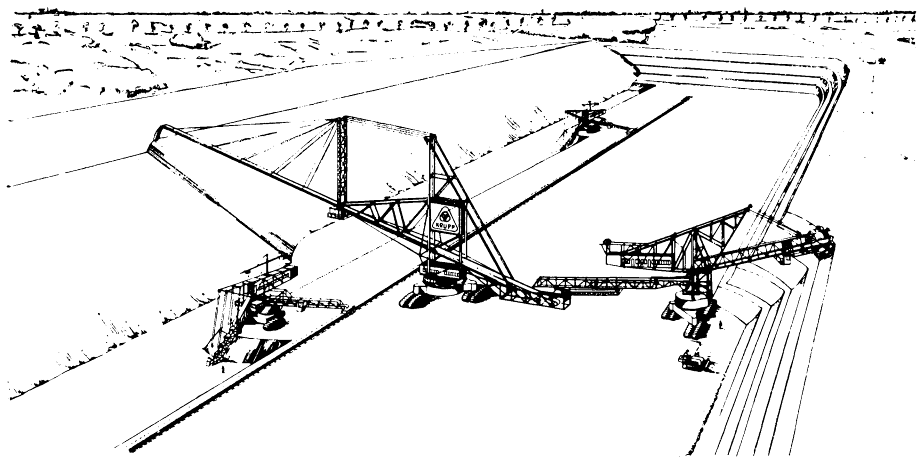


FIG. 15. Scheme of operation – bucket and wheel excavators at lignite mine, Arjuzanx, France.[12]

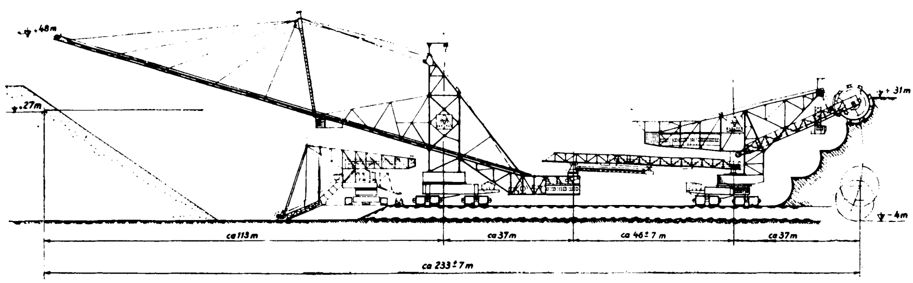


FIG. 16. Wheel and bucket excavators work together, Arjuzanx, France.

U.S. Wheel Excavators

The type of machine developed in the United States also has a wheel digging device and an internal conveyor system.

All existing U.S. wheels are mounted on stripping-shovel bases with four-corner support. A hydraulic jack at each corner allows the machine to be leveled.[11] Average bearing pressures exerted by the crawlers on the ground surface are about 45 psi since these machines operate from the coal surface.

The digging wheel and the stacker boom cannot be independent in these machines and as the digging wheel swings through the arc of its cut the stacker boom also swings through this same arc.

Digging wheels have been developed to handle the variety of overburden material which is found in Fulton County, Ill. This material grades from sticky clays containing some floating boulders, to shales of varying degrees of hardness and varying composition. In addition the wheels must also be able to dig several inches of frost which forms during winter operations.

Domestic wheel units are designed to be employed in tandem stripping operations. At present each one is working with a stripping shovel. The pit development and cut widths follow the orthodox pattern of prairie type shovel stripping operations. To allow the shovel and wheel excavator to pass at will pit widths are about 110 ft.[11]

Operation

The wheel excavator precedes the shovel and takes the top portion of the cut, leaving the lower for the shovel. The shovel digs the lower, which is usually material which has been blasted, and builds a spoil pile immediately adjacent to the coal

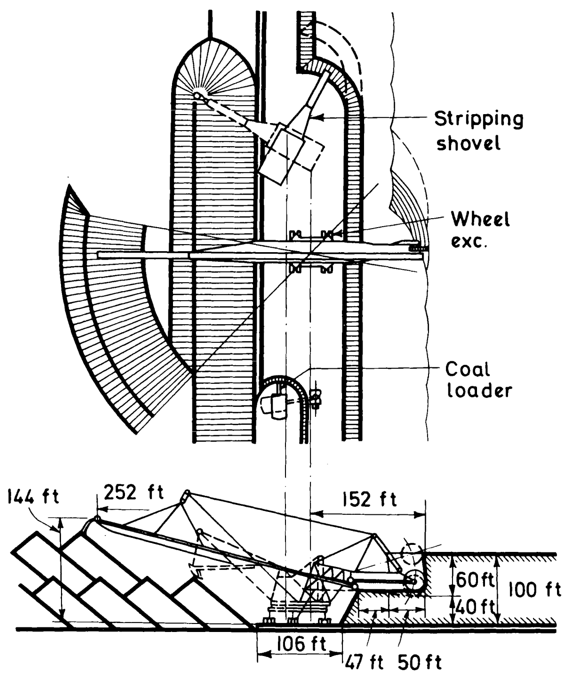


FIG. 17. Pit plan for wheel excavator puts the shovel in the lead, digging rock bench left by the wheel in the previous pass.[2]

rib. When taking the top of the succeeding cut the wheel places its spoil well back in heaps which resemble in plan a series of overlapping crescents. It should be noted that although this is a tandem operation each machine handles a portion of the spoil and that no spoil is handled more than once.

Figure 17 shows a typical shovel and wheel excavator stripping operation.

At one operation in Illinois a wheel strips and spoils 10,000 yd³ per shift, moving the top 25–30 ft and leaving the bottom 30–35 ft for a 33-yd shovel. Cuts taken by both machines are 45 ft wide, but the wheel excavator cut is offset from 10 to 15 ft from the shovel cut to provide a bench that prevents spillage of unconsolidated top material onto the uncovered coal.[2]

MISCELLANEOUS STRIPPING EQUIPMENT

Bulldozer-shovel Stripping[2]

A possible combination for stripping up to 35 ft of softer material is the small shovel and the bulldozer. With this type of setup the bulldozer works across the outcrop and takes off 10–12 ft of loose material – sometimes up to 20 ft. The shovel is used to remove the more solid material down to the top of the coal.

At a Pennsylvania mine operating in 36-in. coal a bulldozer with a hydraulic tilt blade takes the initial cut to within a few feet of the solid sandstone. A 6-yd diesel powered dragline handles the remainder.

A bulldozer plays an important role at an Ohio mine recovering 30-in. coal under 60 ft of shale and sandstone. The unit cuts down 12 ft of shale and makes a level bench for a 5-yd shovel which removes the remaining overburden. After stripping is completed to a 60-ft high-wall, the bulldozer levels spoil for seeding.

The shovel–bulldozer setup is not designed for high output but can be used effectively where cover is relatively soft and a large capital expenditure is not feasible.

Bulldozer Stripping

Where stripping is done by bulldozers alone a minimum of two should work together. For efficient materials handling an average of not more than 35 ft of cover should be moved and the terrain should be gently rolling or hilly to permit easier movement of overburden.

After the initial cut is made the material should be pushed at right angles away from the outcrop, and the bulldozers should work together, one following the other, and slightly overlapping the path of the leading unit to pick up side spillage.

After the pit is filled sufficiently the bulldozers should start pushing to the main spoil area away from the high-wall.

Stripping Anthracite

In the anthracite districts of Pennsylvania coal seams which are accessible from the surface usually pitch at fairly steep angles. This means that stripping pits are usually relatively narrow and deep, and that overburden cannot be gotten rid of merely by casting operations. Consequently overburden must be hauled and dumped outside the pit area.

FIG. 17A. Bulldozer with bucket cleaning bottom. (International Harvester Co.)

Figure 18[13] shows the plan being followed for the stripping and mining of thick anthracite veins. Overburden is removed in 40-ft benches; excavation is by $6\frac{1}{2}$ yd^3 shovels and haulage by 22-ton trucks.

The pit will eventually reach a depth of 750 ft.

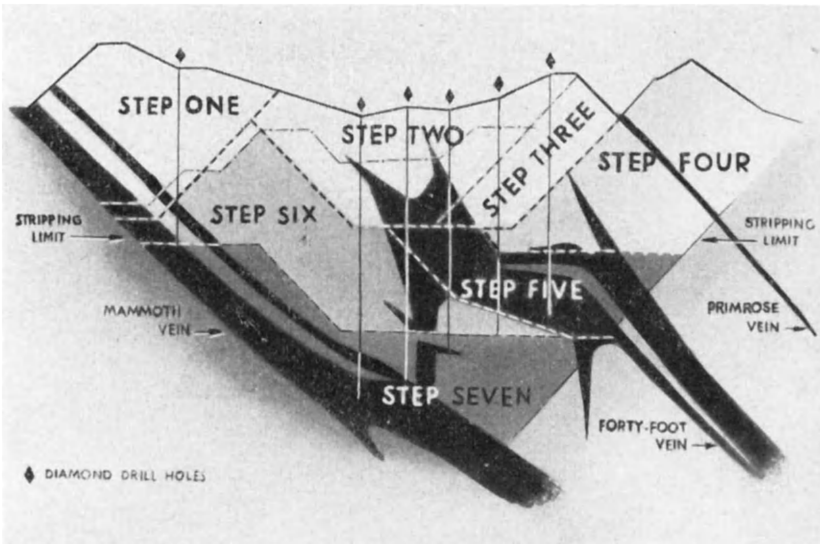


FIG. 18. At this anthracite operation overburden is removed and coal is recovered in seven planned steps. The completion of Step 7 is at a depth of 750 ft below the original surface.[13]

Loading Coal from the Seam

After the coal seam has been stripped and cleaned the coal is loaded onto trucks, or onto tractor–trailer units for transportation to the cleaning plant. Shovels with dipper capacities of 1–7 yd^3 are commonly used for digging the coal from the seam and loading it onto the haulage units.

If the coal seam is hard and unbroken it may be necessary to loosen it by ripping with a bulldozer so that it can be loaded.

Special equipment may be needed to clean the final layer of shale or clay from the top of the coal seam before it is loaded. Scoop loaders, backhoes, or power sweepers may aid in the final cleaning of the coal seam; the first two pieces of equipment have been used in recovering thin layers of coal which may stick to the bottom after the rest of the seam has been loaded.

Scraper Stripping

Rubber-tired tractor–scraper units are used as auxiliary equipment when it is necessary to move a portion of the overburden for a distance of several hundred feet. They find application in moving the loose soil and the relatively soft upper portion of the overburden.

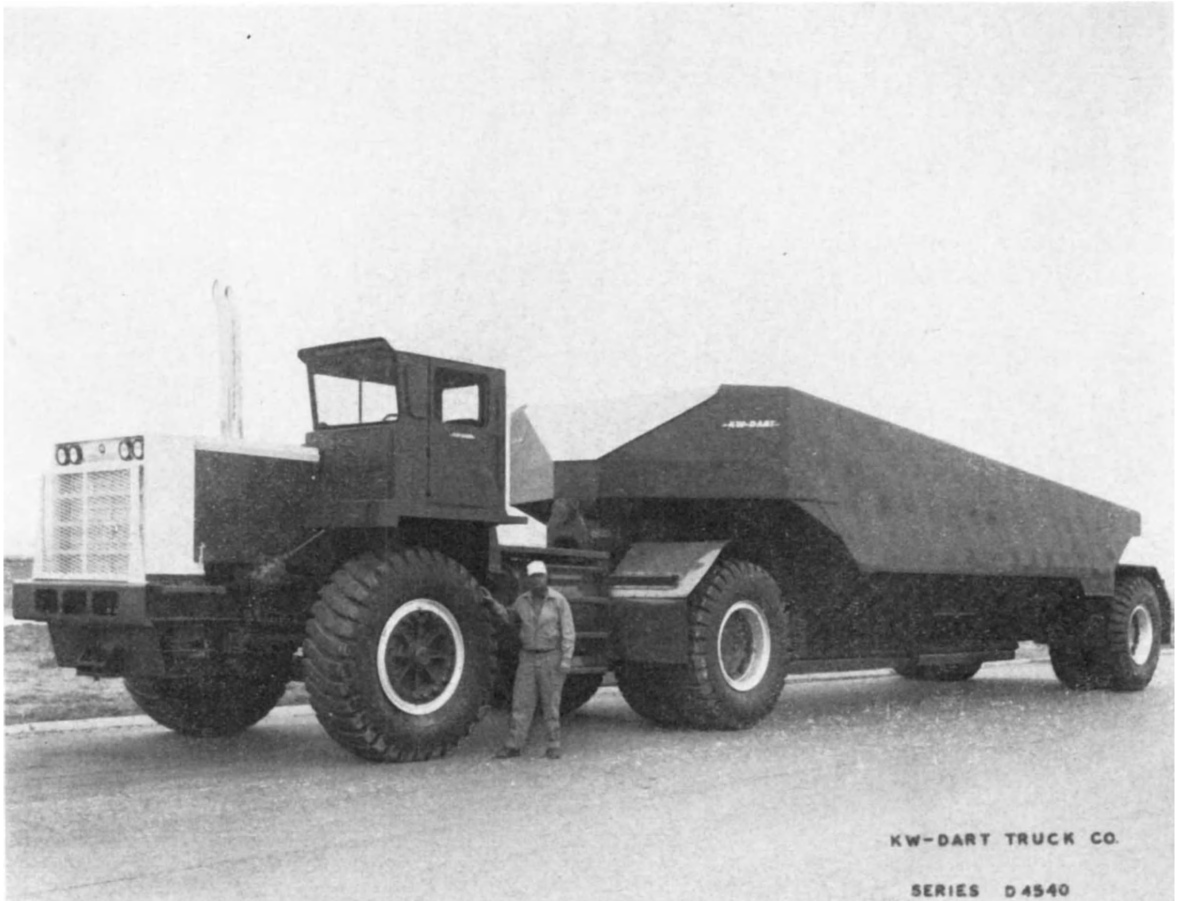


FIG. 19. 100-ton bottom-dump haulage unit. (KW-Dart Truck Co.)

Scrapers, push-loaded by big bulldozers, slice off overburden in 25-yd^3 bites at an eastern Ohio operation. Six units remove 1800–2000 yd^3/hr while working banks 50–60 ft high. Ripper equipped tractors precede the scrapers to break the friable sandstone for the scrapers.[2]

At another operation five tractor–scrapers aided by a rooter have removed 30–35 ft of cover working to a 75-ft high-wall and producing 1700 tons/day of coal. Shovels remove the lower portion of the overburden.

HAULAGE EQUIPMENT

Sizes of Haulage Units

The size of haulage units used in strip mining ranges from that of a small 2-ton standard dump truck up to the 110-ton tractor trailer units used in some of the large strip mines. A unit in common use in larger operations is the diesel tractor pulling a bottom-dump semi-trailer of 40–60 tons capacity.

The size of a haulage unit should be matched to the capacity of the coal loading shovel. For example a 5- to 7-yd shovel works well with a 40-ton truck and a 3- to 4-yd shovel teams well with a 25-ton hauler. A good rule of thumb is to use trucks with four to five times the dipper capacity of the loading shovel.[2]

In general a 20-ton haulage unit may be equipped with an engine of about 200 h.p., a 40-ton unit with an engine of 300 h.p., a 50-ton unit with 400 h.p., and a 70-ton unit with about 450 h.p. Larger units with capacities in the 90- to 100-ton range may be equipped with engines of up to 700 h.p.

When engine sizes exceed 600–700 h.p. mechanical problems in transmitting this power to the drive wheels increase rapidly with engine size. For applications where haulage grades are steep, as in some open pit metal mining operations, haulage units have been built with an individual electric motor drive on each wheel. This arrangement eliminates the mechanical problems with transmission and differential.

Economics of Haulage Units

In 1947 Central Ohio Coal Co. purchased 20-ton bottom-dump coal haulers for operation in the Ohio No. 6 coal field which weighed 36,000 lb. empty and were equipped with 200 h.p. diesel engines. These units were equipped with 21:00 by 24:00 by 20-ply single tires on the tractor drive and on the trailer. Over a 14-year period of operation the average tire cost per ton-mile was \$0.009 and the average total operating cost was \$0.053.

In 1955 the company purchased 40-ton units which were equipped with 300-h.p. engines. The company later purchased 40-ton units[14] which were equipped with 300-h.p. engines, 50-ton units equipped with 400-h.p. engines, and 70-ton units equipped with 450-h.p. engines.

After 1953 as the units with capacities larger than 20 tons began to be put into service it was noted that labor and maintenance costs decreased as did fuel oil costs but that tire costs increased.

Figure 20 shows in graphical form how the cost per ton-mile decreased from a peak of 5.5 cents per ton-mile in 1953 to 4.7 cents per ton mile in 1960 as the size of coal haulers and the quantity of coal hauled increased.

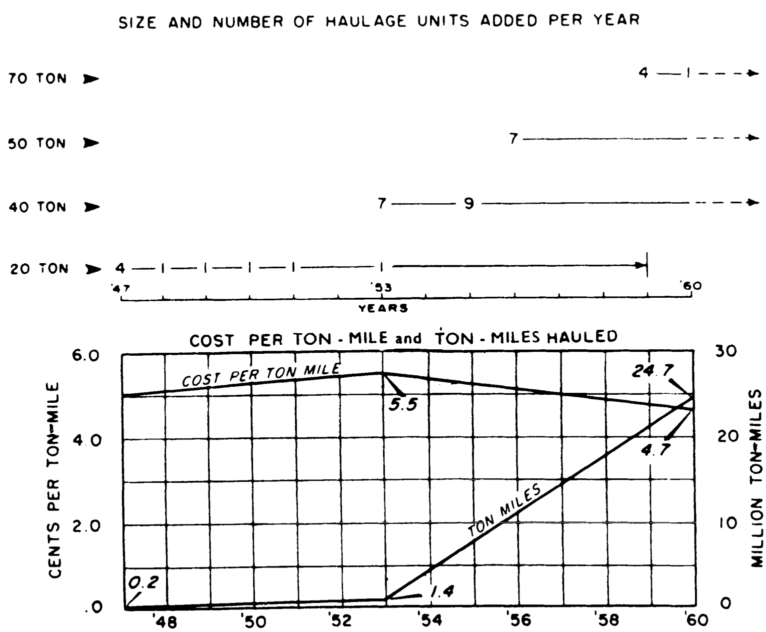


FIG. 20. Sizes of haulage units ton-miles hauled vs. cost per ton-mile.[14]

Another coal company has in service tractor–trailer haulage units of 40, 50, and 90 tons capacity. The 90-ton units are powered by 700 h.p. engines which operate at speeds of 50–55 mph empty and 45–47 mph loaded on level main road sections.

The 90-ton units average 1.36 mpg (miles per gallon) of fuel as compared with 1.77 mpg for the 50-ton units and 1.78 mpg for the 40-ton units.

Overall costs per ton-mile, including road maintenance, tractor repair and maintenance, trailer repair and maintenance, lubrication, fuel and tire costs, but excluding depreciation were 47 per cent less for the 90-ton units as compared with the 40- and 50-ton units.

Roads

It is apparent that the roads constructed for trucks hauling from 60 to 100 tons of coal at speeds from 30 to 40 miles per hour must be very well constructed and maintained in order to avoid disintegration of the road surface and excessive wear on haulage units.

Portions of such roads are relocated at frequent intervals as the pit face advances. The following features are common to well constructed haulage roads.[1]

(1) Sub-base is well prepared by removing top soil on original ground and compacting all filled areas.

(2) Roads are made extra wide to prevent single-plane rutting.

(3) Side ditches are made extra deep to remove all surface water and underground seepage.

(4) All adverse grades are broken at minimum distances of 1000 ft with level stretches 500–1000 ft in length being provided.

(5) All curves are laid out as long radius curves.

(6) Base material is made from 1 to 3 ft thick, depending upon weight of haulage units.

(7) Large size base material is used, usually broken slate, limestone, or sandstone, as available.

(8) The material for the road surface should be graded material containing fines to permit packing and to prevent water absorption.

(9) The surface cross-section should be arched to permit immediate surface water runoff.

COAL AUGERS

Augering has grown to the point where more than 8 million tons of coal a year is produced by this means.[2]

Auger-mining of coal from the surface can be divided into two types of operation.[15]

(1) Augering of high-walls prepared, or left, by stripping operations. These augering operations do not usually require any extensive pit cleanup and serviceable haulage roads are already in existence. In addition the pit is usually wide enough to accommodate any of the standard augering units available.

When teamed with stripping equipment augers permit higher banks to be stripped because the combined cost of auger coal and strip coal can be made to equal or better the cost when strip mining alone is done under thinner cover.

(2) Augering of outcrop coal usually requires the preparation of working benches and roads. These preparations for augering will probably add 10 to 15 cents a ton to the overall mining cost, depending upon the seam thickness. Width of benches required will vary from 20 to 100 ft depending on the augering equipment and the haulage methods used.

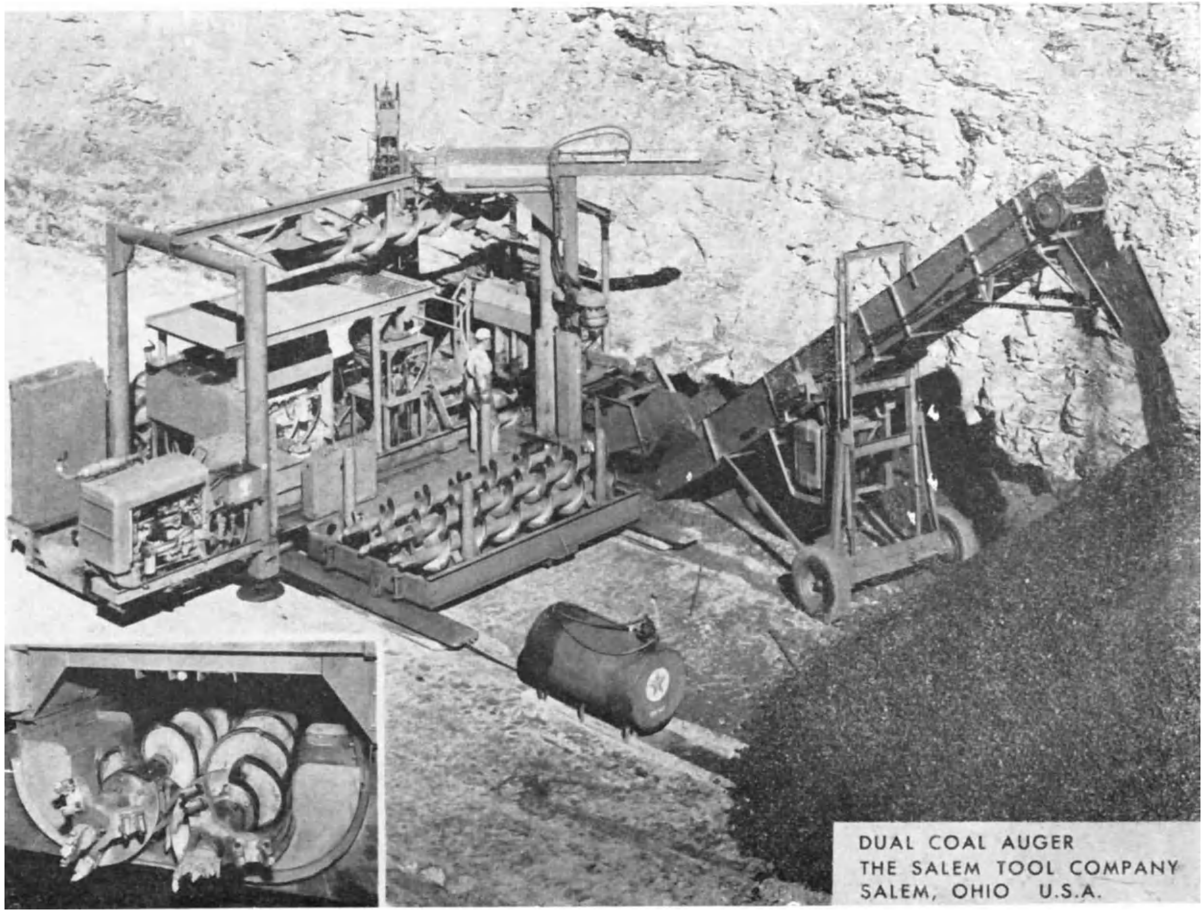


FIG. 21. Dual coal auger. (Salem Tool Company.)

Characteristics of Coal Seam

To be susceptible to profitable augering a coal seam should lie relatively flat or should have a constant pitch and be free from rolls and faults. The seam should not contain wide or excessively hard horizontal bands of sandstone or other abrasive material which will cause excessive wear and breakage of bits, pilots, and heads.

Augering Equipment

Augers are available in diameters from 16 to 84 in. and the larger ones are capable of producing coal at the rate of about 25 tons/min.

In the United States augers are available from three manufacturers:

Salem Tool Co., Salem, Ohio

McCarthy coal recovery machines are available in a complete line of single and double auger units ranging in auger size diameters from 18 to 48 in. and in flight lengths from 6 to 12 ft depending upon the requirements.

These machines are self propelled, reducing the capital outlay required, and are operated by a two-man crew for the single-auger machine and a three-man crew for the dual auger units.

The dual auger units are designed so that the augers rotate in opposite directions, thereby eliminating the common fault of one cutting head "climbing over" the other. This feature increases the depth of hole which it is practical to bore into the seam.

Long-Airdox Co.

Cardox auger miners are produced in three models. Model 65 is a lightweight unit for thin seams up to 24 in. thick. Model 155 has 6-ft auger sections capable of drilling 42-in. holes, while Model 235 has 12-ft auger sections and diameters up to 48 in.

These units are self-moving and self-positioning, capable of drilling holes up to depths of 120 ft or more. Models 65 and 155 can be operated on narrow benches since they are less than 16 ft in length.

Joy Manufacturing Co., Compton Division

Compton auger units have been manufactured since 1950 and the company produces single, double, and triple auger units ranging in size from 18 to 84 in. and with flight lengths from 12 to 40 ft.

The 7-ft diameter auger is powered with a 700 h.p. diesel engine and a 160-h.p. auxiliary engine. The auger sections are 30 ft in length and weigh 5 tons. The auger operates on a 100-ft bench and drills straight holes to a depth of approximately 180 ft.

Preparing for Augering

When a coal seam is to be augered in conjunction with a stripping operation care should be taken in blasting so that the high-wall will be left in the best possible condition, and loose material should be cleaned down so that crews will not be endangered by slides or falling rock.

The pit should be left clean for a width sufficient to accommodate the augering equipment. Augering operations should be carried out as soon as possible after stripping is completed before the high-wall begins to weather and to slough. If augering is to be carried out on an old high-wall then a bulldozer and/or a shovel may be needed to clean up the site for augering.

Augering Operations

The facing up operation and bench preparation should be done in such a manner that the augering machine will have solid footing and the bench area will have natural drainage.

The site for each auger should be selected so that as much as possible of the seam height can be removed from one hole. However, if the seam thickness is such that one auger cannot recover most of the coal, a second hole, above the first, can be drilled.

Pillars of 4–12 in. should be left between holes. The pillar thickness should never be less than 4 in. and a rule of thumb is to leave $1\frac{1}{2}$ in. of pillar for every 10 in. of auger diameter.[15] It may be necessary to leave a 3 or 4 ft wide pillar every fifty holes to prevent a general subsidence of the roof and resultant squeezing action on the augers.

Crew Requirements and Productivity

Crew requirements vary, depending on the type of equipment.[15] McCarthy auger units which are self-moving are usually operated with a two-man crew for the single auger units and three-man crew for the dual auger units. Compton auger units require a three-man crew for the single auger units and a four-man crew for the multiple auger units. These units require a bulldozer to move the auger from hole to hole. Some Compton units have been equipped with skids. Cardox auger units require two or three men depending on the model.

Figure 22 is a chart from which the amount of coal which an auger will produce in a shift may be estimated.

In addition to the auger crew a bulldozer operator or shovel operator may be required for preparing the face and the site for augering operations. In addition trucks and drivers will be required to transport the mined coal to the cleaning plant.

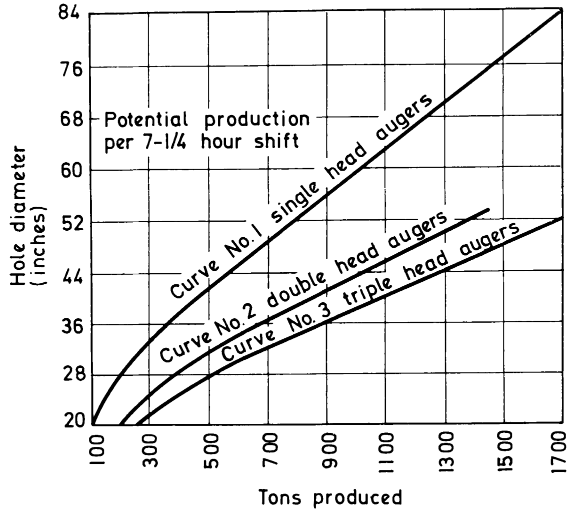


FIG. 22. Auger production vs. auger diameter.

Production per $7\frac{1}{4}$-hr shift may be estimated from the chart. Two-man crews are required for the self-moving single auger units; three-man and four-man crews are required for the multiple auger machines.[15]

Examples of Auger Installations[2]

An Ohio strip mine operator supplements his operations with high-wall augering. A 30-in. auger unit operated by a four-man crew adds 250–300 tons/day to the mine output. The augers penetrate to a depth of 200–220 ft.

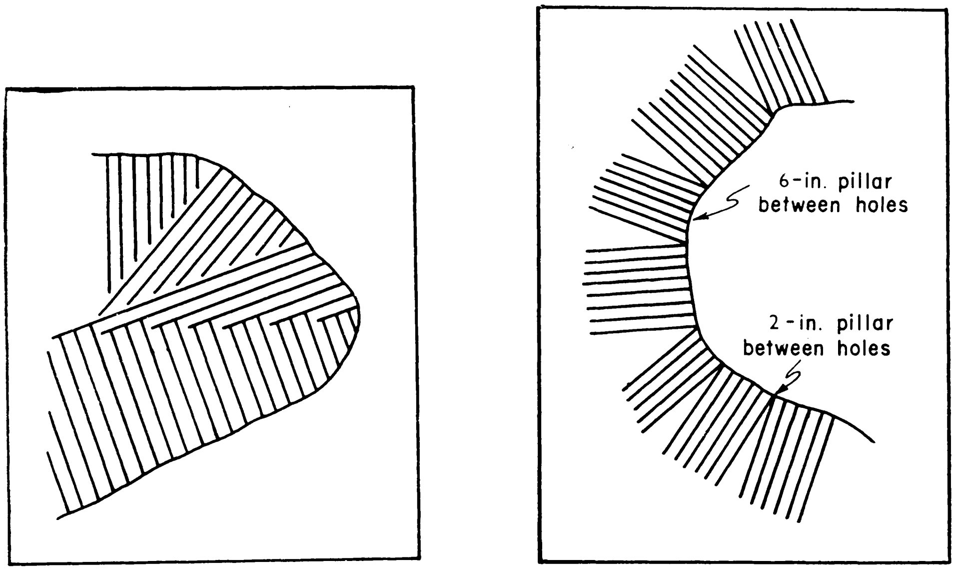
A crew of four men operates a 30-in. self-moving auger in an Alabama pit to produce 140–150 tons of coal per shift. The auger penetrates to a depth of 125 ft. One of the four-man crew drives loaded trucks to the preparation plant; one truck serves as a surge bin while the driver makes a round trip to the preparation plant with the other.

A self-moving 47-in. unit works in a mountain-top pit only 26 ft wide. Three augers, two full time and one spare, are used at this property to recover coal that could not be stripped because of the hazard of spoil rolling or sliding downhill and damaging mine installations.

In Ohio a 30-in. self-moving auger produces 400–500 tons/day in a three-shift operation. Working in areas previously deep mined, the unit has produced up to 275 tons in a single shift.

REMOTELY CONTROLLED MINING SYSTEMS

The first remotely controlled mining machine was developed by the Union Carbide Corp. in 1946 in connection with experiments with underground coal gasification.[17] Two machines were built to make 36-in. diameter air passages in the seam, the second machine being remotely controlled. Although this company's experiments with underground gasification were discontinued the remotely controlled tunneling machine worked so well that it was decided to build a larger model.



Blocking on outside turns allows for maximum extraction

Sets of parallel holes are used on inside turns

FIG. 23. Patterns for auger holes on turns.[16]

The major problems to be solved were in the design of a device to indicate to the operator the position of the machine with respect to the top and bottom of the seam and devices to keep the machine level crosswise, to avoid spiralling, and to keep the holes straight, parallel and at the proper spacing.

The new machine went into service in October, 1949. It bored a roughly oval hole, 38 in. high and 116 in. wide. A second pass, below the first, recovered the coal in seams of greater height than 38 in. The cutting heads consisted of four overlapping cutting heads driven at 60 rpm through a speed reducer by two 60 h.p. motors in parallel. The cusps between the holes were removed by fixed blades at the top and the bottom to produce an even floor and roof.

The machine was propelled on crawler tracks powered by a $7\frac{1}{2}$ h.p. variable speed d.c. motor.

Horizontal steering was accomplished by moving a pair of guide shoes which bore against the sides of the hole.

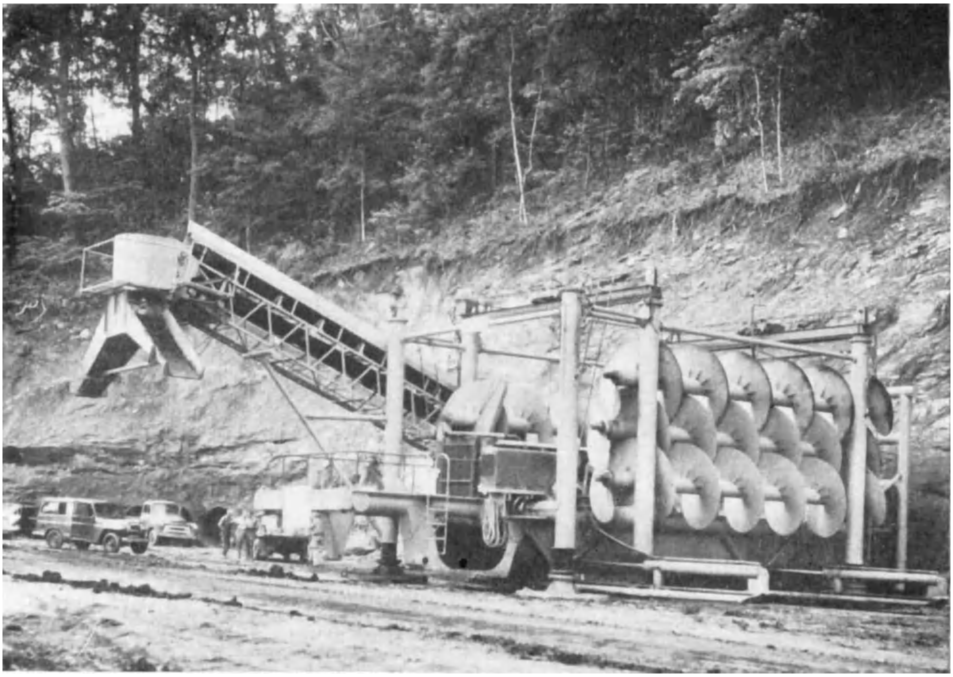


FIG. 23A. Joy 84-in. auger. (Joy Manufacturing Co.)

A train of portable belt conveyors, each 30 ft long, and mounted on two rubber-tired wheels, was used to transport the coal from the miner to the surface. As the machine advanced it pulled the train of conveyors after it.

Several devices for obtaining the information needed by the operator in order to steer the machine were provided. A "stratascope", or device for measuring the pressure on a cutter bit was mounted on the two outer cutting heads. The pressure changed as the bit cut through the various layers in the seam. An electrical signal from each stratascope was shown on an oscilloscope screen, in the form of an irregular circular trace. The operator soon learned from the shape and location of these irregularities whether the seam was rising or dipping, and steered the machine to closely follow the top.

A pair of pendulums sent out electrical signals to show the inclination of the machine both fore and aft, and crosswise.

An electric drill at the side of the machine was used to measure rib thickness. Each time the machine was stopped to add a conveyor this drill was actuated. It drilled through the rib in about 40 sec and indicated to the operator the distance at which it broke through into the adjacent hole.

Maximum depth of hole bored with this miner was 690 ft, limited by the number of portable conveyors available. It was normally operated at a speed of 2 ft/min, corresponding to a production of about $2\frac{1}{2}$ tons/min.

FIG. 24. Joy "Pushbutton" Miner. (Joy Manufacturing Co.)

The normal operating crew was three men, consisting of a machine operator, portable crane operator, and helper.

This experimental machine, while not a commercial success, pointed the way to the development of a practical system for mining by remote control.

A new machine was built and went into operation in 1953. This machine bores the same size hole but is heavier and more powerful than the previous model. The cutter heads are driven by a single 200-h.p. motor at a speed of 60 rpm. A rotary drum type cutter is used to remove the roof cusps.

A major improvement was made in the method of conveying coal from the machine to the surface. Instead of stopping to add sections of belt conveyors as needed the conveyors are assembled into a continuous train, 800 ft long, which follows the machine without stopping. The train extends along the high-wall when not in the bore hole. Another improvement was made in the conveyors themselves. Instead of 24-in. wide belt conveyors these are 60-in. wide chain and flight conveyors. "Minor" roof falls up to 6 ft wide, 8–10 in. thick, and 100 ft long are caught by these wide conveyors and carried to the surface without incident. This eliminates the need for men to go underground to clean up such falls.

Instrumentation on this machine was refined and made more rugged and reliable than that on the older model. Automatic controllers maintain the machine on a predetermined course at the desired angle to the horizontal, correct automatically for "spiral" and steer the machine to bore an absolutely straight hole.

The machine operator sits at a desk watching the oscilloscopes which tell him where the machine is with respect to the top and bottom of the seam. In addition another operator is required to observe the action of the conveyor train. This two-man crew can mine and load into trucks an average of more than 400 tons of coal per shift, working in a 44–48 in. seam.

An improved model of this machine, known as the Pushbutton Miner, and built by the Joy Manufacturing Co., was placed in trial operation in 1961.[18]

The boring unit and conveyors are essentially as in the previous model but the penetration depth has been increased to 1000 ft and as the conveyors retract they are stored in a circular ramp structure or Heli-Track. This is a mobile crawler-mounted structure 45 ft high, 77 ft long, and 48 ft wide which weighs more than 600 tons which, in addition to storing the conveyors and borer, houses the control room from which it is operated, and a repair shop.

The boring unit removes the coal for a width of 9 ft 9 in. Pillars between adjacent holes are between 2 and 4 ft wide, depending upon strata conditions.

Production with the Pushbutton Miner has been $6\frac{1}{2}$ tons/min. in a 48-in. seam, the machine advancing at the rate of 4 ft/min.

LAND RECLAMATION

Planning Reclamation[2]

The two principal problems involved in the reclamation of an area which has been strip mined are:

(1) Restoring the surface to usefulness after mining is completed.

(2) Preventing pollution of streams by acid runoff waters.

The type of restoration work depends upon such factors as contour of the land, type of overburden, and the State laws regulating restoration.

In some cases the spoil will not support plant life until it has been decomposed by weathering for several years. In such case the only immediate steps which can be taken to restore the land are to backfill and level it. In any event the material in spoil banks should be analyzed to see whether it will support plant life before any planting is done.

Sourness of soil acidity is probably the biggest problem in land reclamation. One method of combatting an acid soil condition is to work only the top layer of the spoil in preparing a seedbed.

Where grass or legumes are to be planted, the area should be prepared by harrowing, rather than by plowing. This technique avoids bringing the acid soil to the surface. The hard, compact, and dry soil structure left after levelling with a bulldozer is a further handicap.

For most spoil areas tree planting is recommended. Where the spoil is too steep for planting tree seedlings, it may be seeded, preferably with Black Locust and *Sericea lespedeza*.

Among the seedlings, Locust, Shortleaf Pine, Scotch Pine, and White Pine have shown the most promise, with Black Locust making the most rapid growth.

Autumn olive appears to be a most promising shrub, coupling fast growth with heavy production of fruit for wildlife.

Contour furrowing for tree planting has been practiced in some areas of West Virginia. Such furrowing results in the collection and retention of moisture which stimulates faster growth; but if the soil is too loose it can result in excessive build-up around the seedlings. As a consequence, they may wind up too deep in the ground.

Crown vetch has proved to be especially valuable on slopes for erosion control and is also an excellent forage crop for cattle.

Strip Mining Laws

A new strip mining law was enacted by the Pennsylvania State legislature and became effective on January 1, 1962.(19) The law contained the following provisions pertaining to the bituminous coal fields:

On "productive land" and in built-up areas, strip pits must be backfilled at a slant of 45° from the bottom of the excavation to the high-wall of the cut. "Productive land" is defined as that which has produced farm crops within the five previous years.

When pits extend to highways, public buildings, churches, or community buildings, backfills must be restored to the original contour within 100 ft.

For any remaining distance up to 750 ft, there must be a 45° backfill.

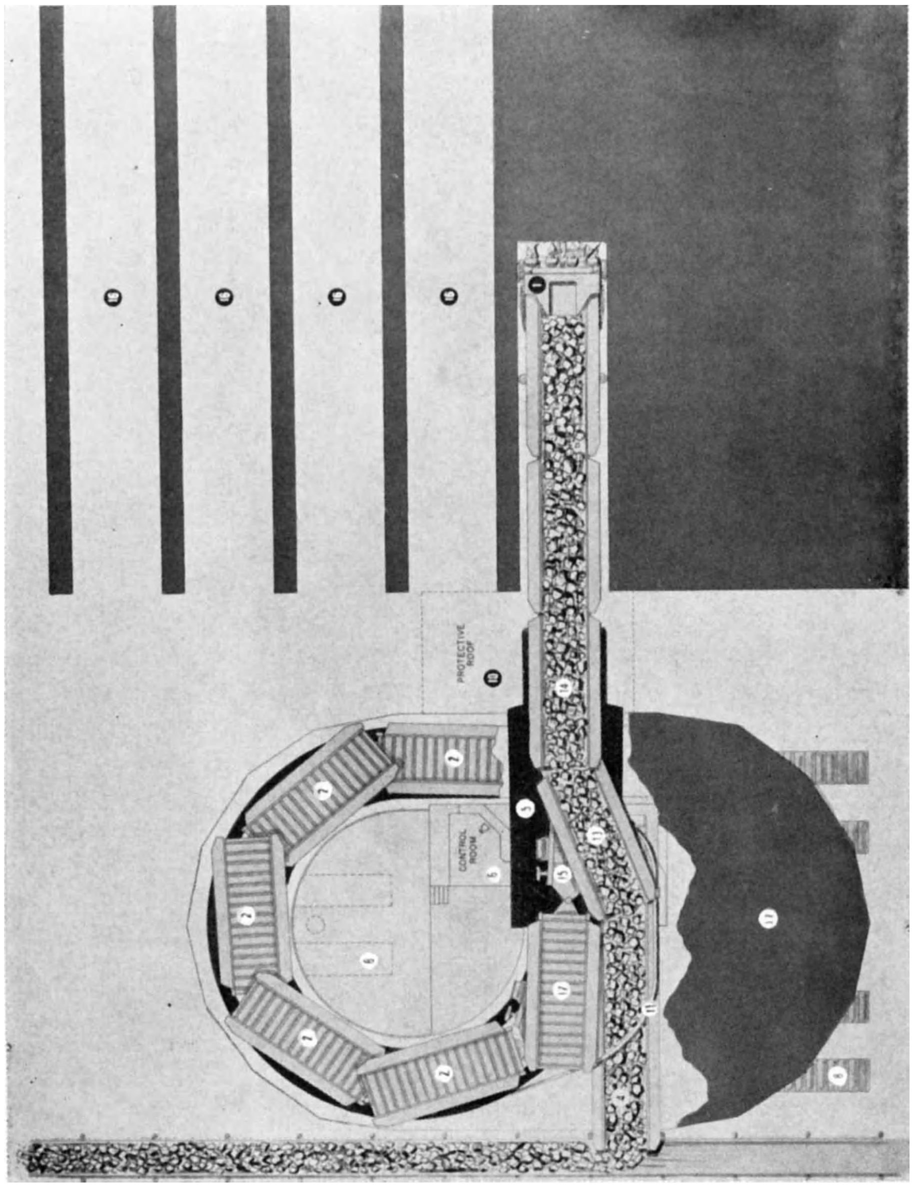
In non-productive areas – such as woodlands and wilderness – operators must backfill at an angle of 70° to the top of the highest cliff and the bottom of the pit must be covered with 5 ft of earth.

Bond requirements for all strip mine operators are increased to $400 an acre, or a minimum of $4000 to assure their compliance with restoration laws.

DRAINAGE

A strip mine is a natural man-made reservoir and a constant struggle is required to keep water out of the pit. Every effort should be made to keep water from entering the pit by ditching above the high-wall and by diverting natural streams to new channels, if necessary.

When water does enter the pit it should be removed as much as possible by gravity flow. It is desirable, when possible, to leave drainage channels open through the spoil piles; but if this is not possible then culverts or drain pipes may be installed at intervals.


PROTECTIVE ROOF
CONTROL ROOM

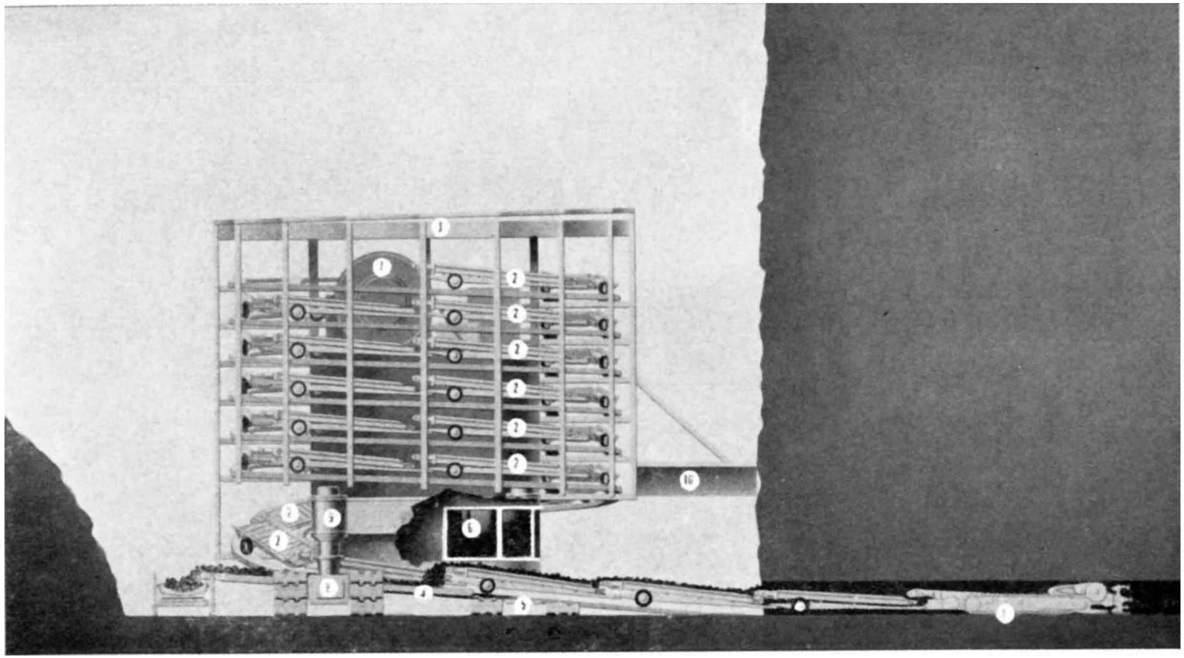


FIG. 25. Joy "Pushbutton" Miner. (Joy Manufacturing Co.)

(1) Boring machine in coal. (2) Conveyor train which follows borer into coal is carried on 1000-ft. long spiral storage runway of (3) Heli-Track structure cascades coal back to (4) ramp or discharge conveyor. (5) Launching ramp positions borer for exact entry into seam. (6) Air-conditioned control cab from which operator controls all functions of the Heli-Track, boring machine and conveyor train through positioning, entry and the full 1000-ft. underground mining run. (7) Reels which store and feed cable for power and control. (8) Crawlers by which Heli-Track moves along highway for new entry into coal. (9) Self-leveling hydraulic jacks. (10) Screen protects in event of rock falls. (11) Guide rail which moves top section of each conveyor as it moves down the runway into discharge position at the launching ramp, then guides it back into straight-line position for entry into hole. (12) Conveyor picked up by guide rail. (13) Conveyor in discharge position. (14) Conveyor returned to straight-line position. (15) Understructure of train conveyor unit. (16) Previously mined holes. (17) Roof of Heli-Track.

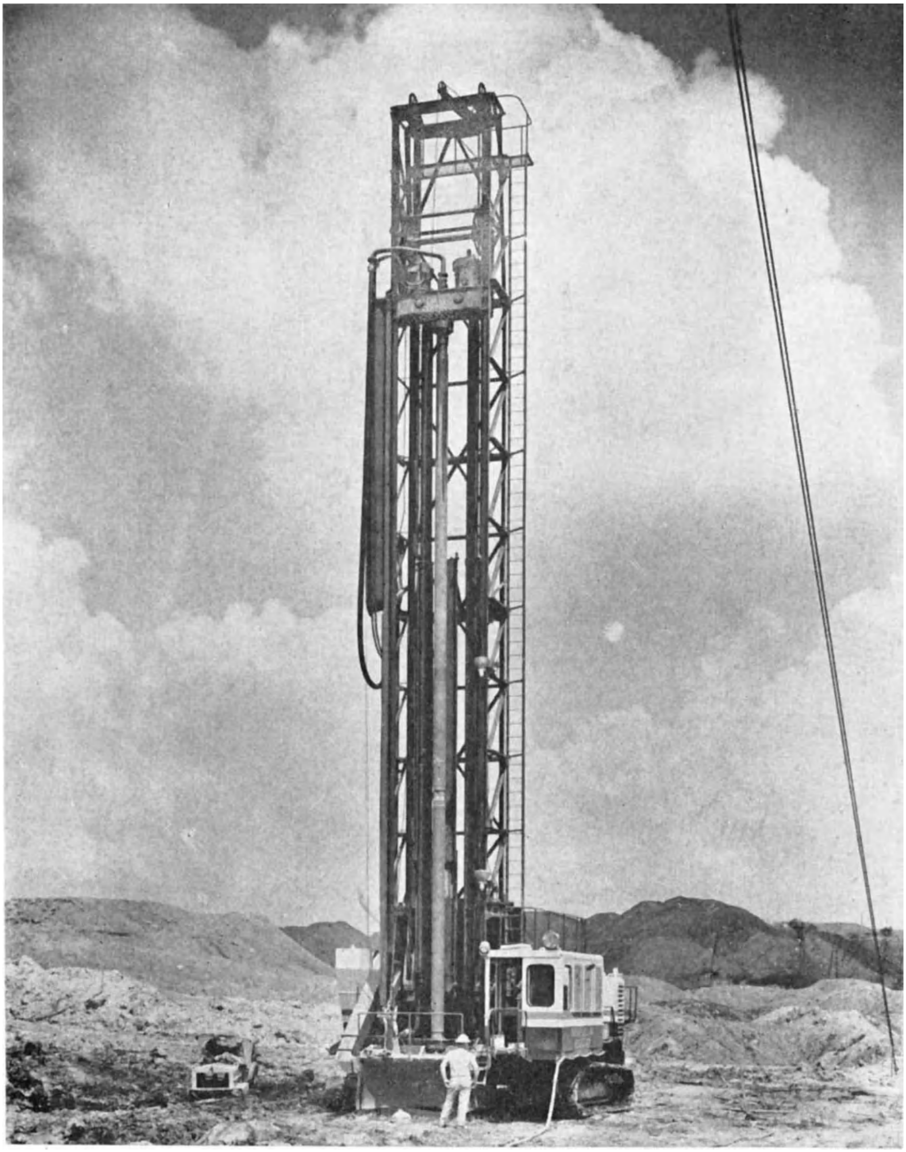


FIG. 26. Rotary blasthole drill capable of putting a 15-in. diameter hole down to a depth of 150 ft in 15 min. (Bucyrus Erie Co.)

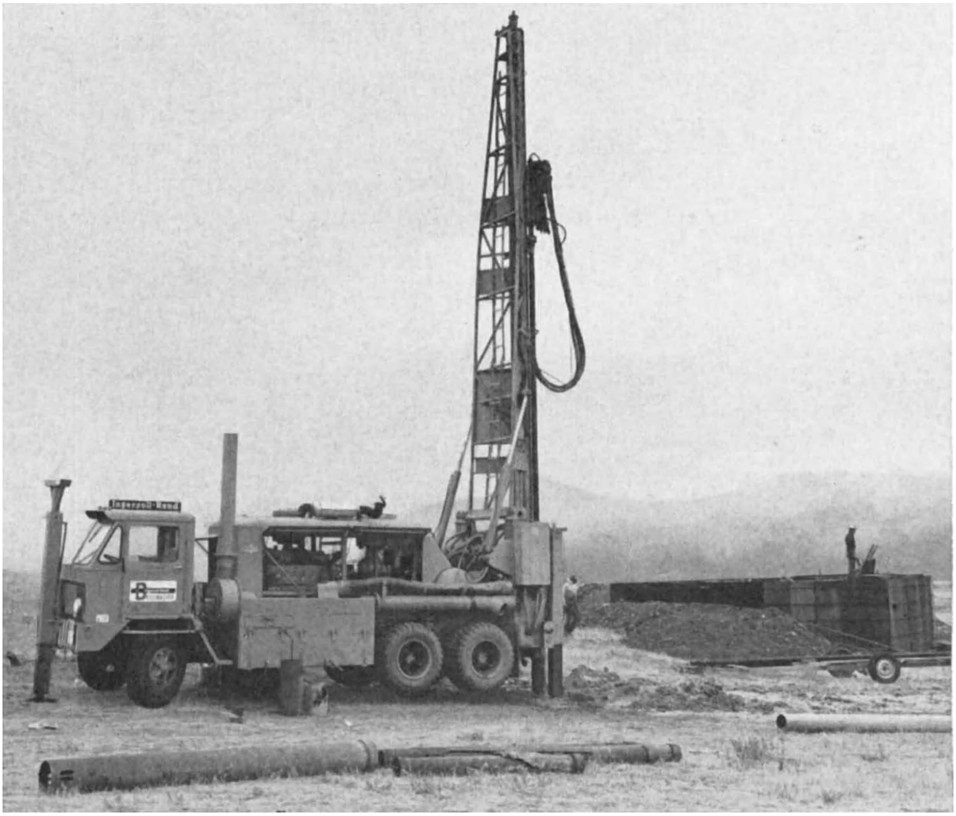


FIG. 27. Truck mounted drill. (Ingersoll-Rand Co.)

Since the high-wall advances constantly, portable pumps are most commonly used when pumping is required. Sumps are excavated at intervals along the pit for installation of these pumps.

Pumps may be mounted on skids or on wheels and in the larger sumps pumps may be float mounted.

Pumps may be driven by electric motors or by diesels. Air cooled diesels are becoming increasingly popular because of their portability and low maintenance requirements. Plastic pipe and aluminium pipe is coming increasingly into use as discharge lines since they are light and easily portable.

DRILLING BLAST HOLES

The two types of drills in most common use in strip mines are: (1) the rotary, and (2) the auger.

Rotary Drills

Rotary drills are of the dry type, using compressed air to remove the cuttings from the holes, and using tri-cone bits for cutting. A large variety of rotary drills is on the market, including sizes to bore holes from 5 up to 15 in. in diameter. Some are mounted on crawlers while others are truck mounted.

In operations where the overburden is thick operators generally favor the heavy crawler mounted machines which can drill holes up to 12 in. in diameter.

These large diameter holes enable operators to break more ground with fewer holes and also make it possible to use low cost explosives such as ammonium nitrate.

Augers

Augers are also available in a variety of sizes and capacities. These machines are usually truck mounted. One heavy duty auger can bore an 8-in. hole up to 150 ft deep or a 12 in. hole to a depth of 60–80 ft. A variety of bits are available for various types of material to be drilled, and bits are available which will cut limestone and coarse grained sandstone.

Drill Performance[2]

To provide sufficient pit width for a 65-yd shovel one company uses a rotary drill to drill holes on three levels in hilly country. Roadways are on 27-ft centers, with holes spaced 27 ft apart on each level. Abrasive action of sandstone wears out a bit after 7000–9000 ft of drilling. In an average shift, two men sink 535 ft of 9-in. hole.

At another operation an average of 1100 ft of 9-inch hole is drilled in 8 hr while drilling with a rotary drill in 50 ft of medium hard shale covered by 12 to 40 ft of sandstone. Bit life averages 21,000 ft of hole.

At another stripping operation 625 ft of $10\frac{1}{2}$-in. hole is drilled per shift with a rotary drill in hard sandy shale that ranges from 45 to 70 ft thick. Bit life is 13,700 ft.

Two men handle the drilling and shooting assignments at a Pennsylvania mine recovering two seams. They employ a truck mounted rotary drill to sink $6\frac{1}{2}$-in. holes at the corners of 15×15 ft squares in 48 ft of clay and sandstone. In an average shift they drill and shoot thirty holes.

At one Ohio mine two crews use auger type machines to sink an average of 600 ft of hole each per shift in sandstone overburden.

At another Ohio mine two drill 400 ft of 8-in. hole per shift and also help charge the holes with AN–oil mixture at the end of the shift.

Vertical vs. Horizontal Drilling

Although most blast holes are drilled vertically there are special situations in which it may be more economical to use blast holes drilled horizontally into the high-wall. Some factors which favor horizontal drilling are:

(1) Overburden is of such nature the drill roads are difficult to build, as in rough terrain or there is soft ground in which drills may become mired. The horizontal drill operates from the floor of the pit and eliminates the necessity for building drill roads at the top of the high-wall.

(2) The cover is thin and the rock is hard. In this case long horizontal holes will give a better breaking action than short vertical holes.

(3) There is a layer of tough rock close to the coal seam. Horizontal holes allow the explosive to be concentrated in or adjacent to this hard rock for better breaking effect.

Both augers and rotary drills are available mounted for horizontal drilling. Heavy augers are available which will bore 12-in. diameter holes horizontally to depths of up to 150 ft. These augers may be truck mounted or may be mounted on a self propelled rubber-tired chassis.

At an Indiana mine a rotary drill equipped with a single mast bores horizontal holes 9 in. in diameter and 48 ft deep without adding drill sections. Penetration rate is 50 in./min. In the best drill shift 816 ft of holes were drilled by one man.

This drill operates one full shift and one part shift 6 days a week and drills enough holes to prepare overburden for operation of a 40 yd^3 shovel 24 hr a day and seven days a week.

Inclined Drilling

Some of the newer drills are so designed that their masts can be tilted to drill holes at angles up to 30° from the vertical. Advantages claimed for inclined drilling are the following:[(2)]

(1) Toe and back breakage can be eliminated.

(2) Fragmentation is better because of better use of explosive energy as well as reduced resistance at the bottom of the hole.

(3) Less footage of hole and less explosives are required per ton of rock broken.

(4) Smaller diameter holes can be used.

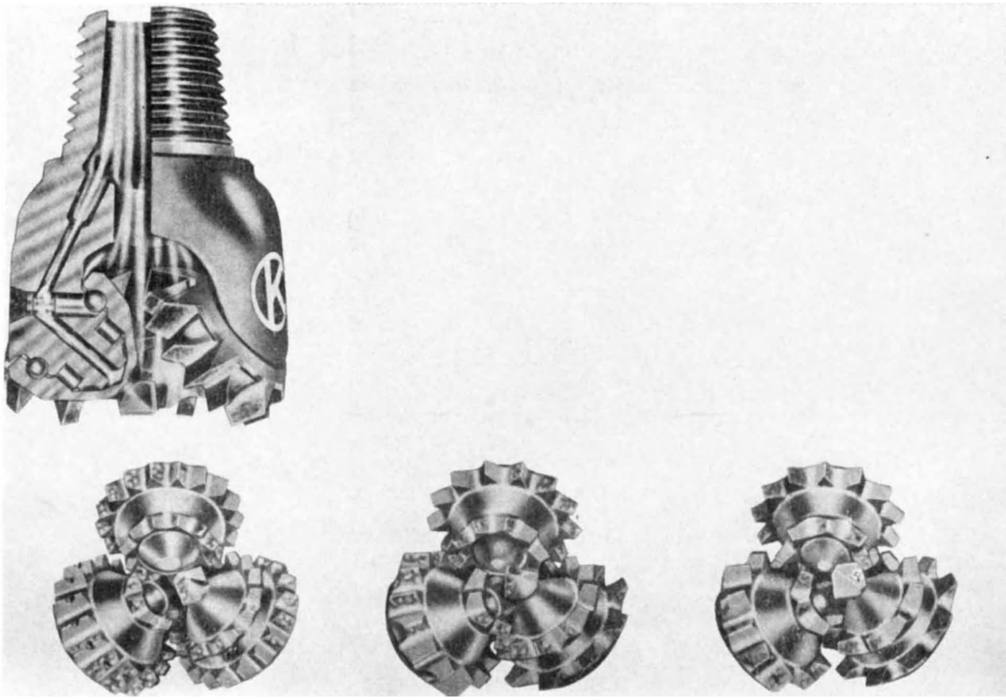
(5) The overburden will be thrown a greater distance.

The disadvantage of drilling inclined holes is that they are more difficult to load as the material being charged may tend to hang up in an inclined hole whereas it would drop to the bottom of the hole in a vertical hole.

Percussion Drills

Percussion and rotary-percussion drills are not extensively used in bituminous coal stripping operations because the overburden to be drilled usually consists of the softer shales and sandstones. Drills of this type are used extensively in quarrying operations in the igneous and metamorphic rocks and to some extent in the metamorphic rocks encountered in anthracite stripping operations.

Down-the-hole (DHD) drills have largely supplanted the hammer and drill steel type for large diameter and deep blast holes. With the down-the-hole drill the



The "K-1"

"K-1," the hard formation bit, "The Mule," designed and built to drill holes in most hard and abrasive formations, such as hard sandstone, chert, granite, dolomite, and other hard abrasive formations. This bit has the chipping, crushing, rolling action. The tearing-scraping is limited to a minimum.

The "K-2"

"K-2" medium formation bit, "The All Formation Bit" maintaining almost the speed of the "K-3." Designed for drilling in hard, non-abrasive formations, such as hard lime, hard shale, dolomite lime, hard anthracite.

The "K-3"

"K-3," the most versatile bit, "The Speedy" for its performance in soft drilling, such as unconsolidated formations, soft shales, clay, red bed, and lime stone.

FIG. 28. Typical bits for rotary drilling. (Oil Tool Sales Co.)

drill unit with the rock bit attached goes down the hole so that all of the striking energy of the pneumatic hammer is transmitted to the rock bit through a short rod instead of the pneumatic portion of the drill remaining on the surface and the hammer blows being transmitted to the bit through a long drill steel.

Drill Bits

Rotary drills usually employ tricone bits of the type which was originally developed for the oil industry. A tricone bit consists of three cones free to rotate about their axes which are face studded with cutting teeth. For soft formations these teeth are rather long and are spaced relatively far apart. For harder formations the teeth are made shorter and spaced closer together. For extremely hard rock the cones are studded with rounded tungsten carbide knobs which serve as the breaking or crushing teeth.

To obtain optimum operating results with rotary bits it is necessary to apply heavy pressure to them. For 9-in. bits or larger in hard formations it is customary to apply approximately 6000–7000 lb./in. of bit diameter, and to rotate these bits at approximately 40–50 rpm.[20]

Bit Life

Experimental drilling is desirable to determine the optimum combination of bit size, speed of rotation, and bit weight (amount of push on the bit).

A quarry in medium-hard limestone employed a drill which had the ability to apply approximately 45,000 lb. weight to the bit and was equippped with an air

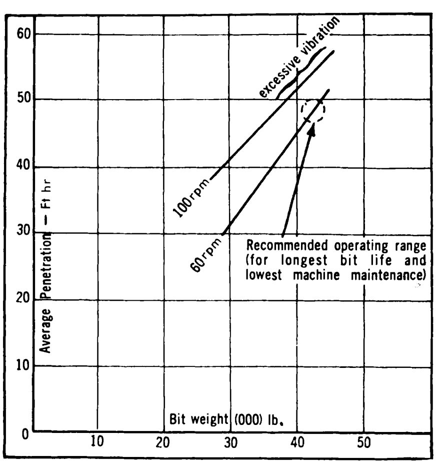


FIG. 29. Results of typical test drilling to determine the most economical bit weight and rpm in medium hard limestone using a $7\frac{7}{8}$-in. bit (designed for rock of medium hardness) and a drill of 45,000-lb. bitweight capacity.[20]

compressor with enough capacity to clean cuttings from a $7\frac{3}{8}$ in hole.[20] This machine had been drilling with $6\frac{1}{4}$-in. bits and bit life had been approximately 700 ft per bit with an average penetration rate of about 10 in./min. By changing to $6\frac{3}{4}$-in. bits and increasing the bit weight from 30,000 to 38,000 lb., the penetration rate increased to 12 in./min. Previously the $6\frac{1}{4}$-in. bits had always failed due to worn bearings, but the bearings in the $6\frac{3}{4}$-in. bits were larger and even at the increased bit weight provided a bit life of 1600 ft. Further experimentation revealed that it was possible to increase the bit life to 3000 ft per bit with an average penetration rate of 8 in./min by lowering the revolutions per minute and reducing the bit weight.

Figure 29 shows graphically the results of tests on $7\frac{7}{8}$-in. bits.

In some instances where soft but abrasive materials are to be drilled it may be more economical to use less expensive bits which can be discarded when they become dull.

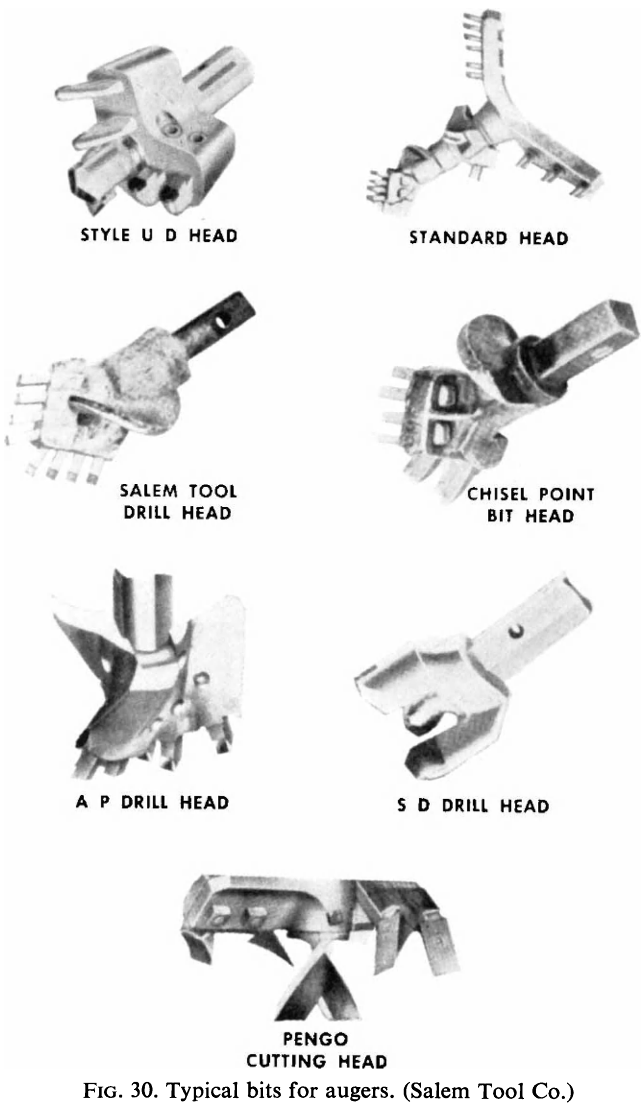


FIG. 30. Typical bits for augers. (Salem Tool Co.)

At one stripping operation where blast holes were drilled in a very abrasive sandstone it was found that tricone bits would wear out after only 90–110 ft of drilling.[21] When a "finger type" auger bit head with replaceable bits was substituted for the tricone bits, bit costs were reduced by two-thirds, to less than 4 cents per foot of hole drilled.

Auger Drill Heads

Bit heads for augers generally are of the type which have two or more projecting "fingers" which may be faced with tungsten carbide and which act as cutting tools to groove and rip the rock at the bottom of the hole. Figure 30 shows some of the drill heads available for augers.

Factors Affecting Costs

Bit Costs

The general features of larger bits are more rugged and the size of the bearings increases with the increase in bit diameter. This permits application of higher bit weight per inch of bit diameter and results in higher rates of penetration and longer bit life. Although the larger bits are more expensive, the longer bit life often accounts for a lower bit cost per foot.[20]

Labor

The drilling crew remains the same regardless of the size of hole drilled, therefore the labor cost per foot of hole drilled varies inversely with the penetration rate.

In the overall picture it is the cost per ton of rock broken which determines the economy of the drilling process. Thus the number of holes required to break a given tonnage is also a factor to be considered. Since larger holes can hold more explosive, fewer holes are required to break a given tonnage of rock.

Thus the relative costs of drilling various sizes of holes varies with various types of drills and bits. Thus the overall cost of overburden preparation depends upon a number of interrelated factors which can only be evaluated after testing a variety of drill types, sizes, and hole spacings.

BLASTING

Explosives (See also page 483)

Ammonium nitrate (with carbonaceous material added) is the most commonly used blasting agent in strip mining operations. Although fertilizer grade ammonium nitrate had long been known to be explosive under certain conditions it was not until 1954 that the Maumee Collieries Co., of Terre Haute, Indiana developed a mixture of ammonium nitrate and carbon black which was packed in polyethylene bags and used for blasting. This mixture was patented and sold under the trade name Akremite.

A mixture of granular ammonium nitrate and fuel oil is presently in extensive use in strip mining operations. These ingredients are usually mixed in the propor-

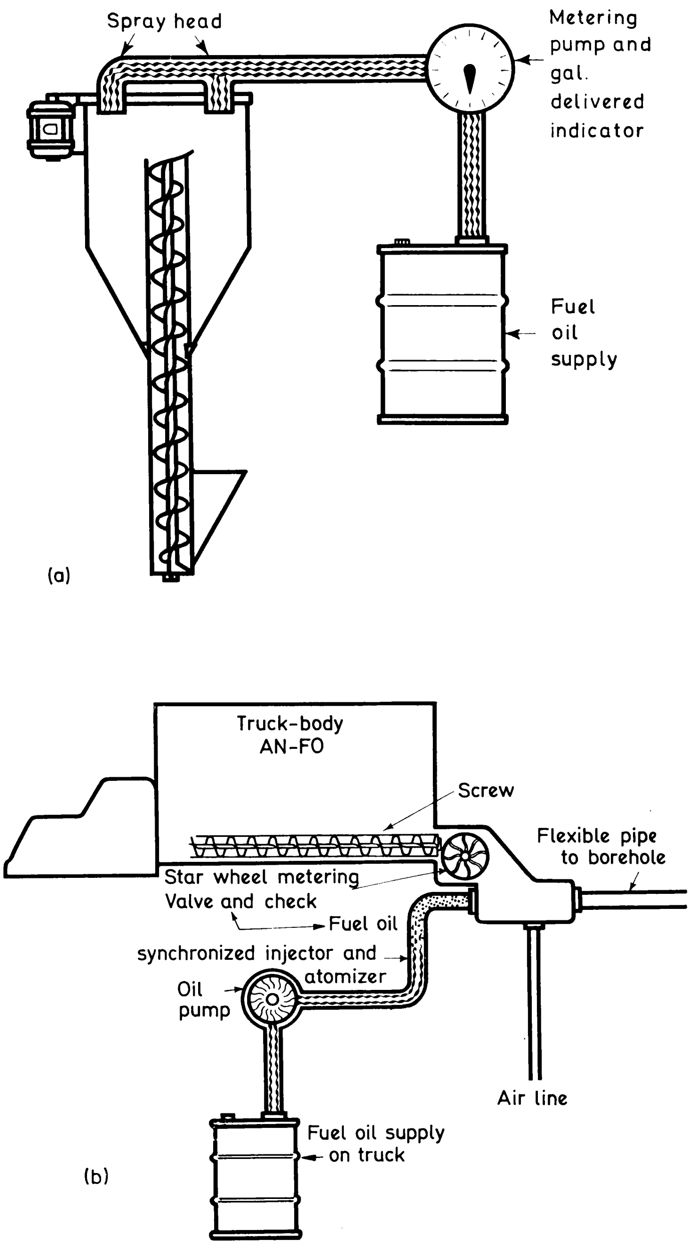


FIG. 31 (a) Popular design for a permanent-type plant for blending nitrate and fuel oil provides for thorough mixing.[2]

(b) Typical design of portable mixing equipment includes nitrate hopper, oil supply, mixer, and air-placement unit.[2]

tion of 100 lb. of ammonium nitrate to 5 or 6 lb. of fuel oil. The resulting "AN–FO" mixture may be bagged for transportation to the drill holes or it may be transported in bulk and poured or blown into the holes by means of compresssed air.

Ammonium nitrate is a relatively slow velocity explosive and explosion must be initiated with high explosive primers. Ammonium nitrate also detonates better

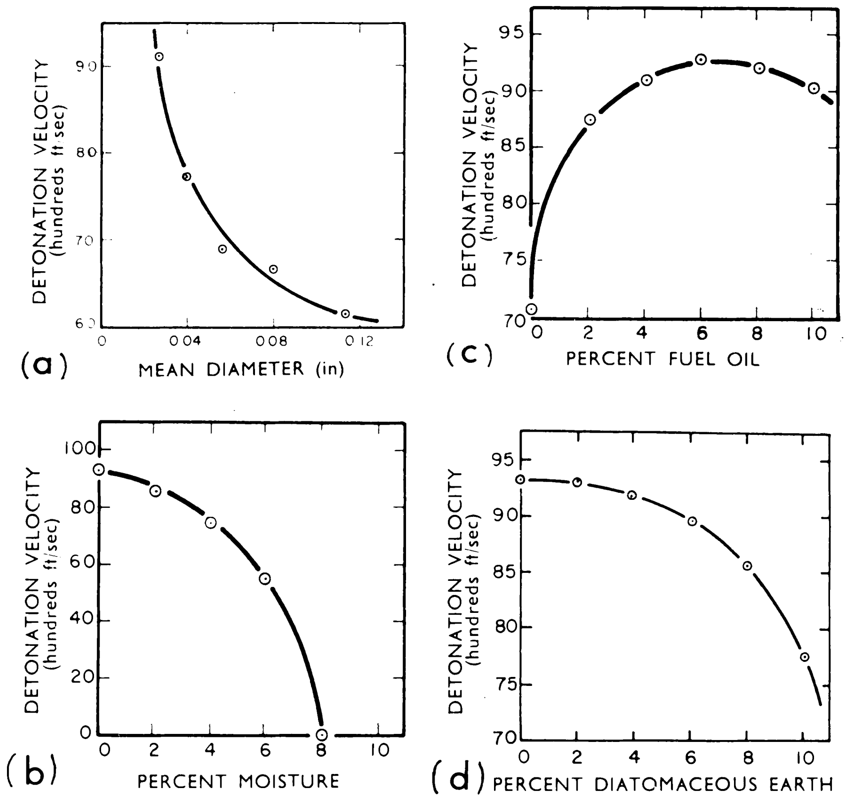


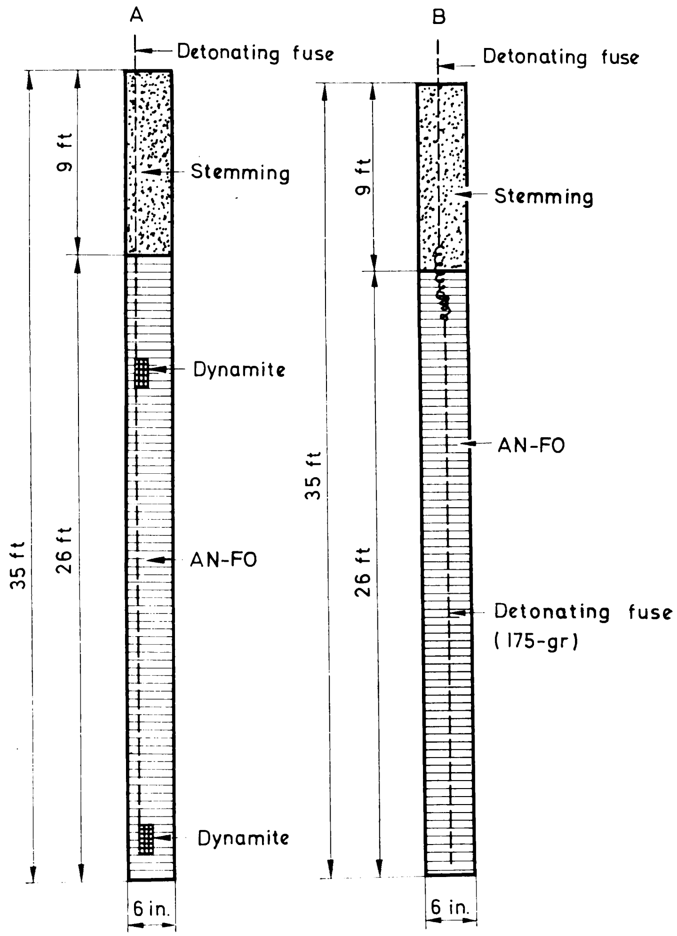
FIG. 32. Characteristics of ammonium-nitrate explosives.[22]

(a) The detonation velocity is increased as the size of prill is decreased.
(b) Water has a marked effect on velocity; failure occurring at 8 per cent moisture.
(c) Velocity falls faster with fuel-lean than with fuel-rich mixtures.
(d) Above 4 per cent diatomaceous earth, the detonation velocity falls rapidly.

when it is loaded in large diameter drill holes and difficulties are sometimes experienced in getting complete detonation in drill holes which are of small diameter and relatively long.

Many of the explosive manufacturers are producing variations of the "AN" mixtures. These may contain technical grade ammonium nitrate, mixed with a carbonaceous dust and a sensitizer, such as nitromethane.

In addition the conventional high explosives, such as dynamite, may be used in special situations when extremely hard or tough strata must be broken.



Cost with Dynamite Primer (A)

250 lb. AN at $0.04/lb.	$10.00
35 ft 50-grain det. fuse at $0.032/ft	1.12
2−2×8 sticks of 60−75% dynamite, approximately $0.36 each	0.72
2.5 gal No. 2 fuel oil at $0.10/gal	0.25
Total	$12.09

Cost with Detonating Fuse (B)

250 lb. AN at $0.04/lb	$10.00
20 ft 150–175-grain det. fuse average at $0.057/ft	1.14
15 ft 50-grain det. fuse at $0.032/ft	0.48
2.5 gal No. 2 fuel oil at $0.10/gal	0.25
Total	$11.87

FIG. 33. Comparison of the costs of priming with dynamite and with detonating fuse.[2]

Mixes

Most companies making their own AN–FO mixes use either a "prilled" or grained ammonium nitrate with No. 2 fuel oil. Approximately 4 quarts of oil are added to each 100 lb. of ammonium nitrate.[2] (Note: "prills" are pallets.)

Many of the larger companies have their own mixing plants where the ingredients can be metered as they flow to a mechanical mixer. Some companies use the product immediately after it is packaged while others prefer to let the mixture season before using it. The seasoning period varies from several hours to several days.

Other producers prefer to mix the nitrate and oil at the hole site. Oil is poured over the opened bags at each hole then left to percolate down through the ammonium nitrate for a short period before holes are charged.

The most recent development is a slurry type blasting agent. One new slurry consists of a mixture of ammonium nitrate, sodium nitrate, high-explosive sensitizer, and water. It has a density of about 1.5 and a rate of detonation of about 17,000 ft/sec.

Primers

Prilled ammonium mixtures react differently with different priming systems.

Dynamite used as a primer causes the AN–FO to release a larger volume of gas which is desirable for heave and throw of the bank while a detonating cord causes a sharp cracking impact which is desirable when a hard rock must be shattered.

One operator uses about 1 lb. of primer for each 150 lb. of AN–FO mixture in dry holes and about 1 lb. of primer for each 50 lb. of AN–FO in wet holes.

Loading Holes

At smaller operations the AN–FO mixtures are usually bagged and the bags are charged into the blast holes. At larger operations, where consumption of nitrate is several hundred tons or more annually, nitrate may be transported in bulk in enclosed hopper trucks and mixed with fuel oil as it is discharged into the holes.

Horizontal holes may be loaded with bagged nitrate using mechanical tampers to push the bags into the hole. A more efficient system for charging horizontal holes is that which employs a truck-mounted unit to blow the bulk mixture into the holes through a plastic tube.

Bagged drill cuttings are commonly used for stemming material for horizontal holes. Split plugs and wedges have also been used to stem horizontal holes because they are easier to place than the bagged cuttings.

Drill cuttings are commonly used for stemming in vertical holes.

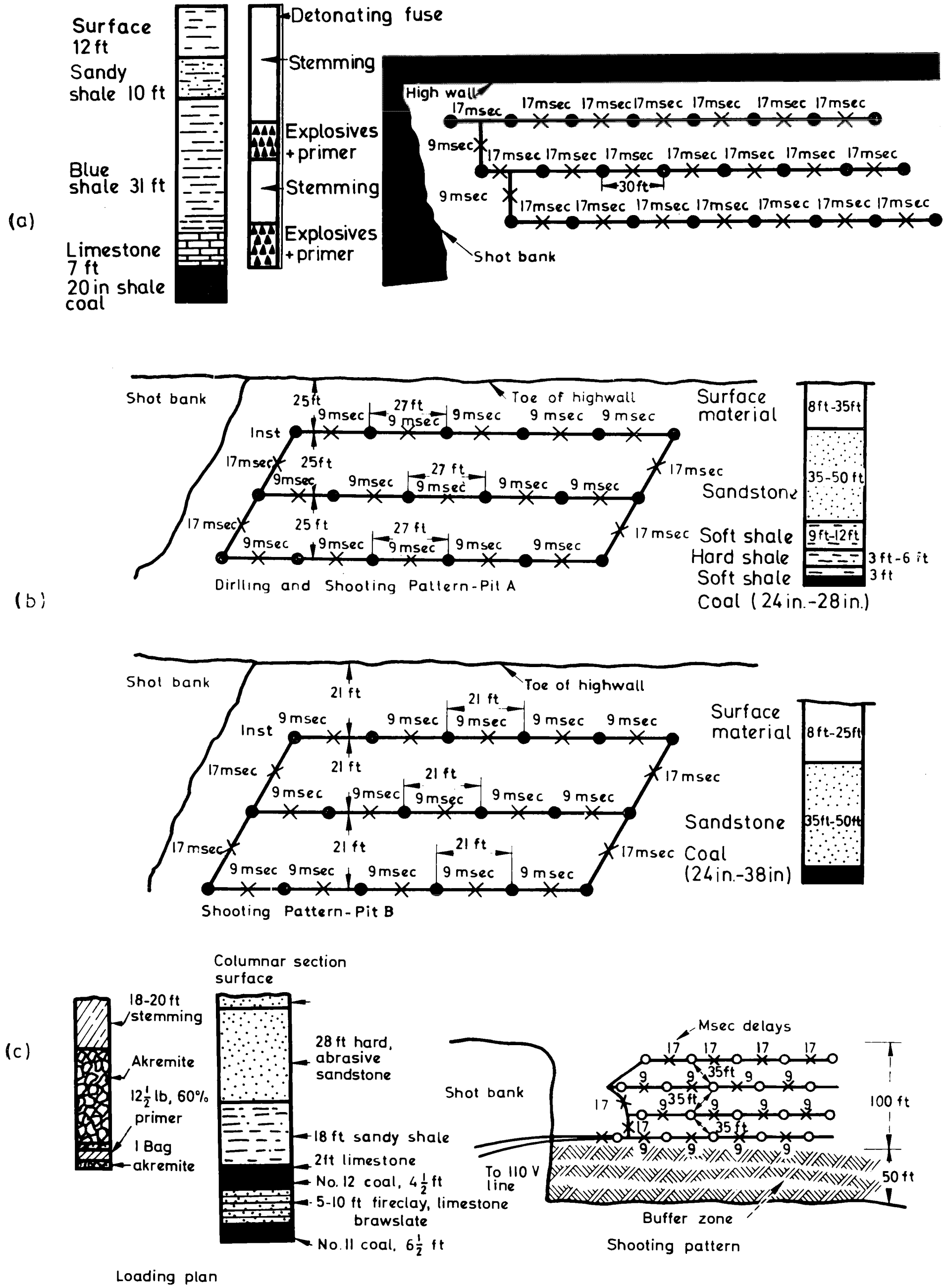


FIG. 34. Blast hole patterns and spacing.[2]

(a) Deck loading concentrates part of the charge in the upper portion of thick overburden for better fragmentation.
(b) Deck loading is also advantageous in overburden consisting of rocks with varying hardnesses. Millisecond delays increase the effectiveness of explosives.
(c) Buffer shooting makes possible a uniform drilling pattern, eliminates large chunks which are sometimes produced when shooting against an open face.

Where blast holes pass through strata of varying degrees of hardness an effort is usually made to obtain the greatest concentration of explosive in the toughest strata. A denser mixture may be formulated for placement in such rock, with increased density being obtained by proper gradation of the grain size of the mixture of ammonium nitrate granules and prills.

Blast Hole Patterns and Spacing

The blast hole pattern used and the spacing of holes depends upon a number of factors including thickness of overburden, type of rock in overburden, and size of the blast holes.

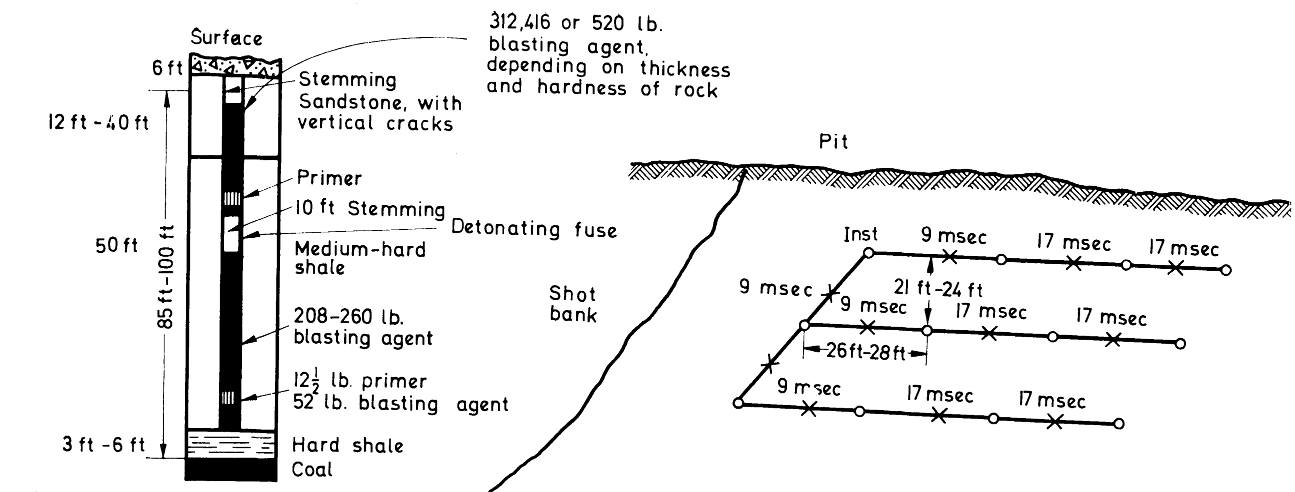
Spacing between the rows of holes which parallel the high-wall may be anywhere from 15 to 25 ft and spacing between individual holes in a row may be from about 20 to 30 ft.

The accompanying Figures show typical drill hole patterns and spacing in use at several operating mines.

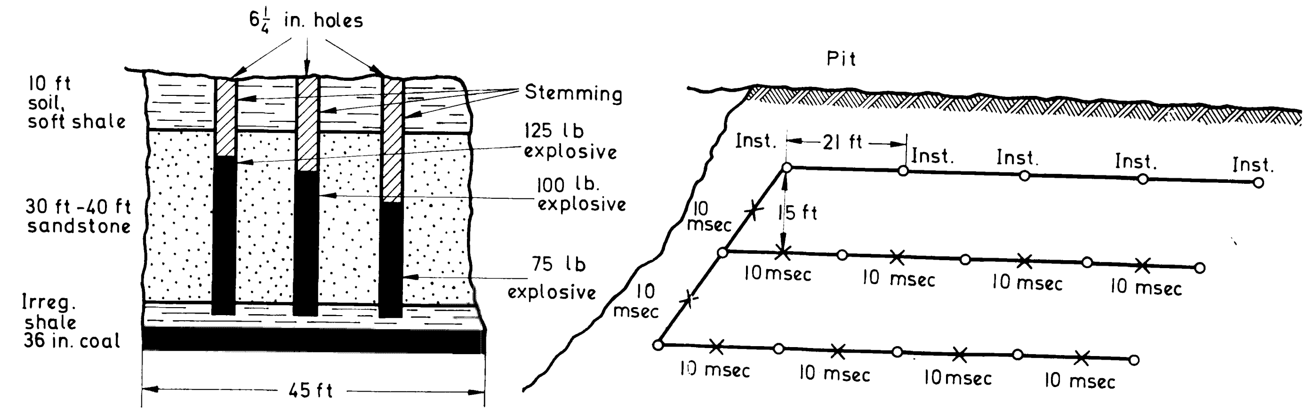
A formula based on the size of hole, strength of rock, and type of explosive has been developed. From this formula the maximum allowable hole spacing may be computed.[23] Table 10 has been derived from the formula. Hole spacing may be computed if the diameter of the hole and the type and condition of the rock is known. The values in the table are based upon the use of an oxygen-balanced, prilled ammonium-nitrate fuel oil mixture.

TABLE 10. TYPICAL BOREHOLE SPACING[23]

Rock type	Tensile strength (psi)	Spacing (ft/in.) of borehole diam
Strong granite	1298	1.92
Strong anhydrite	1220	1.97
Strong limestone	890	2.30
Average granite	888	2.32
Marble	860	2.37
Weak anhydrite	800	2.45
Graywacke	700	2.62
Strong sandstone	583	2.85
Average limestone	480	3.15
Marlstone	480	3.15
Weak granite	422	3.37
Average sandstone	412	3.40
Salt (potash-halite)	400	3.46
Greenstone	380	3.55
Weak limestone	280	4.12
Weak sandstone	280	4.12


Surface
6 ft
12 ft - 40 ft
50 ft
85 ft-100 ft
3 ft - 6 ft
Stemming
Sandstone, with vertical cracks
312,416 or 520 lb. blasting agent, depending on thickness and hardness of rock
Primer
10 ft Stemming
Detonating fuse
Medium-hard shale
208–260 lb. blasting agent
12½ lb. primer
52 lb. blasting agent
Hard shale
Coal
Pit
Shot bank
Inst
9 msec
17 msec
17 msec
9 msec
21 ft-24 ft
9 msec
17 msec
17 msec
26 ft-28 ft
9 msec
17 msec
17 msec

(a)


6¼ in. holes
10 ft soil, soft shale
30 ft - 40 ft sandstone
Irreg. shale
36 in. coal
Stemming
125 lb explosive
100 lb. explosive
75 lb explosive
45 ft
Pit
Inst.
21 ft
Inst.
Inst.
Inst.
Inst.
10 msec
15 ft
10 msec
10 msec
10 msec
10 msec
10 msec
10 msec
10 msec
10 msec
10 msec

(b)

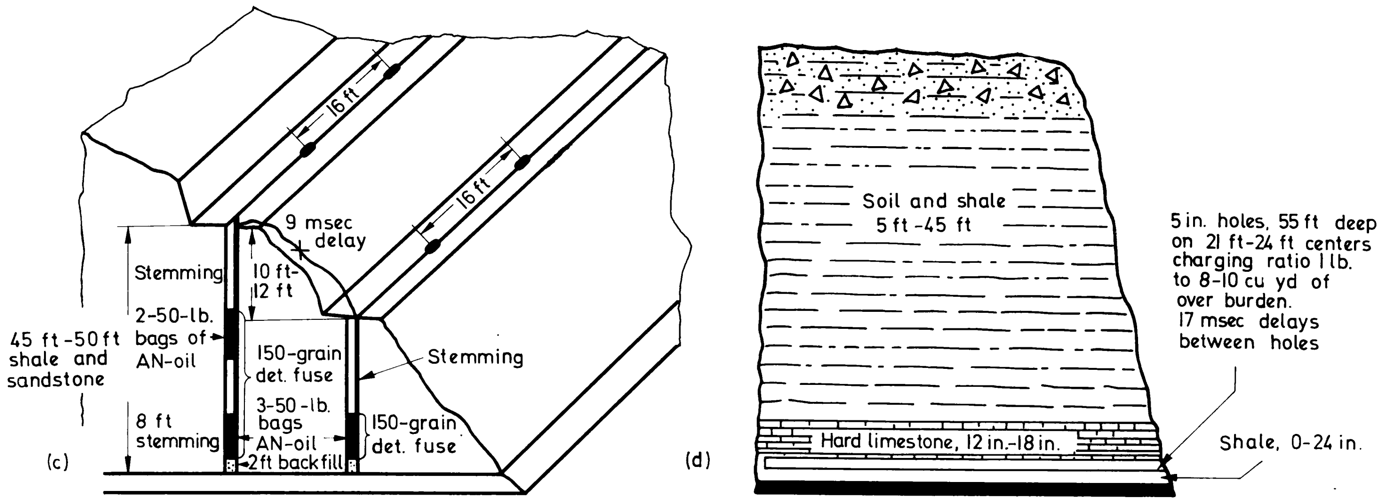


FIG. 35. Blast hole patterns and spacing.[5]

(a) Drilling and shooting procedures in tough rock are designed to get maximum fragmentation at lowest cost. Delays improve shooting results.
(b) Multiple rows of vertical holes with varying charges are effective in breaking bank composed mainly of sandstone.
(c) Two-level drilling is a common practice in hillside stripping. Deeper holes receive heavier charge.
(d) Horizontal holes make it possible to concentrate explosives near hard layers close to the top of the coal.

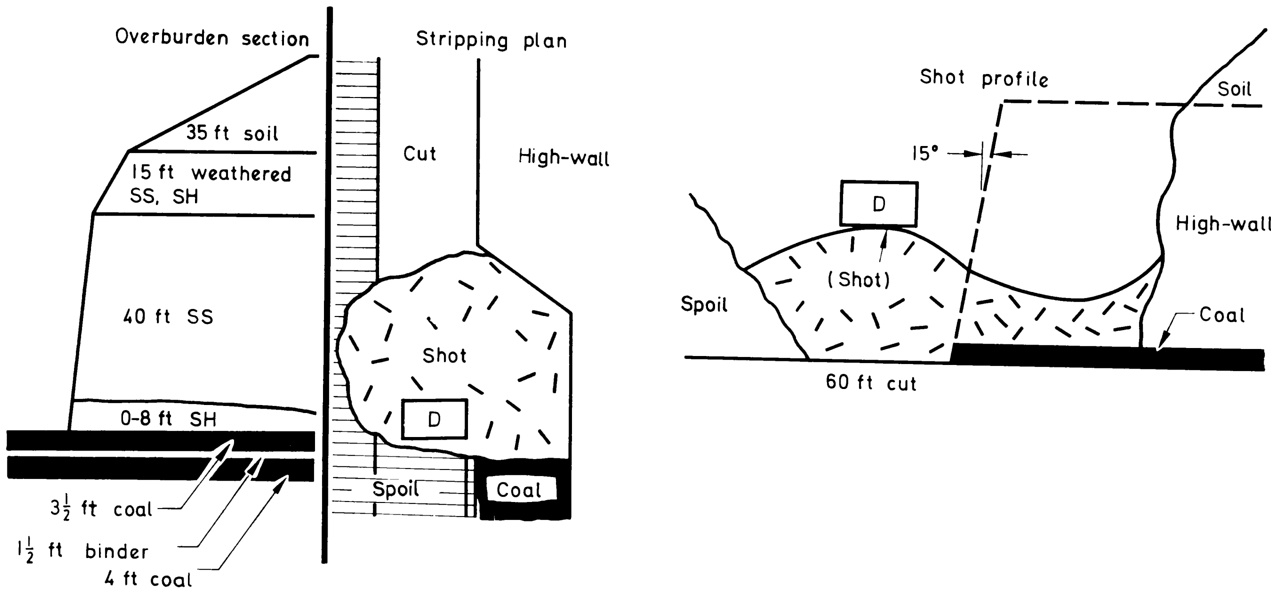


FIG. 36. Overburden casting with explosives.[24] Plan and profile of a typical application of explosives to overburden casting shows a minimum of material to be handled by excavating machines.

CASTING OVERBURDEN WITH EXPLOSIVES[24]

In explosives casting large amounts of low-cost ammonium nitrate mixtures are loaded into medium sized drill holes in a usual ratio of more than 1 lb. of powder per cubic yard of overburden. The explosive charges are detonated through millisecond delay electric blasting caps. When the shot is fired, a large part of the overburden is blasted into the pit away from the high-wall and up on the spoil pile where it attains a favorable angle of repose.

One stripping operation with a 50-ft sandstone overburden prepares its bank for shooting by drilling five rows of holes on 12 ft burdens with 15 ft spacings. Four rows are $6\frac{1}{4}$-in. diameter holes while on the outside row $7\frac{3}{8}$-in. diameter holes are used to insure the desired movement of the toe. Hole depths average 45 ft. The powder factor averages 1.2 lb./yd^3.

This blasting operation moves a minimum of 35 per cent of the overburden without the use of equipment and uncovers 35–40 per cent more coal in the same time. Even though drilling and blasting costs are higher, mining profits are greater because of the increased production.

Factors Favoring the Use of Explosives Casting

(1) Deep, hard overburden requiring extensive shooting.
(2) Dumping radius of primary stripping unit less than 150 ft.
(3) Narrow, steep cuts, 60–100 ft wide.
(4) Undercapacity of primary stripping unit.
(5) Overcapacity of coal mining unit.
(6) Ability to use least expensive AN–FO explosive.

Factors Unfavorable for Explosives Casting

(1) Overburden is shallow and easily excavated.
(2) Cuts are more than 100 ft wide.
(3) Conditions are poor (water, etc.) for the use of bulk AN–FO.
The haulage road may have to be run past the stripping pit.

RIPPING OVERBURDEN

With the development of large size crawler tractors and heavy tractor-mounted hydraulically-operated rippers it has become possible to substitute ripping for blasting as a means of breaking sedimentary rocks so that they can be excavated.

A 335-h.p. tractor equipped with three ripper arms spaced at 53 in. apart can in one pass loosen rock to a depth of 12 in. over a width of 9 ft. Under favorable conditions such a unit can produce 300–600 yd^3/hr.

Experience in several limestone quarries indicates that ripping costs may run from 5.2 cents per yd^3 to 11.5 cents per yd^3 depending upon the hardness of the rock. In excavating sandstone the ripping costs have ranged from 2.1 cents per yd^3 in soft sandstone to 15.0 cents in hard rock. Ripping costs on these jobs were less than the costs for drilling and blasting methods.

Generally shales and sandstones which overlie bituminous coal beds are amenable to ripping and this method warrants consideration when mining methods are chosen.

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CHAPTER 2

OPEN-PIT MINING

THE mining of metalliferous ores by surface mining methods is commonly designated as "open-pit mining" as distinguished from the "strip-mining" of coal and the "quarrying" of other non-metallic materials such as limestone, building stone, etc.

The larger portion of U.S. iron ore is produced by open pit operations and almost 80 per cent of U.S. copper is produced by this method.

Other metalliferous ores produced by open pit operations include tungsten, mercury, uranium, cobalt, nickel, and gold. Open-pit zinc and lead mines have been operated at some locations but they are a rarity.

The principal rock materials produced by quarrying are limestone, and various types of stone which are crushed for use as road material and concrete aggregate.

Open-pit operations range in size from those where a few men with jackhammers, a bulldozer, and a truck produce a few tons of ore daily up to operations like those of the Utah Copper Division of the Kennecott Copper Corporation at Bingham Canyon, Utah which mined more than 100 million tons of rock (ore and waste) in 1962 and which has mined more than 2,200,000,000 tons of rock (ore and waste) in 58 years of operation.

PLANNING

Before any mining is started it is desirable that the orebody be outlined by drilling so that its size and shape are known. Layout of the operations should be planned to minimize haulage distances for waste and ore and to keep the ratio of waste to ore as small as possible.

If the orebody is located in rough country considerable thought must be given to minimizing the vertical distances to which waste must be raised to disposal dumps as well as to horizontal haulage distances.

Waste to Ore Ratios

An analysis of the production records of several large open-pit copper mines showed a production of 1.6 tons of waste for each ton of ore in the period from 1955 to 1959, and 2.1 tons of waste per ton of ore in 1959. Several medium sized open-pit mines produced 1.0 tons of waste per ton of ore for the 1955–1959 period and

1.6 tons of waste per ton of ore for 1959. Three small open-pit mines produced about 0.26 ton of waste per ton of ore during normal operations.[1]

The large low grade copper deposits, as well as a number of iron ore deposits, are usually of a nature which is suitable for mining by block caving, and in the years between 1920 and 1940 there were several large block caving operations in such deposits.

Earth moving units of newer and larger designs and improved blast hole drills, and the introduction of low-cost explosives have combined to reduce the cost of breaking and removing a ton of rock so that the balance has been tipped in favor of mining large deposits by surface mining methods unless the depth of the barren cover is excessive.

The cost of mining 1 ton of rock by block caving is about the same as the cost of mining 4 tons of rock by surface mining methods. On this basis it would be economical to mine all that ore which did not require a stripping ratio in excess of 3 : 1 by surface mining methods.

An important consideration in the choice of the open-pit method is the preliminary stripping, which must be completed before ore can be produced at planned capacity. The average amount of rock stripped at several mines indicates that waste equal to about one-fifth of the total estimated ore reserve has to be removed before the mine can produce at planned capacity.[1]

Three relatively new mines opened during the period 1955–1960 and having an average daily output of 14,000 tons of ore, yielded no ore during the first year; yielded ore at the rate of 25 per cent of planned output by the end of the second year; and at the rate of 75 per cent of planned capacity by the end of the third year. All plants reached design capacity during the fourth year after about 30 per cent of the total waste had been removed.[1]

Mining Cycle

The mining cycle consists of drilling, blasting, loading, transportation of ore and waste. Drilling is broken down into two categories, primary and secondary. Primary drilling is accomplished by sinking vertical, or near vertical, holes behind the open face of an unbroken bank. Secondary drilling is required when boulders too large for the shovels to handle are produced by the primary blasting or when hard unbroken points of rock project above the digging grade in the shovel pit.

Benches

Benches in open-pit copper mines normally range from 25 to 60 ft high. The height is selected to give maximum digging efficiency but is sometimes limited by ore waste separation. The height normally is reduced where the rock is hard and difficult to break and may be increased in easy digging areas.[1]

DRILLING BLAST HOLES

Blast holes in open-pit copper mines are generally from 6 to 12 in. in diameter and are drilled to depths of 2–15 ft below bench grade.[1] Smaller diameter holes spaced more closely are used in harder ground, and larger diameter holes spaced farther apart are used in easily broken ground. Distances between holes range from 12 ft for 6-in. holes in very hard ground to as much as 50 ft for 12-in. holes in easily broken ground.

In taconite ores of the Mesabi Range holes are jet pierced with a minimum diameter of 6 in. and chambered at the bottom to a diameter of 12 in. Theses holes are arranged in patterns with spacings from 21×21 ft to 21×26 ft. A staggered 20×24 ft spacing is common. Bench heights are about 40 ft.

In other types of quarrying hole diameters may vary from 3 in. on a 6×6 ft spacing up to 9 in. on a 30×30 ft spacing. Generally the smaller holes and closer spacing have been used for the shallower holes, with larger holes being used where greater depth is required.

Types of Drills

The churn drill, which was formerly used extensively for boring large blast holes in surface mining operations has been largely superseded during the past few years by other types of drills which are capable of drilling several times as much footage in a shift.

The churn drill (or cable-tool drill) consists basically of an engine which drives gears which in turn cause a "walking beam" to reciprocate in a vertical plane. A heavy string of cutting tools with a chisel-type rock bit is suspended from the end of the walking beam and is alternately lifted and dropped to "chop" away the rock at the bottom of the drill hole. At suitable intervals the tool string is removed from the hole and the drill cuttings are bailed out.

The following types of drills are presently in use in open-pit mining and in quarrying operations:

(1) Rotary drills.
(2) Down-the-hole (DHD) drills.
(3) Wagon drills.
(4) Jet-piercing machines.

Rotary Drills

These have previously been described in the Section on "Drilling and Blasting" under Chapter 1, "Strip Mining of Coal". In strip mining operations the overburden to be drilled is sometimes soft enough that finger-type auger bits can be used. In open-pit mining, however, the ground to be drilled is usually harder and tricone roller bits are most commonly used.

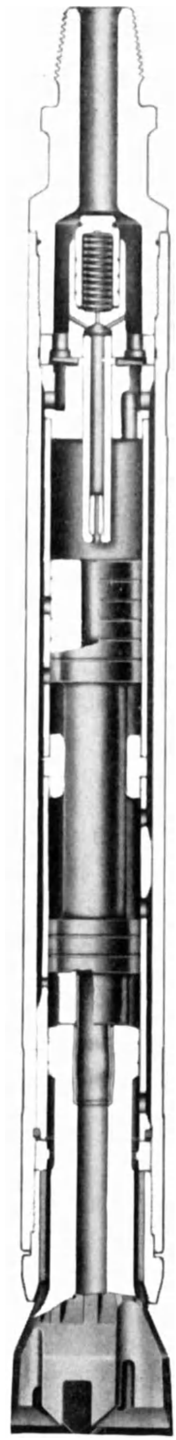


FIG. 1. A down-the-hole drill. (Ingersoll-Rand Co.)

Down-the-hole drills

These are a fairly recent development, and for larger holes have replaced the heavy wagon drill. The DHD is a pneumatic percussion drill whose percussion mechanism has been designed to fit in a cylinder a few inches in diameter so that it can enter the drill hole and follow the bit down the hole. The percussion mechanism is closely coupled to the rock bit and the usually long drill rod is eliminated. This eliminates the large energy losses which accompany the transmission of percussive energy through a long drill steel and also eliminates the drill steel breakage problem.

TABLE 1. DRILLING RATES AT BAGDAD MINE[12]

Type of rock	Feet per shift	
	Churn drill (7-in. hole)	Rotary drill ($6\frac{3}{4}$-in. hole)
Basalt	60	200
Gila conglomerate	110	350
Monzonite:		
Capping	85	275
Sulfide	50	160

DHD drills may be used in rock which is too hard to be economically drilled by rotary drills. Because of the size of the drill mechanism which must enter the hole it is not feasible to drill holes smaller than about 6 in. in diameter with DHD drills, and most such blast holes are 6 or 7 in. in diameter.

Wagon Drills

These are heavy wheel-mounted percussion drills which use the conventional drill steel and rock bits. They are used to bore blast holes up to 3 in. in diameter in the harder rocks.

Jet-piercing machines

The principal use for these machines lies in making blast holes in the extremely hard iron ores which are characterized by the taconite deposits of the Mesabi Range in Minnesota.

Taconite is a ferruginous chert or slate in the form of a compact siliceous rock in which the iron oxide is so finely disseminated that substantially all of the iron bearing particles of merchantable grade ore are smaller than 20-mesh. This finely

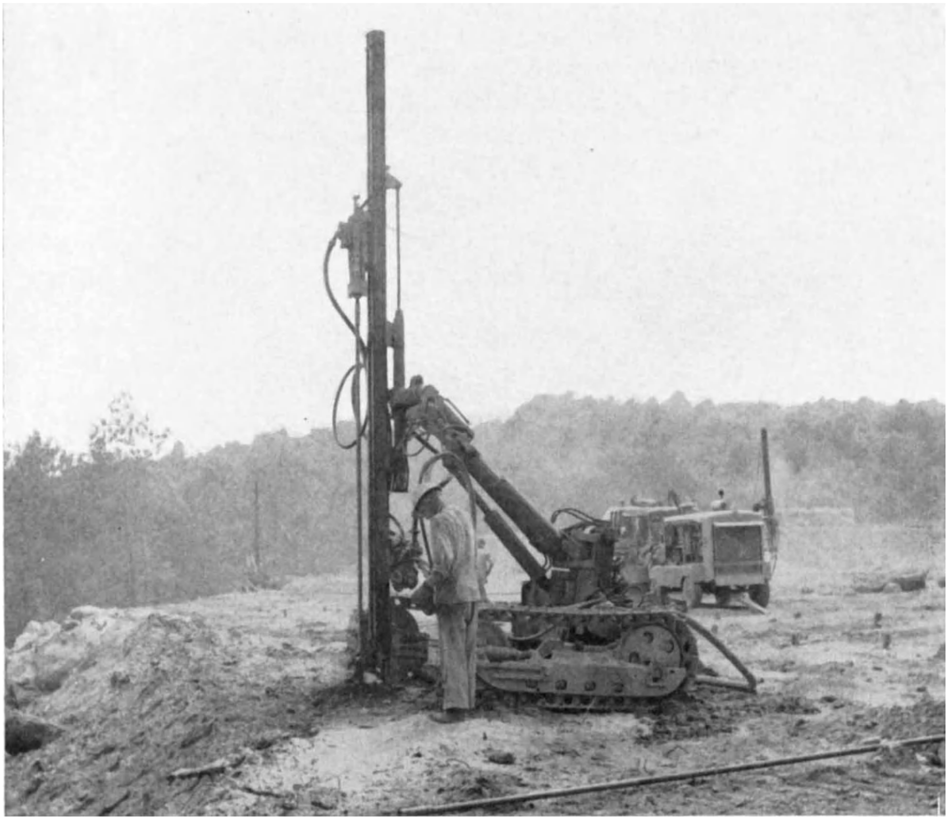


FIG. 2. Crawler mounted drill. (Ingersoll-Rand Co.)

disseminated iron-silica rock has a hardness of about 600 Brinell – harder than most steels and almost as hard as tungsten carbide.[2]

At the Peter Mitchell Pit of Reserve Mining Co., the original churn drill bits had to be resharpened after 6–12 in. of drilling and it took a churn drill and a four-man crew a full shift to drill 10–12 ft.

In the jet-piercing operation an oxygen–fuel oil flame is played on the bottom of the hole along with a water spray. The resulting heating and cooling action causes the rock to spall off in flakes which are carried out of the hole by the stream of escaping steam and exhaust gasses.

The operator of the piercing rig sits in a comfortable seat inside the rig and runs the machine from a control panel on which fuel and water consumption, burner temperature, burner rotation are instrumented. Generally the operator controls the piercing rate according to the rock resistance and fuel flow rates are kept constant. The piercer which can spall holes up to 50 ft deep, works at the rate of 4–10 in./min and produces an irregular-diameter corrugated shaped hole.

An advantage of the piercer is that almost any shape of hole can be produced. Chambers are formed at the bottoms of blast holes by lifting the burner 4 or 8 ft from the bottom of the hole and making two extra passes to enlarge the chamber to about 12 in.

TABLE 2. DATA ON JET-PIERCING RIGS, RESERVE MINING CO.[2]

Flame temperature	4500°F
Burner rotation	30 rpm
Weight	42 tons
Height to top of mast	69 ft
Power	2 25-h.p. motors
Power supply	440 V
Oxygen flow rate (original)	10,000 ft^3/hr
Water flow rate (original)	1000 gal/hr
Fuel oil flow rate (original)	38.8 gal/hr
Oxygen flow rate (present)	12,000 ft^3/hr
Water flow rate (present)	1200 gal/hr
Fuel oil flow rate (present)	49.5 gal/hr
Minimum hole diameter	$6\frac{1}{2}$ in.
Average hole diameter	$9\frac{1}{2}$ in.
Average chamber diameter	$12\frac{1}{2}$ in.
Length of chambers	4 to 8 ft
Average piercing speed, 1952	7.7 ft/hr
Average piercing speed, 1959	14.5 ft/hr
Average piercing speed, present	16.0 ft/hr

Overburden Drilling

A relatively new technique developed in Sweden and known as "overburden drilling" involves the sinking, by percussive-rotary drilling, of a drill casing through the overburden to where it seats in the underlying rock. A rotary percussion drill hole is then continued to the desired depth in the rock.

While the casing is being sunk through the overburden it is coupled to the drill rod and rotates and reciprocates with it. The rock bit on the end of the drill rod projects about an inch beyond the end of the ring bit with which the casing is fitted and acts as a pilot bit for the casing bit.

This equipment can sink holes through overburden containing cobbles, boulders, and fractured rock and once the casing is seated in solid rock the smaller hole can be extended into the rock.

Overburden drilling allows the underlying rock to be blasted without the necessity of first stripping overburden.

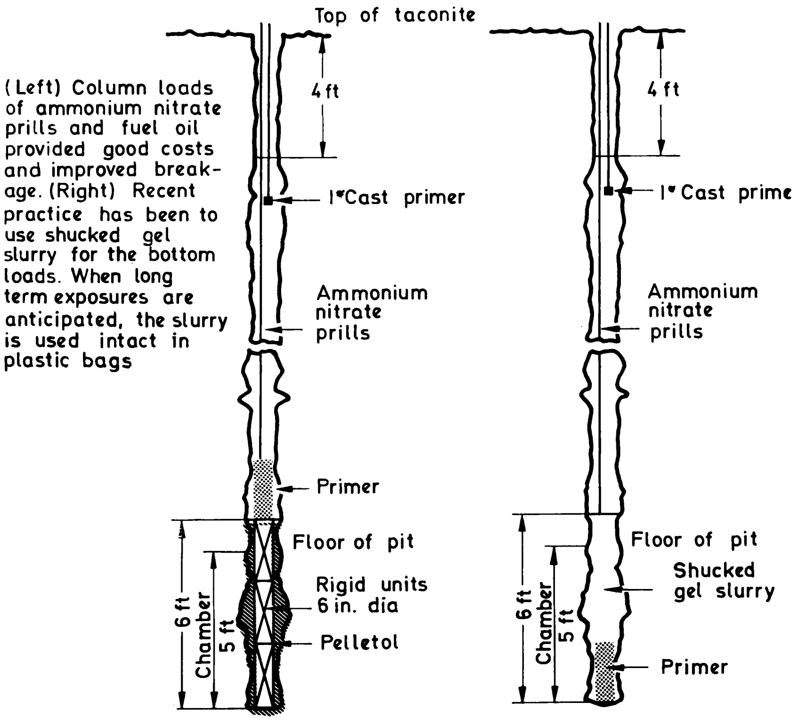


FIG. 3. Loading jet-pierced blast holes.[3]

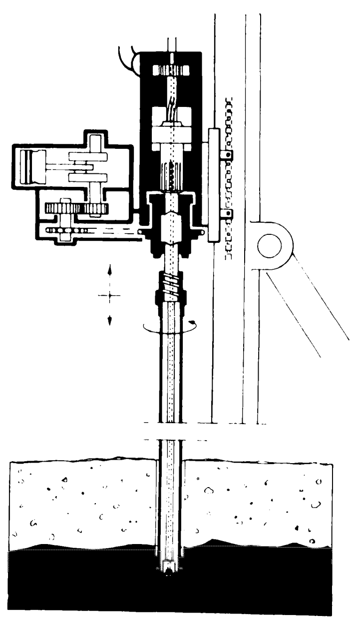


FIG. 4. Overburden drilling. (Atlas Copco.)

Secondary Drilling

For secondary drilling in the open-pit copper mines portable compressors and wagon drills have been largely replaced by self-contained units with the compressor and drill installed as integral parts.[1]

The drills are mounted on hydraulically controlled arms that can be adjusted to drill horizontal or vertical holes. The units are designed for high travel speed. The drilling performance of these machines is two or three times better than the performance of a jack-hammer crew. If the density of the boulders is extremely high, the drop-ball is used. Where adaptable, it is an efficient tool for reducing oversize boulders. In mines where only a few boulders occur mudcap or adobe shots are used.

BLASTING

Blasting Theory

Reflection Theory of Rock Breakage

Recent experimentation in the field of rock blasting indicates that most of the rock breakage which is produced by an explosive charge confined in a drill hole is the result of tensile strain "waves" reflected from the free surface of the rock rather than being the result of the push of expanding gases as is commonly assumed.[6, 6A]

Detonation of an explosive charge in a drill hole in rock creates a large quantity of gas at high temperature and pressure in a very short time. The gas pressure, acting against the rock and the column of stemming, generates a compressive stress (or strain) pulse which travels radially outward in the surrounding rock with a speed equal to, or greater than, that of sound in the rock. Associated with the radial compressive strain pulse is a tangential tensile strain pulse. These pulses are characterized by a single compression of short duration having a steep front or rapid rise time and a slower decay or fall time.

Near the explosion the amplitude of the compressive stress pulse is greater than the strength of the rock in compression, and the rock is crushed and fractured. As the pulse travels outward its amplitude decreases rapidly because of the divergence of the wave and absorption of energy by the rock. Thus the extent of the crushed zone is relatively small.

The gas pressure in the drill hole also acts against the stemming material compressing it and generating a compressive stress pulse which travels up the column of stemming. Because of loss of heat and increase of volume, mainly resulting from compaction of stemming material and crushing of rock, the gas pressure in the drill hole decreases rapidly from its peak value.

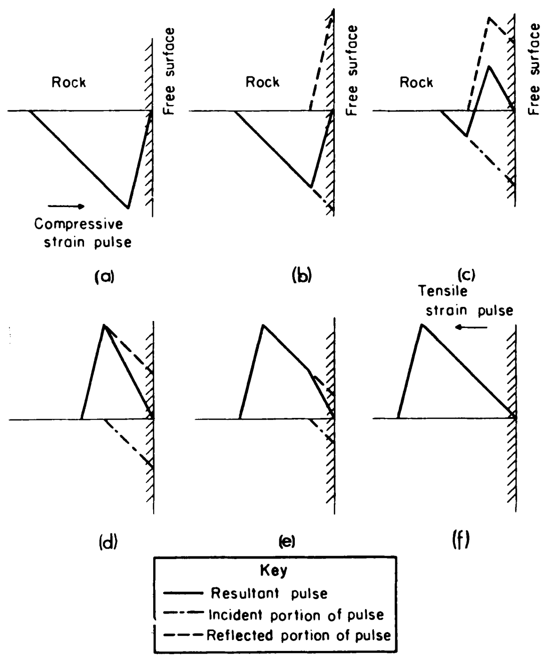


FIG. 5. Reflection of a triangular compressive strain pulse.[6]

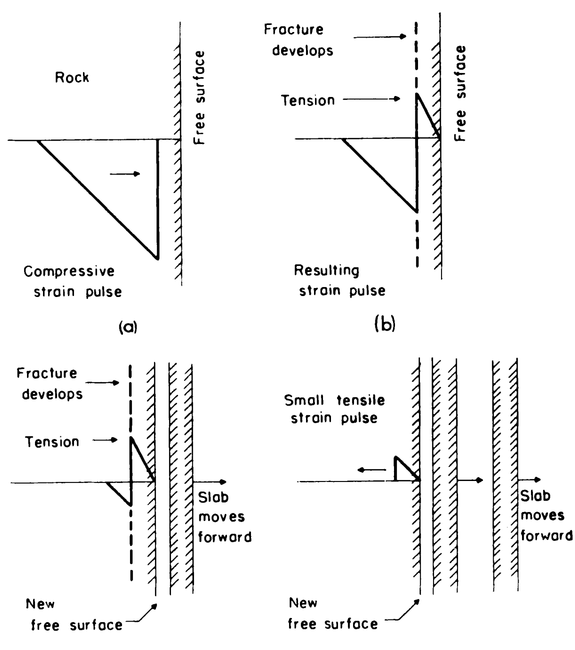


FIG. 6. Tensile fracture by reflection of a compressive strain pulse.[6]

When a longitudinal compressive stress impinges on a free surface, two reflected pulses are generated: (1) a reflected longitudinal tensile stress pulse, and (2) a reflected shear stress pulse. The amount of energy in each of these reflected stress pulses is a strong function of the angle of incidence of the impinging stress pulse. The effects of the reflected longitudinal pulse are manifested first, as the propagation velocity of longitudinal pulses is always greater than that of shear pulses. Also, as the strength of rock in tension is much less than the strength of rock in compression or shear this reflected tensile pulse is able to break the rock in tension as it moves back into the solid rock.

The reflection of a plane longitudinal pulse at a free surface is represented in Fig. 5. A triangular compressive strain pulse moving to the right impinges on a free surface. The dotted lines below the baseline represent the incident compressive pulse and the dotted lines above the baseline represent the reflected tensile pulse. The solid line is the resulting strain pulse in the medium. The maximum tensile strain is developed at a distance from the free surface equal to half of the full length of the incident compressive pulse.

The process of tensile fracture by reflection of a compressive strain pulse is illustrated by Fig. 6. When the resulting tensile strain pulse is equal to the tensile breaking strain of the rock a crack develops at the point of maximum tensile strain. This crack will act as a new free surface from which the impinging strain pulse will reflect. Also the slab of rock so produced will move forward because of the energy trapped in the slab in the form of particle velocity normal to the crack.

As the first slab moves forward and the new free surface reflects the remaining portion of the impinging compressive strain pulse the resulting tensile strain is again enough to crack the rock in tension. The process continues until the tensile strain developed is less than the tensile breaking strain of the rock, after which no further cracking will be produced.

Figure 7 shows test craters formed in chalk. These craters were formed entirely by surface spalling of the chalk as there was no breakthrough from the explosion chamber to the crater.

Formulae Can Help Plan Efficient Blasts*

Calculation of permissible burdens and the required amount of explosive for an efficient blast is still not a simple matter of inserting values in a standard, universal formula. But current research—especially on the reflection theory—appears to have tied the variables down rather closely and it may not be long before we do have such a formula. Nevertheless, at present there are several empirical guides that can aid planning.

* Reprinted from Ref. 6A.

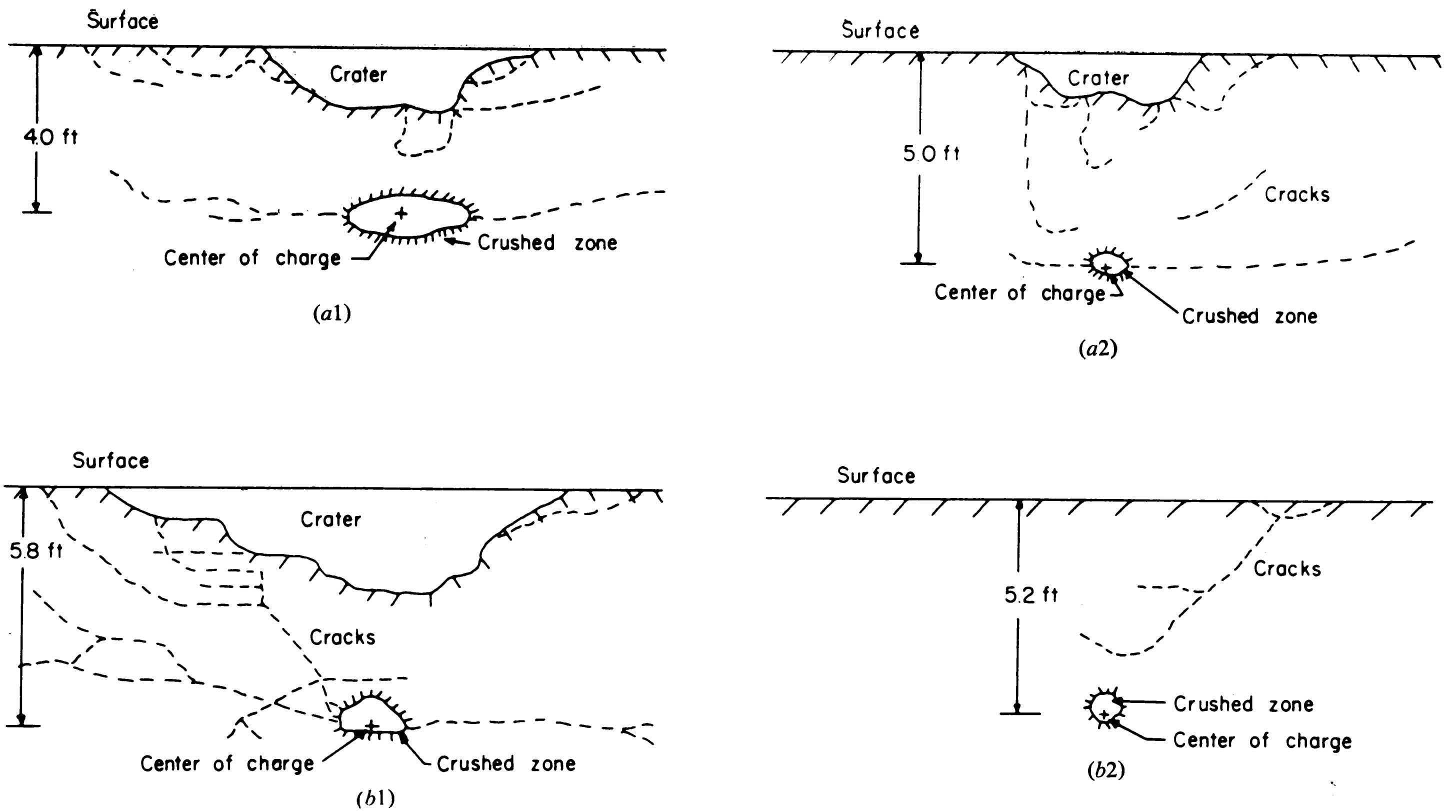


FIG. 7. Sections – horizontal-hole crater tests in chalk: (*a*1) 3.5 scaled charge depth, 1.5 lb. charge weight. (*b*1). 4.0 scaled charge depth, 3.0 lb. charge weight.
(*a*2) 5.4 scaled charge depth, 0.8 lb. charge weight. (*b*2) 7.0 scaled charge depth, 0.38 lb. charge weight.

A number of simple formulae have been based on the relationship that the quantity of explosive is directly proportional to the quantity of rock broken. Quantity is generally measured by weight, which is a function of volume, and volume, in turn, varies as the cube of a length. Thus the required weight, Q, of a single concentrated (i.e. crater) charge breaking to a plane surface will vary as the cube of the burden distance, B: $Q = KB^3$, where K is an experimentally determined numerical factor that takes care of variations in rock and explosive characteristics. For a column charge in a drill hole, the formula could be $Q = KB^2$ per ft and, knowing the density of the explosive, the size of hole required could be calculated. With such formulae it has been found that K for soft, tough rock will be about twice the K for hard, brittle rock. Likewise, an increase in the number of faces to which the charge can break will reduce K; thus K for a cube (six faces) is only one-fourth the K for a bench blast (two faces).

Other formulae have been based on incorrect but useful assumptions. Some are based on assuming that the blast will shear at a constant angle with the face—usually 45°—and give an explosive factor of 0.5–1.0 lb./ton. One bench blasting formula assumes that the blast must overcome the tensile strength of rock over the new face area plus the shear strength of the rock over the new floor area plus the frictional resistance of the whole block moving across the new floor area. And there is a coyote tunnel blasting formula based simply on shearing the new floor area using 2–2½ lb. of powder per square foot of floor area.

Probably the most direct formula is that of O. Anderson (Blast hole burden design, *Proc. Australian IMM*, No. 166–167, Melbourne, 1952). It is based on, and conforms to, a great number of underground and surface practices in a wide variety of ground all over the world. It is a very simple formula:

$$B = \sqrt{dL},$$

where B is burden (ft), d is diameter of blast hole (in.), and L is length of hole (ft) on single hole shots or when holes are at the recommended maximum spacing $S = 1.5B$. If a smaller spacing, S_2, is used then the new burden

$$B_2 = \frac{-S_2 + \sqrt{S_2^2 + 4A}}{2}$$

where $A = B^2 + SB$. If free face is less than $2B$ then the new burden should also be used: $B_2 = \sqrt{Bx}$, where x is half the length of the free face. To improve fragmentation—which is adversely affected by deep holes, wide spacing and large burden—burden may be reduced to two-thirds of that calculated.

A markedly similar formula can be derived from one which is used to establish a rock characteristic called resistance to blasting by K. H. Frankel (Factors influencing blasting results, *Manual on Rock Blasting*, Section 6 : 02, Stockholm, 1952).

Solved for maximum burden it gives:

$$B = \frac{RL^{0.3}l^{0.3}d^{0.8}}{50}$$

where B is burden in meters, R is resistance to blasting (experimental factor, one for difficult-to-break to six for very-easily-broken rock with two being a rough mean of frequent figures), L is length of hole in meters, l is length of charge in meters and d is diameter of hole (mm). It appears even more similar when the following "practical" values are used: reduce actual B to $0.8B$ and limit it to less than $2/3L$; use $l = 0.75L$; and keep spacing under $1.5B$. Frankel says this has been used since 1944 with 35 per cent LFB explosive, a common Swedish ammonia gelatin dynamite that is widely used in mining.

A burden formula based on physical characteristics of rock and explosive was offered by G. E. Pearse (Rock blasting, *Mine & Quarry Engineering*, London, January 1955). It is:

$$B = Kd\sqrt{\frac{P_s}{T}}$$

where B is critical radius (or maximum burden) in inches, K is a constant based on rock characteristics and varies from 0.7 to 1.0 with an average of 0.8 (it can be calculated using Poisson's ratio and the damping characteristic of the rock), d is the cartridge diameter (or hole diameter with proper confinement by tamping and stemming) in inches, P_s is reaction stability pressure of explosive (in psi) and T is ultimate tensile strength of rock (in psi).

Other formulae are worthy of review in a thorough study, but many become highly involved. One that has received some recognition was developed by U. Langefors (The calculation of charges for bench blasting and stoping, *Manual on Rock Blasting*, Section 6 : 05, Stockholm, 1954).

Vibration produced by a blast has come in for considerable study and a simple formula for planning purposes has been suggested by R. Westwater (Heading blasting, *Mine & Quarry Engineering*, London, July 1957). It is:

$$A = \frac{K\sqrt{Q}}{D}$$

where A is amplitude of vibration in 0.001 in., K is a ground factor varying from 100 in hard rock to 300 in wet rock or clay, Q is pounds of explosive and D is distance between blast and point of interest in feet. He mentions that amplitudes of 0.008 in. will not damage normal structures but that it should be kept below 0.003 to prevent human sensing and complaints. Other studies have emphasized the critical effect of acceleration rather than just amplitude.

Blasting Agents (See also page 457.)

Blasting agents used in open-pit copper mines usually range from 40 to 70 per cent equivalent strength, but the higher strengths ranging from 60 to 70 per cent are the most popular.(1)

Commercial ammonium nitrate, sensitized by the addition of a little diesel fuel or some other suitable petroleum product, has come into use since 1955 and is at present the blasting agent most widely used in surface mining operations. Procedures used with AN–FO blasting are described more fully in Chapter 1, "Strip Mining of Coal".

Generally it is true that the charge of maximum density will give the greatest amount of blast energy per unit volume.

In the case of ammonium nitrate (AN) and fuel oil (FO) mixtures this does not always hold true because the efficiencies of these explosions are influenced by the grain size of the mixture as well as the intimacy of the AN–FO mixture.

A less dense mix which is composed of porous AN prills capable of absorbing fuel oil may release more energy than a denser mix which is composed of impervious crystals which are only coated with fuel oil.

Table 3 and Figs. 8 and 9 show the characteristics of some AN–FO mixtures.

TABLE 3. CHARACTERISTICS OF AN–FO EXPLOSIVE
(Spencer Chemical Co.)

Particle size* U.S. standard sieve nos.	Number of tests	Charge density (g/cm^3)	Mean† detonation velocity (ft/sec)	Mean blast pendulum (°)
−8 to +10	10	0.94	9380	15.3
−10 to +12	10	0.96	10,000	14.8
−12 to +16	10	0.96	10,600	16.4
Hammermill Product	10	0.98	13,800	14.5
Standard N-IV Prill	6	0.88	8500	16.4

* All materials tested prepared from Spencer N-IV ammonium nitrate.
† Velocities measured in 2-in. diameter steel pipe.

Blasting Costs

Studies of the relative costs for breaking rock with various types of explosives and various blast hole sizes have been made in Sweden.(5)

Costs for using the pre-fabricated explosives dynamite, Ammonit, and Nabit were studied as well as the costs for using the "mix-it-yourself" types of explosives AN–oil and the AN–TNT slurries.

Nabit is a powder explosive in cartridge form while Ammonit is a pre-fabricated free running ammonium nitrate explosive.

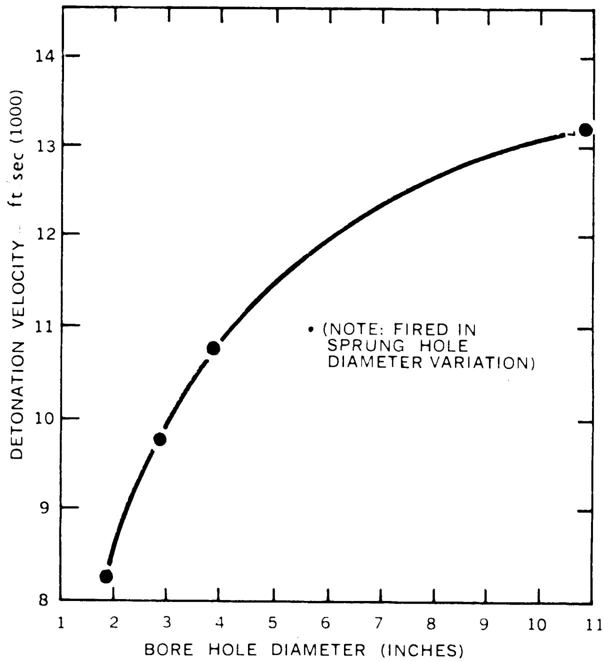


FIG. 8. Characteristics of AN–FO mixtures; detonation velocity vs. bore hole diameter. (Spencer Chemical Co.)

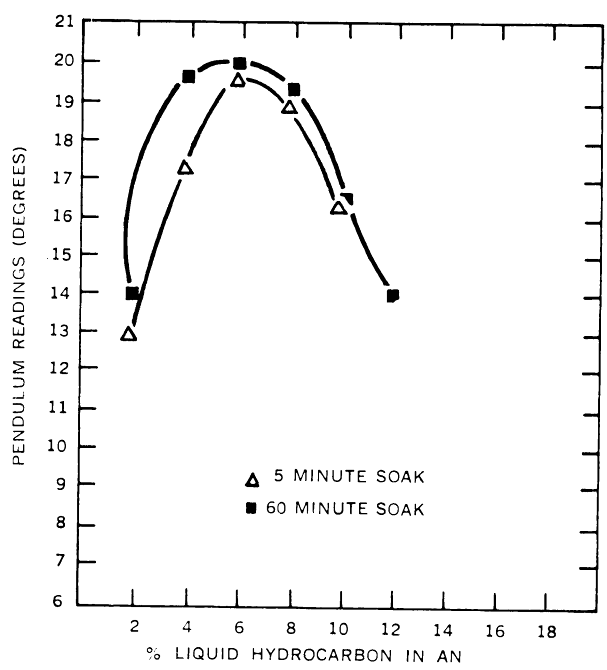


FIG. 9. Characteristics of AN–FO mixtures; blast output vs. oil content. (Spencer Chemical Co.)

The AN–oil explosives consist of fertilizer-grade ammonium nitrate mixed with several percent of fuel oil while the AN–TNT slurry consisted of a mixture of ammonium nitrate, TNT, and water in the proportions 65:20:15. AN–TNT slurry mixed in these proportions can be detonated successfully in holes more than 4 in. in diameter, but if slurries are to be used in smaller holes the percentage of TNT must be increased to 30–35 and this increases the cost of the explosive.

Table 4 (a) shows the relative costs of the various types of explosives used in open-pit mining in dollars per kilogram. Table 4 (b) shows the relative energy content and gas volume generated by each type of explosive. Table 4 (c) shows the strength

TABLE 4.[5]

(a) Cost of explosive in $ per kg.

Cost	Dynamite	Ammonit	AN–oil	AN–TNT-slurry	Nabit
Price	0.43	0.17	0.11	0.18	–
Freight, storage, transportation, mixing	0.07	0.03	0.02	0.02	–
Total	0.50	0.20	0.13	0.20	0.38

(b) Content of energy and gas volume

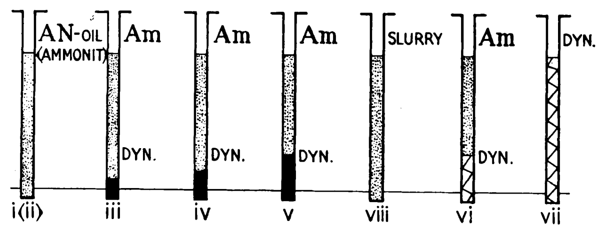
	Dynamite	Ammonit	AN–oil	AN–TNT–slurry	Nabit
Energy (kcal per kg)	1160 (1.00)	–	900 (0.78)	757 (0.65)	–
Volume of gas, (dm^3)	850 (1.00)	–	973 (1.15)	950 (1.12)	–
Calculated strength	1.00	0.87	0.84	0.74	0.90

(c) Weight strength and density

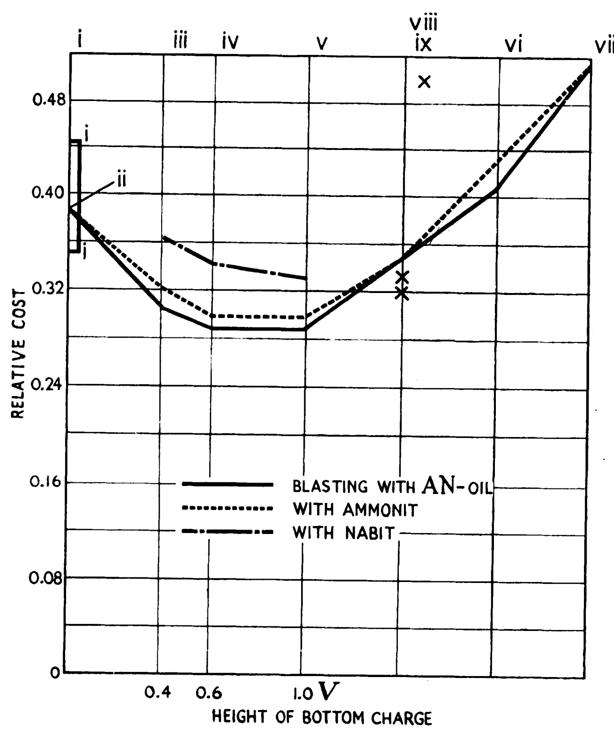
	Dynamite	Ammonit	AN–oil	AN–TNT–slurry	Nabit
Weight strength, experimental, s	1.00	0.89 ± 0.04	0.86 ± 0.04	(0.74 ± 0.04)	0.90
Density, ϱ	1.45	0.9	0.8–1.0	1.4	1.0
Density in drill hole,* ϱ			1.0		
(a) without loading device	1.0–1.2	1.0	(0.9 prills)		
(b) with loading device	1.5	1.1	1.2	1.55	1.0
$\varrho.s.$	1.0–1.5	1.02	1.08	1.21	0.90

* Calculated for a volume $\frac{\pi d^2}{4}$ per metre of hole depth.

per unit weight and the densities obtained when the explosives are loaded into the holes by pneumatic machines and when they are hand loaded and tamped. Also shown is the ϱs factor which is the strength per unit weight multiplied by the density to give a factor for strength per unit volume; it will be noted that the ϱs factor varies with the density obtained in the hole and that the density is greatest for pneumatically loaded holes.



(a)



(b)

FIG. 10. (a) Various alternatives for loading in bench blasting.
(b) Total relative cost for drill hole and explosive in different cases, (i) to (ix), according to (a). Drilling cost taken as \$0.60 per dm³. (Note: 1 ft³ is equal to 28.3 dm³; \$0.60 per dm³ is equivalent to \$1.50 per ft for 4-in. hole).[5]

The velocity of detonation of an explosive is also an important factor in determining its effectiveness in breaking extremely hard rocks. The gelatin dynamites have higher detonation velocities than the ammonium explosives and are somewhat more effective in delivering a "shattering" blow in hard and brittle rocks.

Blasting Costs

The cost of breaking a given tonnage of rock includes the cost of drilling the holes and loading them as well as the cost of the blasting agents. Langefors[5] has studied the cost effects of varying blast hole sizes and various combinations of blasting agents which may be used to load the holes. Figure 10 has been derived by assuming that it costs \$0.60 per dm^3 to drill blast holes and studying the effects of various combinations of blasting agents loaded into the holes. (Note: 1 ft^3 is equal to 28.3 dm^3; \$0.60 per dm^3 is equivalent to \$1.50 per ft for a 4-in. hole.)

It will be noted that the lowest blasting costs are achieved when the upper portion of a hole is loaded with an ammonium explosive (Am indicates either AN–oil or Ammonit since characteristics of both explosives are approximately the same) and the lower portion is loaded with dynamite in such manner that the dynamite is packed tightly to obtain a high density in the hole. Cases (vi) and (vii) illustrate the loss of efficiency when dynamite is loosely packed in the holes.

From these studies it appears that the most economic loading method is a concentrated bottom-charge of dynamite with a column of AN explosive above it. The studies also indicated that the dynamite charge could be substituted by an AN–TNT slurry for holes larger than 4 in. in diameter.

Effect of Hole Size and Bench Height on Breaking Costs

Figure 11 shows the effects of varying blast hole sizes and bench heights on breakage costs. In this case the drilling cost per unit volume of hole is taken to be proportional to $1/\sqrt{d}$. Costs are shown from holes with diameters from 2 to 12 in. and for bench heights of 33 and 50 ft.

Inclined Drilling and Blasting

Laboratory experiments by B. J. Kochanowsky[7] have demonstrated that an increased amount of rock can be broken with less explosive by using inclined rather than vertical blast holes.

In the basalt quarry of the Rock Hill Plant belonging to the General Crushed Stone Co., the powder factor was improved from 2.97 to 3.66 tons per pound of explosive when blast holes were inclined 20° from the vertical rather than being drilled vertical (in a 40-ft high bench). At the same time hole spacing was increased from 18 × 18 ft to 18 × 20 ft.

Figure 12[8] illustrates the transmission of energy through the rock from vertical and from inclined holes. With the inclined holes less explosive energy is wasted in the rock under the toe and more energy is used in breaking rock above the level of the floor.

Figure 13 shows the blast hole patterns and delays used at the Jeffrey Pit. Figure 14 shows details of the placement of explosives used in blast holes in various types of

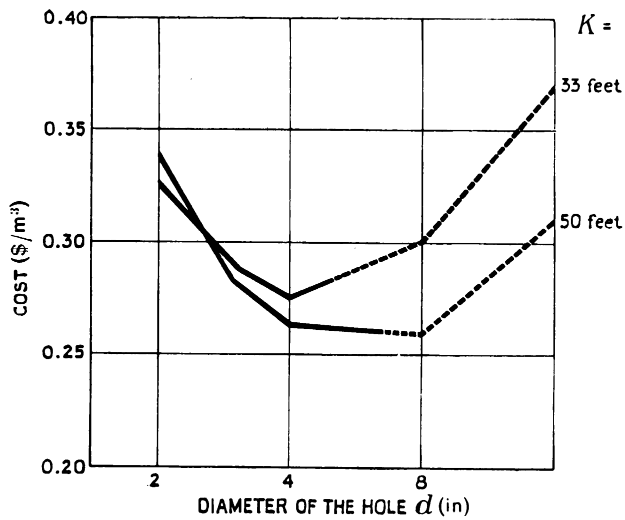


FIG. 11. An example of total cost of drill hole and explosive as a function of drill hole diameter, d, at different bench heights, K, and a drilling cost per volume of hole proportional to $1/d^{\frac{1}{2}}$. One row of holes assumed. Width of bench taken as $B = 40$ m (130 ft).[5]

rock. Holes are 4 in. in diameter and are drilled with percussion drills. Rock types vary from a soft serpentine or dunite in which drill penetration is 76 in./min to biotite granite in which penetration rate is 4 in./min.

Figure 15 is a nomograph for determining drilling and explosives costs.

Breakage Costs vs Handling Costs

The total costs for mining and processing a material will be the sum of drilling, blasting, loading, haulage, and crushing costs. It will frequently be found that closer spacing of drill holes, or heavier loading of the holes will produce better fragmentation of the rock so that costs of handling by loading shovels and crushing costs will be reduced by much more than the increased costs of drilling and blasting.

The McIntyre Development at Tawahus, N.Y. mines magnetite–ilmentite ore by open-pit methods, at the rate of 8500 tons/day and in addition handles about 8500 tons of waste daily.

In 1958 the angle hole was introduced with holes being drilled at an angle of 10° from the vertical. This resulted in a reduction of backbreak and reduction of production of oversize chunks. Blast holes were $6\frac{1}{2}$ in. in diameter and were drilled on 18-ft centers by down-the-hole drills.[9]

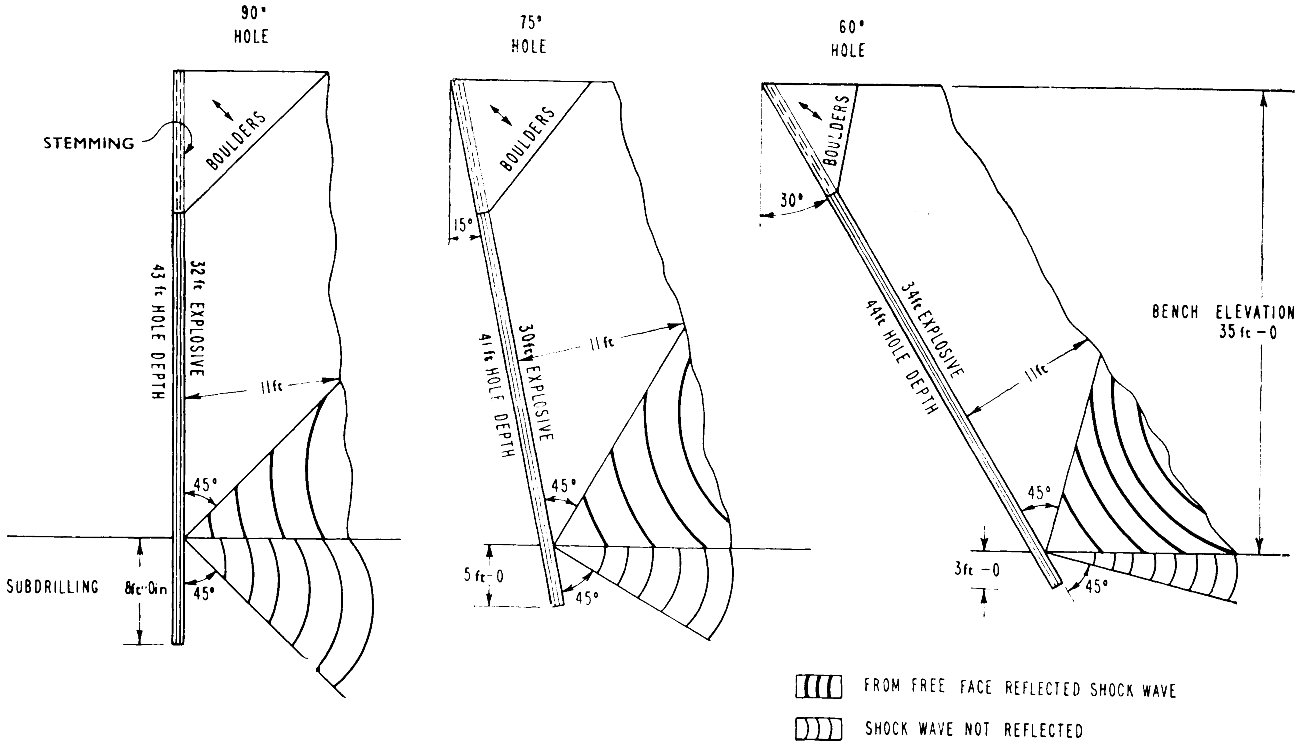
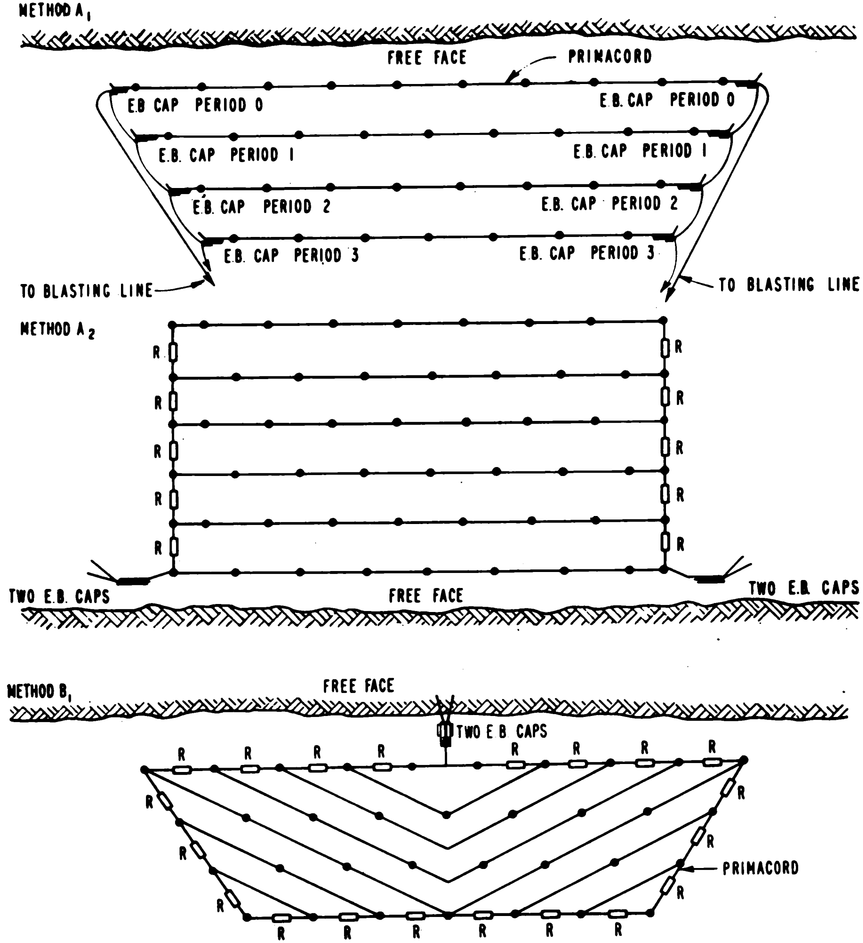


FIG. 12. At bench level, more shock waves are reflected in inclined holes; less subdrilling is needed; charge concentration is less.[8]


METHOD A1
FREE FACE
PRIMACORD
E.B CAP PERIOD 0
E.B. CAP PERIOD 0
E.B. CAP PERIOD 1
E.B. CAP PERIOD 1
E.B. CAP PERIOD 2
E.B. CAP PERIOD 2
E.B. CAP PERIOD 3
E.B. CAP PERIOD 3
TO BLASTING LINE
TO BLASTING LINE
METHOD A2
R
TWO E.B. CAPS
FREE FACE
TWO E.B. CAPS
METHOD B1
FREE FACE
TWO E.B. CAPS
R
PRIMACORD

(a)

(b)

(c)

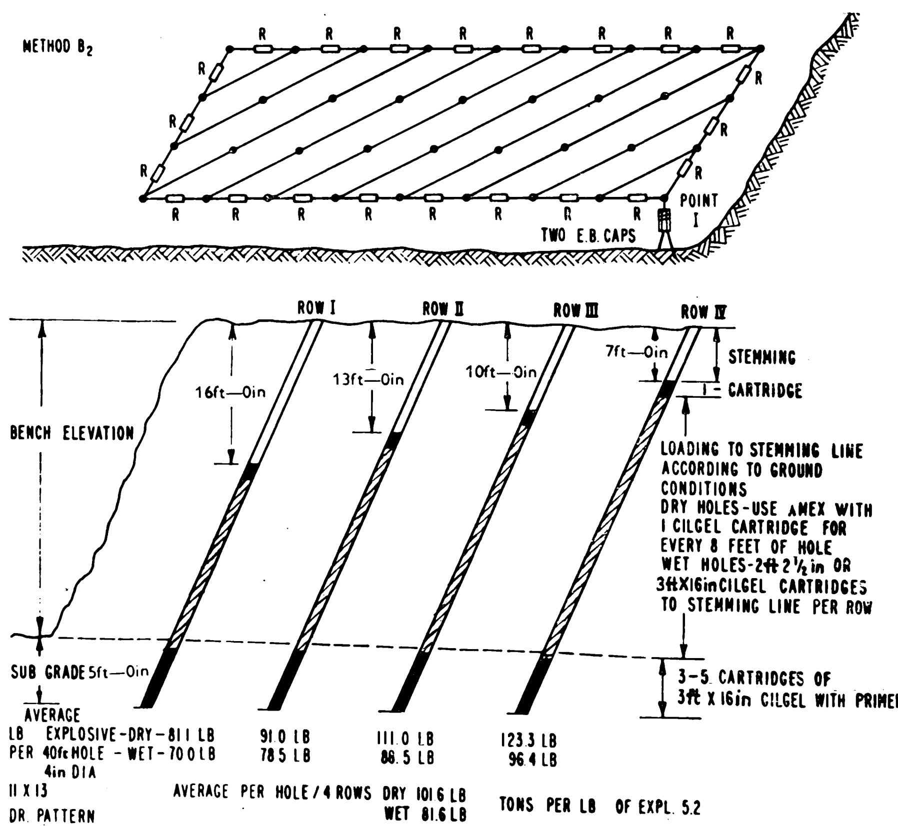
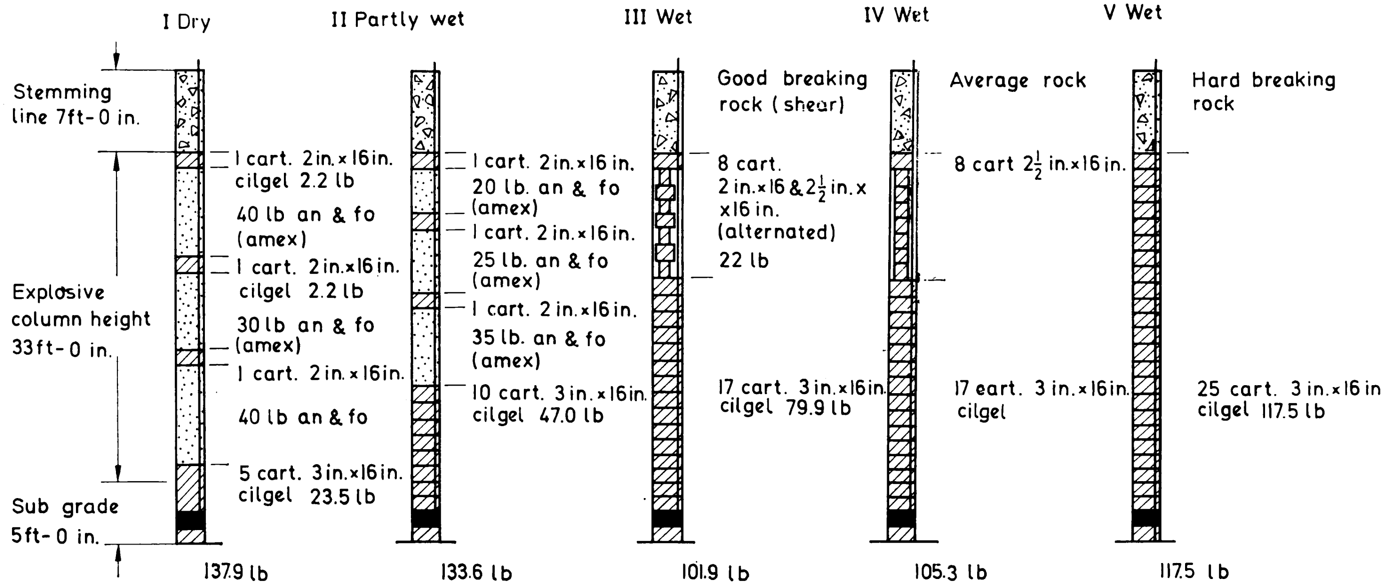


FIG. 13. (a) Parallel pattern wiring methods for 11 × 13 ft hole spacing; method A_1 is standard; method A_2 is multirow using millisecond connectors; both are initiated at both ends using eb caps.

(b) Patterns for 11 × 13 ft hole spacing. B_1 is a V-blast, initiated at centers; B_2 is an "end" blast, initiated at point *I*. Both are diagonal shots, using millisecond connectors (relays).[8]

(c) Cross-section of bench showing typical hole loading.[8]



I

110 lb. AN and FO	at $ 0.09/lb	=	$ 9.90
30 lb. Cilgel	at $ 0.18/lb	=	$ 5.40
50 ft Primacord	at $ 0.04/ft	=	$ 2.00
Labor	at $ 0.60/hole	=	$ 0.60
	Total cost/hole		$17.90

II

80 lb. Amex	=	$ 7.20
53.6 lb. Cilgel	=	$ 9.65
50 ft Primacord	=	$ 2.00
Labor	=	$ 0.60
		$19.45

III

101.9 lb. Cilgel	=	$18.34
50 ft Primacord	=	$ 2.00
Labor ...	=	$ 0.60
		$20.94

IV

105.3 lb. Cilgel	=	$18.95
50 ft Primacord	=	$ 2.00
Labor ...	=	$ 0.60
		$21.55

V

117.5 lb. Cilgel ...	=	$21.15
50 ft Primacord	=	$ 2.00
Labor ...	=	$ 0.60
		$23.75

FIG. 14. Explosive loading methods – fourth and succeeding lines: Costs are highest in the fourth and succeeding lines which are loaded with 33 ft of explosives. A more concentrated charge is provided in the bottoms of holes. Figures are valid as of May, 1959.[8]

In 1959 it was decided to attempt to improve fragmentation in order to reduce rock handling and crushing costs. Accordingly hole spacing was reduced to a 11-ft spacing with a 15-ft burden. This resulted in increased drilling costs but allowed the use of a larger proportion of low cost ammonium nitrate explosive (increased

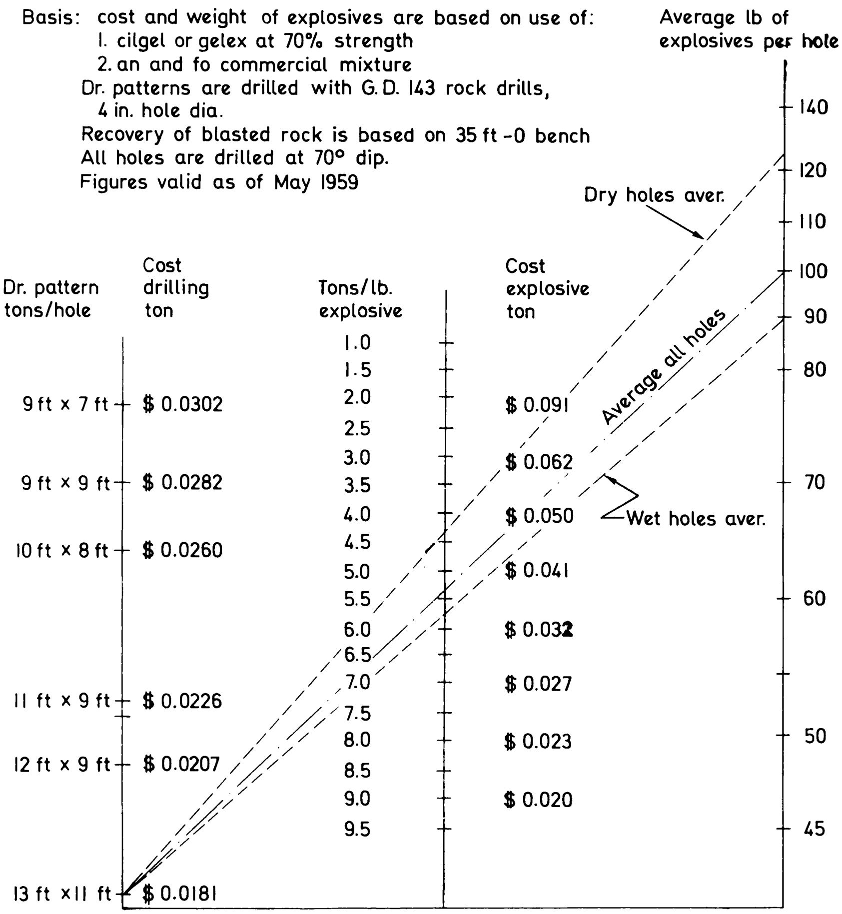


FIG. 15. Nomograph for determining cost of drilling and explosives at Jeffrey Pit.[8]

from 38 to 63 per cent of total used) and reduced the overall explosive costs in spite of the fact that the powder factor increased from 0.270 to 0.329 lb./long ton.

With the better fragmentation the crusher production increased from 383 to 472 long tons/hour. A capital expenditure of more than $1 million would have been required to obtain a similar increase in capacity on the basis of the broken ore previously delivered to the crusher.

Table 5(a) to (d) shows the capacity increases achieved and the comparative costs before and after the change in blasting procedures.

TABLE 5.[9]

(a) Crusher production statistics

	1957	1958	1959	1960
Tons per operating hour	452	436	470	570
Tons per scheduled hour	328	357	383	472
Mine controllable delay (% of scheduled time)				
Ore delivery delay	11.6	5.9	7.4	7.4
Jaw plug delay	6.1	7.7	6.6	3.3
Delay within crusher plant	9.6	4.4	4.7	6.5
Total delay	27.3	18.0	18.7	17.2

(b) Comparative drill, breakage, and crusher costs per ton*

Phase	1957	1958	1959	1960
Drilling	112	100	98	129
Blasting and dropball	114	100	100	90
Crushing	104	100	96	81
Total	107	100	97	90

(c) Comparative crusher performance and cost*

	1957	1958	1959	1960
Comparative performance (tons per scheduled hour)	92	100	107	132
Comparative cost per scheduled hour	95	100	103	107
Comparative cost per ton	104	100	96	81

(d) Ore shovel performance and cost

	1957	1958	1959	1960
Shovel performance (tons per scheduled (hour)	180	184	211	229
Comparative performance*	98	100	115	124
Comparative cost per hour*	100	100	102	109
Comparative cost per ton*	102	100	90	88

* Base: 1958 cost and performance = 100.

ROCK RIPPING

A large proportion of the sedimentary rocks can be broken by ripping with a "tooth" towed behind a heavy crawler tractor. This method has been used in construction and quarrying operations in limestones, sandstones, and shales. The metamorphic and igneous rocks are generally much harder than the sedimentary rocks and are not generally susceptible to ripping unless they have been weakened by weathering or unless they contain numerous planes of weakness such as joints, and shear or cleavage planes.

Generally the greater the number of natural planes of weakness the rock contains the more susceptible it is to ripping although the orientation of the planes of weakness also effects its resistance to ripping.

Table 6 shows ripping costs as compared with drilling and blasting costs on some applications in the U.S.

TABLE 6. RELATIVE COSTS OF RIPPING VS. DRILLING AND BLASTING ON APPLICATIONS IN THE U.S.A.[10]

Material	Ripping production (yd^3/hr)	Ripping cost (cts./yd^3)	Drilling and blasting cost (cts./yd^3)
Limestone	196	11.5	19.3
Limestone	250	7.3	17.3
Limestone	350	5.2	15.1
Limestone	460	6.5	—
Sandstone	300	8.5	15.7
Sandstone	333	5.7	13.8
Sandstone	400	15.0	30.0
Sandstone	1000	2.1	11.7

(Source: Caterpillar Tractor Company, Peoria.)

LOADING ORE AND WASTE

In the open-pit copper mines ore and waste are loaded by full-revolving electric shovels. At most mines shovels with a 4–6 yd^3 capacity are used, but larger mines have shown a tendency to use 7–10 yd shovels.[1]

The 4–6 yd shovels generally load from 3000 to 6000 tons/operating shift, and the larger shovels load as much as 15,000 tons/operating shift. The larger shovels are particularly useful in waste areas where large boulders can be loaded and dumped without secondary blasting. Ore boulders must be broken to a size that the crushing facilities can handle.

ESTIMATION OF SHOVEL AND DRAGLINE OUTPUT FOR SYSTEMS ANALYSIS[11]

The following article is a reprint of a paper prepared by Elmer R. Drevdahl, Associate Professor, College of Mines, University of Arizona, which was titled "Estimation of shovel and dragline output for systems Analysis", and which appeared in *Proceedings of Symposium of Surface Mining Practices*, University of Arizona, October, 1960.[11]

There are many ways put forth in current literature to obtain the approximate output of a shovel or dragline under various conditions. The method developed in this paper uses a basic formula for the calculation of shovel output which includes all the necessary multipliers to correct theoretical output to approximate actual output under the operating conditions that prevail. This method allows one to estimate output using the standard multipliers given in the reference material in this paper. More accurate results can be obtained by determining the value of various multipliers for a specific operation by physical test and time studies of the operation. The following formulas (1) and (2) are used to determine output in "loose" measure in cubic yards per hour, or in "bank" or "solid" measure in cubic yards per hour. Formulas (3), (4), and (5) are used in calculating the approximate number of trucks needed to service a shovel or dragline with an output as computed in formulas (1) or (2).

It will be necessary to give a detailed explanation of various components of the output formulas and to explain the use of reference data included before one can use the formulas easily.

Shovel and Dragline Output Formula

$$\text{Loose measure yd}^3/\text{hr} = \frac{(3600)(C_d)(E)(F)(D)(A)}{t_s} \tag{1}$$

$$\text{Bank measure yd}^3/\text{hr} = \frac{(3600)(C_d)(E)(F)(D)(A)(S)}{t_s} \tag{2}$$

Truck Shovel Ratios

$$\text{Number shovel dippers/truck, } n = \frac{C_t}{(C_d)(F)} \tag{3}$$

$$\text{Corrected truck cycle time, } t_t = \frac{\dfrac{d_h}{V_1} + t_1 + \dfrac{d_r}{V_2} + t_2}{E} \tag{4}$$

$$\text{Number trucks/shovel} = 1 + \frac{60(t_t)(A)}{(n)(t_s)} \tag{5}$$

where

C_d = dipper or bucket capacity (yd^3);
C_t = truck capacity (yd^3);
D = depth of cut correction;
E = efficiency factor (time utilization factor);
F = fill factor, e.g. dipper or bucket efficiency;
S = swell factor;
V_1 = maximum haul speed times speed factor (average speed) (ft/min);
V_2 = maximum return speed times speed factor (average speed) (ft/min);
d_h = haul distance (ft);
d_r = return distance (ft);
n = shovel passes required to fill truck;
t_s = shovel or dragline cycle time (sec);
t_t = approximate corrected truck cycle time (min);
t_1 = turning and dumping time (min);
t_2 = spotting time (min);
A = angle of swing correction (from 90° standard).

Explanation of Output Formulas

(3600): This number divided by the cycle time in seconds gives the number of cycles per hour that the shovel or dragline will make. C_d is the dipper or bucket capacity of the shovel in cubic yards. E is the efficiency factor which corrects the shovel's working time from a theoretical 60 min per hr to the estimated actual working time. In most operations there are one or two factors involved and the value for E is a combination of these factors. Table 7 gives the approximate equipment efficiency obtainable under various conditions. Table 8 lists the effect of management on the job when the shovel is operating as part of a materials handling system and not as an independent unit. When the shovel is operating as an independent unit, such as in stripping operations where the material is cast and not loaded into trucks the efficiency factor in Table 7 is used. When the shovel is part of a system both factors must be considered. Table 9 gives the value of the combined factors of Table 7 and Table 8. The values from Table 9 are applicable to most conditions found in mining operations. Table 10 gives the conversion of actual working time per hour to efficiency and this table may be used to convert time study averages to values of E.

TABLE 7. EQUIPMENT OPERATING EFFICIENCY(11)

	Percent	Factor
Good	90	0.90
Average	80	0.80
Poor	70	0.70

TABLE 8. JOB-MANAGEMENT EFFICIENCY(11)

	Percent	Factor
Good	100	1.0
Average	85	0.85
Poor	65	0.65

TABLE 9. TABLE OF COMBINED EFFICIENCY FACTORS(11)

Equipment operating efficiency	Job-management efficiency		
	Good	Average	Poor
Good	0.90	0.77	0.59
Average	0.80	0.68	0.52
Poor	0.70	0.60	0.45

TABLE 10. MINUTES OPERATING TIME TO OPERATING EFFICIENCY(11)

Minutes	% Efficiency	Minutes	% Efficiency
60	100	35	58
55	92	30	50
50	83	25	42
45	75	20	33
40	67	15	25

TABLE 11. TABLE FOR ESTIMATION OF JOB-MANAGEMENT EFFICIENCY[11]

Job elements to consider	Relative value job-management efficiency 25:45:65 Poor	75:80:85 Average	90:95:100 High
1. *General economy* (local)	*prosperous*	*normal*	*hard times*
business	vigorous	normal	depressed
production volume	high	normal	low
unemployment	low	normal	high
2. *Labor supply*	*poor*	*average*	*good*
training	poor	average	good
pay scale	low	average	good
available workers	scarce	normal	surplus
3. *Supervision*	*poor*	*average*	*good*
training	poor	average	good
pay scale	low	average	good
availability	scarce	normal	surplus
4. *Job conditions*	*poor*	*average*	*good*
management	poor	average	good
location or operation	unfavorable	average	favorable
workmanship required	first rate	regular	passable
length of operation	short	average	long
5. *Weather conditions*	*bad*	*fair*	*good*
rain	much	some	occasional
cold	bitter	moderate	occasional
heat	oppressive	moderate	occasional
snow	much	some	occasional
6. *Equipment availability*	*poor*	*normal*	*good*
applicability	poor	normal	good
condition	poor	fair	good
maintenance, repairs	slow	average	quick
7. *Delays*	*numerous*	*some*	*minimum*
job flexibility	poor	average	good
delivery of supplies	slow	normal	prompt
expediting	poor	average	good

In many situations there may not be adequate production records of time study information on which to estimate job-management efficiency factors, and to this end Table 11 has been prepared as an estimating guide. The table and example calculations are self-explanatory so no further explanation will be given.

(*F*): The fill factor is commonly called the dipper factor for shovels or the bucket factor for draglines. This factor is the approximate load the dipper actually is carrying expressed as a percentage of the rated capacity. Digging conditions are usually classified as easy, medium, medium hard, or hard digging. Table 12 lists the approximate fill factors for shovels and draglines under various digging conditions. The table also lists examples of the types of material that come under the various classifications. It can be seen from the table that the various types of blasted rock come under hard digging and that there is a wide range in the factors. This range can be explained by the relationship of the size of the shovel to the blasted material. If the size of the blasted rock remains essentially the same, the digging efficiency of the shovel will increase as the size of the shovel increases. Table 16 lists the approximate dipper factors in quarry production for various shovel sizes. In a detailed analysis actual field tests should be made to determine the average fill factors for a specific job.

Use of Approximate Job-management Efficiency Table

Example

The average value for job efficiency is determined after careful consideration of the seven production elements in the following manner:

(1) Develop a relative value for each of the seven production elements.

(2) Obtain overall efficiency by averaging the seven production elements.

(3) The job efficiency factor will indicate the approximate correction to output necessary to allow for the general efficiency of the overall operation and management. The correction is in addition to the correction for operating efficiency.

Example Calculation:

(1) General economy	80*	
(2) Labor supply	75	
(3) Supervision	85	
(4) Job conditions	60	
(5) Weather conditions	70	
(6) Equipment availability	75	
(7) Delays	80	
	7/525	
	75	Job efficiency

*General economy

Business	90
Production volume	70
Unemployment	80
	3/240
	80

TABLE 12. APPROXIMATE DIPPER AND BUCKET EFFICIENCY FOR VARYING CLASSES OF MATERIALS
Conditions: Digging face of sufficient length to allow dipper or bucket to obtain load as given. Allowance must be made for smaller dipper or bucket loads when digging in shallow bank, by application of height of bank factor.[11]

Easy Digging Shovel dipper factor 95–110 per cent Dragline bucket factor 95–110 per cent Loose, soft, free running materials. Close lying, which will fill dipper or bucket full.	*Medium Digging* Shovel dipper factor 85–90 per cent Dragline bucket factor 80–90 per cent Harder materials that are not difficult to dig without blasting but break up with bulkiness, causing voids in the dipper or bucket.
Easy Digging Dry sand or small gravel Moist sand or small gravel Loam Loose earth Muck Sandy clay Loose clay gravel Cinders or ashes Bituminous coal	*Medium Digging* Clay—dry or wet Coarse gravel Clay gravel, packed Packed earth Anthracite coal
Medium Hard Digging Shovel dipper factor 70–80 per cent Dragline bucket factor 65–75 per cent	*Hard Digging* Shovel dipper factor 50–75 per cent Dragline bucket factor 40–65 per cent
Materials requiring some breaking up by light blasting or shaking. More bulky and somewhat hard to penetrate, causing voids in dipper or bucket.	Blasted rock, hardpan, and other bulky materials, which cause considerable voids in dipper or bucket and difficult to penetrate or load.
Well broken limestone, sand rock and other broken rocks. Blasted shale Ore formations (not of rock character) requiring some blasting. Heavy, wet, sticky clay Gravel with large boulders Heavy, wet gumbo Cemented gravel	Hard, tough shale Limestone Trap rock Granite Sandstone Taconite Conglomerate Caliche rock All above blasted to large pieces mixed with fines and dirt. Tough, rubbery clay that shaves from bank.

(*D*): This factor is a correction factor to be used when a shovel or dragline is working at a depth of cut other than optimum. The optimum depth of cut is that depth of cut which produces the greatest output for a given shovel size and type of material. The optimum depth of cut does not have any relationship to the maximum digging range of the machine. In practically all mining operations the blasted bench is considerably higher than the optimum depth of cut and the blasted bench approaches optimum loading conditions. If the higher bench does not hang up, no depth of cut correction is necessary and a value of 1.0 is used for (D). Table 13 lists the optimum depth of cut for various sizes of shovels and draglines in different types of materials. Most shovels of a size larger than that listed in the table are used in blasted rock and require no correction or they are custom designed machines tailored to a specific job and the correction factor is not applicable.

TABLE 13. OPTIMUM DEPTH OF CUT (IN FEET)*[11]

Class of material		Dipper or bucket capacity (yd^3)								
		$\frac{3}{8}$	$\frac{1}{2}$	$\frac{3}{4}$	1	$1\frac{1}{4}$	$1\frac{1}{2}$	$1\frac{3}{4}$	2	$2\frac{1}{2}$
Moist loam or light sandy clay	Shovel	3.8	4.6	5.3	6.0	6.5	7.0	7.4	7.8	8.4
	Dragline	5.0	5.5	6.0	6.6	7.0	7.4	7.7	8.0	8.5
Sand and gravel	Shovel	3.8	4.6	5.3	6.0	6.5	7.0	7.4	7.8	8.4
	Dragline	5.0	5.5	6.0	6.6	7.0	7.4	7.7	8.0	8.5
Common earth	Shovel	4.5	5.7	6.8	7.8	8.5	9.2	9.7	10.2	11.2
	Dragline	6.0	6.7	7.4	8.0	8.5	9.0	9.5	9.9	10.5
Hard tough clay	Shovel	6.0	7.0	8.0	9.0	9.8	10.7	11.5	12.2	13.3
	Dragline	7.3	8.0	8.7	9.3	10.0	10.7	11.3	11.8	12.3
Clay, wet, sticky	Shovel	6.0	7.0	8.0	9.0	9.8	10.7	11.5	12.2	13.3
	Dragline	7.3	8.0	8.7	9.3	10.0	10.7	11.3	11.8	12.3
Blasted rock	In practically all mining operations the blasted bench approaches optimum loading conditions and no optimum depth of cut correction is necessary and the value for 100% of optimum is used.									

* Table adapted from tables published by the Power Crane and Shovel Association.

(A): This is the correction factor to correct the shovel cycle time to swing conditions other than 90°. Most shovel data list the shovel cycle time on the basis of a standard swing of 90°, and, with loading conditions other than this, one must correct the estimate of output to reflect the increase or decrease in output resulting from a shorter or longer swing time. Table 14 lists the correction factor for angle of swing and depth of cut other than optimum. Either Table 14 can be used to find a combined (DA) factor or each correction can be determined separately.

(S): This is called the swell factor and it is a function of the percent increase in volume in a material from the "in place" to the "loose" state. In many operations it is desirable to compute equipment output in "bank" or "solid" measure even though the machine is working with loose material. When the bank measure output is desired it is necessary to multiply the output by a swell factor to determine the bank yards that are equivalent to the loose measure output of the machine. The swell factor

$$(S) = \frac{100}{100 + \text{percent swell}} \quad \text{or} \quad (S) = \frac{\text{loose unit weight}}{\text{in place unit weight}}$$

Table 15 lists the approximate in place unit weight, loose unit weight, percent swell, and swell factors for a variety of materials. This table can be used as a guide to estimate the characteristics of various materials; however, actual field determinations will be more reliable for detailed computations on a specific job.

TABLE 14. EFFECT OF DEPTH OF CUT AND ANGLE OF SWING ON POWER SHOVEL AND POWER DRAGLINE OUTPUT*

Conversion Factors

Draglines

Depth of cut in % of optimum	Angle of swing (in degrees)							
	30°	45°	60°	75°	90°	120°	150°	180°
20	1.06	0.99	0.94	0.90	0.87	0.81	0.75	0.70
40	1.17	1.08	1.02	0.97	0.93	0.85	0.78	0.72
60	1.24	1.13	1.06	1.01	0.97	0.88	0.80	0.74
80	1.29	1.17	1.09	1.04	0.99	0.90	0.82	0.76
100	1.32	1.19	1.11	1.05	1.00	0.91	0.83	0.77
120	1.29	1.17	1.09	1.03	0.985	0.90	0.82	0.76
140	1.25	1.14	1.06	1.00	0.96	0.88	0.81	0.75
160	1.20	1.10	1.02	0.97	0.93	0.85	0.79	0.73
180	1.15	1.05	0.98	0.94	0.90	0.82	0.76	0.71
200	1.10	1.00	0.94	0.90	0.87	0.79	0.73	0.69

Shovels

Depth of cut in % of optimum	Angle of swing (in degrees)						
	45°	60°	75°	90°	120°	150°	180°
40	0.93	0.89	0.85	0.80	0.72	0.65	0.59
60	1.10	1.03	0.96	0.91	0.81	0.73	0.66
80	1.22	1.12	1.04	0.98	0.86	0.77	0.69
100	1.26	1.16	1.07	1.00	0.88	0.79	0.71
120	1.20	1.11	1.03	0.97	0.86	0.77	0.70
140	1.12	1.04	0.97	0.91	0.81	0.73	0.66
160	1.03	0.96	0.90	0.85	0.75	0.67	0.62

* From the Power Crane and Shovel Association.

TABLE 15. APPROXIMATE MATERIAL CHARACTERISTICS*[11]

Material	Weight in place per cubic yard (lb.)	Percent of swell	Swell factor	Loose weight per cubic yard (lb.)
Clay, dry	2300	25	0.80	1840
Clay, light	2800	30	0.77	2160
Clay, dense, tough or wet	3000	33	0.75	2250
Coal, anthracite	2200	35	0.74	1630
Coal, bituminous	1900	35	0.74	1400
Earth, dry	2800	25	0.80	2240
Earth, wet	3370	25	0.80	2700
Earth and sand and gravel	3100	18	0.85	2640
Earth and rock mixture such as unclassified excavation	2500–3000	30	0.77	1920–2310
Gravel, dry	2450	10–15	0.87–0.74	1250
	to			to
Gravel, wet	3900	10–15	0.91–0.87	3550
Earth (loam or silt) dry	1700	15–35	0.87–0.74	1250
	to			to
Earth (loam or silt) wet	3500	25	0.80	2800
Rock, hard, well blasted	4000	50	0.67	2680
Rock and stone, crushed	3240–3920	35	0.74	2400–2900
Shale or soft rock	3000	33	0.75	2250
Ashes, hard coal	700–1000	8	0.93	650–930
Ashes, soft coal with clinkers	1000–1515	8	0.93	930–1410
Ashes, soft coal, ordinary	1080–1215	8	0.93	1000–1130
Bauxite	2700–4325	33	0.75	2020–3240
Concrete	3240–4185	40	0.72	2330–3000
Granite	4500	50–80	0.67–0.56	3000–2525
Gypsum	4300	30	0.77	3300
Iron ore, hematite	6500–8700	67–122	0.60–0.45	3900
Kaolin	2800	30	0.77	2160
Limestone blasted	4200	67–75	0.60–0.57	2400–2520
Limestone, marble	4600	67–75	0.60–0.57	2620–2760
Mud, dry (close)	2160–2970	20	0.83	1790–2460
Mud, wet (moderately packed)	2970–3510	20	0.83	2470–2910
Sand, dry	2200–3400	10–15	0.91–0.87	1900–3100
Sand, wet	2450–3900	10–15	0.91–0.87	2150–3550
Sandstone	4140	40–60	0.72–0.63	2980–2610
Shale, riprap	2800	33	0.75	2100
Slate	4590–4860	30	0.77	3530–3740
Trap rock	5000	50	0.66	3300

* The weight and swell factor of a material will vary with such factors as grain size, moisture content, degree of compaction, etc. If an exact material weight must be determined, then a test must be run on that particular sample.

(t_s): This is the average time in seconds necessary for one complete cycle of the shovel or dragline with a swing of 90°. Tables 16 and 17 list the average cycle time on a 90° swing for various types of digging conditions. In Table 17 you will note that for the larger size draglines the cycle time is given for a swing of 110°. In dragline operations the larger size units are used generally in casting operations, and for this work the swing usually approaches 110° so cycle times are listed for this swing. It should be understood that these cycle times are only approximate and that actual measured average times for a specific job will give more accurate results.

TABLE 16. SHOVEL SELECTION GUIDE[11]
Approximate Power Shovel Production (Bank Measure)

Dipper size (yd³)	Cycle time (sec) 90° swing — Digging conditions			Production yd³/hr efficiency 80% — Digging conditions			Quarry production yd³/hr efficiency 80%		
	Easy	Medium	Hard	Easy	Medium	Hard	Dipper factor (%)	Cycle time	Production
0.375	14	19	24	59	34	18			
0.5	14	19	24	78	45	24			
0.75	15	21	26	109	61	33			
1	15	21	26	146	81	45			
1.5	16	22	28	205	116	62			
2	16	22	28	274	155	83	55	28	76
2.5	18	23	27	304	185	107	55	27	98
3	18	23	27	365	222	129	60	27	129
3.5	18	23	27	426	259	150	60	27	150
4	18	23	28	486	297	165	65	27	186
4.5	20	25	29	493	307	180	65	29	195
5	20	25	29	547	341	200	70	29	233
5.5	20	25	29	602	375	220	70	29	256
6	21	26	30	625	393	231	75	28	310
6.5	21	26	30	677	426	251	75	28	336
7	21	26	30	730	459	269	75	28	362
8	22	27	31	796	505	299	75	29	399
9	22	27	31	895	568	336	75	29	449
10	22	27	31	995	631	373	75	29	499
20	42	45	50	1042	758	463	75	48	603
30	45	48	54	1459	1066	643	75	50	868

The problem of selecting the proper size shovel or dragline to provide a given output can be simplified by the use of Tables 16 and 17. These tables give the approximate output of various shovels and draglines and they have been computed using formula (2). These tables can be used as a selection guide to estimate the equipment type that is most likely to yield the desired production. When the most likely size has been selected, it should be confirmed by using the output formula with the factors that are applicable to the job in question.

Once the loading equipment output has been established it will then be necessary to estimate the number of trucks or wagons necessary to service the shovel or dragline. In selecting truck size it has been found that a ratio of 4–6 passes of a shovel or dragline to load the truck will generally give best results. The actual computation of the number of trucks necessary can be done using formulas (3), (4), and (5).

TABLE 17. DRAGLINE SELECTION GUIDE
Approximate Power Dragline Production (Bank Measure)

Bucket size (yd^3)	Boom length (ft)	Digging conditions			Production yd^3/hr Bank measure—efficiency 80% Digging conditions		
		Easy	Medium	Hard	Easy	Medium	Hard
		Cycle time (sec), 90° swing					
$\frac{3}{8}$	28	20	24	30	41	35	12
$\frac{1}{2}$	30	20	24	30	55	33	16
$\frac{5}{8}$	35	21	26	30	65	38	20
$\frac{3}{4}$	35	21	26	30	78	46	24
1	40	23	28	32	95	57	30
$1\frac{1}{4}$	45	24	28	32	114	71	38
$1\frac{1}{2}$	45	26	30	34	126	80	43
$1\frac{3}{4}$	50	27	31	35	142	90	48
2	50	28	32	37	156	100	52
$2\frac{1}{2}$	60	30	34	39	182	118	62
		Cycle time (sec), 110° swing					
3	70	30	35	42	219	137	69
$3\frac{1}{2}$	60	32	37	42	239	151	80
4	70	35	40	45	250	159	86
5	80	35	40	45	313	199	107
6	400	40	45	50	328	213	116
7	140	45	50	55	341	224	123
10	160	50	55	60	433	291	161
12	215	60	63	68	437	304	170
14	180	60	63	68	511	355	199
16	225	70	75	80	500	341	193
20	200	70	75	80	625	426	241

The necessary information can be easily obtained for a particular operation by time study method as well as theoretical computations. The following problem will demonstrate the use of the truck—shovel ratio formulas to calculate the number of trucks necessary assuming (t_t) the corrected truck cycle time is known.

The following detailed problem solution will illustrate the use of the output formula and help in clarifying any difficulties encountered. In this problem all data have been taken from the included tables and one should have no trouble duplicating the answer. The problem has been selected to demonstrate the use of all factors and is not typical of mining operations.

Sample Problem

Determine the bank measure output per hour of a $2\frac{1}{2}$-yd shovel operating in a 9-ft cut in common earth. The cut is narrow and loading will require a 180° swing to load the truck. The equipment operating efficiency is good and the management of the job is considered average.

Solution:

$$\text{Bank measure output cubic yards per hour} = \frac{(3600)(C_d)(E)(F)(D)(A)(S)}{t_s}$$

(1) Shovel size $2\frac{1}{2}$ yd	$C_d = 2.5$
(2) Equipment operating efficiency (good) (Table 7)	$E_1 = 0.90$
(3) Job management efficiency (average) (Table 8)	$E_2 = 0.85$
(4) Combined factor (Table 9)	$E = 0.77$
(5) Percent swell earth (Table 15)	25
(6) Swell factor (Table 15)	$S = 0.80$
(7) Optimum bank height 11.2 ft (Table 13)	
(8) 9-ft bank is 80 per cent of optimum (Table 14)	$D = 0.98$
(9) Angle of swing correction for 180° (Table 14)	$A = 0.71$
(10) Combined (DA) factor (Table 14)	$DA = 0.69$
(11) Digging conditions – easy digging (Table 12)	$E = 1.0$
(12) Cycle time, easy digging conditions (Table 16)	$t_s = 18$ sec

$$\text{Bank measure output} = \frac{(3600)(2.5)(0.77)(1.0)(0.98)(0.71)(0.80)}{18} = 214 \text{ yd}^3/\text{hr.}$$

Explanation of Shovel and Dragline Selection Guide

Note: The output figures on shovels and draglines in the preceding tables are given as average values of the outputs that may be expected under the conditions shown, but they are in no way guaranteed to apply to any particular job or excavator. The figures should be modified for conditions other than those stipulated and for dipper and bucket sizes other than those shown.

In estimating outputs for shovels, consideration must be given for such delay factors as ability of operator, length of swing, character of material, height of bank, voids in dipper, slope of ground on which machine is standing, and supply of trucks or cars. Shovel outputs are based on the net operating or digging time after allowances have been made for these factors.

The tables are to be used primarily as a selection guide to enable the engineer to select the size of equipment most likely to meet his requirements. The size selected should then be checked using the basic output formula with the proper factors to suit conditions to obtain an accurate estimate of the machine's productive capacity on the job.

TABLE 18. FACTORS USED IN BASIC FORMULA TO COMPUTE TABLES 16 AND 17[11]

Conditions	Average swell factor	Dipper factor (%)	Bucket factor (%)
Easy digging	0.80	95	95
Medium digging	0.74	80	75
Hard digging	0.67	60	50
Quarry production	0.67	Variable according to shovel size	Not applicable

Sample Problem

Assuming a corrected truck cycle time of 15 min, calculate the number of 10-yd capacity trucks necessary to service the shovel in the preceding problem.

Solution:

$$\text{Number of the trucks per shovel} = 1 + \frac{60(t_t)(A)}{(n)(t_s)} \tag{5}$$

$$\text{where } n = \frac{C_t}{(C_d)(F)} \tag{3}$$

Explanation of factors:

(1) (C_t) truck capacity given at 10 yd^3.
(2) (C_d) dipper capacity given at 2.5 yd^3.

(3) (F) fill factor from previous problem, 1.0.

$$n = \frac{10}{(2.5)(1.0)} = 4 \text{ passes to load.}$$

(4) (t_t) corrected truck cycle time = 15 min.

(5) (t_s) shovel cycle time (Table 16) = 18 sec.

(6) (A) angle of swing correction (Table 14) = 0.71.

$$\text{Number of trucks} = 1 + \frac{(60)(15)(0.71)}{(4)(18)} = 1 + 8.9 = 10 \text{ trucks.}$$

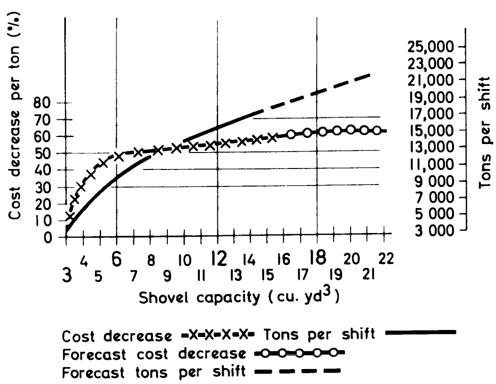


FIG. 16. Decrease in loading cost as shovel capacities are increased. (Based on the performance of a 3-yd^3 shovel for comparison.[13])

The preceding shovel and dragline output calculations are not intended to give exact answers to the mining operator, but to be used in the preliminary design or analysis of mining systems, for equipment comparisons, and for operations research analysis of loading problems.

HAULAGE OF ORE AND WASTE

As a general rule large open-pit copper mines or those with long hauls have rail haulage; medium-sized mines have truck haulage in combination with some other type of haulage. Auxiliary types of haulage which are used are inclined skips, vertical skips, inclined belt conveyors, standard gage railroad cars, large-capacity tractor type trucks, and tractor-drawn scrapers.

Bulldozers are used to push material for distances up to 200 ft; scrapers are used for distances from 200 to 1500 ft; trucks and shovels are used for distances ranging from 600 ft to 1 mile; and rail haulage is used for distances of more than 1 mile. Performance with bulldozers ranges from 500 to 1500 tons/shift; for scrapers from about 400–1500 tons/shift; for large trucks from 500 to 2000 tons/shift; and for rail haulage from about 1000 to 3000 tons/locomotive for distances of about 3 miles.[1]

Table 19. Shovel loading costs at Bagdad Mine[12]

	Shovel capacity, yards								
	Diesel-powered shovels			Electric-powered shovels					
	$2\frac{1}{2}$	3		$3\frac{1}{2}$		$4\frac{1}{2}$		6	8
	Ore	Ore	Waste	Ore	Waste	Ore	Waste	Waste	Waste
General information:									
Shifts operated	7.75	57.25	68.0	92.0	5.0	32.5	149.25	199.75	161.75
Material loaded (tons)	12,993.0	145,198.0	226,278.0	222,799.0	15,419.0	104,398.0	647,411.0	1638,666.0	1,536,578.0
Material loaded per shift (tons)	1677.0	2536.0	3328.0	2422.0	3084.0	3212.0	4338.0	8204.0	9500.0
Cost per ton:									
Labor	\$0.0185	\$0.0186	\$0.0142	\$0.0207	\$0.0162	\$0.0128	\$0.0095	\$0.0048	\$0.0045
Payroll taxes	0.0045	0.0012	0.0010	0.0014	0.0011	0.0006	0.0005	0.0002	0.0003
Vacation and bonus	—	—	—	—	—	0.0002	0.0005	0.0001	0.0002
Lubrication	0.0095	0.0026	0.0020	0.0016	0.0013	0.0015	0.0011	0.0006	0.0006
Miscellaneous	0.0355	0.0060	0.0046	0.0078	0.0061	0.0041	0.0031	0.0017	0.0025
Maintenance and repairs	0.1435	0.0464	0.0354	0.0642	0.0504	0.0475	0.0362	0.0083	0.0203
Industrial insurance	0.0056	0.0015	0.0012	0.0017	0.0014	0.0008	0.0006	0.0003	0.0004
Power	0.0036	0.0060	0.0046	0.0033	0.0026	0.0024	0.0018	0.0015	0.0018
Total	0.22	0.08	0.06	0.10	0.08	0.07	0.05	0.02	0.03

Rail Haulage

In the larger open pit copper mines broken ore is loaded by electric shovels with 6, 7, or 8 yd dippers into rail cars with capacities of about 70–80 tons. Trains are pulled by electric or diesel-electric locomotives of 1200–1750 h.p.

About ten dipper loads of material are required to fill an 80-ton car and the shovel moves an average of about three times in filling the car.

Track Maintenance

Track maintenance is an important item because the tonnage hauled over any given section of track is usually very high and because portions of trackage are built on newly placed material which settles continuously.

Track laying, track shifting, and maintenance operations are highly mechanized. Machinery used includes gang tampers, tie replacers, track lining machines, power brooms, etc.

Truck Haulage

Haulage units have undergone a continual gradual size increase. Capacities presently available from manufacturers include 34-, 40-, 45-, 50-, 60-, 62-, 65-, 70-, 90-, and 100-ton units.

These units vary in type from two and three-axle end dumps, to semi-trailers equipped with either end-dump or side-dump and to wagon trains consisting of a number of small side-dump units moved by one prime mover.

In 1962 there were more than 165 units with capacities of 50 tons or more, at work in various open-pit mines.(13)

Diesel powered trucks powered with 700 hp engines are presently available. Difficulties with transmission of the engine power to the wheels increase rapidly with engines larger than this. To overcome this problem and to control the transmission of power to the wheels the electric wheel has been developed. The electric wheel contains the motor and all required gearing so that it is an independent drive unit and a vehicle may be equipped with four such units to obtain four-wheel drive.

Calculation of Haulage Capacities

The charts and tables on the following pages may be used as a basis for estimating haulage capacities of a typical truck under varying road and grade conditions. These are re-printed from *Engineering and Mining Journal*, June, 1960, and were taken from the book titled *Basic Estimating—Production—Costs of International Construction Equipment*, which is issued by the International Harvester Company, Construction Equipment Division. The basic formula is $P = EIH/C$.

The accompanying tables illustrate production and cost values for a typical truck. Only the necessary tables for this particular item of equipment are included. The basic production formula is explained in the following text, where production is expressed as a combination of the efficiency hour (E), in-bank correction factor (I) of the material, heaped capacity of the machine (H), and cycle time (C) of the operation.

TABLE 20. IN-BANK CORRECTION FACTORS FOR VARIOUS MATERIALS AND IN-BANK WEIGHTS PER CUBIC YARD

Material	Approx. in-bank weight (lb./yd^3)	Approx. in-bank correction factors
Bauxite	2700–4325	0.75
Clay, dry	2300	0.85
Clay, light	2800	0.82
Clay, wet	3000	0.80
Copper ore	3800	0.74
Earth, dry	2800	0.85
Earth, wet	3370	0.85
Earth, with sand and gravel	3100	0.90
Gravel, dry	3250	0.89
Gravel, wet	3600	0.88
Granite	4500	0.76–0.56
Iron ore, hematite	6500–8700	0.45
Limestone, blasted	4200	0.60–0.57
Rock and stone, crushed	3240–3920	0.74
Sand, dry	3250	0.89
Sand, wet	3600	0.88
Shale, soft rock	3000	0.75
Slate	4590–4860	0.77
Trap rock	5075	0.67

Note: Whenever dirt loads are obtained by shovel, dragline or elevating grader, they will usually be less dense and weigh less than loads obtained by the boiling action of payscrapers or tractor-drawn scrapers. To compensate for this difference, when top loading multiply applicable in-bank correction factors by 0.90 for sandy soils, 0.80 for average soils and 0.70 for clay soils.

Production (P) is calculated in cubic yards (tons) per hour and is equated to four basic factors which come into play wherever earth moving and material handling are encountered. Production determines the number and types of machines required and the related cost factors.

The efficiency hour (E) accounts for unavoidable time delays such as machine adjustment, lubrication, cable changes, stops made by the operator, shovel moves, blasting, detour traffic, haul road maintenance, and unbalance of "helper" equipment such as motor graders, rollers and pushers.

Crawler equipment will normally operate at the rate of 50 min of work time out of each 60-min hour or at 83 per cent efficiency. Rubber-tired hauling units will normally operate at the rate of 45 min of work time out of each hour, or at 75 per cent efficiency. However, under excellent conditions a 55-min hour (92 per cent efficiency) can be obtained for crawlers and a 50-min hour (83 per cent efficiency) for rubber-tired equipment. On the other hand, this efficiency drops to 75 per cent (45-min hour) for crawlers and 67 per cent (40-min hour) for rubber-tired equipment operating under unfavorable or night conditions.

The in-bank correction factor (*I*), or swell, is applied to account for voids present in material disturbed by digging or loading. Wet clay, for example, will normally swell 25 per cent when loaded. Table 20 shows the in-bank correction factors for some materials encountered in mining.

TABLE 21. FIXED TIME FACTORS FOR TRUCK

Spotting	Depends on number of hauling units per loading unit. Should not exceed 0.50 min.; usually is not less than 0.15 min.
Loading	See Table 22
Turning	Not applicable (time required included in shovel spotting and dumping time)
Dumping	Depends on whether material is dumped over fill, windrowed (paywagons) or into grizzly. Bottom dump—0.25–1 min. Rear dump—0.50–1.50 min.
Reversing direction of travel per cycle	Not applicable (time required included in shovel spotting and dumping time)

The heaped capacity (*H*) of the machine is normally included in specification sheets and is usually known by the operator. Heaped capacity should not be confused with struck capacity or maximum payload ratings.

The cycle time (*C*) is the time required for a machine to obtain its load, move it to the place of dump and return to the loading point. Total cycle time is the sum

TABLE 22. SHOVEL OUTPUT
60 min hr—90°

Dipper cap. (yd^3)	$\frac{3}{4}$	1	$1\frac{1}{4}$	$1\frac{1}{2}$	$1\frac{3}{4}$	2
Moist loam or light sandy clay	2.75	3.42	4.17	4.75	5.34	5.93
Sand and gravel	2.54	3.34	3.83	4.50	5.00	5.56
Good common earth	2.25	2.92	3.50	4.00	4.50	5.00
Clay, hard tough	1.83	2.42	3.00	3.50	3.92	4.42
Rock, well blasted	1.58	2.08	2.54	3.00	3.42	3.83
Common excav. with rocks and roots	1.33	1.75	2.17	2.54	3.00	3.34
Clay, wet sticky	1.17	1.58	2.00	2.42	2.75	3.09
Rock, poorly blasted	0.83	1.25	1.58	1.94	2.33	2.67

of travel time factors and fixed time factors. Fixed time factors are spotting, loading, turning, dumping and reversing the direction of travel. Travel time factors are altitude, machine weight, rolling resistance, grade resistance, coefficient of traction, and average speeds.

The fixed time factors for a truck are summarized in Table 21. The time necessary to load the truck may be estimated from Table 22 and calculated by the formula:

$$\text{Time to load machine} = \frac{\text{machine capacity in cu bank yd}}{\text{loading unit production in cu bank yd per minute}}$$

Travel time factors offer more variables and depend strictly on the conditions at any one place. The basic formula, without modification, is:

$$\text{Travel time (in minutes)} = \frac{\text{Distance}}{\text{mph} \times 88}$$

However, the following factors retard the mph rate:

Altitude

For machines with naturally aspirated engines (not turbocharged), derate lb. pull as shown on specification sheets 3 per cent for each 1000 ft altitude above 3000 ft. For machines with turbocharged engines derate specification sheet pulls 3 per cent for each 1000 ft altitude above 5000 ft.

Machine Weight

Empty machine operating weights are shown on specification sheets. Loaded machine weights can be found by use of the formula

$$\begin{matrix}\text{loaded}\\ \text{machine}\\ \text{weight}\end{matrix} = \begin{pmatrix}\text{machine heaped}\\ \text{capacity,}\\ yd^3\end{pmatrix} \times \begin{pmatrix}\text{in-bank}\\ \text{correction}\\ \text{factor}\end{pmatrix} \times \begin{pmatrix}\text{in-bank}\\ \text{weight of}\\ \text{material, lb.}\end{pmatrix} + \begin{pmatrix}\text{empty}\\ \text{machine}\\ \text{weight, lb.}\end{pmatrix}$$

(IN-BANK YD³/MIN.)
swing)

$2\frac{1}{2}$	$2\frac{3}{4}$	3	$3\frac{1}{2}$	4	$4\frac{1}{2}$	5	$5\frac{1}{2}$	6	$6\frac{1}{2}$
6.76	7.27	7.75	8.75	9.68	10.6	11.4	12.3	13.2	14.0
6.50	7.00	7.50	8.92	9.26	10.0	10.8	11.6	12.3	13.1
5.73	6.34	6.76	7.58	8.50	9.34	10.1	10.8	11.4	12.1
5.17	5.58	6.00	6.76	7.50	8.16	8.84	9.50	10.1	10.7
4.59	5.00	5.34	6.08	6.83	7.58	8.33	9.00	9.58	10.2
4.08	4.50	4.83	5.58	6.34	7.00	7.68	8.33	9.00	9.58
3.83	4.17	4.50	5.17	5.75	6.42	7.00	7.58	8.16	8.68
3.25	3.59	3.72	4.50	5.08	5.67	6.25	6.83	7.33	7.84

TABLE 23. TRAVEL TIME FACTORS

Factor	Effect on mph
Altitude	Reduces available pull power which may require lower gear speeds than at sea level.
Machine weight	Increasing machine weight by loading requires lower gear travel speeds.
Rolling resistance	The higher the rolling resistance the lower the gear range which must be used.
Grade resistance	The steeper the adverse grade the lower the gear range which must be used.
Coefficient of traction	May eliminate travel in selected gear range if slippage results.
Average speeds	Maximum possible speeds should be reduced to average speeds.

Rolling Resistance (RR)

The sum of the forces which resist forward motion of a machine is its rolling resistance. They include ground conditions, tire flexing and power train friction. The sum of these resistances is expressed in pounds as a percentage of machine weight. It is generally found that to overcome such resistances as normal tire flexing and power train friction will require pull power equal to 2 per cent of a machine's total weight. Resistance offered by ground conditions, however, will increase this figure to as much as 16 per cent of the machine's total weight depending on the softness or irregularities of the surface to be traveled. Table 24 lists some of the conditions and resistances encountered in operations.

TABLE 24. ROLLING RESISTANCE AND COEFFICIENT OF TRACTION FACTORS

Type of haul road surface	Rolling resistance (% machine total weight)		Coefficient of traction (% of machine weight on drive components)	
	Rubber-tired machines (%)	Crawler tractor (%)	Drive tires	Crawler tracks
Concrete, rough and dry	2	–	0.80–1.00	0.40–0.50
Compacted dirt and gravel, well maintained, no tire penetration	2	–	0.50–0.70	0.80–0.90
Dry dirt, fairly compacted, slight tire penetration	3	–	0.45–0.65	0.80–0.90
Firm, rutted dirt, tire penetration approx. 2 in.	5	2	0.40–0.50	0.70–0.80
Soft dirt fills, tire penetration approx. 4 in.	8	4	0.40–0.50	0.70–0.80
Loose sand or gravel	10	5	0.20 0.30	0.30–0.40
Deeply rutted dirt, spongy base, tire penetration approx. 8 in.	16	7	0.10–0.20	0.30–0.40

TABLE 25. FACTORS FOR CONVERTING MAXIMUM HAULING UNIT SPEEDS TO AVERAGE SPEEDS

Haul section distance in (ft)	Speed factor
500–1000	0.46–0.78
1000–1500	0.59–0.82
1500–2000	0.65–0.82
2000–2500	0.69–0.83
2500–3000	0.73–0.83
3000–3500	0.75–0.84
3500–4000	0.77–0.85

Rim pull (RP) or drawbar pull (DBP) required to overcome rolling resistance (RR) may be found by use of the formula:

$$RP \text{ or } DBP \text{ required to overcome } RR = \text{total machine weight (lb.)} \times RR \text{ Factor}$$

Grade Resistance (GR)

For each 1 per cent of adverse grade, the machine must develop an additional pull equivalent to 1 per cent of the machine's total weight in pounds. This is expressed by the formula:

$$RP \text{ or } DBP \text{ required to overcome } GR = \text{total machine weight} \times GR \text{ in } \%$$

Coefficient of Traction

Table 24 shows the percentage of weight on tracks or drive tires which can be utilized as pull power on various ground surfaces before slippage occurs. A traction check may be made by use of the formula:

$$\text{Usable } RP \text{ or } DBP \text{ before slippage} = \begin{pmatrix}\text{machine weight on tracks} \\ \text{or drive tires}\end{pmatrix} \times \begin{pmatrix}\text{coefficient of} \\ \text{traction factor}\end{pmatrix}$$

Average Speeds for Rubber-tired Hauling Units

In order to adjust maximum attainable speeds to estimated average speeds, the speed factors shown in Table 25 can be used to advantage. Maximum speeds may be converted to average speeds by use of the formula:

$$\begin{matrix}\text{Average} \\ \text{speeds}\end{matrix} = \begin{pmatrix}\text{maximum} \\ \text{speeds}\end{pmatrix} \times \begin{pmatrix}\text{speed} \\ \text{factor}\end{pmatrix}$$

To estimate total travel time, it is necessary to break the haul road into sections wherever grade, rolling resistance, or both change. Then apply the travel time formula to each haul road section. The total of the time spent on each section is the total travel time.

Finally, the above data may be summarized and a production vs distance graph prepared for all items of equipment considered for any particular job. From this, the type and number of units necessary can be quickly and efficiently determined.

Types of Power Plants

Most of the large haulage units presently in service are powered by diesel engines.

A few experimental haulage units equipped with electric wheel drive have been put into service in recent years. One such ore hauler has a capacity of 55 tons and is equipped with a 700 h.p. diesel engine which turns the generator which supplies power to the electric motors in the wheels.

Anaconda Company at Butte, Mont., has under test a 670 h.p. truck capable of hauling 63 tons up a 15 per cent grade. This truck draws primary power to turn a 400 h.p. electric motor in each wheel from an overhead trolley when it is climbing steep grades and depends upon power from its diesel engine when maneuvering off the trolley.

Figure 17[14] shows results of test made with various types of trucks at the Berkeey pit.

Other power systems which are under study for possible future application by manufacturers of heavy haulage equipment include gas turbines driving generators and diesel engines driving hydraulic pumps which in turn will drive a hydraulic motor in each wheel.

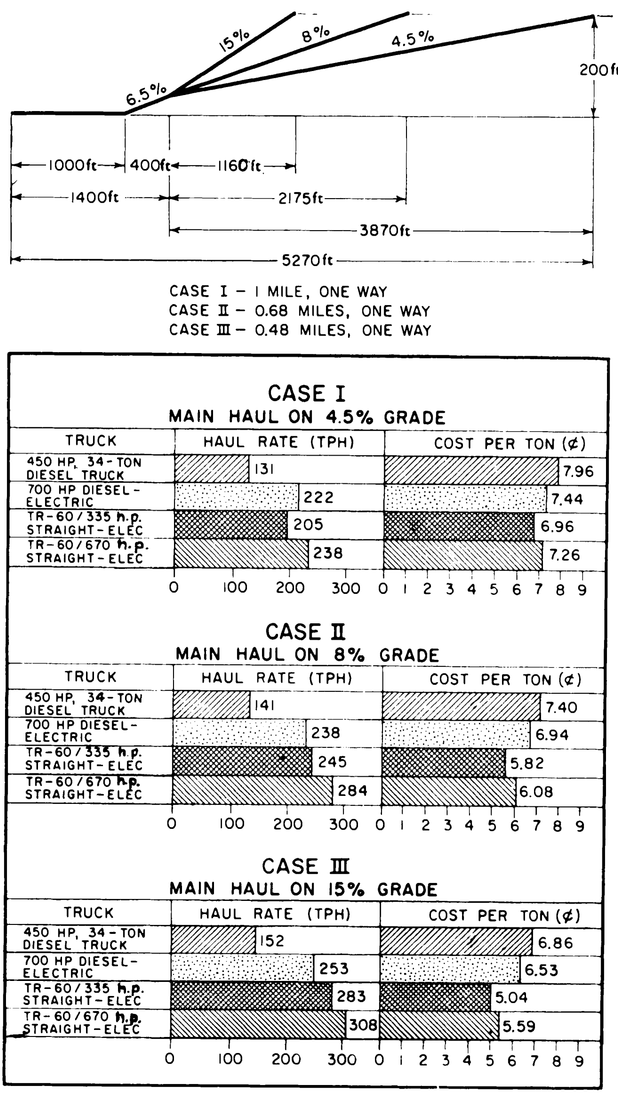


FIG. 17. Results of tests made with various types of trucks at the Berkeley Pit of the Anaconda Company.(14)

TABLE 26. COMPARISON OF PERFORMANCE AND COST PER TON ESTIMATES (BERKELEY PIT)[14]

	450 h.p. straight diesel	700 h.p. diesel-electric	TR-60/335 straight electric	TR-60/670 Straight electric	TR-60/670 Straight electric	700 h.p. diesel-electric
Truck empty weight, tons	35	57	56	57	57	57
Shovel bucket capacity, yd	6	6	6	6	10.5	10.5
Shovel passes per truck	4	7	7	7	4	4
Payload tons per truck	36	63	63	63	63	63
Gross vehicle weight, tons	71	120	119	120	120	120
Loading time, including delays	2.53	4.24	4.24	4.24	2.94	2.94
Trucks loaded per shovel-hour	23.6	14.1	14.1	14.1	20.4	20.4
Haul capacity, tons per hr	850	888	888	888	1285	1285
Maximum shovel rate per hr	1148	1148	1148	1148	1836	1836
Shovel capacity, tons per shift	6375	6660	6660	6660	9180	9180
Shovel—fleet operating effic.	74%	77.5%	77.5%	77.5%	70%	70%
Length of haul one-way, miles	1.0	0.68	0.48	0.48	0.48	0.68
Main haul grade	4.5%	8%	15%	15%	15%	8%
Elevation gained, ft	200	200	200	200	200	200
Round trip time, min	13.68	13.24	11.11	10.22	8.92	11.94
Tons hauled per 50-min hour	131	238	283	309	353	264
Trucks required per shovel	6.7	4	3	3	3–4	5
Estimated Truck Costs						
Cost per operating hour	\$10.43	\$16.91	\$14.26	\$17.28	\$15.20	\$16.59
Cost per ton hauled, cents	7.963	6.937	5.038	5.592	4.307	6.284
Percent variance from 34-T diesel	–	−12.88%	−36.73%	−29.77%	−45.91%	−21.08%

Skip Haulage

Ore and/or waste may be hoisted from deep narrow pits by means of skips operating on inclined railways. Such a system by hoisting ore or waste directly up the slope may eliminate the need for several miles of railroad or truck haulage road which would otherwise be needed for the climb out of the pit.

Tables 29 and 30 list the characteristics of skip haulage systems recently installed at two open pit copper mines in the southwestern U.S.

Conveyor Belts

Conveyors are sometimes used as the principal means of transporting ore from the pit bottom to the mill, or to a main haulage level. Since conveyors are not adapted to handling large rock fragments it may be necessary to install a crusher in the pit bottom to break up any large pieces before the rock is fed to the conveyor.

TABLE 27

	End-dump			
	20-ton	22-ton (with torque converter)	22-ton (without torque converter)	30-ton
General Information:				
Material hauled (tons)	147,228.0	3,100,105.0	535,933.0	5,346,676.0
do. (hours)	1767.0	31,793.0	6161.0	39,284.0
do. (miles)	7701.9	180,317.1	27,293.4	261,576.6
do. (units)	5.0	10.0	4.0	10.0
Cost per mile:				
Labor:				
operating	$0.5352	$0.4121	$0.4958	$0.3465
repair	0.1340	0.0347	0.0677	0.0238
Parts	0.3931	0.1273	0.3028	0.1247
Tires and tubes	0.3383	0.2970	0.5343	0.2161
Oil and grease	0.0370	0.0242	0.0375	0.0219
Fuel	0.2465	0.1951	0.2725	0.2030
Total cost per mile	1.68	1.09	1.71	0.94
Total cost per ton	0.088	0.063	0.087	0.046

TABLE 28. COMPARISON OF TRUCK AND CONVEYOR HAULAGE COSTS (BAGDAD MINE)[12]

	Truck		Conveyor
	30-ton	40-ton	
Material hauled (tons)	1,548,290	932,560	1,351,513
Material hauled per hour (tons)	127.5	151.05	254
Costs per ton:			
labor	$0.01964	$0.01781	$0.01394
payroll tax	0.0098	0.00082	0.00052
vacation and bonus	0.00039	0.00076	–
tires and tubes	0.01794	0.01065	0.00017*
lubrication	0.00197	0.00196	0.00001
miscellaneous	0.00479	0.00347	0.01228
maintenance and repairs	0.02655	0.02688	0.01002
industrial insurance	0.00122	0.00102	0.00080
fuel	0.01379	0.01372	0.00185
Total	0.08727	0.07729	0.0396

* Material and supplies.

HAULAGE COSTS (BAGDAD MINE)[12]

trucks			Turnarockers		
50-ton	Diesel, 40-ton	Butane, 40-ton	40-ton	50-ton	All haulage
756,670.0	3,538,162.0	59,172.0	25,414.0	60,573.0	13,569,933.0
3732.0	21,881.0	4311.0	208.0	442.0	109,766.0
20,203.1	112,724.3	21,833.2	1012.9	1913.0	635,863.6
1.0	6.0	2.0	1.0	1.0	
$0.4637	$0.4897	$0.4947	$0.5183	$0.6775	$0.0485
0.0668	0.0368	0.0140	0.1622	0.1402	0.0340
0.3144	0.1543	0.0603	0.3843	0.2628	0.1459
0.7460	0.2180	0.0326	1.3480	—	0.2659
0.0431	0.0318	0.0183	0.0175	0.0364	0.0257
0.2898	0.2679	0.3409	0.2342	0.3021	0.2234
1.92	1.20	0.96	2.66	1.42	0.74
0.051	0.038	0.036	0.106	0.045	0.051

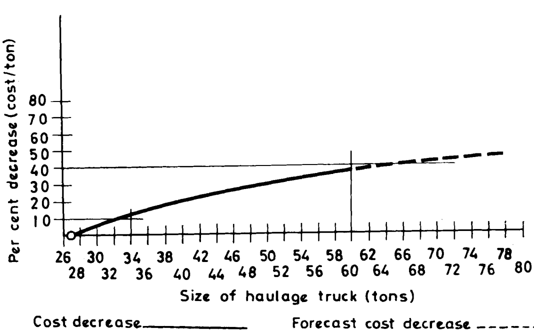


FIG. 18. Decrease in haulage costs as size of haulage unit is increased. (The costs of working 27-ton haulage units is used as a basis for the comparison).[13]

Combination Surface–Underground Haulage

A number of open-pit operations employ a combination of haulage methods to transport the ore from the pit bottom to the main haulage level. Some of these operations drop the ore down a shaft or raise it into cars which transport it to the

FIG. 19. Heavy haulage unit. (International Harvester Co.)

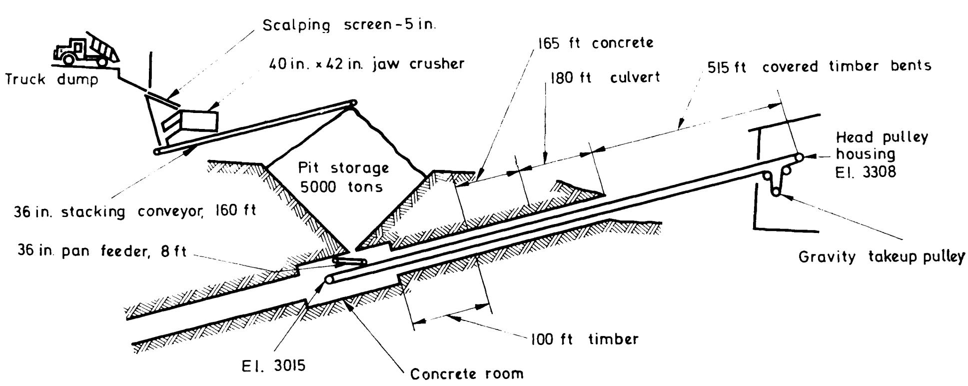


FIG. 20. Pit conveyor system – Bagdad Mine(12)

TABLE 29. SKIP HOIST STATISTICS AND OPERATING DATA, LIBERTY PIT, RUTH, NEVADA, JUNE 1959[15]

Elevation loading station (top)	6704 ft
Elevation headframe (base)	7033 ft
Total hoisting distance	1234 ft
Slope of skip trackway	19° 04′
Total vertical rise	405 ft
Weight of ore (wet)	50,000 lb
Weight of skip	39,490 lb
Weight of rope	5.6 lb/ft
Rope diameter	$1\frac{7}{8}$ in.
Drum diameter of hoist	11 ft 0 in
Hoist type—double cylindrical drum clutched	
Hoist drive—twin pinion, four 300-h.p. 1150 rpm motors in a single series loop with two 500 kW generators, driven by one 1250 h.p. leading power factor synchronous motor.	
Rope speed	1630 ft/min
Acceleration time	12 sec
Retarding time	10 sec
Rest time	7 sec
Total cycle	65 sec
Maximum capacity (47 min operating hour)	1120 wet tph

TABLE 30. SKIP HOIST STATISTICS AND OPERATING DATA, SANTA RITA PIT, N. MEXICO, JANUARY 1962[16]

Skipway slope	27° 19′	Hoisting speed maximum	1270 ft/min
Present skipway length	1384 ft	Cycle time (to waste headframe)	94.9 sec
Eventual skipway length	1763 ft	Trips per hour (present loading station)	37.9
Skipway track gauge (center to center)	13 ft	Capacity	1516 tons/hr
Hoist motors	Four 500 h.p.	Cycle time (to ore headframe)	77.3 sec
Type of motor	D.C.	Trips per hour (present loading station)	46.5
Type of control	Modified Ward Leonard	Capacity	1860 tons/hr
M-G set synchronous motor	2000 h.p.	Skip loading pocket capacity	40 tons
M-G set generators	Two 800 kW	Ore bin capacity	1000 tons
Hoist rope size	$2\frac{1}{4}$ in.	Waste bin capacity	350 tons
Skip size	1000 ft³	Cost (approximate)	$2,250,000
Skip rated capacity	40 ton		

FIG. 21. Skip haulage installation at a copper pit. (National Iron Co.)

surface while others transport the ore to a shaft where it is loaded into skips and hoisted to the surface.

Table 31[17] is a list of transportation means used at some open-pit mines where combination surface–underground haulage is used.

TABLE 31. SURFACE–UNDERGROUND HAULAGE SYSTEMS[17]

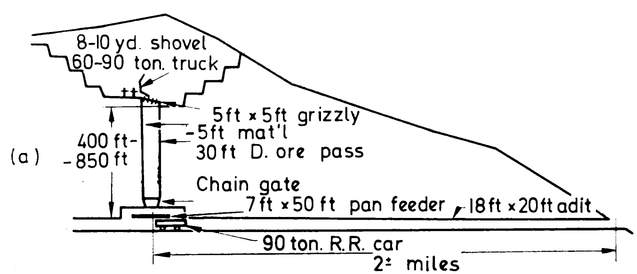
A. Underground Transport Method	
(1) Conveyor	Spruce, Susquehanna, Algoma, Caland.
(2) Railroad adit	Kiruna, United Verde, Sullivan; Carol Project (under consideration).
(3) Shaft	Susquehanna, Caland.
B. Crushing	
(1) Surface	Spruce, Susquehanna, Kiruna.
(2) Underground	Algoma.
(3) None	Caland, United Verde, Sullivan, Carol Project (under consideration).
C. Ore Passes	
(1) Vertical	Spruce, Susquehanna, Algoma, Kiruna; Carol Project (under consideration).
(2) Inclined	Caland, Sullivan, United Verde.
(3) With knuckle at bottom	Algoma, Sullivan.
(4) With bulldozing chamber and storage bins	United Verde.
(5) With inspection raises and drifts	Sullivan, United Verde. Verde.

TYPICAL EXAMPLES OF OPEN-PIT OPERATIONS AND COSTS

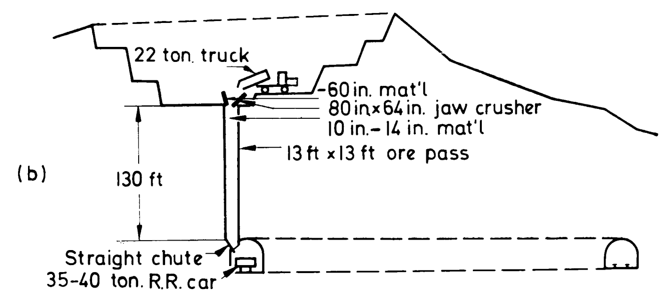
Small Scale Operations

Open-pit mining operations on a very small scale are, typified by those of the Copper Butte Mine near Ray, Arizona. These operations have been described in *U.S. Bureau of Mines Report of Investigations* 3914,[18] and by Hardwick.[1]

The mine which is located in the Mineral Creek mining district about 6 miles west of Ray, Ariz., has been operated for about 20 years and has yielded about



CAROL PROJECT



KIRUNA MINE

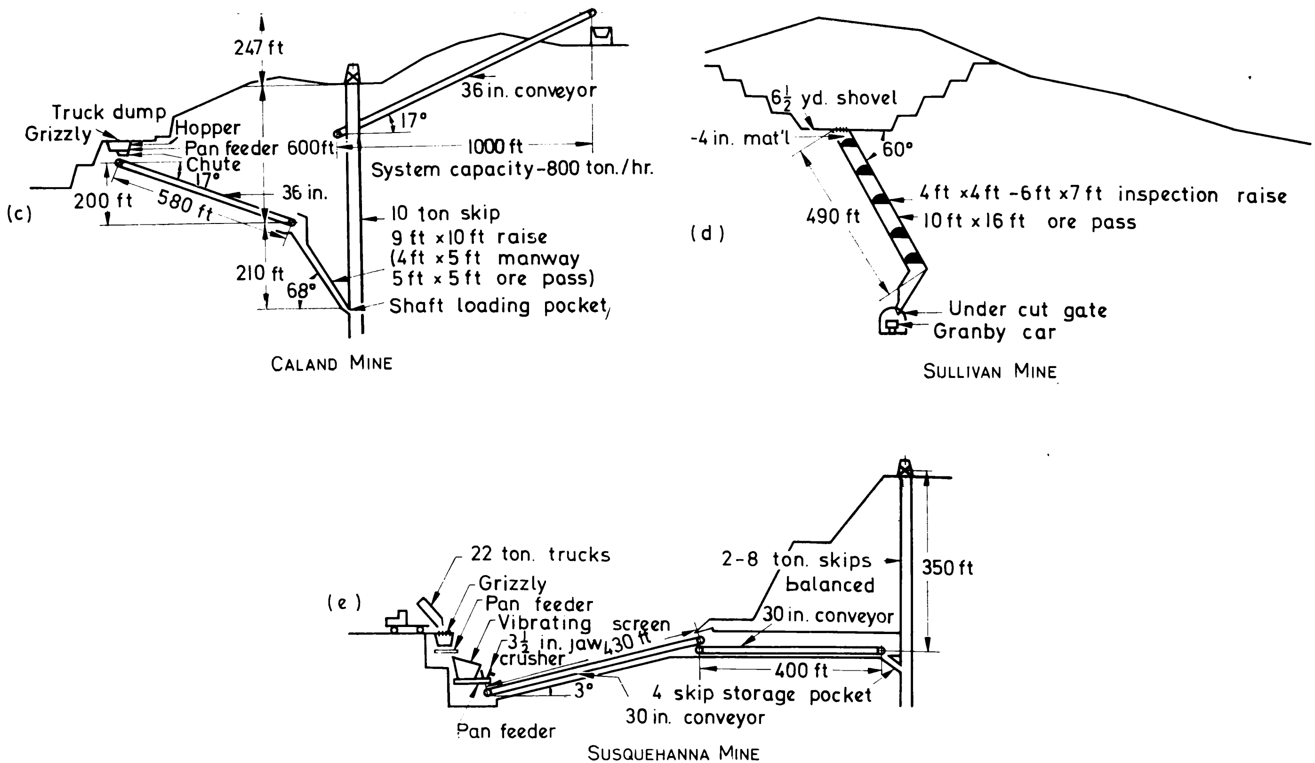


FIG. 22. Surface–underground haulage systems.

TABLE 32. COMPARISON OF ORE TRANSPORTATION METHODS AT THE CAROL PROJECT, LABRADOR(17)

	Truck–ore pass–tunnel	Truck–ramp–surface RR	Direct into surface RR
*Capital Requirements**			
Railroad trackage:			
main line (miles)	6.6	10.1	10.1
yard and benches (miles)		11.2	16.6
tunnel (miles)	3.8		
Rolling stock:			
locomotives	4 (60 tons each)	7 (125 tons each)	10 (125 tons each)
Cars (90 short ton capacity)	95	220	220
Trucks:			
40-ton rear dump†		10	†
60-ton trailer type	11		
Ore transfer raises	4		
Personnel Required for Ore Transportation Only	94	120	98
Cost Estimate‡			
Initial capital costs (millions of dollars)	15.1	15.7	17.4
Relative unit costs:§			
capital write-off including interest (%)	35	37	37
Operating labor (%)	15	20	16
Operating supplies and maintenance charges (%)	50	64	52
Total *relative* unit costs (%)	100	121	105

* Capital requirements are based on estimated production rate of 13,400 tons per shift.

† 40-ton rear dump trucks used only in opening-up benches.

‡ Cost estimates are based on an initial annual capacity of 5.5 million long tons of concentrates.

§ For comparative purposes, unit costs of the three systems are given as percentages adjusted to a base represented by the "truck-ore pass–tunnel" system.

125,000 tons of ore. The ore averages about 3 per cent copper and is shipped directly to the smelter.

The mine is operated by a force of six to eight men, including the owner. Mine equipment consists of two portable compressors, a wagon drill, a tractor-mounted $1\frac{1}{8}$ yd^3 loader, and dump trucks of 6-tons capacity equipped for highway travel.

The ore is broken from small benches with vertical holes and loaded into trucks with a loader. Generally one truck operates between the loader and a bin on the property. Other trucks are loaded from the bin and haul the ore to the railway siding about 15 miles away, from whence it is hauled by rail to the smelter at Hayden, Arizona, 18 miles away.

Medium-sized Operations

A typical medium-sized open-pit copper mine is the Copper Cities mine of Miami Copper Co. in Gila County, Ariz., which has been described by Hardwick.[1,19]

The mine is located on the south slope of Sleeping Beauty Mountain, about $3\frac{1}{2}$ miles north of Miami, Ariz. The ore body was estimated to contain 43 million tons and overburden was estimated to be 34 million tons, of which 20 million tons had to be moved before production could begin. About sixty-five men are employed in the mine-operating department, and about 12,000 tons of ore and 8000 tons of waste are mined each day.

The general mine layout is that of a cresecent-shaped pit advancing into a hillside. Waste is disposed of on each side of the pit. Generally three levels are operated —one level approaching completion and final pit limit, one level in full production, and one level just being started.

TABLE 33. PRODUCTION BY OPEN-PIT MINE (BAGDAD MINE)[12]

Period	Material mined (tons)			Production (lb.)	
	Waste	Leach ore	Sulfide ore	Copper	MoS_2
1940–1948	1,343,015	784,151	—	—	—
1948	1,071,273	620,996	1,037,427	14,422,718	—
1949	2,635,414	87,414	1,058,311	15,985,329	—
1950	2,607,572	549,572	1,250,892	22,529,680	—
1951	3,052,968	979,722	1,231,613	18,445,523	—
1952	2,652,856	1,307,543	1,221,220	18,236,496	248,748
1953	3,212,557	1,678,869	1,232,590	20,381,907	230,173
1954	5,023,438	1,621,614	1,300,454	18,440,052	293,520
1955	6,878,079	2,297,668	1,351,513	22,950,982	264,360
1956	8,291,041	4,474,317	1,363,505	14,912,211	253,906
1957	4,878,842	5,024,458	1,487,994	20,518,119	242,100

Nine-inch diameter holes are drilled with a rotary drill along predetermined lines to a depth of 6 ft below the 45-ft banks. The holes are spaced at 25- to 30-ft centers along rows that are 35–40 ft apart. The rotary drill average 461 ft of hole per shift, and an average of 85 tons of rock is broken per foot of hole. The average bit life is 1660 ft. About 7 tons of rock is broken per pound of explosive used. Only a small amount of secondary drilling is required to break lumps resulting from the failure of primary blasts to break the rock to digging grade.

TABLE 34. SUMMARY OF MINE EQUIPMENT IN USE AT BAGDAD MINE[12]

	Number of units
Drilling:	
Diamond drills, skid-mounted, surface type	2
Churn drills:	
1500-lb. tools	2
2000-lb. tools	3
Rotary drills ($6\frac{3}{4}$-in. holes)	2
Compressor, 225-cfm., trailer-mounted	1
Loading:	
Shovels:	
$2\frac{1}{2}$-yd, track-mounted, diesel-powered	1
3-yd, track-mounted, diesel-powered	1
$3\frac{1}{2}$-yd, track-mounted, electric-powered	1
$4\frac{1}{2}$-yd, track-mounted, electric-powered	1
6-yd, track-mounted, electric-powered	1
8-yd, track-mounted, electric-powered	1
Front-end loader, $1\frac{1}{2}$-yd.	1
Haulage:	
End-dump trucks:	
2-ton	19
30-ton	10
50-ton	1
55-ton	8
Tournarockers:	
40-ton	2
50-ton	2
Tractors:	
Large, track-mounted	5
Small, track-mounted	1
Large, rubber-tired	3
Miscellaneous:	
Scraper, 11-yd	1
Ripper	1
Road graders	3
Trucks:	
drill service	1
sprinkler, 4500- and 2500-gal capacity	2
grease	1
Boom crane	1
Forklift	1

TABLE 35. ANALYSIS OF STRIPPING COSTS BASED UPON 4,065,457 TONS MOVED IN 4 MONTHS OF 1957 AT BAGDAD MINE (12)

	Percent of total cost		Percent of total cost
Direct labor	3.06	Loading	17.35
Other direct charges	0.39	Haulage	45.78
Primary drilling	9.57	Dozers and graders	8.40
Secondary drilling	1.11	Miscellaneous	3.06
Blasting supplies	11.28	Total	100.00

TABLE 36. MINING AND MILLING COSTS FOR 4 MONTHS OF 1957 AT BAGDAD MINE

	Percent of total cost		Percent of total cost
Mine (ore and waste):		*Concentrator:*	
Operating:		Operating:	
breaking ground	7.13	primary crushing	1.55
loading and hauling	8.60	primary conveyor	1.81
general mine expense	2.32	secondary crushing	4.42
Total direct cost	18.05	grinding	9.17
		flotation	7.96
		filtering	1.04
Other:		molybdenum section	1.57
diamond drilling	2.04	tailings disposal	0.89
advance stripping	47.19	water supply	1.10
Total other costs	49.23	general mill expense	4.21
Total mining cost	67.28	Total concentrating cost	32.72
		Grand total operating cost	100.00

Three 5-yd full revolving electric shovels are available, but only a total of three shovel-shifts are worked per 24-hr period. Ore and waste are hauled in heavy trucks which carry an average of 41 tons/trip and make about thirty trips per shift, for an average of about 798 ton-miles/shift.

The principal equipment used at this mine includes one 9-in. rotary drill, three full-revolving electric 5-yd shovels, eight 40-ton rear-dump trucks, seven bull-dozers, one motor grader, one 2-ton flat-rack truck for supplies, one 2-ton flat-rack truck for explosives, one 4-yd dump truck for moving stemming material, and one 3000-gal sprinkler truck. A 12,000-ton/day concentrator is operated in conjunction with the mine.

TABLE 37. MISSION OPEN-PIT DATA[21]

(a) How Rock is Mined

Item	Ore	Waste
Bench height	40	40
Drill	Be 40R or Reich 850	
Hole size	9-in. diameter	9-in. diameter
Hole depth	50 ft	50 ft
Pattern	24 ft hole spacing in rows 20 ft apart	
Feet hole per shift	300	350
Powder	NH_4NO_3 Use polyethelene bags in wet holes	
Blasting	Primacord, double primed, ProCore No. 3 and Pentalite boosters	
Powder factor, per yard	0.8 lb.	0.8–1.0 lb.
Shovel	P & H 1800	P & H 1800
Dipper fill factor	80–85%	80–85%
Tons per shift	11,000	11,000
No. dippers per truck	5	5
No. trucks per shovel	6–9	8–10
No. loads per shift	25–30	22–25
Truck factor, tons	58	55–60
Haul distance, miles	0.9	1.0–1.7

(b) How Alluvium is Mined

Item	Gravel
Bench height	50 ft originally, now 40
Drill	Reich 850
Hole size	12-in. diameter
Hole depth	50 ft
Pattern	24 ft rows 30 apart
Feet hole per shift	800
Powder	NH_4NO_3
Blasting	Primacord, double primed, ProCore No. 3 and Pentalite boosters
Powder factor, per yard	0.4–0.5 lb.
Shovel	190—B (10 yd)
Dipper fill factor	100
Tons per shift	13,500
No. dippers per truck	4
No. trucks per shovel	6–8
No. loads per shift	35–40
Truck factor, tons	60
Haul distance, miles	1.0

TABLE 37 *(cont.)*. *(c) How Conglomerate is Mined*

Item	Conglomerate
Bench height	40
Drill	Reich 850 or BE 40R
Hole size	9 or 12 in. diameter
Hole depth	25–50 ft
Pattern	17-ft centers, single line
Feet hole per shift	200–400
Powder	NH_4NO_3
Blasting	Primacord, double primed, ProCore No. 3 and Pentalite boosters
Powder factor, per yard	0.8 lb.
Shovel	P & H 1800
Dipper fill factor	75–80
Tons per shift	9000
No. dippers per truck	5.5
No. trucks per shovel	6
No. loads per shift	25
Truck factor, tons	58
Haul distance, miles	1.0–1.7

Another typical medium-sized open-pit copper mine is the Bagdad Mine in Yavapai County, Arizona.[12]

Tables 33 to 36 show annual production, mine equipment in use, and a breakdown of the stripping costs at this mine.

Large Open-pit Operations

The Morenci open pit of the Phelps-Dodge Corp., in Greenlee County, Arizona is typical of large operations. This operation has been described by Hardwick.[1, 20] The current open-pit mining operations at Morenci was begun in 1937 on an ore body roughly oval in plan, about 1 mile wide, and $1\frac{1}{4}$ miles long. The average thickness of the overlying waste was 216 ft, and its maximum thickness was about 500 ft. It took approximately 4 years, from 1937 to the end of 1941, to strip 49 million tons of waste and prepare the ore body for mining. By September, 1956, this operation had produced 205 million tons of ore and removed 420 million tons of waste. The mine continues to yield about 52,000 tons of ore per operating day.

Twelve-inch diameter holes are drilled with rotary drills along the crest of 50-ft banks at altitudes ranging from 4350 to 5650 ft. Mining benches advance into a high ridge. Some waste from the higher levels is hauled to the waste dumps by truck, but the larger part of the rock is loaded by 5- to 9-yd capacity shovels into

TABLE 38. SUMMARY OF BERKELEY PIT COSTS, SIX-MONTH AVERAGE[22]

	Cost per ton of material
Blasting:	
Blasthole drilling	0.0032
Blasthole drilling power	0.0004
Blasting	0.0131
Repairs to drill equipment	0.0010
Total blasting	0.0177
Shovel Loading:	
Shovel loading	0.0141
Shovel loading power	0.0012
Repairs to shovels	0.0087
Total shovel loading	0.0240
Truck Haulage:	
Truck haulage	0.0558
Road maintenance	0.0021
Dump maintenance	0.0028
Repairs to trucks	0.0285
Repairs to tractors	0.0067
Repairs to graders	0.0007
Repairs to service trucks	0.0014
Total truck haulage	0.0980
Crushing, Conveying and Loading Ore:	
Crushing, conveying, and loading ore	0.0080
Crushing, conveying, and loading ore power	0.0012
Repairs to diesel locomotives	0.0001
Repairs to crushing plant, conveyor, and bins	0.0030
Total crushing, conveying, and loading ore	0.0123
Miscellaneous Expenses:	
Superintendence, technical and clerical	0.0112
Service and other miscellaneous	0.0148
Service and other miscellaneous power	0.0003
Fire expense	0.0001
Move power lines	0.0004
Maintain airways	0.0007
Maintain airways power	–
Move miscellaneous equipment	0.0001
Holiday pay	0.0017
Workmen's compensation	0.0020
Vacation allowance	0.0017
Unemployment tax	0.0009
Federal insurance contribution	0.0020
Total miscellaneous	0.0359
Total	0.1879
General expense	0.0406
Total	0.2285
Amortization of Development:	0.0616
To combine development and mining costs	0.0657
Total all expenses	0.2244

standard-gage railroad cars which are pulled by 1750-h.p. diesel-electric locomotives.

Twenty full-revolving electric shovels were used to load approximately 52,000 tons of ore and 118,000 tons of waste a day into 43-yd^3 railroad cars and into 25- or 35-ton rear-dump trucks. The average rail haul was about 3 miles. About 1000 men are employed in the mining operation.

The Mission Mine of the American Smelting and Refining Co, near Tuscon, Ariz., produces copper ore at the rate of 15,000 tons/day. It was necessary to remove 44 million tons of waste before the full daily production could be sustained.

Table 37(a), (b), and (c) gives pertinent data on breakage, loading and haulage at the Mission Mine.[21]

Open-pit Mining Costs

Table 38[22] is a tabulation of costs at the Berkeley Pit of the Anaconda Company at Butte, Mont. It will be noted that the largest single cost item is haulage.

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CHAPTER 3

THE STABILITY OF PIT SLOPES

THE angle at which cut slopes will stand determines the ratio of waste stripping to ore production in open pit mines. For an open-pit mine $1000 \times 1000 \times 400$ ft deep an additional one million tons of rock will have to be moved to flatten the average cut slope by an amount of 1°. This would be a small or medium sized pit. An open-pit mine in the porphyry copper ores of the southwestern U.S. may be 4000×5000 ft in lateral dimensions and more than 1000 ft deep. In a pit of this size a 1° change in the average slope will involve 20 million tons of material.[1]

TYPES OF SLOPE FAILURE

Coates[2] classifies slope failure under the following four types:

(1) Rock Fall

Rock fall is simply the fall of loose material to the foot of the slope where the angle is greater than the angle of repose of the blocks. The cause of this slope failure can be considered as a failure in tension of the rock or the cement between blocks of rock.

To diminish this type of failure the tensile strength of the rock mass should be preserved. The deterioration of surface rock can be due to water action -- weathering of joint material, freezing and thawing of joint water, and expansion of joint materials on exposure. The blasting of benches is possibly of more significance. Any development leading to the reduction of shock into the wall rock would assist in preserving the strength of the material. For example by using optimum or smaller burdens and hole spacing or by using inclined holes, or possibly slower explosives.

The problems arising from this type of failure are generally distinct from those arising from other modes of failure. However, in some cases the overall pit wall slope will be controlled by the accumulation of this loose material. In one instance a pit slope 520 ft high in siltstone breccia cannot be steepened to more than 45° because of the large amounts of loose rock which run down to flood the benches, even though the intact rock is of such nature that a higher slope angle could probably be maintained without producing a deep seated failure.

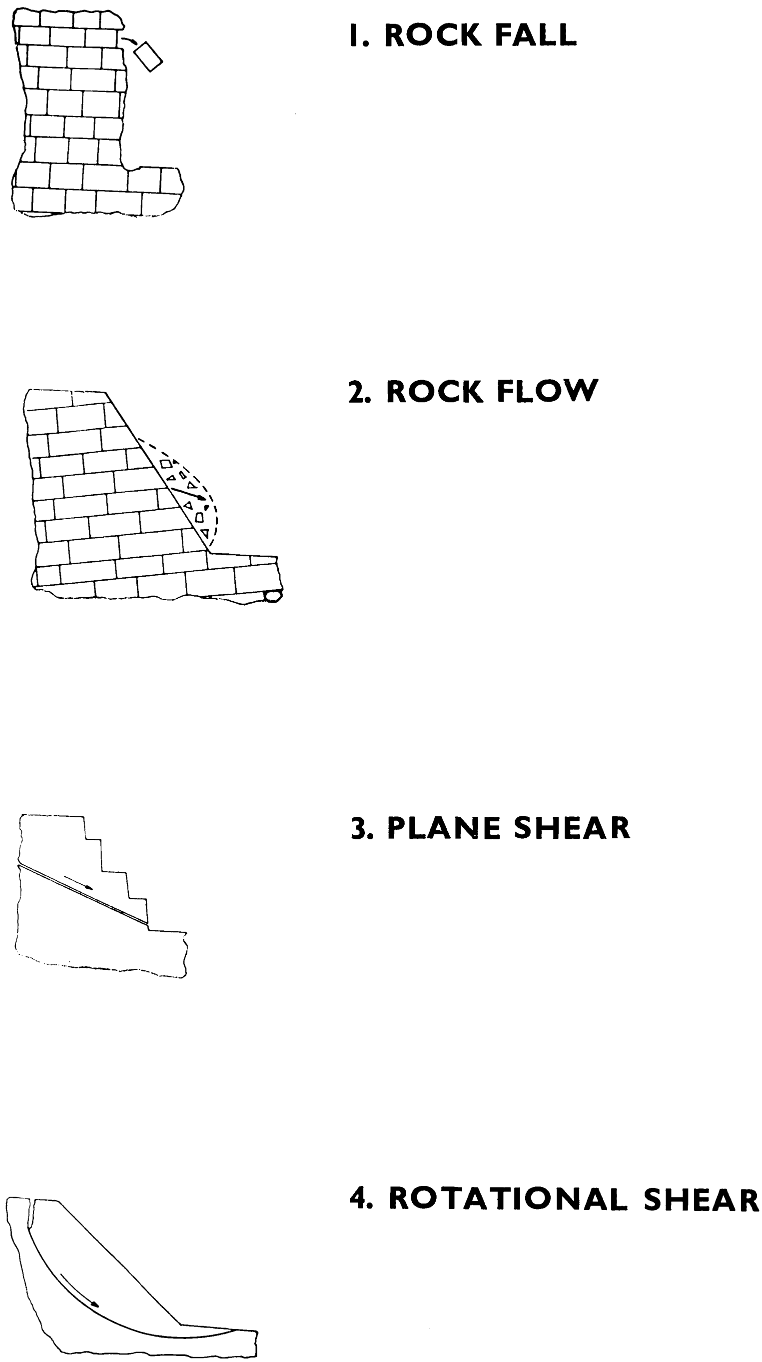


FIG. A. Four types of slope failure.

(2) Rock Flow

Block flow is the term given to a slope failure when there is a general breakdown of the rock mass. Rocks nearly always contain a family of joints which divide the mass into a system of irregular blocks. The blocks may or may not be cemented together, however, the strength of the cement between the rocks is normally less than that of the rocks themselves.

When such a rock mass is subjected to shear stresses sufficient to break down the cement or to cause crushing of the angularities and points of the rock blocks the blocks will move as individuals and the mass will flow down the slope, or will slump into a more stable slope position.

(3) Plane Shear

Plane shear failure results when a natural plane of weakness such as a fault, a shear zone, or bedding plane exists within the slope and has a direction such as to provide a preferential path for failure. Large intact portions of the slope rock may slide along this plane surface.

Such geological planes of weakness may not be serious as long as water is excluded from them. However, if seepage water finds its way into such planes it tends to soften and make lubricants of any clay minerals present, but if present in sufficient quantities it also exerts a hydrostatic uplift on the material lying above the plane of weakness and reduces the effective stress on the plane thereby reducing the friction which prevents sliding.

(4) Rotational Shear

Failure by rotational shear produces a movement of an almost undisturbed segment along a circular or "spoon-shaped" surface and occurs in comprehensive, uniform material. This material would not be affected by geological planes of weakness.

Failure of this type can occur from causes which either increase the shear stresses or which decrease the shear strength of the material. The shear stresses may be increased by removal of ground from the toe of the slope, by pressures of seepage water acting parallel to the slope, or by hydrostatic pressure acting in fissures or cracks in the slope. Strength of the material may be decreased by decomposition of material, by weathering action, or by vibration effects.

METHODS FOR PREDICTING SLOPE STABILITY

Soil Mechanics Methods

When the dimensions of a slope are large as compared with the dimensions of the units of the material composing the slope then conditions favor the analysis of slope stability by the principles of soil mechanics.

Such conditions are common in the open-pit copper mines in the porphyry copper ores of the south-western United States. These ores are usually brecciated so that there are not large blocks of intact rock, and in addition most of these pits attain depths of hundreds of feet so that the dimensions of the slopes are measured in thousands of feet.

Some of the open-pit iron mines are situated in slates, quartzites, and volcanics, containing bedding and joint systems which allow them to be classified as fragmented materials and make them susceptible to analysis by the principles of soil mechanics.

Rock Mechanics Methods

When a rock slope is composed of rock blocks which are large relative to the height of the bank or slope then the principles of rock mechanics can be better applied to predicting the steepest slope on which the rock will stand without failure.

In some instances a combination of the two methods may be required as when certain known faults form planes of weakness along which failure may occur as "plane shear" but the slope is composed of fractured material which may fail along a circular arc in the classical manner if the material becomes saturated with seepage water.

SLOPE STABILITY ANALYSIS BY SOIL MECHANICS METHODS

Stresses in Embankments and Slopes

Consider a vertical bank composed of a homogeneous cohesive material, as, for example, a cohesive soil or an unfractured rock. The height to which such a bank can be cut depends upon the shear strength of the material. In strong rock natural cliffs hundreds, or thousands, of feet high are not uncommon.

In contrast to this, a granular, cohesionless material, such as a dry sand or gravel, cannot be cut to form a vertical bank, but instead tends to assume a slope with a maximum inclination usually between 35 and 45°. The exact value depends upon the shapes of the individual grains, or fragments, and their tendency to interlock.

In strip-mining of coal, vertical walls from 60 to 100 ft high are cut and remain standing long enough to allow the coal to be removed.

Open-pit mining operations in the metalliferous mining field involve much deeper excavations. Some of these pits reach depths of 1200–1500 ft. Despite the fact that they are excavated in rock, the maximum slopes which will remain stable in the deeper pits do not exceed an average of about 35°. Thus, it appears that the rock involved resembles structurally a granular or fragmented material rather than a cohesive solid.

If we consider the relative dimensions of the individual "fragments" composing a rock mass and the dimensions of the slope, it is apparent that a slope 10 ft high in gravel might correspond to a slope 1000 ft high in a fractured rock. The individual fragments composing the rock mass may possibly be several feet in nominal diameter but such dimensions are small when compared with the overall dimensions of the exposed slope. Thus, the rock fragments may behave in a manner similar to the rock fragments composing a gravel bank.

Types of Slope Failure

Homogeneous, Elastic Material

Consider a vertical bank cut in unfractured rock or cohesive soil. The maximum principal stress is a vertical compressive stress on a plane at the base of the bank. When the bank reaches a certain height the maximum compressive stress at the base of the bank exceeds the shear strength of the material and shearing progresses from the toe of the bank as shown in Fig. 1.

The slight vertical yield allows the overlying rock to subside slightly and to pivot. This results in lateral tensile stresses in the upper portion of the block.

As excavation proceeds downward the sheared zone at the toe progresses inward and upward, allowing the overlying block to subside and to tilt further. Tensile stresses at the surface eventually create a fracture and separation and the additional weight must be supported by potential shear surfaces. When the weight of the block exceeds the resistance along these surfaces failure takes place, with the block sliding down along the sheared surfaces.

A theoretical analysis of the stability of a vertical cut follows.[3]

Assuming that failure occurs along a plane surface such as 1–3 in Fig. 1 at an angle θ to the horizontal then the length of line 1–3 will be $h/\sin\theta$. The weight W per foot depth of the sliding wedge is

$$W = \frac{\gamma h^2}{2 \tan \theta}$$

where γ is the effective unit weight of the soil.

The component of W tending to produce sliding will be $W \sin \theta$. This force will be resisted by friction and by cohesion. Thus at failure:

$$W \sin \theta = \frac{ch_{cr}}{\sin \theta} + W \cos \theta \tan \phi$$

or

$$\frac{\gamma h_{cr}^2}{2} \frac{\sin \theta}{\tan \theta} - \frac{\gamma h_{cr}^2}{2} \frac{\cos \theta \tan \phi}{\tan \theta} = \frac{ch_{cr}}{\sin \theta}$$

or

$$\frac{\gamma h_{cr}^2}{2} (\sin \theta \cos \theta - \cos^2 \theta \tan \phi) = ch_{cr}$$

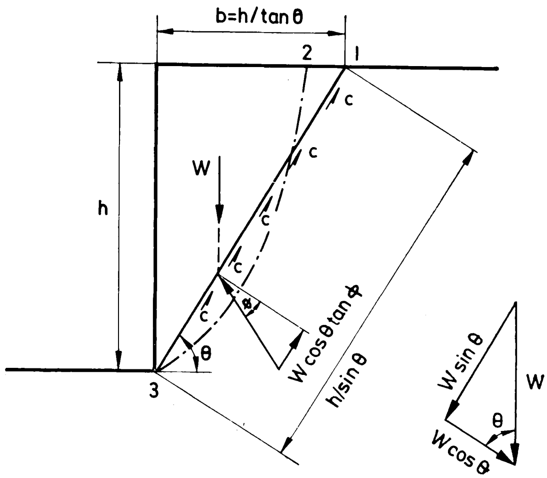


FIG. 1. Stability analysis of a vertical cut.[3]

In these equations c is the cohesion, $\tan \phi$ is the coefficient of internal friction, and h_{cr} is the limiting height or "critical height" of the vertical bank. The limiting height of slope, h_{cr}, is minimum when the expression

$$\sin \theta \cos \theta - \cos^2 \theta \tan \phi$$

is maximum. It can be shown (3) that this expression is maximum when

$$\theta_{cr} = (45° + \phi/2)$$

substituting this value in the above equation

$$\sin \theta_{cr} \cos \theta_{cr} - \cos^2 \theta_{cr} \tan \phi = \frac{1}{2 \tan \theta_{cr}}$$

and substituting again:

$$\frac{\gamma h_{cr}^2}{4 \tan (45° + \phi/2)} = ch_{cr}$$

$$h_{cr} = \frac{4c}{\gamma} \tan \left(45° + \frac{\phi}{2}\right) = \frac{4c}{\gamma} \left(\frac{\cos \phi}{1 - \sin \phi}\right)$$

This is the equation for the theoretical maximum height of a vertical cut in cohesive material assuming a plane surface of failure. Field observations show that slides occur along curved surfaces of a type corresponding to 2–3 in Fig. 1. Analyses of such curved surfaces have indicated[3, 4] that the computed values for h_{cr} are only about $3\frac{1}{2}$–5 per cent less than those obtained with plane failure surfaces.

The preceding analysis did not consider the fact that tilting of the wedge creates tensile stresses near the ground surface. These tension cracks extend for some distance down from the ground surface and reduce the strength of the bank; that is the shear strength of the soil is destroyed in the cracked zone.

Terzaghi has estimated that the maximum depth to which tension cracks may extend is equal to one-half the unsupported height of the cut.

Cohesionless Material

Materials such as dry sand, gravel, or talus cannot form vertical slopes because they retain stability only by virtue of the friction forces between surfaces of individual fragments. The coefficient of friction, μ, between fragments may be represented by tan ϕ, where ϕ is designated as the angle of internal friction.

As shown in Fig. 2(a), the tendency of a block to slide down an inclined plane is resisted by the frictional force between the block and the plane. The force f due to the weight of the block can be resolved into the force f_p parallel to the plane, and the force f_n normal to the plane. Frictional resistance to sliding is proportional to f_n, that is,

$$R_f = f_n \mu = f_n \tan \phi$$

As β increases f_n decreases and f_p increases until a point is reached at which f_p exceeds R_f and the block slides.

Similarly, when a cut is made in cohesionless granular material which is in a loose condition (e.g. not compacted) the maximum slope which can be maintained will have an inclination approximately equal to the angle of internal friction ϕ.

The angle of internal friction for such granular materials varies with the nature and the initial density of the material. For sand the angle may vary between extreme limits of 30–50°.[4] The difference in the angle of internal friction of a given sand in its densest and its loosest state may be as much as 15°. According to Terzaghi[4] the angle of repose of dry sand has a fairly constant value and is approximately equal to the angle of internal friction of the sand in its loosest state. Such a slope is shown in Fig. 2(b).

According to Wilson[5], the angle of internal friction varies from a minimum of about 32 or 33° for a well-rounded sand in a loose state to something in excess of 45° for a compact angular crushed rock.

Since the fissured rock in an open-pit mine is in a compact state more dense than anything which could be produced in the laboratory, it should be possible to

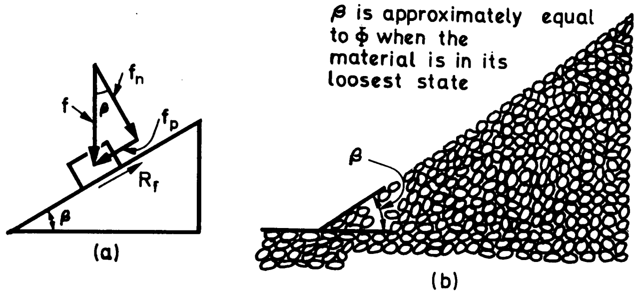


FIG. 2. Stability of slopes in cohesionless granular material.

excavate slopes as steep as 45°. Often, however, these slopes are unstable at inclinations between 30 and 40° and in a few instances slopes appreciably flatter than 30° are in the process of sliding. Wilson ascribes these phenomena to the effect of hydrostatic uplift pressures due to seepage in reducing the effective friction forces along sliding surfaces.

Typical Stability Computations

Numerous field observations have shown that slope failure in a homogeneous cohesive material usually occurs by sliding along a surface which is approximately a section of a cylinder; that is, the surface along which movement takes place is approximately represented, in cross-section, by a portion of a circle. The customary method for computing the stability of a slope consists of a series of calculations to determine, by trial and error, the circular arc along which resistance to sliding is the least.

The mass of sliding material may be considered as revolving about a central axis just as a portion of a cylinder revolves about the axis of the cylinder. The stability of a potential slide is calculated by determining the moments which tend to produce rotation about such an axis and comparing them with the moments which resist such rotation.

Potential failure surfaces with various radii are assumed to exist and the relative stability of each is calculated until that potential failure surface is found for which the factor of safety is the smallest. The procedure is as follows:

A cross-section of the slope is drawn to scale and a circular arc is drawn through it, as shown in Fig. 3. This cross-section is then divided into a number of segments and the effective total weight of each is computed. In Fig. 3 the effective weight, W_8, of Block 8, for example, when multiplied by its lever arm, r_8, gives the contribution of this block toward the total moment which tends to produce rotation of the entire block *ABC*, and sliding along the surface *AB*. On the other hand, the effective weight, W_2, of Block 2, creates a moment W_2r_2, which resists movement.

Thus the net moment tending to produce movement will be the algebraic sum of moments, or

$$M_{\text{mov}} = \sum_{4}^{10} Wr - \sum_{1}^{3} Wr$$

Resisting moment is also furnished by cohesion along the curved failure surface *AB*, and by friction along the same surface. For Block 8 the resistance along the slippage surface due to cohesion is cl_8 and the resisting moment is Rcl_8 where c

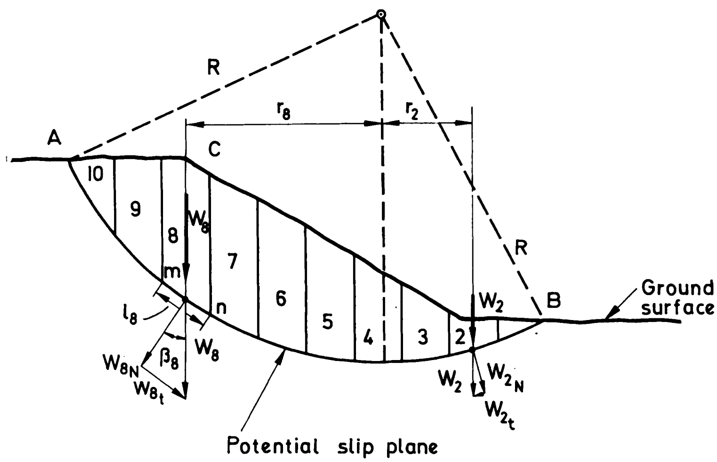


FIG. 3. Method for determining the stability of a slope.

is unit cohesion and l_8 is the distance measured along the potential failure surface, *mn*, at the bottom of the block, and R is the radius to the center of rotation. The resistance due to friction is $W_{8N} \tan \phi$ and the resisting moment is $RW_{8N} \tan \phi$, where $W_{8N} = W_8 \cos \beta_8$ is the component of the weight acting normal to the slippage surface and $\tan \phi$ is the coefficient of friction along this surface.

The total resisting moment is then

$$M_{\text{res}} = R \sum_{1}^{10} cl + R \sum_{1}^{10} W_N \tan \phi$$

and the factor of safety against movement is

$$F_s = \frac{\text{Total resisting moment}}{\text{Total moment tending to cause movement}} = \frac{M_{\text{res}}}{M_{\text{mov}}}$$

Effect of Seepage Water

In the foregoing example it was assumed that no seepage water was present. Such subsurface water has an important effect on the stability of soil and rock slopes. Take for an example the slope of Fig. 4(a) in rock, within which there exists a

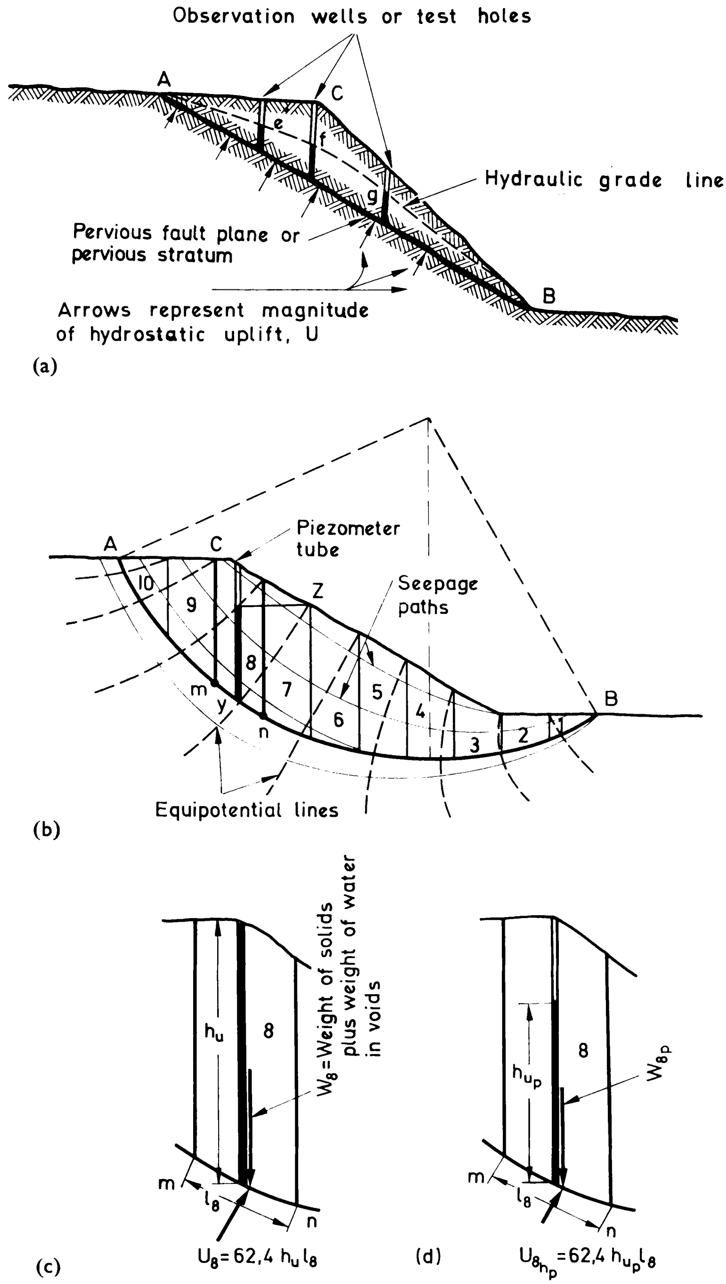


FIG. 4. (a) Effect of seepage water confined to a single pervious plane in otherwise impervious material.
(b) Effect of seepage water in pervious materials.
(c) Uplift forces – piezometric head assumed to be at the ground surface.
(d) Uplift forces – based on actual measured piezometric head.

natural plane AB along which ground water can percolate. Such a plane may be a fault plane, or a pervious sedimentary stratum such as sandstone. The water exerts an uplifting effect on the overlying rock block ABC. The magnitude of this uplift depends upon the hydraulic gradient along the pervious plane, and the hydraulic gradient in turn depends upon the size and roughness of the pervious channel and the amount of water supplied at the top of the slope.

The profile of the hydraulic grade line can be determined by sinking a series of test wells along the slope and noting the height to which the water rises in each hole.

If the lower end of the seepage channel were nearly closed, so that water could escape only very slowly, then the hydraulic grade line would rise at every point to the same elevation as that of the ground surface at point A, and the trapped water would exert the maximum amount of hydrostatic uplift on the block ABC. Such an unfavorable situation is rarely encountered in nature, but conditions similar to that represented by the hydraulic grade line $AefgB$ in Fig. 4(a) are rather common. If holes are bored from the surface to tap the water-bearing stratum, then water will stand in the holes to the depths indicated by this line.

The uplift pressure at each location is equal to the hydrostatic pressure in the pervious stratum at that location. The hydrostatic pressure along the pervious plane acts in a direction normal to the plane and its effect is to reduce the normal component W_N. This reduces the frictional resistance, $W_N \tan \phi$, but does not reduce the force W_t, which tends to produce movement parallel to the plane. The presence of seepage water along AB reduces the factor of safety which opposes the sliding of the block ABC.

Seepage Effects on Stability of Pervious Soils and Rocks

In the previous example it was assumed that rock was impervious except along the plane followed by the seepage water. This condition can exist in rocks *in situ* because the porosity and the permeability of most unfractured rocks are relatively low.

If we deal with disturbed or unconsolidated materials, then the porosity (that is, the percentage of void space per unit volume of material) may range up to almost 50 per cent. This is equivalent to a void ratio (that is, the ratio of the volume of the voids to the volume of solid material) of 1.00.

For a porous material the total weight, W_8, of the block, such as No. 8 in Fig. 4(b), consists of the weight of the solid material plus the weight of the water contained in the voids of the solid material. However, that portion of the block which is situated below the groundwater table will be buoyed just as if it were in a tank of water. The total uplift due to this buoying effect may be taken as equal to the total weight of the water displaced by the portion of the block which is in the saturated zone.

If we assume that rainfall is intense enough to completely saturate the block in question, then the saturated zone extends completely to the surface. In Fig. 4(c) the weight W_8 of Block No. 8 would be the sum of the weight of solids in the block and the weight of the water in the voids.

As an example, assume a void ratio of 0.8, which is typical of some loose sands and gravels. Then the porosity is equal to $0.8/1.0+0.8 = 44$ per cent, so each cubic foot of soil contains:

$$(0.44)(62.4) = 27.5 \text{ lb. of water}$$
$$\text{and} \quad (0.56)(165) = 92.5 \text{ lb. of solids,}$$

making the weight of the saturated soil 120 pcf.

The weight of the soil dry is 92.5 pcf, and this is increased to 120 pcf when saturated. On the other hand the uplift effect (or buoying effect), is equivalent to 62.4 lb. per ft of head, h, above the slippage plane. Thus, the normal pressure, produced by the dry weight of Block No. 8 on plane *mn*, is reduced when the block becomes saturated because the net weight increase is 27.5 pcf due to water in the voids, while the net uplifting force, U_8, normal to this plane is equal to 62.4 h.

In rock which is relatively intact and has a low porosity, the maximum uplift effect occurs along the permeable planes because the uplifting effect of 62.4 h_8 is not partially compensated by weight of water in voids (see Fig. 4(a)).

A slope stability analysis for pervious material is made in the same manner as is shown in Fig. 3, except that the additional weight of seepage water contained in the sliding block must be included in W and the hydrostatic uplift pressure along potential failure plane AB must be deducted from the total normal pressure due to W to obtain the true "intergranular" pressure W_N which contributes to frictional resistance.

Hydrostatic Head Contributing to Uplift

It may be difficult to determine the actual hydrostatic head h_u which contributes to the uplift pressure U.

As an approximation, for a saturated block the hydrostatic head may be taken as equal to the height of a column of water extending from the potential slippage plane to the ground surface. This is represented by the water column h_u in Fig. 4(c).

If test holes or wells are bored, then the surface defined by the water levels in the holes is the upper surface of the saturated zone, or "water table". This surface may be used for determining the value of h_u. Piezometers may also be installed to find the actual head contributing to uplift.

Piezometric Head

Typical piezometers are shown in Fig. 5. A common type of piezometer consists of a small cylinder of porous stone to which is connected one (sometimes two) plastic tubes. The porous stone is sealed into a borehole and the tubes are con-

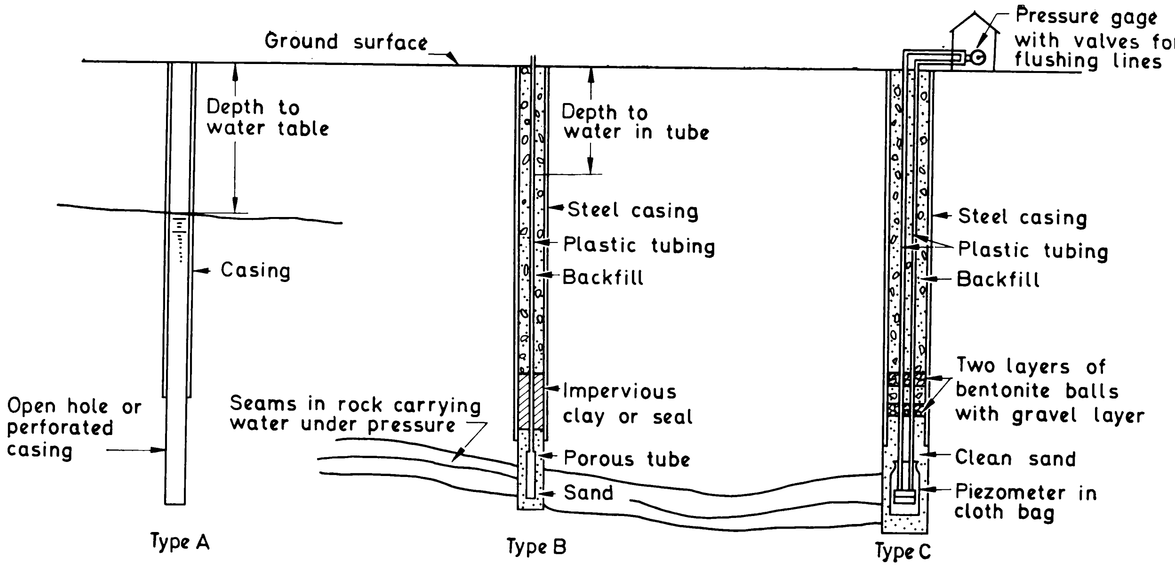


FIG. 5. Piezometer types for measuring uplift pressures.(5)

nected to gages on the surface. The apparatus measures the hydrostatic head at that point where the porous stone is sealed in. The level of the water in a piezometer tube rises to the same elevation as that at which the equipotential line intersects the ground surface.

From the results of piezometric measurements a "flow net" may be sketched. Such a sketch is shown in Fig. 4(b). A flow net consists of flow paths and "equipotential lines"; that is, lines which pass through all those points having the same hydraulic grade line. In other words, if piezometers were installed at intervals along a specific equipotential line, water would rise to the same height in each of the piezometer tubes.

In Fig. 4(b) the equipotential line yz intersects the surface at z. With the piezometer installed at point y water in the piezometer tube rises to the same elevation as point z. Thus the actual head contribution to uplift is the head h_{u_p} shown in Fig. 4(d), rather than the head h_u (Fig. 4(c)), which was used as an approximation. If a hole is uncased and open for its full length, then water will rise to the top of the saturated zone; that is, to the actual level of the water table.

As a rough approximation, the directions of equipotential lines in homogeneous material may be taken as being approximately normal to the slope along its middle portions.

A complete presentation of the theory of seepage flow nets, and their equipotential lines is beyond the scope of this book. Among references which treat this subject more thoroughly are Refs. 3, 4, and 6.

Faults

In addition to its "uplift effect", groundwater also lubricates potential planes of sliding. Clay surfaces, which may be resistant to movement when dry, become soft and slippery when saturated. For this reason faults are important as potential sliding surface. The effect of faults is three-fold:

(1) A fault furnishes a surface of weakness which possesses continuity of direction and inclination.

(2) The fractured zone adjacent to a fault may form a good path along which seepage water can percolate into an otherwise nearly impervious rock.

(3) Fault clay, or fault gouge, acts as a lubricant when it is wet.

PREDICTION OF SLIDES

Apparatus has recently been developed which can be installed in boreholes in slide material to locate planes of sliding. The initial position of the casing is computed from dial readings and subsequent readings show changes in inclination at every point. The change of inclination is fastest in the vicinity of the plane of sliding. Figure 6 shows cross-sections through a moving rock mass at a large open-pit copper mine. The slide occurred along the surface of two intersecting faults which approximately paralleled the normal slope of the pit. The actual surfaces of movement were delineated from information obtained by slope indicators installed in boreholes designated by *TM* in Fig. 6. For *TM* 3 the depth of the plane of movement was established the first day after installation of the observation wells, although

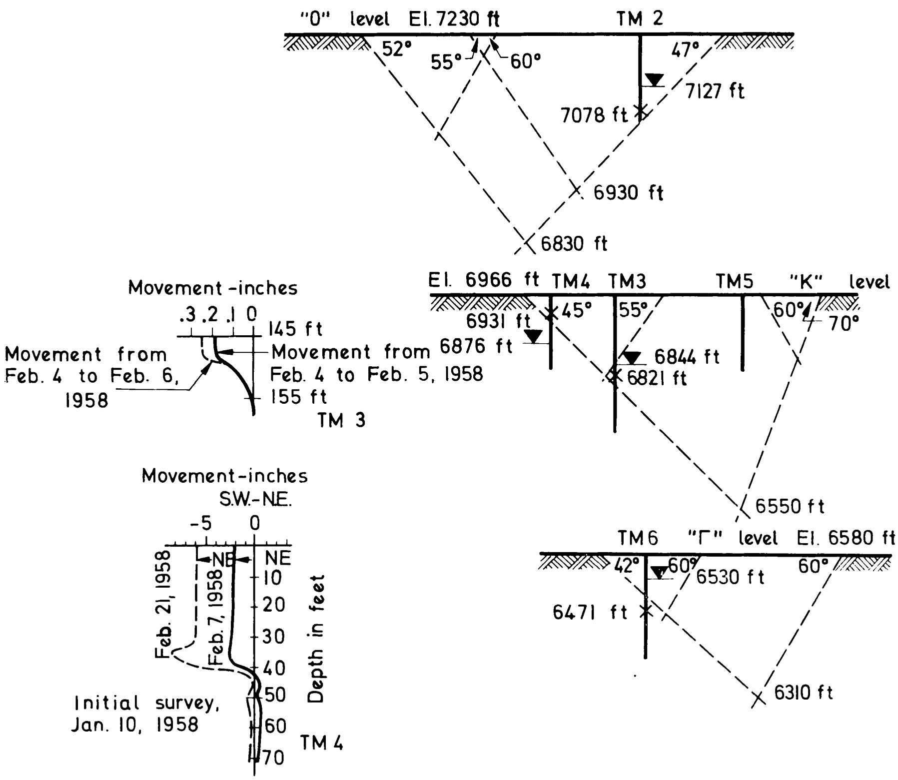


FIG. 6. Movement data and section through slide, Pit "A".[5]

the actual movement at that time was less than 0.2 in. Eventually, the casings were sheared off at the depths indicated by the *X*'s in Fig. 6. The ground-water surface is indicated by the black triangles in the same figure.

The longitudinal section shown in Fig. 7 has been plotted from this data.

Figure 8 shows stability computations (after Wilson) for this slide, where:

W = the weight of a slice of rock of unit length;

N = the component of W normal to the slide axis;

T = the component of W which is parallel to the direction of sliding (this is the force which induces movement);

K = the ratio of width and depth of saturated zone to width and depth of the triangular block of rock;

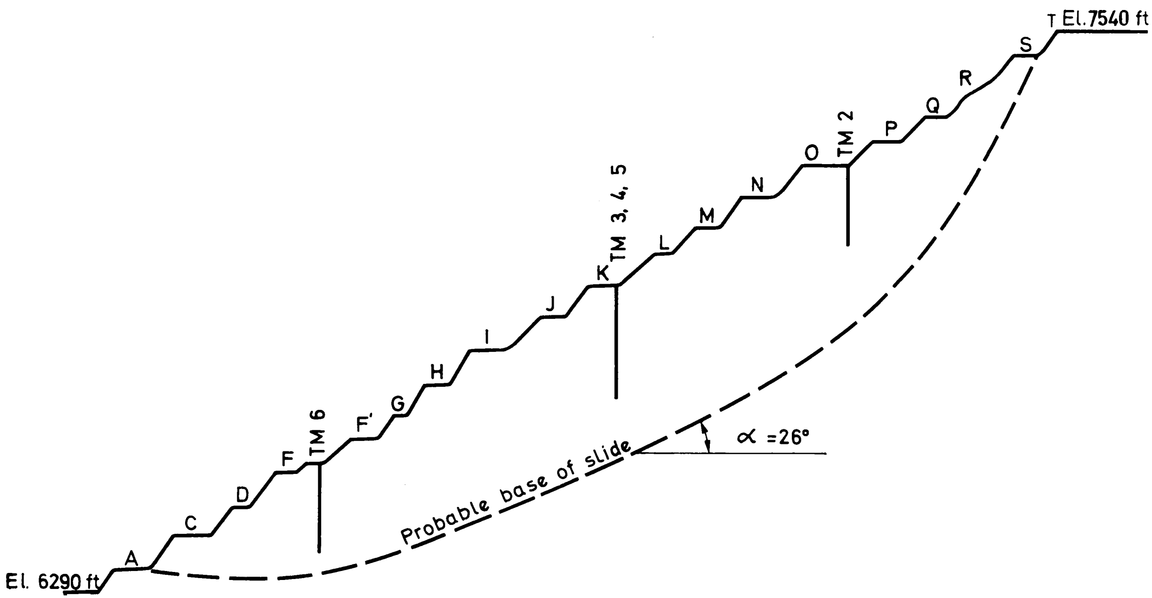
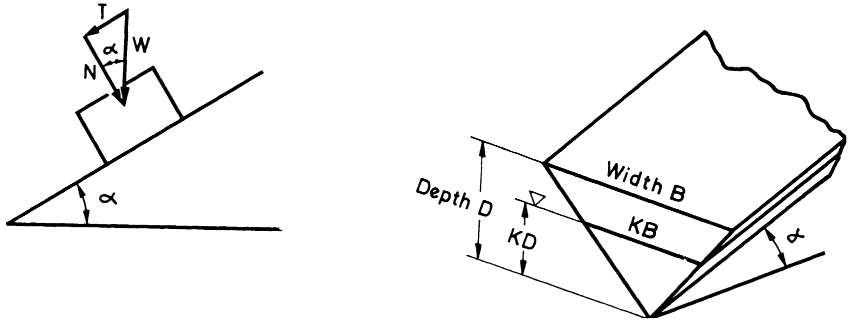


FIG. 7. Longitudinal section through slide, Pit "A".[5]



ASSUMPTIONS:

$\alpha\ = 26°$
γ_s = Rock density
$= 162$
$\gamma_w = 62.5$

TRIANGULAR SECTION

STABILITY COMPUTATION
(Based on unit length)

$$W = \frac{BD}{2}\gamma_s$$

$$N = W\cos\alpha$$

$$T = W\sin\alpha$$

$$U = \frac{BDK^2\,\gamma_w}{2\cos\alpha}$$

$$FS = \frac{(N\text{-}U)\tan\phi}{T} = (2\text{-}K)^2\tan\phi$$

COMPUTED RELATIONSHIP
between K and ϕ for $F.S. = 1$

$K =$	0.4	0.6	0.8	1.00
$\phi =$	28.5°	31.5°	36.5°	45°

COMPUTED RELATIONSHIP
between K and $F.S.$
for ϕ 36.5°

$K =$	0.8	0.7	0.6	0.5
$F.S. =$	1.00	1.11	1.21	1.29

FIG. 8. Analysis of slide, Pit "A".(7)

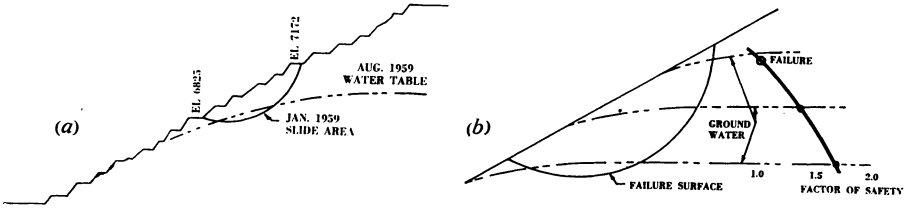


FIG. 9. Influence of ground water on slope stability, Pit "A".(7)

(a) Longitudinal section through Pit "A".
(b) Effect of rise in ground water table on the slope stability.

B = the width of the triangular section;
D = the depth of the triangular section.

For this particular set of conditions, and setting $\infty = 26°$, the equation for the factor of safety, F.S. reduces to

$$\text{F.S.} = \frac{\text{N}-\text{U}}{\text{T}} \tan \phi$$

or

$$\text{F.S.} = (2-K^2) \tan \phi$$

Assuming that F.S. = 1.0 (that is, the slide is on the verge of movement), then the calculated relationship between K and ϕ is shown in Fig. 8.

Setting $\phi = 36.5°$, then the computed relationship between K and F.S. is also shown in Fig. 8.

With a measured value of $K = 0.8$ and a F.S. of 1.0, then the angle of friction on the sliding planes would be 36.5°.

It will be noted that moments were not taken about a center of rotation in the preceding analysis, but rather the component parallel to the surface of sliding was used as the driving force. For the type of slide which approximates a portion of a cylinder the value of the tangential driving force varies continuously along the sliding surface. It is possible to obtain the total driving force for the cylindrical type of slide by computing the tangential force for each block separately and summing them, rather than by taking moments.

The analysis assumed zero porosity of the rock; that is, the total uplift effect of the saturated zone is taken as being equal to the volume of the saturated zone multiplied by the unit weight of water. In a relatively compact rock *in situ* this is a legitimate assumption.

When dealing with a pervious material, such as a sand or gravel, the uplift effect of the water saturating a seepage zone is equal to the weight of the water displaced by the solids. If 50 per cent of the material is void space, then only 50 per cent of a unit volume is occupied by water, and the uplift effect is only one-half of that occurring with a solid body.

There was a definite correlation between the amount of rainfall and the rate of slide movement as measured by Wilson. This is shown in Fig. 10. The time lag of several months between the periods of heavy rainfall and periods of increasing rate of slide movement indicated that the seepage water in this slide migrated from some other catchment area to this location.

METHODS OF STABILIZATION

In the preceding example sliding occurred along fault planes and the movement was caused by seepage water and resultant uplift pressures which reduced frictional resistance along the fault planes.

In such instances one remedy is to provide drainage facilities which prevent the water from reaching the potential sliding block. Possible methods for accomplishing this are shown in Fig. 11. Such measures are expensive, but where long-cut slopes are involved the steepening of a slope may reduce the required amount of stripping enough to pay for such drainage facilities.

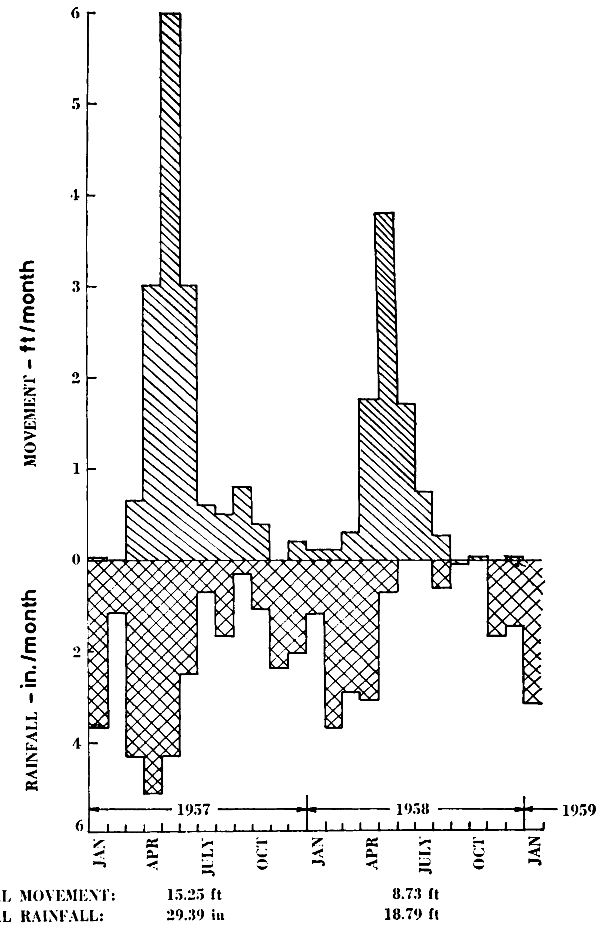


FIG. 10. Relationship of rainfall and slide movement, Pit "A".(7)

It was noted by Wilson that the most stable area in one open-pit mine was an area that had been mined at one time by block caving and was therefore broken up. Such disturbances of the ground had provided good drainage and had lowered the water table.

In addition to improving the drainage, such disturbances to natural ground also break and displace fault surfaces and weak or soft strata so that their continuity is destroyed. Thus, the opportunity for movement along continuous surfaces is reduced.

EXAMPLES OF PIT SLIDES

Studies have been made of actual slides in some Canadian open-pit iron mines.[8] Table 1 lists pertinent data for nine slides which occurred in paint rock at the Errington and the Hogarth Pits at the Steep Rock mine.

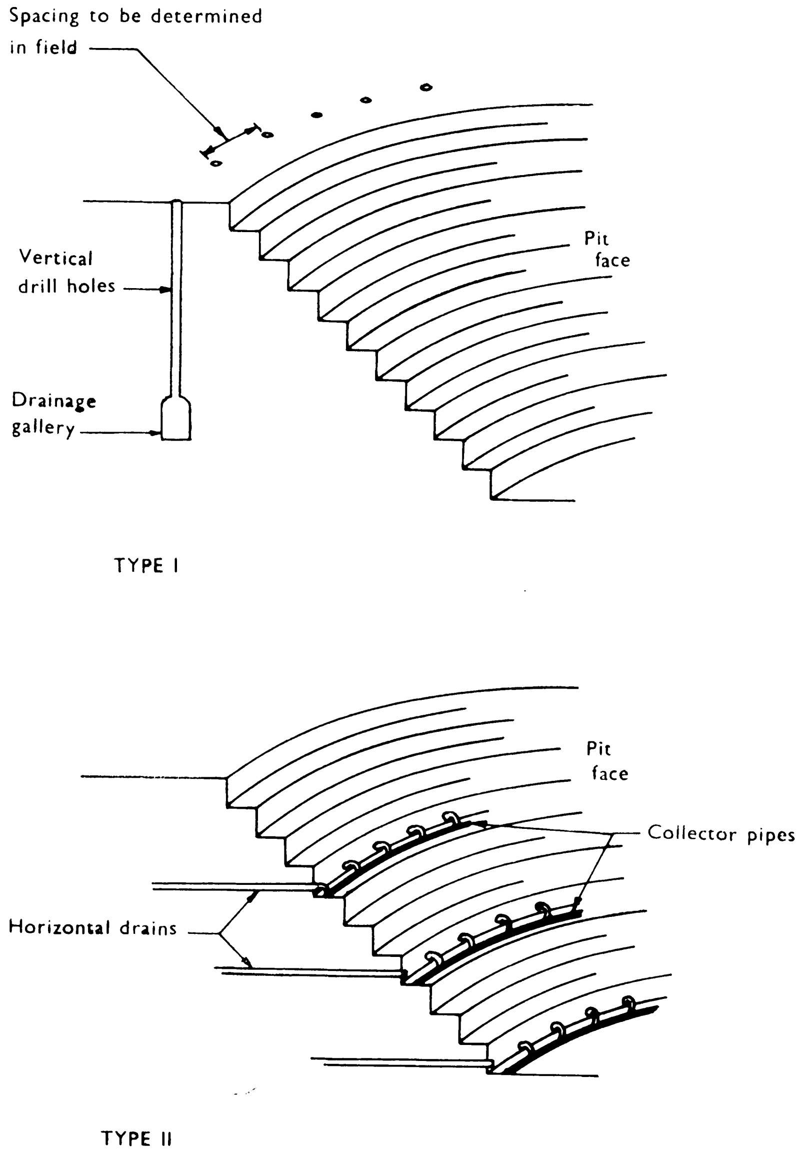


FIG. 11. Two types of corrective treatment to relieve uplift pressures in back of slope face.[5]

Type I: Gallery with vertical holes.
Type II: Horizontal drains.

The "paint rock" is a soft, incompetent, fine grained mass of quartz, pyrolusite, and kaolin with subangular fragments of chert, hematite, and goethite. The formation is generally made up of laminations less than 1 cm thick.

The height recorded is that of the actual slide zone before failure. The cohesion listed was calculated by using an effective angle of internal friction of 37° and assuming the ground water level behind the slope to be half the height of the slope. Circular arcs of failure with tension cracks equal to 50 per cent of the height of the slope were assumed in making these analyses.

Table 2 lists data for six slides which occurred in the pit walls of the Ruth Lake Mine of the Iron Ore Company of Canada in northeastern Quebec and Labrador. These slides occurred in the Ruth slate and the Wishart quartzite. The Ruth slate is finely laminated, ferruginous, and is commonly interbedded with thin layers of chert. Near the ore the slate is composed chiefly of an intimate mixture of very fine grained quartz, sericite, kaolinite, and iron oxides. On a sample passing the No. 4 sieve about 55 per cent of the particles are smaller than 200 mesh. The Wishart quartzite is generally composed of equi-granular quartz grains cemented by an

TABLE 1. PAINT ROCK SLIDES (STEEP ROCK MINE)[8]

Slide No.	Date	Height (ft)	Width (ft)	Slope angle	Cohesion (psf)
1	11.3.60	215	290	51°	1490
2	1.60	163	240	51°	1150
3	–	51	300	60°	685
4	24.11.60	173	120	50°	935
5	6.12.60	138	125	56°	1250
6	11.1.60	119	100	54°	950
7	4.3.61	86	150	49°	660
8	7.52	115	50	57°	825
9	11.48	95	100	66°	1195

TABLE 2. SLATE AND QUARTZITE SLIDES (RUTH LAKE MINE)[8]

Slide No.	Date	Height (ft)	Slope angle	Cohesion (psf)
1	29.6.58	100	36°	410
2	19.7.61	136	41°	765
3	17.11.60	114	42°	665
4	13.9.61	132	40°	695
5	4.11.61	140	43°	860
6	31.5.62	106	44°	690

extremely fine-grained quartz. Near the ore bodies the rock is altered by the partial removel of the quartz cement, leaving a rock that is more or less friable.

The cohesion was calculated assuming the effective angle of internal friction to be 34° (minimum as determined by laboratory tests) and the ground water level to be one-half of the slope height and that tension cracks extended to a depth of 25 per cent of the slope height.

TABLE 3. "ASH ROCK" SLIDES (STEEP ROCK MINE)[8]

Slide No.	Date	Height (ft)	Width (ft)	Slope angle	Cohesion (psf)
1	7.52	160	350	48°	840
2	8.59	100	200	56°	820
3	6.53	85	50	61°	780
4	27.10.60	85	200	63°	1160
5	12.12.60	145	125	40°	610
6	2.3.61	85	125	46.5°	410
7	–	445	450	51°	2620

Table 3 lists data for seven slides which have been recorded in the "ash rock" which forms the hanging-wall of the Steep Rock ore zone. This rock varies from a soft altered material near the iron ore to a more competent, brittle, somewhat schistose material. A typical specimen contains dark green to black lenticular, aphanitic, serpentinized fragments generally less than $\frac{1}{2}$ in. in size in a greenish schistose matrix. The rock is probably a pyroclastic of an unusually basic type.

The cohesion was computed assuming that the effective angle of internal friction was 40°, that the ground water level was at one-half of the slope height, and that tension cracks extended to 50 per cent of the depth of the slope, and that failure occurred along circular arcs.

Figure 12 illustrates the relationship between height of slope and slope angle for various conditions. The "no slide" conditions shown by curve I would be maintained by flattening the slopes to approximately 31° when the pit depth reached 500 ft. If the pit slope were steepened to $33\frac{1}{2}°$ then slides could be expected to occur along 10 per cent of the slope length as indicated by Curve II. If the groundwater level could be lowered to below the bottom of the slope then the pit slope could be steepened to about 37° before slides would be expected to occur along 10 per cent of the slope length as shown by Curve III.

ROCK MECHANICS ANALYSIS APPLIED TO SLOPE STABILITY

In order to predict slope stability by the principles of rock mechanics a detailed field investigation is required. Joints, faults, bedding planes must be mapped and orientation recorded so that information on all these features will be available for study.

Slopes in Mechanically Anisotropic Rock

"Unfractured" rock does not exist in nature. Even the most massive granite such as is encountered in the better quarries is separated by occasional joints. Thus, the high cliffs and escarpments which occur in nature exist because the planes of weakness in the rock do not coincide with the orientation of potential failure planes. A pile of bricks dumped in a heap with random orientation will be limited

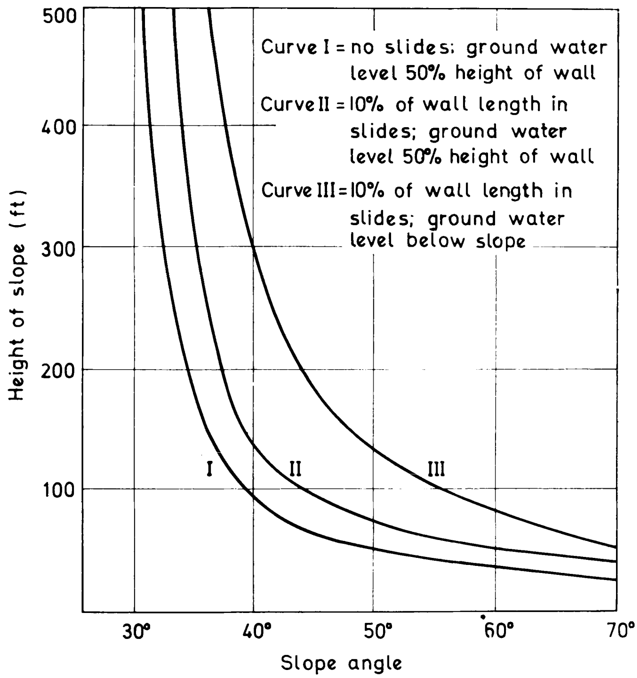


FIG. 12. Typical stability curves for incompetent rock.(8)

to the slope determined by the angle of friction. If the same bricks are piled symmetrically, they can form a vertical escarpment many tens of feet high.

To obtain a basis for calculating the maximum stable slope in mechanically anisotropic rocks it is necessary to determine the directions, the extent, and the relative structural importance of the various planes of weakness in a rock mass. The basis for such "Statistical Joint Measurement" has been laid by Stini (Vienna) and Sanders (Innsbruck).(9)

The result of "mechanical anisotropy" of rock can be observed in nature where a mountain, for instance, is shaped differently in different directions. For that reason an engineer must design the slopes of open pits with different slopes at different locations if he wants to give the same degree of stability to every section.

Figure 13 shows a non-symmetric open-pit mine with rock slopes designed according to varying structural characteristics of rock at different depths.

Determination of Maximum Slopes

The mechanical anisotropy of rock, as well as discontinuities, makes it much harder to describe in mathematical terms than is the case with soils.

Surface mapping will not provide sufficient structural information. Such information should include the orientation, extent of joints, stratification, and foliation lines in each individual zone, and should be complete enough so that a statistical

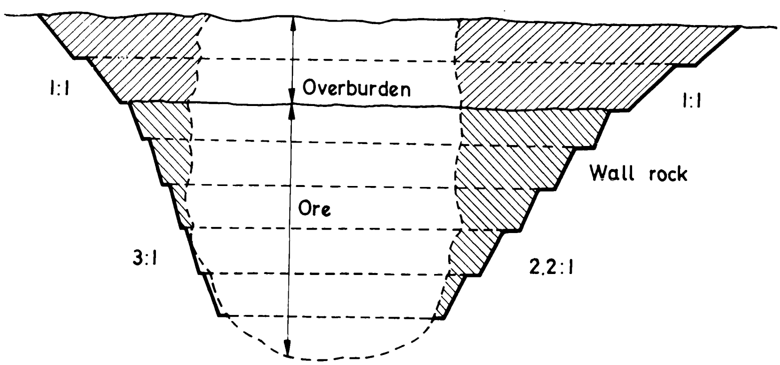


FIG. 13. Unsymmetrical open pit mine.[9]

analysis can be made of the planes of weakness. Numerous boreholes will be required, and core must be mapped inch by inch. Walls of boreholes may be photographed with a borehole camera, or scanned with a television borehole camera. Boreholes should be oriented to obtain the maximum amount of information relative to the planes of weakness. Hundreds (or thousands) of individual observations may be required to obtain enough information to allow plotting and statistical analysis to determine directions and magnitudes of principal structural weaknesses.

The relative stability of the rock and the maximum permissible angle of slope are determined by a number of factors. Muller[9] lists the following:

(1) The type of rock.
(2) The strength of the rock.
(3) Stratifications and foliation.
(4) Mechanical fragmentation.
(5) Chemical defects.
(6) Positional relations between slope plane and structural elements.
(7) The time factor.
(8) Water in rock joints.
(9) The effect of vibrations.

The analysis of the stability of a rock mass requires that the data on orientation, extent, and width of planes of weakness be expressed by graphical means. This may be done by plotting these weaknesses so that these planes pass through the center of an imaginary sphere. The points at which these planes intersect the surface of the sphere are then projected onto a circle equal in area to a hemisphere and the relative "density" of the points indicates the orientation of the planes of the most structural importance.

In addition to the orientation of weaknesses, the amount of separation between surfaces is also of interest. In other words, are the planes open joints, or are they tight, such as cleavage planes? The different types may be designated by letters.

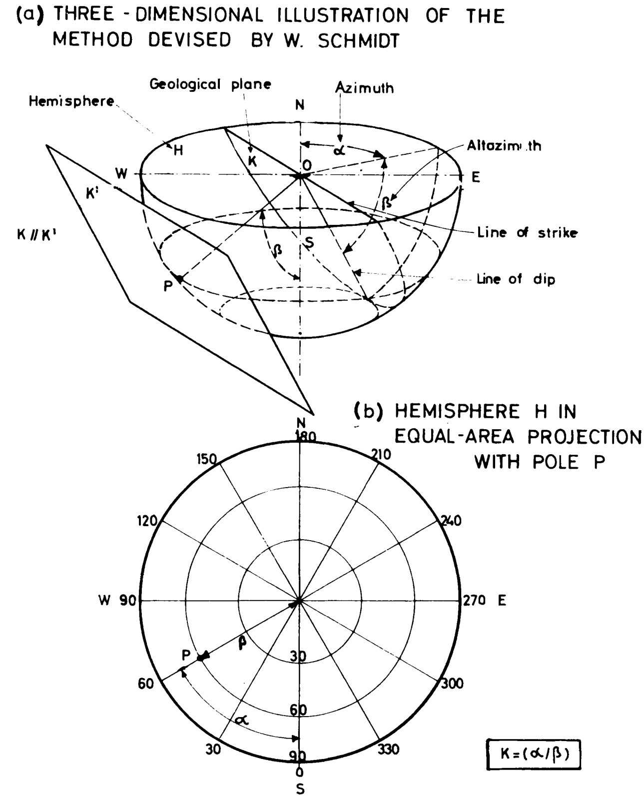


FIG. 14. Representation of geological planes. (After W. Schmidt)(10)

After establishing the direction of orientation of the most numerous, or the most important, planes of weakness, the next step is to determine the direction of the maximum shearing stress and its relationship to the planes of weakness. The rock will evidently have the least structural strength when the orientation of the pre-existing fractures is the same as the orientation of the maximum shearing stress. A rock will have the greatest structural strength when the pre-existing planes of weakness are orientated approximately normal to the directions of the maximum shearing stress.

Orientation of Slopes

Another method for increasing slope stability involves the lateral orientation of slopes. In some instances it is possible, by re-orienting the strike of a bench (that is, by turning it by 10 or 20°) to greatly increase the angle at which a slope will stand. The structural investigations carried on before mining starts should be thorough enough to indicate the most favorable orientation for benches.

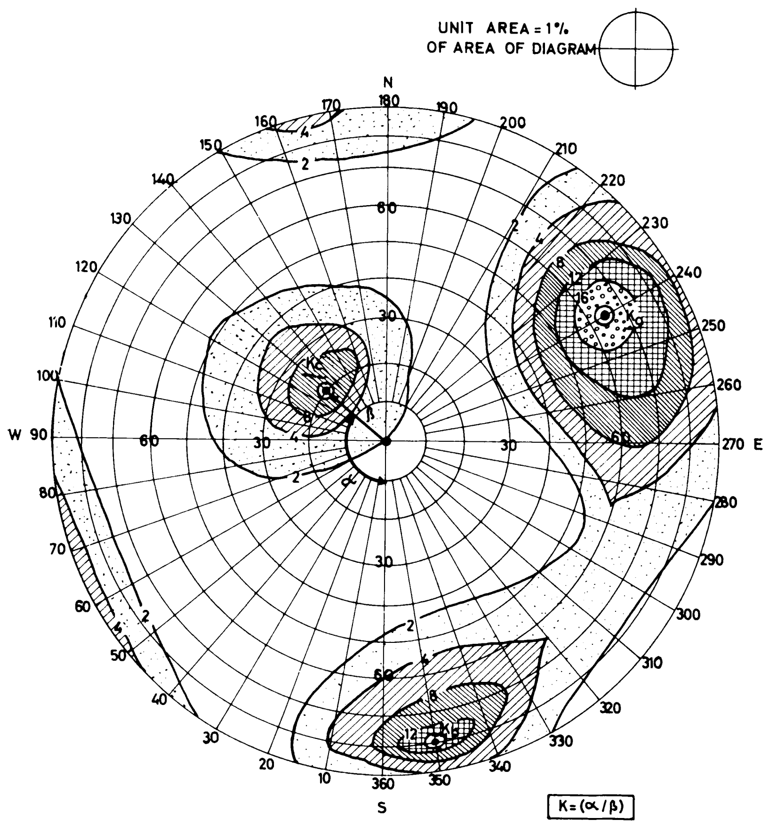


FIG. 15. Contour diagram representing the orientation of geological joint system.[10]

Theoretical Methods in Rock Mechanics

John[10] has presented a comprehensive treatment of methods for studying stability conditions of rock slopes. Figure 14 illustrates one of the polar projection methods used by geologists for graphically representing the planes of weakness in a rock mass. This diagram represents three-dimensional planar features, such as joints, faults, dips, and vectors on a two-dimensional chart.

Referring to Fig. 14 the joint plane K is positioned at the center of the hemisphere H; the line O–P, normal to the plane K, will pierce the hemisphere at the pole P, thus representing the orientation of plane K. The poles of all joints surveyed are plotted on an equal-area projection of the hemisphere, resulting in a point diagram. From this, a contour diagram as shown in Fig. 15 is developed.

The contour diagram utilizes an equal-area projection of the lower hemisphere, with the indication of the lines of equal numbers of joint measurements per unit area. This system produces a statistically representative distribution of the points representing the orientations of the individual joints of the system.

The hypothetical joint system illustrated features two sets of predominant joints, K_a and K_b, which are approximately normal to each other, and one set of less important joints, K_c, in an approximately horizontal orientation. Their orientations, as indicated in Fig. 15, are expressed as α/β, $K_a = (240/65)$, $K_b = (350/80)$, and $K_c = (130/20)$.

Monahan[11] has commented as follows regarding such polar diagrams:

> A polar diagram is frequently all that is needed to serve as a base for the judgment of slope stability because, if natural or proposed slopes are also plotted on it, a glance will show if the proposed slope will undercut a joint system. Such qualitative appraisal can, however, be applied only to simple cases and, being qualitative, can not give means of obtaining a relative safety factor to be assigned to the different slopes under study.

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